SME Mining Engineering Handbook

2nd Edition
Volume 1

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Cosponsored by
Seeley W. Mudd Memorial Fund of AIME

Published by
Society for Mining, Metallurgy, and Exploration, Inc.
Littleton, Colorado · 1992

2nd printing: December 1996

Printed in the United States of America by
Cushing-Malloy, Inc., Ann Arbor, MI

Library of Congress Catalog Card Number 92-61198
ISBN 0-87335-100-2

Preface

It was in 1918 that Robert Peele, assisted by 46 collaborators, edited the first edition of the classic *Mining Engineers' Handbook.* Like the second and third editions, released in 1927 and 1941, respectively, John Wiley & Sons, Inc., was the publisher. No doubt, there are a number of mining engineers in practice today who remember and still refer to Peele's highly regarded work.

By good fortune, in 1967, when a new handbook was sorely needed and Wiley declined to publish a new edition, the Society of Mining Engineers of AIME (now the Society for Mining, Metallurgy, and Exploration, Inc.) rose to the occasion. With Arthur B. Cummins as chairman of the Editorial Board and Ivan A. Given as editor, a first edition of the new *SME Mining Engineering Handbook* appeared in late 1973. Timely and authoritative, it has served the profession well for nearly two decades.

There is now need for a new and completely revised edition of the *SME Handbook.* Anticipating my coming retirement from The University of Alabama, I proposed to the SME editorial staff in 1987 that a second edition be published, and that I would be willing to serve as senior editor. Accordingly, we assembled an Editorial Board of 6 associate editors and 20 section coordinators, established a target release date of 1992, and set to work recruiting a collaborative team of 242 chapter authors. To the credit of all contributors — and under the resolute direction of first Marianne Snedeker and then Barbara Dygert, SME Managers of Book Publishing —this latest edition of the *SME Mining Engineering Handbook* is being published, on schedule, in 4 1/2 years!

In one of its first transactions of business, the Editorial Board grappled with a statement of purpose for the proposed Handbook:

> "By our thinking, a handbook is a comprehensive reference work, containing the distilled body of knowledge that characterizes a disciplinary field. While it may serve incidentally as an advanced textbook for students, its primary function is to provide professional practitioners with an authoritative reference and design source. In a field as practice-oriented as mining engineering, that is both a definitive and demanding charge.
>
> "To a lesser extent, the handbook should also serve nonprofessionals who seek technical knowledge of the field of mining. These include government officials, journalists, attorneys, economists, and other interested individuals.
>
> "It is the careful selection of pertinent subject matter and the manner and style in which it is presented, more than its organization, that distinguishes a handbook. However, the outline adopted for the revised *SME Handbook* benefits from a logical, generic subdivision of mining engineering knowledge; it is not unlike that of a comprehensive textbook. But the contents of a handbook are quite different, because design principles, relevant equations and tables, conceptual methods, illustrative examples, and case studies are stressed."

Intentionally, this statement of purpose has guided our labors throughout.

Further objectives related to subject-matter coverage and organization are as follows:

1. Appropriate attention is given to all branches of mining — metal, coal, and nonmetal — and to all locales of mining — surface, underground, and hybrid.
2. While major emphasis is placed on American mining, numerous references are made to practice abroad.
3. To facilitate international usage of the *Handbook,* mathematical and dimensional notations are cited in dual units, generally English first followed by SI in parentheses.
4. Appropriate references and/or bibliographies are grouped with each chapter for convenience.
5. Appendices follow the main body of the *Handbook.* They consist largely of tables of data, supplementing the text and useful for problem solving.

Credit for any eminence these volumes may attain belongs to the individuals who shared their knowledge, time, and expertise gratuitously. But recognition for a unique contribution properly goes to our professional organization, the Society for Mining, Metallurgy, and Exploration: it is the Society's enlightened policy of disseminating and publishing knowledge for the profession that makes this *SME Mining Engineering Handbook* possible.

Howard L. Hartman, *Senior Editor*
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Sacramento, California
August 1992

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Introduction

Section 1 Introduction to Mining

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Chapter 1.0
INTRODUCTION

HOWARD L. HARTMAN

It is appropriate in this opening section to introduce the broad topic of mining and mining engineering—the general subject of this entire *Handbook.* Hence, Section 1 in three chapters explores the history of mining, outlines the elements of mining, and discusses mineral engineering education. Coverage of history and education appears only in this section.

1.0.1.1 History of Mining

Contributions made by mining have played a much more significant role in the development of civilization than is generally conceded by historians or recognized by ordinary citizens. In modern society, mined products pervade all industry and the lives of all civilized people. Early man relied largely on stone and ceramics, and eventually metals, to fashion tools and weapons. Civilization was advanced by discoveries such as abundant supplies of high-quality flint in northern France and southern England and firesetting to break rock. Middle Eastern cultures flourished not only because of agriculture and trade, but also because of mineral-rich deposits nearby.

The earliest miners date back perhaps to 300,000 BC; their quest was for nonmetallic minerals (chert, flint, obsidian) suitable for utensils and eventually for weapons. Other rocks and minerals (ceramics, clay, salt, meteoric iron) attracted the miners for jewelry, cosmetics, construction materials, food seasoning, and coinage. At first, their excavations were confined to the surface, either pits or placers. But by about 40,000 BC, mine workings had been extended underground as short adits or shafts, and by 8000 BC as elaborate interconnected openings 300 ft (90 m) in depth.

Metallurgical separation of metals from their ores and their subsequent fabrication evolved gradually over the centuries, copper being the first liberated (c. 7000 BC) followed by lead, silver, gold, and iron. Man enhanced both the sophistication of his utensils and the lethalness of his weapons by an order of magnitude when his mineral frontiers first extended beyond the nonmetallics to the metallics. And the discovery and utilization of the first of the mineral fuels (coal) in the late 13th century AD carried civilization another quantum leap forward.

Much is learned both of mining development and human civilization when plotting a chronology of historical events. It is astonishing how well they correlate. In fact, one can track the major migrations of civilization westward and the discovery of the New World with the insatiable lust for mineral wealth.

1.0.1.2 Elements of Mining

Mining and mining engineering are similar but not synonymous terms. *Mining* consists of the processes, the occupation, and the industry concerned with the extraction of minerals from the earth. *Mining engineering,* on the other hand, is the art and the science applied to the processes of mining and to the operation of mines. The trained professional who relates the two is the *mining engineer;* he/she is responsible for helping to locate and prove mines, for designing and developing mines, and for exploiting and managing mines.

The essence of mining in extracting minerals from the earth is to drive (construct) an excavation or an opening to serve as a means of entry from the existing surface to the mineral deposit. Whether the openings lie on the surface or are placed underground fixes the locale of the mine. The specific details of the procedure, layout, equipment, and system used distinguish the mining method, which is uniquely determined by the physical, geologic, environmental, economic, and legal circumstances that prevail.

Using scientific principles, technological knowledge, and managerial skills, the mining engineer brings a mineral property through the four stages in the life of a mine: prospecting, exploration, development, and exploitation.

NOTE: Chapter 1.2 follows the outline of the *Handbook*, previewing in turn the six major parts of the volume, subdivided into 25 sections.

1.0.1.3 Mineral Engineering Education

The training of mining engineers was one of the first specialized fields in engineering education. Originating in 1716 at the academy in Joachimstal, Czechoslovakia, mining and mineral engineering education is now offered at institutions of higher learning on a worldwide basis. In the United States, 37 colleges and universities currently award ABET-accredited degrees in mining and related fields of engineering (ABET is the Accreditation Board for Engineering and Technology, Inc.).

Disciplines encompassed by the generic term *mineral engineering* include mining, geological, environmental, mineral processing, and metallurgical and materials engineering, Numbers of mining engineers graduating in the United States range from 200 to 800 per year (BS, MS, and PhD degrees).

Mineral engineering is a broad educational field, in part because accreditational standards for engineering education are extraordinarily wide ranging. In addition to mathematics, the basic sciences, and professional courses, mineral engineers must master a variety of engineering sciences ranging from electrical circuits to thermodynamics and strength of materials. At present, nearly all US undergraduate curricula are four years in duration, although increasingly the bachelor's degree is followed by a year or two of graduate study.

As mineral engineering grows ever more complex and technologically sophisticated, there is greater emphasis in the curriculum on computers, systems, and related topics. Likewise, there is a liberalizing effort underway to "humanize" and to stress social responsibility in the engineer's education.

Chapter 1.1
HISTORY OF MINING

WILLARD C. LACY AND JOHN C. LACY

1.1.1 A CHRONOLOGY OF EVENTS

History is much more than dates of political events; it is written in civilization's tools, weapons, workshops and factories, roads, bridges, canals, railways, laboratories, churches, housing and schools, laws, organizations, books, art and music, and medical and dental care. It is shaped by scientists, engineers, farmers, industrialists, and entrepreneurs. It is driven by economic concerns, fixed and circulating capital, supply and demand, wages and prices, expansion and contraction of markets, competition and monopolies, shortages, gluts, substitutions, trade cycles, crises, restrictions, diplomacy, and war. It has also been critically influenced by the availability to mankind of industrial minerals, metals, and fuels.

Table 1.1.1 attempts to list chronologically and geographically many historical developments, both those that have influenced the growth of mining and the mineral industries, and those that mining has influenced.

Viewing developments of mining technology purely on the basis of chronology, however, is misleading because technology advanced and declined in irregular geographic patterns throughout the world. For example, while Greek mining technology was well advanced by 500 BC, Britain remained in a primitive stage until Roman civilization arrived at the end of the 1st century AD, and Australia remained in a Paleolithic state until British colonization. Gunpowder, though in use by the 8th century in China, was not introduced in Europe until the 13th century, and still was not recorded as being used in mining until the 18th century. Steam as a source of power for pumping, first harnessed in the 17th century, was not effectively used in mining until a century later.

1.1.1.1 The Progress of Metallurgy

The earliest miners, dated back to perhaps 300,000 BC, were concerned with chert and flint for tools and weapons. Their quarries and pits led first to adits and shafts and finally to underground mining during the Neolithic Period (8000 BC to 2000 BC). Using crude stone picks and hammers, these early miners surprisingly reached depths of 300 ft (90 m) in the soft chalk of northern France and southern England. With this technology, mankind also directed its attention to metallic ores.

Metals were first appreciated only as stones, and from 7000 BC to 4000 BC, metallurgy centered on copper and gradually evolved from simple hammering to hammering and annealing, to melting and casting, and finally to alloying to produce desired characteristics of melt, hardness and flexibility. The pyrometallurgical technology of the West had its origin in the Near Eastern zone of ancient Anatolia (Turkey), Syria, Egypt, Iraq, and Iran. By the 6th millennium BC, these craftsmen were able to produce furnace temperatures of more than 1981°F (1083ºC) using forced draft, a technological breakthrough that launched the Metal Eras.

Ancient metallurgy entailed selection of compatible ores and fluxes and avoidance of incompatible ones. By the 2nd millennium BC, the eastern Mediterranean people were able to engage in mass production of copper, lead, and silver from oxide and sulfide ores and improve physical properties by addition of alloying elements. It was also found that by blending of ores from different localities, the metal product could be improved and controlled, and that ores containing iron-rich minerals, sea shells, or silicate minerals fluxed and aided the smelting process. Thus they were added if not already present in the ores.

Iron came into use as a byproduct of the smelting process for other metals in Anatolia where gossans were used as fluxes and iron formed as a part of the slag. China, however, was the site of great improvement in iron smelting and casting technology during 475 BC to 220 BC where pig iron was produced containing 3.5 to 4.5% carbon at a melting point of approximately 2282ºF (1250ºC). The Chinese also at this time developed new innovations in gilding bronze and inlaying of bronze and iron implements with gold and silver.

1.1.1.2 Mining Methods

The silver-lead mines of Laurium, near Athens, Greece, were first worked and abandoned by the Myceneans in the 2nd millennium BC, but opened again by the Athenians beginning about 600 BC. The earliest workings were open cast with short adits. Later, more than 2000 shafts were sunk and connected by drifts. Shafts were sunk in pairs with parallel drifts driven from them with frequent connecting crosscuts to aid ventilation. Stoping of ore bodies was either by overhand or underhand methods, and room and pillar methods were used for the larger stopes. Progress was slow, and it has been calculated that in shaft sinking a miner averaged about 5 ft/month (1.5 m/month).

It was in water-pumping devices that Roman mining showed the greatest advance. They drained the copper mines of Rio Tinto, Spain, and others as far away as Britain. The most important of these dewatering devices was the water wheel and the Archimedean screw.

The migrations of the Celts, originally from south-central Europe, was probably the single most important factor in the dissemination of mining technology throughout Europe. These migrations and general nomadic tendencies of the Celts not only spread knowledge of mining techniques, but many legal concepts can be traced to their tribal customs. The Celts, who permanently settled in the metal-rich areas of central Germany, became the Saxons and led the way to advances in mining during the Middle Ages, not only in their own country, but throughout Europe. They began mining at Schemnitz, Czechoslovakia, as early as AD 745; at Rammelsberg, Saxony, by 970; at Freiberg, Saxony, by 1170; and Joachimsthal, Bohemia, in 1515. Thereafter, when mines were opened in Spain and the famous silver mines at Konigsberg, Norway, Saxons skilled in mining and metallurgy were sent from Germany. By the 16th century the terms "miner" and "Saxon" had become almost synonymous.

Knowledge of mining technology was essentially a closely held craft that was dispersed by the mobility of miners. For example, the Celts that settled in Cornwall were the early tin miners. In 1240 a "tinner" fleeing England after having committed murder is reported to have taught the Germans to prospect for, mine, and concentrate metal ores; in 1562 the technology returned as Queen Elizabeth of England sent back to Germany for miners to introduce better mining and metallurgical practices in Devon and Cornwall. During the gold rushes of the 19th

Table 1.1.1. Chronology of Events Related to Mining

Period—Stage	Near East—Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
BC Paleolithic c.20,000 Late Paleolithic	c.40,000 Hematite mined for ritual painting (Africa) c.9500 Copper pendant. (Iraq)	300,000-100,000 Surface mining of flint (N. France, S. England) c.30,000 Use of fire, lamps, cave art, hunting with projectiles End of Ice Age c. 10,000 Gold ornaments		c.500,000 Use of fire (China)
c.9000 Mesolithic (Human Power)	c.8000 Development of agriculture (Egypt, Mesopotamia) c.7000 Burning of lime c.6500 Farming introduced (Greece) —Copper tools (Anatolia) c.6000 First pottery (Catal Huyuk) c.5000 Meteoric iron beads (Egypt) —Lead in use (Egypt) —Turquoise mining (Sinai) —Gold mining (Nubia) —Emerald mining (Red Sea) shafts to 300 m		c.6500 Farming introduced	c.8000 Development of agriculture (China)
c.4000 Neolithic—Chalcolithic	c.4000 Bronze casting (Egypt) —Copper smelting (Timna) c.3500 Wheel and plow invented (Mesopotamia) —Gold vessels (Iraq) —Stone quarrying (Egypt) c.3000 Bitumen mortar (Ur)	c.4000 Introduction of plow c.3500 Flint mines with shafts and galleries (France, Britain)		c.4000 Technology migration along trade routes
c.3000 Metal Age I Bronze	c.3000 Copper and stone artwork —Lost wax casting —Phoenician traders bring tin from Cornwall c.2700 Copper mined in Cyprus c.2600 Tin bronzes in use c.2500 Lead in use (Troy)	c.3000 Bronze weapons in use —Spread of copper mining —Gold from Ireland	c.3000 First pottery (Equador, Colombia) —Old Copper Culture, Lake Superior (US) —Hammering, annealing, grinding of copper (US)	c.3000 Bronze casting (Ban Chiang, Thailand) c.2800 Emperor Shen Nung discovers smelting (China)
c.2000 Metal Age II Bronze	c.2000 Pre-Hittites use iron —Silver separated from lead by cupellation (Anatolia) —Laurium Mine worked by Mycenians 1750 Hammurabi's code of laws (Babylonia) —Cu,Ag,Pb,Fe mined and smelted by Hittites c.1400 Gold mines of Nubia exploited —Use of bronze chisels 1280 Hebrew flight from Egypt taking metal technology		c.2000 First metalworking of gold (Peru)	c.2000 Gold and silver acupuncture needles (China) —Compound casting (China) —Coal dug and used as fuel —Miner lowest stratum of society (China) 1600-1400 Iron smelting and forging (Ban Chiang, Thailand)

Table 1.1.1. Chronology of Events Related to Mining (Continued)

Period—Stage	Near East—Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
Early Iron	c.1200 Jason uses fleece to recover fine gold c.950 Rio Tinto Mine operated by Phoenicians for silver c.670 Introduction of ironworking (Egypt) 650 First coins of silver and gold (Lydia, Turkey) c.600 Hanging Gardens of Babylon floored with lead 600-500 Bitumen mined (Baku) 510 Private leases granted at Laurium, royalties to citizens of Athens	c.700 Iron tools used in salt mining (Hallstatt, Austria)		c.800 Iron and bronze coins introduced (China) —Iron chains used for suspension bridges (China) c.600 Discovery of oil and gas while drilling for salt (China)
c.500				
Late Iron (Water Power)	c.500 Wootz steel made in India —Iron-making techniques spread to sub-Sahara 490 Athenian senate uses royalties from Laurium to finance naval construction 334 Alexander invades Asia Minor, conquers Egypt c.330 Aristotle writes *Meteorologica,* on how stones originate c.300 Theophrasus writes *Concerning Stones* (Greece) —Alchemy begins (Alexandria) 265 Punic wars begin for control of silver deposits of Iberia, Spain c.100 Water wheels in use—horizontal by Greeks, vertical by Romans	c.400 Celts mine placer tin in Cornwall using horn picks and wooden shovels with free miners	267 Platinum worked at Esmeraldas, Equador	c.500 Tombs contain gold (China) c.400 Steel weapons (China) 214 Great Wall commenced (China) c.200 Blowing of iron used 112 Opening of Silk Road
c.50 **AD**				
Greco-Roman	c.20 Romans use brass coins 23-79 Pliny writes books on earths, metals, stones, gems 117-138 Hadrian introduces strict regulations for treatment of miners —*Lex Metalli Vespacensis* recognizes miners rights, half to Crown —Villeinage introduced —Use of skilled artisans as miners c.220 Roman currency debased 320 Theodosian Code introduced (Roman Empire)	43 Roman invasion of Britain 112 Coal used by Romans in Britain, surface mines	c.0 Gold technology, lost wax method reaches Colombia	105 First use of paper (China) c.200 Iron casting a well developed art Han Dynasty, (China) 200-600 Gold and silver mining in China
500				
	528-533 Justinian compiles *Corpus Juris Civilis* in E. Roman Empire		c.500 American Indians use cinnabar and carnotite as decoration	c.500—Roman coins abundant (China)

Table 1.1.1. Chronology of Events Related to Mining (Continued)

Period—Stage	Near East—Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
Dark Ages	c.600 Use of windmills (Persia) —Gold and silver mines of Spain reopened by Moors c.800 Charlemagne renovates Roman mines in Italy c.1000 First Iron Age settlement in Zimbabwe	745 Mining begins at Schemnitz (Czechoslovakia) c.800 *Bergbaufreiheit,* right of free miner (Saxony) 938 Rammelsberg mine discovered (Saxony) 965 Gold strike in Harz Mountains (Saxony) c.990 Danelaw Courts of Aethelred II (England)	c.700 Gold working reaches Mexico c.900 Hopi Indians mine coal	c.710 Printing begins (China) c.900 Porcelain made (China) c.1000 Great age of Chinese ceramics and painting c.1045 First movable type (China)
c.1050				
Middle Ages	1185 Treaty of Bishop of Trent frees miners (Italy), expands miner's rights 1208 Mining customs announced in Trent (Italy) c.1215 Mining regulations of Massa (Italy) 1250 Forerunner of stock company in Genoa (Italy) 1298 Marco Polo writes *Description of the World,* includes formula for black powder (Italy)	1150 First adit in Ram melsberg 1170 Erzgeberge silver discovered 1195 Charter of Rights of Sovereign Princes recognizes discovery rights (Germany) 1198 Tin mines of Devon placed under supervision of warden (England) 1201 John I decree permits entry of unoccupied land for mining (England) 1210 "Sea coal" grant to monks of Holyrood Abbey by Wm. the Lion (Scotland) 1217 Forest Law of Henry III gave people right to use coal (England) 1219 Harz silver mining charter (Germany) 1238 First collieries established in Newcastle (England) 1240 Monks of Newminster Abbey granted rights to use "sea coal" for forge 1249 Iglau code (Moravia) 1250-1350 Energy crisis in Great Britain c.1250 Active coal mining at Liege, Belgium 1267 First recorded reference to steel (England) 1277 Oldest English brass 1288 Inquisition at Ashbourne defining miners' rights (England) c.1300 Philip **IV** frees serfs ending forced labor (France) —Coal diggings as trenches, irregular diggings, "bell pits;" coal raised in corves and windlass or backs of women (Britain)	1300 Indians in N. America learn to burn "black rock"	1275 Marco Polo arrives in China c.1300 Waterwheels used to power bellows (China)

Table 1.1.1. Chronology of Events Related to Mining (Continued)

Period—Stage	Near East—Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
		—Coal used by brewers and dryers 1305 Edward II issues stannary charters for Cornwall and Devon (England) 1306 Edward II prohibits artificers from using coal (England) 1307 Commission of Oyer and Terminer appointed to enforce proclamation of Edward II c.1340 Mining by pit and adit, vertical shaft and horizontal gallery, manual windlass or jackroll (Britain)		
	1344 Publication of *Las Siete Partidas* (Spain)	14th century Code of Freiberg (Saxony) 1356 "Golden Bull" of Charles IV recognizes free prospecting and working of discovery (German Empire) 1370 Human-powered water lifting devices used at Rammelsberg mine (Saxony) 1379 First tax on coal (England)		
	1387 Decree of Juan I permits mineral exploitation on land of others, ⅔ to Crown (Spain)			
c.1400 Rennaissance		1413 Charles IV claims sovereignty over all mines, tithe to Crown, damages to landlord (France) 1415 Great Barmote Court (England) 1450 Gutenberg's printing press (Germany) 1451 Funken separates copper and silver (Germany) 1454 Rag and chain pump installed at Rammelsberg		
	1463 Water-powered blast furnace in use at Ferriere (Italy)	c.1475 Mining tools: pick, hammer, wedge, wooden shovel —Two opposing water wheels used for driving hoist (Hungary)		
	1492 Columbus sails west (Spain)	c.1500 Windlass used for hoisting buckets or baskets —Surface haulage by panniers on horseback —Miners form guilds for mutual aid and benefit (England)	1492 Americas discovered	
	1509 Portugese defeat Saracen fleet closing Indian Ocean	1509 Watch invented by Peter Henle (Germany) 1516 Silver strike at Joachimstal (Czechoslovakia) 1550 First lift pump, Joachimstal (Czechoslovakia)	1521 Cortes enters Mexico 1524 Mining of copper in Cuba	

Table 1.1.1. Chronology of Events Related to Mining (Continued)

Period—Stage	Near East—Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
	1584 Philip II of Spain issues code of laws to govern overseas mining	1553 Railroads introduced in mining (Czechoslovakia) 1556 1st edition *De Re Metalica,* Agricola (Germany) 1558 English law prohibits cutting of trees for iron smelting 1563 Scottish parliament prohibits transport of coal outside of realm 1565 Hemp ropes replace iron chains at Rammelsberg (Saxony) 1568 Justices and barons of Exchequer rule ownership of gold and silver extends to privately owned land (England) 1590 Patent issued Dean of York for coking "pit coal" c.1600 Energy Crisis (England) —Sir Wm. Gilbert recognizes earth's magnetic field (England)	—Copper, tin, silver mined at Tazco, Mexico, by Spaniards 1545 Discovery of silver at Potosi (Bolivia) 1550 Mendoza's Mining Code (Mexico) 1554 "Patio" method developed at Pachuca (Mexico) 1574 Francisco Toledo's Mining Code (Peru, Bolivia, Chile, Argentina)	
1600 Rise of Science Coal Age		1600 Founding of English and Dutch East India Trading Co 1605 Furnace with covered crucibles to protect from noxious coal fumes introduced for production of sheet glass (England) 1610 Scientific Revolution in Europe begins: Kepler, Bacon, Galileo, Descartes, etc. —Patent to Wm. Wright for furnace to melt "bell metal" with coal (Germany) 1612 First use of reverbatory furnace (England) 1613 Bore rods in use in exploration for coal —Use of gunpowder in mining (Germany) 1614 Napier discovers logarithms (France) —Robert Mansell granted a monopoly for use of coal in glass manufacture (England) 1615 Salomon de Caus suggests raising water by expansive power of steam (France) 1619 Patent to "Dud" Dudley for smelting iron with "pit coal" (England) 1627 First use of drilling and blasting at Oberbeiberstollen mine, Schemnitz (Hungary)	1603 Philip II of Portugal issues code of laws to govern mining (Brazil) 1606 Charter granted to London and Plymouth Co. by James I specified royalties $1/5$ precious metals, $1/15$ copper 1609 Barba invents "kettle and cooking" process for silver ores (Bolivia)	

Table 1.1.1. Chronology of Events Related to Mining (Continued)

Period—Stage	Near East—Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
	1669 Steno writes *De Solium Naturalitar Contento* (Italy)	1630 First use of railway, Beaumont mine, Schemnitz 1640 Chimney tax imposed (England), repealed by Wm. III c.1650 Longwall mining introduced at Shropshire colleries, bord and pillar in N. England 1659 Papin's steam engine (France) 1660 Founding of Royal Society, chartered 1662 by Charles II (England) 1665 John Woodward writes "An Essay Toward a Natural History of the Earth and Terrestrial Bodies, Especially Minerals" (England) 1671 Railway and wagons used to convey coal from Ravensworth to Team Staith (England) c. 1675 Methane explosions in English coal mines 1689 Drilling and blasting introduced at Cornwall 1692 Lead smelted with coal 1694 British patent to extract tar and oil from stone 1698 Sir Humphery Mackworth employs coal in smelting copper (England) 1699 Thomas Savery reads paper to Royal Society on vacuum principle engine used at Grentwork mine, Cornwall	1636 Founding of Harvard College 1646 First successful use of blast furnace in N. America at Saugus, MA 1679 Coal discovered in Illinois 1692 Lead discovered in Mississippi Valley by Nicholas Perot 1693 Gold discovered at Minas Gerais (Brazil)	
1700				
		1709 Abraham Darby uses coke to smelt iron (England) —Kaolin mines established near Meissen, Germany 1712 Newcomen engine installed at Wolverhampton, Shropshire (England) 1713 Coke used at Coalbrookdale foundry (England) 1714 Henry II prohibits foundries from using coal (France) 1716 Mining school established at Joachinstal, Czechoslovakia 1718 Cornish tin mines dewater by pumping 1720 Zinc smelting at Swansea, Wales —"Bubble Act" limits stockholders to eight (England) c.1730 Cast iron used for mine pumps, pipes, and steam engine cylinders (England)	1718 SW Missouri lead mines commence work at La Motte mine 1727 Diamonds identified (Brazil) 1730 Brazilian diamonds declared property of Crown of Portugal	

Table 1.1.1. Chronology of Events Related to Mining (Continued)

Period—Stage	Near East—Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
		1740 Crucible steel method rediscovered by Huntsman (England) —Zinc produced on commercial scale (Wales) 1743 Marggraf publishes method to produce zinc (Germany) 1744 Gold mining in Russia 1747 Hutton writes *Theory of the Earth* (Scotland) 1749 Roebuck introduces lead chamber process for sulfuric acid manufacture (Britain)		
1750 Industrial Revolution (Britain) (Steam Power)	1752 Large-scale copper leaching at Rio Tinto (Spain) 1783 Tungsten discovered in Spain	1751 Crucible steel commercially established (Britain) 1759 Lebon's coal-gas patent (France) 1767 Cast iron rails for railroads introduced (England) 1769 Watt patents steam engine (England) 1770 Reverbatory furnace introduced (England) c.1770 Period of great advance in science: Priestley, Lavoisier, Volta, Watt, Harrison (Britain, France, Italy) 1774 Emancipation Act abolishing villeinage (Scotland) 1775 Abraham Werner inaugurates "Neptunist" school of geology at Freiberg (Germany) —Underhand cut and fill stoping used at Cornwall 1768-82 Watt and Boulton introduce double-acting condensing steam engine for dewatering Cornish tin mines 1781 Cornish pump introduced 1783 Henry Cort patents rolling mill (England) 1784 Henry Cort introduces puddling process for malleable wrought iron (England) 1789 Wheeled trams on light rails used underground (England) —Uranium discovered in pitchblende by Martin Klaproth (Germany) c. 1790 "Friendly Societies," forerunners of union, organize (England)	1770 Artur de Sa Mining Code for working mines at Minas Gerais (Brazil) 1771 Diamond Code (Brazil) 1783 Charles III (Spain) approves mining ordinances for "New Spain" 1785 Land Ordinance reserves 1/3 gold, silver, lead, and copper mines to Confederation (US) 1788 Benjamin Waterhouse begins course in geology and mineralogy at Harvard (US) 1792 Royal Seminary of Mining established in Mexico	

Table 1.1.1. Chronology of Events Related to Mining (Continued)

Period—Stage	Near East—Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
1800				
Industrial Revolution (United States)		c.1800 Underhand cut and fill replaced by overhand cut and fill at Cornwall and Saxony 1800 Trevithick introduces high-pressure steam engine at Cornwall 1802 Trevithick builds locomotive for travel on roads (England) 1804 Trevithick builds railway locomotive at Cornwall 1807 France codifies partnerships and commercial companies 1809 Davy names and attempts to isolate aluminum 1812 George Stephenson alters pumping engine to make it haul coal (England) —First locomotive and first successful steamboat in Europe —Cylinder printing press invented (England) —Trevithick applies plunger pump with Cornish pump at Prince Henry mine (Cornwall) 1813 Trevithick introduces boring machine 1815 Davy and Geo Stephenson invent first successful mine safety lamp (Britain) —R. W. Fox utilizes earth electrical currents to locate sulfide ore body (Cornwall) 1820 Coal gas used for lighting conveyed through iron pipes (England) 1821 Faraday invents electric motor and generator (England) —Berthier discovers bauxite at Les Beaux, France 1825 First passenger railway (England) —Oersted obtains aluminum powder (France) —Labor unions given legal status (Britain) 1827 Robert Stephenson designs the "Rocket" locomotive (England) 1829 Jigs introduced in Britain 1831 Bickford develops safety blasting fuse at Cornwall 1833 Pattison's process for silver extraction (Britain) —"Man-lift" invented at Clausthal (Saxony) —Wire rope used for hoisting at Clausthal	1803 Alvaria implements new mining code (Brazil) 1804 Evan's high-pressure steam engine (US) 1807–1846 Leasing of lead mines in Illinois, Iowa, and Missouri (US) 1811 New York allows corporations with limited liability 1828 Gold discovered in Georgia (US)	

Table 1.1.1. Chronology of Events Related to Mining (Continued)

Period—Stage	Near East—Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
		—British regulations for industrial working conditions 1834 Introduction of guided cage hoisting of coal carriages and flanged wheels on carriages (England) 1835 *Mining Journal* begins publication (England) 1836 Sorel's galvanized iron (France) 1838 First electric telegraph in Britain 1843 Brunley launches iron ship, "Great Britain" (England) —Legislation against employment of women and boys under 10 underground in mines (Britain) 1844 Rittinger invents shaking table (Britain) 1845–49 Irish potato famine 1846 Standard railway gage introduced in Britain 1847 Elie de Beaumont writes "Sur les Emanations Volcaniques et Metaliferes" (France)	1837 First edition *System of Mineralogy* by Dana (US) —Otis invents steam shovel "Yankee Geologist" (US) 1840 Copper discovered on Keweenaw Peninsula, MI, by Douglas Houghton (US) 1842 Geological Survey of Canada established 1844 Cliff mine begins operation (Lake Superior) 1845 Chief Manjekijik showed S. Carr iron deposit in Michigan (US) —Sault Ste Marie Canal begun 1848 Gold discovered by James Marshall at Sutter's Mill, California —Patent granted on percussion drill capable of operating other than vertical (US)	1842 Treaty of Nanking opens ports in China and allows exit of low-cost labor 1843 Copper mining commences at Burra Mine (Australia)
1849 Rush for Gold		1850 Act for safety inspection of coal mines (England) —London School of Mines established —First mechanized rock drill (France) 1851-52 Derbyshire lead mining laws become a part of common law (England) —Longwall mining of coal in general use in Europe —First Mining Institute at Newcastle on Tyne 1854 Deville's aluminum displayed at Paris Exposition 1855 Luigi Palmieri invents seismograph (Italy) 1856 Bessemer process introduced for mass production of steel (England) —Siemens patents open hearth furnace for steel (England)	1850 Gold rush mining district regulations for California gold placer deposits (US) 1852 Extralateral right established, Nevada County, CA 1853 Polytech College of Pennsylvania founded; first institution in the US to offer degree in mining 1854 Gadsden Purchase from Mexico by US 1855 Roebling's wire rope bridge over Niagara (US)	1851 Gold discovered in New South Wales by Hargreaves (Australia) 1854 Mathew Perry opens Japan to foreign trade —Victoria Gold Regulations and Eureka Stockade riot (Australia)
1857		1855-62 Limited Liabilities Act made public financing possible (England) 1859 Von Cotta publishes *Treatise on Ore Deposits* (Germany)	1857 Wm. Kelly granted patent in US for air-blown iron converter process in steel production 1859 Discovery of oil at Titusville, PA	

Table 1.1.1. Chronology of Events Related to Mining (Continued)

Period—Stage	Near East—Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
Age of Steel		1860 Exploitation of Stassfurt potassium deposit 1862 Introduction of steel rails in England 1863 T. Harrison and Wm. Baird & Co. design chain cutter for use in Gartsherrie Collieries (England)	1860 Discovery of Comstock Lode, NV —Development of Pittsburgh coal seam (US)	1860s Discovery of gold in Queensland (Australia)
Petroleum Age		1864 Leschot invents diamond drill (France) 1865 Electrolytic refining of copper in Britain —Nobel invents dynamite (Sweden)	1864 Gold discovered at Butte, MT	
		1866 Siemens-Martin open hearth steel (Germany) —Prussian Mining Law (Germany)	1866 Lode Mining Act (US) —First packaged powder used, New Almaden mine, CA —First issue *American Journal of Mining* (Later E&MJ) (US)	
		1867 Mendeleev's classification of elements (Russia)	1867 Purchase of Alaska by US from Russia —J. H. Rae patents cyanide extraction of gold from ores (US)	
		1868 Siemens suggests gasifying slack and waste coal in situ (Britain)	1868 First dynamite produced in the US by Giant Powder Co. 1869 First diamond drill in US at Bonne Terre mine, MO	
		1870 Germany permits limited liability companies	1870 Standard Oil Co. incorporated with J. D. Rockefeller as president (US) —Copper discovered (Chile) —AIME founded —First manufacture of Portland cement in US at Allentown, PA —S. Ingersoll patents rock drill with universal tripod mounting (US)	
(Petroleum Power)			1872 Mining Act of 1872 (US), discovery and development required for possession 1874 First electric train (NY)	
		1876 Pneumatic drill introduced at Rammelsberg mine (Germany)	1876 Bell patents telephone (US) —Homestake mine discovered (SD)	
		1878 First oil tanker (Russia)	1877 Hodges and Armil inaugerate mechanical strip mining with Otis steam shovel (KS) —E. Schieffelin discovers Tombstone mine (AZ) 1879 USGS established —Edison demonstrates lighting system (US)	
1880				
Age of Electricity		1880 Ball mill and jaw crusher introduced at Cornwall —Steel structure used in buildings (England)	1880 G. Eastman makes popular photography possible (US) 1883 20-mule team hauls borax to Mohave Junction, CA	1883 Discovery of Broken Hill silver (Australia)

Table 1.1.1. Chronology of Events Related to Mining (Continued)

Period—Stage	Near East—Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
			1884 Codigo National Minera (Mexico)	
	1886 Gold discovered near Johannesburg, S. Africa	1886 P. Heroult codiscovers direct electrolysis of aluminum in molten cryolite (France) 1887 *Mining Yearbook* established (England) —McArthur & Forrest patent cyanide leaching and zinc precipitation of gold (Scotland)	1886 C. Hall codiscovers direct electrolysis of aluminum in cryolite (US) 1885 Wooden dredge on wheels used to strip overburden by Consol Coal Co. (IL) —First profitable gold discovery in Yukon	
Aluminum Age	1888 Formation of DeBeers Consol Mining Ltd. (S. Africa)	1888 Karl Bayer develops process for production of alumina (Germany)	1888 First electric hoist at Aspen, CO —Homestake mine introduces amalgamation (SD) —Discovery of Florida phosphate —Geol. Soc. Am. established	
		1889 Eiffel Tower constructed (France) —Fougue & Levy establish time/distance seismic plots (France) 1890 R. von Etvos develops torsion balance to measure gravity (Hungary)	1889 *Mining Record* began publication, Denver, CO 1890 H. Frasch patents sulfur extraction technique (US) —Brunton compass introduced (US) —Sherman antitrust legislation (US)	1889 Charles Potter, brewer, devises flotation process at Broken Hill (Australia)
			1892 Ley Minera (Mexico)	1892 Gold discovered at Kalgoorlie, W. Australia 1893 Aus. Inst. Mining and Met. founded (Australia)
(Electric Power)			1895 Sullivan mine discovered (Canada) 1896 G. Carmack makes gold discovery on Bonanza Creek (Klondike) 1897 J. K. Leyner introduces new drill (US) —American Mining Congress founded —Wilfley table invented (US)	
		1898 Pierre and Marie Curie observe radioactivity (France) 1899 Heroult produces first commercial steel with electric arc furnace (France)	1898 D. C. Jackling issues feasibility study on Bingham Canyon deposit (UT) 1899 "Cyanide Charlie" Merrill awarded contract to install cyanide processing at Homestake mine (SD) —Amer. Smelting & Refining organized (US)	
1900				
	1900 Copper mining begins at Katanga (Zaire)	c. 1900 Acetylene lamps introduced (Britain) 1902 Coal-face conveyor introduced by W. C. Blackett at Durham, England	1900 First mine geological department organized at Butte by R. Sales and H. Winchell —Great Mexican oil boom —Discovery of Gulf Coast salt dome sulphur deposits 1903 First flight of petroleum powered aircraft by Wright brothers, Dayton, OH —Discovery of Cobalt mine (Canada)	1901 C. V. Potter and G. V. Delprat patent flotation process (Australia) 1902 First commercial use of flotation at Broken Hill to recover sphalerite from tailings (Australia)
		1904 First electric locomotive introduced (Britain) 1905 Electric winder introduced to England from Europe	1904 First Dorr classifier (US) 1905 Mining commences at Bingham Canyon, UT —Soc. Econ. Geol. founded (US)	

Table 1.1.1. Chronology of Events Related to Mining (Continued)

Period—Stage	Near East—Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
			—A. Einstein proposes relation between energy and matter (US) 1907 E. L. Oliver introduces continuous vacuum filter (US)	
	1908 Oil discovered, Zagros Mts. (Iran)	1908 H. Geiger and E. Rutherford invent geiger counter (Britain) 1910 Synthetic cryolite patented (France)	1909 Ley Minera (Mexico) 1910 USBM founded —Mining commences at El Teniente mine (Chile) —Sudbury mine discovered (Canada)	
		1912–13 C. & M. Schlumberger demonstrate use of conductivity in locating ore bodies (France) 1913 Schilowsky demonstrates use of electromagnetic effect (Russia)	1911 Marion 250 stripping shovel introduced (US) —Breakup of Standard Oil 1913 Henry Ford develops conveyor assembly of Model-T (US) 1915 Electric coal-cutting machine introduced by Westinghouse Air Brake Co. (US) —Mining commences at Chuquicamata (Chile) 1916 Shrinkage stoping evolves into block caving, El Teniente mine (Chile) 1917 WAAIME founded (US) —AAPG founded (US) 1920 Mineral Leasing Act (US)	
			1923 LeTourneau introduces self-propelled scraper (US)	1923 Mount Isa mine discovered (Australia)
Atomic Age	1925 First off-shore oil drilling in Caspian Sea (Baku)	1925 Turbine drill introduced (Russia)	1925 Codigo Saavedra (Bolivia) 1926 Ley de Industras Mineralas (Mexico)	
Uranium Age		1928 C. & M. Schlumberger develop electrical logging of formations (France)	—Potash discovered (NM) 1929 Wall Street crash (US) 1930 Thermal blasthole drilling introduced at Mesabi (MN) —Potash fields in E. NM developed —Ley Minera (Mexico) 1932 Eimco introduces rocker shovel using compressed air (US) 1933 Euclid produces its first rubber-tired truck (US)	
		1936 Hydraulic coal mining used successfully (Russia) 1937 Jet engine first tested (Britain) 1939 Beginning of World War II	1936 Wilhelm Kroll produces pure ductile titanium (US) 1938 Nationalization of oil in Mexico 1939 Dart introduces electric-drive off-highway truck (US)	1937 Mount Isa's first dividend (Australia)
		1941 First electromechanical digital computer (Germany)	1940 Continuous coal mining introduced in US 1942 Gold mining restricted (US) 1945 Atomic bomb exploded (NM) 1946 A. Brant and E. A. Gilbert patent IP method of ore search (US)	1941 Japan attacks Pearl Harbor 1945 Atomic bomb dropped on Japan

Table 1.1.1. Chronology of Events Related to Mining (Continued)

Period—Stage	Near East—Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
(Atom Power)	1948 Birth of ARAMCo 1950 Uranium production begins at Witwatersrand (S. Africa)	1947 Marshall Plan for economic reconstruction of Europe 1950 First nuclear power station in Britain 1952 Oxygen steel process developed (Austria) —British computer LEO introduced	—Cerro Bolivar iron deposit discovered (Venezuela) 1946 First electric computer (US) 1948 Transistor invented (US) 1950 Codigo Minera (Peru) —General use of rock bolt support (US) —ANFO used in surface mining 1951 UNIVAC-1 installed at US Bureau of Census 1952 Patent filed for solution mining of uranium (US) —Nationalization of tin mines (Bolivia) 1952 Paley Commission Report 1953 First taconite concentration and pelletizing plant at Hoyte Lake (MN) —J. S. Robbins & Assoc. introduces tunneling machine (US) 1955 Successful synthesis of diamonds by General Electric (US)	1950 Discovery of iron ore in W. Australia 1955 AuslMM chartered (Australia)
1957 Space Age Age of Computers	1958 Bucket excavators installed by N'Changa Cons. Copper Mines Ltd. (Zambia) 1960 OPEC organized 1973 Arab oil embargo	1957 First space satellite launched (Russia)	1957 AEC "Plowshare" program established (US) 1959 J. L. Mero's thesis on "Economic Analysis of Mining Deepsea Phosphate" (US) 1960 H. S. Hess & R. S. Dietz propose ocean floor spreading (US) —Regular use of ANFO underground at Stanrock Uranium mine (US) 1961 Carlin gold discovered (NV) 1968 220-yd walking dragline excavator built by Bucyrus-Erie for coal stripping (US) —National Wilderness Preservation Act (US) 1970 National Environmental Policy Act (US) —Mining and Mineral Policy Act (US) 1971 US abandons gold standard 1972 Clean Water Act (US) —First LANDSAT spacecraft launched by NASA for remote sensing of earth resources 1973 Endangered Species Act (US) 1976 Toxic Substance Control Act (US) —Federal Land Policy and Management Act (US)	1966 China's "Cultural Revolution" 1975 Lake Argyle diamonds discovered (Australia)

Table 1.1.1. Chronology of Events Related to Mining (Continued)

Period—Stage	Near East—Ñ Mediterranean—Africa	Central and Northern Europe and Great Britain	North and South America	Australasia
		1986 Chernobyl nuclear reactor disaster (Russia)	1977 Surface Mining and Control Reclamation Act (US) 1979 Three-Mile Island accident in nuclear power plant (US) 1982 Commercial production of electricity by solar energy (US)	

Sources: See references.

century, the Cornish and German miners supplied mining technology throughout the world.

The first comprehensive record of mining and metallurgical methods was published in 1556 in Agricola's *De Re Metallica*, which describes mining and metallurgical technology as it had developed in the Ore Mountains of central and eastern Germany by the mid-16th century. Agricola's work was disseminated wherever there were miners, and its compendium of recorded knowledge provided a strong foundation for a rapid advance of mining education and technological advancement.

During Agricola's time, ground was broken by hammer and chisel (then and now the universal symbol for mining operations); fire setting was used in stoping rather than in driving, and then only when timber support was not required. Other notable advances were the first use of drilling and blasting at Schemnitz in 1727; the introduction of the pneumatic drill at Rammelsberg in 1876; and the replacement in hoisting of "kibbles" of wood bound with iron by hemp ropes first with wagons riding in cages hoisted by iron chains, then by wire rope in 1833 at Clausthal, Germany. Illumination evolved from tallow or oil-dip lamps used by the early Saxon miners, candles fixed in clay on helmets, ladders, or on rock faces by l8th-century Cornish miners, to 20th-century illumination provided by acetylene lamps and electric cap lights.

1.1.1.3 Milling and Smelting

The early miners and metallurgists were fortunate in having rich oxidized ores near the surface from which they could achieve adequate concentration by hand picking or simple washing. However, as grades declined and mineralogy became more complex with mineral grains finely interlocked, hand breaking and sorting were no longer possible. The most primitive mechanical grinding device used throughout Asia and the Near East was the Korean mill, a boulder rotating in a cupped stone. The arrastra, dating back to the Roman Empire, consisted of "drag stones" pulled over a circular paved area; the Chilean mill was similar except circular stone wheels were used in place of drag stone. Stamps came into use for primary crushing. All these devices were largely replaced by jaw and cone crushers and rod and ball mills about the turn of the 19th century.

The oldest method of concentration was by washing with water to remove light minerals and collect the heavy ones. This was accomplished by hand panning, rockers, or sluices with riffles or by passing the materials over a special cloth, as was the case with the fleece used by Jason and his Argonauts. The jig was the earliest mechanized gravity concentration device, developed about AD 1830, followed by the shaking riffled table. Magnetic separation was the second ore-concentration method to be developed in the 19th century.

Chemically induced concentrating began with amalgamation of gold on mercury-coated copper plates and dates back to the 4th century BC. Solution extraction on a large scale probably began with heap leaching and precipitation of copper at Rio Tinto, Spain, about AD 1752; solution of gold in cyanide solutions was known in Germany as early as 1805, but it was not developed as an ore treatment technique until 1887. It was this development that made the fine-grained gold of the Witwatersrand an economic source of gold.

Finally, the greatest breakthrough in the history of mineral beneficiation occurred at the turn of the 19th century when nearly simultaneously in Britain, Australia, and Italy, the technique of froth flotation evolved.

The mineral industry in the 20th century has been characterized by the recognition of economics of scale and development of technology for large-scale mining, bolstered by development of mineral flotation and hydrometallurgical techniques.

1.1.2 THE MINER'S CONTRIBUTION TO SOCIETY

The contribution of mining has played a bigger part in the development of civilization than is usually conceded by the historian or recognized by the ordinary citizen. In fact, products of the mineral industry pervade the lives of all members of our industrialized society.

Early man relied upon wood, bone, stone, and ceramics to fashion tools, weapons, and utensils. Civilization was advanced by the discovery of abundant supplies of high-quality flint in northern France and in the chalk beds of southern England. Culture after culture occupied the sites around the Acheuleum communities over a span of 200,000 years. Clay deposits supplied material for storage vessels as agriculture was introduced, and the metallic residues from pigments in the potters' kiln may have provided the first clue to these ancient peoples of the secrets of extraction of metals through smelting. Likewise, salt was recognized as essential in the human diet and, along with flint, became a prime medium of exchange that dictated early trade routes. During the initial development, the use of metallic minerals was in the form of pigments, decorative beads, and native metals that could be shaped into simple objects by hammering.

Most discoveries of these useful minerals were made by accident along trade routes. However, Egypt, which was not well endowed with mineral resources, sent out expeditions exploring for turquoise and gold as early as 4500 BC, resulting in an era of warfare for the acquisition of metals. The Mycenaeans followed by the Phoenicians broke this cycle of war and became wealthy, exchanging minerals for goods. These traders/prospectors sought deposits of silver, tin, lead, copper, and gold, acquiring them by barter rather than by conquest. By 1200 BC they had sea trade routes throughout the Mediterranean world, acquiring lead and silver from Spain, copper from Cyprus, and tin from Cornwall.

By 100 BC trade routes between China and the West, primarily for silk and spices, were well established. The roads passed through many countries and disseminated knowledge of "seric" iron (steel) and metallurgical technology to the known world. By 620, during the T'ang Dynasty, China had become the most advanced society in the world culturally and technologically. The fact that mining technology never fully developed in China can probably be attributed to Guatarma (563-483 BC), who taught that "suffering is caused by the craving for that which one has not," resulting in governmental policies that alternately discouraged and encouraged mining.

The discovery of copper on Cyprus c. 2700 BC resulted in the fabrication of tools, weapons, and household utensils made of metal and turned the island into an important trading center. Wealth poured into the island allowing for luxuries and artistic and religious development.

Work in the mines by the Greeks and Romans was first done by slaves, either prisoners of war, criminals, or political prisoners. Easily exploitable deposits were eventually exhausted and mine economics demanded mining skills. As a result, beginning with the reign of Hadrian (AD 138), the Roman Empire began to recognize a degree of individual ownership and permitted mining by freedmen in increasing numbers. There was gradual improvement of mining technology through the Roman Empire that accompanied replacement of slaves by skilled artisans, though villeinage was still practiced.

One legacy largely the result of Phoenician trading was to create a system whereby power and prosperity could thereafter be measured in terms of actual, exchangeable wealth. In this capacity, gold and silver throughout history have been universally accepted coinage. Thus debasement of the Roman denarius resulted in its loss of creditability as the standard of exchange, contributing to the fall of the Roman Empire, and by the end of the 6th century, the Latin West reverted to an agrarian economy and abandoned coinage and trade. The center of culture and technology shifted to the Byzantine and Islamic empires.

Charlemagne (768-814) recognized the need for metals and began the mining of lead, silver, and gold at Rothansberg, Kremnitz, and Schemnitz by enslaved captives. Charlemagne also reformed the coinage of his Holy Roman Empire, and these actions set in motion the establishment of new mints during the 10th century in Eichstadt (908), Cologne (960), Hildesheim (977), and Saxony (990), creating new and geographically dispersed demand for metals. Thus as Charlemagne's Empire gave way to more local kingdoms, a demand for precious metals had been created that aroused the spirit of enterprise and awakened interest in the development and use of metals; Europe saw a birth (or rebirth) of the traditions originally carried by the Celts of nomadic mining expertise. This rebirth was characterized as "bergbaufreiheit," or the rights of the free miner, whereby the poorest villein could become his own master merely by marking his own mining claim and registering its boundaries after making a discovery—subject to a tribute or royalty paid to the royal land owner. The miner thus ceased to be a serf and became a free man. The evidence of the foundation of this concept of self-initiation of rights to develop mineral ground includes a treaty initiated by the Bishop of Trent in 1185, where miners were invited to explore and mine that region of northern Italy as free men with rights of discovery; the charter of rights granted to miners by the various princes in the Germanic empire in 1209; and the results of the inquisition ordered by Edward II of England in 1288 to memorialize the ancient customs and practices of the miners within his realm. Thus the right of ownership based on discovery by a free miner became the foundation for mining laws carried by individual miners throughout Europe, then to the Americas, Australia, and South Africa.

Discoveries lured hordes of prospectors and miners, followed by farmers and merchants, eastward into Saxony and beyond to become settlers and developers of the land. Expansion east of the Rhine into the rich metalliferous province of Saxony resulted in discovery and development of mines at Schemnitz, Kremnitz, and Rammelsberg, and marked an awakening of metal mining, the revival of industry and trade, and the end of the Dark Ages in Europe. The metal from central Europe moved directly south to Venice and was largely responsible for the conversion of this poverty-stricken village of the 9th century into the richest port in southern Europe.

The affluence created by this industry had ultimate consequences in the arts, as Emperor Frederick II, supported by the wealth of Rammelsberg, became a noted patron of literature and science and contributed substantially to the Renaissance.

As mining extended underground, the free miner found that he could do little by himself, so he formed a partnership. As the operation grew, other men were required, and self-governing associations were born whose ownership and financial stake were supported by contributions memorialized in a "cost-book." The cost-book association formed the model for company organization before the practice of issuing stock as evidence of proprietorship. In the 13th century, the German cost-book association usually consisted of 16 able-bodied men. As the scale of operation increased, it was necessary to add additional participating shares, and Agricola notes in his time that the number of shares at Achneeberg was 128, of which 126 were private owners in the mine, one to the state, and one to the church.

Initially, production was divided among the shareholders, but as treatment and marketing became more complex, the sale became centralized. When a profit was made, it was divided among the "adventurers," but when losses were experienced the adventurers were required to contribute in proportion to their holdings or risk loss of their ownership. Rarely was any money set aside as a reserve, and consequently, a decline in metal prices or grade generally resulted in mine closure.

Growing demands for capital forced a search for outside capital, and gradually operators lost control to investors. The miners became contract workers. Guilds, originally organized by miners for charity and insurance, assumed objectives of industrial aggression.

When public financing in Britain was made possible through the enactment of the Limited Liabilities Acts of 1855-1862 and repeal of the Bubble Act that had limited stockholders to eight, British capitalists came to the forefront in financing mineral development worldwide. Goldsmiths assumed a banking function and issued printed receipts (or notes) payable to any bearer—the forerunner of present paper currency.

During the 18th century, iron metallurgy made great strides and made possible the Industrial Revolution in Britain. Village craftsmen evolved into the factory system and the "Friendly Societies" legally took on the function of the trade unions after 1825.

An industrial revolution is a period during which the economy of an underdeveloped country is transformed into an industrial economy, stimulated by availability of energy sources and metal resources. This change took place in Britain during the 18th and early 19th centuries and spread to France, the United States, Germany, Japan, Russia, Sweden, Canada, Taiwan, and

Korea in approximately that order. The developing technology was accompanied by a revolution in science and engineering, with empirical contributions from alert and observant workmen.

The machine age introduced by the Industrial Revolution of the late eighteenth century also required minerals as raw materials and as a source of energy. Industrial power thus became a measure of political and military power, and exploration for the acquisition of minerals resources extended to nearly all parts of the world. Nations' economies became interdependent. In an attempt to control the large-scale international flow of mineral resources, various commercial and political measures have been tried: monopolies, cartels, tariffs, subsidies, and quotas to name a few. The final result was that political and commercial control over mineral resources and their distribution played a leading role in both the maintenance and destruction of world peace (Leith et al., 1943).

No mention of coal mining appears in the historical record of the West until late in the thirteenth century. Early references are either to "quarries" or "drift" material. Development of coal mines in Britain, Europe, and the United States supplied energy that powered the industrial revolutions.

The structure of the coal industry traditionally was one of many small, low-capital, independent operators who supplied the retail and industrial markets, and a certain percentage of captive mines that supplied railroads, steel and electrical power requirements—all operating within their national boundaries. Depressed coal prices resulting from competition from oil and gas forced reorganization of the industry through aggregation by mergers and acquisition of small operations, mechanization, and a move to surface mining operations by larger companies able to afford this capital-intensive approach. This first occurred in the United States during the late 1950s and early 1960s, and internationally during the late 1960s and 1970s as coal markets were likewise expanded.

The petroleum industry not only shared in the technological developments, but along with coal was in a large part responsible for supplying abundant, cheap, and flexible energy and chemical raw materials. It created demand as well as responding to demand.

The story of aluminum is one of human ingenuity in the creation of a new metal, a new way of life, and a spate of new industries and technologies, as well as combined chemical, technological, and geological cooperation and discovery. Aluminum was not isolated as a metal until 1825 and has been in commercial production for only 100 years. Its discovery was made possible through development of the dynamo by Faraday and Edison.

Uranium, consistent with the history of copper and iron, was first used as a weapon. Its technology began in 1896 as a curious clouding of a photographic plate and evolved into a weapon, less than a half century later, of horrible destructive dimensions, first witnessed on the New Mexico desert in 1945 with the man-made thunder and lightning that was the atomic bomb.

Finally, fascination for gold has lured explorers, invaders, investors, settlers, and "con men" to all parts of the world. It has served, and continues to serve, as an international medium of exchange and a measure of a nation's wealth and financial stability.

1.1.2.1 Minerals and National Policy

With the final peace settlement after World War I, Germany lost 68% of its territory, all of its gold, silver, and mercury deposits, 80% of its coal mines and iron-producing capacity, and entered into a period of depression and starvation. The German economy managed to recover with imported ores and a high degree of technical skill and efficient labor.

The depression years of the 1930s resulted in economic nationalism and protective tariffs, and many markets were effectively closed. Since Germany and Japan were both dependent upon international trade, their standard of living plunged, and hunger, bitterness, and resentment flared. The Nazis came to power in Germany with promises of work, food, and prestige; rearmament began in 1933, and Japan followed suit shortly thereafter, leading the world into World War II (Lovering, 1943).

The world's mineral resources since the latter part of the 19th century have been primarily developed by Britain, the United States, the Soviet Union, Japan, West Germany, and France. These countries have furnished the necessary science, technology, and capital and have supplied the markets.

Local mineral wealth throughout history and social development has made first one nation rich and powerful, then another. The Phoenicians established worldwide trade and gained great wealth by developing and exchanging minerals for all manner of goods. Athens financed its ancient wars and "Golden Age" with silver from Laurium, Alexander funded his early conquests with gold from Macedon, the Romans expanded their Empire to acquire the silver of Carthage and the copper of Spain, and the Catholic crown of Spain became a world power by the exploitation of gold and silver from the New World. During the Middle Ages, Germany became the center of lead, zinc, and silver production and the leader in mining technology. Britain moved into the forefront during the Industrial Revolution of the 19th century and was successively the world's leading producer of tin, copper, lead, and then coal. Her resources were bolstered by those of a vast empire, and she became the wealthiest nation in the world. The greater resources of the United States subsequently supported its advance to become the richest nation; however, the future is already foreshadowed. Most of the Greek, German, and British high-grade mines are exhausted, and the United States is fast becoming dependent upon imports and preservation of peaceful world trade. Near East countries have experienced a rapid rise to great wealth based upon petroleum resources. This has been important in technological developments, but historically is of short duration. New discoveries of high-grade metal deposits are very likely in the Soviet Union and in China but less likely in the United States.

1.1.2.2 Future Contributions of the Minerals Industry

With few exceptions, no nation can achieve a high level of prosperity without a reliable source of minerals to supply its manufacturing industry. Through mining, emergent (Third-World) countries can finance growth progressively by the export of raw mineral resources, then by processing these raw materials prior to export, and finally by achieving progressive industrial development (Fig. 1.1.1).

Mineral reserves, upon which the future of the human race depends, occupy less than 0.1% of the continental areas. Unfortunately, we are not at present sufficiently skilled to determine exactly where they occur or how large they may be. They remain elusive targets.

Research in mining and metallurgical technology is essential. A new discovery may locate a mine, but a technological breakthrough can open up mines all around the world.

The economic evolution of society that began in Neolithic prehistory was based then, as it is now, on minerals, and has led man into modern times. The 104 elements of the periodic table, all but a few of which are recovered from widely spaced, often remote, mineral deposits using a variety of complex mining and metallurgical techniques, form the foundation of modern society.

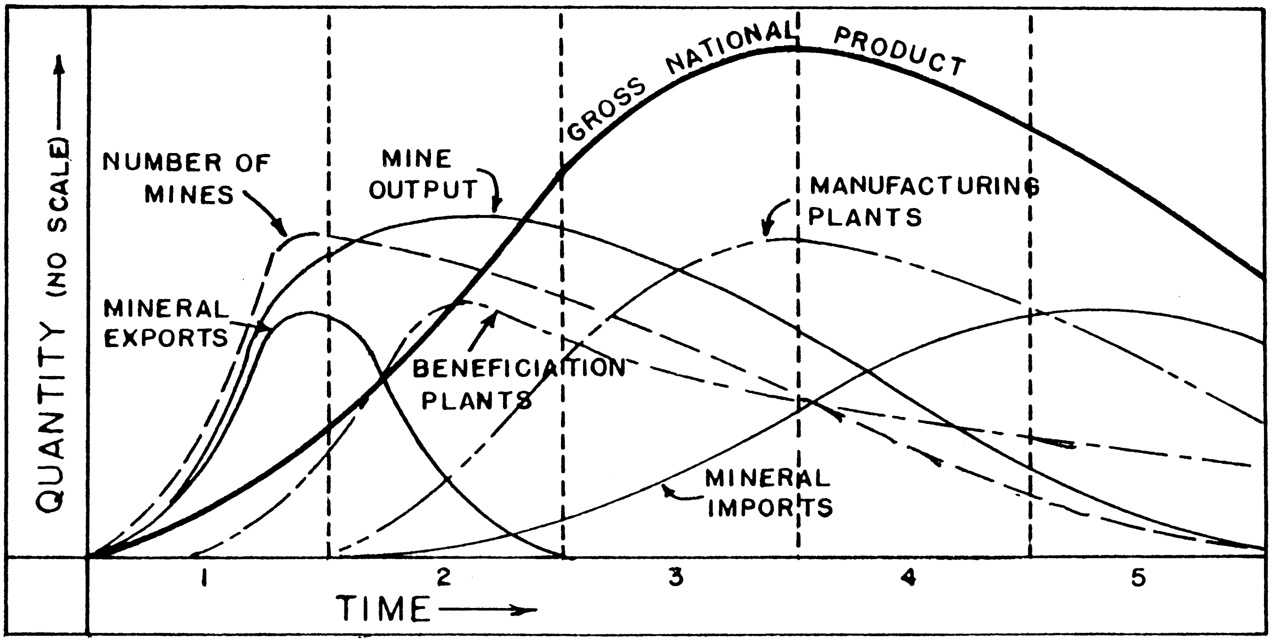

Fig. 1.1.1. Stages of mineral and metal production of an industrial country. (Modified from Lovering, 1943, p. 18.)

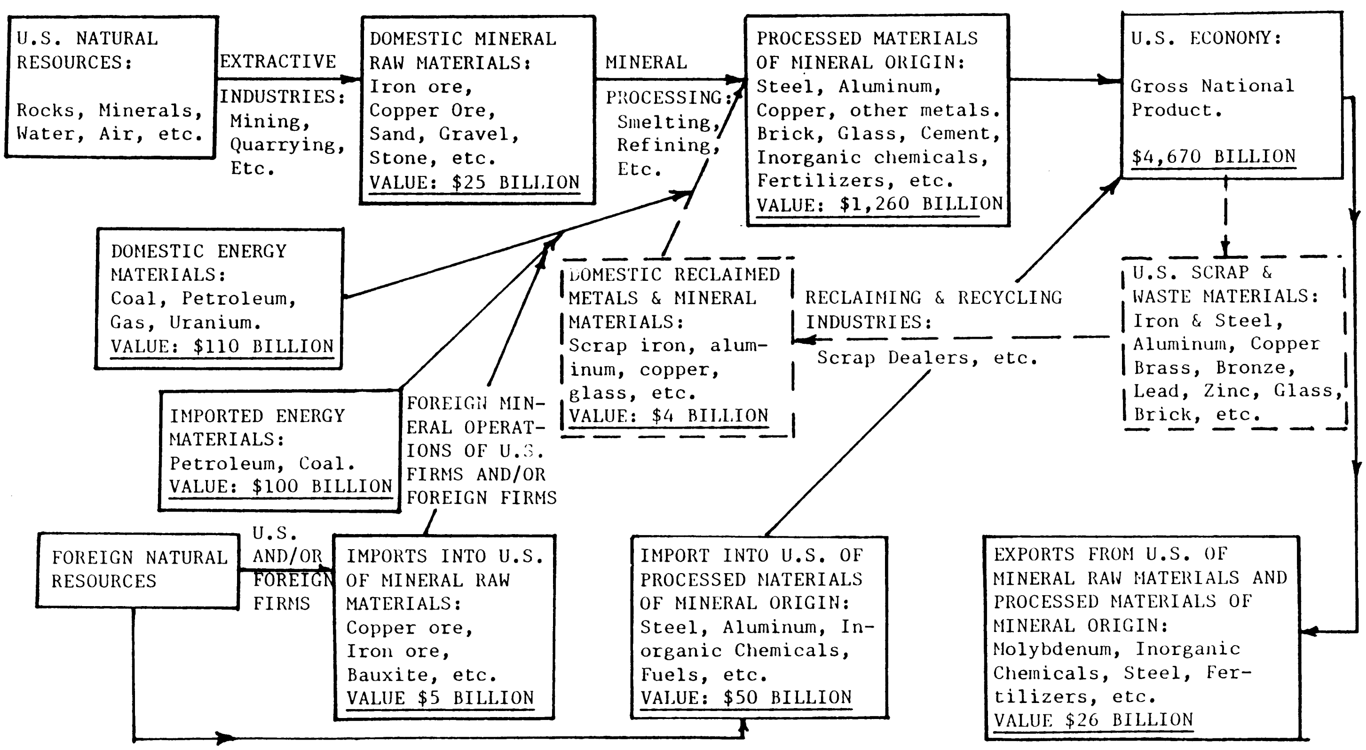

Fig. 1.1.2. The role of fuel and nonfuel minerals in the US economy (estimated values for 1990). (Modified from annual publications of the US Bureau of Mines.)

They provide its heat, light, shelter, transportation, communication, and food. The standards of living of the industrialized nations—which developing nations are striving to attain—are based upon minerals, and societies could not continue in their present state without them (Tables 1.1.2 and 1.1.3 and Fig. 1.1.2).

Mineral deposits within the border of any country represent potential national wealth: they can be transformed into actual

Table 1.1.2. Per Capita Consumption of Minerals in the United States (1970)

Commodity	Quantity	Major Uses
Steel	1400 lb	Transportation
Aluminium	44 lb	Kitchenware, buildings
Copper	20 lb	Electrical appliances
Tin	1 lb	Cans
Petroleum	3.4 tons	Transport, heating, industrial
Natural gas	2.5 tons	Heating, industrial
Coal	2.3 tons	Electricity generation, steel production
Salt	440 lb	Chemicals
Sulfur	70 lb	Fertilizer
Sand and gravel	4 tons	Roads, buildings

Modified from McDivitt and Manners.
Conversion units: 1 lb = 0.4536 kg
1 ton = 0.9072 t

Table 1.1.3. Source of Power (1987). United States Utility Companies Generated 2.5 Trillion Kw-Hr of electricity in 1987

Source of Power	kWh in Billions
Coal	1464
Nuclear power	455
Natural gas	273
Hydroelectric	250
Petroleum	118
Geothermal, others	12

Source: Energy Information Administration, *U.S.A. Today.*

material wealth (and contribute to the gross national product) only by being mined. Among the benefits to the state are an increase in employment levels (one mining job carries approximately a 5:1 multiplier effect), an enhanced level of self-sufficiency, and improved balance of trade. The latter results from fewer imports and greater exports of commodities mined, a spirited search for more minerals, a build-up of technical manpower levels by in-service training, attraction of overseas investment capital, and creation of national wealth (Gregory, 1980).

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Chapter 1.2
ELEMENTS OF MINING

HOWARD L. HARTMAN

This chapter is both an introduction to the *Handbook* and an overview of the sections that follow. Each main segment of Chapter 1.2 correlates to one of the six major parts of the volume, and each subdivision to one or more of the 25 sections (corresponding part and section numbers appear in brackets following the headings). In this manner, it is possible to preview a portion or all of the *Handbook* or to identify where subject matter of interest is located.

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1.2.1 PRELIMINARY TOPICS [Part I]

1.2.1.1 Introduction to Mining [Sec. 1]

History of Mining: The chronological development of mining technology bears an important relation to the history of civilization. In fact, as one of the earliest of man's enterprises, mining and its development correlate closely with human progress. It is no coincidence that the cultural ages of man are associated with minerals or their derivatives (e.g., Bronze Age, Nuclear Age). Today products of the mineral industry pervade the lives of all mankind.

Mining began with Paleolithic man, perhaps 300,000 years ago during the Stone Age, when flint implements were sought for agricultural and construction purposes. Primitive miners first extracted and fashioned the stone raw materials that they needed from deposits at the surface, but by the beginning of the New Stone Age (c. 40,000 BC), they began to mine underground as well.

Although records are nonexistent, human fossils and artifacts substantiate an early record of mining all over the world. Like other aspects of human civilization, mining originated in Africa; at first, it was done crudely and then with some technological sophistication. For example, early miners devised ways to chip and free fragments from the solid, to hoist ores by simple lifts, to illuminate their workings by torches and lamps, and even to ventilate underground openings.

Eventually, the first technological breakthrough that significantly advanced mining occurred in the breakage of rock in place. Fire setting, applying heat to expand rock and water to quench, contract, and crack it, was discovered by an unknown miner It was a revolutionary advance in geomechanics, one not surpassed in mining history until the deployment of explosives to break rock in the later Middle Ages.

Elements of Mining: A distinction is drawn between mining and mining engineering. *Mining* consists of the processes, occupation, and industry concerned with the extraction of minerals from the earth. *Mining engineering,* on the other hand, is the art and science applied to the processes of mining and to the operation of mines. The trained professional who relates the two is the *mining engineer;* he/she is responsible for helping to locate and prove mines, designing and developing mines, and exploiting and managing mines.*

The essence of mining in extracting minerals from the earth is to drive or construct an excavation, an opening to serve as a means of entry from the existing surface to the mineral deposit. Whether the openings lie on the surface or are placed underground fix the locale of the mine. The specific details of the procedure, layout, equipment, and system used distinguish the mining method uniquely determined by the physical, geologic, environmental, economic, and legal circumstances that prevail.

Using scientific principles, technological knowledge, and managerial skills, the mining engineer brings a mineral property through the four stages in the life of a mine: prospecting, exploration, development, and exploitation. Ever more advanced training is required for the professionals who direct mineral enterprises—which is the next topic discussed.

Mineral Engineering Education: Training engineers for the mineral industries is a specialized branch of the engineering profession. Originating at the academy in Joachimstal, Czechoslovakia, in 1716, mining and mineral engineering education is now offered worldwide. In the United States, 37 institutions of higher learning currently award accredited degrees in mining and related fields of engineering.

Disciplines encompassed by the generic term *mineral engineering* include mining, geological, environmental, mineral processing, and metallurgical engineering. Numbers of mining engineers graduating in the United States range from 200 to 800 per year (BS, MS, and PhD).

Mineral engineering is a broad educational field, in part because accreditational standards for engineering education are extraordinarily wide ranged. Additionally, mineral engineers must master a variety of engineering sciences, ranging from electrical circuits to thermodynamics and strength of materials. At the present time, nearly all US undergraduate curricula are four years in duration, but they are likely to be supplemented with a year or two of graduate study.

As mineral engineering grows increasingly more complex and technologically oriented, there is greater emphasis in the curriculum on computers, systems, and related topics. There is also a broadening effort underway to "humanize" and to add social responsibility to the engineer's education.

1.2.1.2 Mineral Economics [Sec. 2]

Because of their utility and value, minerals have been integral and essential to man's existence. Their uses are myriad: tools and utensils, weapons, ornaments, currency, structures, machines, and energy. Consequently, mining ranks with agriculture as one of man's two basic, earliest industries. And also like agriculture, mining is one of two human endeavors capable of generating new wealth (Beall, 1973).

Mineral wealth is, of course, neither abundantly nor uniformly distributed. Only a fraction of 1% of the earth's surface is

*The nomenclature used in the *Handbook* follows publications of the US Bureau of Mines and the Society for Mining, Metallurgy, and Exploration, Inc. (e.g., Thrush, 1968; Hustrulid, 1982).

underlain with mineral deposits that currently are of commercial value. Yet the annual mineral production (excluding petroleum and natural gas) of the United States currently exceeds 3.5 billion tons (3.2 t), valued at over $50 billion (Anon., 1984a). With value added in processing, the contribution of the mineral industry to the US gross national product approaches $300 billion, or approximately 8%. In developing countries, minerals' share of the GNP may reach 25%.

Consumption of minerals increased to such an extent in modern times that the United States alone has consumed more mineral products since World War II than were mined in the entire previous history of the world (Anon., 1983a). Since the Industrial Revolution, the average rate of increase in US mineral consumption has averaged 5%, and since 1950 the use of minerals has increased twice as fast as the total consumption of all other raw materials.

Currently, the United States leads the world in the mine production of bituminous coal, lead, molybdenum, natural gas, phosphate, salt, and sulfur. It also produces a significant amount (over 60%) of the copper, gold, gypsum, iron, and nitrogen that it consumes. But on balance, the United States has become a net importer of minerals: imports now exceed exports (on a dollar basis) by a 2:1 margin. We import 50 to 100% of 15 key minerals, including many critical to national defense or food production (e.g., aluminum ore, chromium, industrial diamonds, manganese, nickel, potash, tin). Growing US dependence on foreign sources for its mineral needs has both created a troubling defense concern and contributed to a soaring international trade deficit.

The shifting complexion of the US mineral industry has also raised environmental and conservation dilemmas for the nation. These issues are widely debated. Controversies often arise between profit-oriented mining corporations and conservation or wilderness groups, some of whom advocate extreme preservationism, not conservation. Increasingly, though, when new mining projects undergo environmental review, voices of reason prevail on both sides, allowing compromises to be reached without costly litigation or abandonment of objectives.

The uniqueness of mineral deposits accounts in large measure for the complexity of mineral economics and mining enterprise (Vogely, 1985; Strauss, 1986). Minerals are immobile and, unlike agricultural or forest products, cannot reproduce or be replaced. A mineral deposit may be viewed as a depleting or wasting asset whose production is restricted to the locality in which it occurs. These factors impose limitations on a mining company in the area of business practices and financing as well as in production operations. Because its mineral assets are continually being depleted, a mining company must discover additional reserves or acquire them by purchase to stay in business.

Other peculiar features are related to operations. Production costs tend to increase with the depth of the mine and the declining grade of ore, creating technological and financial problems with which every mine eventually is confronted. Financial hazards are great since estimates of the ore supply, market price, mining cost, or other factors may prove to be lacking in accuracy or sufficient detail.

The law of supply and demand likewise complicates the economics of the mining industry, because the price of minerals varies more sharply than the price of commodities manufactured by the consumers of raw materials. The output of minerals from byproduct producers and foreign sources can create an oversupply that depresses the market. Some minerals, such as precious metals, iron, and certain base metals, are recycled and in a sense never expended because of their utility as scrap. Reservoirs of scrap—lead is the extreme case at 50% of primary consumption—may depress the market, and stockpiles of strategic minerals maintained in the national interest may act as buffers. Certain minerals are exceptions to economic laws because their prices are fixed by government decree or private cartels. Official prices of gold, silver, and uranium historically have been regulated by statutes (although their market prices currently fluctuate in Free World markets), and cartels strongly influence the prices of industrial diamonds, mercury, oil, and tin. In addition, substitutes for a particular mineral may be developed, especially if the price of the mineral remains at a high level (e.g., aluminum for copper, plastic for metal). Market trading and speculation affect the prices of minerals as they do most other commodities.

Stockpiling of strategic minerals by the federal government became a common practice after 1939, and the practice was sharply increased after World War II. The Federal Emergency Management Agency is responsible for procuring certain minerals as part of the program of national preparedness and enters into purchase agreements with individual producers, at negotiated prices, to meet its stockpiling objectives. In recent years, US government stockpile purchases have declined under provisions of the Strategic and Critical Materials Stockpiling Act of 1979 (Dorr, 1987). The mineral industry has often been critical of the government's stockpiling policy, since sudden large purchases or sales from the stockpile can have drastic artificial effects on the price and demand for a commodity. Economists generally tend to favor private-sector management of inventories.

A final related aspect of mineral economics concerns financing and marketing of mines and mineral properties (also see Sec. 6). Mining enterprises are financed in much the same manner as are other businesses (Gentry and O'Neil, 1984; Wanless, 1984; Tinsley et al., 1985). Because of great financial risks, however, the expected return on an investment is higher and the payback period shorter in a mining enterprise. Mineral properties as well as mines are marketable. The selling price is determined generally by a valuation based on the report of qualified engineers; the value of future earnings may then be discounted to the date of purchase in computing the present value of the property.

1.2.1.3 Government Role and Influence in Mining [Sec. 3]

Governments and their agencies exert many influences on the mining industry. In the United States, these take the form of various statutes and regulations pertaining to land use, mineral rights, taxation (Sec. 2), quotas, tariffs, financial incentives, antitrust constraints, stockpiling (Sec. 2), safety and environment, and expressed or implied mineral policies.

Laws governing the acquisition of mineral rights in the United States have developed from the common law of England, the laws and statutes of the federal government, and the laws of the various states. Although the federal Mining Law of 1872 has been somewhat modified by later legislation, it remains the recognized and pertinent statute. It provides for the location of claims for mineral deposits located in the public domain, the performance of annual assessment work to retain rights to a claim, and the acquisition of title to claims. Certain nonmetallic minerals such as coal, gas, oil, phosphates, sodium compounds, and sulfur are exempted from this act and are governed by a leasing law, the Mineral Leasing Act of 1920. Uranium is subject to a leasing arrangement also, under the Atomic Energy Act of 1954. Many states have also enacted legislation to provide mineral rights within their boundaries.

A mining company is subject to the same forms of taxation upon income as any other business and, in certain states, to production, royalty, and severance taxes as well (Anon., 1983b).

In the federal corporate income tax law, however, the Internal Revenue Service wrote in two provisions that are advantageous to mining companies. The first is a depletion allowance, similar in effect to a depreciation charge, that permits a deduction from taxable income in recognition of the gradual exhaustion of ore. The second allows the deduction of exploration and development costs over a period of time.

Many imported minerals and processed metals are subject to tariff duty. The mineral industry, like manufacturing, has consistently sought import quotas or tariff protection from foreign producers, and the US government has generally recognized the importance of encouraging a strong domestic mining industry, notwithstanding other commerce policies generally supportive of international free trade.

The need for safety and environmental regulation arises because of some of the less favorable impacts of mining. Sometimes these are direct and obvious, but more often they are considered side effects. Typical impacts include (1) accidents and health hazards, (2) land-use and environmental impacts, and (3) economic-political-social-psychological effects.

Accident and health hazards in mining are of vital concern to the industry as well as to regulatory bodies of the government and the public at large. Mining practices are regulated by individual states and by the Mine Safety and Health Administration under Title 30 of the *Code of Federal Regulations,* based mainly on legislation enacted in the Coal Mine Health and Safety Act of 1969 and the Mine Safety and Health Act of 1977 (Anon., 1984b). While mining's safety record is among the poorest of all US industry, due in part to an inherently more dangerous environment, it has improved significantly since the 1960s. Greater industry diligence, government intervention, and union criticism are variously credited for the improvement. The consequences of poor health and safety practices in industry are costly, both in terms of loss or harm to life and property damage, and mining is beginning to exercise the initiative required to improve its record (Hansen, 1973).

Physical, chemical, and biological changes in the environment often result from mining. They are usually the most evident and serious of mining's side effects. Examples are disturbance of the surface, subsidence, water and air pollution, consumption of irreplaceable resources, threat to endangered species, and preemptive use of land (Parr and Ely, 1973; Brooks and Williams, 1973; Parr, 1982). Federal legislation (e.g., the Clean Water and Clean Air Acts of 1977, the Endangered Species Act of 1973) now requires the containment or correction of any of these effects that violate environmental standards. Conflicts over land use increasingly are being resolved in ways that provide for orderly, multiple use of the land; applicable legislation is contained in the Multiple Surface Use Act of 1955. Restoration of the surface following coal mining is now required under the federal Surface Mining Control and Reclamation Act of 1977.

Finally, there is a variety of indirect effects, often more subtle and less susceptible of measurement, that may be associated with mining. They are grouped into a third, omnibus category of economic-political-social-psychological effects (Weinreach and Fagan, 1975). Often they result from either initiation or termination of mining operations, when drastic changes occur in manpower-employment levels in nearby communities. The primary effects of opening a mine are largely beneficial, of course, but there may be deleterious secondary ones that create economic and political strains, require social readjustments, and cause psychological stress among the population. These are multiplied when a mine closes.

The anticipation of unwanted, indirect consequences is the most important and difficult challenge that mining confronts in managing its various side effects. Generally, a mining company writes a comprehensive environmental impact statement (EIS) prior to undertaking a mine development project. The National Environmental Policy Act stipulates that an EIS must be filed when "federal action" is involved, that is, approval of a lease, permit, right of way, or mining plan (Parr, 1982). In this way, cost/benefit analyses can anticipate mining's consequences in advance. Older mining operations face no such restraints, however, and consequently environmental abuses do exist in mined-out areas.

In spite of these direct and important involvements of government, many in the mineral industry rightly allege that the United States, as a nation, lacks a coherent, definitive mineral policy (Dorr, 1987). Long advocated by trade associations such as the American Mining Congress (Anon., 1988), a US national mineral policy in reality exists only in de facto form.

1.2.2 STAGES OF MINING [Part II]

The overall sequence of activities involved in modern mining can be expressed as *stages in the life of a mine.* There are four: prospecting, exploration, development, and exploitation. *Precursors* to actual mining, prospecting and exploration are closely linked stages, transitional, and often considered a combined activity (as they are treated here). Likewise, development and exploitation, which constitute *mining proper,* are inherently related. The great preponderance of the *Handbook* is devoted to these latter two stages.

Table 1.2.1 summarizes the four stages in the life of a mine, plus an evaluation step. Included are procedures, time durations, and cost ranges for each.

1.2.2.1 Stages 1 and 2: Mineral Prospecting and Exploration [Secs. 4, 5]

Prospecting, the first stage, is the search for metallic ores or other valuable minerals (coal or nonmetallics). Because mineral deposits are found at or beneath the surface of the earth, both direct and indirect techniques are employed, although geology is the basic science of all prospecting. In the United States, over the past 50 years, geology has accounted for three-quarters of all mineral discoveries (Derry and Booth, 1978).

The *direct method* of discovery, normally limited to surface deposits, consists of *visual examination* of either the exposure (outcrop) of the deposit or of the loose fragments (float) that have weathered away from the outcrop. *Geologic studies* of an area augment this simple, direct technique. By means of aerial photography and with topographic and structural maps of a region, the geologist gathers further evidence by direct methods to locate areas of ore deposition. Precise mapping of rock formations and their peculiar structures in the field, supplemented by analytic and microscopic studies of samples in the laboratory and aided by geologic inference, can enable the geologist to locate hidden as well as surface ore bodies.

A valuable scientific tool being employed in the *indirect search* for or exploration of hidden ore bodies is *geophysics,* a method that detects anomalies caused by the presence of mineral deposits through the analysis of gravitational, seismic, magnetic, electrical, electromagnetic, and radiometric measurements (Anon., 1983a). It is suitable for airborne, surface, and subsurface use. Three methods lend themselves to simultaneous application from aircraft: magnetic, electromagnetic, and radiometric. Geophysics applied from the air or space through remote sensing enables vast areas to be prospected and explored. On the ground and in logging boreholes, it provides more definitive information. The magnetic, electrical, electromagnetic, and radiometric meth-

Table 1.2.1. Stages in the Life of a Mine

Stage/ (Project Name)	Procedure	Time	Cost/Unit Cost
Precursors to Mining			
1,2. Prospecting and Exploration (Name: Prospect)	Search for ore a. Prospecting methods Direct: physical, geologic Indirect: geophysical, geochemical b. Locate favorable loci (maps, literature, old mines) c. Air: aerial photography, airborne geophysics Space: satellite d. Surface: ground geophysics, geology e. Spot anomaly, analyze, evaluate Define extent and value of ore (examination/evaluation) a. Sample (drilling or excavation), assay, test, log b. Estimate tonnage and grade Feasibility study: make decision to abandon or develop property a. Evaluate deposit (by formula or discount method), present value = annual cash flow discounted to the present	2-8 yr	\$0.5-\$15 million 10¢-\$1.50/ton (9¢-\$1.40/t)
Mining Proper			
3. Development (Name: Project)	Open up ore deposit for production a. Acquire mining rights (purchase or lease), if not done in Stage 2 b. Prepare budget, obtain financing c. File environmental impact statement, technology assessment, permit d. Construct access roads, transport system e. Locate surface plant, construct facilities f. Excavate deposit (strip or sink shaft)	2-5 yr	\$10-\$250 million or 25¢-\$5/ton (23¢-\$4.50/t)
4. Exploitation (Name: Mine)	Produce ore on large scale a. Factors in choice of method: geologic, geographical, economic, environmental, societal, safety b. Types of mining methods Surface: open pit, open cast, etc. Underground: room and pillar, block caving, etc. c. Monitor costs and economic payback (3-10 yr)	5-30 yr	\$5-\$50 million/yr or \$2-\$100/ton (\$1.80-\$90/t)

Source: Hartman, 1987.

ods are the most popular ground methods. *Geochemistry,* the microquantitative analysis of soil, rock, and water samples, and *geobotany,* the study of vegetational and plant growth patterns, also are employed as prospecting tools.

The second stage in the life of a mine, *exploration* determines as accurately as possible the size and value of mineral deposit, utilizing techniques similar to but more refined than those used in prospecting. The line of demarcation between the two is not sharp; in fact, a distinction between the two stages is usually not made. The locale in exploration shifts more from the air to the surface and subsurface, both with geology and geophysics. In addition, more positive information of the extent and richness of the deposit is obtained by representative and systematic *sampling,* subjecting mineral specimens to chemical, X-ray, spectrographic, or radiometric analyses. Samples are obtained systematically by chipping or trenching outcrops and by drilling and excavating below the surface; additionally, borehole logs may be taken by geophysics. These are several common drilling methods; diamond drills provide core samples, and rotary or percussion drills produce chips or cuttings. Coring is more useful but most expensive; rotary accounts for 70% of exploration drilling (Martens, 1982).

An evaluation of chip or core samples or logs enables the geologist or mining engineer to calculate the tonnage (extent) and grade (richness) of the deposit. He or she establishes the economic value of the ore, estimates mining costs, and assesses all other foreseeable factors in an effort to reach an accurate conclusion concerning the merits of a given deposit and the profits likely to be realized. This entire procedure consists of *reserve estimation and examination and valuation* of the mineral deposit. A complete ore estimate provides a breakdown of several categories of reserves (proven, probable, or possible), based on geologic and economic evidence. Many of the advanced phases of exploration constitute *project and mining geology.*

Discovery and location of an ore deposit have been likened to the search for the proverbial needle in the haystack. A mineral deposit is a geologic anomaly, while an ore deposit is a freak of nature. The odds against a mineral deposit evolving into a mine—of progressing successively from stages 1 to 4—are variously estimated as 1000 to 10,000:1 (Anon., 1980; Anon., 1983a).

The staggering costs involved in prospecting and exploration (Table 1.2.1) reflect these odds. Further, the complexity of search procedures and the need for a multidisciplinary team in mineral exploration have all but ruled out the solitary prospector as a viable alternative.

1.2.2.2 Mine Evaluation and Investment Analysis [Sec. 6]

At the conclusion of the prospecting and exploration stages in the life of a mine, a thorough *feasibility study* is conducted to determine the potential of developing the mineral deposit into a producing mine. The outcome of this study is a decision to abandon or proceed with the project.

The feasibility report produced is primarily an economic one, but legal, political, technological, geologic, environmental, and sociopolitical considerations are involved as well. In a typical study, all the information assembled by the exploration team of geologists and others is turned over to an evaluation group of engineers, mineral economists, and legal experts.

The formal feasibility study includes an economic analysis of the rate of return that can be expected from the mine at a certain production rate (Anon., 1983a). Some of the factors considered during such an economic analysis are

- Production
  - Reserve tonnage in the deposit
  - Mill recovery
  - Production rate, tons (tonnes) per day
- Costs
  - Exploration and development costs
  - Capital cost of the mine
  - Capital cost of the mill
  - Mining cost per ton (tonne)
  - Processing cost per ton (tonne)
  - Miscellaneous costs of operation
  - Royalties
  - Taxes (federal, state, and local)
- Revenues
  - Sale price of the metal or mineral
- Financing
  - Working capital necessary
  - Depreciation method used
  - Depletion allowance

In many cases, this information is processed by a computer to calculate the dollar value of annual gross sales, operating costs, operating income, depreciation, depletion, income tax, net income after taxes, cash flow, and after-tax rate of return on investment.

Each mining organization has a minimum acceptable rate of return. The cost of borrowing capital for the mine or of generating the needed capital internally within the company must be considered. If a company has a number of attractive investment opportunities, the rate of return from the proposed mine venture may be compared with the rate expected on a different mining venture elsewhere, or with some other business opportunity unrelated to mining.

1.2.2.3 Stage 3: Mine Development [Sec. 7]

In the third stage of mining, *development*, work is performed to open a mineral deposit for exploitation. With it begins mining proper. Access to the deposit must be gained either (1) by *stripping* overburden, the earth and/or rock covering the mineral deposit, to expose near-surface ore for surface mining; or (2) by *excavating* openings from the surface to more deeply buried deposits to prepare for underground mining.

In either case, certain preliminary development work, such as preparing an environmental impact statement, acquiring mineral rights and financing, and providing access roads and other transportation, power sources, ore processing facilities, dams, and waste disposal areas, nearly always precede the actual mining. These and other sequential steps in mine development are often programmed by operations research techniques such as CPM or PERT to conserve time and expense.

Stripping of the waste material overlying the ore body then commences if the mine is to be a surface one. The cycle of operations to break up and remove the overburden may be the same as that employed in exploitation of the ore, or it may differ, depending on the characteristics of the waste and the ore. Fig. 1.2.1 illustrates development for surface mining, using the open cast method. One or more bench faces are established, which permits mining on multiple levels.

Development for underground mining is generally more complex and expensive. It requires careful planning and layout of access openings for convenience, safety, and permanence. The principal opening to the surface is usually a shaft, which may be circular or rectangular in cross section, vertical or inclined (called a slope), and of sufficient size to allow passage for men and machines. In areas of high relief, horizontal openings called adits or tunnels may be used to reach the deposit. Mining of massive or steeply pitched underground deposits of minerals (usually metallic) is carried on from horizons, or levels, located at regular intervals in a vertical plane. The openings on each level consist of main arteries called drifts and numerous secondary, connecting crosscuts. Vertical openings (raises or winzes) or inclined ones (ramps) provide access between the levels. All these development openings connect with large exploitation chambers called stopes, from which most of the mine's mineral production is obtained (Fig. 1.2.2).

Coal and most nonmetallics in this country are often found in flat-lying, bedded deposits and are mined from systems of connected horizontal openings called entries or crosscuts and rooms or longwalls.

1.2.2.4 Stage 4: Mine Exploitation [Sec. 8]

Exploitation, the fourth and final stage of mining, is associated with the actual recovery in quantity of mineral from the earth. While some exploration and development work necessarily continues throughout the life of a mine, the emphasis in the exploitation stage is on production. Only enough development is done prior to exploitation to ensure that production, once started, can continue uninterrupted throughout the life of the mine. The transition through the four stages from prospect to producing mine for an actual case is shown in Fig. 1.2.3.

The mining method selected for exploitation is determined mainly by the characteristics of the mineral deposit and the limits imposed by safety, technology, and economics. Geologic conditions, such as deposit dip and shape and strength of the ore and wall rock, play a key role in selecting the method. *Traditional exploitation methods* fall into two broad categories based on locale: surface or underground. Surface mining includes mechanical extraction methods such as open pit and open cast and aqueous extraction methods such as placer and solution mining. Underground mining is usually classified into three groups of methods, including unsupported (e.g., room and pillar mining, sublevel stoping), supported (e.g., cut and fill stoping, square set stoping), and caving (e.g., longwall, block caving).

In addition to these traditional exploitation methods, novel or *innovative mining* methods are continually evolving. They are applicable to unusual deposits or employ unusual techniques or equipment. Examples are automation, rapid excavation in hard rock, underground gasification, and marine mining (see Sec. 22).

A scheme to classify the mining methods referred to in this *Handbook* is shown in Table 1.2.2. Distinctions are made on the basis of degree of acceptance (traditional or novel), locale (surface or underground), and class and subclass (extractive features). The table also provides information concerning application (commodities mined and relative cost).

Other topics covered in this section include mine surveying, systems engineering, computer methods, labor relations, management, and mine closure.

1.2.3 UNIT OPERATIONS OF MINING [Part III]

During the development and exploitation stages of all mining when natural materials-rock or soil, ore or waste-are extracted from the earth, remarkably standardized unit operations are employed. The *unit operations of mining* are the basic steps employed to produce mineral from the deposit, together with

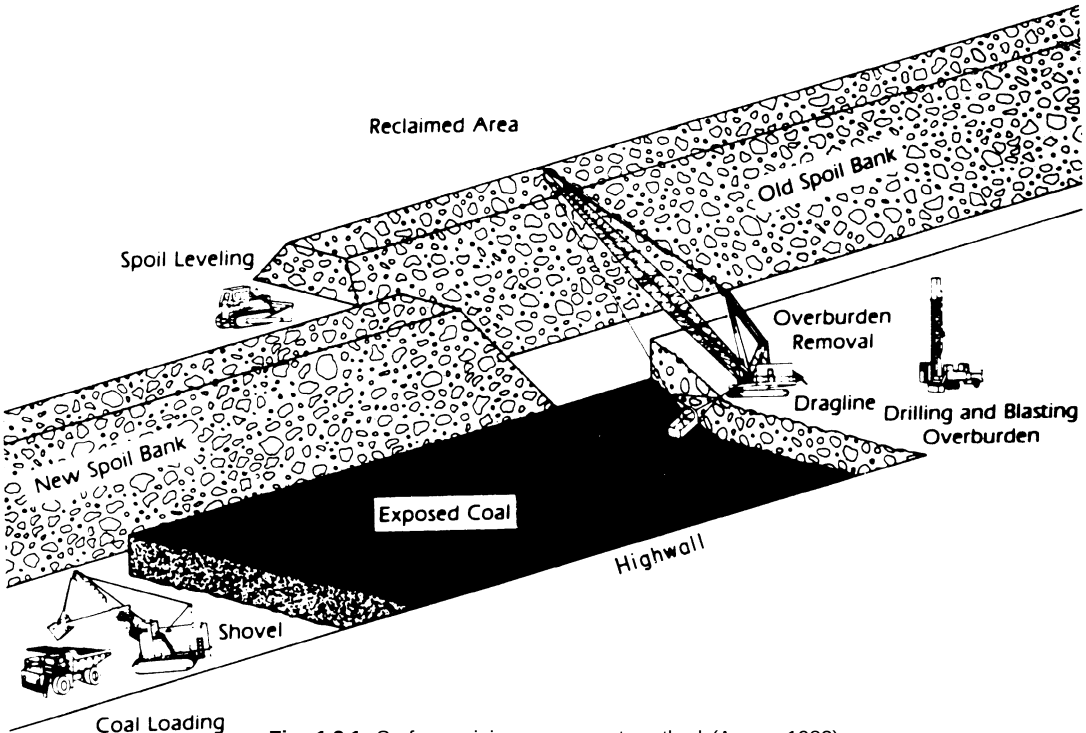

Fig. 1.2.1. Surface mining, open cast method (Anon., 1982).

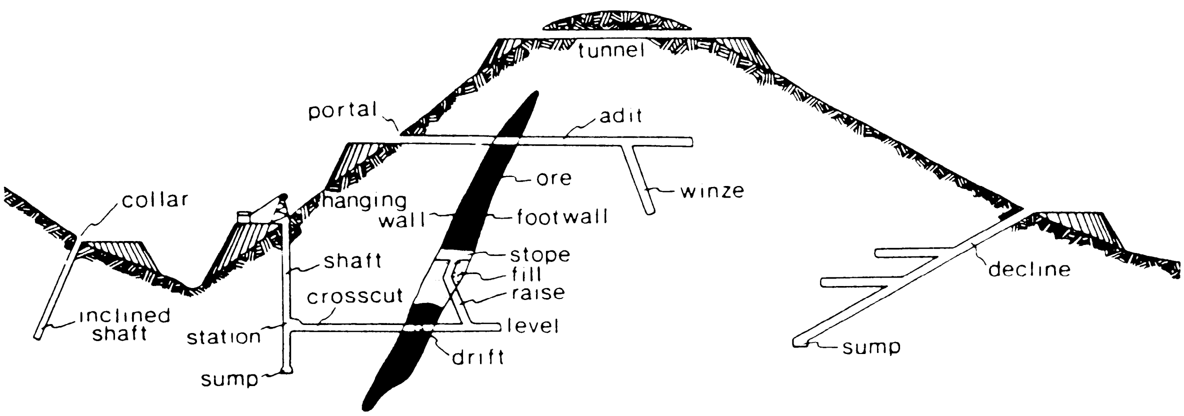

Fig. 1.2.2. Underground mining, stoping method (Anon., 1983a).

the auxiliary steps involved. Those steps contributing directly to mineral extraction are *production operations,* comprising the production cycle of operations. Those ancillary steps that support the production cycle are called *auxiliary operations.*

1.2.3.1 Production Operations [Sec. 9]

The production cycle employs unit operations that normally are grouped in two functions: rock breakage and materials handling. *Rock breakage* includes a variety of mechanisms but is usually accomplished by drilling and blasting, sometimes preceded by cutting in underground coal mining or replaced by channeling in quarrying. *Materials handling* generally encompasses loading or excavation and haulage (horizontal transport), with hoisting (vertical or inclined) optional. Thus the production cycle in mining consists of these unit operations:

General cycle = cut + drill + blast + load + haul + hoist

which may be abbreviated in many mines (especially noncoal or surface) to

Conventional cycle = drill + blast + load + haul

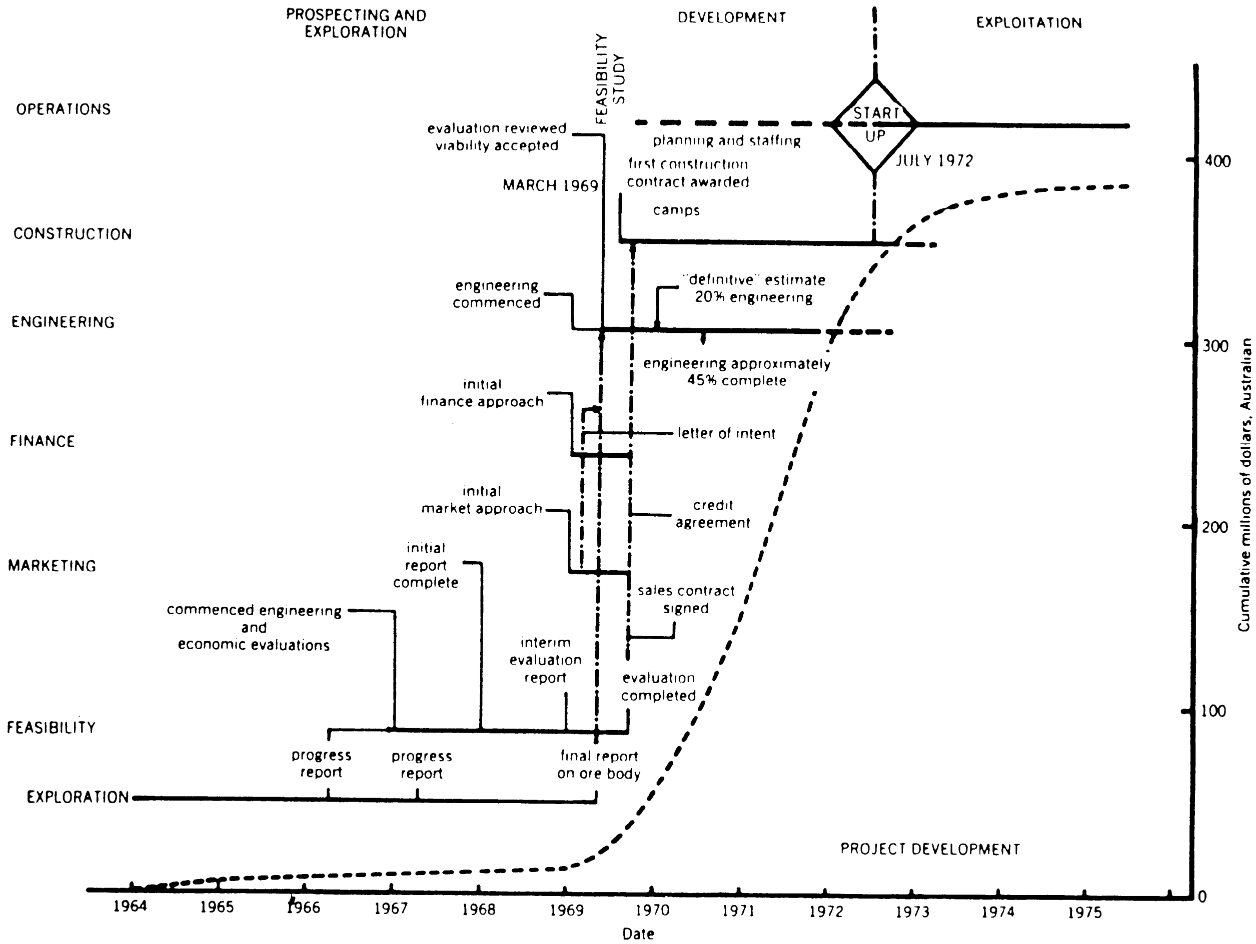

Fig. 1.2.3. Stages in the life of a mine. Relationship between planning steps during exploration and development and expenditures preparatory to mining a large copper open pit—Bougainville mine, Papua New Guinea. (Hope, 1971. By permission from Institution of Engineers, Barton, Australia.)

While production operations tend to be separate and cyclic in nature, the modern and future trend in mining and tunneling is to eliminate or combine functions and to increase continuity. For example, soil may be excavated by a machine (bucket wheel excavator) which requires no drilling or blasting. If loosening is necessary, it may be accomplished without explosives by ripping prior to loading. In coal or soft ores, continuous miners break and excavate mechanically and thus eliminate drilling and blasting; boring machines perform the same tasks in soft to medium-hard rock. The production cycle in these cases further simplifies to

Continuous cycle = mine + haul

The cycle of operations in surface and underground mining is distinguished mainly by the scale of the equipment. Specialized machines have evolved to meet the unique needs and conditions of the two regimes. In modern surface mining, blastholes several inches (tens of millimeters) in diameter are bored by mobile rotary or percussion drills for the placement of blasting agents or high explosives, essentially all now ammonium-nitrate based, when consolidated rock must be excavated. The charge is then inserted and detonated to reduce the ore or waste to fragments. The broken material is loaded by power excavators of the shovel, dragline, or bucket wheel type into haulage units—railroad cars, belt conveyers, or, most usually trucks—or cast on a waste (spoil) bank. Soil and coal are mined in a similar way, although blasting is often unnecessary. In quarrying dimension stone, the blocks are freed without blasting by channeling machines or saws.

In underground mining, the cycle differs little, although scaled-down equipment is usually employed. Smaller drillholes are bored for blasting, and compact loading machines and down-sized trains, trucks, shuttle cars, or conveyors are used to haul the ore or coal in or from the mine. To facilitate breakage in coal, salt, or potash mines where blasting is minimized to prevent methane ignition and excessive degradation, the process of cutting a kerf into the mineral face with a special machine precedes blasting. Hoisting by skip, cage, or conveyor may be the final operation.

In designing a production cycle for balanced operation, once individual machine capacities are established, the number of units (e.g., drills or trucks) can be determined from the required mine output. Ideally, the units of the system should be matched

Table 1.2.2. Classification of Mining Methods

Acceptance/ Locale	Class	Subclass	Method	Commodities	Relative cost, %
Traditional					
Surface	Mechanical	—	Open pit mining	Metal, nonmetal	10
			Quarrying	Nonmetal	100
			Open cast (strip) mining	Coal, nonmetal	10
			Auger mining	Coal	5
	Aqueous	Placer	Hydraulicking	Metal, nonmetal	5
			Dredging	Metal, nonmetal	<5
		Solution	In situ techniques	Metal, nonmetal	5
			Surface techniques	Metal	5
Underground	Unsupported	—	Room and pillar mining	Coal, nonmetal	30
			Stope and pillar mining	Metal, nonmetal	30
			Shrinkage stoping	Metal, nonmetal	50
			Sublevel stoping	Metal, nonmetal	40
			Vertical crater retreat mining	Metal, nonmetal	35
	Supported	—	Cut and fill stoping	Metal	60
			Stull stoping	Metal	70
			Square set stoping	Metal	100
	Caving	—	Longwall mining	Coal, nonmetal	20
			Sublevel caving	Metal	50
			Block caving	Metal	20
Novel	—	—	Rapid excavation	Noncoal (hard rock)	
			Automation, robotics	All	
			Hydraulic mining	Coal, soft rock	
			Methane drainage	Coalbed methane	
			Underground gasification	Coal	
			Underground retorting	Hydrocarbons	
			Marine mining	Metal, nonmetal	
			Nuclear mining	Noncoal	
			Extraterrestrial mining	Metal, nonmetal	

Source: Hartman, 1987.

in capacity so there is a uniform, uninterrupted flow of material from the working face to the surface disposal point (plant, loading pocket, or dump).

1.2.3.2 Auxiliary Operations [Secs. 10, 11, 12]

In addition to the productive phases of the actual mining cycle, certain auxiliary unit operations must be performed. Underground, these auxiliary operations consist of providing and maintaining adequate health and safety, roof support, ventilation and air conditioning, power supply, pumping, maintenance, lighting, noise abatement, communications, and handling of supplies. In surface mining, most functions remain the same, but slope stability, waste disposal, and land reclamation must be practiced instead of roof support and air contaminant control in place of ventilation.

Certainly the most important auxiliary operations in all mining-generically speaking-are (1) *health and safety,* (2) *ground control,* and (3) *atmospheric environmental control.* Specialized fields of engineering analysis and design (e.g., geomechanics for ground control) have matured around them.

In planning production cycles, most auxiliary operations are scheduled so as to support but not interfere with production operations. A few may be conducted as an integral part of the cycle if they are essential to health and safety or overall efficiency.

1.2.4 SURFACE MINING [Part IV]

Surface mining, in which excavation is carried on aboveground, is the predominant exploitation method, domestically and worldwide, producing in the United States nearly 85% of all minerals, excluding petroleum and natural gas (Pfleider, 1968). Almost all (96%) of the nonmetallic minerals, 87% of the metallic ores, and 60% of the coal in the United States are now mined from the surface-and the large preponderance by two methods (open pit or open cast mining). By their very nature, the mechanical extraction surface methods (except quarrying) are large-scale, mass-production techniques. The sheer magnitude of the volume or tonnage of material broken and handled in surface mining is staggering (over 12 billion tpy, or 11 Gt/a). In a recent year, surface methods account for 95% of all ore and waste extracted in US mining.

In importance, surface mining clearly ranks ahead of underground mining, if one compares tonnage or value of current annual production. In spite of its many attractions, however, there are some serious limitations to surface mining, not the least of which are depth, selectivity and flexibility, and environmental impact.

1.2.4.1 Surface Mine Development [Sec. 13]

Certain factors in mine development (Sec. 7) receive special attention in preparation for surface mining. Of the locational factors, climate is of more critical concern in surface operation than underground. Today, harsh climates at high altitudes or in northern latitudes rarely mitigate against surface mining, but they can be detrimental. Among natural and geologic factors, terrain, depth and spatial characteristics of the deposit, and presence of water are most important in surface mining. Among environmental factors, certainly antipollution and reclamation requirements rank highest as concerns in surface mining.

In the sequential steps of mine development, there are three that are unique to surface mining:

1. Initiation of a land reclamation plan, during and after mining, as part of the requirement to implement the EIS at the mine.
2. Provision of topsoil stockpiles and waste-disposal dumps.
3. Performing advanced stripping of overburden to gain access to the deposit.

They, too, must be incorporated into the development/exploitation schedule of operations. The major engineering design task in the development of a surface mine is planning the pit; three groups of factors are involved (Soderberg and Rausch, 1968; Atkinson, 1983):

1. *Natural and geologic factors:* geologic conditions, ore types, hydrologic conditions, topography, and metallurgical characteristics.
2. *Economic factors:* ore grade, ore tonnage, stripping ratio, cutoff grade, operating cost, investment cost, desired profit, production rate, and market conditions.
3. *Technological factors:* equipment, pit slope, bench geometry, road grade, easements and property lines, and pit limits.

Pit planning and design—partly because of the immensity of the scale of operations—is crucial to the success of a surface mine. It is predicated on several objectives and broken down into short-range and long-range planning. In both phases, the calculation of stripping ratios and pit limits is required.

Location of the ultimate pit limits is based both on technological and economic constraints. Equipment and method limitations govern absolute depth capability (see Fig. 1.2.1). The maximum allowable *stripping ratio,* a break-even ratio based solely on economics, is typically expressed in units of cubic yards (cubic meters) or tons (tonnes) of overburden per ton (tonne) of ore; it determines the areal pit boundaries. Magnitudes of the actual overall stripping ratio range from as high as 45-to-1 yd^3/ton (38-to-1 m^3/t) in coal mining to as low as 1:1 in metal and approach 0:l in nonmetal. Extensive calculations and computer plotting may be necessary to define both short-range and long-range objectives and limits in surface mining.

1.2.4.2 Surface Mining Methods [Secs. 14, 15, 16]

Two classes of methods are employed in surface mining: mechanical extraction and aqueous extraction. The former is by far the more prevalent (over 90% of US surface production), the latter being limited to applications where water is instrumental to exploitation.

Mechanical Extraction Methods: The *mechanical extraction class* employs mechanical processes in a nominally dry environment to free minerals from the earth. Four methods comprise this class: open pit mining, quarrying (of dimension stone), open cast mining, and auger mining.

In *open pit mining,* a thick deposit is generally mined in benches or steps, although a relatively thin deposit may be mined from a single face, as in quarrying, augering, or open cast mining. Any overburden must be removed by a stripping process before or during mining. In *open cast (or strip) mining,* however, overburden is removed, usually by casting into mined-out areas, and mineral (commonly coal) recovered in successive operations. Open pit or open cast mining is used to exploit a deposit near the earth's surface that has a relatively low stripping ratio, is preferably large in extent, and is reasonably uniform in value. These methods necessitate a large capital investment but generally result in high productivity, low operating costs, and good safety conditions. *Quarrying,* a highly specialized method and the only one intended to produce both a sized and shaped product, is slow, small scale, and (along with square set stoping) the most expensive of all mining methods. *Augering* is utilized in recovering coal from the highwall at the pit limit; it, too, is specialized but a low-cost method.

Broadly applicable, open pit and open cast methods employ a conventional mining cycle of operations to extract mineral: rock breakage is usually accomplished by drilling and blasting, followed by the materials handling operations of excavation and haulage. Quarrying and augering are specialized and less frequently used methods where breakage is achieved by alternative means and explosives are essentially eliminated.

Aqueous Extraction Methods: The *aqueous extraction methods* are uniquely reliant on water or an aqueous mixture during mining and processing to recover the valuable mineral by jetting, slurrying, dissolving, or melting. They are grouped in two subclasses: (1) placer mining or related methods and (2) solution mining methods.

Placer mining is used to exploit mineral deposits that are loosely cohesive or are nonconsolidated, such as sand and gravel or alluvium that contain a valuable heavy mineral in a free state. Native gold and platinum, diamonds, tin in the form of cassiterite, and titanium as rutile and ilmenite commonly are found in placer form. Two historical placering methods have been modernized and find application for a variety of mining purposes; they are hydraulicking and dredging. *Hydraulicking* (also called *hydraulic mining)* utilizes a high-pressure stream of water that is directed against an exposed bank, thereby undercutting it and causing it to collapse. *Dredging* accomplishes extraction of the ore minerals mechanically or hydraulically, normally from floating vessels. In both of these methods, if the objective is extraction, the valuable mineral constituent, generally heavier than the waste material, is removed from a water-base slurry by concentration. On a tonnage basis, however, both of these methods find widest application in mining fields other than placering and for many purposes other than mineral extraction (e.g., tailings transport, ore slurrying, overburden stripping, land reclamation, etc.).

Solution mining includes both *in situ techniques* and *surface techniques.* Examples of the former are salt wells, uranium dissolution, and the Frasch process to melt sulfur. Surface techniques principally involve solvent leaching of mineral values from heaps or dumps or an insoluble matrix or host rock.

Hydraulicking, dredging, and the solution mining methods are the most economical of all exploitation methods but can be used only for mineral deposits that are easily excavated and susceptible to aqueous (solution) attack. They employ unique and dissimilar cycles of operations and bear little resemblance to the mechanical extraction methods. Placer mining is applicable to the recovery of heavy minerals from shallow alluvial and other unconsolidated deposits; it lends itself to large-scale, continuous operation, especially dredging. Solution mining, on the other hand, is employed both for surface and deeply buried deposits of small size; hence it is a hybrid method. Generally, no personnel are exposed to underground operations, however, so it is properly regarded as a surface method.

Two nonmining applications of the aqueous methods are worth mentioning: channel dredging and creation of storage openings and waste repositories by solution mining.

A comparison of all surface methods is contained in Sec. 16.

1.2.5 UNDERGROUND MINING [Part V]

If the appeal of surface mining lies in its mass production and minimal-cost capabilities, then the attraction of underground mining stems from the variety and versatility of its methods to meet conditions too demanding and extreme for surface exploitation. True, underground mining cannot compete with surface mining today in its share of US mineral production. But the United States depends heavily on underground mining for certain essential and/or strategic minerals: all or most of its fluorspar, lead, potash, trona, and zinc come from underground mines, plus a significant part of its bituminous coal, gold, molybdenum, salt, and silver. Regardless, then, of present status and past trends, it seems safe to conclude that (1) underground mining still occupies an essential role in mineral exploitation, and (2) no drastic diminution in application is foreseeable.

While it is always risky to attempt to predict trends and the future, indications seem to favor an eventual return of underground mining to the prominence it once held. Reasons include (1) increasing deposit depths, (2) limited mobility of large surface machines, (3) ever-tightening environmental constraints, and (4) promising advances in underground rock-boring and continuous mining equipment. We have only to remind ourselves that the ultimate technological limit in all mining is depth, and that underground exploitation effectively postpones the inevitable. (Economics, of course, may impose a shallower limit than technology but never a deeper one.)

1.2.5.1 Underground Mine Development [Sec. 17]

In preparing a mineral deposit for exploitation, development in underground mining requires certain considerations that surface mine development does not. A review of governing factors indicates the least concern for locational criteria (climate, in particular, can almost be neglected, unless the mine requires heating or cooling). The most critical are natural and geologic factors: ore and rock strength, the presence of groundwater, and the rock-temperature gradient must be evaluated carefully (terrain is less important, because the surface plant is less extensive in underground mining). Social-economic-political-environmental factors can pose a plethora of problems in underground mining: a more skilled labor force must be recruited, financing may be more difficult because of the higher risk involved, and subsidence may occur.

The extent of access development performed prior to exploitation also differs. Surface mining requires considerable excavation if overburden exists, as is the normal case, and extensive surface area may be tied up with stripping activity and waste disposal prior to the commencement of actual mining. On the other hand, only limited excavation and relatively small openings are required in developing for underground mining. Overall excavation costs may not be too dissimilar, however, because of the vast differences in opening advance rates and unit excavation costs. Further, in underground mining, more careful attention must be given to siting, lifetime, and the construction scheduling of development openings.

All of the steps comprising the general sequence of mine development apply to and are usually performed in underground mining. One unique environmental feature—carried out as an auxiliary operation—is the necessity to provide an artificial atmosphere as a means of life support for the miners. The mine ventilation system utilizes access and production openings to distribute fresh air of the quality and quantity desired to all working places. Other than that requirement, underground development openings provide access to the mineral deposit in the broad sense, permitting entry of miners and materials (equipment, supplies, power, and water) as well as egress for the product mined and any attendant waste.

On occasion, underground development openings double for exploration purposes, and vice versa. Those openings driven in advance of mining can provide valuable exploration information and afford suitable sites for exploration drilling and sampling. Likewise, openings constructed for exploration purposes sometimes can be utilized later as development workings.

Mine development in the underground locale is more specialized, extensive, and expensive than on the surface. Development openings are classified (by rank order of importance) as primary or main, level or zone, and lateral or panel. Primary access is provided by a shaft, slope (decline or incline), or drift or adit (see Fig. 1.2.2). Secondary openings include crosscuts, laterals, raises, winzes, and ramps. Design factors to be taken into account in mine development are the type of mining method, production rate, mine life, and interval between levels. The overall physical plant required to conduct subsurface mining has three components: surface, shaft, and underground. Of these, the hoist plant is unique and a major task of engineering design.

1.2.5.2 Underground Mining Methods [Secs. 18, 19, 20, 21]

Mineral exploitation in which extraction operations are carried out beneath the earth's surface is termed *underground mining* (Hustrulid, 1982). Underground methods are employed when the depth of the deposit, the stripping ratio of overburden to ore (or coal or stone), or both become excessive for surface exploitation. Once the economics has been established, then the selection of a proper mining method hinges mainly on (1) determining the appropriate form of ground support, if necessary, or its absence, and (2) designing an appropriate opening configuration and extraction sequence to conform to the spatial characteristics of the mineral deposit.

Reflecting the importance of ground support, underground mining methods are categorized in three classes on the basis of the extent of support utilized. They are unsupported, supported, and caving, with individual methods differentiated by the type of wall and roof supports used, the configuration of production openings, and the direction in which mining operations progress.

Unsupported Methods: The *unsupported class* consists of those underground methods that are essentially self-supporting and require no major artificial system of support to carry the superincumbent load, relying instead on the walls of the openings and natural pillars. (The superincumbent load is comprised of the weight of the overburden and any tectonic forces acting at depth.) This definition of unsupported methods does not preclude the use of rock or roof bolts or light structural sets of timber or steel, provided that such artificial support does not significantly alter the load-carrying ability of the natural structure.

Theoretically, the unsupported class of methods can be used in any type of mineral deposit (except unconsolidated or placer) by varying the ratio of span-of-opening to width-of-pillar to achieve the desired mine life expectancy. Since the stable size of opening is determined by the depth and the mechanical properties of the ore and overlying rock, the safe span conceivably could range from a few feet (meters) to over 100 ft (30 m). Practically, the unsupported methods are not universally applicable and are limited to deposits with favorable characteristics. The unsupported class, however, is still the most widely used underground, producing over 80% of the ore and mineral from US subsurface mines.

Unsupported methods of mining are used to extract mineral deposits that are roughly tabular, flat or steeply dipping, and generally in contact with competent wall rock. This class consists of five methods: room and pillar mining, stope and pillar mining, shrinkage stoping, sublevel stoping, and vertical crater retreat mining.

Room and pillar mining is adaptable to regular flat-lying deposits, with the advance horizontal; support of the roof is provided by natural pillars of coal or ore that are left standing or recovered in a systematic pattern, and rooms are cut from access entries to provide working faces. When necessary, additional support is supplied by roof bolts or timbers. *Stope and pillar mining* (a stope is a large production opening) is a similar method used in noncoal mines where thicker, more irregular ore bodies occur; the pillars are usually spaced randomly and consist of waste or relatively low-grade ore, since the richer ore is extracted in the stopes. These two methods—room and pillar and stope and pillar—account for approximately 75% of all underground mining in the United States. In *shrinkage stoping,* mining progresses upward, with slabs of ore being broken along the length of the stope. The broken ore is allowed to accumulate in the stope to provide a working platform for the miners and is thereafter withdrawn through chutes into haulage drifts on the level below. *Sublevel stoping* differs from shrinkage by providing several working benches, aligned vertically or staggered, with breast (horizontal) mining on each bench. Long blastholes are drilled into the ore in a parallel or fanlike pattern to fracture the rock. *Vertical crater retreat (VCR) mining* is one of the few patented mining methods, originating from sublevel stoping. Large, parallel, vertical drillholes permit placement of nearly spherical explosive charges, the ideal shape for blasting; horizontal slices of ore are then broken into an undercut. The VCR method is applicable to ore of only moderate strength.

Unlike surface mining, there is little distinction in the cycle of operations for the various underground methods (except in coal mining), the differences occurring in the direction of mining (vertical or horizontal), the ratio of opening-to-pillar dimensions, and the nature of artificial support used, if any. Of the unsupported methods, room and pillar mining and stope and pillar mining employ horizontal openings, low opening-to-pillar ratios, and light-to-moderate support in all openings. Shrinkage and sublevel stoping and VCR mining utilize vertical or steeply inclined openings (and gravity for the flow of bulk material), high opening-to-pillar ratios, and light support mainly in the development openings.

Supported Methods: The *supported class* of underground mining methods consists of those methods that require substantial amounts of artificial support to maintain stability in exploitation openings and systematic ground control throughout the mine. Supported methods are used when production openings will not remain standing during their active life and when major caving or subsidence to the surface cannot be tolerated. In other words, the supported class is employed when the other two categories of methods—unsupported and caving—are not applicable.

Support systems for production workings are chosen to provide varying degrees of controlled wall closure and ground movement. Next to pillars, the most satisfactory form of support is backfill, which approaches 100% in its ability to support the superincumbent load without yielding. In certain instances, some yielding is acceptable and, in fact, preferable because artificial support cannot hold the superincumbent load. Heavy support systems of this type include timber stulls and cribs, timber or steel sets and trusses, and steel jacks, props, arches, chocks, shields, and canopies. Timber is weaker and yields more than steel (sometimes a desirable feature) but is readily available, flexible, workable, easy to install, and economical.

The supported class of mining methods is intended for application to rock ranging in competency from moderate to incompetent. (A competent rock is defined as rock that, because of its physical and geologic characteristics, is capable of sustaining openings without any heavy structural supports.) There is one major method in this class—cut and fill stoping—and two minor ones—stull stoping and square set stoping. They find application in metal (and nonmetal) mining but account for only a small percentage of US underground mineral production. All are vertical stoping methods.

Cut and fill stoping is usually employed for weak tabular deposits. As mining progresses, normally upward, sand, tailings, or waste backfill is placed in the stope to provide support for the walls. The ore, recovered in horizontal slices, is moved to chutes or orepasses mechanically, and the waste is usually distributed hydraulically. *Square set stoping,* a timbered-support method, likewise involves backfilling; however, it also relies on timber sets to support the walls during mining. These timber sets are assembled in a continuous support structure to form skeletal prisms that are subsequently filled with waste material for long-term support. Since it (with quarrying) is the costliest of all methods, it is generally used only in rich mines having very weak structure and is nearly obsolete today. *Stull stoping,* also a timbered method, is a small-scale, supported method using single timbers of rock bolts in narrow, tabular, pitching ore bodies. Cut and fill and stull stoping are intended for moderately competent rock, while square set stoping is suitable for the least competent rock.

The supported methods have declined in use in the decades since World War II (to an estimated 5% in US underground mines). Only cut and fill stoping lends itself to mechanization; consequently, costs of the other methods have risen relatively. Also the ranges of application of the unsupported and caving classes have tended to broaden in recent years, overlapping the former province of the supported class.

Caving Methods: The two classes of underground methods just discussed focus on maintaining exploitation workings open, essentially intact, for the duration of mining. If the ore and rock are sufficiently competent, unsupported methods are adequate; if ore and rock are incompetent to moderately competent, then supported methods may be used. There is also a class of methods in which the exploitation workings are designed to collapse; that is, caving of the ore or rock or both is intentional and the very essence of the method.

Caving methods may be defined as those associated with induced, controlled, massive caving of the ore body, the overlying rock, or both, concurrent with and essential to the conduct of mining. Subsidence of the surface eventually follows. There are three major caving methods: longwall mining, sublevel caving, and block caving. Longwall mining is used in horizontal, tabular deposits, mainly coal; the others have application in inclined, vertical, or massive deposits, almost exclusively metallic or nonmetallic. The caving class accounts for about 15% of US underground mineral production, a sharply increasing amount. In cost, this class includes a moderately priced method as well as the two cheapest of all underground methods.

Longwall mining is a caving method particularly adapted to thin seams, usually coal or nonmetallics at some depth. In this method, a face of considerable length (a "long" wall) is maintained, and, as the mining progresses, the overlying strata are caved, thus promoting the breakage of the coal itself. Widely used abroad, longwall mining for coal production is growing rapidly in popularity in this country. A different method, *sublevel caving,* is employed for a dipping tabular or massive deposit. As mining progresses downward, alternate slices of ore are mined out and the intervening layers of ore recovered by caving. The overlying rock is also subsequently caved. *Block caving* is a remarkable, large-scale, mass-production method that is highly productive, low-cost, and conceptually ideal for massive deposits that must be mined underground. A large block of ore, a few hundred feet (meters) to a side, is undercut and thereby caused to cave. As the block fragments and collapses, the ore is drawn off through chutes or loading points into haulage drifts. Block caving, with longwall, is the most economical of all underground methods because production is high and, except during the undercutting operation, manpower requirements are low. It is adaptable to weak or moderately strong ore and rock bodies and also to massive or dipping tabular deposits of considerable size that are cavable.

Because exploitation openings are deliberately destroyed in the progress of mining, the caving class is unique. Rock mechanics principles are applied to ensure that caving, in fact, does occur—rather than to prevent the occurrence of caving. In effect, the cross-sectional shape of the undercut area (i.e., the width-to-height ratio) is sufficiently elongated to cause failure of the roof or back. Further, development openings have to be designed and located to withstand shifting and caving ground, as well as subsidence that usually extends to the surface. Production must be maintained at a steady, continuous level to avoid disruptions or hangups in the caving action. Good mine engineering and supervision are indispensable to a successful caving operation.

The various underground methods are compared in Sec. 21.

1.2.6 SUPPLEMENTAL TOPICS [Part VI]

1.2.6.1 Novel and Innovative Mining Methods [Sec. 22]

There are several unique mining methods that are not included among the traditional surface and underground methods just described. They are termed *novel methods,* defined as methods that employ new or innovative principles or technologies, or exploit uncommon resources, and that are not yet widely accepted in practice.

The distinction between traditional and nontraditional methods is not as sharp as we might at first expect. Just as classical methods evolve, are modified or combined with other methods, or become obsolete and fall into disuse, so novel methods may in time receive the acceptance that warrants their reclassification into one of the traditional categories. Good examples are auger mining and solution mining, which a relatively short time ago were exploitation curiosities. Some of the novel methods examined are on the verge of winning wide enough acceptance to justify a change of status; others will sink into oblivion. Further, other methods, as yet only concepts or undiscovered, will most certainly emerge to supplement the novel methods now recognized.

How do novel mining methods originate? In past times, they evolved almost entirely from operating experience within the industry. That is not as true today. Technology transfer is occurring from other industries and endeavors. Military and space hardware and concepts frequently find application in diverse branches of industry, including mining. Also look for research and development within the mineral industry to contribute to the adoption of new methods in the future, both traditional and novel (e.g., VCR mining and mechanized sublevel caving both resulted from industry R&D).

Table 1.2.2 lists as examples nine current nontraditional mining methods and the mineral commodities to which they are applicable (relative costs are omitted because reliable data are lacking). They may be classified as to the likelihood of their eventual commercial application as (1) limited existing use, (2) promising but not yet in use, and (3) questionable or unlikely use. While some novel methods are intended for surface exploitation, all but one are applicable to underground mining. However, they all tend to be restrictive or specialized methods, limited as to conditions of use. Some comments on the principle, importance, and status of the major novel methods follow.

Existing Methods: 1. *Rapid excavation:* Still more concept than practice, rapid excavation is intended to replace the intermittent operations of rock breakage and materials handling in hard-rock mining with a system of continuous extraction. It seeks to develop boring-machine technology to achieve truly rapid advance and continuous operation in low-drillability rock. Not so much a mining method as an improved cycle and system of operations, rapid excavation offers revolutionary prospects in many fields of mining, including the boring of tunnels and shafts as well as raises. Truly continuous extraction and handling systems for hard-rock mining await a breakthrough and remain a distant possibility; but progress is being made, and the legitimacy of the goal is now widely accepted for both development and exploitation.

2. *Automation and robotics:* Evolving from cost-driven concepts of mechanization and automation, humanless or remote control in mining is especially attractive for reasons of safety. Widespread adoption depends upon more technological ruggedness, especially for the underground regime, which in turn should produce economic feasibility.

3. *Hydraulic mining:* Applications of water-jet and borehole-slurry technology are advancing slowly into various unit operations (penetration, fragmentation, and handling), toward a clear goal of an integrated mining system. Extension from coal to harder rock is a companion objective.

4. *Methane drainage:* Signs are favorable for rapid expansion of coalbed degasification throughout the underground coal-mining industry, in part on safety grounds but also with economic justification. Drainage from seams that are not actively being mined is equally attractive and coming to the fore as a competitive source of natural gas.

Promising Methods: 1. *Underground gasification:* Ripe with promise for difficult natural conditions, in situ coal gasification and combustion has been fraught with economic risk and technological difficulties. It involves the partial combustion of coal in place, generally through boreholes, with the collection of a low-quality gas at the surface.

2. *Underground retorting:* In situ oil shale retorting, in which pyrolysis of kerogen occurs in place, faces some technological uncertainties but, more serious, has yet to demonstrate economic viability. Unfortunately, its future is tied to that of the synthetic fuels industry, which presently is held economic hostage by the international oil cartel.

3. *Marine mining:* There are intriguing technological possibilities for mining rich unconsolidated nodule and mud deposits in the deep oceans. However, for deposits located in international waters, political and legal risks are too great until an acceptable treaty of the seas is negotiated. Exploitation appears much more likely for resources located within the so-called exclusive economic zone (EEZ) declared by the United States. Deposits of interest include cobaltiferous seabed crusts, massive sulfides, and deep offshore placers. Worldwide interest in the various marine resources is high.

Questionable Methods: 1. *Nuclear mining:* No applications are likely as long as radiation hazards are uncontained (and the Limited Nuclear Test Ban Treaty remains in effect), in spite of technological promise and economic attractiveness for certain fragmentation applications in underground mining.

2. *Extraterrestrial mining:* The furthest out of all the novel methods, colonization of outer space (most likely site: the moon) is a must to justify risky, untried extraterrestrial mining. Launching of a US space station would revive interest in the concept.

Other Methods: A variety of other emerging concepts in mining is also discussed.

1.2.6.2 Evaluation of Mining Methods and Systems [Sec. 23]

Earlier discussions dwelt on individual mining methods, or classes of methods, their characteristics and conditions. Finally, an overall comparison and evaluation and some selection procedures are needed, limited here to traditional methods.

Method Features: It is not possible to compare all the features associated with surface and underground methods, but one can note the principal advantages and disadvantages of the two locales.

1. *Mining cost:* Except in rare cases, relative costs (quarrying is an exception) are significantly less for surface mining; underground costs are higher but variable, with caving lowest and supported highest.
2. *Production rate:* All surface methods (except aqueous and quarrying) moderate to high; underground low to moderate (except high for caving and some unsupported).
3. *Productivity:* Surface much higher than underground in nearly all cases.
4. *Capital investment:* Generally small for aqueous and large for other surface, but larger for underground; surface equipment more expensive, but underground development costlier.
5. *Development rate:* More rapid for surface.
6. *Depth capacity:* Limited for surface (except for solution mining); range from limited (unsupported) to somewhat unlimited (supported) underground.
7. *Selectivity:* Generally low for surface, variable underground.
8. *Recovery:* Generally high for surface (except aqueous), variable from low to high underground.
9. *Dilution:* Generally less for underground (except for caving).
10. *Flexibility:* Underground tends to offer more flexibility than surface, although surface may be more adaptable to change.
11. *Stability of openings:* Generally higher for surface; more difficulty to attain and maintain underground.
12. *Environmental risk:* Substantially higher for surface, except that subsidence may be severe with underground methods.
13. *Waste disposal:* May be serious problem for surface, minor underground.
14. *Health and safety* (including atmospheric control): Vastly superior for surface.

Of the method features noted, the most important favoring surface mining are cost, production rate, productivity, recovery, and health and safety. Those supporting underground mining are depth tolerance, selectivity, dilution, environmental risk, and waste disposal.

Overall Considerations: Subjectively, we may conclude from the preceding comparison that, excluding depth limitations, surface mining is usually preferred over underground. There are certain significant factors that favor underground over surface mining, however, as noted previously, and these may govern in certain circumstances. Final judgment in a specific case awaits determination of costs and an economic analysis of competing candidate methods.

Cost Analysis: The ultimate basis for decision making in selecting a mining method is economics. Assuming that safety and technological considerations are satisfied, cost estimates are prepared for all candidate methods in order to make a final choice. Usually the process is performed in two stages: (1) if the deposit depth is shallow to moderate, compare approximate costs first for the general categories of surface vs. underground mining; and (2) once that has been resolved with some certainty, compare specific costs for promising, individual mining methods. Generally, direct mining costs including prospecting, exploration, development, and exploitation will suffice, but in some instances, consideration of all production costs (overhead, mineral processing, smelting, transportation, etc.) is necessary. Inherently, underground mining costs typically exceed surface mining costs by a margin of three to four to one. Relative costs in Table 1.2.2 provide an indication of the range.

Selection Procedures: A variety of procedures, including decision-making matrices, have been devised to aid in the selection of the most suitable mining method. Most are subjective (Peele, 1941; Young, 1946; Lewis and Clark, 1964; Hamrin, 1982; Hartman, 1987), but in recent years numerical techniques utilizing the computer have been developed (Nicholas, 1981). By permitting consideration of virtually an unlimited range of factors, a quantitative selection procedure is far more likely to result in the optimum choice.

1.2.6.3 Openings for Nonmining Purposes [Sec. 24]

Often, excavations in the earth are employed for purposes other than mineral extraction. These include both *civil and military works* in which the objective is to produce a stable opening of desired size, orientation, and permanence. Examples are vehicular tunnels, storage reservoirs, waste disposal chambers, and military installations. They are excavated using methods that are borrowed from mining. Since the objective is the excavation or opening itself rather than the mineral extracted, however, other kinds of conditions or circumstances may govern, such as time, shape, or life. Because the excavation technology is so similar to that used in mining, it is only mentioned here. Rather, attention is given to the variety of openings created, design criteria, and utilization factors.

Tunnels, Sewers, and Water Diversion Openings: The majority of civil construction is directed toward openings in this category. While tunnels serve several purposes, most today are driven for vehicular use (e.g., automobile, rail, subway). Sewers and water diversion openings are similar in size and appearance. Tunneling methods consist both of soft-ground and rock excavation, plus some specialized techniques such as cut-and-cover, shield, caisson, and immersed tube. Like mining, tunneling has become completely mechanized and employs continuous, rapid excavation technology whenever possible (tunnel boring machines, in fact, were originally developed for civil works). Governing factors in tunneling, again similar to those in mining, are safety and cost. Because of their greater permanence, however, civil openings tend to be much more expensive, to be more time-consuming to construct, and to require lining or more elaborate support systems.

Storage and Power Generation Openings: Underground openings constructed for storage and power generation purposes are generally larger than tunnels and hence less costly per unit volume of rock broken. In hard rock, because of their larger size, they must be excavated by conventional drill-blast techniques rather than by rapid excavation. In soft rock, susceptible to solution attack (e.g., salt beds and domes), large cavities can be constructed more readily and cheaply. Ground stability is of the utmost importance because of the long spans involved; lining and support systems are expensive to install and usually avoided by driving openings in competent rock. Underground storage is utilized for a host of materials, both solid and fluid, packaged and bulk. Gases stored include compressed air, methane (natural gas), helium, and nitrogen, generally under moderate to high pressure. Petroleum liquids and water are also stored in underground chambers. The newest application is for packaged storage, often of paper goods and records, warehouse inventory, and food stuffs. Power generation openings are usually constructed in conjunction with dams and hydroelectric projects, providing water-conveying pumped-storage facilities.

Waste Repositories: Like other large underground structures, waste repositories are located in competent rock to provide stability and to obviate the requirement of ground support. With the advent of the nuclear age and the need to dispose of dangerous radioactive wastes, underground repositories have assumed new prominence. Subsurface disposal of wastes of all kinds has been employed since the Industrial Revolution first created the need. Originally confined to wellbores, disposal in large chambers is now commonly practiced, often specially constructed to provide security for toxic and radioactive wastes. Disposal of high-level radioactive wastes from nuclear reactors poses a unique public health problem of national dimensions, one currently unresolved; underground repositories are presently the favored solution.

Military and Defense Installations: Underground facilities serve many military purposes, including materials storage, personnel protection, concealment, weapons testing and emplacements, and troop infiltration. Simple shallow structures are often adequate, but deeply buried ones may be necessary to withstand heavy bombardment or nuclear attack. Some large and very elaborate facilities, such as strategic factories, command headquarters, and submarine pens, have been constructed underground. Standard excavation techniques are employed to construct military installations, complicated by exacting specifications and the frequent need for secrecy and security.

1.2.6.4 Postmining Operations [Sec. 25]

All the steps in processing raw minerals that occur following extraction from the earth comprise *postmining operations.* They result in over a tenfold enhancement in value of US mineral production. It is not within the scope of the *Handbook* to treat postmining subjects at length, but they are introduced and their relationships established to the functions of mining proper.

Storage and Transportation: Materials handling as a unit operation continues beyond mining through processing. At each phase or transition, bulk-material storage and transportation must be provided. Bins, silos, hoppers, and stockpiles may be required for storage, with attendant transfer feeders, stackers, and reclaiming machines. Transportation occurs by rail, road, barge or ship, or conveyor (belt, hydraulic, or pneumatic). Design of these facilities is not often the responsibility of the operating mineral or metallurgical engineer, but selection is. Because of the specialized nature of the equipment, consultants, manufacturers, or engineer-constructors often provide the expertise.

Mineral Processing: It is termed cleaning or washing if the mineral is coal and milling or concentrating if it is an ore. Interfacing directly with mining, mineral processing requires close communication and coordination with the extraction activities. Almost continuous monitoring of the run-of-mine product as to tonnage and grade is mandatory; feedback control loops permit adjustments in mining practice to meet processing demands. Mineral processing today may be an intricate succession of treatment stages: comminution is followed by screening or classification, then by one or more beneficiation processes, and finally by agglomeration, dewatering, and drying. The treatment flowsheet is designed uniquely for the mineral commodity being processed. Disposal of wastes (tailings, culm, reagents) must always accompany mineral processing.

Chemical and Electrolytic Processing: Metallic ores often incur further processing during leaching, solvent extraction, ion exchange, electrowinning, and electrorefining. These traditional steps are classified as chemical and electrolytic processing, and the treatment plant to accomplish them may or may not be located at the mine or with the mineral processing facility. Environmental control is a necessary auxiliary operation for this step.

Sales and Marketing: Following one or more steps of processing, the final coal/mineral/metal product is ready for market. Many large organizations are vertically integrated, conducting their own exploration, mining, processing, and marketing. Smaller companies tend to be limited to a single stage of operations and must sell their product to others for processing and consumer marketing. Marketing practices vary for each of the three major mineral commodity groups—fuel, metal, and nonmetal—in accordance with the economic uniqueness of each industry (Strauss, 1986).

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Chapter 1.3
MINERAL ENGINEERING EDUCATION

LAURENCE H. LATTMAN

1.3.1 INTRODUCTION

The educational disciplines covered in this chapter include mining engineering, geological engineering, environmental engineering, and metallurgical (and materials) engineering. The general term *mineral engineering* is here used to describe these fields, although mineral engineering is a separate program at a few schools. The boundaries between some of these disciplines are not sharply drawn, and what one school calls metallurgical engineering may largely overlap what another school calls mineral engineering. In addition, data on each of these areas of study are not equally complete or precise. For example, mining engineering enrollment information is extensive and accurate, whereas enrollment data on metallurgical engineering may be combined with materials engineering and thus may require interpretation. Nevertheless, general trends in enrollment at the undergraduate and graduate levels and changes in programs are available and worth noting.

There are today 37 accredited schools offering engineering degrees in the mineral fields (Table 1.3.1). The complete listing is issued annually by the Accreditation Board for Engineering and Technology (ABET), Inc. (Anon., 1987). A guide to mineral schools, giving faculty and enrollment data, is published by the Society for Mining, Metallurgy, and Exploration (SME), Inc. (Anon., 1988). Several schools report consideration being given to dropping or combining various mineral engineering programs, but no major changes were made in the 1987-1988 academic year. These considerations have been driven by significant enrollment drops since the early 1980s.

1.3.2 ENROLLMENT

1.3.2.1 Trends

While historically very cyclic, the most striking feature of enrollment trends in the past several years has been the decline. As shown in Fig. 1.3.1, the decrease in undergraduate enrollment in mining engineering has been steady since 1980–l981 and particularly rapid during 1983 to 1986. Graduate enrollment has remained essentially constant during that time. The numbers of graduating mining engineers (BS degrees) peaked in 1983-1984 and has been dropping steadily since. The projected graduates are expected to decrease through 1990-1991. In number, advanced degree graduates have remained close to constant.

A fact not brought out by Fig. 1.3.1 is that enrollments in mineral engineering have not dropped similarly at all schools. Some have kept enrollment reasonably steady, whereas others have had a very large drop-as high as 75% in some cases. Ashworth (1986) has given an excellent summary of mining engineering enrollment through 1985.

The data cited apply to mining engineering, but telephone discussions with colleagues indicate that all mineral engineering programs have had essentially a similar recent enrollment history.

Many schools report unused scholarships. Thus it is concluded that lack of financial support is not a contributor to the enrollment decline. Rather, it is almost universally attributed to an actual or perceived decrease in job opportunities for mineral engineers.

1.3.2.2 Types of Students

There appears to be a change, difficult to document, in the type of student enrolling in undergraduate and graduate mineral engineering programs. The number of international students is increasing. Anecdotal data on all mineral engineering programs indicate that from one-third to one-half of the undergraduate group are now international students, dominantly from Canada, the Pacific Rim countries, and Central and South America. Some schools state that they are actively recruiting international students.

At the graduate level, over one-half (perhaps 60%) of the students are international, and many do not plan to remain in the United States after graduation.

From this information, it is apparent that the numbers of US citizens in mineral engineering programs are probably decreasing at a faster rate than the total enrollment data alone indicate.

1.3.3 BASIC REQUIREMENTS FOR THE BACHELOR'S DEGREE

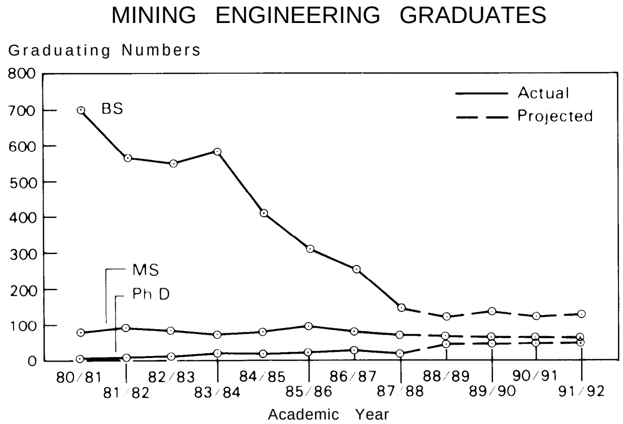
Most mineral engineering schools design their undergraduate programs to meet ABET criteria for accreditation. For the 1988-1989 academic year, program criteria have been developed for geological engineering; metallurgical, materials, and similarly named engineering programs; and mining and similarly named engineering programs. ABET notes that materials and similarly named engineering programs were separated from metallurgical and related engineering programs.

Details of accreditation requirements for any program can be obtained from ABET (Anon., 1987b). The requirements include faculty size and qualifications as well as curriculum needs in mathematics, basic sciences, and engineering sciences and design. The fundamental requirements are set by ABET and amplified by various professional societies. For example, SME-AIME formulates the program criteria for mining and geological engineering, expanding on ABET's basic requirements in science, humanities, laboratory experience, computer-based experience, etc.

Discussion of ABET accreditation criteria is continuous in academic circles, and the criteria commonly undergo modification. They should be carefully reviewed when seeking initial accreditation or renewal.

It is usually a principle in academe that a student should be held only to the graduation requirements contained in the school's catalog at the time that the student enrolled. This is to prevent changes prolonging the time required for a degree. Many schools, however, while not requiring students to change plans, strongly recommend such changes during the student's education, if these changes are part of new ABET criteria. This potential problem can best be resolved on an individual basis.

Individual program criteria are not discussed here as they change frequently and are available from ABET (Anon., 1987b).



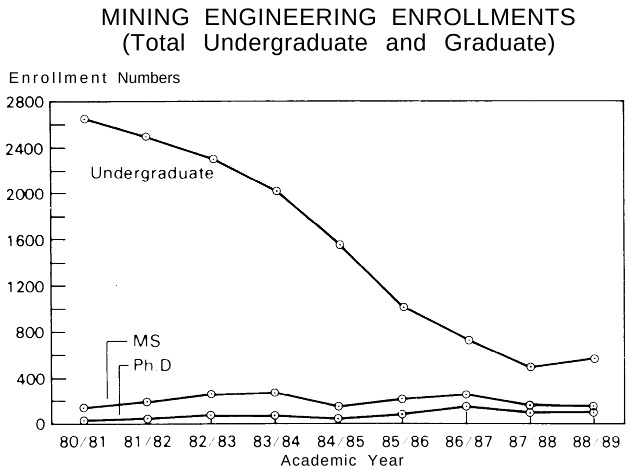


Fig. 1.3.1. Mining engineering enrollments and graduates. (Rahn, 1988. Used by permission.)

Table 1.3.1. Number of Colleges and Universities with Selected Accredited Engineering Programs

Engineering Programs	Number of Colleges and Universities with Accredited Programs
Environmental Engineering	18
Geological Engineering	16
Materials Engineering[a]	22
Metallurgical Engineering[b]	37
Mineral Engineering	2
Mineral Processing Engineering	3
Mining Engineering	21

Source: Anon., 1987a.
[a] Combined with Materials Science and Engineering.
[b] Combined with Metallurgy, Metallurgy and Materials Science Engineering, and Extractive Metallurgy.

Clearly, faculty should take an active and progressive interest in these requirements through their professional societies as well as within their departments. Some recent program changes and trends are discussed in the next section.

1.3.4 SOME RECENT CHANGES IN EDUCATIONAL EMPHASIS

Many schools report changes in faculty thinking about undergraduate and graduate educational requirements in the mineral sciences. As is so often the case, many of these changes require new courses and can only be accommodated by dropping existing courses or creating a five-year undergraduate program. There is considerable reluctance by schools to force mineral engineering into a five-year program because of increased costs, possible further loss of student interest, and lack of support from industry.

The major single change, begun some years ago and continuing apace, has been the increased emphasis on computer training and application. Computers are now an integral part of mineral engineering and extend into reserve estimation, design, planning, automatic control, and operations. There is no doubt that computing and automation have made great changes in mineral engineering in the past decade and will continue to cause major curriculum changes for some years to come. It is considered impossible to graduate a well-educated mineral engineer today without significant understanding and skill in these areas.

In addition to the fundamental changes mentioned, there is increased concern in mineral engineering education today for coverage of the environment (including reclamation), some study of laws and regulations that have so major an impact on the industry, and at least an introduction to finance and management. These areas, while not new to mineral engineering education, are receiving increased emphasis. New courses are being added and are not infrequently being taught by persons outside the degree-granting department. Existing courses are undergoing major revisions.

To accommodate all these changes, particularly computer applications, automation, and environmental and reclamation emphasis, some standard older courses are being dropped. One school reports that students now enrolling in mining and geological engineering will no longer have to take a separate petrology course. Petrology will be subsumed under several other courses.

Obviously, mineral engineering education is undergoing great change with new and increased resultant demands on students, faculty, laboratory equipment, and space. The problem reported by many schools is that these increased needs come at a time of decreasing enrollment causing, in some cases, a reluctance on the part of central administrations of universities to fund such changes.

The changing emphases noted apply to graduate as well as to undergraduate programs. Schools report especially that these changes are clearly manifest in graduate research and thesis efforts. The traditional greater flexibility of graduate programs has, however, resulted in less major changes in degree requirements.

1.3.5 PROBLEMS OF MINERAL ENGINEERING EDUCATION

In common with all engineering education in the United States, mineral engineering education faces two pressing problems. These are a limited pool of potential students and a probable lack of qualified future faculty.

1.3.5.1 Pool of Students

Several recent studies have indicated that the United States faces a shortage of 300,000 to 500,000 engineers by the year 2010, if universities continue to attract only the traditional engineering student. The most obvious source of additional students is from an increase in the number of women and ethnic minority students entering the engineering profession. It is estimated that the number of women must be doubled and the number of ethnic minority members quadrupled to forestall the impending severe shortage of engineers. Detailed figures as to the needed increase to prevent a shortage of mineral engineers, in particular, are not available.

Nevertheless, it is clear that women and ethnic minority students must be attracted in increasing numbers to the mineral profession. Nearly all engineering schools have active recruitment efforts underway to increase participation of these groups, and the mineral engineering efforts are strong. Industry and professional societies have worked closely with the universities in sponsoring summer short courses, special high school programs and talks, literature, scholarships, and other efforts to recruit women and minorities. These efforts have resulted in some gains, but most mineral engineering schools feel that the efforts must continue to be strengthened and plan to do so.

1.3.5.2 Faculty Shortages

Another concern common to all engineering disciplines is a predicted shortage of qualified faculty. Because over one-half of current graduate students are international students, it is believed that inadequate numbers of future faculty will be available. Although most schools express concern about this problem, there does not appear to be a coordinated, or even a broad, effort to address it. In fact, the percentage of international students in graduate programs has been slowly and steadily increasing over the last several years. Some schools are aiding outstanding foreign graduate students to meet residency requirements and remain in the United States.

Although the great majority of schools seek young faculty holding the PhD degree, there is a growing recognition of the need to obtain help from experienced, part-time, older engineers as faculty members. Retired persons or engineers granted time by industry and government can be of very great help, but schools continue to be concerned about shortages in the traditional, young, PhD-holding faculty pool.

1.3.5.3 Funding Problems

Besides the two major problems discussed, many schools report concern about funding problems in mineral engineering. This concern is felt in two areas—faculty salaries and general support funds.

In some schools, mineral engineering faculty salaries have fallen behind those in more “glamorous” fields of engineering. Such disciplines as electronics, computer, and biomedical engineering are attracting more university salary support, perhaps at the expense of mineral engineering.

Also funds for new equipment, maintenance, and replacement equipment are in increasingly short supply. Concern over the impact of this on instruction and research is frequently expressed by academic people.

1.3.6 FUTURE OF MINERAL ENGINEERING EDUCATION

The section which follows is based largely on informal discussions with colleagues and does not necessarily represent a consensus of mineral engineering educators.

1.3.6.1 General Comments

Without doubt, computer use and instruction will increasingly become a part of course work in mineral engineering programs. The objective will be to give the student a firm familiarity and skill in computer application in addition to formal course work in computer science taken outside the mineral departments. The concentration of instructional effort appears to be on the increasingly powerful PCs and work stations, but some work with larger computers is included. It is believed by many faculty that some computer use will appear in at least one-half of all specialized courses taught in mineral engineering programs.

Automation, which includes process control of all types, is another area that will receive increased instructional and research attention. The object will be to give the student familiarity with the great potential of automation to increase efficiency and reduce costs. It is recognized that this is a specialized area of expertise, but an understanding of principles and general applications must be a part of the mineral engineering graduates’ background. Some departments indicate a strong interaction with chemical engineering departments in instruction in automatic process control.

1.3.6.2 Specific Needs

Foreign Exploration: As new mineral exploration efforts are likely to be concentrated outside the United States, those students who will work in exploration will need to receive training in the economic geology of the Pacific Rim countries and South America, for example. Such courses are already planned at some schools.

Environmental Education: Although environmental engineering and reclamation will become more and more a specialized profession, all mineral engineers will need some education in these important areas. Students specializing in these fields will have course work in such disparate areas as aquatic chemistry, hydrogeology, and air pollution. Legal and regulatory aspects of environmental problems are already covered in some courses. Schools report increasing industrial interest in students receiving such training. The entire field called *waste isolation* will become part of the mineral engineers’ background.

It is believed that course work in this area will require several instructors in each course because of the range and complexity of material covered. Also, graduate student research in environmental studies will become increasingly common. Several schools believe job opportunities in environmental and reclamation specialties are very inviting to increasing numbers of students. As shown in Table 1.3.1, there were 18 accredited programs in environmental engineering in the United States as of 1987, although all were not closely allied with the mineral field.

Materials: In many, but not all, schools materials, engineering has traditionally been combined with metallurgy or extractive metallurgy. Increasingly, materials science and engineering will become a stand-alone program. In this respect, ABET has established separate accreditation criteria for materials and similarly named engineering programs, and metallurgical, materials, and similarly named engineering programs. The rapidly expanding academic efforts in ceramics, polymers, and composite materials are in large part responsible for this change. Metals are no longer the dominant subject in many materials engineering programs. Extractive metallurgy really does not fit into the new materials engineering degree programs at all.

The outcome of what is essentially a reorganization will be different at various schools, and a realignment of departments is inevitable. Nevertheless, in some schools, there is a desire to have mineral engineers receive some familiarity with the new materials being developed, especially in their application to corrosion problems, grinding, transportation, etc. This is simply another example of the increasing range of subjects with which educators want mineral engineering students to have some mastery.

Special Subjects: Schools have expressed growing interest in their mineral engineering students having an introduction to such diverse subjects as budgeting, management, and public relations. Efforts in these areas vary widely among schools and are commonly driven by the concerns of one or two faculty members. There does not appear to be, however, an increasing interest in including them in graduate programs.

Graduate Studies and Research: Historically, a student specializes at the graduate level, and today the student's efforts are more and more controlled by perceived job opportunities. In addition to thesis research in traditional mineral engineering subjects, there is expected to be an increased graduate interest in the subjects given in this segment. Obviously, the input and desires of industry will play a key role in graduate students' efforts.

One other factor has lately appeared. Research funding, as for example, from the US Bureau of Mines and other federal agencies, has been curtailed. As a result, expensive graduate research has been diminished and more and more effort is going into less expensive research, particularly computer modeling. Faculty note this trend away from equipment-intensive research. It is a trend expected to continue.

Continuing Education and Technology Programs: A considerable amount of thinking and planning is going on in the fields of continuing education and engineering technology. There are nine accredited mineral technology programs in the United States (Anon., 1987c). Continuing education is commonly handled in-house by the large companies in the form of specialized training for their engineers, for example, in safety. Upgrading the education of practicing engineers in computer applications and instrumentation is carried out by short courses and, in some cases, by sending engineers back to universities for one or two terms. There is interest by schools in being involved in these efforts for engineers and technicians, and many discussions are underway. Televised courses to plants or offices are also being discussed as a way to reduce lost employee time but achieve the desired education. As yet these efforts in the mineral fields are not widespread or well established, but it is the consensus that they will become increasingly important. Educational institutions are developing continuing education courses in a wide variety of fields.

1.3.7 SUMMARY AND CONCLUSIONS

Clearly, mineral engineering education faces a period of significant change in the near future. This change is driven by a wide variety of factors, chief of which are

1. Declining enrollments at least for the foreseeable future.
2. In some cases, declining university financial support.
3. Some university plans concerning merging or elimination of programs.
4. Some splintering of existing programs such as separation of metallurgical and materials engineering and formation of new programs such as environmental engineering out of older programs.
5. Increased need to recruit women and ethnic minority students.
6. Coming possible shortage of qualified faculty.
7. Major changes in instructional efforts, such as vastly increased computer usage and education and automation and process control receiving strong new emphasis.
8. Calls for more instruction in areas of recent concern such as reclamation and environmental problems, knowledge of laws and regulations, and public relations, resulting in proposals to consider five-year programs.

Mineral engineering programs have traditionally been of high quality and are technically sound at American universities. The decisions made in the next few years may have a very significant influence on degree requirements and methods of instruction, but all educators contacted have insisted vigorously that the quality will remain very high.

There appears to be an increasing tendency for interaction among universities, to exchange data and information so that all mineral engineering programs may benefit from the thinking and experience of others.

Having completed the study for this chapter, the writer cannot help but be very optimistic that mineral industry education, although perhaps involving fewer programs, will continue to produce high-quality, well-trained, modern engineers for the profession. The next generation of engineers will have a significantly different education than its predecessors but will be a major strength to industry and the country.

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Section 2 Mineral Economics

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Chapter 2.0
INTRODUCTION

MICHAEL RIEBER

Mineral economics interfaces the mineral sciences and engineering with finance and economics in the analysis of appropriate questions facing the minerals and energy industries. Its practitioners include those trained in the earth sciences; mining, petroleum, and geological engineering; metallurgy and mineral preparation; as well as finance, information systems, economics, and statistics—most usefully in one or more combinations. Allowing for some overlap, three general areas may be identified: evaluation, economic analysis, and resource appraisal.

Evaluation begins with mine investment analysis (exemplified by Gentry and O'Neil, 1984) and process costing, for which the STRAMM Handbook (Anon., 1977) provides a well-known basis for mining and beneficiation. But it extends from property appraisal, often used for the determination of taxes and royalties, through the determination and comparison of transport costs (e.g., Rieber and Soo, 1977), and to benefit/cost analyses of externalities. Economic analysis, the subject of this section, is displayed in rigor and extent both theoretically and as applied to commodity classes in Vogely (1985), but it is implicit though apparent throughout Lefond (1983). The text edited by Vogely places minerals in their aggregative context with economic growth and trade, proceeds to the theoretical analysis of minerals and their markets, analyzes the structure and performance of the major mineral sectors, and concludes with policy issues: fuel, nonfuel, and environmental. Resource appraisal employs quantitative methods for the estimation of mineral and energy resources. It moves through geologic, geophysical, and geochemical properties to the identification of estimation procedures for mineral endowment and for the economic measure of stocks and flows of specific minerals (Harris, 1984). It can be used to estimate the probable amount of a specific resource remaining in a given region and has been used to project sites in a specific quadrangle with the highest probability of an economic occurrence.

Explicitly or implicitly, the tools of economics are used by all in the mineral industries. Resource appraisal seeks to exclude their implicit use by geologists in the estimation procedures, but explicitly includes it to obtain some of the final results. Among the economic tools required for evaluation procedures, the estimation of revenue and cost streams, the determination of the cost of capital, risk adjustments, the determination of the internal rate of return and present discounted value, and the analysis of the impact of taxes are familiar to all. Indeed, the very concept of a minable reserve has an economic component, as do the related concepts of mining grade, cutoff grade, mill head feed grade, concentration, and the rates of production and depletion. Smelter returns are not invariant with respect to market forces nor is the increasing production emphasis in the copper industry from open pit mining to leach-solvent extraction/electrowinning.

The economics interface exhibits irregularities in the area of economic analysis; in some cases, it has been abrasive. This is a function of the users, not the tools. Problems arise because of the subject matter. Economists, including some mineral economists, tend to be interested in (1) forecasting (supply, demand, price, and the cost of capital), (2) the structure of mineral and commodity industries and the competitive behavior of the firms comprising those industries, and (3) in public policy, especially as related to energy, the environment, and land management. At the more aggregate level, the issues include the relation between minerals and domestic economic growth, the development of mineral resources in the Third World, and international trade policy.

When performed by outsiders or academics, the results of these investigations have arguably not always been well received by the mineral industries. Some have been proven wrong; others have run counter to the natural optimism of any established industry. A few have been detrimental to the perceived interests of particular industries or firms. A common complaint is that the work does not fully comprehend the complexity and detail of the industry involved. There is merit in the complaints, especially when due to bias or ill-conception the results adversely affect the interests of a firm or industry. Nevertheless, it is not the economic tools that are at fault; more usually it is a logical consequence of the initial conditions or relationships assumed or estimated by the user. When externalities are important and when the benefits of an action or policy are received by some and the costs borne by others, even conceptually, agreement is rare.

Of necessity, everyone forecasts; history suggests that no one and no group has proven to be prescient. The difference between a formal and an informal forecast may simply be the time horizon, but is more closely related to the rigor with which the variables of interest are estimated and the basis for the decision upon which their inclusion is made (the defensibility in the statistical sense of the check list). At issue is whether addition or deletion of a fact or data series improves forecasting ability or reduces it, and how do we know? Errors arise due to faulty or incomplete data, a misunderstood change in basic industry relationships, or simply due to technical change. An econometric forecast, relying on a long period of stable relations among the variables to provide the basis for extrapolation, may provide a fine fit for the historical data, yet may prove incorrect in the face of major industry changes. Thus a benefit of formal forecasting is that the use of estimated models in the process of model building may force a more critical examination of assumptions and data than that which takes place in informal forecasting. Finally, the confidence limits of the forecasts are often neglected. Pin-point accuracy, a unique estimate, is not possible, and its use is misleading.

Competitive behavior and the structure of industry is widely investigated, not least by those companies with an interest in marketing and the market penetration of their product. Similarly, the economics inherent in the public policy issues and the aggregate relations between the mining industry and the national or international economies are widely discussed by academics, government personnel, and not least by industry representatives as seen, for example, in the articles and editorials contained in the *American Mining Congress Journal.*

If competitiveness is the watchword, economic as well as geologic factors are important. Grade-tonnage relationships are decisive in the long run, but that time horizon is usually longer than the planning period of a firm. In the interim, economic considerations are likely to be decisive. If the cutoff grades for Zairian and Zambian copper are higher than the Chilean and US mining grades, why are the former not the low-cost producers? Alternatively, if their mining grades were not so high, would the deposits be exploited?

Clearly, no study by outsiders is likely to utilize the wealth of detail or information available to onsite practitioners. But, the needed detail and the complexity of the relationships depend upon the issue to be resolved. Some broad relations, patterns, or trends may be clear only when details are aggregated. As both the level of aggregation and the degree of generality increase, the level of abstraction from reality also increases. For both economists and engineers, the world is not made up of a mass of special cases. The trick is to reduce the problem to manageable proportions, consistent with the desired accuracy of the result, without sacrificing anything essential to the result.

Mineral economics is inextricably associated with exploration, mining, mining engineering, processing, product sales, and residuals management. The chapters of this section comprise a relevant, analytical subset of these areas. The authors represent industry, government, and academia.

2.0.1.1 Chapter Overviews

The mineral economics section begins with Chapter 2.1, John Tilton's exposition of mineral supply, demand, and price formation employing the conceptual tools of the mineral economist's trade. For the economist, it is a useful review and check list. For the engineering nonspecialist, it is a lucid statement, with applications, of the theoretical underpinnings of practical analysis. Read in conjunction with the succeeding chapters, it bares their implicit theoretical assumptions and relationships.

Of the major elements that distinguish mineral economics from economics in general (derived demand for minerals, mineral substitution, co- and byproduct supply, recycling or secondary production from scrap, nondegrading stockpiles or inventories, a market structure often dominated by a few suppliers, depletion, and the importance of the time dimension), only depletion is omitted (it is covered in Chapter 2.4). Recognizing the diversity among the minerals and the need for familiarity with their production processes and markets, Tilton concentrates on the commonality of the economics principles needed for their analysis. From the firm to the industry level, the important elements of supply/demand analysis are explained as is the basis for their importance.

As most minerals are not directly consumed by final users but rather by processors or manufactures for final products, demand is treated as derived from the demand for the final products—a factor in the latter's production process. As it is the attributes of the minerals that are important, the role of substitution among minerals in subsequent manufacturing or use is highlighted.

The response of mineral supply to capital and variable costs over the several planning periods is next demonstrated. To this is added the often important role of co- and byproduct production of primary minerals and the secondary metals available from the recycling of scrap. In both supply and demand, but particularly the former, the role of market structure (the number and size distribution of firms) in price formation and the supply response to price changes is demonstrated.

The chapter concludes with two useful applications: the basis for mineral commodity market price instability, and the use of the incentive price technique for forecasting long-run mineral prices. The latter is the price level that ensures the industry level production capacity required to meet a specified or estimated level of demand.

The diversity of the minerals industry is displayed in Anthony Cammarota's commodity review (Chapter 2.2) of the nonfuel minerals of commercial importance in the United States. It provides the basic familiarity with the production processes and markets for individual minerals necessary for effective economic analysis. Some of the groups into which the individual minerals fall are familiar: base metals, precious metals, ferrous and their alloying metals, and the light metals (aluminum, magnesium, and beryllium). The nonmetallics fall into construction materials and agricultural commodity classes, both based on their demand characteristics. Two interesting categories are then presented; commodities produced principally as byproducts (gallium, arsenic, selenium and tellurium, germanium and indium), followed by a discussion of advanced materials (metals, ceramics, and polymers and their composites, e.g., aluminum-lithium alloys).

That diversity does not lead to singularity of the analytical approach to each commodity, even across commodity groups, is demonstrated by the format of the discussion. A general introductory segment shows the relevance to supply and demand of cost changes, economic cycles, environmental control impacts, and foreign competition. Individually, all commodities are discussed in terms of their location, supply sources, principal processing techniques, and byproduct credits as determinants of supply. Demand is related to the commodity attributes of use to their several consuming sectors. Market structure provides a relation between supply and demand. Transport costs help define market extent, the location of downstream processing, and are related to commodity exports and imports.

The chapter concludes with a discussion of import dependence. The analysis is in terms of the geologic, economic, and political reasons for the growing US dependence on imports for an increasing list of mineral commodities.

Mine management, production planning, and mine investment analysis all require a knowledge of present prices and the ability to forecast future prices. The publicly available price series upon which this depends, however, are often quoted in terms of refined products rather than ores or concentrates, and may be quoted differently on one or more international markets. Some mineral prices are quoted at several stages of processing; the quotations need not move in tandem in the short run. For both trade and planning purposes, it is necessary to understand how prices are determined and what they actually represent. In Chapter 2.3, Simon Strauss provides the required background for pricing and trading in metals and minerals.

Strauss begins with the generic issues: the effects of inflation on prices over time and the impact of volatile currency exchange rates on the prices of internationally traded minerals as separately seen by the buyer and seller. The body of the discussion is devoted to an analysis of the methods used to determine mineral market prices and why particular methods are used for different classes of minerals. Discussed separately, with examples, are the bases for pricing at various stages of processing, and the methodology for the determination of (and the reasons behind) producer pricing, prices as reported by independent agents, negotiated prices, and formal commodity market price formation.

Producers and some consumers prefer price stability. The reasons for this and some of the methods used to attain this end are outlined. Using the tin agreement as an example, Strauss shows the pitfalls in such stabilization efforts and why they are short-lived: market forces may be resisted, they cannot be abrogated. The chapter concludes with a view of the shifting focus of trade and processing towards the new sources of mineral mining.

John Whitney and Garth Sibbold (Chapter 2.4) present an overview, not a primer, of taxation in relation to mineral mining. Both federal and state taxes are discussed, the latter being necessarily generic rather than state specific. The context is cash flow analysis as, for example, the expected return on investment as used for project evaluation and the need for an after-tax determination of net present value or the internal rate of return for proper comparison across projects. Federal taxation with respect to exploration and development expenditures, depreciation, and depletion are emphasized. The interrelations between the taxes and business decisions are discussed. Importantly, the tax alternatives available are laid out with their implications for other tax payments shown.

State taxes—property, severance, income, sales, and use—are described along with their implications for both the mining industry and the taxing authority. The former react through adjustment of exploration, development, and production plans as well as by cutoff grades and delineation of minable reserves. The latter are concerned with the amount and predictability of tax revenues.

Any investment is risky. The length of time to profitability, the size of the investment relative to firm size, its front-end loading, and the inherent uncertainties of prices, costs, and political decisions put mine investment in the high-risk class. Chapter 2.5 by Dennis Arrouet, Investment Strategy for Mining Projects, defines the process of mining investment strategy for both US and foreign projects. Risk cannot be eliminated, but it can be identified, adjusted for, and ameliorated.

Market (product) risk arises because mineral demand is derived from that for the final goods containing or using the mineral. Country risk is exacerbated by the high profile of a mining project. Project risk is a function of the capital size, technical considerations, and the time horizons: to explore, to develop or build, to recoup the investment, to profit. Risk reduction is shown to depend on new corporate strategies: low-cost leadership, product and market differentiation, condensation of the time horizon, and a market structure that does not compete away legitimate economic rents.

As profitability must be shown, the discussion proceeds with several illustrative examples to demonstrate the calculations required to determine, given the uncertainties, if the rate of return on the project is adequate. The techniques used to compensate for risk are listed.

Projects require significant capital infusions, leading to an illustrated discussion of capital costs and their relation to the capital structure of the firm over the relevant period. Given the financing method, the means of calculating the contribution of project investment's risk to the firm's financial risk is demonstrated. This leads to the discussion and calculation of the acceptable level of risk-project and financial, as related to the firm's portfolio of projects.

Throughout, mine investment decisions are shown to be a process of the organization, one with steps to follow, check points, and a corporate ethos. Even failure, through painful, is shown to provide lessons. The chapter ends with two extremely valuable check lists: (1) a consideration of the various project participants, the suppliers of good, services, and finance; and (2) an extensive project check list.

Since biblical times, government stockpiles have been viewed by producers alternately as a source of additional demand bolstering prices, especially when excess supply threatens to depress prices, and as price depressing burdens overhanging the market when excess demand creates potential price and profit opportunities. As it is both a function and the nature of a stockpile or any inventory (whether voluntary for business purposes or involuntary due to production exceeding sales) to perform in this manner, irrespective of the objective for which the storage was created, this result is not surprising. In Chapter 2.6, J. Wayne Kulig and John Babey analyze the development of the US National Defense Stockpile, probably the largest and certainly the best known of the current stockpiles.

From the producer's standpoint, stockpile procedures and objectives; purchase and disposal provisions as they apply to mineral supply, demand, and price; purity considerations; and other restrictive provisions are important to the expectations of their industry and of their own firm. The changes in these considerations, 1939–1990, are the subject of the first part of the chapter. The changes have been extensive and intensive. For those who must plan with one eye on Washington, this history bears careful reading.

The final part of the paper includes a sectional analysis of the current (1990) Act, an analysis of its operation (who is responsible for what and what are the connections), and a precis of currently planned future developments to the Act. Two useful appendices provide the provisions of the Strategic and Critical Materials Stockpiling Act and the currently included materials.

Chapter 2.7 by William Holroyd provides an overview of some of the particularities encountered in the trading of minerals. Included are the types of sourcing and contracts, material quality, and financing. The importance of transport provisions, particularly for low value, bulk material, is shown. As trade is not necessarily direct from mine to processor or to manufacturer and need not be conducted for present or future cash payment, both barter and compensation trade are discussed. Finally, the role of minerals trading companies as both agents and dealers is assessed.

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Chapter 2.1
ECONOMICS OF THE MINERAL INDUSTRIES*

JOHN E. TILTON

Mineral commodities are normally separated into three generic classes—metals, nonmetals, and energy minerals including oil and gas—and encompass a large number of different substances. Some are extracted from large open pits, others are dug out of underground mines or pumped from deep wells, and still others are processed from the sea. Extraction and processing can be relatively uncomplicated and inexpensive, though in most instances, sophisticated technology is necessary and the costs are high. Some mineral commodities are produced mainly as byproducts of other commodities. Some are recovered in large quantities from new and old scrap. Some are found in only a few locations and traded worldwide; others are produced in many different countries. Some are sold by numerous firms at fluctuating prices determined on competitive commodity exchanges, while others are produced by only a few firms and sold at stable producer prices.

Although this diversity makes the mineral industries interesting, indeed fascinating to study, it also poses problems. There is no general model or economic analysis applicable to all mineral commodities. Rather each mineral commodity must be considered individually, so that the analysis takes explicit account of its particular features.

This means that a single chapter cannot begin to cover comprehensively the economics of all mineral industries, and no attempt to do so is made here. Instead, we will concentrate on illustrating the usefulness of relatively simple economic principles, particularly those associated with supply and demand analysis, in understanding the behavior of mineral markets.

There is one particularly important conclusion that the following pages should make clear. The simple tools of economics can provide powerful insights into the operation and behavior of the mineral industries, but only if the analyst applying these tools has a firm understanding of the important technological and institutional relationships governing the market he or she is examining, and can tailor the analysis to take these relationships explicitly into account. Good analysis requires knowledge of both economics and the particular mineral commodity of interest.

2.1.1 MINERAL DEMAND

Mineral commodities rarely are final goods. Rather they are in demand because they possess certain qualities or attributes, such as strength, resistance to corrosion, or Btus, that are needed in the manufacturing of final consumer and producer goods. This means that mineral demand depends on the demand for final goods, and for this reason is often characterized as a derived demand. Since demand is really for a set of attributes, rather than for a mineral commodity per se, in many end uses one mineral produced can replace another. The importance of mineral substitution is highlighted in the discussion that follows on the major determinants of mineral demand.

*This article is a revised and updated version of another work (Tilton, 1985).

2.1.1.1 Major Determinants

Literally thousands of factors affect mineral demand-cold weather in the Northeast of the United States, perestroika in the Soviet Union, a decision by the French government to modernize part of its naval forces, a World Bank loan to Brazil to build a dam and hydroelectric power station. In analyzing mineral demand, it is not possible to take account of all possible determinants. There simply are too many. Moreover, the effects of most are so trivial they can be safely ignored, and indeed should be ignored to avoid needlessly complicating the analysis. The problem is deciding which features are of such importance they need to be considered. The answer depends not only on the minerals of interest, but also on the purpose and time horizon of the analysis. Technological change, for example, is not likely to alter greatly the demand for zinc over the next three months, but would require careful consideration for an analysis of zinc demand in the year 2020.

The choice of which factors to consider and which to ignore will to a large extent determine the quality of the analysis. In this regard, it is useful to review the determinants often considered in mineral demand studies.

Income: Mineral commodities are used in the production of consumer and producer goods. So changes in the output of these goods have a direct and immediate impact on mineral demand. In this connection, two types of changes in aggregate production or income are often distinguished: relatively short-run changes that come about largely as a result of fluctuations in the business cycle; and longer-run changes caused by secular growth and structural change in the economy.

As income is one of the most important variables affecting metal demand, its influence is almost always taken into consideration. In many studies, gross domestic product (GDP) or industrial output is employed for this purpose. More disaggregated measures of income are also used. For example, in assessing the demand for copper wire, we might use the production of electrical and electronic equipment to capture the effect of income fluctuations.

Own Price: A mineral commodity's own price is also normally an important determinant of demand. Demand tends to fall with an increase in price and rise with a decline in price. There are two reasons for this inverse relationship. First, a higher mineral price increases the production costs of the final goods in which it is used. If these costs are passed on to the consumer, demand for the final goods will fall. Second, and usually of greater consequence, firms may respond to the higher price by substituting another mineral commodity whose price has not risen. It is important, however, to note that mineral substitution takes time. New equipment is often necessary, personnel may have to be recruited or retrained, and production techniques must frequently be altered. Consequently, the initial effect on mineral demand of a change in a price may be quite modest, and a number of years may be needed before the full effect is realized.

Prices of Substitutes and Complements: The demand for a mineral commodity may be affected by prices other than its own. Most compete with other mineral commodities for their end-use markets, and so a change in the price of any such substitute can affect the demand for the others. The sharp rise in the world price of oil during the 1970s, for example, increased the demand for coal and natural gas.

In some instances, a fall in the price of one mineral commodity may actually increase demand for another. In such cases, we say the two are complements. For example, a fall in the price of steel tends to increase the use of tin plate, since tin plate is composed primarily of steel. This, in turn, stimulates the demand for tin.

As with changes in own price, changes in the prices of substitutes and complements affect mineral demand primarily by inducing firms to alter the nature of their production processes. Consequently, some time is required for the full impact of price changes on demand to be realized.

Technological Change: New technology can alter demand in several ways. First, it can reduce the amount of a mineral commodity required in the production of specific items. For example, new alloys and other technological developments have significantly reduced the amount of aluminum in a beer can.

Second, new technology can affect the ability of a mineral commodity to compete in particular end-use markets. This is nicely illustrated by the water-pipe market for home construction, where innovations in the production of polyvinyl chloride (PVC) plastic pipe have allowed this material to capture a sizable market share over the last 25 years. In the process, the demand for copper and other traditional pipe materials has suffered.

Finally, new technology can change the number and size of end-use markets. The advent of the automobile, for instance, gave rise to a major new market for petroleum, steel, and lead.

Since measuring technological change is difficult, some studies ignore this particular determinant. This may not be serious when assessing demand over a very short period, but over the longer term, it is much harder to justify its omission. Other studies simply assume that technological change is closely correlated with time. This allows the use of a time trend to capture the effects of technological change. While such a procedure may be acceptable in some situations, in most the influence of technological change is too random and discrete. The tremendous impact that electrolytic tinning had on the demand for tin was basically a once-and-for-all event. The effects of such major innovations are not likely to be closely correlated with time and should be explicitly and individually taken into consideration.

Consumer Preferences: Changes in consumer preferences affect the end-use markets in which metals are consumed. When preferences shift toward small cars, for instance, the demand for steel, copper, aluminum, and other metals by the motor vehicle industry tends to fall.

Consumer preferences vary over time and among countries for various reasons. The age distribution of the population, for example, can be important. Between the ages of 18 and 35, many individuals are engaged in setting up new family units and spend a relatively large proportion of their income on housing, automobiles, refrigerators, and other consumer durables.

Per capita income and the overall level of economic development also influence consumer preferences. The poor have to spend their limited incomes almost entirely on basic necessities, while the rich can indulge in more luxuries. The rich also tend to save a larger portion of their income. So a shift of income in favor of the poor is likely to reduce the amount of total income saved and invested. Since investment stimulates the construction, capital equipment, and other material-intensive sectors of the economy, such a redistribution may reduce mineral demand.

New technology, by making new and better products available, also causes shifts in consumer preferences. The rapid growth of the airline industry over the last 50 years has substantially increased the use of aluminum and titanium in this market, while reducing the consumption of steel in railroad passenger cars and ocean liners.

Finally, consumer preferences can change simply in response to shifts in personal tastes. In some instances, these shifts are influenced by advertising and psychological considerations that are not fully understood.

Government Activities: Government policies, regulations, and actions constitute another major determinant of metal demand (Chapter 3.2). This is perhaps most dramatically and starkly illustrated when government policies lead to war.

In peacetime, changes in government expenditures on education, defense, research and development, highways, and other public goods alter the output mix of the economy. Fiscal, monetary, and social welfare policies affect income distribution and the overall level of investment and economic growth. Worker health and safety legislation, environmental standards, and other governmental regulations may proscribe certain minerals in particular end uses.

2.1.1.2 The Demand Function

The relation between the demand for a mineral commodity and its major determinants, such as those we have just discussed, is given by the demand function. This economic relationship is often expressed mathematically. In some analyses, for example, demand during year t (Q_t^d) is assumed to depend on only three variables—income during year t (Y_t), own price during year t (P_t^o), and the price of its principal substitute during year t (P_t^s):

$$Q_t^d = f(Y_t, P_t^o, P_t^s) \tag{2.1.1}$$

Several things about Eq. 2.1.1 are worth noting. First, it recognizes only three variables affecting demand. In most situations, as we have seen, there are other important determinants that belong in the demand function.

Second, this equation considers only the immediate- or short-run effects on demand of changes in its determinants. For the income variable, this is not a serious shortcoming, since mineral demand tends to respond rather quickly to changes in the overall level of economic activity. This is not the case for prices. Producers take time to substitute one material for another and in other ways to respond fully to a change in a commodity's own price or that of its principal substitutes. This means that demand this year depends not only on prices this year, but also on prices a year ago, two years ago, and so on for as far back as past prices affect current demand.

Third, while Eq. 2.1.1 identifies important variables presumed to influence demand, it does not specify the nature of the relationship. Normally, a rather simple specification between demand and its determinants is assumed. A linear or log linear relationship, similar to those shown in Eqs. 2.1.2 and 2.1.3, are particularly popular, primarily because they are relatively simple and easy to estimate:

$$Q_t^d = a_0 + a_1 Y_t + a_2 P_t^o + a_3 P_t^s \tag{2.1.2}$$

$$\log Q_t^d = b_0 + b_1 \log Y_t + b_2 \log P_t^o + b_3 \log P_t^s \tag{2.1.3}$$

Unfortunately, such specifications entail rather strong assumptions about the nature of the demand function, whose validity is often difficult to assess.

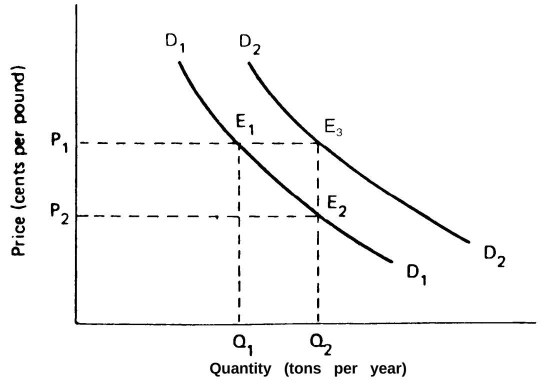

Fig. 2.1.1. Movements along and shifts in the demand curve.

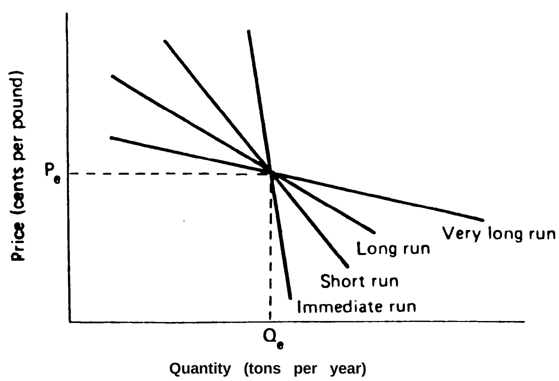

Fig. 2.1.2. Demand curve in the immediate, short, long, and very long run.

2.1.1.3 The Demand Curve

In analyzing mineral markets, we at times focus on one particular variable and try to assess how it alone affects demand. For example, if the US economy is expected to grow by 4% over the coming year, aluminum firms need to know how this will alter their demand.

Another variable of special interest is price, particularly a commodity's own price. The demand curve shows how much of a commodity can be sold at various prices over a year or some other time interval, on the assumption that income, the prices of substitutes, and other determinants of demand remain fixed at certain designated levels.

Normally, demand curves are drawn with a downward slope, like those shown in Figs. 2.1.1 and 2.1.2. Intuitively, one would expect demand to fall as price rises. Still, there can be exceptions. In special circumstances, the demand curve, at least over a significant segment, can be vertical (implying that consumers want a particular amount of the commodity, no more and no less, regardless of its price); horizontal (implying that above a particular price consumers demand none of the commodity while below that price their demand is insatiable); and upward sloping (implying that consumers actually increase their demand as price goes up). Such situations are rare, but when they do occur they are likely to be of considerable interest and importance.

Several other characteristics of the demand curve should also be noted:

1. A movement along the curve reflects the effect of a change in a commodity's own price. A change in any of the other variables influencing demand causes a shift in the curve itself. In Fig. 2.1.1, for example, demand can increase from Q_1 to Q_2 because price falls from P_1 to P_2, causing a movement along the curve DD_1 from point E_1 to E_2. Or the same increase can occur because the demand curve shifts from DD_1 to DD_2, causing the equilibrium point to move from E_1 to E_3. Such a shift in the demand curve can occur in response to a rise in income, a new technological development, an increase in the price of a substitute commodity, or a change in one or more of the other demand variables. In using demand curves, it is important to keep clear the distinction between a movement along the curve and a shift in the curve.

2. The same commodity may have many different demand curves. At the most aggregate level is the total demand curve, which indicates how much all buyers are willing to buy at various price levels. On the buyers' side of the market, we can define demand curves for individual buyers, for regional or national markets, for a country's imports, and for particular consuming sectors or industries. We can distinguish between the demand curve for consumers and the demand curve for speculators and hoarders. On the seller's side of the market, a similar breakdown is possible. So for refined copper, we can identify a demand curve for the world as a whole, for the United States, for the telecommunication sector, for AT&T, for speculative stocks, for US imports, for US exports, for US producers, for Cyprus Minerals Co., and so on.

3. The demand curve—and the demand function as well—indicate the demand for a commodity and not its consumption or production (even though the horizontal axis on the demand curve is sometimes identified as output). *Demand* is the quantity of a commodity that can be sold at a particular price in a given market over a year or some other time period. If the US government is selling tin from its strategic stockpile, or if speculators or other private stockholders are drawing down their inventories, production may be considerably below demand. Even in the case of a demand curve specifically for producers, production will be less than demand when producers are liquidating their inventories and more than demand when they are building up inventories. Similarly, consumption will be above demand when consumers and other buyers are decreasing their stocks and below demand when they are increasing their stocks.

Over a number of years, the differences between consumption, production, and demand are small and can safely be ignored. This is because inventory changes over, say, 10 years will largely cancel out, and any remaining differences will be small compared to cumulative demand over such a period. Usually, however, demand curves indicate how much of a commodity is needed over a year or shorter period, and so changes in stocks can cause sizable discrepancies among production, consumption, and demand.

4. The demand curve does not indicate how the effect of a price change varies with respect to time. Rather, it assumes, explicitly or implicitly, one specific adjustment period. In this connection, economists typically distinguish between (1) the *short run,* a period sufficient for firms to adjust output by altering their labor, raw material, and other variable inputs, and (2) the *long run,* a period long enough for firms to vary their fixed inputs, such as plant and equipment, as well as variable inputs.

In examining mineral markets, it is useful at times to consider what we will call the *very long run,* which provides time not only for all inputs to change, but also for the development and introduction of any new technology induced by price changes. At the other end of the spectrum, the *immediate run* is needed in addressing certain mineral issues. It provides so little time for adjustment that firms find it infeasible to alter their output. Only changes in inventories are possible. As illustrated in Fig. 2.1.2, the responsiveness of demand to price increases with the adjustment period.

Just how long the immediate, short, long, and very long runs are in practice is complicated by the fact that no one answer is valid for all situations. The lag between an infusion of variable inputs and increased output depends on the manufacturing process and may even vary over time for the same process. Similarly, new capacity can be built more quickly in some industries than others. Normally, we would not expect the immediate run to last for more than several months, and the short run for more than about several years. The shift from the long to very long run is more difficult to pin down. Some of the new technology induced by a price change may occur quickly, indeed within a year or two, but other developments may take decades.

The time dimension introduced by the immediate, short, long, and very long runs should not be confused with the time interval over which demand is measured. All of the curves drawn in Fig. 2.1.2 presume that demand is in tons per year. The very-long-run demand curve does not indicate how much of a commodity will be demanded over a very long period, for example, over the next 20 years. Rather, it indicates how much will be demanded per year in 20 years time as a consequence of a price change today, assuming prices stay at the new level and all other determinants of demand also remain unchanged.

Since neither of these conditions will hold for 20 years, it is best not to think of the very-long-run demand curve as showing annual demand 20 years from now. What it shows is the new equilibrium towards which demand is moving over the very long run in response to the price change. Long before this equilibrium is actually reached, price and other determinants will change again, causing the trend in demand to shift course and follow a new path towards a different equilibrium.

5. The downward-sloping demand curve, as commonly drawn, implies that the relationship between price and demand is continuous and reversible. Continuity means that the demand curve is smooth, like those drawn in Figs. 2.1.1 and 2.1.2, without any kinks or breaks. Reversibility means that if price, after an upward or downward movement, returns to its original level, demand will also return to its original level. In other words, one can move up and back down the same curve in response to price changes without causing the curve itself to shift.

For the immediate and short-run demand curves, reversibility seems reasonable. If the desired level of stocks that consumers and others wish to hold declines by a certain amount as price rises, the reverse is likely when price eventually falls. Mineral substitution that can occur in the short run by its nature involves changes that can be made quickly with minimal costs and disruption. After such a switch, it should be relatively easy to switch back to the original mineral commodity.

Reversibility is less likely with the long-run demand curve. Here mineral substitution will entail new equipment, lost production, and other conversion expenses. As a result, a firm will not switch back to a mineral product until its price falls considerably below the level at which its replacement became attractive. In the very long run, the assumption of reversibility is even more doubtful, for now price-induced innovations may substantially change the underlying technical and economic conditions governing the demand for a mineral commodity.

The assumption of continuity may also not hold, particularly for those commodities whose consumption is concentrated in a few major end uses. Over a wide range, price may rise with little or no effect on demand. Then at a particular threshold, an alternative becomes more cost effective in a major application, causing demand to drop sharply. Such discrete jumps or breaks may be found in both short- and long-run demand curves. They are particularly likely to characterize the very-long-run demand curve, as price induced innovations by their nature are discrete events. When they do occur, they can have a substantial impact on demand.

2.1.1.4 Demand Elasticities

In addressing many mineral issues, we need to know how sensitive demand is to a change in price. The measure economists use for this purpose is the elasticity of demand with respect to own price, or simply the *price elasticity of demand.* As Eq. 2.1.4 indicates, it is defined as the negative of the partial derivative of demand with respect to own price times the ratio of own price to demand. Since an increase in price normally produces a decrease in demand, this derivative is itself negative, making the price elasticity a positive number.

$$E_{Q_t^d,\ P_t^o} = -\frac{\partial Q_t^d}{\partial P_t^o} \cdot \frac{P_t^o}{Q_t^d} \tag{2.1.4}$$

$$= -\frac{\text{percent change in } Q_t^d}{\text{percent change in } P_t^o} \tag{2.1.4a}$$

For many, it is easier to remember the price elasticity of demand as the percentage increase in demand resulting from a 1% reduction in price, as shown in Eq. 2.1.4a. If the increase in demand is greater than l%, the elasticity is also greater than one, and we say that demand is elastic. When the elasticity is less than one, demand is inelastic.

Since the derivative of demand with respect to price is equal to the inverse of the slope of the demand curve, where two curves cross, the elasticity of demand will be lower for the curve with the steeper slope. This means that demand at the point where the curves intersect in Fig. 2.1.2 is most elastic in the very long run, and becomes increasingly less elastic in the long, short, and immediate runs. This, of course, is exactly what we would expect, for consumers have more opportunities to increase or decrease the usage of a material in response to a price change the longer the period they have to adjust.

If the relationship between demand and price is linear, as is assumed in Fig. 2.1.2 (and earlier in Eq. 2.1.2), the slope of the demand curve is the same at all points. This means that the price elasticity of demand decreases as one moves down the demand curve, and the ratio of price to demand falls. Consequently, other than at their intersection point, we must be careful in comparing the demand elasticities of two curves. The steeper curve will not necessarily have the lower elasticity everywhere.

At times, the relationship between demand and price is assumed to be linear in the logarithms, as in Eq. 2.1.3. In this case, the elasticity does not vary with the level of price and demand; a 1% decrease in price produces the same percentage change in demand over the entire demand curve. The latter, if drawn using logarithmic scales, is a straight line whose slope alone determines the price elasticity of demand. So one can easily compare the elasticities of two curves, even at points where they do not intersect. These properties make the logarithmic relationship popular in analyzing material demand. However, as stressed earlier, its use is appropriate only if there are good reasons to believe it reflects the true relationship between demand and its determinants.

Up to this point, we have considered only the price elasticity of demand. It is possible to define a separate elasticity for every variable affecting demand, though in practice we normally encounter only two others—the elasticity of demand with respect to the price of substitutes and the elasticity of demand with respect to income.

The elasticity of demand with respect to the price of a substitute, often called the cross (price) elasticity of demand, measures the percentage increase in demand for a material caused by a 1% increase in the price of a substitute. It too will be larger in the long and very long run than in the immediate and short run, since the opportunities to respond to a change in a substitute's price grow with the adjustment period.

The income elasticity of demand similarly measures the percentage increase in demand caused by a 1% rise in the GDP or some other measure of income. Since the demand for final goods, and in turn the demand for the raw materials needed to produce these goods, responds fully to a change in income rather quickly, the income elasticity does not increase with the adjustment period. There is consequently no need to distinguish among the immediate, short, long, and very long runs, as is the case for own and cross price elasticities.

Another distinction, however, is significant. Earlier, we noted that a change in income can be separated into two parts: a cyclical component caused by short-term fluctuations in the business cycle, and a secular component caused by long-term growth trends. Which of these is primarily responsible for an income change will affect the magnitude of the demand response and the size of the income elasticity.

The demand for many mineral commodities is particularly responsive to income changes caused by business cycle fluctuations. Metals and other materials, in particular, are consumed primarily in the capital equipment, construction, transportation, and consumer durable sectors of the economy, which use them to produce automobiles, refrigerators, homes and office buildings, new machinery, and other such items. These sectors boom when the economy is doing well, and they suffer severely when it falters. Since small fluctuations in the business cycle cause major change in their output and in turn the demand for materials, the income elasticity is normally greater than one when the business cycle is responsible for changes in income.

When income changes are the result of secular growth trends, the traditional and still very common presumption is that mineral demand grows or declines in direct proportion with income. The income elasticity of demand in such situations is thus one.

In recent years, this assumption has come under attack, in part because energy and metal consumption has not kept pace with income growth in many countries. As income grows, the desired mix of final goods may change, affecting mineral usage and causing the income elasticity of demand to deviate from unity. Indeed, Malenbaum (1975, 1978) and others writing over the last two decades on the intensity of material use provide a rationale for expecting just such a shift.

They contend that countries in early stages of economic development with low per capita incomes are largely agrarian. Their intensity of material use, defined as the amount of material consumed per unit of GDP, is quite low. As such countries begin to industrialize, they invest in basic industry, infrastructure, and other material intensive projects, which cause their intensity of use to rise. As development proceeds, the demand for factories, water and sewer systems, roads, housing, schools, and automobiles is gradually satisfied, and the composition of final production shifts away from manufacturing and construction and toward services. For this and other reasons, they believe the relationship between the intensity of material use and per capita income follows an inverted U-shaped curve similar to that shown in Fig. 2.1.3 for steel use in the United States. This implies that the income elasticity of demand is greater than one for developing countries operating on the rising portion of the intensity of use curve, and less than one for developed countries on the declining portion.[a]

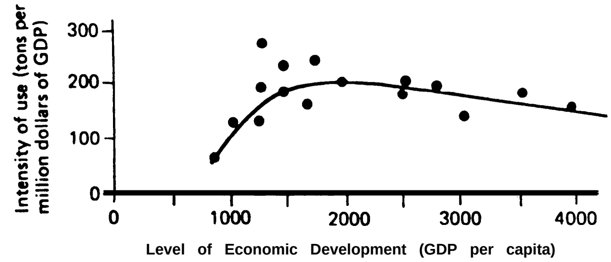

Fig. 2.1.3. Relationship between intensity of steel use and per capita income in the United States, 1888-1967 (Anon., 1974, p. 58). GDP is measured in constant (1963) dollars. Points shown in the figure are five year averages, through which a free hand curve has been drawn.

Canavan (1983), Radcliffe et al. (198l), Roberts (1985), and others have raised some serious questions regarding the intensity of use hypothesis, though most of the criticisms concern its use for forecasting. The evolution it anticipates as a country develops and per capita income rises in the importance of mineral intensive goods in overall GDP, while far from proven, certainly seems plausible.

In summary, the income elasticity of demand for mineral commodities depends on several considerations. When the business cycle produces a change in income, for metals and other mineral materials, the elasticity will normally be greater than unity. When secular growth causes change in income, the elasticity is likely to be greater than unity only if growth is concentrated in developing countries on the upward sloping portion of their intensity of use curve. It will be less than unity if growth is concentrated in developed countries on the downward sloping portion of their intensity of use curves.

Since the full impact of a change in income is quickly transmitted to the demand for mineral commodities, the income elasticity of demand is the same in the immediate, short, long, and very long run. This is not the case, however, for the elasticity of demand with respect to own price or the price of substitutes. With price changes, the longer the adjustment period, the greater the demand response. Consequently, these elasticities are often less than one in the immediate and short run and greater than one in the long and very long run.

2.1.2 METAL SUPPLY

Many mineral commodities, such as oil, coal, bauxite, and iron, are typically extracted as single or individual products. Others are produced as joint products. For instance, molybdenum and gold are often found in porphyry copper deposits, and nickel sulfide mines may produce copper as well.

[a]This assumes that part of any rise in income is accompanied by an increase in per capita income. It is possible, though unlikely, for income to increase solely as a result of population growth with per capita income remaining stagnant. In this case, there is no movement along the intensity of use curve and no change in the ratio of material usage to GDP. A 1% increase in income causes a 1% increase in material consumption, and the income elasticity of demand is one.

Where joint production occurs, main products, coproducts, or byproducts may be recovered. A main product is so important to the economic viability of a mine that its price alone determines the mine's output. A byproduct on the other hand is so unimportant its price has no influence on mine output. When prices of two or more joint products affect output, they are coproducts.

Once processed and consumed in the production of final goods, some mineral commodities, specifically metals, are often recovered and reused. Most of the gold ever mined, for example, is still in use. Recycling is called secondary production, not because recycled or secondary materials are inferior, but because the scrap from which they are made is not the original or primary source of the product.

This segment examines mineral supply, looking first at the primary production of individual products. It then considers how the recovery of byproducts and coproducts and secondary production supplement supply.

2.1.2.1 Individual Products

In examining mineral supply, we again want to concentrate on the few most important variables. Just which variables are worthy of consideration varies with the commodity, the source of supply, the time of adjustment, and other factors, and calls for considerable judgment on the part of the analyst. While no single list is appropriate for all situations, the following variables are often important.

Own Price: Firms have an incentive to increase their output up to the point where the costs of producing an additional unit just equals the extra revenue they receive from selling that unit. Consequently, a rise in the price of a mineral commodity normally increases its supply, while a fall in price reduces its supply.

In the short run, however, the response of supply to a price change may be constrained by existing capacity. It takes time to develop new mines, drill new wells, and build processing capacity, and so producers may need 5 to 7 years to respond fully to a price increase. An even longer time may be required to adjust fully to a price decrease. Extraction and mineral processing are capital intensive activities, requiring equipment and facilities with long productive lives. Firms will remain in production, despite a fall in price below average costs, as long as they are recovering their variable or out-of-pocket costs. Only when existing plants and equipment need to be replaced will they cease production.

Input Costs: The costs of labor and other inputs used in extraction and processing also affect profitability and, in turn, supply. For example, the rise in world oil prices during the 1970s sharply increased the costs of producing aluminum in Japan. Again the long lags in adjusting capacity to new conditions mean that the full effect of a change in costs on supply may take a number of years.

Technological Change: Advances in technology that reduce the costs of extraction or processing also affect mineral supply. For example, the mining of copper from large open pit porphyry deposits became feasible in the early years of the 20th century as a result of the introduction of the flotation process for concentrating such low-grade ores. Advances in excavating capability—more powerful blasting techniques, bigger trucks, and stronger shovels—have since helped keep low-grade deposits economic at constant or even lower real prices.

Strikes and Other Disruptions: Industry-wide strikes have in the past closed down the US copper industry and the Canadian nickel industry for months. Inadequate precipitation in the Pacific Northwest has curtailed hydroelectric power generation, and in turn aluminum production, in that region. Rebel invasions into the Shaba Province of Zaire, the world's largest producer of cobalt, have on several occasions disrupted world supplies of this important metal. Strikes, mine accidents, natural disasters, civil disturbances, and other such disruptions can affect the supply of a metal by interrupting either its production or transportation.

Government Activities: Government actions influence metal supply in a variety of ways. Environmental regulations and state-imposed severance taxes tend to increase costs and reduce supply. Abroad some countries require that mining companies purchase supplies from domestic producers, process ores and concentrates domestically, and employ nationals for managerial and technical positions, even though these restrictions may reduce efficiency and increase costs. Alternatively, governments may stimulate supply by subsidizing new mines or processing facilities.

Market Structure: Where a few firms or countries account for most of a mineral's production, they may maintain a producer price or, in the case of oil, a cartel price. As discussed later, this alters the nature of supply.

In addition, over the last 30 years, the number and importance of state-owned mining companies have grown. In their production and marketing decisions, such firms may be less concerned about earning profits than maintaining employment, foreign exchange earnings, and other public goals. If so, their market supply is likely to respond less to price signals, particularly to low prices during market recessions.

The relation between the supply of a metal and its principal determinants, such as those just discussed, is given by the supply function. Normally, it is expressed mathematically. Equation 2.1.5, for example, is the function for a metal whose supply Q_t^s depends on its price P_t^o, the wage rate paid by producers W_t, the cost of energy E_t, and strikes S_t.

$$Q_t^s = g\,(P_t^o, W_t, E_t, S_t) \tag{2.1.5}$$

This is a rather simple supply function. It does not consider technological change and certain other variables that often affect supply. It contains no lagged values of the price or cost variables, and hence takes account only of their short-run influence. Finally, its exact specification is not indicated.

The relationship between a commodity's price and its supply is often of special interest. It is portrayed by the supply curve, which shows how much producers will offer to the marketplace at various prices over a year or some other time period, on the assumption that all other variables affecting supply remain at some specified level.

The supply curve is normally drawn sloping upward, indicating that supply increases with price. This positive relationship seems plausible for reasons already mentioned, though it can be derived in microeconomics from the theory of the firm.

In special circumstances, however, the curve can, over the relevant portions, be horizontal (implying that sellers are willing to provide the market with as much as they have to offer at a particular price and with nothing below that price), vertical (implying that sellers will provide to the market a given quantity, no more and no less, regardless of the price), or downward sloping (implying that sellers will offer more to the market the lower the price).

Such behavior can occur for various reasons. Firms may maintain a producer price at which they are prepared to sell all of their available supplies. In other instances, a change in price may not alter the output because firms are already operating at full capacity. Some firms, for instance, state enterprises, may continue to produce at or near capacity, even though it would be more profitable to reduce output in order to avoid laying off employees. Some may even attempt to increase production when price falls if they feel responsible for maintaining their country's foreign exchange earnings.

While such situations do occur, they are unusual. Normally, the supply curve is upward sloping, and in this important respect differs from the demand curve. Other characteristics of the supply curve, however, are the same or similar to those discussed for the demand curve.

For example, a movement along the supply curve reflects a change in price, while a shift in the curve itself reflects a change in one of the other determinants of supply. There are also many different supply curves. For refined copper, for example, separate curves are possible for the supply of all producing firms, for the supply of US producing firms, for the supply of Cyprus Minerals Co., for the supply of US exports, for the supply of US imports, for the supply from US government stockpiles, and so on.

Supply reflects how much sellers are willing to offer in the marketplace, and so like demand should not be confused with consumption or production. Where the supply and demand curves intersect, the quantity desired by buyers and the quantity offered by sellers are equal, and at that price the market clears.

As normally drawn, the supply curve assumes that the relationship between price and supply is continuous and reversible. In practice, neither condition always holds. Some mines and smelters operate on a very large scale. When they begin or stop production, supply experiences a discrete jump. Similarly, when price goes up, it may induce higher wages, shifting the supply curve to the left, which makes it impossible to move back down the original curve. Alternatively, higher prices may stimulate new technology and lower costs, causing the supply curve to shift to the right. In this situation, should price return to its initial level, supply would be greater, not less, than originally.

The supply curve also assumes that metal producers have a certain amount of time to adjust to changes in price, input costs, and other determinants. Here, as with demand, it is useful to distinguish four adjustment periods and in turn four types of supply curves: the immediate, short, long, and very long run.

In the *immediate run,* firms do not have time to alter their rate of production. Consequently, supply cannot exceed current output plus available producer inventories or stocks. If demand is weak, they can build up inventories for sale at a later time. So the immediate-run supply curve is not everywhere vertical or nearly vertical, as one might first think.

Before assessing the general shape of the immediate-run supply curve, we need to distinguish two types of metal markets, producer markets and competitive markets, for the supply curve is different for each. Firms in producer markets quote the price at which they are prepared to sell their product. These markets, normally characterized by a few major sellers, have relatively stable prices, though when demand is weak, actual prices may fall below quoted producer prices as a result of discounting and other concessions. Steel, aluminum, and magnesium are a few of the metals sold in producer markets.

In competitive markets, price is determined by the interplay of supply and demand and is free to fluctuate as much as necessary to clear the marketplace. Many buyers and sellers are typically active in competitive markets, and price is often set on a commodity exchange, such as the London Metal Exchange (LME) or the New York Commodity Exchange (Comex). Tungsten, manganese, and silver are metals sold in competitive markets.

Producers are price takers in competitive markets and have no influence over the going market price. Nevertheless, they still control their own supply.

An immediate-run supply curve for producers selling in a competitive market is shown in Fig. 2.1.4a. At very low prices, it

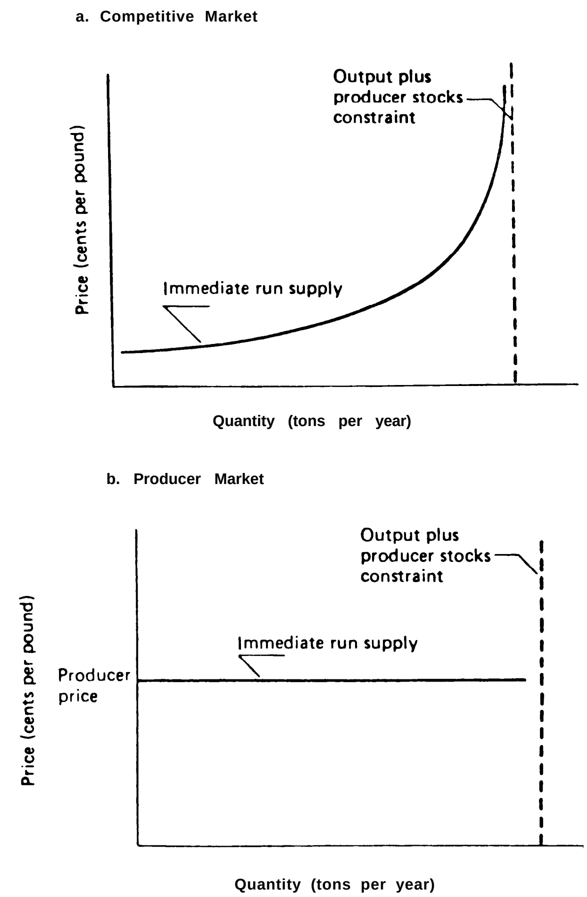

Fig. 2.1.4. Supply curves in the immediate run.

indicates that no supply is forthcoming as production is withheld from the market in anticipation of higher prices in the future. At some point as price rises, supply begins to come onto the market. Once this threshold is reached, supply at first expands greatly in response to higher prices as more and more current production is offered for sale. Eventually, however, further increases in supply must come from inventories. Since producers will deplete their stocks only at high prices, and since stocks can normally augment supply by only modest amounts compared to current output, the supply curve shown in Fig. 2.1.4a becomes quite steep at high prices, and finally vertical. As the curve approaches the constraint imposed by current production plus producer stocks, firms are unwilling or unable to add further to supply by drawing down inventories.

An immediate-run supply curve for a producer market is illustrated in Fig. 2.1.4b. It is simply a horizontal line at the producer price that extends from zero to an amount equal to current production plus the stocks producers are willing to sell. Since firms cannot increase their output in the immediate run, current production plus available producer stocks impose a constraint on supply. Figure 2.1.4b shows the curve stopping slightly before this barrier, as producers usually are not willing to sell all their available stocks.

The supply curve in Fig. 2.1.4b assumes that firms faithfully adhere to the producer price. If this is not the case, if some or all firms discount the producer price at times of weak demand, the curve is not perfectly horizontal, but instead drops somewhat at lower quantities reflecting these price concessions.

The curves shown in Fig. 2.1.4 indicate that, in both competitive and producer markets, metal supply is quite responsive to price until supply approaches current output plus producer stocks. At this point higher prices attract little or no additional supply into the market.

To assess the responsiveness of supply to price, economists use the elasticity of supply, defined as the partial derivative of supply with respect to price times the ratio of price to supply, as shown in Eq. 2.1.6.

$$E_{Q_t^s,\ P_t^o} = \frac{\partial Q_t^s}{\partial P_t^o} \cdot \frac{P_t^o}{Q_t^s} \tag{2.1.6}$$

$$= \frac{\text{percent change in } Q_t^s}{\text{percent change in } P_t^o} \tag{2.1.6a}$$

This measure reflects the percentage increase in supply produced by a 1% rise in price.

We say that supply is elastic when the elasticity is greater than one, and inelastic when it is less than one. Where the supply curve is vertical (as on the right side of Fig. 2.1.4a), or where it simply ends (as on the right side of Fig. 2.1.4b), supply is completely unresponsive to price and the supply elasticity is zero. Where the supply is flat or horizontal, supply is highly or infinitely responsive to price and the elasticity is very large.

In the *short run,* producers have time to change output but not capacity. So the supply curve is constrained by existing capacity rather than current production. If the industry is not operating at full capacity, this means the supply curves shown in Fig. 2.1.4 for both producer and competitive markets are extended in the short run by the amount of available idle capacity.

In the *long run,* new mines can be developed, wells drilled, and processing facilities built. Firms can also expand the capacity of existing operations. Consequently, the relatively flat or elastic portions of the supply curve encompass far more output than in the immediate or short runs. Only after all known deposits or fields are in operation does the supply curve stop or become vertical.

In the *very long run,* even the constraint imposed by existing deposits no longer holds, as firms have the time to conduct exploration and find new reserves. New technology may also permit the exploitation of new types of deposits.

So in the very long run, no barrier or constraint forces the supply curve to terminate or become vertical.[b] Consequently, the very-long-run supply curve, in contrast to the other supply curves we have considered, may be relatively flat or elastic over its entirety with no terminal point. In fact, for a number of mineral commodities sold in competitive markets, it may become more elastic at higher prices and quantities, because more costly sources of supply or deposit types are found in greater numbers and contain on average more metal. Large porphyry copper deposits containing about 0.4% copper, for example, are fairly abundant, and many have been found over the last 30 years. Should the price of copper reach the level needed to make such deposits attractive, supply would expand greatly.

The major differences just discussed among the four time-related supply curves are highlighted in Fig. 2.1.5a for competitive markets and in Fig. 2.1.5b for producer markets. It is particularly important to notice how the supply curves in both types of markets expand over increasingly greater quantities as the time permitted for adjustment increases, allowing first the output constraint, then the capacity constraint, and finally the known deposits constraint to be overcome. So in the very long run, no constraint on supply may exist, and supply may be elastic over the entire range of plausible outputs.

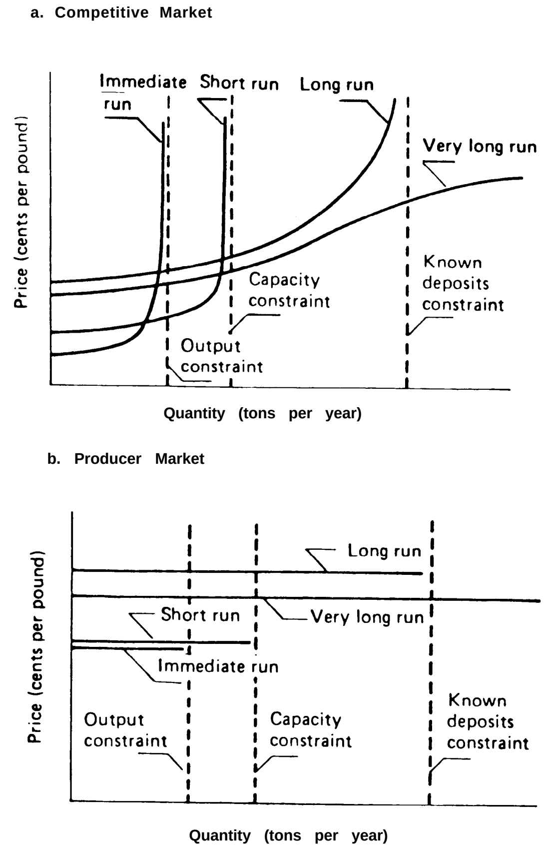

Fig. 2.1.5. Supply curves in the immediate, short, long, and very long run.

The relative vertical positions of the curves shown in Fig. 2.1.5 are based on certain presumptions about the influence of production costs on supply. In the case of producer markets, for example, we assume the producer price (which determines the height of the supply curve) is set on the basis of average total production costs at a standard or representative level of capacity utilization. These costs are likely to be similar in the immediate and short run, since capacity is fixed. In the long run, as new and presumably higher cost deposits are brought into production, average total costs will probably rise. In the very long run, though, new discoveries and the exploitation of new types of deposits should keep average total costs below those of the long run. Consequently, in Fig. 2.1.5b, the immediate- and short-run supply curves are drawn at about the same height, the very-long-run curve somewhat higher, and the long-run curve even higher.

Unfortunately, we still have much to learn about how producer prices are set and the extent to which they are actually tied to production costs. While average total costs may often be the major determinant, there are presumably instances when this is not so and when other factors are important. So the relative

[b]It is true that the earth is finite, and so the amount of oil, copper, and other minerals it contains is fixed. But the quantity of every mineral commodity found in the earth's crust compared to the amount supplied annually (which is what the supply curve relates to price) is so enormous that this ultimate constraint is not relevant here.

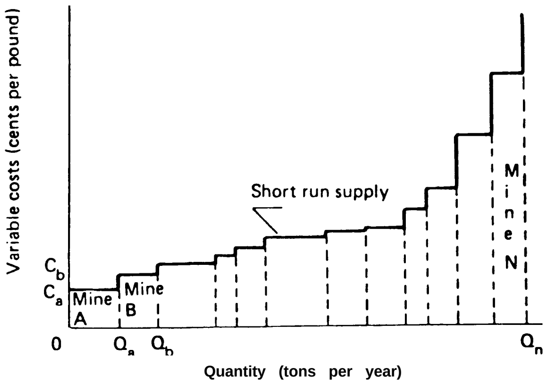

Fig. 2.1.6. Short-run supply curve derived from capacity and variable cost data.

heights of the curves in Fig. 2.1.5b should be considered simply as plausible and illustrative.

For competitive markets, the immediate-run supply curve is drawn below the short-run curve in Fig. 2.1.5a until output approaches the output constraint. Firms have two options available in the short run for reducing their supply—they can cut production in addition to building up inventories—and so are more likely to supply less at any particular price when demand is low.

The short-run curve, in turn, lies below the long-run curve until output approaches the capacity constraint. This is because firms in the short run, in contrast to the long run, have an incentive to continue production when price is below their average total costs so long as they are recovering out-of-pocket or average variable costs.

The long-run curve in Fig. 2.1.5a is also shown above the very long-run curve, for in the very long run the discovery of low-cost deposits and the development of new deposit types should reduce costs and in turn the price required to elicit any level of supply.

Economists have much more to say about the relationship between prices and costs in competitive markets than in producer markets. In the short run, according to microeconomic theory, competitive firms will have, as pointed out earlier, an incentive to produce and supply a commodity as long as price covers average variable costs. Since variable costs do not include the costs of capital and other fixed inputs, in the short run firms may remain in production even though they are losing money. This is because fixed costs must be paid whether a firm produces or not, and so losses are minimized by staying in the business as long as price is above variable costs.

This suggests that one might estimate the short-run supply curve for a metal sold in a competitive market by determining the average variable costs and capacity associated with each operating mine. This information can then be arranged as in Fig. 2.1.6, so that mine A with the lowest average variable costs *(OC_a)* and capacity *(OQ_a)* is shown first, mine B with the second lowest variable costs *(OC_b)* and capacity *(Q_aQ_b)* next, and so on. The resulting curve under certain conditions traces out a short-run supply curve. It approximates the short-run marginal costs curve for the industry, and shows how costs increase as the industry expands output by bringing back into operation mines with increasingly higher production costs.

While this procedure is used by mineral firms and others (Anon., 1987; Torries, 1988) and can provide useful insights into the nature of the short-run supply curve, it is predicated on several assumptions. All producers must have similar shutdown and start-up costs and share similar views regarding future price movements. Otherwise, some mines may remain operating while lower-cost competitors shut down because they have higher shutdown and start-up costs, or because their managers anticipate a rapid recovery of prices. In addition, governments must not provide subsidies or other public assistance when prices decline to keep marginal mines from closing. And, all producers must be primarily interested in maximizing profits. In practice, of course, these conditions may not be met. State-owned enterprises in foreign countries, for example, may remain in production even when the market price drops below their variable costs because employment and foreign exchange earnings are more important than profits. When this is the case, the actual short-run supply curve lies below that shown in Fig. 2.1.6.

Long-run supply curves for metals sold in competitive markets can also be derived using a similar procedure. The US Bureau of Mines (Anon., 1987), Torries (1988), and others have estimated the average total costs, including a competitive rate of return on invested capital, associated with all known deposits and their annual production capacity for copper and other metals. With this information, the industry's long-run supply curve is approximated in a manner similar to that illustrated in Fig. 2.1.6. Though, of course, the long-run supply curve is based on average total costs, not average variable costs, and must take account of all known deposits, including those that have not yet been placed into production.

Again this approach can provide useful insights into the nature of supply, but it does presume that undeveloped deposits will be brought into operation in order of their average total costs and only after price rises to a level that covers these costs. In practice, this is not always so. Some deposits come on-stream sooner because host governments are willing to provide expensive infrastructure and in other ways subsidize their development. On the other hand, political risk, heavy taxation, and other adverse public policies may delay the development of other deposits.

2.1.2.2 Byproducts and Coproducts

So far, we have taken account of only the primary supply of individual products. Often, however, particularly in the case of metals, two or more commodities are recovered from the same ore body. In such instances, as pointed out earlier, we can distinguish among metals mined as main products, coproducts, and byproducts. A *main product* is by definition so important that it alone determines the economic viability of a mine. When two metals must be produced to make a mine economic, both influence output, and they are *coproducts.* A *byproduct* is produced in association with a main product or with coproducts. Its price has no influence over the mine's ore output, though normally as we will see it does affect byproduct production.

Main product supply is quite similar to that for individual products discussed previously, and so is not considered further here. Instead, this segment, drawing on Brooks (1965), highlights the differences between the supply of byproducts and coproducts on the one hand and the supply of individual products on the other.

Before proceeding, however, we should point out that some metals, such as gold, silver, and molybdenum, are main products at some mines, coproducts at other mines, and byproducts at still other mines. To determine the total primary supply for such metals, one needs to assess main product, byproduct, and coproduct supply and then add them together. Moreover, gold and other metals may at some mines be a byproduct at times

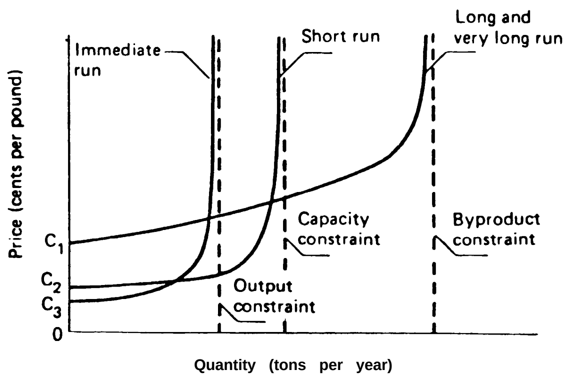

Fig. 2.1.7. Supply curves for byproduct metal in a competitive market.

and a coproduct or even main product at other times, if the price of gold and associated metals varies greatly.

Byproduct Supply: There are two important differences between the supply of byproducts and the supply of individual or main products. The first is that byproduct supply is limited by the output of the main product. The amount of molybdenum recoverable as a byproduct of copper production, for instance, cannot exceed the physical quantity of molybdenum actually in copper ore. As production approaches this constraint, the byproduct supply curve turns upward and becomes vertical. This is because a higher byproduct price does not increase the output of ore or of main product. Otherwise, it would not be a byproduct. So at some output, supply even in the very long run becomes unresponsive or inelastic to further increases in byproduct price.

This characteristic of byproduct supply is illustrated in Fig. 2.1.7. Both the long- and very-long-run supply curves, shown as the same curve, are quite elastic with respect to price until output approaches the byproduct constraint, which is imposed by main product output. Thereafter, little or no increase in supply is possible. Since normally byproduct producers are competitive firms, Fig. 2.1.7 shows the byproduct supply curve for only the competitive market. The same constraint, however, would apply in a producer market.

Since it is the price of the main product, rather than the byproduct, that stimulates supply in the very long run, through exploration and development of new technologies for exploiting alternative ores, the long- and very-long-run byproduct supply curves may actually be the same, as shown in Fig. 2.1.7. This, however, need not be the case. A high byproduct price, for example, may encourage new technology that allows a greater recovery of the byproduct. In this case, the byproduct constraint becomes binding at a greater output in the very long run, causing the long- and very-long-run byproduct supply curves to deviate from each other.

The constraint imposed by main product output shifts in response to changes in main product demand and supply. If the demand for copper goes up, for example, this causes the price and output of main product copper to rise, increasing the amount of ore from which byproduct molybdenum can be recovered. For this reason, the supply function of a byproduct should normally contain the output (or price) of the main product, in addition to its own price and the other important supply variables discussed earlier. Since changes in main product output are independent of the byproduct price, they reflect a shift in the byproduct curve itself, rather than a movement along it.

At times, main product ores may contain more of the byproduct than existing capacity can treat. This constraint, however, applies only in the immediate and short runs, as new capacity can be installed in the long run. Such a situation, where capacity limits byproduct supply in the immediate and short runs, is illustrated in Fig. 2.1.7. The immediate-run supply curve in Fig. 2.1.7 is further constrained by current output (plus available producer stocks), implying that the industry is not operating at full capacity. Otherwise, the output and capacity constraints would be the same, and the immediate- and short-run curves would both become inelastic at about the same output.

A second important difference between byproduct and individual (or main) product supply is that only costs specific to byproduct production affect byproduct supply. Joint costs, those necessary for the production of both the main product and the byproduct, are borne entirely by the main product and do not influence byproduct supply.

This means that byproduct supply curves for competitive markets, such as those drawn in Fig. 2.1.7, reflect the marginal costs of byproduct production exclusive of all joint costs. As a result, byproduct supply until production approaches the constraint imposed by main product output is often, though not always, available at lower costs than the same metal from main or individual product supply.

It is sometimes assumed that byproducts are basically free goods and that the byproduct supply curve is simply a vertical line at that output reflecting the amount of byproduct in the main product ore. For this to be true, however, two conditions must be satisfied: (1) the production of the main product must require the separation of the byproduct, and (2) no further processing of the byproduct must be necessary after separation. The first condition often is satisfied. It is the second condition that normally gives rise to specific byproduct costs. Byproducts will not be recovered and supplied to the market unless their price covers these specific costs. In Fig. 2.1.7, the lowest cost byproduct producer has specific costs equal to OC_1 cents per pound. Once byproduct capacity is in place, the market price can fall below specific costs and production will continue in the short run, as long as price covers the minimum variable or out of pocket costs specific to byproduct production (OC_2). In the immediate run, the price can even decline further (OC_3). Over the long run, however, capacity will not be replaced and byproduct production will cease if price remains below the minimum specific cost (OC_1).

Since byproduct production tends to occur first where main product ores are particularly rich in the byproduct mineral or are for other reasons less costly to process, the marginal costs specific to byproduct production usually rise with output. For this reason, the long- (and very-long) run supply curve shown in Fig. 2.1.7 has an upward slope over its relatively flat or elastic segment.

Coproduct Supply: Coproducts are in many respects between byproducts and main products. Their price influences mine output, but so do the prices of associated coproducts. Joint production costs must be shared, as no single coproduct can support them alone. This means that a coproduct's price must cover its specific production costs plus some but not all of joint costs.

Consequently, a coproduct's supply function includes its own price, the price of other coproducts, its specific costs, and joint costs, as well as possibly other factors. A change in any of these supply determinants, other than own price, causes a shift in the supply curve. An increase in specific or joint costs, for example, shifts the curve upward, while an increase in the price of an associated coproduct shifts it downwards.

Coproduct supply curves have the same general shape as those illustrated for individual products in Fig. 2.1.5. In the immediate, short, and long runs, coproduct supply is similarly constrained by current output, capacity, and known deposits.

Since a coproduct must bear only a part of joint production costs, supply may be available at lower costs than from main or individual product output. This reduces the height of the supply curve in competitive markets and possibly in the producer markets as well.

2.1.2.3 Secondary Production

Secondary production adds to the supply of some mineral commodities by recycling new and old scrap. *New scrap* is generated in the manufacturing of new goods. The aluminum sheet that remains after stamping the round tops for soft drink cans is part of the supply of new metal scrap. *Old scrap* comes from consumer and producer goods that have come to the end of their useful lives because they are obsolete, worn out, or for some other reason no longer of use. An empty soft drink can is part of old scrap supply. In recent years, the recycling of old scrap alone has accounted for about 20% of US aluminum consumption, 25% of US copper consumption, and 50% of US lead consumption.

Secondary supply differs in several respects from primary supply, particularly that for individual or main products. For example, secondary producers are for the most part highly competitive and rarely support a producer price. So in considering secondary supply, we need not be concerned with supply curves for a producer market.

In addition, it is the availability of scrap, rather than the availability of known deposits, that limits secondary supply in the long run. Since the important factors determining the availability of new and old scrap differ, secondary supply from these two sources are best considered separately.

Secondary Supply from New Scrap: The amount of new scrap available for recycling depends on three factors—current overall metal consumption, the distribution of this consumption by end uses, and the percentage of consumption resulting in new scrap for each end use. A rise in overall copper consumption, due to an increase in GNP or a change in consumer preferences, increases the availability of new copper scrap, causing both the constraint and the long-run supply curve for secondary copper metal produced from new scrap to shift to the right. On the other hand, improved manufacturing techniques that reduce the percentage of copper scrap generated in the manufacturing of electric wire or other fabricated products shifts the constraint and the curve to the left.

This means that the supply function for secondary metal from scrap, in addition to technological change and other determinants discussed earlier, should include a variable for the availability of new scrap. Often metal consumption is used as a proxy for the availability of new scrap. This is appropriate, however, only if its distribution by end uses and the proportion of consumption resulting in new scrap for each end use remain unchanged. Otherwise, these variables as well belong in the supply function.

The shape of the long-run (and very-long-run) supply curve for secondary metal from new scrap is shown in Fig. 2.1.8 and reflects the cost of collecting, identifying, and processing new scrap. The scrap which is the least costly to recycle will be processed first. These costs, given in Fig. 2.1.8 as OC_1 cents per pound, determine the point where the curve intersects the vertical axis. As most new scrap is relatively inexpensive to recycle, the slope of the curve rises from this point very gently over the range of possible outputs. Only as the constraint imposed by the availability of new scrap is approached does the supply curve turn upward.

The low cost of recycling new scrap compared to alternative sources of supply means all or almost all new scrap is recycled.

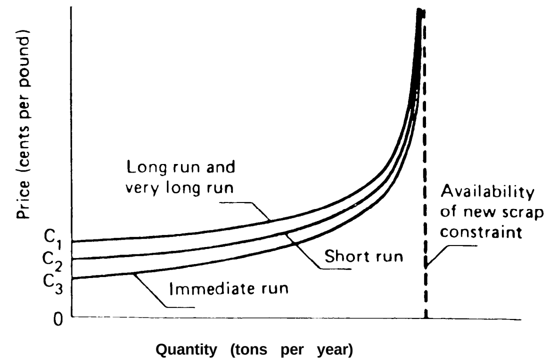

Fig. 2.1.8. Supply curves for secondary metal from new scrap in a competitive market.

So over the range of normal prices, little additional supply from new secondary is possible, making supply price inelastic. However, at very low prices, those approaching the cost of recycling new scrap, supply is—as Fig. 2.1.8 illustrates—quite elastic with respect to price.

The fact that almost all new scrap is recycled means that the constraints limiting supply in the immediate and short run, namely output and capacity, are not likely to differ significantly from the constraint in the long run imposed by the availability of new scrap. So although the immediate- and short-run curves lie below the long-run curve, they turn upward and become vertical at about the same output.

If current technology allows the full recovery of the metal content of new scrap, the very-long-run supply curve is also constrained by the availability of new scrap. In this case, it and the long-run curve coincide, as illustrated in Fig. 2.1.8. If this is not the case, if a high metal price induces over the very long run new technology that allows more metal to be recovered from the available scrap, the constraint on supply would be further to the right in the very long run. It would, however, not be eliminated. At some output the supply of secondary metal from new scrap, like that for byproduct supply, becomes inelastic to price even in the very long run and in this regard differs from the supply of individual and main metal products.

Secondary Supply from Old Scrap: The availability of old scrap during any particular year depends on (1) the flow of metal-containing products reaching the end of their service life during the year, and (2) the stock of metal-containing products no longer in use or service at the beginning of the year, but which have not yet been recycled. The number of old automobiles available for recycling, for example, includes those scrapped during the year as well as those scrapped in earlier years but which for one reason or another have yet to be recycled.

The flow of old metal scrap depends on the number and types of goods in use throughout the economy at the beginning of the year, their metal composition, their age distribution, the mean age at which they come to the end of their service life, and the frequency distribution around this mean. Since these factors together determine the flow of old scrap, they can be included in the supply function for secondary metal produced from old scrap in place of the latter.

The stock of old metal scrap depends on the accumulated past flows of products coming to the end of their service life. From this total, the quantities already recovered through recycling must be subtracted. This means that over time, the stock of old scrap will grow if the amount recycled is less than the incoming flow.

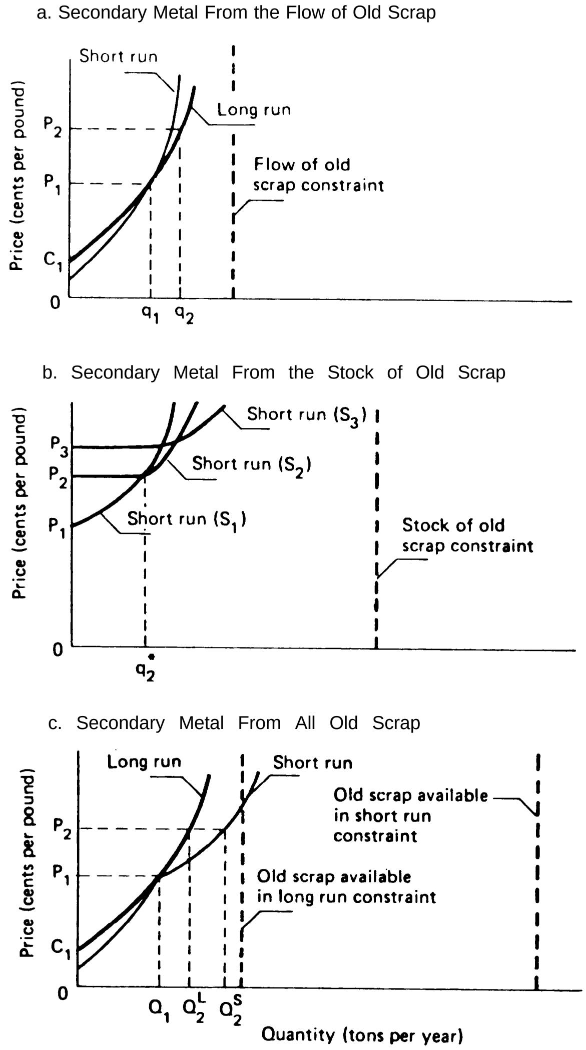

Fig. 2.1.9. Supply curves for secondary metal from old scrap in a competitive market.

Even though it depends on two flow variables—the accumulated flow of old scrap and the accumulated recycling of this scrap in the past—the stock of old scrap is as stated "a stock" and not a flow. Consequently, what is recycled this year is not available for recycling in the future. This has several important implications, and makes it necessary to distinguish between the supply of secondary produced from the flow and from the stock of old scrap.

A long-run supply curve for the former is shown in Fig. 2.1.9a. At the price P_1, this curve indicates that the quantity q_1 of the secondary metal is recovered from the incoming flow of old scrap. The remainder of the incoming flow of old scrap is not recycled, but rather added to the stock of old scrap available for recycling in the future.

The curve begins as the vertical axis at a fairly low price, reflecting the fact that some old scrap is of very high quality. It can be recycled at a relatively low cost (OC_1) per pound of contained metal. However, in contrast with new scrap, costs rise notably as more and more of the old scrap flow is recycled. This is because some scrap is scattered geographically, and so collection costs are high. Some is a mixture of various scrap types, requiring expensive identification and sorting techniques. Some is highly contaminated with rubber, glass, wood, and other waste material, making treatment costs high. Indeed, in some uses, such as lead in gasoline, the metal is so dissipated after use that recycling is simply too costly to contemplate. It is for such reasons that much of the old scrap flow is not recycled for many metals.

Fig. 2.1.9a also illustrates a short-run supply curve for secondary metal produced from the flow of old scrap. This curve is drawn beneath the long-run curve at prices below P_1, on the assumption that secondary producers will continue to operate and supply the market in the short run as long as they can cover variable costs. Since fixed costs tend to account for a relatively small share of total production costs, particularly in comparison with primary production, the short-run curve lies relatively close to the long-run curve.

At prices higher than P_1, Fig. 2.1.9a shows the short-run curve above the long-run curve, since producers at some output encounter capacity constraints. However, capacity tends to be more flexible in secondary than primary production. It is easier to raise output by increasing the number of shifts and by augmenting labor in other ways. For this reason, the short-run supply curve above the price P_1 is drawn quite close to the long-run curve.

This, though, need not necessarily be the case. Under certain circumstances, the short- and long-run curves may lie quite far apart. For example, at particularly high metal prices, certain products that account for a significant part of the total old scrap flow may become economic to recycle. The capacity to handle these products, however, may not exist, causing the short-run supply curve to become vertical considerably before the long-run curve.

Conversely, short-run supply can exceed long-run supply at relatively high prices, with the reverse being true at low prices, as a result of the "accelerated scrapping" phenomenon. The latter occurs when products close to the end of their service life are scrapped earlier than they otherwise would be as a result of high metal prices. For example, at times, old or obsolete machines are held in reserve for use during peak production periods, for emergencies, or are stored away and cannibalized for their parts. High metal scrap prices encourage the premature recycling of such equipment. When metal prices are particularly low, the costs of keeping such equipment in terms of the scrap revenues forgone are quite modest. This implies that in some circumstances, the constraint imposed by the flow of old scrap may not be invariant in the short run to price but rather may increase with price at least over a range.

So far we have focused entirely on secondary supply from the flow of old scrap. Fig. 2.1.9b illustrates three short-run secondary supply curves from the stock of old scrap. The first curve S_1 indicates that at the price P_1, no metal is recovered from the stock of old scrap. This price simply does not cover recycling costs. At higher prices, however, some of the available old scrap stock can be economically processed. At P_2, for example, the old scrap stock will produce an amount of metal equal to q^*_2.

Over the long run, however, this output is not sustainable, because the old scrap stock recoverable at costs under P_2 is depleted. So if price remains at P_2, the short-run curve soon becomes truncated in a manner similar to that illustrated by the curve S_2. Above price P_2 this curve lies somewhat to the right of the original curve, reflecting the fact that the flow of old metal scrap with recycling costs above P_2 has over the intervening years not been recycled, but rather added to the stock of old scrap. Below price P_2, however, the new supply curve S_2 indicates that no supply from the stock of old scrap is now forthcoming, as the material that can be recycled economically at this price has been exhausted. Similarly, if the price rises to P_3, the short-run supply curve will soon approach the shape indicated by the S_3. This means that normally there exists no long-run supply curve for secondary metal from the stock of old scrap.[c]

Although the size of the stock of old scrap does impose a constraint on the ultimate supply of secondary metal, this constraint is not nearly as binding as is the availability of scrap in the case of supply from the flow of old scrap or from new scrap. Long before the stock of old scrap is exhausted, processing is constrained by high costs. Most of the available old scrap stock has not been recycled precisely because at prevailing prices it has been uneconomical to do so. Other sources of supply—individual product, main product, coproduct, byproduct, secondary from new scrap—for the most part have been cheaper.

As Fig. 2.1.9c illustrates, the secondary supply curve from all old scrap can be derived by combining or, more precisely, by adding horizontally the secondary supply curve for the old scrap flow and for old scrap stocks. The long-run curve is simply that for secondary from the old scrap flow shown in Fig. 2.1.9a, as there is no long-run secondary supply curve from the stock of old scrap.

The short-run curve is derived by adding the appropriate short-run curve for the stock of old scrap, assumed to be the curve S_1 in Fig. 2.1.9b, to the short-run curve for the flow of old scrap. Since the short-run curve for the stock of old scrap intersects the vertical axis at P_1, implying that below this price no secondary metal from old scrap stocks is forthcoming, the short-run secondary curve for all old scrap below this price is simply the short-run curve for the flow of old scrap.

Fig. 2.1.9c highlights two interesting facets of secondary supply from old scrap. First, the constraint posed by the availability of scrap is actually less binding in the short run than in the long run. This, of course, is because the stock of old scrap, if exploited in the short run, is not available in the long.

Second, an increase in price, for example from P_1 to P_2, may actually produce an increase in supply that is greater in the short run than in the long run. In the short run, some of the stock of old scrap may be recycled, adding to supply. It is presumably for this reason that efforts to measure the price elasticity of secondary metal supply from old scrap (Bonczar and Tilton, 1975; Fisher et al., 1972) have found the elasticity to be greater in the short run than in the long run, which is just the opposite from what one normally finds with other sources of supply. However, as Fig. 2.1.9c indicates, this unusual result should be expected only if the market price is above the price at which secondary supply from old scrap stocks is forthcoming, that is, above the price P_1 in Fig. 2.1.9. When this is not the case, the figure suggests a change in price will produce a greater increase in supply in the long run than in the short run. Though as pointed out earlier, even here the phenomenon of accelerated scrapping can cause the response of supply to an increase in price to be greater in the short than the long run.

Nothing has been said so far about the immediate-run and very-long-run supply curves for secondary metal from old scrap, in large part because the curves are not particularly unusual. The immediate-run curve has the general shape illustrated for the competitive market in Fig. 2.1.5. At very low prices, producers will save much or all of their current production, in hope of higher prices in the future. As price increases, however, and

[c]It is possible that at very high prices, large reservoirs of old scrap might become economical to process. For example, at some price, certain metals found in dumps and landfills could presumably be extracted profitably. In such instances, the depletion of the available stock of old scrap could take a number of years, and make a long-run supply curve possible.

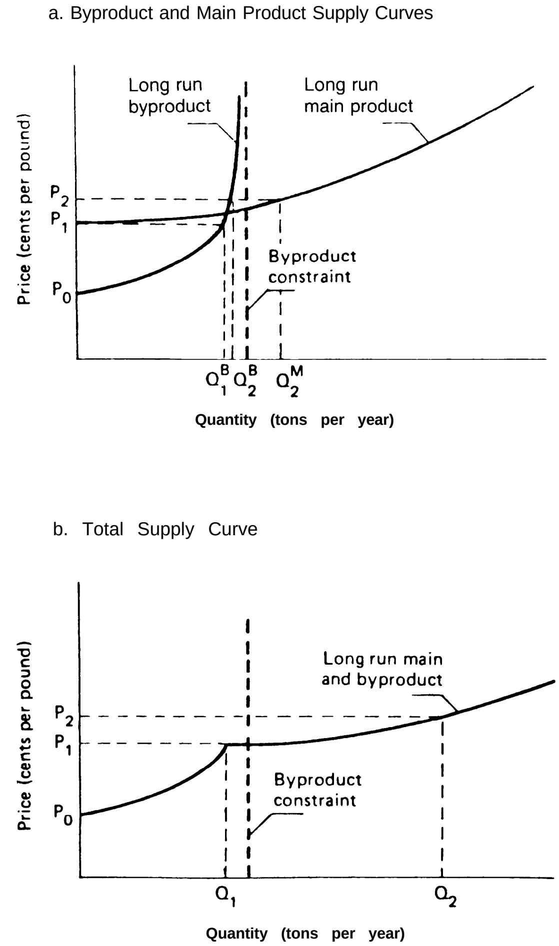

Fig 2.1.10. Total long-run supply curve for a metal produced as a byproduct and a main product.

supply approaches the constraint imposed by current production, the supply curve turns upward and becomes quite inelastic.

The very-long-run supply curve has the general shape of the long-run curve and is similarly constrained by the metal content of the flow of old metal scrap. Indeed, in some circumstances, the two curves may coincide. Where high metal prices, however, stimulate old scrap recovery technology and thereby permit a greater proportion of the flow of old metal scrap to be economically recycled at any particular price, the very-long-run curve will lie somewhat to the right, and perhaps below, the long-run curve.

2.1.2.4 Total Supply

Individual products, main products, coproducts, byproducts, and secondary from new and old scrap are all potential contributors to a mineral commodity's total supply. To derive the total supply curve, we must add horizontally the individual curves for all significant sources (in a manner similar to that just used to derive the secondary supply curve for all old scrap). For example, if the byproduct and main product supply curves for a mineral commodity are as shown in Fig. 2.1.l0a, and if other sources of supply are unimportant and can be ignored, the horizontal summation of these two curves gives the total supply curve shown in Fig. 2.1.l0b.

Among its many uses, the total supply curve provides an indication of the relative competitiveness of the different sources of supply and, in turn, their relative importance at different market prices. For example, byproduct production is initially the most competitive and cheapest source of supply for the commodity whose total supply curve is shown in Fig. 2.1.l0b. When market price is below P_1, it is the only source of supply. As the market price rises above P_1, however, main product production begins and becomes increasingly important. At the price P_2, main product supply exceeds byproduct supply.

In deriving the total supply curve, a question sometimes arises as to whether the secondary supply curve for new scrap should be included. New scrap, after all, is generated in the manufacturing process and depends on other sources of supply. If metal fabrication becomes more efficient and generates less new scrap, this does not mean total metal supply has declined.

Whether to include secondary supply from new scrap depends on the purpose for which total supply is being considered. In assessing the extent to which the United States is vulnerable to interruptions in supply from certain foreign producers, for example, we would not want to include secondary supply from new scrap. To do so would ignore the fact that this metal is generated from other sources of supply, and so would not be available in their absence. Including it would underestimate US dependence on foreign producers. On the other hand, in assessing the competitiveness of secondary metal markets, we would normally want to consider secondary supply from new scrap.

Regardless of how secondary supply from new scrap is treated, it is important that total supply and demand be consistent in this regard. If the demand curve takes account only of metal actually contained or embodied in final products, and excludes metal that ends up as new scrap and is recycled, then total supply should also exclude new scrap. Conversely, if demand includes the demand for all metal, including that which ends up as new scrap, then the total supply curve should include new scrap as well.

2.1.3 APPLICATIONS

This final segment illustrates the usefulness of the supply and demand principles examined in the preceding segments. It uses these concepts to analyze the problem of mineral market instability and the incentive price technique for forecasting long-run prices for mineral commodities.

2.1.3.1 Market Instability

Mineral markets are well known for their instability, for their feast-or-famine nature. In an effort to stabilize commodity markets, particularly for the benefit of producers in the developing countries, the United Nations Conference on Trade and Development (UNCTAD) has over the last 30 years pushed for the creation of an Integrated Program for Commodities. Among other measures, this program proposes to establish a common fund on which international commodity agreements can draw to support market stabilization measures. While the proposed program has encountered a number of difficulties, it does reflect the concern on the part of both producing and consuming countries over the instability that plagues mineral markets.

Nor is this instability new. One of the major driving forces behind the multiple mergers in the American steel industry at the turn of the last century, which culminated in 1901 with the creation of the US Steel Corp., possessing at that time some two-thirds of the country's steel-making capacity, was a desire to control the volatile steel market. Gyrations in steel prices during the 1880s and the 1890s had created severe problems for all producers.

A highly concentrated market structure where one or a few major producers dominate the market and set a producer price does not, however, eliminate market instability (though as we shall see, it does alter the ways in which market instability manifests itself). This is because the following three characteristics of short-run metal supply and demand, which are responsible for market instability, are present no matter how concentrated the market.

First, as output approaches the capacity constraint, total supply becomes increasingly price inelastic. In a competitive market, as shown earlier, the short-run supply curve turns upward and at some point becomes vertical. In a producer market, the curve simply ends when major producers no longer have sufficient supply to satisfy demand at the producer price.

Second, demand also tends to be price inelastic in the short run. So the slope of the demand curve is quite steep.

Third, demand is often highly elastic to changes in national income over the business cycle. The consumption of most metals and many other mineral commodities is concentrated in four sectors—construction, capital equipment, transportation, and consumer durables—whose output is particularly sensitive to fluctuations in the business cycle. During a recession, these sectors suffer far more than the economy as a whole. During a boom, their sales soar. As a consequence, the demand curve for most metals shifts considerably over the business cycle.

These characteristics of short-run supply and demand are illustrated in Fig. 2.1.11a for a commodity sold in a competitive market and in Fig. 2.1.11b for a commodity sold in a producer market. In both instances, it is assumed that supply comes from individual or main product production. This simplifies the analysis, but does not alter the conclusions, since total supply, regardless of the combination of sources from which it is derived, is at some output constrained in the short run by the available production capacity. As supply approaches this constraint, it becomes inelastic to price.

The two characteristics of demand—its low elasticity with respect to price and its high elasticity with respect to income-are portrayed in Figs. 2.1.11a and 2.1.11b by the steep slope of the demand curves and by the shifts in the demand curves over the business cycle. The curve DD_t reflects demand at the trough of the cycle, the curve DD_m at the midpoint of the cycle, and the curve DD_p at the peak of the cycle.

One of the important consequences of market instability for copper, tungsten, and other mineral commodities sold on competitive markets is the severe fluctuation in market price. It varies in Fig. 2.1.11a from a high of P_p at the peak of the business cycle to a low of P_t at the trough. The output that producers supply to the market also varies greatly, from a high of Q_p (approximately the maximum possible given the industry's production capacity) to a low of Q_t. At the latter level, producers are burdened with either shutting down much of their capacity or adding a large part of their output to their inventories.

Fig. 2.1.11a also indicates that when market instability is caused by shifts in the demand curve (rather than by shifts in the supply curve, which is more typical for agricultural commodities), the quantity sold and price move together. When one is down, so is the other. Consequently, total revenues and in turn profits tend to be highly volatile.

For mineral commodities sold in producer markets, the situation is somewhat different. If all firms faithfully adhere to one producer price, there is no price instability. Even if this is not the case, if some open or secret discounting occurs, or if the producer price is reduced when the market is weak, price instability will generally be less than in competitive markets.

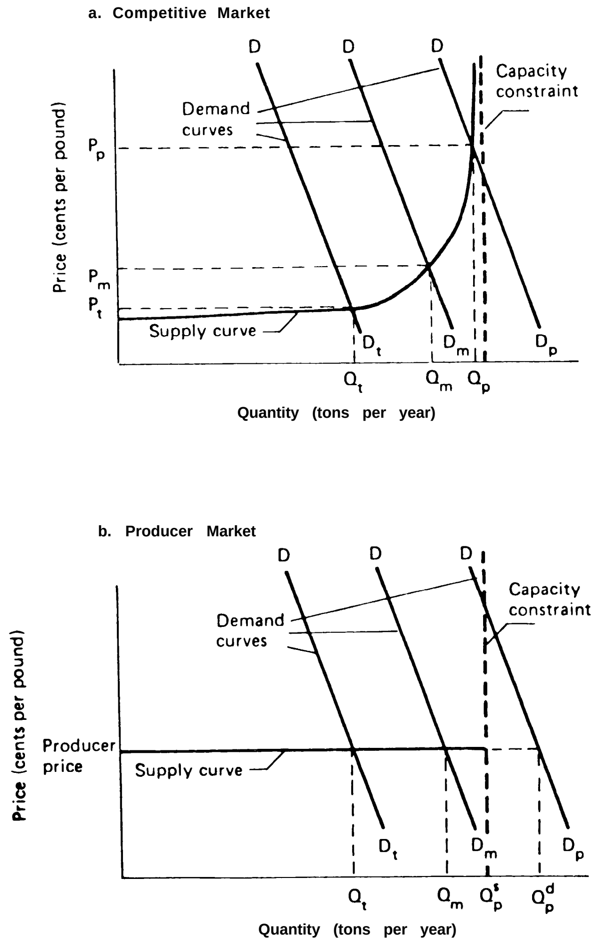

Fig. 2.1.11. Short-run supply and demand curves.

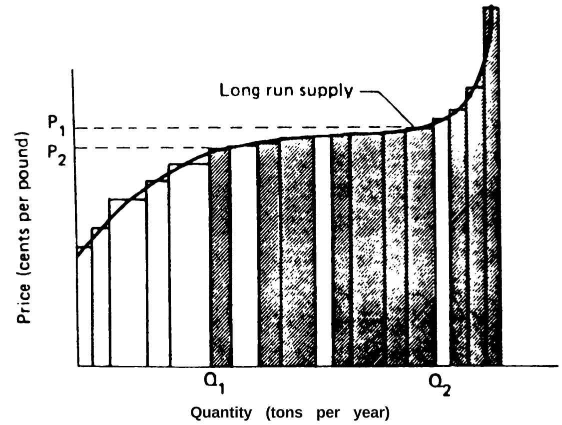

Fig. 2.1.12. Long-run supply curve for a metal with many underdeveloped deposits of similar quality. (Shaded areas represent known but undeveloped deposits.)

However, physical shortages, where the available supply is insufficient to satisfy demand at the prevailing price, can occur in a producer market. In Fig. 2.1.11b, for instance, the quantity demanded during the peak of the business cycle is Q_P^d while the maximum amount the industry can supply is only Q_p^s. The shortfall requires that producers allocate or ration their limited supply to customers.

Despite the greater price stability, firms in producer markets still suffer from sizable fluctuations in total revenue and profits over the business cycle as a result of the instability in metal demand and the impact that this has on sales. In these respects, the adverse effects of market instability are similar for firms in both producer and competitive markets.

2.1.3.2 Forecasting Long-Run Metal Prices

Firms studying the feasibility of developing new mines or of expanding existing facilities make forecasts of long-run commodity prices. Similarly, manufacturers contemplating the substitution of one material for another rely on price forecasts. Governments whose tax receipts and foreign exchange earnings follow trends in export markets and local communities whose economy depends on mineral production also have an interest in future price forecasts.

The incentive price technique is one method of making such forecasts. It assumes that over the long run, a commodity's price must approach that level that provides the necessary incentives to ensure production capacity just sufficient to satisfy demand.

As long as entry into the industry is not severely restricted, this condition is likely to be met. When market price is above the incentive price, new firms will enter the industry and existing firms will expand their capacity. The resulting increase in supply will eventually reduce price, and this downward trend will continue until the incentive price is reached and the expansion of existing capacity ceases.

Conversely, when the market price is below the incentive price, neither new firms nor established producers will be motivated to invest in new capacity. As existing facilities become obsolete and are retired from service and as demand grows over time the market price will rise. This upward movement will continue until the incentive price is reached and once again firms are willing to build new capacity.

Success in applying the incentive price technique lies in identifying the incentive price itself. While this can be extremely difficult for some mineral commodities owing to the nature of their long-run supply, it may be relatively simple.

Fig. 2.1.12, for example, portrays the long-run supply curve for a commodity produced largely as an individual or main product and sold in a competitive market. This curve is derived from the average total costs and annual production capacities associated with available deposits, both those actually in operation and those that are undeveloped but known to exist. The undeveloped deposits, denoted by the shaded areas in Fig. 2.1.12, are among the higher costs sources of supply, since lower cost deposits tend to be exploited first.

Because a very wide range of supply (from Q_1 to Q_2) is available in the long run over a narrow band of prices (from P_1 to P_2), the incentive price for the metal portrayed in Fig. 2.1.12 may be relatively easy to approximate. The long-run supply curve has a relatively flat segment, such as that shown, when a number of large deposits of comparable quality and costs are known to exist and so are available for development.

If this flat segment of the long-run supply curve is broad enough, it may cover all likely points of intersection with the long-run demand curve. The incentive price can then be approximated from the estimated average production costs, including an adequate rate of return on invested capital, associated with those marginal deposits determining the relatively flat and wide portion of the long-run supply curve.

Using this procedure, Radetzki (1983) has estimated that the incentive price for aluminum is between 80 and 100 ¢/lb (176 and 220 ¢/kg) and that for copper between 120 and 150 ¢/lb (265 and 331 ¢/kg) (both in 1980 dollars). At the time of his study, prices for both of these metals were substantially below these figures, allowing him to predict that aluminum and copper prices would rise over the long run.

The incentive price technique is a useful forecasting method when the incentive price itself is easily and accurately determined. In the following circumstances, however, this is not the case.

First, when entry into the industry is restricted and one or a few firms set a producer price, the long-run price may not approach the average production costs of the marginal deposits. In such situations, the determinants of long-run price are more difficult to identify and assess.

Second, the long-run supply curve may not have a broad flat segment. This depends on the quality differences among known deposits. If there is no tendency for a sizable number of large deposits to share the same quality—the same ore grade, ease of access, and processing costs—there will be no narrow price band over which large quantities of supply are forthcoming in the long run.

Third, even if the long-run supply curve possesses a broad flat segment, the range of possible intersection points with the long-run demand curve may fall outside this segment. Some forecasters, for example, are now suggesting that large porphyry copper deposits may not be needed in the foreseeable future, in part because the growth in copper demand has slowed in recent years, and in part because more copper supply will be coming from secondary and coproduct production. As a consequence, the long-run demand curve may intersect the long-run supply curve before the latter flattens out in response to the availability of porphyry copper deposits of comparable quality.

Fourth, the discovery of new deposits and especially the development of new technology can cause the very-long-run supply curve to lie appreciably below the long-run supply curve. Forecasting on the basis of the incentive price derived from the long-run supply curve will, as a result, substantially overestimate future price in some instances.

These considerations reflect violations of the four necessary conditions for obtaining reliable forecasts with the incentive price technique. It is important to consider whether these conditions are satisfied before using the technique to predict future metal prices.

Market instability and long-run price forecasting are only two examples of the usefulness of economic principles in analyzing mineral markets. Many other possibilities exist. We could, for instance, have assessed the market impact of a substantial release of tin from the US government stockpile, the effects of civil disruption in South Africa on the export price for platinum or coal, the consequences of seabed mining for land-based nickel producers, or the costs to domestic consumers of protecting the US steel industry.

The two examples, however, suffice to demonstrate the usefulness of relatively simple economic principles of supply and demand for the analyst investigating mineral markets and industries. The examples also indicate that these conceptual tools must be combined with specific knowledge about the nature of mineral supply and demand. It is not enough to know that the demand curve in most markets is downward sloping. Some idea of just how responsive or unresponsive demand is to price, and how this responsiveness varies from the immediate to the very long run, is essential. Some applications even require that the total demand be broken down into its various components, such as the demand for consumption, for inventories, and for government stockpiles.

Similar information is also needed on the nature of supply. To what extent, for example, does supply come from individual and main product output, from secondary production, from byproduct and coproduct production? How does market structure and the manner in which prices are determined affect supply? How responsive is supply to a change in price? How does this responsiveness change as supply approaches the constraint imposed by current production in the immediate run, by existing capacity in the short run, and by the availability of mineral deposits in the long run? How does the output or price of a main product affect the supply of its associated byproducts? How does secondary supply vary with metal consumption, prices, and the flow of old scrap?

No single set of answers to such questions is valid for all mineral commodities. The answers may even change for the same commodity, as technology and other conditions evolve over time. So the good analyst of mineral industries must know economic principles, but he or she must also know how technology, institutions, market structure, and government policies shape mineral supply and demand.

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Chapter 2.2
COMMODITY REVIEW

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2.2.1 INTRODUCTION

Of the approximately 100 nonfuel mineral commodities produced worldwide, more than 70 are mined in the United States. Domestic production represents many thousands of operators, ranging from comparatively small but numerous sand and gravel producers to large but relatively few integrated multinational producers of steel, copper, aluminum, and other metals and chemicals. For some commodities, there is little or no US production. The focus of this chapter is on nonfuel minerals of significant commercial importance, with the emphasis on the US position in these materials.

Stone was probably the first mineral commodity used by man. From stone, man produced tools, weapons, and shelter. Stone continues to this day as the largest-valued mineral product in the United States. With time, man learned to use the metals that occur in nature in elemental form such as copper and gold (Muller-Ohlsen, 1981). Later man learned how to produce metals such as iron and bronze by heating certain materials. Increased metallurgical knowledge led to the production of other mineral commodities, such as aluminum that requires the use of electricity and titanium that involves complex chemical reactions and is carried out under an inert atmosphere.

In this period, cartel action of certain world petroleum producers caused energy price increases that profoundly affected the world economy, including minerals production and use. The economic performance of the minerals industry traditionally has been linked closely with the business cycle. As the price of energy increased, the industrial economy underwent tremendous structural change to lower manufacturing costs and conserve energy. For example, the automotive industry designed smaller automobiles to meet mandated fuel efficiency standards, making increased use of lighter materials such as aluminum, magnesium, and plastics and higher-performance steel alloys, in place of copper, zinc, and iron.

In the early 1970s, growth in the mineral industry contributed significantly to the economy (Hodel, 1986). However, in the mid-1970s the recession that followed the petroleum embargo and sharp energy price increases affected the mineral industry, especially the metals sector, more severely than the rest of the economy. Employment and production in the minerals industry fell, as did capacity utilization. For the remainder of the decade, growth rates in mineral production and consumption were lower than those which had been experienced in the years following World War II, and did not keep pace with the growth rates in the general economy. Concern for the environment became a major factor for the minerals industry during this decade. The high cost of environmental compliance, in addition to the increased cost of energy, made some facilities obsolete and forced them to close. Dependence on imports began to increase as the value of mineral imports as a percentage of Gross National Product increased from about 0.6% before 1970, to 0.8% in 1975–1976, and to 0.9% in 1977–1979.

The economic recession of the early 1980s saw mineral production, prices, and employment fall, especially in the metals sector. However, in the period of economic rebound that followed, the mineral industry recovered better than the economy. Foreign competition in metals increased. In addition to the higher energy cost, rising environmental regulation costs, and the recession, the US minerals industry has been affected by high labor cost, declining ore grade and reserves of some minerals, the relative value of the US dollar vs. other world currencies, and the trend of developing countries to process their own ores into industrial products and finished goods.

The United States has historically been a major producer of nonfuel minerals, and was the world's leader in aggregate value of production from about 1900 to the late 1950s when it was displaced by the USSR. In recent years the United States has been losing its world ranking in specific commodities, such as iron ore, steel, lead, potash, and zinc (Table 2.2.1). The United States has essentially maintained its rank in aluminum, cement, copper, phosphate rock, and sulfur, while increasing its position in gold by surpassing Canada.

US demand compared with the base period 1966–1970 is shown in Table 2.2.2 for selected commodities that are used in a wide variety of applications. In general, those commodities such as aluminum, columbium, germanium, rare earths, and platinum-group metals—which are associated with advanced material applications, which have substituted for other materials, or whose uses have benefited from changing demand patterns within the consuming industries—tend to show an uptrend in demand. However, commodities such as iron, manganese, molybdenum, cadmium, lead, and zinc, which are related to manufacturing and the steel industry, or those which have been particularly impacted by environmental constraints, tend to show a decline in demand. Some of this decline in demand can be attributed to increased US imports of finished goods such as automobiles. Agricultural chemicals, some ferrous-alloying metals, and copper have remained relatively flat. The reasons for these differences among commodities will be discussed in the subsequent segments.

The future supply of some minerals may be augmented through ocean mining (see Chapter 22.8). On Mar. 10, 1983, the United States declared an Exclusive Economic Zone (EEZ) that extends 200 nautical miles offshore. Exploration has uncovered many mineral occurrences including sand and gravel, gold, tin, platinum, titanium, zirconium, rare earths, copper, lead, zinc, cobalt, nickel, chromite, phosphate, and manganese. More than 70 coastal countries have taken advantage of EEZs to increase the area available to them for mineral development. In spite of the quantity of minerals that may be available in the EEZ, factors such as their location, extent, characteristics, extractive technology, and competition with land-based mineral extraction will ultimately decide the economic future of seabed mining. Offshore gold placers in Alaska appear promising, whereas offshore titanium and chromite-rich sands appear to be economically less appealing (Anon., 1987a). Although sand and gravel are low-value materials, marine deposits may be in a good competitive position because of land use restrictions that limit onshore development and the characteristics of marine sand (Anon., 1987b). Ferromanganese nodules and crusts in the deep sea contain copper, nickel, cobalt, and manganese, while polymetallic sulfide deposits contain the base metals.

Table 2.2.1. US Production Trends for Selected Nonfuel Minerals (thousand t, except as noted)

	Annual average US production (in thousands), 1966-70	Index of average annual US production (1966-70 = 100)				Average annual US production as a percent of world production					US rank in the world					Top 3 producers in 1988
		71-75	76-80	81-85	1988	1966-70	71-75	76-80	81-85	1988	1970	1975	1980	1985	1988[a]	
Aluminum	3,130	124	138	120	125	38	33	31	26	23	1	1	1	1	1	US, USSR, Canada
Cement	68,700	107	106	95	103	13	11	9	7	7	2	3	4	4	3	China, USSR, US
Copper:																
Mine	1,240	116	110	95	116	24	21	18	15	17	1	1	1	2	1	US, Chile, Canada
Metal[b]	1,240	117	106	89	153	22	21	17	14	20	1	1	1	1	1	US, Japan, Chile
Gold, mine	1,670[c]	76	61	114	360	4	3	3	4	12	4	4	5	4	3	S. Africa, USSR, US
Iron ore	89,100	93	85	56	67	13	10	9	6	6	2	4	4	5	5	USSR, Brazil, Australia
Crude, steel	121,000	100	96	69	69	23	19	16	12	12	1	2	3	3	3	USSR, Japan, US
Lead:																
Mine	378	148	144	117	103	12	16	16	13	12	1	1	1	3	3	Australia, USSR, US
Metal[d]	471	128	120	103	81	16	18	18	15	12	1	1	1	2	2	USSR, US, Japan
Molybdenum	44,000	115	136	97	77	62	62	60	46	40	1	1	1	1	1	US, Chile, Canada, USSR
Phosphate rock, marketable	35,700	110	139	131	129	44	40	40	33	30	1	1	1	1	1	US, USSR, Morocco
Potash (K_2O content)	2,700	87	82	61	56	16	11	9	6	5	4	4	5	6	6	USSR, Canada, German Democratic Republic
Silver	39,100[c]	95	93	107	115	14	12	11	11	10	1	4	5	4	4	Mexico, Peru, USSR
Sulfur	9,580	113	119	112	110	26	23	22	20	19	1	1	1	1	1	US, USSR, Canada
Tungsten	3,790	85	75	44	6	12	9	6	4	1	3	3	5	12	20	China, USSR, Rep. of Korea
Zinc:																
Mine	497	89	71	55	49	10	8	6	5	4	4	4	5	6	8	Canada, USSR, Australia
Metal[d]	890	61	47	30	30	20	10	9	6	4	1	4	5	7	6	USSR, Canada, Japan

Source: Adapted from Hodel, 1986.

[a]Estimate.
[b]Primary and secondary refined production.
[c]Troy ounces.
[d]Primary refined production.
Conversion factors: 1 ton = 0.907 t, 1 troy oz = 31.103 g

Table 2.2.2. US Demand for Selected Nonfuel Mineral Commodities

Commodity	Average Annual US Demand 1966-1970		Index of average annual US demand (1966-70 = 100)[c]			
	Quantity	Units	71-75	76-80	81-85	88[a]
Arsenic	21,000	metric tons	93	60	81	113
Asbestos	700,000	metric tons	101	82	34	12
Aluminum	3,850,000	metric tons	130	147	141	148
Cadmium	5,610	metric tons	89	80	67	66
Chromium	486,000	metric tons	97	108	71	108
Cobalt	7,590	metric tons	111	113	88	115
Columbium	2,240	metric tons	108	130	145	152
Copper	1,970,000	metric tons	100	111	104	116
Germanium	21	metric tons	91	114	181	191
Gold	7,180,000	troy ounces	82	62	45	45
Gypsum	13,550,000	metric tons	122	136	143	168
Indium	572,000	troy ounces	129	108	110	157
Iron ore	75,000,000	metric tons-cont. iron	100	92	58	65
Iron & steel	108,000,000	metric tons	116	115	96	99
Lead	1,200,000	metric tons	110	110	94	98
Manganese	1,150,000	metric tons	106	103	58	65
Mercury	71,000	flasks	76	86	74	66
Molybdenum	27,200	metric tons	105	116	81	62
Nickel	211,000	metric tons	103	103	87	86
Phosphate rock	24,200,000	metric tons	119	151	150	168
Platinum group	1,380,000	troy ounces	115	151	156	159
Potash	3,610,000	metric tons (K_2O equiv)	133	156	143	136
Rare earths (incl Yt)	8,110	metric tons (RE oxides)	153	195	222	196
Selenium	522	metric tons	105	78	b	126
Silver	185,000,000	troy ounces	89	83	64	65
Sulfur	9,320,000	metric tons	111	52	38	131
Tantalum	480	metric tons	121	124	112	86
Tellurium	116	metric tons	123	145	b	b
Tin	73,800	metric tons	85	76	57	75
Titanium	440,000	metric tons	110	107	114	141
Tungsten	7,060	metric tons	116	134	117	142
Vanadium	5,500	metric tons	128	134	b	b
Zinc	1,350,000	metric tons	100	82	76	82

Source: US Bureau of Mines.

[a]Estimated.
[b]Data not available due to withholding of proprietary demand data for some years in the period.
[c]Period average annual demand divided by 1966-1970 average annual demand.
Conversion factors: 1 ton = 0.907 t, 1 troy oz = 31.103 g.

2.2.2 COMMODITY DISCUSSION

2.2.2.1 Base Metals

The base metals are usually considered to be copper, lead, zinc, tin, mercury, and antimony, some of which have been known since ancient times. The ores of these metals usually are the source of a number of byproduct metals, such as precious metals and the minor metals such as arsenic, bismuth, indium, selenium, and tellurium. Base metals, both in elemental and chemical form, have an extremely wide range of industrial uses; copper, lead, and zinc are also used in very large quantities.

Copper: This metal is found in its elemental state, which accounts for its ancient use, but more commonly as sulfide or oxide minerals. The metal is recovered by smelting the sulfide ore to produce an impure blister and anode copper, which is then electrolytically refined to pure copper. In recent years, copper ores have been increasingly smelted in the countries where they are mined. Japan, however, smelts most of its copper from imported concentrate while Europe maintains a smelting industry that also relies significantly on imported concentrates.

Copper is the third most widely used metal, after iron and aluminum, owing to its combined properties of good electrical and thermal conductivity, corrosion resistance, and formability. Western Europe and the United States account for about one-half of world demand. Electrical and electronic uses of copper have continued to be a major growth area and now account for about three-quarters of the total demand for copper.

Copper is mined in approximately 60 countries throughout the world, with Chile, the United States, USSR, Canada, Zambia, and Zaire accounting for about 60% of production. The market economy countries contribute about 80% of world production. World copper mine production has more than doubled over the past 25 years, reaching about 9.4 million tons (8.5 Mt) in 1988. The US share of world mine production was maintained at about 25% through the 1960s, declined to about 14% by the mid-1980s as mine capacity developed in other countries, but recovered to almost 20% by the end of the decade. The largest growth in copper mine production, outside the centrally planned economies, has occurred in countries of South America, the Near East, and Oceania. Mine production in Chile has increased from 880,000 tons (800 kt) in 1975, to 1.54 million tons (1.4 Mt) in

Table 2.2.3. US and World Reserves for Selected Nonfuel Minerals

Commodity	Units	Total US	Total World	Other Major Countries and Percent of World Total
Arsenic	metric tons	50,000	1,000,000[b]	Chile (26); Mexico (6); Canada (5)
Asbestos	thous. metric tons	4,000	110,000	Canada (36); Republic of South Africa (4)
Bauxite	thous. metric tons	38,000	21,800,000	Guinea (26); Australia (20); Brazil (13); Jamaica (9)
Cadmium	metric tons	70,000	535,000[c]	Canada (15); Australia (10); Mexico (7); USSR (7); Republic So. Africa (7)
Chromium	thous. metric tons	0	1,030,000[d]	Republic of So. Africa (80); USSR (10); Finland (2); Zimbabwe (2)
Cobalt	metric tons	0	3,300,000	Zaire (41); Cuba (32); Zambia (11); New Caledonia (7)
Columbium	thous. metric tons	0	3,450	Brazil (93); Canada (4); Nigeria (2); Zaire (1)
Copper	thous. metric tons	57,000	350,000	Chile (24); USSR (11); Zaire (7); Canada (5)
Germanium	metric tons	450[c]	[a]	United States, Zaire
Gold	million troy oz	136	1,416[e]	Republic of So. Africa (54); USSR (14); Canada (3); Australia (4)
Gypsum	thous. metric tons	725,000	2,360,000	Canada (19)
Indium	thous. troy oz	7,000	54,000[c]	Canada (15); USSR (11); Peru (6); Japan (4)
Iron Ore	million metric tons	3,360	65,000[f]	USSR (35); Brazil (15); Australia (14); India (7)
Lead	thous. metric tons	11,000	75,000	Australia (21); USSR (16); Canada (11); Republic of So. Africa (5); Mexico (4)
Manganese	thous. metric tons	0	907,000	Republic of So. Africa (41); USSR (33); Gabon (11); Australia (8)
Mercury	flasks	100,000	3,700,000	Spain (59); USSR (8); Mexico (4); Turkey (3)
Molybdenum	thous. metric tons	2,720	5,530	Chile (20); Canada (8)
Nickel	metric tons	0	49,000,000[e]	Cuba (37); Canada (17); USSR (13); New Caledonia (9)
Phosphate rock	thous. metric tons	1,300,000	13,855,000	Morocco and Western Sahara (51); Republic of So. Africa (18); USSR (9)
Platinum group	thous. troy oz	8,000	1,810,000	Republic of So. Africa (88); USSR (10); Canada (0.4)
Potash	thous. metric tons	60,000	7,000,000[g]	USSR (43); Canada (19); German Democratic Republic (11); Fed. Rep. of Germany (7)
Rare earths (RE oxide content)	thous. metric tons	5,500	45,000	China (80); India (4); Australia (2)
Selenium	metric tons	12,000	80,000[b,e]	Chile (21); Canada (14); Zambia (9); Mexico (5)
Silver	million troy oz	1,000	9,000	USSR (16); Mexico (13); Canada (13); Peru (9)
Sulfur	thous. metric tons	148,000	1,300,000	USSR (27); Canada (13); Iraq (12); Mexico (6)
Tantalum	metric tons	0	22,000	Thailand (33); Australia (21); Nigeria (15); Zaire (8); Canada (8)
Tellurium	metric tons	3,700	22,000[b,e]	Canada (4); Peru (4)
Tin	metric tons	20,000	4,260,000	Malaysia (26); Indonesia (16); Brazil (15); China (9); USSR (7)
Titanium	thous. metric tons	7,350	172,000[h]	Brazil (20); Republic of So. Africa (14); India (12); Australia (11); Norway (11)
Tungsten	metric tons	150,000	2,700,000	China (45); Canada (13); USSR (11); Australia (5)
Vanadium	thous. metric tons	136	4,300	USSR (62); Republic of So. Africa (20); China (14); Australia (1)
Zinc	thous. metric tons	20,000	147,000	Canada (16); Australia (13); USSR (7); Peru (5)

Source: Anon., 1985, 1989.

[a]Not available.
[b]Based on content in copper ores.
[c]Based on content in zinc ores.
[d]Gross weight of shipping grade ore. Quantity normalized to 45% Cr_2O_3 for virtually all deposits.
[e]Excludes some centrally planned economies.
[f]Iron content.
[g]Potash (K_2O) equivalent.
[h]Titanium content of ilmenite, rutile, and anatase.
Conversion factors: 1 ton = 0.907 t, 1 troy oz = 31.103 g.

1988, resulting in Chile and the United States vying as the largest copper mine producer.

In the early 1950s, eight privately owned North American and Western European companies controlled about 75% of market-economy country production. However, since that time, most mines owned by these companies in the less developed countries (LDCs) have been expropriated or nationalized. The result has been that more than 50% of world copper mine production is by government-owned or controlled corporations, and thus may not always be subject to the full impact of market forces.

World reserves of copper are estimated to total about 375 million tons (340 Mt), 85% of which are located in the market economy countries (Table 2.2.3). In the United States, massive porphyry copper deposits in Arizona, New Mexico, and Utah account for most of the reserves and production. Exploitation of these deposits became possible with the development in the early 1900s of the froth flotation process for concentrating sulfide ores. Subsequently, the tendency has been away from the small, high-grade deposits to the large porphyry deposits that are suitable for large-scale open-pit or underground mining methods. About 85% of domestic production is from open pits. While most of the ores are insoluble sulfides that are concentrated by flotation, oxide ores and tailings may be leached with sulfuric acid and the dissolved copper recovered through precipitation or solvent extraction and electrowinning.

Following the recession of 1982, world inventories of copper rose to record levels, and copper prices on the world markets

Table 2.2.4. US Secondary Recovery of Selected Nonfuel Minerals

Commodity (unit)	Secondary Production from Old Scrap				Percent of Demand Filled by Secondary Production			
	1970	1975	1980	1988[e]	1970	1975	1980	1988e/
Aluminum (thous. metric tons)	179	305	617	960	5	7	12	17
Chromium (thous. metric tons)	53	34	48	132	11	9	9	25
Cobalt (metric tons)	31	155	537	1,360[a]	1	2	7	19
Copper (thous. metric tons)	457	335	614	520	24	23	28	23
Gold (thous. troy oz.)	850	1,122	2,184	1,700	14	28	68	53
Lead (thous. metric tons)	459	512	581	635	38	44	55	54
Nickel (thous. metric tons)	44	38	45	45	22	19	24	25
Platinum-group metals (thous. troy oz.)	350	270	331	200	26	21	15	9
Silver (million troy oz.)	56	51	53	25	43	32	43	21
Tin (metric tons)	12,083	15,869	11,665	11,000	18	30	25	20
Zinc (thous. metric tons)	65	70	66	90	5	5	7	8

Source: US Bureau of Mines.
[e]Estimate
[a]Increase over 1980 reflects greater industry coverage.
Conversion factors: 1 ton = 0.907 t, 1 troy oz = 31.103 g.

fell to levels that were below the average production costs in many countries. Consequently, domestic production declined by one-third from 1981 to 1983. Since that time, through restructuring, capital investment, and labor concessions, the US industry has reduced its production costs from 83¢/lb ($1.83 kg) in 1981 to 58¢ ($1.28 kg) in 1987, doubled its productivity, and become one of the most efficient and lowest-cost producers in the world.

Historically, copper smelting and refining have been done in the industrialized nations of the world. However, since 1980, in order to gain the added revenue of downstream processing and to avoid increasing transportation costs, some of the traditional concentrate exporting countries, including the Philippines, Mexico, and Chile, have been developing their own smelting, refining, and semi-fabricating industries. However, Japanese and European smelters and refiners, who lack their own ore resources and have invested heavily in foreign copper mines, still tend to dominate the world trade in copper concentrates. The newly industrializing countries, including the Republic of Korea, Taiwan, and China, have recently invested in smelting, refining, and semifabricating capacity.

Copper is one of the most extensively recycled metals. Copper from old scrap satisfies about one-quarter of US demand (Table 2.2.4). About 60% of the copper recovered from old scrap is consumed in the production of refined copper, the remainder mostly being consumed in alloy form in the production of brass and bronze.

Lead: Lead is mined in about 50 countries, and except for unique deposits in Missouri, Morocco, and the Republic of South Africa, lead is essentially a coproduct of zinc mining or a byproduct of copper and/or gold and silver mining. These complex ores are also the source of byproduct metals such as bismuth, antimony, silver, copper, and gold. Australia, Canada, China, Mexico, Peru, the USSR, and the United States traditionally supply about two-thirds of the world's lead mine production. In the United States, over 90% of the mine production comes from lead-sulfide stratabound deposits in Missouri.

Primary lead smelting is mainly a pyrometallurgical process, as is the refining process, except in Canada, China, Italy, Japan, Korea, and Peru, which have significant electrolytic refinery production associated with cheap coal or hydroelectric power or subsidized power. Secondary lead production is a significant portion of total lead production in several countries, most notably Japan, the United States, and the USSR. Worldwide, secondary lead accounts for over 40% of refined lead production (Anon., 1987c). Secondary production grew through recognition of the fact that lead is readily recyclable, mainly as spent lead-acid storage batteries of all types, while its dissipative uses in chemicals, paints, and gasoline additives declined.

Lead is the most corrosion resistant of the common metals, and, in terms of tonnage, lead ranks after aluminum, copper, and zinc among the nonferrous metals in total usage. About three-quarters of its use is in transportation, mainly in batteries, that make it less susceptible to recession than most other metals. Other uses are in lead alloys for the construction, electrical, and metal fabricating industries. Since the mid-1970s, its use as an antiknock additive in gasoline has substantially declined.

Zinc: Zinc ore is mined in about 50 countries, with about one-half the total coming from Australia, Canada, Peru, and the USSR. In the United States, most zinc is mined in Tennessee where the ore is relatively low grade and in Missouri where zinc is a byproduct of lead mining. Zinc concentrates, produced by flotation of sphalerite in the ore, are processed to produce metal by electrolytic deposition from a sulfate solution or by reduction and distillation in retorts or furnaces. For either method, concentrate is roasted to eliminate most of the sulfur that is recovered as sulfuric acid and sold mainly to the chemical industry.

Until recently, much concentrate produced in the LDCs was shipped to custom smelters, many of which were in Europe, for processing. However, the trend has been for smelters to be constructed within the mining country to achieve the value added. Countries such as India, Republic of Korea, Mexico, Peru, and Thailand have become new zinc metal producers or increased capacity significantly to become exporters of metal. Consequently, less concentrate is internationally traded, while more metal and zinc alloys find their way to the marketplace. The United States, which in the 1960s imported about 440,000 to 550,000 tpy (400 to 500 kt/a) of zinc in concentrate and derived almost 50% of its slab zinc production from foreign material, had by the late 1980s reduced its imports to less than 110,000 tons (100 kt), and only about 25% of its production of metal was from foreign material. During this same time many mine and smelter facilities closed due to environmental and technological factors, and, as a result, US reliance on imports of zinc materials (Fig. 2.2.1) rose from about 50% in the 1960s to 70% in 1988, mainly as metal.

About 60% of world zinc reserves occur in ores associated with coproduct precious metals, lead, and copper, while the remaining zinc reserves are ores in which zinc is the principal valuable component. In addition, it is conceivable that a potential exists for millions of tons of zinc to be recovered from sphalerite-bearing coals in the midcontinent United States.

Zinc metal uses are based on a number of properties, mainly its low melting point, that facilitates shaping by casting, its high electrochemical activity, whereby zinc provides cathodic protection for iron and steel and electrical energy in the ubiquitous zinc-carbon batteries, and its ability to alloy readily with copper to make brass. Galvanized steel products used in motor vehicles and construction account for zinc's greatest use, followed by zinc alloys for die casting, brass for a myriad of uses, coinage, zinc chemicals in agriculture, pharmaceuticals, and pigments, and zinc dust in paints and coatings for corrosion protection.

An important consideration in zinc mining and smelting is the extraction of valuable byproducts that can be credits in the production cost cycle. Most common byproducts include cadmium, germanium, lead, silver, and sulfur, although indium, gallium, copper, and gold may also be recovered. These elements are recovered during ore beneficiation or from residues generated in the zinc roasting and refining process.

Tin: Tin ore is mined in about 35 countries by both placer mining and by lode deposit mining, depending on the region of the world. About 75% of the total world production of tin ore comes from Brazil, China, Indonesia, Malaysia, Thailand, and the USSR. In the United States, there is no substantial amount of tin ore mined, with only a few small mines in Alaska and New Mexico. Almost all tin reserves occur in ores in which tin is the only or dominant metal; some coproducts include antimony, bismuth, columbium, and tantalum.

Until the 1960s, much of the tin concentrates produced in the LDCs was shipped to custom smelters, many of which were in Europe, for processing. However, the trend here, as with lead and zinc, has been for smelters to be constructed within the mining country to achieve the added value. Countries such as Australia, Bolivia, China, and Thailand have increased their smelting capacity in recent years and have become significant exporters of tin metal. As a result, less tin concentrate is traded internationally, and most trade is in tin metal. The United States has long had one primary tin smelter, located in Texas, which has traditionally depended on tin concentrates from several foreign countries, especially Bolivia and Peru. Secondary production of tin is derived from processing wastes and obsolete consumer products such as tin cans.

Up through the mid-1980s, Bolivia and Malaysia tended to be the leading suppliers of tin metal to the United States, but since then, Brazil and China have emerged as the major suppliers to the United States.

Tin metal uses are varied, and most tin uses are as a protective coating or as an alloy with other metals. The major applications are in tinplate, which is used extensively for food cans, and as an alloy with lead (solder), which is widely used as a joining material for metals, especially in electrical and electronic applications. Tin chemicals are used in fluoridated toothpastes, wood preservatives, heat stabilizers for making polyvinylchlorides, and marine paints as an anti-foulant for the hulls of ships.

2.2.2.2 Precious Metals

In addition to the traditional precious metals of silver and gold, the platinum-group metals including platinum, palladium, rhodium, iridium, ruthenium, and osmium are part of this group. Silver and gold have been used since ancient times for coinage, jewelry, and ornaments and as a storehouse of wealth. As monetary metals, their prices were controlled or manipulated by governments. With the demonetarization of gold and silver, their industrial applications have assumed increasing importance. The platinum-group metals have also undergone a change. These metals, sometimes called noble metals, were initially industrial metals used in catalysts and in corrosion resistant materials. In the 1980s, several countries began issuing coins made of platinum.

Silver: Silver is most often found associated with copper, gold, lead, or zinc ores. Although most of the silver in the United States is recovered from silver ores, approximately two-thirds of the world's silver reserves are estimated to be contained in copper, lead, and zinc deposits, with the remainder primarily in silver or gold deposits. Some silver has been found associated with certain aluminum or tin ores or with fluorspar.

Silver-bearing ores are mined using open pit or underground methods. Nearly all US mine production of silver comes from states west of the Mississippi River, mainly Nevada.

Worldwide, over 50 countries produce silver, with the largest producers traditionally being Australia, Canada, Mexico, Peru, the United States, and the USSR. Current treatment of silver-containing ores is almost entirely by a flotation process, which results in the recovery of silver from intermediate products of lead, zinc, or copper smelting. Gold and silver ores are generally treated by cyanidation, followed by precipitation of the metals from solution and electrolytically refined to pure metal.

The major primary silver-refining countries are Australia, Canada, Japan, Mexico, Peru, the United Kingdom, the United States, and the USSR. In the United States, most refiners of primary materials are located in the western states, while most secondary refiners are located in the eastern states, and along the west coast. Old scrap typically provides one-third to one-half of the annual US refinery production of silver. Price plays an important role in the quantity of scrap available for recycling.

Gold: From time immemorial, gold has played a prominent role in world economic and political events. Most of the gold mined or accumulated over the past 6000 years exists mainly in the form of refined gold held by governments as monetary reserve assets or by individuals in the form of jewelry, bullion coins, or small bars held as insurance against currency devaluation. In modern usage, gold is also consumed worldwide in numerous electronic, industrial, and dental applications, but more than three-fourths of the total annual demand goes toward the fabrication of jewelry and the minting of coins.

Although gold has been mined at one time or another in nearly every country in the world, deposits mined in the Republic of South Africa, beginning about 1886, have accounted for about 40% of the world's total cumulative production, estimated at about 3.3 billion troy oz (102.63 kt). By the late 1980s South Africa was producing about 40% of the world's annual production of over 50 million oz (1555 t); the second largest producer, the USSR, accounted for about 15%, followed by the United States with nearly 10%. In the United States, Nevada has been by far the nation's dominant producer during the 1980s.

As a consequence of the rapid and sustained increase in the world price of gold since the mid-1970s plus the development of a generally favorable margin between gold's market price and the cost of its production, exploration worldwide for new deposits has been both intense and highly successful. Gold exploration and mining is pursued by both individual miners and large multinational corporations with interests on several continents. Mining operations range from hand-dug pits and deep underground mines to highly mechanized surface lode or placer mines capable of handling large volumes of low-grade materials. Gold may also be recovered as a byproduct of various base metal deposits, principally copper. Recently, heap leaching, generally combined with open pit mining and relatively simple recovery methods, has become a popular and cost-effective method of extracting gold from very low-grade ores (Chapter 15.2). Crude or impure gold is refined to commercial specifications by chlorination or by electrolysis. Gold recovered in nugget or specimen form generally commands premium prices when sold in specialty markets. Substantial quantities of secondary material such as gold-bearing scrap are processed for recovery as a refined product; however, some material may also be reused directly as in local jewelry fabrication.

Gold is traded widely in all forms with substantial volumes traded daily for future delivery on various world commodity markets. Prominent gold trading centers, such as London, Zurich, New York, Hong Kong, and Singapore, account for much of the trading activity. However, clandestine or smuggled gold, can also account for a substantial portion of the daily movement. Gold contained in secondary or scrap products may be widely traded, especially during periods of heightened demand when prices tend to escalate.

Platinum-group Metals: The two most important metals of the platinum group (PGM) by quantity produced are platinum and palladium. PGM are concentrated in copper-, nickel-, and iron-bearing sulfide minerals in the Bushveld Complex in the Republic of South Africa and in copper-nickel sulfide deposits near Noril'sk in the USSR. Together, South Africa and the USSR account for almost all the world's reserves of PGM, with most of the remainder shared equally between the United States and Canada. In South Africa, PGM ores contain about 47% platinum, 32% palladium, 11% ruthenium, 7% rhodium, 2% iridium, and 1% osmium; while in the USSR, ores contain about 25% platinum, 67% palladium, 3% rhodium, 2% ruthenium, 2% iridium, and 1% osmium.

Together, South Africa and the USSR mine about 94% of the world's annual production of PGM. Almost all processing and refining is done within the borders of each country, although South Africa has historically relied on the United Kingdom for some of its PGM refining. In South Africa, the ore is processed in such a way as to produce byproduct copper, nickel, and cobalt, in addition to all the precious metals.

Secondary refining of PGM is extensive because of the high value of the metals. Secondary recovery is highest in the petroleum refining, chemical, glass, and the dental industries. Less recycling occurs from used automobile catalytic converters, electronic components, and jewelry. In the case of automobile catalytic converters and electronic components, the scrap is widely scattered and/or low in concentration, thus adding to the cost of processing and refining.

The PGM are refractory and chemically inert at elevated temperatures to a wide variety of materials, display excellent catalytic properties, and are attractive in appearance. These properties are the basis for the principal uses: as catalysts in the automotive, chemical, and petroleum refining industries; as corrosion-resistant materials in the chemical, electrical, glass, and dental-medical industries, and as beautiful objects and collectibles in the jewelry and investment industries. In the western world, 35% of the platinum is consumed annually in automobile catalytic converters, 30% in jewelry, 15% in investment, and the remaining 20% for all other uses. The investment uses of platinum came into being in 1982. In the case of palladium consumption, the electrical industry uses 50%; the dental industry, 31%; and all other industries, 19%. Rhodium usage is about one-tenth that of platinum, with about three-quarters going to automobile catalytic converters. About half of ruthenium usage is for electronics, one-third for electrochemical uses, and the remainder to other uses. For iridium, about one-quarter of the usage is evenly divided between the electrochemical industry and the petroleum refining industry, and three-quarters to other industries. Consumption of osmium is very low, mostly for the medical industry.

2.2.2.3 Ferrous Metals

Iron and steel are the fundamental metals of every modern industrialized country in the world. These commodities are critical to manufacturing, mining, construction, energy production, transportation, and agriculture. Iron is the least expensive and most widely used metal.

Iron: Iron ore is mined extensively in Africa, Australia, Canada, China, South America, the United States, and the USSR. Major world producers of iron and steel include the European Economic Community, Japan, the United States, and the USSR. About 98% of the iron ore produced in the United States is used for the manufacture of iron and steel. Most iron ore is mined from open pits. Continuously charging iron ore along with coke and limestone into a blast furnace is the primary method of iron production. In the United States, the main method used to convert molten blast furnace iron into steel is the basic oxygen furnace. Some open hearth furnaces are also used for this purpose.

US production of iron ore is based mainly on low-grade ore of the taconite type. These taconite pellets made up almost all US production in the 1980s, with very little direct-shipping ore being produced. In contrast, there was no commercial production of pellets 30 years earlier and direct-shipping ore made up almost three-fourths of US production. The change was brought about by depletion of reserves of direct-shipping ore and increased reliance on taconite to maintain production. Similar changes occurred in Canada and the USSR, but ore grades remain high in Australia, Brazil, India, and South Africa. Almost three-fourths of the world's production of iron ore is government-controlled.

The US iron and steel industry has reduced the amount of energy required per ton of raw steel production by about 20% between 1972 and 1988. During this period, the steel industry shifted its energy consumption away from coal and fuel oil toward natural gas and electricity. This shift occurred principally because of the increased fraction of steel produced by electric furnaces. Installation of heat recuperation systems has also reduced energy requirements.

About 99% of world demand for iron ore is from the iron and steel industry. Iron ore demand from steel production has declined because the proportion of steel produced in scrap-based electric furnaces has increased, adoption of continuous casting in integrated plants has resulted in higher yields, and a number of other technical advances have increased scrap consumption in the basic oxygen furnace.

Up to 10% of the tonnage of iron ore and agglomerates annually consumed in blast furnaces at US iron and steel plants is derived from secondary materials such as flue dust, mill scale, and steel-furnace slag. Ferrous scrap is also charged to blast-, steel-, and foundry-furnaces in varying amounts.

Manganese: The USSR is the world's largest producer of manganese ore. Manganese is an essential element in steelmaking and iron castings production. These industries, chiefly steelmaking, account for over 90% of manganese consumption. Manganese usage in the production of ferrous alloys is mostly as an intermediate product such as a manganese ferroalloy into which the ore is first smelted. Relatively small amounts of manganese are used in the form of metal to alloy with several nonferrous metals, chiefly aluminum. A variety of nonmetallurgical uses exist for manganese, mostly as natural and synthetic oxides and manufactured chemical compounds. Manganese dioxide is a key ingredient in dry cell batteries.

Smelting of ore into ferroalloys is becoming a more significant activity for ore-producing countries, especially Brazil and the Republic of South Africa. Among nonore-producing countries, France, Japan, and Norway are still leading world producers of manganese ferroalloys. The USSR is the world's largest producer of ferromanganese. Manganese smelting into ferroalloys has dwindled to a fraction of the scale once practiced in the United States. Since the late 1970s, imports of manganese in upgraded forms, such as ferroalloys and metal, exceeded those of manganese in ore.

The need for manganese in steelmaking and lack of significant economic domestic deposits make security of supply a national concern in such principal steel-producing areas as the United States, Japan, and Western Europe. The United States has been totally dependent since 1970 on foreign sources for ore products containing at least 35% manganese in addition to having lost most of its capability for producing manganese ferroalloys.

The strong reserve positions in 1990 of the Republic of South Africa and the USSR (Table 2.2.3) help explain the central importance of these two countries to present and future manganese supply. Currently, world trade in manganese is mostly either between market economy countries or between centrally planned economy countries, with relatively small flows between the two types of economies.

Manganese is not recycled per se, but rather is reprocessed mainly as a constituent of iron and steel scrap, steel slag, and aluminum alloys. These materials are recycled primarily for their contents of iron, slagging ingredients, and aluminum, respectively. The net effect on manganese supply is minor.

Chromium: Chromium is used primarily as an alloying element. It is also used in substantial quantities in the chemical and refractory industries. About one-half the chromium consumed in the United States is for the production of stainless steel. Other metallurgical uses of chromium include alloy steel (except stainless), superalloys (nickel, iron-nickel, and cobalt base alloys), and in copper, aluminum, and titanium alloys.

Chemical uses of chromium include wood preservation, pigments, plating solutions, and leather tanning. A small amount of chromium is used to make refractory bricks.

Chromium is mined in the form of the mineral chromite in about 20 countries, mainly the Republic of South Africa and the USSR. In the metallurgical industry, chromite is smelted in electric furnaces to produce ferrochromium that is subsequently used in iron-base or iron-containing alloys. Ferrochromium is also used to make chromium metal by an electrolytic process that is used for making alloys that do not contain iron. In the chemical industry, chromite is roasted with soda ash to produce sodium bichromate, the starting point for the manufacture of numerous chromium-containing chemicals. Secondary chromium from stainless steel scrap accounts for about 20% of US consumption.

Ferrochromium was originally produced in the major stainless steel producing centers of Europe, Japan, and the United States. Chromite-mining countries have installed ferrochromium smelters, while traditional producers have been closing excess capacity. This trend of vertical integration of the chromium industry in chromite-producing countries permits those countries to earn more from their resources.

Nickel: Nickel is primarily used in alloys with other elements, where it adds strength and corrosion resistance over a wide range of temperatures. About one-half the nickel used is for stainless steel. Other uses include electroplating, chemicals, superalloys, and magnets.

About 80% of the nickel reserves are found in laterite deposits, with the balance in sulfides ores. Laterites are highly weathered ultramafic rocks, which can contain nickel, cobalt, iron, and other elements. Typically, ferronickel is produced from laterite deposits with the attendant loss of any cobalt or other metals. Nickel sulfide deposits usually contain copper, cobalt, and sometimes platinum-group metals. Although most nickel reserves are in laterites, almost three-quarters of nickel mining is conducted on sulfide deposits that are readily amenable to concentration by flotation and can be reduced to metal using less energy than with laterite material. No effective method exists to concentrate nickel laterite ore early in the stages of processing.

Aside from small quantities of byproduct nickel produced from copper smelting, there is no US production of nickel. Secondary production, derived primarily from stainless steel scrap, provides about one-quarter of the total annual demand of nickel in the United States. Canada and the USSR are the largest producers of nickel, together mining and refining about 45% of the world's nickel. Australia mines and refines 11% and 6% of the world's nickel, respectively. Norway refines nickel matte produced in Canada and elsewhere. Indonesia and New Caledonia, which refine little nickel, together mine 15%. Japan has no mining capacity but refines 13% of the world total.

Cobalt: The primary use of cobalt metal is in superalloys. These alloys were developed for use in high temperature and corrosive environments. The major application of superalloys is in gas turbine engines for aircraft and industrial use. Cobalt is also used in cemented carbides for cutting and wear-resistant tools and in magnetic alloys for permanent magnets in electrical equipment. The chemical industry consumes significant quantities of cobalt in catalysts, in paints, as inorganic pigments and decolorizers for glass and ceramics, and in various other applications. The United States is a major cobalt consumer, using about one-third of the world's supply annually.

Cobalt is almost always mined as a byproduct of other more abundant metals, usually copper or nickel. More than one-half of the world's supply of cobalt comes from the copperbelt of Zaire and Zambia, where it is mined and refined by government-owned companies. The USSR is a major cobalt producer, mostly as a byproduct of its nickel mines, but is a net importer. Canada produces cobalt, also as a byproduct of nickel mining.

In terms of substitution, cobalt-base superalloys can be replaced by nickel-base superalloys in some applications, and cobalt levels can be reduced in some nickel-base superalloys. Magnetic alloys made of iron, neodymium, and boron can replace cobalt magnets in most cases.

A major transportation issue relating to cobalt is the export routes from Zaire and Zambia. Within Zaire, metals are transported over a complex system of rivers, roads, and railroads. Presently, Zairian cobalt is exported through the Republic of South Africa. Zambia is able to export its cobalt by railway to the port of Dar-es-Salaam in Tanzania. The political situation in southern Africa can have a tremendous impact on the supply, and therefore the price, of cobalt.

Presently, domestic production is limited to recovery of cobalt from scrap which meets almost one-fifth of US demand. Superalloy scrap generated during the making of superalloys, machining of engine parts, or from the replacement of components in jet engines is processed and reused in the production of new superalloys. Other sources of cobalt-bearing scrap are cemented carbide scrap, spent petroleum catalysts, and grindings from the production of cobalt-containing magnets.

Molybdenum: Molybdenum occurs in nature only in combination with other elements. The most common occurrence is molybdenite (molybdenum sulfide), which is widely distributed throughout the world. Commercial deposits have been identified in about 25 countries. About 90% of the total production comes from Canada, Chile, the USSR, and the United States.

As a result of the concentration of mine production capability, international trade in molybdenum materials consists primarily of exports from Chile, Canada, and the United States to industrialized nations that lack mine production. The major importers are West European countries and Japan. The United States exports more than 50% of its mine output, mostly as molybdenum concentrate or oxide.

Molybdic oxide is produced in several grades that are the precursors for all types of molybdenum salts and compounds, and it is also added to steel, cast irons, and superalloys, either with the furnace charge or to the molten bath, to make various alloys. It enhances hardenability, strength, toughness, and wear and corrosion resistance. Purified molybdenite concentrate is also used in the manufacture of molybdenum disulfide lubricants.

Columbium and Tantalum: Columbium (niobium) is a refractory metal used mostly as an alloying element in steels and superalloys. The largest demand for columbium in superalloys has been in a nickel-base superalloy which is used for a number of jet engine parts. Over 40% of the columbium consumed in the United States is used for construction purposes, such as high-strength steels in permanent structures. The next most important use of columbium is in the transportation and aerospace industries with about 25% of the total, followed by the oil and gas and metalworking industries. Tantalum is a refractory metal with approximately 60% of US consumption in electronic components, mainly capacitors. The remainder is consumed in the aerospace and other transportation applications, in the form of tantalum carbide in the metalworking industry, and in chemical processing equipment.

The major producers of columbium mineral concentrates are Brazil, Canada, and Thailand, while tantalum is produced mainly in Australia, Brazil, Canada, and Thailand. Tin slags containing columbium and tantalum are produced mainly by Brazil, Malaysia, and Thailand. Beginning in 1975, synthetic concentrates from the Federal Republic of Germany became an additional source material for columbium and tantalum production. The synthetic concentrates are produced from low-grade tin slags rather than from mined ore. The United States continues to be a major processor of imported feedstock to make columbium and tantalum end products. Brazil is the largest producer of ferrocolumbium for steelmaking. Large amounts of ferrocolumbium are also produced in Japan, the United States, the USSR, and Western Europe, using largely imported raw materials.

Australia, Brazil, and Thailand account for over 70% of the free-world production of tantalum feed material. Thailand is the major source, accounting for about 40% of total production. Thailand's production comes mainly from tin slags derived from the smelting of tin concentrates.

As domestic columbium and tantalum resources are of low grade and not currently commercially recoverable, the United States depends on imports for its columbium and tantalum feed material supply. Processing facilities in the United States are capable of producing about 50% of US demand for columbium products. However, columbium and tantalum processing is being increasingly conducted in countries in which the feed materials originate, such as in Australia, Brazil, and Thailand. Brazil and Canada account for 90% of total US columbium imports, and Thailand accounts for about 30% of total tantalum imports.

Tungsten: Tungsten ores are mined in approximately 30 countries covering nearly all continents. Occurrence of the tungsten-bearing mineral is usually in association with minerals of copper, tin, bismuth, or molybdenum. Most tungsten concentrate is converted to ammonium paratungstate, metal powder, tungsten carbide powder, ferrotungsten, or other tungsten chemicals before being consumed in end-use products. The exception is high-purity scheelite concentrate, which is added directly to steelmaking processes.

China has become a major supplier of tungsten materials to the United States, as US mine production declined drastically during the 1980s to just several hundred tons of tungsten per year. Accompanying this increase in imports from China, trade problems related to price and oversupply have developed requiring special agreements between the two countries. About one-third of the US supply is estimated to be derived from recycled material, mainly cemented tungsten carbide parts.

Because of tungsten's outstanding physical characteristics of high melting point and high density, a significant quantity is used as tungsten carbide in cutting and wear-resistant applications and is generally pressed and sintered (cemented) with cobalt powder for such purposes. The extreme hardness of cemented tungsten carbide at temperatures exceeding 1830°F (l000°C) makes it a preferred metalworking material for the cutting edge of machine tools. Mill products made from tungsten metal powder are used as lighting components and electrical contact points. Wrought tungsten metal is employed as heat and radiation shielding, welding electrodes, and heating elements. Tungsten is also used as an alloy constituent in stainless steels, superalloys and nonferrous alloys. Nonmetallurgical uses of tungsten chemicals include phosphors, catalysts, corrosion inhibitors, and fireproofing agents.

Vanadium: Vanadium occurs in uranium-bearing minerals of Colorado and Utah, in copper, lead, and zinc vanadates of Africa and with certain phosphatic shales and phosphate rocks in the western United States. Certain petroleum crude oils, especially those from South America, contain varying amounts of vanadium compounds. The vanadium is concentrated into the coke, fly ash, and boiler slag upon refining and combustion of the crude oils.

Most of the vanadium reserves are in deposits in which the vanadium would be a byproduct or coproduct with other minerals, including iron, titanium, phosphate, and petroleum. Vanadium is recovered domestically as a coproduct or byproduct from uranium-vanadium ores, and from ferrophosphorus slag as a byproduct in the production of elemental phosphorus.

The steel industry accounts for the majority of the world's consumption of vanadium as an additive to steel in the form of a ferrovanadium alloy. The addition of vanadium to steel produces a fine grained steel which exhibits greater toughness, hardness, impact and wear resistance, weldability, and good high temperature strength. The principal application of vanadium in nonferrous alloys is in a titanium alloy, which is important in the production of supersonic aircraft where strength-to-weight ratio is a primary consideration. Vanadium compounds are used in several permanent-magnet alloys containing cobalt, iron, and nickel. Vanadium metal and several vanadium compounds are used as catalysts in certain chemical and petrochemical reactions.

About one-half the total world reserves of recoverable vanadium is contained in titaniferous magnetite-ilmenite ores, certain magnetite-hematite ores, and titaniferous iron sands. The largest reserves of magnetite-ilmenite ores are in the Bushveld Igneous Complex of the Republic of South Africa and the Kachkanar Gabbro-Pyroxenite Massif of the USSR. Other countries with reserves of vanadium-bearing magnetite include Australia, Brazil, Chile, China, Finland, and India. About 6% of the total reserves are contained in crude petroleum and tar sands.

Silicon: Silicon never occurs free in nature but combines with oxygen and other elements to form oxides or silicates. Deposits of silicon-bearing minerals are widely distributed around the world.

Many countries produce ferrosilicon and silicon metal; however, Brazil, Norway, the United States, and the USSR account for over one-half of all production based on silicon content. Supply and demand of silicon-containing alloys and metal are related primarily to the requirements of the iron and steel, aluminum, and chemical industries, and prices are influenced by the cost and availability of transportation and energy. Ferrosilicon and silicon metal producers are generally located near a relatively inexpensive hydroelectric power source. Raw materials are usually obtained from within a 500-mile (805-km) radius of the plant. Economic factors such as accessibility to low-cost energy, transportation costs, and a ready market for the product determine resource development. Although US silicon resources are abundant, the United States imports ferrosilicon and metallurgical-grade silicon metal because smelting costs are lower in some other countries.

Most silicon is used as ferrosilicon for deoxidation and alloying in iron and steel production. Another significant portion of silicon demand is the silicon metal needed in the aluminum industry for alloying and to improve casting fluidity. The chemical industry uses silicon metal as the raw material for the manufacture of silanes from which silicones and semiconductor grade silicon are produced.

2.2.2.4 Light Metals

In the age of energy conservation and aerospace, light metals have achieved a prominent place in our industrial society. These metals are relatively new in their industrial applications, having been isolated in elemental form less than 200 years ago.

Titanium: The present mineral sources of titanium are rutile (93 to 96% titanium dioxide, TiO_2), ilmenite (44 to 70% TiO_2), and leucoxene, an alteration product of ilmenite, with up to about 90% TiO_2. Ilmenite is produced from both hard-rock and sand deposits, but rutile is currently produced only from sand deposits. A large proportion of the world's ilmenite production is upgraded by smelting or chemical processing to form high-TiO_2 slag or synthetic rutile.

Titanium dioxide pigment is produced commercially by either the sulfate or the chloride process, requiring either ilmenite or rutile, respectively. The major environmental concern in the titanium industry is the disposal of waste solutions from the sulfate pigment process that contain sulfuric acid and iron sulfate. To minimize this problem, US producers of sulfate process pigment operate waste treatment plants that neutralize such effluents with limestone and lime, and produce gypsum and iron oxide byproducts.

Principal world producers of ilmenite are Australia, Canada, Norway, the Republic of South Africa, the United States, and the USSR. Main producers of rutile are Australia, Sierra Leone, and the Republic of South Africa. Titanium metal is produced mainly by China, Japan, the United Kingdom, the United States, and the USSR. Scrap titanium generally accounts for about one-half the ingot production in the United States. Major producers of TiO_2 pigments are France, the Federal Republic of Germany, Japan, the United Kingdom, and the United States.

Because of the high strength-to-weight ratio of titanium alloys and their resistance to corrosion, titanium metal is an important strategic and critical material and is widely used for high performance military and commercial aircraft in both airframes and engines, in electric generating plants, and for a wide variety of chemical processing and handling equipment. Titanium metal is produced by highly sophisticated chemical processes and is more expensive than most other structural metals such as aluminum and steel. Only about 5% of the world's annual production of titanium minerals goes to produce titanium metal. The remainder is used primarily to make white TiO_2 pigment. Because of its whiteness, high refractive index, and resulting light-scattering ability, TiO_2 is the predominant white pigment for paints, paper, plastics, rubber, and various other materials. Other applications of titanium and its minerals include use in welding rod coatings, titanium carbides, ceramics, and chemicals.

Aluminum: Aluminum has been produced in commercial quantities only in this century. This versatile metal of widespread use is exceeded only by iron in world consumption.

Aluminum metal is produced in electrolytic cells by reduction of high-purity aluminum oxide (alumina), an intermediate-stage raw material produced by refining bauxite ore. Although the principal use of bauxite is in the production of aluminum, up to 15% of world production is consumed in refractories, abrasives, chemicals, or in alumina for nonmetallurgical uses. Since 1985, virtually all bauxite mined in the United States has been processed for specialty uses other than aluminum metal production.

The major producers of primary aluminum metal, including those in the United States, are dependent on imports for most of their bauxite and alumina. Large quantities of electrical energy supplied at a constant rate without interruption are required for the reduction of alumina to aluminum metal; thus aluminum smelters are built near sources of abundant electric energy. Until the early 1970s, aluminum metal production was located primarily in developed countries, close to sources of low-cost energy. When the cost of energy increased in the 1970s, metal production shifted to areas where the energy could still be obtained relatively cheaply. New major aluminum plants were built in Australia, Brazil, Canada, and Venezuela.

The United States continues to be the world's largest consumer of aluminum metal. Domestic production, however, has not been capable of meeting demand. Imports of aluminum ingot, semifabricated materials, and scrap supplement domestic production. Canada has traditionally been the largest supplier of metal to the United States. The largest domestic market for aluminum metal is the container and packaging industry. Aluminum metal is also widely used in transportation, building and construction, electrical applications, consumer appliances, and machinery and equipment.

While low cost and ease of fabrication contributed to the tremendous growth in the use of aluminum beverage cans, the recyclability of aluminum played an even more important role. Approximately 50% of all aluminum beverage cans are recycled in the United States. Cans have been the major reason behind the tripling of secondary aluminum production during the past decade (Table 2.2.4). Plastics and paper compete with aluminum for the container and packaging market. Aluminum's high strength-to-weight ratio has encouraged its use in the aircraft and aerospace industry. Aluminum's use in transportation has grown as manufacturers seek ways to reduce the weight of land vehicles to improve fuel efficiency.

Magnesium: Magnesium metal and magnesium compounds are produced from seawater, well and lake brines and bitterns, as well as from minerals such as magnesite, dolomite, and olivine. The largest tonnage use of magnesium in compounds is in refractories consumed primarily by the iron and steel industry. Magnesium compounds are also used in such varied materials as animal feed, cement, rubber, rayon, pharmaceuticals, and ceramics. Olivine, a magnesium-iron silicate, is used mainly as a foundry sand. The primary use of magnesium metal is as an alloying addition to aluminum. Aluminum-magnesium alloys are used in products such as two-piece beverage cans, automobiles, aircraft, and machinery. Magnesium metal and its alloys are also used as structural components in transportation equipment and in machinery.

China, North Korea, and the USSR account for almost one-half the world's magnesite production. China, Ireland, Israel, Japan, Mexico, the United Kingdom, and the USSR produce magnesium compounds from seawater. The United States is the largest producer of magnesium metal from seawater, brines, and magnesite, and with Norway and the USSR, accounts for about three-quarters of the world's magnesium output. Brazil, Canada, China, France, Italy, Japan, and Yugoslavia are the other magnesium metal producers.

Until the 1980s, the United States supplied most of its need for magnesium compounds with domestically produced material. With the decline in the US iron and steel industry in the late 1970s and the rise in the value of the US dollar, many magnesium compound producers shut down capacity or operated at significantly reduced capacities. Imports, principally from China and Greece, have replaced domestically produced magnesia in some of the refractory markets.

As the world's largest magnesium metal producer, the United States has been a net exporter of magnesium for many years. Domestic production, including secondary magnesium from scrap, have been sufficient to supply US demand, although the United States imports some magnesium, primarily from Canada and Norway.

Beryllium: This metal has become a necessary industrial metal only over the last 60 years because of its high strength, low weight, and high thermal conductivity. It is used in the form of metal, beryllium-copper alloys, or oxide. About two-thirds of consumption is in the form of copper alloys that have electrical, electronic, and aerospace applications. The metal accounts for about 20% of consumption in aerospace and defense applications, with the remainder being used in the form of beryllium oxide ceramics for electronics.

In the United States, beryllium resources consist primarily of bertrandite, while in the rest of the world the resource is beryl. Principal world beryl producers are Brazil, China, and the USSR, but the United States is the only integrated producer of beryllium products, including beryllium metal. Reserves exist in only about a dozen countries. Energy requirements for the production of beryllium materials are high, as processing requires the use of induction furnaces that consume large quantities of energy. Because of its high cost and potential health and environmental problems, beryllium materials are specified only when there is a clear technical and economic advantage over other materials.

2.2.2.5 Construction Materials

Construction materials are for the most part industrial minerals. They are high-volume, low-unit-value materials that are usually mined and consumed locally. Processing of the mined material to a usable product is usually a simple process. Imports tend to be a small part of supply and recycling represents a small but growing part.

Sand and Gravel: Sand and gravel, one of the most accessible natural resources, is used mostly for construction purposes, mainly as aggregate in concrete, as road material, and as fill. A significantly smaller volume of high purity sand known as industrial sand or silica sand is used in the glassmaking, foundry, abrasives, and oil industries. Because the uses, specifications, prices, producers, and consumers of construction sand and gravel differ substantially from those of industrial sand, the two types are treated separately.

Construction sand and gravel is a low-value product characterized by its "place value," a term that describes the importance of the location of the geologic deposit in reference to the market. The industry is geographically widespread and self-contained. Mine production is directly related to the amount of construction work in any area. But despite its low unit value, the construction sand and gravel industry in the United States showed a significant growth in production, from 209 million tons (190 Mt) in 1928 to about 882 million tons (800 Mt) in 1988. Annual sand and gravel production tonnage ranks second in the industrial minerals industry.

Construction sand and gravel is the only mineral commodity produced in all 50 states. In 1986, about 4300 companies with about 5800 operations were active in the United States. The individual operations range in size from those producing less than 11,000 tpy (10 kt/a), to those up to millions of tons annually. While most of the operations are relatively small, turning out one product or a limited range of products, most of the tonnage comes from large operations.

The total sand and gravel resources of the United States are very large. However, the geographic distribution and/or the quality of reserves and the environmental as well as zoning restrictions imposed by society are a very significant factor in determining if a particular deposit can be developed for production.

Sand and gravel are usually used together, but some uses require only sand or only gravel. Sand is used in mortar as concrete aggregates, plaster, gunite, and for snow and ice control, while gravel is used in drainage control and as covering and stabilizer on unpaved roads.

US consumption of sand and gravel almost equals that of production because of the very small export trade. The construction industry is by far the greatest consumer and, therefore, the primary factor that determines the demand and supply of sand and gravel for any given period.

The sand and gravel industry is a very competitive industry, largely because sand and gravel are abundant in most areas of the country. Production costs for sand and gravel are determined to a large extent by the cost of labor, equipment, energy, and water, in addition to the cost of compliance with the environmental and safety regulations. These costs vary depending upon the nature of the deposit, the geographic location, and the type and number of products produced. But despite the rising cost of labor, energy, and land value, the competition among sand and gravel producers has kept the relative cost of sand and gravel low. Increased operation costs have been somewhat offset by automation and other means of increasing operating efficiency. The availability of automated and therefore relatively expensive equipment is very important. For this reason, large companies are able to better compete in the sand and gravel business. Smaller, less efficient operations become less economical as operation costs, as well as costs associated with meeting various federal, state, and local regulations, continue to increase.

Stone: Despite the low unit value of crushed stone, the industry in the United States showed a tremendous growth in production, from 130 million tons (118 Mt) in 1928 to about 1.2 billion tons (1.1 Gt) in 1988, mainly due to rapidly increasing demand for railroad, highway, and other construction work. The amount of crushed stone produced is greater than that of any other mineral commodity mined in the United States.

The crushed stone industry is widespread and varies by size of operations, kind of stone, and type of material produced. The size of individual companies ranges from small independent producers with one quarry to large diversified corporations with 50 or more crushed stone operations, from which most of the production comes.

Reserves of stone have not been systematically investigated, partly because of the size of the undertaking and partly because a true measure of the total reserves available held by stone producers would have no particular value when compared with total potential resources, which are, on a national basis, unlimited.

Crushed stone in its different forms is used in a wide variety of applications in industries including construction, chemical, metallurgical, and agriculture. The most common use of crushed stone is for construction purposes. About 65% of the total US production of crushed stone is used as construction aggregates in residential and nonresidential construction and highway construction, of which 60% is unconsolidated aggregates and 40% is concrete and bituminous aggregates.

About 12% of the total crushed stone, mostly limestone, is used for cement and lime manufacturing and 2% for agricultural purposes. The remaining 21% is used for a wide variety of applications ranging from metallurgical flux to manufacture of glass, ceramic pottery, paper, and as fillers and extenders in asphalt, paint, rubber, plastics, and abrasive. A growing amount of limestone is being used in removal of sulfur oxides from stack gases, primarily from coal-burning electric generating stations, and for mine dusting to enhance mine safety by reducing explosion risk of highly combustible coal dust.

Cement: Portland cement is produced by nearly 2000 plants in more than 130 countries around the world. China is the world's largest producer followed by the USSR, the United States, and Japan. Together these four countries account for nearly two-thirds of the world's production. Portland cement is produced by pulverizing clinker consisting essentially of hydraulic calcium silicate and containing various forms of calcium sulfate. Limestone comprises about 85% of the raw material for making cement. World cement capacity was about 1.3 billion tons (1.2 Gt) in 1988, 40% of which is concentrated in Asia, with China and Japan accounting for one-half the total. In the United States, production capacity has been declining since 1985. The decline is attributed to closure of older, less efficient, more energy intensive plants that can no longer compete with more modern plants, both domestic and foreign.

Because cement is a high-bulk low-cost item, it was not always regarded as a commodity that would be traded internationally. However, due to the cyclical nature of cement consumption, many producers in highly developed countries found that domestic demand was not keeping pace with supply, thereby resulting in excess capacity in their own countries. Because of this overcapacity, they began to seek outside markets for their cement. Generally, lesser developed nations with little or no capacity were targets for their excess cement. Some developing countries, notably Mexico and Venezuela, became major producers and exporters. In the early 1980s, however, massive plant construction activity in Middle Eastern countries, and parts of Africa and Asia, made these countries more self-sufficient in cement production. This, coupled with plant construction and modernization projects in North America and Western Europe, contributed to excess world capacity.

The United States has the fastest growing and most stable market of all cement-producing nations and thus has been the growing target of foreign cement producers. Because plant construction is designed to serve strategically defined markets, the cost of transportation becomes an integral part of the planning process. It is uneconomical for cement to be transported long distances overland, thereby causing terminals to be constructed to serve as transfer points to supply markets that cannot traditionally be served by plants. Terminals located near water make it possible for producers to serve more distant domestic markets. At the same time, these terminals make it easier to bring in low-cost imports, thereby increasing the industry's reliance on foreign cement to meet their customer's needs. Shipment of cement from one country to another has become quite prevalent. It is estimated that almost 77 million tons (70 Mt) of cement is traded by world exporting countries each year. The United States is the largest recipient of imported cement receiving about one-quarter of the total. Portland cement manufacturing is one of the most energy intensive of all US industries. Shortages of petroleum and natural gas and escalating fuel costs in the 1970s caused producers to shift to coal and petroleum coke. In addition, countries such as Mexico that provided low-cost fuel to cement manufacturers became international competitors for cement.

Gypsum: Gypsum is the most common of the naturally occurring sulfate minerals, usually associated with limestones, shales and sandstones, marls, and clays. Crude commercial gypsum is generally high-grade material, the major portion of which can be utilized with no beneficiation.

About 80 countries are known to produce gypsum. Because of its wide distribution and plentiful supply, most of the world's production is consumed domestically. Exceptions are Canada and Mexico which export significant portions of their production to the United States; Thailand and Australia which export to much of the southeast Asia market; and Spain which exports principally to the United States and the Scandinavian market.

Crude gypsum is marketed for use in cement, agriculture, and fillers. Commercially calcined gypsum is a manufactured hemihydrate product produced by partial calcination of gypsum. Commonly called plaster of paris, when water is added to form a paste, it quickly sets and hardens to form gypsum again. In most industrialized countries, the major use of gypsum is in the manufacture of prefabricated wallboard products. In developing countries, most gypsum is used in crude form for manufacturing cement. The United States is the leading producer and consumer of gypsum. The domestic gypsum industry is a large, integrated industry in which a few large companies are predominant. These companies operate gypsum board plants with captive mines as a domestic raw material source, but imports of crude ore from Canada and Mexico are substituted for domestic crude gypsum for those plants located on the eastern and western seaboard.

Asbestos: Asbestos is mined in approximately 30 countries. The leading producer of asbestos is the USSR, which accounts for 60% of the world asbestos production. Other countries with significant asbestos production are Brazil, Canada, China, Italy, the Republic of South Africa, and Zimbabwe. The United States accounts for only 1% of the world asbestos production.

Chrysotile, the most common variety of asbestos, is mined in more than 25 countries, with Canada and the USSR accounting for over 70% of the world's production. The Republic of South Africa is the only country producing significant quantities of amosite and crocidolite, two other varieties of asbestos.

Asbestos is used in asbestos-cement products, coatings and compounds, friction materials, packing and gaskets, paper, plastics, roofing materials, and textiles. The properties of asbestos that make it useful in this wide variety of products are high-strength, good chemical and thermal stability, high flexibility, low electrical conductivity, and large surface area.

The largest markets for chrysotile are in asbestos-cement pipe, friction materials such as brakes and clutches, packing and gaskets, and roof coatings. These end uses account for over 80% of the US asbestos consumption. Crocidolite is used primarily in asbestos-cement pipe and accounts for only 1% of total US asbestos consumption.

Domestic demand for asbestos is satisfied primarily by imports, mainly from Canada. Demand for crocidolite is filled through imports from the Republic of South Africa.

Concern over possible health risks associated with asbestos exposure has resulted in an overall decline in the worldwide demand for asbestos in recent years. The most dramatic change has been in the United States, where there has been a tenfold decrease in asbestos consumption since the health issue was raised.

Asbestos consumption has declined in product categories where asbestos substitutes or alternative products are most readily available. These substitute materials are metal, mineral, organic, and glass fibers. In addition to direct substitution, manufacturers also are using alternative products, such as ductile iron pipe, reinforced concrete pipe, wood, or aluminum siding, to replace asbestos-containing products.

Lime: Lime, a manufactured product never found in a natural state, is produced by means of a process known as calcination of limestone or other high-calcium materials. Lime is a basic chemical used for many purposes and is produced in more than 70 countries. The USSR is the leading producer, followed by the United States, Japan, the Federal Republic of Germany, and Brazil. Lime is made from a variety of naturally abundant calcareous materials, such as limestone, marble, dolomite, chalk, shell, coral, aragonite, or byproduct sludge from paper mills, carbide plants, or other industrial plants. Lime in various forms is used in steel refining as a flux to remove impurities, in refractory brick furnace linings, and in a variety of metallurgical processes.

Environmental uses of lime include softening and clarification of municipal potable water, pH control and clarification in sewage treatment, neutralization of mine and industrial wastewater discharges, flue gas desulfurization serving utility and industrial plants, and stabilization of sludges from sewage and desulfurization plants before disposal.

Raw materials for lime manufacture are relatively inexpensive. The manufacturing process is energy intensive and thus expensive compared to raw material costs. The lime industry has one of the highest ratios of energy costs to total material costs found in any manufacturing process. The availability of economic and continuing supplies of fuel used mainly in the calcining operation has been one of the industry's principal concerns. In the past, rising oil and natural gas prices have driven the industry to increase energy efficiency and convert to lower cost fuel, especially coal.

The cost of transportation is also a large factor in the marketing of lime. Market areas and established end-use patterns are very critical to decisions on lime plant locations. Transportation costs restrict international trade in lime. When a demand for lime develops at a distance from an existing production facility, but near raw materials and a satisfactory energy supply, it is sometimes more economical to build a new plant rather than ship the material long distances.

2.2.2.6 Agricultural Commodities

Agricultural minerals, or fertilizers, are bulk commodities with worldwide trade patterns. No living organism can exist without phosphorus, potassium, or nitrogen. Phosphorus must be present in adequate amounts in living cells before cell division will take place. This nutrient has vital functions in photosynthesis, utilization of both sugars and starches, and in energy transfer processes.

Phosphate: Although phosphate minerals are common throughout the world, occurrences that can be profitably beneficiated to a marketable product are limited to Africa, China, the Middle East, the United States, and the USSR. China, Jordan, Morocco, the United States, and the USSR are the principal world producers of marketable phosphate rock. US production of phosphate rock is mainly in Florida. It is estimated that over 90% of production is used to manufacture phosphate fertilizer chemicals. The balance, principally produced by electric furnace smelting, is used to manufacture detergents and other industrial chemicals. Most of the countries with reserves have chemical complexes to convert phosphate rock into phosphoric acid and downstream fertilizers, that is, triple or single superphosphates or ammonium phosphates.

Potash: Potash deposits are not widely distributed throughout the world. Less than 20 countries have deposits of significant size and only two countries have large deposits. Canada and the USSR account for about 60% of the world's production from large, underground, bedded deposits of sylvinite. Most of the rest of the mines in the world are of the same type. Manufactured forms of potash are potassium sulfate and potassium nitrate.

Potash is traded widely and is used primarily as an agricultural fertilizer. Approximately 5% of the world's production enters the industrial chemicals industry. Potash is converted by the chloralkali industry into potassium hydroxide which is the gateway into industrial chemicals containing potassium.

There is a tendency for potash mines to be state-owned, which means that, when supply exceeds demand, the state-owned mines may continue production to maintain employment. LDCs tend to develop marginal mines to create employment, discourage imports, and earn foreign exchange from exports.

US imports of muriate of potash (sylvite) from Canada began in 1962 and have increased on a consistent basis to nearly 95% of total imports and nearly 90% of US apparent consumption. The Canadian mines are closer to the largest consuming region of potash, the US corn belt, than are the US mines. The US producers continue to export to Mexico and the Caribbean because of their natural geographical advantage to those regions. China, India, and Japan are completely dependent on imported potash. Canada, the Federal Republic of Germany, the German Democratic Republic, and the USSR supply most of the potash to the international potash market.

Nitrogen: The saline nitrate deposits of the Atacama Desert of northern Chile are the largest known geologic occurrences of mineralized nitrogen. With the phenomenon of world population explosion and increasing requirements for food and fiber during the past 150 years, nature's bank of available nitrogen has long been exceeded. Commercial production of fixed nitrogen as anhydrous ammonia (82.2% nitrogen) is now the principal source for global nitrogen fertilization. Fundamentally, the process of combining nitrogen from the air with a hydrogen source (usually natural gas) to produce anhydrous ammonia has changed little since commercialization in 1913.

Anhydrous ammonia production facilities are in place worldwide, particularly in countries having access to supplies of natural gas, the major feedstock. Naphtha, refinery off-gases, and condensates, together with coke and coal gases, are other feedstock sources. About 66 countries produce about 132 million tons (120 Mt) of anhydrous ammonia product annually. The People's Republic of China, India, the United States, and the USSR produce just over one-half the total.

Principal downstream nitrogen fertilizer products include anhydrous ammonia for direct application, urea, ammonium nitrate, ammonium phosphates, ammonium sulfate, and nitrogen solutions. About 20% of US anhydrous ammonia production is used for synthesis of nonfertilizer nitrogen products, including plastics, synthetic fibers and resins, explosives, and miscellaneous chemical materials.

2.2.2.7 Byproduct Commodities

These metals are produced as a byproduct of major commodities and therefore their production and supply do not necessarily reflect demand forces. Reserve data are tenuous, with their calculation based on typical content of ore mined for the major metal.

Gallium: Gallium is widely disseminated in the earth's crust; consequently, it is nearly always recovered as a byproduct during processing of ores to recover other metals. The principal materials in which gallium is found are bauxite, coal, phosphate ore, and zinc ore. Of these, gallium is commercially recovered mainly from the processing of bauxite to alumina and the rest from the processing of zinc ore. Although world gallium reserves of over 1.1 million tons (1 Mt) are available from bauxite, much of this bauxite will not be mined for many decades, and only about 40% of the available gallium is recoverable with current technology.

Gallium recovery facilities are centered in Europe, with France and the Federal Republic of Germany as the largest gallium producers. Japan and the United States lead the world in gallium arsenide (GaAs) crystal and device fabrication, with about 85% of the world gallium demand estimated to be in these two countries. The United States has been almost completely dependent upon imports of gallium and relies on the Federal Republic of Germany, France, and Switzerland to supply its demand.

Gallium has limited commercial applications in its metallic form. Its principal use is in the manufacture of semiconducting compounds, mainly gallium arsenide (GaAs) and gallium phosphide (GaP). These semiconducting compounds are used as substitutes for silicon-based devices in optoelectronic devices and integrated circuits. Opto-electronic devices are primarily employed as visual displays in consumer electronic and industrial equipment, as a light source or detector in fiber optic telecommunications systems, and as a power source for satellites. Integrated circuits are used principally in defense applications (Kramer, 1988). Most gallium applications require very high purity levels.

World demand for gallium was about 55 tons (50 t) in 1989, but with the rapid technological progress in GaAs integrated circuit development, the status of world supply and demand is changing dramatically. GaAs has advanced from a laboratory curiosity in the 1970s to a material with distinct applications and almost no effective substitute. Development of fiber optic telecommunications systems, the advent of sophisticated electronic military warfare systems, the widespread use of consumer electronics, and the need to process vast quantities of data in the shortest time possible have provided the impetus for implementing the large number of GaAs research and development programs that are currently underway.

Arsenic: Arsenic occurs commonly in sulfide deposits associated with copper, lead, zinc, cobalt, and gold, and may occur in coal as well. Most arsenic is recovered as a byproduct from copper or lead smelting, though increasingly it is being recovered from the roasting of gold-bearing sulfide concentrates. In general, arsenic is regarded as a troublesome impurity in smelting and refining, and thus high-arsenic source materials may be penalized at the smelter and refinery or avoided at the mine. Arsenic trioxide is produced in about a dozen countries, with France, Mexico, Sweden, and the USSR being the major producing countries.

From 1977 until 1982, there was a world shortage of arsenic trioxide, partly caused by environmental concerns. In the late 1980s, Belgium, Chile, and the Philippines have brought on new capacity. Some companies have actively sought out new sources of arsenic for processing, such as stockpiles of arseniferous dusts, accumulated from various stages of copper, lead, and zinc smelting.

Until 1985, the United States was a producer of arsenic trioxide as a byproduct of copper smelting, principally from imported high arsenic concentrates from the Philippines. The copper smelter and arsenic recovery plants in Washington were shut down because of difficulties in complying with air quality emission standards. Consequently, the Philippines became a new producer of arsenic trioxide in 1985 at its new copper smelter.

An estimated 97% of total western world arsenic production is marketed as trioxide. Most of this trioxide is used in making industrial chemicals, the most important of which is chromated copper arsenate (CCA), which is used as a wood preservative. The use of CCA has grown to be the largest single end use for arsenic trioxide. The next most important use for arsenic trioxide is in the manufacture of agricultural chemicals used in the production of cotton. In the United States, since the 1970s, the uses of inorganic arsenical pesticides have nearly all been phased out. Historically, inorganic arsenical pesticides have been a large end use for arsenic trioxide. There are minor uses for arsenic trioxide and arsenic acid in producing pressed and blown glass.

Arsenic in the form of metal accounts for about 3% of consumption. The major end use for arsenic metal is in copper and lead-base alloys, particularly in lead acid storage batteries to improve the strength of the lead posts and grids. Small amounts of high-purity arsenic metal, 99.999% arsenic or better, are used by the electronics industry in making products that contain gallium arsenide.

Selenium and Tellurium: These trace metals are recovered as byproducts, principally from the processing of copper ores. They accumulate along with the precious metal values in the slimes that are generated during the electrolytic refining of copper. The selenium and tellurium values are not recovered from copper ores processed by leaching technologies nor from copper concentrates processed by fire refining. Much of the selenium and tellurium initially present in the copper ores is not recovered, as concentration circuits are geared toward optimizing copper recovery and slimes circuits are optimized for recovery of precious metals. This is particularly true for tellurium, where recovery of the initial tellurium present in the ore may be less than 5%. Other potential sources of these materials include tellurides of gold and silver, which have been processed for tellurium, and coal resources which contain selenium and tellurium in association with sulfur. Belgium, Canada, Japan, and the United States account for the majority of refined selenium and tellurium production.

The principal uses for selenium are in electronics, where high-purity selenium is used as a photoreceptor on the drums of plain-paper electrophotographic copiers; in heat-stable reddish pigments used in plastics, glass, and ceramics; and in glass manufacturing as a decolorant. Other applications include its use as a feed additive for livestock, selenium being a necessary micronutrient yet toxic in high doses, and as an alloying element in metallurgical applications. Used photocopier drums and manufacturing scrap provide source material for the production of secondary selenium. Tellurium's principal use is as an alloying agent in the production of free-machining steels and copper alloys. It is also used in chemicals and rubber manufacturing and electronic applications including electrophotographic copiers. High-purity tellurium is of strategic significance in cadmium-mercury-telluride used as infrared detectors for military night vision systems.

Germanium: Those producers that currently recover primary germanium rely for the most part on germanium-rich residues derived from past or present base metal smelting operations, mostly zinc. Some germanium is also recovered from scrap. Germanium-rich residues as a byproduct of processing zinc ores mined in Tennessee are shipped to Belgium for germanium recovery and refining. Refined and semirefined germanium is also produced from domestic and imported raw materials in Canada, the Federal Republic of Germany, France, Italy, Japan, and Spain. The USSR and other centrally planned-economy countries produce and refine germanium from domestic sources.

The use of germanium in infrared systems and fiber optics has become the dominant market for germanium. Other applications for germanium include detectors, catalysts, phosphors, metallurgy, and chemotherapy.

Indium: Indium occurs as a trace constituent of other metal deposits, principally zinc, but also lead, tin, tungsten, and iron. It is usually recovered as a byproduct from residues generated in the refining of zinc, so its supply is therefore dependent upon the production of that metal. It is estimated that 90% of the US requirements for indium are accounted for by imports of highly refined indium metal or indium-rich residues and other materials, mainly from Europe. Canada is believed to be the world's largest mine producer of indium and is one of the few countries that processes indium-rich residues produced domestically. Indium-rich materials are usually sent to the major indium-refining countries for processing into high grade material. The major refiners are in Belgium, France, Italy, Japan, and the United Kingdom.

The major use of indium in recent years has been for alloys and solders. Indium is also used in semiconductors, infrared detectors, and atomic indicators, and dental alloys.

2.2.2.8 Advanced Materials

While the metals and industrial minerals as mined or produced normally for commercial use are not usually considered advanced materials, they are the sources of the elements used to make advanced or high technology materials in our modern society. *Advanced materials* can be defined as those materials developed over the past 30 years or so, and being developed at present, that exhibit greater strength, higher strength:density ratios, greater hardness, and/or one or more superior thermal, electrical, optical, or chemical properties, when compared with traditional materials (Sorrell, 1987). Advanced metals, ceramics, and polymers, including composites of these, offer the promise of decreased energy consumption, better performance at lower overall system cost, and perhaps less dependence on imports of materials for which the United States has few or no reserves. Heretofore, materials from high-grade mineral ores, earth minerals, coal, and oil were used as is or processed to produce commodity materials such as metals, glass, ceramics, and plastics for industrial and consumer use. However, technology has now made it possible to design a product and develop new materials with properties needed to perform a specific function and often entirely new functions.

Some of the more important new metals and alloys are aluminum-lithium alloys, with lower density and greater strength than other aluminum alloys; amorphous alloys with superior strength and magnetic properties; magnetic alloys with greatly improved energy product and coercive force; superconducting alloys for high-intensity magnets; super-plastic alloys suitable for new forming processes; improved titanium alloys and coated alloys with greater oxidation resistance; and beryllium-copper alloys with high thermal conductivity and fatigue resistance.

By virtue of their superior heat resistance, wear resistance, and corrosion resistance, structural ceramics are potentially important materials for use in turbine engines, cutting tools, wear parts, and many other applications in which they would replace metals. Some of the important materials for structural ceramics are alumina, zirconia, yttria, thoria, and borides and carbides of silicon, boron, aluminum, and titanium. In general, the raw materials for these materials are abundant, with the possible exception of yttria.

A new class of polymeric materials with exceptional strength and heat resistance is emerging and is increasingly replacing metals in automotive, aircraft, and many other applications. These materials, reinforced with a variety of fibers, are being applied to both exterior and interior components of aircraft and are being evaluated for use in automobiles to replace sheet steel in the body and possibly as structural components.

Stronger, lighter-weight, more heat-resistant materials, such as metal matrix, ceramic matrix, and polymer matrix composites containing particulate, whisker, and fiber reinforcements, have been developed. New fibers of carbon, boron, silicon carbide, silicon nitride, alumina, zirconia, aluminum silicates, and polymers have broadened the range of materials and properties that can be substituted for traditional materials in aerospace equipment.

Electronic, optical, and magnetic materials will require appreciable amounts of relatively rare and less abundant materials, many of them byproducts of the metals already discussed. Among these are gallium, germanium, indium, cadmium, thallium, arsenic, bismuth, zirconium, hafnium, and the rare earths. Because of the byproduct relationship, their future production depends on the extent of major commodity reserves. Consequently, it is imperative that recycling systems exist to recover these materials from waste streams and scrap. Waste materials, such as coal fly ash from power generation and red mud from Bayer processing of bauxite, contain not only appreciable quantities of materials such as aluminum, titanium, and iron, but also valuable minor elements such as gallium and germanium.

2.2.3 US IMPORT DEPENDENCE

Fig. 2.2.1 shows the import dependence of the United States for a number of major commodities expressed as *net import reliance* (imports less exports plus adjustments for government and industry stock changes) as a percentage of apparent consumption. Since World War II, the United States has become increasingly dependent on imports of certain minerals (Cameron, 1986). For the commodities for which reserves are minor or nonexistent in the United States, import reliance is greater than about 75%, and, if recycling plays only a minor part, reliance is nearly 100%. Some commodities, especially cobalt, chromium, nickel, and tin are recovered to a significant degree from scrap, and thus import reliance is significantly lower as secondary production plays an important part in the supply picture.

Since about 1950, the US import dependence has increased substantially for a number of commodities, such as cement, iron and steel, potash, and copper, and to a lesser extent for iron ore, zinc, aluminum, and tungsten. Since the United States has ample reserves and production facilities for most of these commodities, the reasons for increasing import dependence must be sought in the economic and political area.

Prior to the 1980s, cement imports supplemented US production in peak demand years wherever shortages existed. During the 1980s, world consumption declined while US consumption continued to grow. Because of this sustained growth, a strong US dollar in the early part of the decade, cheap water transportation, and in some cases alleged foreign government subsidies and less stringent environmental constraints, it became more economical to import cement. Low cost energy in Mexico for an industry that is one of the most energy-intensive of all led to that country becoming a major US supplier.

The United States was a net exporter of iron and steel in the 1950s, but by the late 1980s was importing almost one-quarter of its needs. By the mid-1960s, Japan and Western Europe had built modern steel industries to compete with the US steel industry. At the same time, the US industry continued to operate aging plants with high labor costs. When steel demand fell in

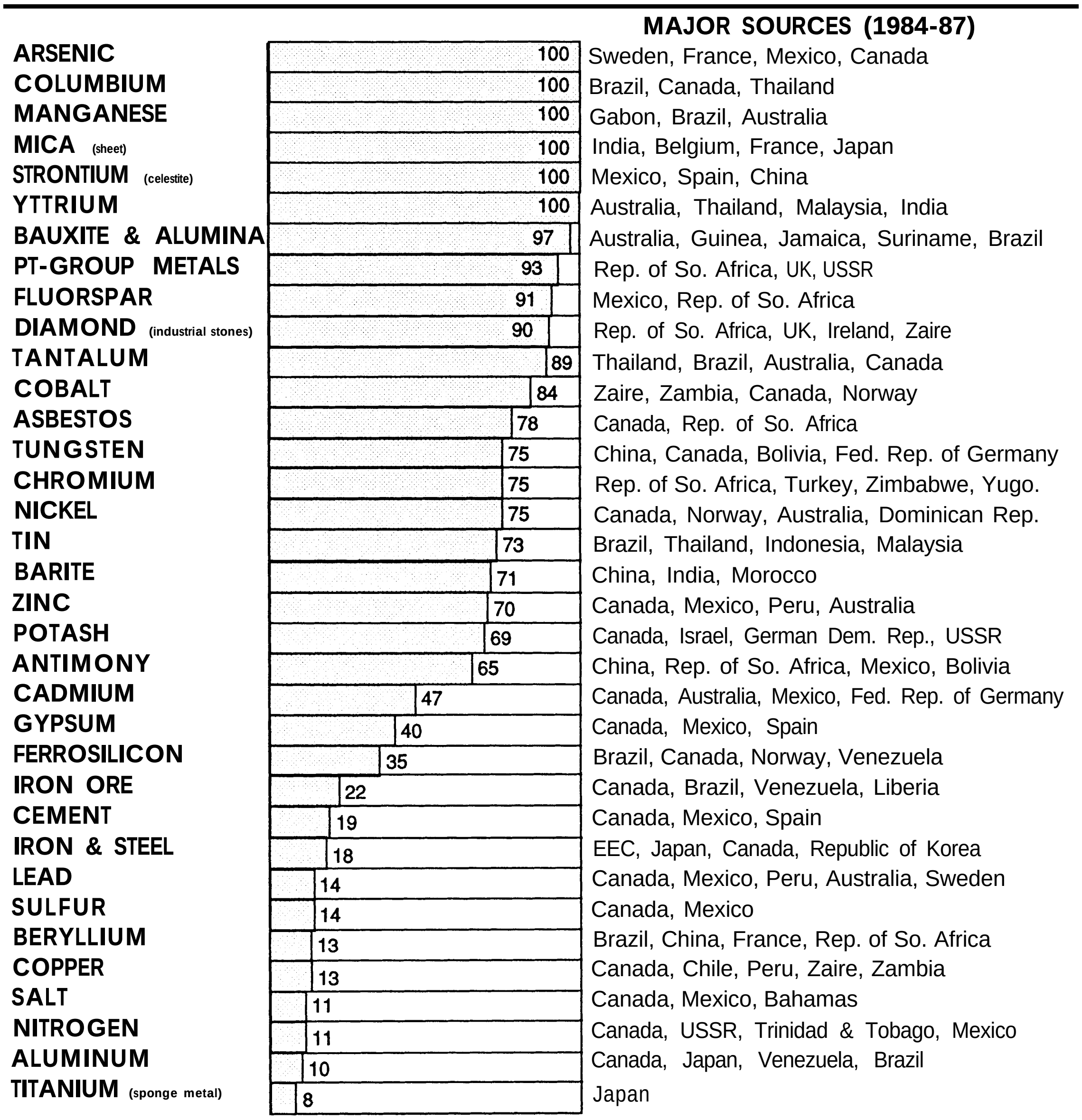

Fig. 2.2.1. 1988 net import reliance[a,b] of selected nonfuel mineral materials as a percent of apparent consumption[c] (Anon., 1989). [a]Estimated. [b]Net import reliance = imports – exports + adjustments for government and industry stock changes. [c]Apparent consumption = US primary + secondary production + net import reliance. Note: For a number of minerals, net import reliance data are withheld or incomplete. However, commodities for which sufficient data are available to indicate a significant degree of import dependency include: andalusite (Republic of South Africa), bismuth (Mexico, Belgium-Luxembourg, Peru, United Kingdom), gallium (France, Switzerland, Federal Republic of Germany, United Kingdom), germanium (Belgium-Luxembourg, China, France, Federal Republic of Germany), graphite (Mexico, China, Brazil, Madagascar), iodine (Japan, Chile, United Kingdom), ilmenite (Australia, Canada, Republic of South Africa) mercury (Spain, China, Algeria, Turkey), rhenium (Chile, Federal Republic of Germany), rutile (Australia, Republic of South Africa, Sierra Leone), selenium (Canada, United Kingdom, Japan, Belgium-Luxembourg), tellurium (Canada, United Kingdom, Peru, Belgium-Luxembourg, and vanadium (Republic of South Africa, South America, European Communities, Canada).

the early 1980s, US production was cut, and the strong US dollar in that period favored imports.

The increase in potash imports beginning in the mid-1960s came about from competition from large, low-cost Canadian mines coming onstream and exporting to the flourishing US agricultural areas. Canadian producers enjoyed a freight advantage in shipping potash to the US grain belt.

The United States evolved from a net exporter of copper through the mid-1970s to a significant importer. Increasing US copper demand coupled with the closure of a number of US mines and smelters due to environmental factors and economic pressures such as high labor costs and a strong dollar contributed to a higher degree of import reliance, which continued through the early 1980s.

The increase in US net import reliance for aluminum, iron ore, tungsten, and zinc has been less dramatic. Nearly all the primary aluminum metal produced in the United States relies on foreign bauxite. Until the 1980s, the US import reliance for aluminum metal was irregular, but with the decline in the 1980s of US smelter capacity due to high labor and energy costs, import reliance exhibited a steady increase. As a result, foreign competitors were better able to supply growing US needs.

Import reliance for iron ore has also fluctuated but was higher in the 1980s than in the 1950s. The depletion of high-grade iron ore reserves in the Lake Superior region was mainly responsible for greater US import dependence. Although the development of the taconite process allowed the use of lower-grade ore, the high impurity level in the pig iron product and environmental factors led to lower US production. In the meantime, many countries through international loans developed their own sizable reserves of high-grade ores that allowed them to enter the international market.

Tungsten dependence has been erratic, mainly because of its relationship to defense efforts and US strategic stockpile activity. With domestic mines operating during the 1960s and early 1970s, supply from US sources was adequate. As prices fell in the 1980s because of a world supply/demand imbalance, lower demand, and increased use of substitutes for tungsten carbide, US production facilities began to close and import reliance increased.

For zinc, total net import reliance has increased, but the reliance on imported zinc concentrate to feed US custom smelters gave way to a reliance on zinc metal imports as US smelters closed due to high costs, obsolescence, and environmental factors. As with a number of other commodities, import reliance was not a matter of lack of domestic resources, but rather one of international competition and political factors that discouraged investments in modern technology in the United States.

For further discussion of imports-exports, see Chapter 2.7.

2.2.4 INTERGOVERNMENTAL MINERAL COMMODITY ORGANIZATION

While most if not all mineral commodities have some type of trade or marketing development association to look after the interests of the industry, some commodities also have intergovernmental organizations with specific mandates. These types of organizations came into being after World War II with the establishment of the United Nations in 1945 and the United Nations Conference on Trade and Development (UNCTAD) in 1964. Developing countries through the auspices of UNCTAD have attempted to form international commodity agreements involving both producing and consuming countries with some type of economic provision for price stabilization through buffer stock management and export controls for bauxite, copper, iron ore, manganese, phosphate, and tungsten. The rationale behind these agreements was to provide an orderly market for these commodities such that developing countries could plan economic growth with assurance of foreign exchange for their commodity exports upon which many were dependent (Buck, 1986). However, the only agreement negotiated was for tin in 1956, with the establishment of the International Tin Agreement (Fox, 1974). In 1985, the buffer stock manager exceeded the financial ability of the implementing International Tin Council to support the tin price. As a result the tin market collapsed and the Council was in bankruptcy.

In the absence of effective economic commodity agreements, intergovernmental producer organizations sprang up for bauxite, copper, iron ore, and sulfur. CIPEC, the Conseil Intergovernmental des Pays Exportateurs de Cuivre established in 1967 by Chile, Peru, Zaire, and Zambia, is perhaps the most well known. Its objectives were essentially the same as for the tin agreement, but without formal economic mechanisms to achieve the end. These organizations also provide much information on the commodity, such as industry statistics worldwide. Committees on Tungsten and Iron Ore exist under UNCTAD as forums for discussion and data dissemination.

The International Lead and Zinc Study Group was formed in 1959, with the anticipation of concluding a commodity agreement. However, it was found that a study group whose function was to provide a forum for discussion, and to collect and disseminate data on the industry so that all members would be free to make their own market decisions, was an acceptable format. Hence market transparency became an objective of commodity groups, and it was this concept in the 1980s that led a number of industrialized nations to foster the formation of "classical" study groups for copper and nickel. In addition, tin-producing and consuming nations began negotiations in 1988 to form a similar organization for tin in order to continue the work of statistical reporting and intergovernmental consultation.

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Chapter 2.3
PRICING AND TRADING IN METALS AND MINERALS

SIMON D. STRAUSS

2.3.1 INTRODUCTION

In any feasibility study for a metals or minerals project, an assumption must be made as to the prices at which the project's products can be sold. This is a difficult assignment. Prices of many mineral commodities are volatile, reflecting unpredictable shifts in the balance between supply and demand. Moreover, with some exceptions, the significant balance is that prevailing in the world market rather than any national market—for most mineral commodities are subject to a high degree of international trade.

Thus the mining engineer involved in the development of a mineral deposit needs to understand the pricing of the minerals that deposit will produce. Above all, he/she needs to recognize the perils involved in forecasting mineral prices. Even the most experienced and sophisticated observer of mineral markets is likely to err in predicting the future course of prices. This is true even though his/her studies have taken into account what appear to be all relevant factors.

Moreover, in projects involving mineral deposits previously unexploited, the forecast of prices must cover an extended period. Typically, a "greenfields" project requires several years for mine development, construction of production facilities, and provision of necessary infrastructure of transport and power supply. Following project completion, the price forecast will cover the years necessary to recoup the original investment and provide an adequate return. For most projects, this means that the forecast will cover a period of 10 to 20 years.

The yardsticks by which prices are measured are vastly different from the yardsticks with which engineers are familiar in the measurement of production, metallurgical efficiency, or labor productivity. Engineers are used to working with yardsticks that are constant—measurements of weight in pounds or kilograms; measurements of distance in feet or meters; measurements of time in seconds, minutes, hours, or days. Recovery of mineral content can be measured in percentages.

The yardsticks for prices are in terms of monetary currencies. The value of a given currency unit tends to change. Thus if the current price of a given mineral is the exact monetary equivalent of the price of that mineral 10 years earlier, it nevertheless has almost certainly changed. Due to the shifting tides of economic fortune, a given amount in dollars and cents today may be worth more or less than it was 10 years earlier. It will be worth more than 10 years ago if the economy has suffered deflation; it will be worth less than 10 years ago if the economy has suffered inflation. During the 20th century, with the exception of the severe depression of the thirties, the general trend worldwide has been inflationary.

To provide a specific illustration of the shifting value of currency units in relation to a mineral commodity, consider the price of copper in 1988 in the United States—it averaged $1.19/lb ($2.62/kg). Fifty years earlier, in 1938, the average price of copper was l0¢/lb (22¢/kg). Thus copper appears to have increased in price nearly twelvefold in 50 years. However, if these two prices are adjusted to allow for inflation, the comparison between them yields a very different picture. A common tool for making such adjustments is the US consumer price index. Taking this so-called CPI with the year 1967 equalling 100, the price of copper in 1938 is measured to have been 23.7¢/1b, not l0¢/lb. And the price of copper in 1988 is measured to have been 33.6¢/1b, not $l.l9/lb. In other words, by restating the prices in terms of a constant deflator, the change in the copper price over this 50-year period was not a gain of 1l00%, but rather 42%.

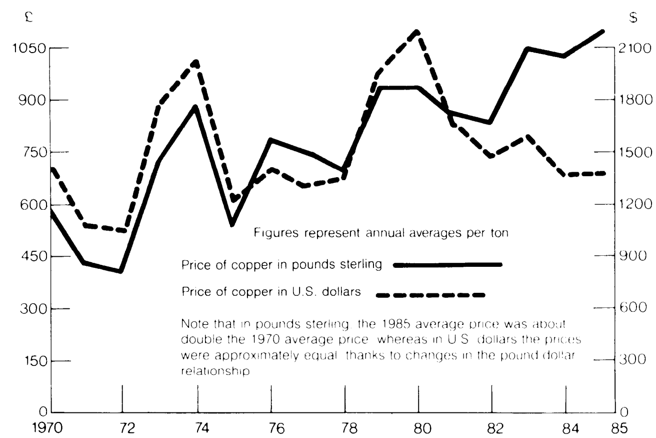

Fig. 2.3.1. London Metal Exchange prices for copper, 1970-1985, expressed in pounds sterling and US dollar equivalents (Strauss, 1986).

The mining engineer who is making a feasibility study of a mineral project obviously cannot be expected to study the broad economic picture. Therefore, his projection of price will be based on what the economists call "money of the day." If he is preparing his analysis in, say, the year 1992, his forecast of prices can appropriately be stated on the basis of 1992 dollars. If future economic trends significantly alter the value of the dollar, the presumption must be that roughly corresponding changes will occur in the prices of mineral commodities—although this may take time.

An additional difficulty in forecasting prices is that the currency of any given nation may lose or gain value at a different rate than the currencies of other nations. This has been particularly true since the early 1970s, when the system of fixed exchange rates instituted by the International Monetary Fund at the end of World War II gave way to a new era of floating exchange rates.

For most mineral commodities, the currency standard in the 20th century became the US dollar, as it was for many years the accepted denominator of international trading values. Thus prices stated in US dollars are the usual measuring stick for such significant minerals and metals as gold, copper, aluminum, nickel, zinc, lead, silver, tungsten, iron ore, sulfur, phosphates, potash, and talc, to name just a few.

But due to the change in the value of other leading currencies in relation to the dollar, it has frequently happened that a rise in the price of a commodity in terms of dollars may equate to a loss in the price of that commodity in terms of a different currency, say Japanese yen or German marks. See Fig. 2.3.1, especially after 1980, for an example of this. To further illustrate, between February 1985 and March 1987, the price of gold rose from \$299/oz (\$10.55/g) to \$409/oz (\$14.43/g), a gain of 37%. But during those 25 months, the value of the dollar fell in relation to other major currencies. In terms of Japanese yen, the price of gold declined by 21%, and in terms of German marks, it fell by 24%.

How then can one describe the gold market during those 25 months. Was the price rising? Most US citizens would say so. Was the price falling? That would be the verdict of the Japanese or the Germans.

The key point is that on any given day, however, the price of gold in the United States, Japan, or Germany was approximately equal—given the high value of the commodity, the ease with which it can be transported, and the absence of import duties. For other bulkier commodities, there may at times be significant differences in prices prevailing in particular markets. And where there are substantial barriers such as tariffs, quotas, or other limiting factors, on occasion a wide gap may exist between the price in one national market as distinct from other countries. However, with the increased trend toward freer trade and the continued improvement in transportation facilities, such isolated instances of large price differentials are becoming increasingly rare. One contributing factor to maintenance of fairly uniform pricing throughout the trading world at any given time is the information explosion resulting from widespread use of computers, fax machines, international telephone communications, and a vigilant trade press. Moreover, a substantial share of the world's mineral trade is carried on by international dealers or brokers who have a keen sense of market developments and hence will focus on sales in strong markets in preference to those where demand is lagging.

2.3.2 HOW ARE MINERAL PRICES DETERMINED?

The methods by which mineral prices are established differ widely. Unique patterns have evolved over the years in which minerals have been traded. Since there are literally hundreds of distinct mineral commodities being bought and sold, it is impracticable to attempt in this presentation to describe the prevailing procedures for each—or even for the 30 or so most widely used minerals in terms of volume or value.

Instead, an attempt is made to describe some of the more widely used methods of arriving at price.

In general, minerals are sold on the basis of weight, varying from carats for gem stones, ounces for precious metals, pounds or kilograms for the more valuable base metals, and on to tons (be they short, metric, or long) for less valuable metals, ores, and most of the industrial minerals. In some cases the price is stated on the basis of the metal content of certain ores rather than gross weight—thus, for example, tungsten is sold on the basis of units of tungsten contained in wolframite or scheelite concentrates.

For any given mineral, market transactions can occur at several stages of production. Thus there is a market for bauxite (the usual source of aluminum), for alumina (an intermediate product), and for aluminum. Prices are quoted for each of these three products and the trends are not necessarily the same. At times, aluminum metal may have been in short supply, but there may have been an abundance of bauxite.

One can identify at least four principal methods by which the minerals industries arrive at their selling prices. There is considerable difference of opinion among producers, consumers, traders, and public officials as to which is the best procedure. All express a desire to arrive at prices that are fair and reasonable and promote stability, yet they differ profoundly in their approaches.

The four broad procedures are

1. Prices established by a producing seller, who periodically announces the terms and conditions, as well as the prices, at which he/she will make sales. These are called producer prices.
2. Prices announced periodically by an independent agency, often a trade periodical, that makes regular surveys among both buyers and sellers to ascertain the basis at which actual transactions have taken place.
3. Prices negotiated directly between seller and buyer. Frequently, but not always, the result is contracts extending over a period of several years with provision for price adjustment under agreed conditions.
4. Prices established on futures markets or commodity exchanges. These facilities, open to all to trade, provide auctions by open outcry, comparable with securities exchanges.

For some commodities, the determination of price in some markets is by one procedure and in other markets by other procedures. For example, for many years in the United States, the price of lead was based on method 1. The major US and Canadian sellers, the chief suppliers of lead in the United States, periodically announced the price at which they would market their product. However, in the European market, the price of lead has long been customarily based on the quotation of the London Metal Exchange. In some of the developing countries of Asia and Latin America, sales of lead are frequently the result of method 3, negotiation between buyer and seller. In such negotiations, due attention is always given to the prevailing prices in the United States and Europe.

Some transactions take place in which two or more of the pricing methods play a role in a single transaction. Typical of these are the sale of copper, lead, zinc, or nickel concentrates by a producing mine to a so-called custom smelter, a processing facility which secures part or all of its feed from independent mines. Method 3 (direct negotiation between the miner and the custom smelter) is universally followed to set the treatment charges involved. These are characteristically stated in terms of an amount per ton of concentrate delivered, modified by credits for byproduct elements that are recovered and penalties for impurities. However, payment for the recoverable metal contained is based on a price determined either by methods 1, 2, or 4. For example, lead content may be paid on the smelter's selling price for lead (method l), a periodical's published price for lead (method 2), or the London Metal Exchange price for lead (method 4). Usually, these are the average price for the month in which the concentrate has been delivered to the smelter or for the month following delivery.

Perceived advantages and disadvantages of each method of pricing will influence both sellers and buyers as to which they select. Method 4 (trading on a commodity exchange) has not been found practicable for most mineral commodities even though it has become a highly acceptable price determinant for several key metals and has long dominated the field of pricing of leading agricultural commodities.

In the following, we discuss the salient characteristics of each of the four methods.

2.3.2.1 Producer Prices

In establishing a producer price, the seller takes into account his cost of production, his potential markets, the position of his competition, and the possibilities of increasing his market share. The competitive factor must always weigh heavily on his decisions.

Historically, there have been two instances of a mineral monopoly, a single significant seller in the world market. One was the Chilean nitrate of soda industry prior to World War I, and the other was the Greenland cryolite industry prior to World War II. In a sense, these two industries were in a position to name their own prices without considering competitive factors. Eventually, however, the very existence of this seemingly unlimited power sparked successful efforts to develop synthetic products that proved to be acceptable substitutes, and the apparent monopoly ended.

Although monopolies are rare, in the mineral field there have been numerous instances of what economists call oligopolies. These are markets dominated by a small number of powerful sellers. As examples of oligopolies, one might cite the aluminum, molybdenum, and nickel markets. As late as the middle of the 20th century, these markets were examples of producer pricing.

In the case of aluminum, a handful of companies accounted for the bulk of world production through the 1960s, notably Alcoa, Alcan, Alusuisse, Kaiser, Pechiney, and Reynolds. In molybdenum, the dominant seller was Climax, subsequently merged into the company now known as Amax. In nickel, the largest seller by far was Inco (formerly International Nickel Co.).

The pricing policies of these companies appeared to be designed for stability (in contrast with the volatile price behavior of such base metals as copper, lead, and zinc) at levels that would encourage maximum expansion of consumption. Indeed, in the third quarter of the century, the consumption of aluminum increased at an average rate of 8% compounded annually, while consumption of nickel and molybdenum both rose by about 6% compounded annually. These figures are for world consumption.

Expanding use of these metals was due to intensive research and promotion efforts by the dominant companies, as well as to the stable price structure. Since success attracts attention and imitation, in the long run what happened was that other corporations were encouraged to enter the aluminum, molybdenum, and nickel industries. Intensive exploration discovered previously unknown deposits in many areas (including byproduct resources of molybdenum in many copper deposits), financing was arranged, and new producers were launched.

With more sellers competing in the marketplace, gradually the ability of the previous major firms to control price was eroded. Stability of price gave way gradually to volatile behavior. By the 1980s, the prices of aluminum, nickel, and molybdenum rose and fell in response to market conditions much as had long been the case with other metals.

Producer pricing has coexisted with prices established on commodity exchanges in the case of copper, zinc, and lead. Producers of these metals have attempted to lessen the extreme swings in price on the exchanges at times in an effort to prevent a perceived threat of substitution by competing materials. When, as has happened, a material discrepancy has developed between the producer price and the exchange price, difficult market distortions have resulted. Increasingly the trend has been away from the producer price and toward the exchange-denominated price.

Two important categories of mineral commodities—the industrial minerals and the so-called minor metals—tend to be sold either on the basis of producer prices or prices as determined by trade periodicals or other independent sources. Recent price histories of representative nonmetallic minerals appear in Table 2.3.1.

Major industrial minerals tend to be bulk materials of relatively low unit value. They include sulfur, potash, phosphate rock, barite, gypsum, salt, fluorspar, diatomaceous earth, talc, and asbestos. An appreciable portion of their cost to the user is absorbed in transportation costs from mine to market. Keen competition prevails in these commodities, but they are considered unsuitable for listing on commodity exchanges because of low value and problems of establishing uniform quality standards.

Minor metals present a contrast: they have fairly high unit value but limited volume. Many are byproducts recovered in processing base-metal ores, for example, bismuth, cadmium, cobalt, germanium, indium, tellurium, and selenium. Antimony and mercury are two exceptions in that much of their production is derived from deposits in which they constitute the chief element of value. A problem arises in that output of the byproduct metals depends on the demand for the major base metals, whereas demand for byproduct metals does not necessarily coincide with demand for major metals. Hence markets for byproduct metals are at times subject to substantial excesses or shortfalls-with drastic effect on price. This price volatility would appear to encourage listing on commodity exchanges, but the prospective trading volume is too small to interest the exchanges.

Producer pricing is not feasible for the two traditional precious metals, gold and silver. Therefore, they have historically had prices established by other procedures. Huge supplies of gold and silver mined in the past exist as bullion, coins, jewelry, and works of art. These holdings dwarf the current newly mined product and exercise a dominant role in price determination. Because supplies of platinum group metals arise chiefly from two sources—the Soviet Union and South Africa—in the past, producer pricing arrangements have existed, yet increasingly the markets are being influenced by commodity exchange trading.

2.3.2.2 Independent Pricing Determination

Responding perhaps to consumer concerns over being subject to prices unilaterally determined by producers, in many minerals pricing is sometimes based on quotations as determined by a source that is neither seller nor buyer. In many but not all cases, this may be a trade periodical, such as *Metals Week* or *American Metal Market* in the United States or *Metal Bulletin* in Great Britain.

The prices quoted in these periodicals are based on canvassing producers, consumers, and merchants dealing in the specific commodity to determine the prices at which actual transactions have taken place at specific dates. The publication then reports these prices as averages for a given day, week, or month. Not infrequently, contracts are made between producer-sellers and consumer-buyers providing that the actual price for a given tonnage of metal or mineral will be the price as reported in the publication. Sometimes a seller will give buyers the option of paying either the seller's price or the periodical's average price, with the option to be decided in advance.

In addition to the trade press, other independent arbiters of price exist. One of the best known is the Handy & Harman daily silver quotation. Handy & Harman is a US silver refiner and fabricator, based in New York, that buys silver from producers and sells fabricated silver products to ultimate consumers. Each day, having determined the quantity of silver it requires to meet that day's sales commitments, Handy & Harman solicits bids from principal silver sellers to supply a matching amount of metal. Its price represents the clearing price at which it can obtain the amount it requires.

In London, a somewhat similar procedure is undertaken daily by the leading London bullion dealers to match inquiries from gold and silver buyers with offers from gold and silver sellers. Representatives of the firms meet at noon to review offers and bids received from worldwide sources. The consequence is announced as the official "fixing" for each of the two metals. Much of the world's commerce in the two metals is based on this daily announcement, although since the mid-1970s, attention is also paid to prices as announced in Zurich, Tokyo, Singapore,

Table 2.3.1. US Prices for Nonmetallic Minerals, 1973–1985

Year	US Gross National Product Deflator Price Index = 100	Barite, $	Boron, $	Diatomite, $	Gypsum, $	Phosphate Rock, $	Salt, $	Sulfur, $
1973	49.5	16.66	88.18	65.04	4.60	6.24	6.82	17.56
1974	54.0	16.77	108.03	83.77	4.86	12.10	7.87	28.42
1975	59.3	17.70	115.74	88.18	5.04	25.35	9.85	44.91
1976	63.1	25.63	121.25	95.90	5.51	21.26	8.62	45.72
1977	67.3	22.33	130.07	109.13	6.12	17.39	9.85	44.38
1978	72.2	22.71	141.09	122.36	6.87	18.56	11.13	45.17
1979	78.6	28.08	184.08	138.89	7.53	20.04	11.03	55.75
1980	85.7	32.39	186.28	160.94	9.18	22.78	16.15	89.06
1981	94.0	39.64	205.03	180.78	9.40	26.63	15.17	111.48
1982	100.0	41.53	221.56	194.00	9.33	25.52	15.31	108.27
1983	103.8	42.69	221.56	203.93	8.68	23.97	14.80	87.24
1984	108.1	36.19	229.28	212.74	8.75	23.99	15.19	94.31
1985	111.7	30.86	229.28	230.38	9.15	23.50	15.43	104.68
Change 1973-85	+125.7%	+85.2%	+160.0%	+254.1%	+98.8%	+176.6%	126.3%	+486.1%

Source: US Bureau of Mines 1986 for prices. GNP Deflator Index as reported by American Bureau of Metal Statistics.
All prices are in $US per ton material, f.o.b. shipping point.

and other bullion-trading centers, as well as to the prices prevailing on the New York Commodity Exchange.

In the case of tungsten, two independent sources of price determination have been established. *Metal Bulletin* in London has long been the source of quotations for tungsten content of scheelite and wolframite concentrates, the form in which most of the world's tungsten production is traded. In addition, as a consequence of meetings arranged under United Nations auspices, a so-called International Tungsten Indicator price is calculated, based on details of tonnages sold and prices paid provided by both buyers and sellers, stated in price per unit (1%) of tungsten content (in US dollars).

2.3.2.3 Negotiated Prices

In many transactions, the applicable price paid for a metal or mineral is the consequence of direct negotiation between buyer and seller, rather than the use of a price previously established by the seller or previously published by an independent agency. The use of negotiated prices is particularly common in the case of long-term contracts covering the supply of raw materials to a processing facility.

Thus, for example, a mine supplying iron ore to a steel mill or one supplying chrome or manganese ores to a ferroalloying plant will typically negotiate the price of its product with the consumer. Because the material involved is bulky and because transportation arrangements must be worked out on the basis of a regular schedule, it suits both the buyer and the seller to commit themselves to a long-term arrangement.

The contract specifies the quality of the product to be delivered: the seller can only offer material of the particular characteristics contained in the deposit it is exploiting. The buyer needs to be certain that his processing facility will receive raw material it can handle efficiently. There is thus a community of interest between the two.

In the price negotiation, each party is cognizant of the other's requirements. The buyer knows that if the price involved is inadequate to insure continued operation by the mine, he may lose a suitable supply of raw material for his plant. To guard against adverse consequences of major inflationary trends on his supplier's operating costs, as a rule he will agree to a provision for price escalation in the event labor, fuel, supply, or transport costs change drastically. The seller has to recognize that if he makes excessive price demands, the buyer will look for alternative sources. Therefore, possible competition influences the seller's negotiation stance.

Concentrates of such base metals as copper, lead, zinc, and nickel are normally sold through long-term contracts negotiated between a miner and a smelter—or perhaps by a miner and an intervening trading firm that then sells to a smelter. In such negotiations, the discussions center about the processing charges for smelting and/or refining of the concentrates. A separate matter is the determination of the value of the recoverable metal content. This is calculated either on the basis of published prices in trade periodicals or on official quotations from the commodity exchanges. The payment for the concentrates is the consequence of deducting the processing charges from the calculated value of the recoverable metals.

Buyer-seller negotiations also frequently determine the price paid in transactions involving nonmetallic industrial minerals—for example, chemical companies purchasing sulfur or other materials required in fertilizers and similar products. Major construction projects are also likely to negotiate on a long-term basis for cement, building stone, and other required minerals.

2.3.2.4 Commodity-Exchange Pricing

Prices of a few important nonferrous metals are either determined on or largely influenced by transactions on commodity exchanges, the two most widely known being the London Metal Exchange and the New York Commodity Exchange. There are other exchanges in operation that trade in metals, but these two are the ones most frequently referred to in the metals trade. The number of metals listed for trading in London and New York changes from time to time. When this chapter was written, London was trading in aluminum, copper, lead, nickel, tin, and zinc. New York was trading in aluminum, copper, gold, and silver. On the New York Mercantile Exchange, platinum and palladium were being traded.

Some description of exchange trading is essential for an understanding of the role played. Essentially, such trading represents an auction at which contracts for a given commodity are offered for sale during prescribed hours of each business day. Trading on an exchange is open to anyone with an appropriate credit rating, but the actual transaction must be conducted through a firm or individual that is a member of the particular exchange.

Exchange contracts specify a standard quantity of the commodity being traded. For example, the London copper contract is for 25 t (metric tons); the New York copper contract is for 25,000 lb. The quality of the metal that constitutes acceptable delivery under an exchange contract is carefully specified. Brands that are produced by known refiners must be approved for listing by the exchange before they can be traded.

Offerings made on the exchange floor are for delivery at a specific time. London contracts are traded for the prompt or “cash” position, requiring immediate delivery or for future delivery on a specified date. New York contracts specify the month of delivery.

Under exchange contracts, the seller may deliver any acceptable brand he chooses at any warehouse site approved by the exchange. This latter option is of great significance since the London exchange has approved warehouses not only in Britain but also at major ports throughout the European continent and even in Singapore. The New York exchange has approved warehouses scattered throughout the continental United States.

If the buyer takes delivery at a warehouse location that is inconvenient for him, or if the brand tendered is not one that he normally consumes in his plant, as a rule he can arrange to make an exchange for a more suitable warehouse site or a more desirable brand through the member firms, but this may involve paying a premium identified as “exchange for physical.”

However, in the vast majority of exchange transactions, physical delivery is not made. Instead, the buyer of a contract may sell or the seller of a contract may buy back the quantities involved in the original transaction prior to the effective date of the contract. What then is the purpose of the transaction? Simply to establish a known price to cover the seller’s output or the buyer’s raw-material costs.

As an illustration, assume a secondary copper smelter buys scrap material to be delivered to its plant in 30 days. After delivery, an additional 45 days may be required for processing. The price of copper may rise or fall substantially between the date of the original purchase and the date, 75 days forward, when metal is available for sale. By selling an equivalent amount of copper on the exchange the day the scrap material is purchased, for delivery 90 days forward, the secondary smelter has fixed the price it will receive based on the position of the market at the time it bought its raw material. When the recovered metal is available, it is sold to a consumer based on the then prevailing price and the outstanding exchange contract is bought back. These two latter transactions offset each other, so the net outcome is based on the original exchange sale even though no delivery is made on the exchange.

In addition to the large volume of exchange transactions based on this type of hedging, there are also trades made by speculators who believe they can realize capital gains through purchasing when a rise in price is probable or through selling when a fall in price seems imminent. Speculators are not actually involved in the metals business. They are as apt to trade in orange juice or hog bellies as they are in aluminum or gold. They too have no interest in holding supplies of metal per se. Therefore, in the great majority of instances, they either liquidate their positions prior to maturity or, alternatively, liquidate nearby positions and roll the contracts forward to more distant dates.

Obviously, much more could be spelled out in regard to exchange transactions—volumes have been written in regard to their operations (see the bibliography for some sources). Because of space limitations, however, a brief summary is all that is offered here.

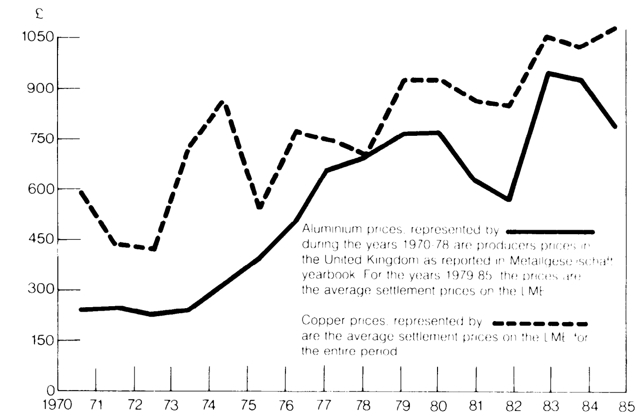

Fig. 2.3.2. Average prices for aluminum and copper, 1970-1985, stated in pounds sterling per ton (Strauss, 1986).

The key to the trend of prices on commodity exchanges is the perceived balance between supply and demand. If supply exceeds demand, inventories will rise and prices will fall. If demand exceeds supply, inventories will fall and prices will rise. The trend of production costs does not immediately affect prices on an exchange.

It is different with a producer price. Costs are an important price determinant. Thus after making an expensive wage settlement, a producer will try to increase its price. By contrast, when a wage settlement is reached, no matter how costly, the commodity-exchange price of the product will tend to fall because the market perceives that the risk of a strike and a reduction in supply has been averted.

As an example, before aluminum had been listed for trading on the London Metal Exchange, the producer price of aluminum was increased in 1975—even though at the time there was a serious economic slump under way. The rationale was that an expensive wage contract had just been concluded. After trading started on the London Metal Exchange during the recession of the early 1980s, the price of aluminum fell sharply—despite further wage increases (Fig. 2.3.2).

Producer prices are still quoted for some metals traded on commodity exchanges, but their significance has diminished. Many producers of the metals traded in London and New York believe that the volatility of prices is increased by the exchange mechanism. Yet they are reconciled to living with the prices quoted on the exchanges and have modified their former marketing strategies to take the exchange prices into account.

2.3.3 EFFORTS TO STABILIZE PRICES

Volatile mineral prices create problems for producers, consumers, and governments. Ore has been defined as mineralized rock that can be profitably extracted. At a given price a deposit may be profitable; at a lower price it will not be. Therefore a mine operator’s estimate of ore reserves is dependent on price and the very existence of his enterprise can be threatened by a sharp fall in price.

Consumers who produce products containing minerals need to establish their own selling prices. Frequent changes in those prices are resented by the buying public. Hence consumers deplore constant fluctuation in the cost of essential raw materials-particularly those raw materials that constitute a large part of their total costs.

Governments—particularly governments of developing countries that depend on mineral exports for a large share of their foreign-currency revenues and of their tax incomes—find it hard to plan their budgets in the face of widely fluctuating mineral prices.

Not surprisingly, therefore, repeated efforts have been made to stabilize mineral prices. Sometimes these efforts are made by private-sector producers of minerals; sometimes they are made by governments through commodity agreements. With few exceptions, these efforts have ultimately been unsuccessful because market forces have proved more powerful than the most ingeniously contrived stabilization schemes.

Perhaps the most extensive and longest lasting of the formal inter-government stabilization arrangements was that undertaken, beginning in 1956, by the International Tin Council. Although the number of member governments varied from time to time, during most of the three decades of ITC operation, there were over 20 tin-consuming and 6 or 7 tin-producing countries formally allied with this undertaking.

The stated objectives of the ITC were to avoid persistent disequilibrium between production and consumption of tin and the accumulation of burdensome stocks, to stabilize employment in the tin producing and consuming countries, to maintain steady export revenues for the producing countries, and to minimize price fluctuations.

Producing and consuming country members shared equally in decisions, but on major issues a two-thirds majority was required, ensuring that approved measures were acceptable to some members from both groups. Individual members had voting rights proportional to their production or consumption.

The Council set both floor and ceiling prices for tin, with the objective of keeping the actual price in the trading zone between floor and ceiling. These prices were subject to periodic reviews and the underlying agreement was itself renewed every five years.

The mechanism by which the Council undertook to keep the price within its desired trading range involved buffer stocks, financed by contributions from the producing-country members, and export quotas to be invoked if demand and prices were weak. Some consumer countries eventually made voluntary contributions to the buffer-stock fund, thus increasing available resources.

The buffer stock was operated by a manager who was instructed to buy tin in the open market if prices dropped toward the stipulated floor and to sell tin if prices rose toward the ceiling. The initial funds raised when the Council began operations were equivalent to about 25,000 tons (22,700 t) of tin, about one-sixth of the then current rate of world production.

During the 29 years of its existence, the floor and ceiling prices inexorably moved higher, reflecting worldwide inflation trends. In this period of time, the floor price was never lowered; the tin-agreement mechanism worked like a jack that steadily ratcheted the price upward. There were two consequences—one was that production of tin by nonmember countries was encouraged; the other was that consumption of tin was discouraged. World tin consumption in the 1980s was roughly at the same level as in the 1950s, whereas consumption of all other base metals had increased sharply.

By 1985, the buffer stock manager had not only exhausted all the funds available from member contributions, he had borrowed heavily using the buffer stock as a collateral to buy still more metal. In the lackluster economic situation then prevailing, supply exceeded demand, and stocks continued to build. In October, operations of the stock were suspended, the price of tin collapsed to levels far below the floor price, and the Council's inability to meet its obligations resulted in staggering losses of hundreds of millions of pounds to lenders and London Metal Exchange member firms. The banks had loaned funds and the traders had assumed that the member governments would stand behind the Council's obligations. They did not. Litigation resulted that continued until 1990, when settlements were made that mitigated a fraction of the losses.

The tin agreement was undertaken in good faith by member governments that operated with the best of intentions. It worked for a while, but eventually ran afoul of the market because good intentions are not enough. Consuming countries, wanting to help the producing nations that were for the most part poor countries with limited resources, had been persuaded to agree to unrealistic floor prices based on the costs of production at the most marginal mines. Once the target was set too high, trouble was inevitable in the long run.

There have been discussions of intergovernmental price stabilization arrangements for several other minerals—copper, bauxite, manganese, tungsten, and iron ore to name a few—but none have gone beyond the talking phase. The collapse of the tin agreement has reduced the interest in similar schemes.

Of course, history has been replete with efforts to stabilize prices through private arrangements. Under existing anti-monopoly legislation in many industrialized countries, such efforts would today be illegal.

One exception that perhaps proves the rule is in the field of gemstones. A high proportion of the world's supply of diamonds is marketed by a group called the Central Selling Organization (CSO), controlled by De Beers, a prominent South African diamond-mining firm. With its affiliates, De Beers today accounts for perhaps somewhat less than one-half of world production of gem and industrial diamonds. A large part of the balance of the production is bought by the CSO for classification and subsequent resale.

De Beers is well financed, able to hold substantial inventories, and prepared to adjust its own production as demand fluctuates. Competitive producers have come to recognize that to market diamonds in competition with De Beers could lead to sharp price declines that, in turn, would destroy the attraction that diamonds possess as long-term investments. They have therefore been willing to dispose of their output through the Central Selling Organization. A case in point is provided by the major Australian diamond deposits developed by firms independent of De Beers in the early 1980s. Originally, the operators planned to do their own selling but before actual production began, they agreed to sell all gemstones and 75% of industrial stones through the De Beers mechanism.

At the time this is written, the diamond market is firmly under CSO control with prices stable—although not inflexible. Public confidence in diamonds as investment has been maintained. Revenues realized by developing countries from their diamond exports have been stabilized. Nevertheless, in the United States, the cartel is considered illegal, but efforts to institute legal proceedings have been unavailing since neither De Beers nor CSO maintains offices within the United States.

The nonmetallic minerals have been less subject to severe price volatility than the metals. As previously commented, they are not traded on commodity exchanges and hence are rarely the target of speculators. Furthermore, the nature of consumption of nonmetallics is different from that of the metals: demand tends to be steadier and less subject to short-term cyclical economic trends. Because most of them sell at low prices in relation to bulk, the threat of substitution by competitive materials is much less pronounced. Indeed, for most of the industrial minerals, the level of world consumption has shown steady but not spectacular gains in line with population trends and overall industrial activity. Hence forecasts of nonmetallic mineral prices can be made with greater confidence than is the case with the metals.

2.3.4 SOURCES OF PRICING INFORMATION

Mining engineers need reliable information about prices. For current prices, the best sources are the three publications already mentioned, *Metals Week* and *American Metal Market* in the United States and *Metal Bulletin* in Great Britain. Commodity exchange quotations are available on a daily basis in the *Wall Street Journal* in the United States and the *Financial Times* in Britain and also in some metropolitan dailies, but these publications carry only limited information about prices of minerals not traded on commodity exchanges.

Other professional and trade publications carry price information (e.g., *Engineering and Mining Journal, Skillings Mining Review),* but it tends to be less current than the data appearing in the five periodicals mentioned.

When long-term historical pricing data are needed for feasibility studies or basic understanding of macro-economic trends, useful tabulations can be found in the yearbooks published by the US Bureau of Mines, the American Bureau of Metal Statistics, and a private German organization, Metallgesellschaft of Frankfurt A.M. The latter is particularly helpful if data are desired for prices within some of the principal consuming countries stated in terms of local currencies.

The World Bureau of Metal Statistics publishes excellent information on production, consumption, and trade in metals, but its yearbook does not contain tabulation of prices. However, it has perhaps more data on production and trade in minerals between the market-economy countries and the nations of the Communist bloc than any other source. Much of the production information is perforce based on informed estimates since official statistics have not been provided by many of the governments of the Communist bloc.

Further discussion of the marketing and sale of minerals is contained in Chapter 25.5.

2.3.5 TRADING IN MINERALS

As commented in the opening paragraph of this chapter, most minerals are subject to a high degree of international trade. This is the consequence of the erratic distribution of viable mineral deposits around the globe. For example, among the commodities that have highly concentrated production, one might cite platinum-group metals (primarily produced in the Soviet Union and South Africa), cobalt (Zaire, Zambia, and Canada), columbium (Brazil), phosphate rock (United States, Soviet Union, and Morocco), and asbestos (Soviet Union, Canada, and Zimbabwe).

Even in the case of minerals that are somewhat more widely distributed—such as tin, nickel, bauxite, and copper—the extent of international trade is huge because much of the production is concentrated in countries that are not large consumers while much of the consumption takes place in countries that have limited production potential. Western Europe and Japan are examples of the latter. Highly industrialized, they have only meager viable mineral resources and must depend on import sources to provide the mineral raw materials they require.

Particularly illustrative of the extent of international trade in minerals is the position in iron ore. Deposits of iron ore are to be found in every country, but the quality and quantity of these resources vary enormously. Great Britain, Germany, and Japan in their early stages of industrial development exploited their domestic iron-ore resources, but the best of these deposits have been long since depleted. Today these countries are huge importers of ore. The United States was at one time a net exporter of iron ore, thanks to the resources of its Mesabi Range, but since World War II has become a major importer.

Among the market economy countries, the largest iron-ore producers in the last three decades have been Brazil and Australia. Although they have substantial domestic iron and steel industries, the bulk of their output is exported. Since they are located at considerable distances from their potential customers, they have developed superb port loading facilities served by huge bulk-carrying freighters to transport their ores to the major consuming centers.

For many of the mineral-exporting countries, a major thrust has been to encourage the development of domestic processing facilities. Thus, rather than export copper concentrates, the preference has been to develop domestic smelting and refining facilities so that exports will be in the form of refined copper or even, perhaps, fabricated copper products. The economics of downstream processing are complex; considerations such as availability of fuel, skilled labor, and transport facilities between mines and processing plants may result in costs that exceed the charges offered by foreign processors—particularly if the initial capital cost of building a new facility is high.

A further complication has been the fact that the raw-material importing country may be encouraging its domestic processors through high duties on refined product and no duties on raw material imports. Thus a considerable share of the world trade in minerals remains in the form of raw materials rather than semi-finished or refined products.

Nevertheless, the long-term trend has been toward an increase in processing in the country where the minerals originate. This has been particularly true for ferroalloying materials. Prior to World War II, most of the trade in manganese and chrome was represented by ores. By the 1970s, a major share of international commerce in these two elements was in ferromanganese and ferrochrome. Indeed, one of the principal manganese exporters—India—prohibited the export of ore at one time and permitted only export of ferromanganese.

Although much of the international minerals trade is carried on through direct contracts between the minerals producer and the minerals consumer, a very large share—perhaps an increasing share—is carried on through the intermediation of trading firms. These enterprises, headquartered in Tokyo, London, New York, Frankfurt, Brussels, Zug (Switzerland), Helsinki, Paris, or other metropolitan centers, maintain offices or agencies in all of the principal producing and consuming centers. They deal in a whole range of minerals and are familiar with customer requirements of quality and physical characteristics. They understand the mechanics of dealing in foreign exchange and of commodity exchange trading. The key to their success lies in their access to information about both production and consumption developments.

Following the break-up of the colonial empires and the granting of independence to most of the developing nations, public-sector companies have taken over the operation of many mineral properties previously owned by private foreign investors. The governments involved have taken an active interest in the marketing of the mineral products. This has introduced a new element into minerals trading because on occasion foreign-policy or social considerations have replaced strictly commercial factors in determining where and how minerals are to be sold.

In the long run, however, market forces will predominate in determining what will happen to mineral prices and where the bulk of mineral trading will occur.

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Chapter 2.4
TAXATION AND DEPLETION

JOHN W. WHITNEY AND GARTH H. SIBBALD

2.4.1 INTRODUCTION

Federal and state tax laws create a complex framework within which the cash flow analysis of a mining project or operation is conducted. Two important components of federal tax law are the capital recovery provisions of the depreciation and depletion regulations. The other important elements of federal taxes are the actual income tax provisions and the minimum tax provisions. Anyone who works for profit needs to be aware of tax concepts in order to determine how they will impact the amount and timing of generated cash flows. The purpose of the chapter is to emphasize the importance of tax planning and to show the effects of taxes in investment evaluation. This chapter describes the federal tax rules that apply to mining and different kinds of state taxes that affect mining operations.

Tax planning requires the legitimate use of all current laws, regulations, and court decisions to reduce taxable income. This is referred to as tax avoidance and enables companies to reduce their taxable income to the minimum allowed by the host country's legal system. Thus careful tax planning permits the retention of a maximum amount of cash for the company to use in its own endeavors.

It is necessary to include the effects of income tax in project analysis. This allows one to evaluate projects in terms of the net-after-tax return on total investment. There are several technical reasons for inclusion of the effects on income taxes, but the most important is to ensure a realistic estimate of the expected return on investment. For example, proposals that involve a mix of foreign and domestic operations will be subject to different tax rates and tax rules than either wholly foreign or wholly domestic corporations. Also there may be alternatives that require choices between projects that are wholly foreign vs. projects that are wholly domestic. In either case, it is necessary to analyze the project returns on an after-tax basis to avoid unrealistic rate-of-return comparisons.

The two most important US tax elements for preliminary financial analysis are depreciation and depletion. Although treatment of these two elements is relatively straightforward, the choice of the depreciation method will often depend on the anticipated mine life, the discount rate, and expected profitability. This is the reason that it is important to have a clear understanding of how depreciation and depletion are calculated and how they interact to affect the net present value and the internal rate of return.

Two categories of expenses, (1) exploration and (2) development, incurred by mineral exploration companies are given special tax treatment. Treatment of these two categories for tax purposes is summarized in this chapter. It is also important to emphasize here that oil and gas exploration expenditures are treated somewhat differently from exploration expenditure for other minerals. This chapter does not provide an analysis of the treatment of expenditures incurred in exploration and development of oil and gas.

Investment strategy and mine financing are discussed in Chapter 2.5 and Section 6.

2.4.2 FEDERAL INCOME TAXATION

2.4.2.1 Exploration Expenditures

Exploration expenditures are defined as expenses incurred by a mining organization in determining the location and quality of mineral deposits that had not previously been commercially exploited. As a general rule, exploration expenditures are capitalized and recoverable through depletion. Taxpayers do have the option to deduct a significant percentage of the exploration expenses in the year they are incurred and then recapture these expenses when the production stage is reached or when the property is transferred.

Two relatively separate stages in the exploration process can be identified, (1) the pre-exploration and (2) subsequent exploration activities. *Pre-exploratory costs* include the costs of reconnaissance, that is, the cost of surveying wide areas to find specific areas of interest. The Internal Revenue Service has ruled that pre-exploratory costs incurred in a project are to be related to the property ultimately acquired; these costs may be allocated on an acreage-based formula. This allocation of costs serves to establish part of the cost basis for the property.

Exploration expenditures are high-risk investments that are essential for the growth and long-run existence of the mining industry. For tax reporting purposes, exploration expenditures are those expenditures, excluding cost of depreciating property, that are paid or incurred by the taxpayer before the beginning of the development stage for the deposit. The *development stage* is deemed to begin when mineral deposits are determined to be of sufficient quality and quantity to reasonably justify commercial exploitation by the taxpayer.

Exploration costs include all expensable costs incurred prior to entering the development stage. These costs include such items as core drilling to ascertain the existence of commercially marketable ore and underground shafts, drifts, and crosscuts for the same purpose. Exploration costs also include the cost of labor, administrative overhead, depreciation of equipment used in exploration, and support facilities.

The taxpayer may elect to capitalize all exploration costs to be recovered from depletion. Under this option, exploration costs are capitalized and included with property costs to establish the cost basis of the property. This cost basis is then used to determine the appropriate cost depletion rate for the mineral deposit.

Exploration costs also may be deducted currently instead of capitalizing them. If this option is taken, the costs must be recovered when the mineral deposit has reached the producing stage. The amount of expenditures during the exploration stage that is deductible depends upon when the expenditures were incurred. Pursuant to the Tax Reform Act of 1986, expenditures made after 1986 are subject to a 70% current deductibility limitation. For expenditures incurred after 1984 but before 1987, deductibility is limited to 80%; for taxable years subsequent to 1982 but before 1985, the corporation can deduct 85% of exploration expenses. The remaining balance can be deducted ratably over the 60-month period beginning with the month in which the costs are paid or incurred.

There are two possible methods for recovering exploration costs: (1) they may be recorded as income in the first year of production, or (2) they may be recovered by disallowing the depletion allowance until the cost is recaptured.

If the taxpayer elects to recapture exploration costs through income, he/she should include in gross income for the taxable year an amount equal to the adjusted exploration expenditures applicable to the mineral property in the year that it reaches the producing stage. The amount so recorded in gross income is then capitalized as part of the cost basis of the mineral property and will be recoverable through depletion.

If the exploration expenditures are not included in income in the first producing year, they must be recaptured directly through the allowance for depletion. The deduction for depletion with respect to the property will be disallowed until recovery of the total amount of depletion expenditures that has previously been expensed.

2.4.2.2 Development Expenditures

Mine development expenditures include those necessary to gain access to an ore body in the preproduction stage, but after the existence of ores in commercially marketable quantities has been disclosed. Development expenditures also include those expenditures necessary to extend production in an existing ore body.

Development expenditures may be either deducted currently or deferred and amortized as ore is sold. Unlike exploration expenditures, development expenditures are not subject to recapture so that it is advantageous to classify expenditures as developmental rather than exploratory.

For development expenditures incurred in taxable years beginning after 1982, the amount currently deductible is limited to 85% of the expenditure. The remaining 15% must be capitalized and recovered over a five-year period. For expenditures incurred after 1984, the amount to be capitalized is increased to 20%. The capitalized development costs, however, are not part of the basis of the property for depletion purposes. For expenditures incurred after 1986, the amount allowable as a deduction for domestic mine development must be reduced by 30%. The amount not allowable as a deduction for any taxable year after 1986 may be amortized over a 60-month period beginning with the month in which the costs are paid or incurred.

2.4.2.3 Depreciation

Depreciation is the periodic write-off of the costs of items of property and certain other long-lived tangible assets. It denotes a periodic cost allocation against revenue of such tangible assets as buildings, machinery, and equipment. Depreciation is also the process by which the capitalized cost of a fixed asset is deducted from taxable income.

The cash flow analysis is primarily concerned with the tax savings that are a result of capital recovery provisions, and the *net present value* of the tax savings. The method used to calculate depreciation for tax purposes determines the value of the resulting tax savings.

The modified accelerated cost recovery system (MACRS) is mandatory for most tangible depreciable property placed in service after Dec. 31, 1986. Under MACRS, the cost of eligible property is recovered over a 3-, 5-, 7-, l0-, 15-, 20-, 27.5-, or 31.5-year period. The class-life of an asset determines its recovery period, the method of depreciation used, and the applicable convention.

The cost of property in the 3-, 5-, 7-, and l0-year classes is recovered using the 200% declining-balance method over three, five, seven, or ten years, respectively. The cost of 15- and 20-year property is recovered using the 150% declining-balance method over 15 and 20 years, respectively. Instead of the applicable depreciation method, taxpayers may elect to claim straight-line MACRS deductions over the regular recovery period.

In addition to the normal depreciation methods, mining companies may also use the ore-reserve or units-of-production method. This method of computing depreciation is similar to the cost depletion computation, except that a different cost basis is used in determining the depletion unit.

2.4.2.4 Depletion

Definition: Congress has recognized the unique risks and special nature of the mining industry by granting it a percentage *depletion allowance* that is not limited to the cost basis of the property. In general, annual depletion deductions are allowed only to owners of an economic interest in mineral deposits or standing timber.

In the case of mineral resources, the allowance for depletion may be calculated either upon the adjusted depletion basis of the property (cost depletion) or upon a percentage of gross income from the property (percentage depletion). The Internal Revenue Code requires taxpayers to use the method that yields the largest deduction for any taxable year. It is based on the concept that mineral deposits are wasting assets that are difficult to replace.

Cost Depletion: Cost depletion is based upon the cost of mineral property, the number of units of mineral or contained metal sold during the year, and the number of units of mineral or contained metal remaining in the deposit at the end of the year. The basis that is established for cost depletion in any one year is the undepleted cost basis of the mining property at the beginning of the year.

The cost depletion allowance for any year is determined in the following manner. First, the undepleted cost basis of the property must be determined. Then subtract the accumulated depletion to date from the original cost basis of the property. This must be divided by the total number of units of mineral remaining at the end of the year plus the number of units sold during the year. The result is then multiplied by the number of units sold during the year to determine the cost depletion allowance for the taxable year. The applicable relationship is

$$CD_n = \left(CB - \sum_{i=0}^{n-1} D_i\right)\left(\frac{U_n}{U_n + U_r}\right) \tag{2.4.1}$$

where CD_n is cost depletion allowance in year n, CB is the original cost basis of the property, $\sum_{i=0}^{n-1} D_i$ is accumulated depletion taken in the preceding years (both cost and percentage), U_n is units of metal sold during year n, and U_r is units remaining at year end (units of metal contained in ore reserves).

Each taxpayer who claims and makes a deduction for depletion of mineral property must keep a record of the cost basis for the mineral property and a record of the sum of depletion credits previously taken. No further deductions for cost depletion will be allowed when the sum of the depletion credit equals or exceeds the cost basis of the property.

Percentage Depletion: Percentage depletion is an allowance expressed as a percentage of the gross income from the mine and varies in amount with the type of mineral (see Table 2.4.1). The percentage depletion allowance is limited to 50% of the taxable income from the property before the allowance is deducted. One of the favorable aspects of the percentage depletion allowance is that it is not limited by the cost basis of the property.

The percentage depletion for year n is calculated by multiplying the applicable percentage depletion allowance for the min-

Table 2.4.1. Statistory Depletion Rates for Various Minerals

1. 22%
   a. Sulfur and uranium; and
   b. If from deposits in the United States, anorthosite, clay laterite, and nephelite syenite (to the extent that alumina and aluminum compounds are extracted therefrom), asbestos, bauxite, celesite, chromite, corundum, fluorspar, graphite, ilmenite, kyanite, mica, olivine, quartz crystals (radio grade), rutile, block steatite talc, and zircon, and ores of the following metals: Antimony, beryllium, bismuth, cadmium, cobalt, columbium, lead, lithium, manganese, mercury, molybdenum, nickel, platinum and platinum group metals, tantalum, thorium, tin, titanium, tungsten, vanadium, and zinc.
2. 15%–If from deposits in the United States
   a. Gold, silver, and copper;
   b. Oil shale (except shale described in paragraph 5); and
   c. Domestic iron ore; for tax years beginning after 1983, the corporate statutory percentage depletion deduction for such minerals is to be reduced by 15% of the excess of the percentage depletion deduction over the adjusted basis of the property as determined at the end of the tax year and without regard to the depletion deduction for that year.
3. 14%
   a. Metal mines [if paragraph (1)(b) or (2)(a) does not apply], rock asphalt, and vermiculite; and
   b. If paragraph 1b, 5, or 6b does not apply, ball clay, bentonite, china clay, sagger clay, and clay used or sold for use for purposes dependent on its refractory properties.
   c. Foreign iron ore; for tax years beginning after 1983, the corporate statutory percentage depletion deduction for such minerals is to be reduced by 15% of the excess of the percentage depletion deduction over the adjusted basis for the property as determined at the end of the tax year and without regard to the depletion deduction for that year.
4. 10%
   a. Asbestos (if paragraph 1 b does not apply), brucite, lignite, perlite, sodium chloride, and wollastonite. The depletion deduction for coal (including lignite) is 10%, but for tax years beginning after 1983, the corporate statutory percentage depletion deduction for such minerals is to be reduced by 15% of the excess of the percentage depletion deduction over the adjusted basis of the property as determined at the end of the tax year and without regard to the depletion deduction for that year.
5. 7.5%
   a. Clay and shale used or sold for use in the manufacture of sewer pipe or brick, and clay, shale, and slate used or sold for use as sintered or burned lightweight aggregates.
6. 5%
   a. Gravel, peat, pumice, sand, scoria, shale (except shale described in paragraph 2b or 5) and stone (except stone described in paragraph 7).
   b. Clay used, or sold for use, in the manufacture of drainage and roofing tile, flower pots, and kindred products; and
   c. If from brine wells, bromine, calcium chloride, and magnesium chloride.
7. 14%
   All other minerals, including, but not limited to, aplite, barite, borax, calcium carbonates, diatomaceous earth, dolomite, feldspar, fullers earth, garnet, gilsonite, granite, limestone, magnesite, magnesium carbonates, marble, mollusk shells (including clam shells and oyster shells), phosphate rock, potash, quartzite, slate, soapstone, stone (used or sold for use by the mine owner or operator as dimension stone or ornamental stone), thenardite, tripoli, trona, and (if paragraph 1 b does not apply) bauxite, flake graphite, fluorspar, lepidolite, mica, spodumene, and talc (including pyrophyllite), except that, unless sold on bid in direct competition with a bona fide bid to sell a mineral listed in paragraph 3, the percentage shall be 5% for any such other mineral (other than slate to which paragraph 5 applies) when used, or sold for use, by the mine owner or operator as rip rap, ballast, road material, rubble, concrete aggregates, or for similar purposes. For purposes of this paragraph, the term "all other minerals" does not include:
   a. Soil, sod, dirt, turf, water, or mosses;
   b. Minerals from seawater, the air, or similar inexhaustible sources; or
   c. Oil and gas wells.
   For purposes of this subsection, minerals (other than sodium chloride) extracted from brines pumped from a saline perennial lake within the United States shall not be considered minerals from an inexhaustible source.

Source: Section 613(b) of the Internal Revenue Code.

era1 by the gross income from mining less royalty payments. The amount of 0.5 times taxable income before depletion must be calculated to determine whether this limits the allowable percentage depletion. This is illustrated by the following relations:

$$PD_n = (MPDA)\ (GIM_n - RP_n) \qquad (2.4.2)$$

If

$$PD_n > (0.5)\ (TIBD) \qquad (2.4.3)$$

then

$$PD_n = (0.5)\ (TIBD)$$

where PD_n is the allowable percentage depletion for the mineral sold during the year, *MPDA* is the mineral's percentage depletion allowance (e.g., copper at 15%), GIM_n is gross income from mineral sales in year *n*, RP_n is royalty payments, and *TIBD* is taxable income before depletion.

2.4.2.5 Alternative Minimum Tax System

For taxable years beginning after Dec. 31, 1986, the Tax Reform Act of 1986 replaces the old "add-on" corporate minimum tax with a new alternative minimum tax. The new *alternative minimum tax* is based on a much broader income base than the regular tax because the preferential treatment accorded many items for regular tax purposes is not allowed in the alternative minimum tax system. This system has separate but parallel rules for several significant calculations, including depreciation, amortization and depletion, net operating losses, and foreign tax credits.

The corporation's regular taxable income is the starting point for the calculation of the alternative minimum tax. The amount of taxable income is recomputed to reflect certain adjustments required for determination of the alternative minimum tax. Some of the major adjustments are to depreciation, mining exploration and development costs, and depletion. Alternative minimum taxable income is taxed at the corporate rate of 20%.

Depreciation: For purposes of calculating alternative minimum taxable income, depreciation of most property is based upon use of the 150% declining-balance method over the class life of the property, switching to the straight-line method when it results in a larger allowance. However, if the taxpayer uses straight-line depreciation for regular tax, he must also use that method for the alternative minimum tax.

Most assets used in the mining industry have class lives of 10 years. Equipment used in smelting and refining of metal generally has a class life of 14 years. For regular tax purposes, these assets would generally be depreciated using the 200% declining-balance method or the straight-line method over a 17-year life. An election will allow the l0- and 14-year lives to be used for regular tax purposes, thereby reducing the adjustment necessary for computation of the alternative minimum tax. However, for the depreciation of buildings, the straight-line method must be used for both alternative minimum tax and regular tax purposes.

Exploration and Development Costs: Mining exploration and development costs that are expensed must be computed under 10-year straight-line amortization for purposes of the alternative minimum tax.

Depletion: The excess of percentage depletion allowable for regular tax over the adjusted basis of each piece of property at the end of the taxable year (determined without regard to the depletion deduction for the current taxable year) is treated as a preference item for purposes of computing the alternative minimum tax and must be added back in the calculation of alternative minimum taxable income.

2.4.3 STATE TAXES

The major types of state taxes affecting mineral production operations are property taxes, severance taxes, income taxes, and sales and use taxes.

2.4.3.1 Property Taxes

A *property tax* is an ad valorem tax levied by the taxing authority against the "value" of the property in question. Property taxes can be levied by cities, counties, states, or other taxing authorities such as school districts.

They can be levied against either real or personal property or both. Many states do not have a general state property tax authority and leave property taxation to the counties and local divisions. Many states have a board of equalization, a body which attempts to equalize taxes so that the tax burden is equitably imposed in each of the counties or other divisions within the state.

Real property is taxed in all states; however, some states do not tax personal property, and intangible personal property is exempt from taxation in most jurisdictions.

The amount of property tax to be paid depends on the tax base and the tax rate. Valuation, usually carried out by local or state assessors, determines the tax base. The tax rate, usually expressed in "mills per dollar," is applied to the assessed valuation.

Property is generally assessed at its "fair market value." The *fair market value* is the value which a willing buyer would pay to a willing seller in a "arm's-length" transaction, that is, in a situation in which neither is under any compulsion to buy or sell, and each is reasonably well informed as to the facts having a bearing on value. Other terms used to describe fair market value are "market value" and "cash value." All of these terms mean virtually the same thing.

There are essentially three methods that property appraisers (assessors) use to determine the fair market value of a piece of property. They are the (1) comparable sales approach, (2) replacement-cost method, and (3) capitalization of income method. A variant of the capitalization of income method is the net present value method. With the comparable sales approach, the assessor uses recent sales of comparable property to determine what it would cost to replace the property with a new piece of like property. Then the assessor subtracts depreciation based on the age of the property in question. Under the capitalization of income method, the present value of the annual net income of the property is determined by using a rate of return acceptable to a normal prudent investor. The figure is then capitalized to produce the market value of the property. For example, if the net income of the property is $12,000/year, and the acceptable rate of return to a prudent investor is 12%, the capitalized value of the property is $100,000.

The net present value method is similar to the capitalization of income method, but is based upon projected future operating income. There is a growing trend to use the net present value method for mine property assessment. There can be problems with this method because future mine income is based on metal prices that fluctuate widely from year to year. Projected after-tax cash flow over a mine's life, based upon mineral reserves and historical performance, is discounted using an appropriate capitalization or discount rate. The mine value is the sum of annual discounted after-tax cash flows over the mine life. For example, if the after-tax cash flow of a property is projected to be $12,000/year, 10% is an acceptable rate of return, and reserves for 5 years remain, the net present value for the current tax year is $10,908 + $9,912 + $9,012 + $8,196 + $7,452 = $45,480.

Although in many states, the state constitution and statutes allow property to be assessed at 100% of fair market value, in practice, property is often assessed at something less than 100%. This lesser assessment rate usually varies from 25% to 50% of market value. The assessment rate is referred to as the *assessment ratio.* After the assessment ratio is applied to the property, the tax is levied.

With mineral properties, the valuation of surface improvements, machinery and equipment, and land poses few problems. On the other hand, estimating the value of a mineral deposit itself is not easy. Two main approaches have been used to value ore bodies. One is to base the value of the ore body on its annual proceeds. Production may be that of the year under consideration or an average level over several preceding years. The valuation basis of the ore deposit may be gross proceeds, net proceeds, or some other criterion based on production. Gross proceeds is generally the value of the ore as reflected by the sale price if it is an arm's-length transaction with no deductions for costs of producing the ore. Net proceeds is the value as reflected by the sales price less allowable costs of production. For example, the costs of producing, transporting, refining, and selling the ore are generally considered to be allowable deductions. The property tax is then levied on the assessed value of the proceeds at the prevailing rate for the locality in question. This approach does not include in the valuation any mineral reserves to be extracted in the future, but provides a "realized value" basis for assessment.

An alternative approach is to use the present value of the projected future earnings of the mining operation to represent the value of the ore body. This application of the net present value method requires assumptions regarding mine life, rates of return, and future minerals prices. This type of tax is difficult to administer due to the many poorly defined technical variables that must be included. The principal advantage to taxing authorities of levying a property tax on the present value of mining operations is that it provides a steady source of revenue. The main disadvantage is that it discourages exploration for and development of reserves ahead of actual mining operations. This approach often results in “double taxation” on the equipment and facilities used in the operation. States that have used this method have experienced declines in their mining sectors over the years because it does discourage ore reserve development.

A direct ore reserve tax represents a fixed cost to the property owner. Ore reserve taxes clearly discourage ore reserve development. A tax on the capitalized value of income has the same negative implications although not to the same extent. In states where this latter type of tax has been applied, mine operators have tended to develop only the minimum amount of ore needed to sustain ongoing operations. The lack of long-range ore development due to the reduced ability of mine operators to adjust their output can be inefficient and may lead to premature mine closures during adverse economic periods. Often such mine closures are permanent because the lack of developed reserves makes it difficult for the mining company to justify the financing necessary to reopen the mine.

The net proceeds type of property tax was developed to overcome some of the foregoing problems. This type of tax applies only to operations that are active and that are profitable on a defined basis. The net proceeds property tax is beneficial because it encourages both ore reserve and mine development. However, in states with a corporate income tax, it may result in double taxation of the profits from mining unless there are specific exemptions.

Because taxes on real and personal property represent a fixed cost regardless of profitability, some states permit exemptions for certain classes of personal and/or real mine property. Such exemptions may be justifiable because mining is capital intensive and because the exemptions increase the amount of ore extractable from the mine by reducing fixed costs. Often the loss of primary property tax revenues is more than compensated for by secondary tax revenues and employment benefits.

2.4.3.2 Mineral Specific Taxes

A *severance tax* is a state tax imposed on the severing of natural resources from the land and is based on the value or quantity of production. In some states, this type of tax is given a different name. Instead of being called a severance tax, it may be called a mine license tax, an excise tax, a net proceeds tax, a production tax, or a royalty tax, depending on the label chosen by the legislators of the particular state. In all cases, however, the tax is based on actual output or production.

These types of taxes are usually calculated either as a flat rate per unit of production (sometimes called a “unit” or “specific” severance tax), or as a percentage of the value of the resource produced (sometimes called an “ad valorem” or “percentage” severance tax). The tax base for an ad valorem severance tax is generally either the gross or net value of resources produced or sold.

With a unit severance tax, there may be an escalation clause such that the tax increases with inflation. Ad valorem severance taxes automatically vary with the value of the mineral. Severance tax payments are altered by exemptions, credits, and exclusions in many states. There may be exemptions for minimum levels of production, credits for payments of ad valorem property taxes, and exclusions for small producers.

States differ widely in the definitions of value they use for calculating the base for an ad valorem severance tax. The tax might be imposed on the value of the resource in the ground (net proceeds), at the point of severance (gross proceeds), or after processing and preparation for shipment. The value is greater at each succeeding point. A severance tax based on gross sales value, or value f.o.b. mill, corresponds to the case where the resource is valued after processing and preparation for shipment. A tax based on gross proceeds generally corresponds to the case where the resource is valued at the point of severance. One way to estimate this value would be to deduct processing costs, transportation costs, certain production taxes, rentals, and royalties from the value at the point of first sale. A tax based on net proceeds corresponds to the case where the relevant value is the value in the ground. To estimate value in the ground, all costs of production, including mining costs, would be deducted.

The chief advantages of a tax of this type relative to many property taxes on ore bodies are that it does not discourage exploration for and development of additional ore reserves and it does not require ‘valuation’ of the ore body with all the attendant difficulties that entails. On the other hand, a severance tax does not create a steady stream of revenue since it requires an operating mine coupled with production or ore sales. Furthermore, an output-related proceeds tax will raise the cutoff grade of ore that will be mined, resulting in a waste of low-grade ore.

The net proceeds tax is preferable to other types of severance taxes from a conservation point of view. If a progressive net proceeds tax were imposed, mining operations might seek to enter lower tax brackets by reducing their rates of recovery; but a purely proportional net proceeds tax should have no effect on the rate of recovery. Indeed, a proportional net proceeds tax should have less effect on the production decisions of a mining operation than a property tax or any other type of severance tax.

Severance taxes offer some advantages from the standpoint of ability to pay. They are due only when a mining operation is actually producing, so there is no penalty if there is a strike or shutdown for market-related reasons. In contrast to property taxes, severance taxes offer opportunities to achieve a more nearly uniform distribution of revenues among areas of the state; and a mine located in a district that lacks other business activity is not so likely to be disadvantaged. A net proceeds tax is optional if ability to pay is an important policy criterion because the tax base most closely approximates profits. Only profitable operations will be required to pay taxes. The least satisfactory type of severance tax from an ability to pay standpoint is a high unit tax. With unit taxes, it often happens that an unprofitable operation whose output has a relatively low unit value is taxed at a higher percentage rate than a more profitable mine of comparable size with higher valued output.

A unit severance tax simplifies administrative problems because the rate schedule depends only upon the amount produced. A severance tax based on the value of output is easily determined if there is a market sale, but in some cases there may be no market transaction. Some barite mines, for example, are part of a vertically integrated drilling-fluids service company. In such cases, it is necessary to determine an accounting price for the barite, causing tax administration to become more complex.

Revenue generation is much more unpredictable and volatile with a severance tax of the net proceeds type than with other severance taxes. During periods of low minerals prices, profits and tax revenues also tend to be low. In addition, severance taxes do not help cope with covering pre-operational costs as well as property taxes do. If there is reliance on severance taxes, in the early stages of mine development, tax revenues may be below the levels needed to finance new services required by the community where the mine is being built.

2.4.3.3 Corporate Income or Occupation Taxes

The *income tax,* like the net-proceeds-type severance tax, is based on the profits of mines with certain statutory deductions and adjustments. The income tax is advantageous from the mining operator's point of view because it is assessed only if the operator makes a profit. The income tax has the least effect on direct and indirect costs. Further, it does not affect the cutoff grade of ore. Therefore, maximum ore recovery is promoted. Both the federal and many state governments impose an income tax.

The most significant income tax paid by mining companies is the federal income tax. Several provisions of the Federal Tax Code specifically relate to the minerals industry. These include the depletion allowance and expensing privileges for exploration and development costs. A percentage depletion allowance represents a deduction over the above recovery of actual costs equal to a fixed percentage of gross receipt and is generally viewed as an allowance for the reduction in asset value as mineral resources are removed. The mineral industry is also provided special tax treatment of costs associated with exploration and development. Exploration and development costs may be expensed as incurred. Most operations benefit from the expensing privilege for exploration costs, but only economically profitable operations benefit from expensing of development costs.

State corporate income taxes are applied to net income (income less statutory deductions). Most states use the federal definition of net income, often with certain adjustments, but the tax rate is much less than the federal rate. Items which are usually deductible in computing net income include business expenses, interest paid or accrued, uninsured losses, bad debts, a reasonable allowance for depreciation, charitable deductions, and net loss carryovers. Most states using the federal definition of net income would also allow the federal depletion allowance. States using a different tax base may or may not allow a deduction for depletion. Rates vary from state to state.

Some states allow deduction of federal income tax liabilities. If a mining operation is in the 34% tax bracket, this deduction effectively reduces the company's tax base by about that amount for state income tax purposes. The actual amounts are determined by application of a simultaneous equation.

Corporations doing business and having income in more than one state are generally taxed by the state in question on both income allocated to the state and an apportioned part of total US income under the unitary tax concept. Apportionment is generally standardized among the various states that adopted the "Uniform Division of Income for Tax Purposes Act" and is based upon a formula relating in-state tangible assets, compensation paid, and sales receipts to total tangible assets, compensation, and sales. In general, nonbusiness income, such as rents and royalties and capital gains from transactions in real or tangible personal property, is allocated wholly to the state in which the tangible property is located or from which it was sold. Interest and dividend income is allocated to the state that is the taxpayer's commercial domicile. Normal business income is apportioned among the states in which the taxpayer does business, irrespective of whether each such state imposes a corporate income tax.

The greatest disadvantage of the income tax from the state's point of view is that it does not provide the predictable, steady stream of revenue that the property tax provides. On the other hand, states can encourage mining development through the use of tax incentives such as deductions for exploration and development costs as well as liberal depletion deductions.

2.4.3.4 Sales and Use Taxes

The purpose of the *use tax* is to prevent users from purchasing property from states having no sales tax or a lesser sales tax and bringing it into a state with a higher sales tax. The use tax eliminates this advantage. Items subject to use taxes are generally not subject to *sales taxes* as well, because most states allow a credit for sales or use taxes paid to the state in question or to another state. For example, assume that a mining company is located in state A, which has a retail sales tax of $0.05/dollar. State B's sales tax is only $0.03/dollar. If the mining company purchases office equipment in state B for use in state A, state A would charge a use tax for the difference, that is, in an amount of $0.02/dollar.

Sales and use taxes produce revenues prior to production start-up and so may help defray the costs of new services early in the project life. Sales and use taxes raise the cost of mine construction and equipment and raise the cost of mine operation. To the extent that costs are increased, there will need to be compensating increases in minable grade of the deposit, and the ore reserves that are minable may be reduced. Such taxes also increase the cost of exploration. In areas where mines are already in operation, increases in sales and use taxes may cause a reduction in modernization and expansion and, to the extent that operating costs are increased, may cause mines to close sooner than they would in the absence of the tax.

Another disadvantage of a sales or use tax is relatively high administrative costs. Revenues from this type of tax tend to fluctuate through investment cycles, reaching their highest levels in years when capital spending by the mining industry is strong.

Sales or use taxes will, however, generate a steadier and more predictable revenue stream than income or net proceeds taxes. Because tax payments are not necessarily directly tied to profits, sales or use taxes do not satisfy most equity criteria as well as income taxes do.

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Chapter 2.5
INVESTMENT STRATEGY FOR MINING PROJECTS

DENNIS ARROUET

Investment decisions in the mining industry, as in any other, normally focus on numerical methods comparing alternative investments. While these methods are important, the process of investment strategy, making the investment, and operating a project during its economic life, involves much more. Many key people in various disciplines provide varying degrees of input over the life of the project, and their input can play a major role in the investment strategy process. This chapter reviews principles and concepts covering investment within a home country and in foreign countries, showing how these many and disparate elements blend together and change over time. It is complemented by coverage in Section 6, Mine Evaluation and Investment Analysis.

2.5.1 MINING INVESTMENTS

Investments in the mining industry have many characteristics not found in most other industries. These include the following.

2.5.1.1 Finding or Exploration

Either generative or opportunistic exploration must be funded for a company to get/renew investments that will be profitable in the future. These programs, which may be structured as individual efforts, partnerships, or joint ventures, take time to develop good prospects/projects.

2.5.1.2 Size

In the past, projects had high ore grades and were easily mined and processed. In the latter part of the 20th century, the norm for mining projects was lower-grade deposits, large-scale mining, and processing using high technology equipment. Often these projects required infrastructure to make them feasible. As a result, mining became highly capital intensive. Projects costing in the hundreds of millions of dollars are more and more common.

2.5.1.3 Life

Projects generally have very long lives. Many years pass between first finding ore-grade material, proving up the reserves, planning the project, and construction. Additional years must pass before capital costs are recouped, and thereafter even more years are needed for equity investors to make an adequate return.

2.5.1.4 Product

By their nature, mining projects produce a major product, although byproduct or coproducts are often produced. In general, the product produced in the mining process is not sold to the final customer. Rather, this raw product is often sold to others who, after adding value to it, may or may not sell to the final customer. As a result, demand for "raw" mining products is "derived" and therefore subject to price and volume swings in addition to those generated by the final customer.

2.5.1.5 Profile

Projects are usually "high" profile wherever they are located because they are "natural resources." In some countries, they often get much more media and political coverage than they might elsewhere. If this situation exists, or may come to exist, attitudes and business ground rules can change very quickly; what appears to be a good investment one day can quickly become less attractive.

2.5.2 INVESTMENT STRATEGIES

Historically, mining investors explored, developed, and operated until the deposit was exhausted. The only question was proving up enough ore to support a profitable investment. While this strategy has been and can be successful, it assumes little change during the life of the project.

Recent dramatic changes in the price and volume of mining products, operating costs, and government attitudes, among other considerations, have led to the reexamination of a once successful strategy. Under the newer thinking, general approaches to a successful mining investment now include these strategies.

1. *Cost Leadership.* The objective is to have all operating costs for a project below that of average costs in the industry. Low costs result from a combination of ore grade, mining difficulty, dilution, economies of scale, development of the operation including infrastructure, technology, low cost of utilities and materials, location relative to markets, and access to low-cost transportation.
2. *Differentiation.* Here, the product offered is high valued and special to the market served.
3. *Focus.* The objective is to fulfill a market need ignored by others in the industry.
4. *Coattails.* When an industry has a single or few competitors, the "new entrant" may be accepted; economic return to the dominant producer(s) can be better (but less than before the new entrant), if the dominant producer(s) cede portions of the market and maintain price leadership rather than lower price to stop the new competitor.
5. *Volatility Exploitation.* The emphasis is on low fixed costs and low start-up and shutdown costs. As a result, in periods of high prices, a mine may be operated very profitably. While this strategy may apply to a good mining project that has successfully operated for many years, investors may be able to buy shut mines at very low cost, operate the property for a short time, and have a very attractive investment. Alternatively, developing ore bodies with these quick in-and-out-of-operation characteristics could prove profitable, depending on circumstances and assumptions.

2.5.3 CALCULATION METHODS

The evaluation of a mining investment starts with the concept that the rate of return over future periods measures the financial productivity of capital. In other words, a return of *x%* or more would be adequate for an investor. Return has been defined many ways over the years. Some of these measures and their peculiarities are detailed in the following.

Modern financial theory for measuring return on investment uses an interest factor to equate cash inflow (outflows) in different time periods. Thus $1 invested at 10% increases to $1.10 at the end of the first year, $1.21 after the second, etc. Similarly, $1.21 received two years hence has a present value of $1 today, "discounted" at 10%.

Mathematically,

$$FV = S_1 (1 + i) + S_2 (1 + i)^2 + \ldots \tag{2.5.1}$$

$$PV = \frac{S_1}{(1 + i)} + \frac{S_2 +}{(1 + i)^2} \ldots \tag{2.5.2}$$

where FV is future value, PV is present value, S is cash inflow (outflow) per period, and i is interest or discount rate. Thus investment returns are calculated by equating cash outflows for investment to cash inflows from operations (revenues less all cash operating costs, interest, taxes, maintenance and capital expenditures, debt repayment), discounted at an appropriate interest rate.

Example 2.5.1.

Project data (in million dollars)

Investment year 1 = $10, year 2 = $10	$20
Operating cash flow year 3, 4, and 5 = $10	$30
Depreciation—life	3 years
—amount	$20
Debt financing	none
Capital investment after start up	none

Solution.

Depreciation per year	$6.7
Average investment. Investment less one-half total depreciation	$10
Average operating cash flow per year	$10
Average net income. Average operating cash flow per year less average depreciation per year	$3.3

	Investment and Operating Cash Flows	Present Value: $1 discounted at 10%*	Present Value: Revalued Cash Flow
Year 1	(10)	0.909	(9.2)
2	(10)	0.826	(8.3)
3	10	0.751	7.5
4	10	0.683	6.8
5	10	0.621	6.2
			3.1

	Investment and Operating Cash Flows	Discounted Cash Flow: $1 discounted at 17.8%	Discounted Cash Flow: Revalued Cash Flow
Year 1	(10)	0.849	(8.5)
2	(10)	0.721	(7.2)
3	10	0.612	6.1
4	10	0.519	5.2
5	10	0.440	4.4
			0.0

*Discount factors assume revenue received once per year. Year 1 equals 1 divided by 1.10; year 2 equals 1 divided by 1.21, etc.

Returns

Accounting

Return on investment	$\frac{\text{Average net income } 3.3}{\text{Investment } 20.0}$	= 16.5%
Return on average investment	$\frac{\text{Average net income } 3.3}{\text{Average investment } 10.0}$	= 33.3%
Payback:	Years needed for cash inflows to equal cash outflows	2 years

Interest

Present value	10% interest rate	= $3.1
Discounted cash flow return		= 17.8%

The controversy over what measure to use has largely been resolved in favor of the discounted cash flow method. Each individual or organization may use one or a number of these return measures to increase communication of "return" to the many parties involved with the investment decision. Which measure(s) parties use relates to factors peculiar to their own circumstances.

The sophisticated decision maker knows that behind these precise calculations are tangible and intangible data that are uncertain. To cope with this uncertainty, decision makers use a number of techniques including the following:

1. *More Accurate Forecasts.* A worthy objective, but uncertainty is not addressed.
2. *Higher Cutoff Rate.* With a higher cutoff rate, a degree of risk is recognized as part of an investment. However, the probability of returns exceeding the threshold is not recognized.
3. *The Base and Most Likely Case.* Returns are calculated using "averages" for input variables. The base case is presumed to be conservative, that is, low risk by implication. The most likely case is moderate risk.
4. *"What If" Scenarios.* A series of returns are calculated using averages for all but one variable. For example, the base case return is run at sales increases/decreases of 5%, l0%, and 15%. While this methodology shows the sensitivity of returns to change in specific variables, the likelihood of these events is not quantified.
5. *Simulation.* Probability curves on each input variable are developed based on estimates of those most knowledgeable in a given area, for example, operating cost, selling price, etc. Thereafter, a probability is randomly selected from each probability curve and then used to calculate a discounted cash flow rate of return. Then the process is repeated many times. The probability of all anticipated returns can then be graphed as shown in Figs. 2.5.1 and 2.5.2. Then assessing the project and comparing it to other projects is easier and more meaningful.

2.5.4 COST OF CAPITAL

For an investment to add value, it must earn more than the cost of capital—the cost of debt and equity each weighted based upon the proportions of debt and equity the corporation targets for its capital structure over the investment period.

While there are several models for calculating the cost of capital, each at best is an estimate. The capital asset pricing model that follows is the most widely used formulation. *For debt cost,* calculate interest on incremental debt with a maturity similar to the investment less tax savings generated by deducting

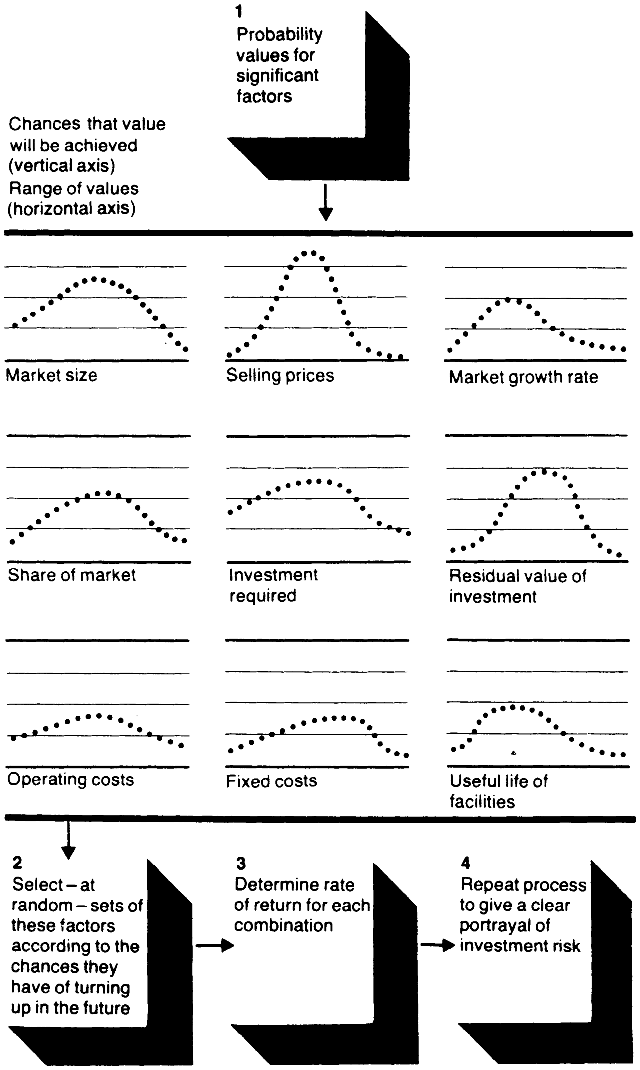
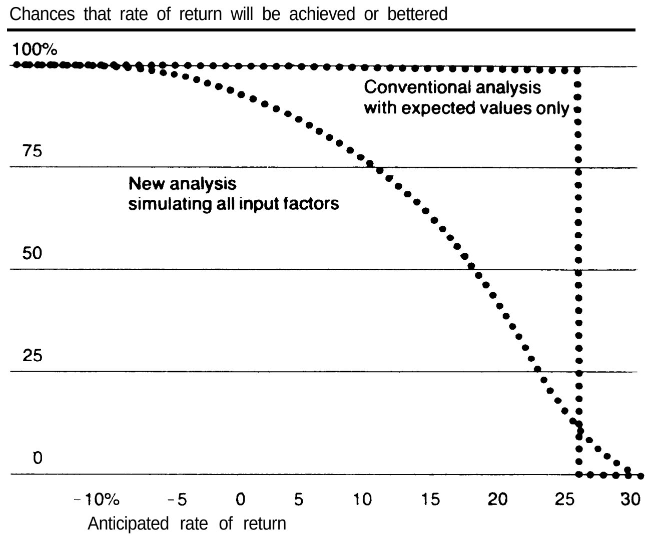


Fig. 2.5.1. Simulation for investment planning (Hertz, 1979).



Percent return	Probability of achieving at least the return shown
0%	96.5%
5	80.6
10	75.2
15	53.8
20	43.0
25	12.6
30	0

Fig. 2.5.2. Anticipated rates of return under old and new approaches (Hertz, 1979).

interest from taxable income. *For equity cost,* calculate the risk free rate of return, the yield on government securities, plus an equity risk premium, for investing in equities. The yield on government securities compensates debt investors for the "real" interest rate plus the expected inflation rate. The equity risk premium compensates the equity investor for added risk and is equal to the expected equity market return less the risk-free rate multiplied by the riskiness of the stock or the beta. Betas are calculated by running linear regression analyses between past returns for a stock and past returns for the market. Then the volatility of the stock is compared to the volatility of the market. Thus a stock with a beta of 1.20 is 20% more volatile than the market. Betas are published by Value Line, Merrill Lynch, and other financial institutions.

In one of the classic studies of long-term investment returns, Ibbotson and Sinquefield (1989) calculated the risk and return of stocks, bonds, and treasury bills and the rate of inflation for 1926–1987. According to this study, the historical (1926–87) arithmetic average equity risk premium of stocks vs. government bonds was 6.8%. Since the equity risk premium is the average annual difference between the rates of return of stocks and government bonds, it is possible to use this factor in calculating a firm's equity cost regardless of the absolute level of interest rates. Specifically, the equity risk premium is multiplied by an individual stock's beta and the result is added to the current risk-free rate of government bonds to determine that company's equity cost.

Example 2.5.2.

Project Data

Risk free government interest rate		9%
Interest rate on new borrowings		10%
Income tax rate		40%
Expected return on equity market		15%
Beta		1.5
Capital mix over investment horizon	debt	40%
	equity	60%

Solution.

	Cost *(%)*	Weight	Weighted Cost
Debt	10% – 10% (0.4)	40%	2.4%
Equity	9% + (15% – 9%) 1.5	60%	10.8%
	Cost of Capital	100%	13.2%

Investment decision making involves more than just deciding to make a project expenditure or not. The process involves determining the amount of acceptable project risk, the financial capability of the investing organization, and the risk profile of each investment in its portfolio. Every situation needs to be analyzed separately.

The corporate objective is to maximize earnings (over the cost of capital) commensurate with acceptable risk. However, at some point in a corporation's life cycle, the risk profile is altered due to price and volume changes, competitor actions, etc. These dynamic changes in the corporation's risk profile are also present in the risk profile of an individual project.

A project's risk exposure is usually greatest prior to completion. But the exposure varies over time, as one would expect, according to the initial risk profile of the project and subsequent changes in the dynamic variables affecting the project. The project's risk profile is superimposed on that of the company, and the relative size of the project will affect its potential impact on the entire company. Upon completion, the project will have a proportionate impact on the overall risk profile of the company and, ideally, the periods of adverse returns on the project will not coincide with the periods of financial difficulty for the rest of the company. Otherwise, a restructuring of the project and/or the company may be necessary. Thus, the corporate decision-making process must explicitly integrate the project risk profile with the corporate profile.

The decision process also encompasses the entire life of an investment. The find, invest, and hold strategy implies one investment decision. However, in a more volatile world, knowledgeable decision makers keep track of the value of their investment with a view to capturing excess valuation. This "recapture" changes the capital structure and the risk profile. It may be done, for example, through the sale of a partial or whole interest in the investment; the excess can be used to adjust the risk profile of the project and thus the corporation and to create new value by developing or buying undervalued investments, repaying financial commitments or paying shareholders.

2.5.5 DECISION MAKING

Decision making for a mining investment is a process. It involves decisions within many separate disciplines that are then linked together. The initial economic decision is to invest or not to invest and thereafter to grow, maintain, or disinvest.

Good communication within the organization and with outside consultants is extremely important. It speeds decision making and it minimizes misunderstandings. But often it is overlooked as a key element that helps the decision process progress smoothly.

Also poor negotiation with outside parties can lead to major problems and unforeseen penalties. The process is critical to every aspect involved in a project. Experienced personnel have a better understanding of what to expect, how to act, and what to demand. They pay attention to:

1. *Organizing Their Positions.* They plan for changing negotiation strength, interference by others in the organization, insufficient internal communication, etc.

2. *Empathizing with Others.* They place themselves in another's position to understand different ways of thinking; they recognize the need for "saving face."

3. *Determining the Role of Government.* They try to understand the politics, the role for private enterprise in the country, status of businessmen, government involvement in negotiations, etc.

4. *Focusing on the Decision-Making Process.* They weigh economic and political criteria, the role of personal relations and personalities and the amount of time assigned to negotiations by others.

As the decision-making process approaches the go/no-go decision, senior management and finally the board of directors become more and more involved. By this time, all work is reviewed to check for consistency, risk, and possible omissions or areas that need more work pending the final decision. "Due diligence" is an investigatory safeguard and is particularly important. It covers reserve verification, project risks, projected capital and operating costs and their variability with volume, competitive assessment, market analysis, and financial analysis. The due diligence process is normally conducted again when, and if, potential equity investors and/or project finance are needed. Then money providers are given up to date factually correct data on the project together with a discussion of potential risks.

2.5.6 PROJECT FAILURES

While failure only happens to others and will not occur in a well-planned project, history teaches us that financial failures do occur, often despite the best plans. Indeed, the risk of failure may increase in the future. First, a number of metal and non-metal mining products that have been relatively price stable now show more significant price volatility (e.g., gold, aluminum, iron ore). Second, low price cycles are relatively longer at lower levels—at least recently—thereby increasing financial stress. Third, increased costs per unit of production and relatively higher interest rates make financial returns more difficult to achieve. Finally, the frontiers for "new" mineral deposits are increasingly in remote areas.

The causes for financial failure of investment in projects can include one or more of the following:

1. Delays in completion causing delay in revenues.
2. Unplanned capital cost overruns.
3. Technical failure.
4. Failure of contractor(s) and supplier(s).
5. Increased price or shortage of raw materials.
6. Estimated grades and amounts unavailable due to insufficient drilling, complex geology, errors, unforeseen faults, rock competences, etc.
7. Technical obsolescence.
8. Operating failure.
9. Loss of competitive position in the market.
10. Uninsured casualty losses.
11. Force majeure.
12. Refinancing risk.
13. Increased taxes, direct and indirect.
14. Foreign exchange, artificial rates, availability, etc.
15. Government interference, changes in laws, rules, regulations.
16. Expropriation.
17. Poor project management.

In hindsight, project failures fit into "nice" categories shown previously. Unfortunately, during a project's demise, the vision is rarely clear. While a check list can be helpful, even in retrospect the root causes are often overlooked. These root causes include:

1. The risk profile for each business variable (reserves, capital and operating costs, competition, etc.) changes during the life of the project.

2. Sponsors often neglect the vested interests of the project decision makers, whose future may be enhanced by a project go ahead, especially when excessive optimism is expressed.

2.5.7 PROJECT STRUCTURE AND PARTICIPANT CHECK LISTS

In order to minimize the downside risk of a project, it is important to focus on certain key issues at the outset. These issues, which relate to the structure of the project and the operational aspects of the project, are summarized in the following in check list format (also see Fig. 2.5.1).

2.5.7.1 Direct Owners

Parent or holding company structure can have undesirable impacts on financing, taxes, risk profile, future restructurings.

2.5.7.2 Ultimate Owners/Completion Guarantors

Completion guarantors could include third parties such as suppliers, offtake purchasers, and governments (particularly in the case of project infrastructure).

2.5.7.3 Domestic Bank Lenders/Guarantors

Domestic bank credit or lending is a major feature of most project financings. Domestic experience with other projects, local practices, etc., can assist in getting the optimal finance package. In addition, domestic institutions can help support the project with foreign lenders.

2.5.7.4 Foreign Bank Lenders/Guarantors

Foreign lenders include all lenders not domestic to the project. Many of these institutions are chosen to lead a project financing if they are well established domestically. In large financings, representation by institutions from many countries is encouraged in less-developed areas. The theory is that the local governments will not expropriate, embargo funds, etc., for fear of reduced funds being available to the country.

2.5.7.5 Guaranteed Lenders

These are generally export credit agencies, regional development banks, other foreign government lenders, institutional lenders, and occasionally debt participants in leveraged leases. The parties advance funds under guarantees or standby letters of credit, which take the project risk and charge a guarantee fee. Concessional rate and term financing is generally achieved in this way, since lenders in this category will not accept project risks.

2.5.7.6 Equipment Suppliers

Use of equipment from different countries should be evaluated. Prices and terms are often more attractive from one country vs. another.

2.5.7.7 Lessors

Leveraged leasing is often used as part of a project financing. Amount and terms will determine the use of leases.

2.5.7.8 Offtake Purchasers

Marketing arrangements can be a critical part of the credit supports needed to finance a project, especially where floor prices or minimum offtake levels are needed to meet debt service. Involving purchasers and lenders or their advisors in the early stages of market planning can be critical.

2.5.7.9 Engineering Consultants/Contractors

Consultants often establish the technical, marketing, and financial feasibility of major projects Sometimes an overall project feasibility consultant is used, but the trend has been towards using specialists.

2.5.7.10 Auditors

Auditors (for accounting and taxes) can provide useful information on the proposed ownership and financing structures. If off-balance-sheet accounting treatment is planned, auditors should be involved early in the project.

2.5.7.11 Legal Counsel

Project development inevitably involves lawyers. Their areas of involvement include agreements with customers, major suppliers, consultants, financial institutions, investors, other partner/venture owners, governments, etc. In most situations, lawyers should have prior mining project experience.

2.5.7.12 Financial Advisors

Investment banks, merchant banks, and commercial banks have undertaken advisory roles to determine the optimal financing for mining projects. For a fee, they will screen, select, and negotiate terms and conditions with financial institutions supplying funds. In some cases, they will supply their own funds.

2.5.7.13 Federal, State, and Local Governments and Their Agencies

Numerous approvals are necessary to obtain a mining lease to construct and operate a mine. Approvals are normally needed before loan drawdowns, and all are required before completion supports are released by lenders. Lenders want to minimize the chance that government intervention will frustrate a project's completion and operation.

2.5.7.14 In-house Expertise

Skills are needed in the areas of geology, mining, milling, metallurgy, operations, maintenance, taxes, legal, financial, governmental, etc.

2.5.8 PROJECT CHECK LIST

Relative to this check list, see Fig. 5.2.2.

2.5.8.1 Technical Considerations

1. Land and minerals: lease, ownership, title, royalties, special arrangements.
2. Reserves: geologic setting, ore types, geologic reserves vs. mineral reserves, proven vs. probable reserves, drilling data, sampling, methodology of reserve calculation.
3. Mining plan: forecasts of grades and material movements, geotechnical test results, equipment to be used, personnel requirements and availability, design assumptions.
4. Processing plan: plant layout, equipment needed, manpower requirements and scheduling, power requirements.

5. Infrastructure: water/power supplies and service availabilities, transport facilities, maintenance workshops, labor housing.

6. Construction: realistic scheduling allowing for weather, terrain, personnel requirements, infrastructure needs, labor agreements.

7. Environment: studies, reports, approvals needed. Limitations and controls affecting: air, water, solid and liquid waste, land, noise, ecology, power, hospitals, schools, roads, community services.

2.5.8.2 Products and Markets

1. Customer buying considerations: spot or term contracts, price—negotiated,tied to other factors such as cost escalators, f.o.b., c.i.f., exchange rate fluctuations and adjustments, volume minimums and maximums, quality premiums and penalties, chances to sell upgraded products, service, engineering, credit, transport.

2. Customers: number and size, location and growth prospects, relationships with other suppliers.

3. Past and prospective price and volume patterns through a complete economic cycle for domestic and world product market served. Usage changes due to obsolescence, use of competitive materials, technological innovation.

4. Competition: existing and potential.
   a. Who are they? What are their strategies for the future?
   b. Current capacity utilization and planned expansions.
   c. Price discipline, price leaders, relationship of price and inventory levels, use of contracts, terms and conditions.
   d. Cost curve for the industry—compare to prior and projected prices.
   e. Project strengths and weaknesses vs. competition.

5. Government regulations on export, approvals, tariffs.

6. Potential product liability.

2.5.8.3 Operating Considerations

1. Management: experience, responsibilities and pay of key people.

2. Labor: skills, availability (now and in the future), union considerations including strikes, prior work practices, pay, incentive compensation, production bonuses.

3. Machinery and equipment: operating availability and costs, flexibility to expand capacity, technical and economic obsolescence.

4. Materials: availability, lead times, cost volatility, purchasing practices.

5. Health, safety, security, and employee rights.

6. Maintenance: onsite vs. offsite workshop, availability of special parts and skilled labor, scheduling.

7. Inventory: amounts and locations, on-site and/or at suppliers.

8. Industry norms: reasons for under/over performance, programs to enhance, maintain or upgrade vs. norms.

9. Environment: studies, reports, approvals needed. Limitations and controls affecting air, water, solid and liquid waste; land; noise; ecology; power; and hospitals, schools, roads, and community services.

10. Equal opportunity employment.

2.5.8.4 Financial Considerations

1. Capital availability: sources, currencies, cost, terms, conditions. Need for refinancings.

2. Capital tied up in assets such as in accounts receivable and inventory, investment in facilities where services can be supplied by others.

3. Repayments: schedules, formula and interrelation with price, volume and margins.

4. Interest rates: variability and protective options.

5. Fees: front-end and continuing.

6. Taxes: land, use, income, royalties, special, direct, indirect.

7. Insurance: costs and coverage.

8. Leases: terms, and conditions on operating and financial leases.

9. Employee costs (other than wages): medical, dental, pensions and other contractual benefits, funding assumptions and requirements.

10. Foreign exchange: availability, quotes, special transactions.

11. Inflation: inclusion in capital and operating costs.

12. Capital additions and replacements.

13. Cost of outside services, travel, and home office.

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Chapter 2.6
THE NATIONAL DEFENSE STOCKPILE

J. WAYNE KULIG

2.6.1 INTRODUCTION

The current Strategic and Critical Materials Stock Piling Act of 1979, as amended, provides for the acquisition and retention of strategic and critical materials in a National Defense Stockpile (hereafter referred to as the Stockpile). The stated purpose of the Stockpile, found in Section 2 of the Act, is "to decrease and to preclude, when possible, a dangerous and costly dependence by the United States upon foreign sources for supplies of such materials in times of national emergency."

The National Defense Stockpile, as planned, should be sufficient to sustain the United States through a three-year national emergency. The Stockpile does this through its inventory of strategic and critical materials and through activities that encourage the conservation and development of scarce domestic materials.

The Stockpile and its related efforts at conservation and development are part of this nation's industrial base and part of its national defense arsenal. The existence of the Stockpile is a visible sign to our allies and our adversaries that this nation has the reserves to sustain the industrial base during an extended national emergency.

To better understand the US Stockpile program and the foundation of its policies, we should first review the history of the Stockpile in the United States.

2.6.2 A BRIEF HISTORY OF THE STOCKPILE

Throughout the history of the Stockpile in the United States, two factors, other than its central purpose, stand out: the far-reaching impact of establishing a stockpile and the changing nature of the program.

Although the National Defense Stockpile is maintained for defense purposes, its presence affects more than the defense sector. Purchases and disposals of Stockpile commodities directly affect the US and the international mining and minerals industry. In fact, the stockpiling program has been a recognized factor in the national economy.

Another salient feature of the Stockpile program is its variability. The Stockpile is not static, but continually shifts to meet projected demands. Throughout its history, the Stockpile has been adjusted to adapt to changing conditions. Changing projections of defense need, obsolescence of materials resulting from new technology, development of new sources of supply, and changes in Stockpile policy regarding the adequacy of the Stockpile have kept the program in a constant state of review.

2.6.2.1 Need for a Stockpile

Although the United States has considerable mineral resources, it is far from being self-sufficient, particularly in time of war. The advent of World War II dramatically brought to the attention of the US government and the US public the need for a cache of materials that could be drawn on in an emergency.

Prior to the war, a law permitting stockpiling was already on the books. In 1939, Congress had enacted legislation that provided for acquisition of certain strategic and critical materials. Shortly thereafter, the 76th Congress had allocated $70 million in appropriations and advance contract authorizations in anticipation of a loss of important imports that could occur if Japan conquered parts of Asia or if war broke out in Europe. Little material was accumulated for a stockpile under that act, and when the war broke out, materials were needed badly for immediate use rather than for stockpiling.

During the war, the United States found itself short of some important commodities and was forced to resort to substitution and rationing to ensure a sufficient supply of commodities to support the war effort. Some valuable resources, such as energy supplies and equipment, were diverted for use in stepping up domestic mineral production of items in short supply. Thus, at the close of the war, the United States was ready to adopt a more formal concept of stockpiling.

2.6.2.2 Stock Piling Act of 1946

At the close of World War II, with the renewed interest in a stockpile, the US government sought new legislation for ensuring a constant supply of strategic and critical materials in case of a national emergency. Congress prepared to revise the 1939 provision in favor of provisions that would better meet the needs of national security. But the proposed new provisions brought a host of considerations that had to be adequately addressed.

One consideration was that of who should control the Stockpile. The question stemmed from what purpose the Stockpile should serve. The expressed purpose was that of providing for a national emergency, so the military seemed to be the logical controller.

Another issue that arose was whether the Stockpile should be a factor for consideration in international, political, and economic policy and whether it should function in compliance with international raw materials controls and price stabilization agreements. In this case, the Department of State should be involved.

Then, again, the domestic ramifications of establishing a Stockpile and meeting the needs of the US mining industry had to be considered. These subjects were of importance to the Departments of Commerce and the Interior.

After much debate, a stockpiling act was created that reflected a compromise among the interested parties. The Secretaries of War, Navy, and Interior were authorized "to . . . determine, from time to time, which materials are strategic and critical . . . and to determine, from time to time, the quality and quantities of such materials which shall be stockpiled."

Responsibility for performing some major functions of the Stock Piling Act was originally given to the Army-Navy Munitions Board and then to the National Security Resources Board. Later, the responsibility was shifted to the Treasury Department. A few years later, partially as a result of a Hoover Commission study, the functions that were being carried out by the Treasury were merged with the newly created General Services Administration (GSA) that was to be a major focal point for federal procurement. Operational responsibilities transferred to the GSA included the following:

1. Purchasing strategic and critical materials, as directed.

2. Storing, securing, and maintaining strategic and critical materials.

3. Arranging commercial refining or processing of materials, as directed.

4. Rotating and replacing materials as necessary to prevent deterioration.

5. Conducting Congressionally authorized disposals of excess materials that were no longer needed.

The Act also contained a provision to prevent undue disruption of the usual markets of producers, processors, and consumers by preventing Stockpile materials from being released indiscriminately. The Stock Piling Act was signed by President Truman on July 23, 1946.

While the Act in itself was straightforward, it engendered a number of complex problems. Soon after the Act was made official, a question arose as to what objectives the Stockpile should meet. The Stock Piling Act had stated that the Stockpile was to be used for national defense purposes. National defense included consideration of the following requirements:

1. Military requirements—materials needed by the military to carry on a war.

2. Domestic nonmilitary requirements—the essential civilian and war-supporting needs of industry and the civilian population that must be met to successfully carry on the war. These requirements assumed substitution and conservation measures would be in effect.

3. Export requirements—the usual peacetime exports to friendly countries.

After vigorous discussion, a "factoring system" was adopted in 1950 to determine the estimated availability of wartime supplies. A comparison with requirements helped determine what materials and what quantities should be stored. The system took into account the possible military accessibility to critical materials, political dependability of potential foreign suppliers, concentration of supplies of needed materials, and other miscellaneous factors, such as vulnerability to sabotage.

Of course, the concept of Stockpile objectives was a long-term issue. More immediate was the question of what to begin purchasing to start meeting those objectives. The major question was whether to work for balance in the Stockpile by purchasing materials that were needed to meet the quantity requirements or by purchasing materials that were readily available.

The original appropriation for the Stockpile was $100 million. With this beginning, the Munitions Board created an ideal procurement schedule. Their plans were subject to self-imposed policies. During the early postwar period, the Munitions Board purchased items conservatively. Any commodity price higher than one and one-half times the prewar price was considered too high. Even when the price was right, the Board did not buy material needed for industrial consumers.

Right after the war, business was active and materials were generally in demand. Prices were high for many items on the Stockpile list. So, as a result of the Board's acquisition policies, procurement was slow during the program's first year. Of the $100 million appropriated, $33 million remained unobligated at the end of the fiscal year.

Closely linked with the early purchasing decisions was the debate over the "Buy American" Act of 1933. The Act stated that only materials or supplies produced in the United States could be acquired for public use unless buying US materials was "inconsistent with the public interest" or the cost was "unreasonable." Under the Act, domestic producers were allowed up to one year for delivery and were normally exempt from the requirement of giving bond to guarantee delivery. Moreover, a 1937 Treasury Department ruling stated that domestic bids were supposed to be accepted if their price was not more than 25% over the nearest foreign bid.

One of the expressed purposes of the Stock Piling Act was to "encourage the conservation and development of sources of these materials within the United States . . ." But President Truman had dealt a blow to the "Buy American" Act on July 23, 1946, when he signed the Stock Piling Act into law. The President stated that he felt the "Buy American" provision of the Stock Piling Act would increase the cost of stockpiling, hamper our foreign economic policy, and deplete domestic reserves of strategic materials.

In the early years of stockpiling, many purchases were made from foreign sources, for two basic reasons. For one thing, two-thirds of the materials to be stockpiled were not produced in the United States. Of the listed materials that were available in this country, many could not be mined and processed economically. Purchasing those materials would have raised costs more than the specified 25%. On the other hand, foreign procurements were important early in the program because the strategy was to purchase some of the foreign-made materials and have them in stock in case an unexpected emergency again cut the United States off from foreign sources.

Naturally, senators and representatives who favored development of domestic mining industries took a dim view of the foreign purchases for the Stockpile. So did the miners, who had originally opposed the stockpiling legislation because they believed it could undercut their business. They, and the Department of Interior, rejected the notion that vigorous domestic mineral production would actually weaken national security by depleting important domestic resources. On the contrary, some congressmen fought for high Stockpile objectives as a means of stimulating domestic mining and regional economies. They felt that stockpiling alone was not enough to ensure readiness, and so they encouraged development of US mining as a supplement.

Two other procurement issues that had to be considered were the quality of material that should be stored and the form in which it should be stored. On the issue of quality, the question was the degree of impurity that was acceptable in stockpiled materials. At first, only materials of the highest purity were accepted for stockpiling on the grounds that in an emergency they could be mixed with available lower-grade materials to produce an acceptable product.

Obviously, this limitation was not palatable to those who advocated the interests of the US mining industry. Interior Secretary Harold Ickes argued that insistence on highest purity would "discourage the utilization of domestic ores, particularly those of a marginal character." In addition, Congress charged that the specifications were deliberately set high to exclude domestic materials from the Stockpile. The Preparedness Subcommittee of the Senate Committee on Armed Services insisted that the adherence to the concept of "universal applicability," that is, stockpiling only the purest materials because they would be suitable for any use, was unrealistic. They suggested stockpiling various qualities for a variety of end uses, thereby enabling more domestic materials to qualify for inclusion in the Stockpile inventory.

Initially, however, the argument for highest purity won, although this standard was later relaxed.

Regarding the issue of form, the question revolved around whether materials should be stored in their raw form or, for example, in upgraded forms or alloys that would be ready for use. On one hand, it was argued that stockpiling raw materials would be less expensive, would ensure a supply of raw materials for whatever was needed, and would prevent waste, if, for instance, some alloys became obsolete. The counter argument proposed by the Department of Interior was that storing already processed materials would save time, labor, and other resources if a war actually occurred. During wartime, having a stock of already processed materials would prevent the necessity of diverting valuable labor and natural or economic resources from the war effort to process raw materials into forms needed for the military effort. Ultimately, however, the decision was made to continue stockpiling primarily raw materials.

2.6.2.3 Defense Production Act

When the Korean War began, Congress recognized the need to expand production of materials quickly to meet the emergency. This necessity led to the Defense Production Act of 1950, an act that gave the President expanded powers for production emergencies. Among those powers were the following:

1. Encouraging banks to lend money and extend credit to speed production and deliveries of defense-related materials.
2. Authorizing broad direct lending to speed government contracts for production of strategic and critical materials.
3. Purchasing metals, minerals, and other raw materials for government use or resale.
4. Encouraging development and mining of critical and strategic minerals and metals, as long as those minerals and metals were not sold below the current US market price.

Naturally, use of such authority benefited the domestic mining industry and eventually created one of the major alternate sources of materials for the Stockpile.

In addition to these powers, the President was also authorized to subsidize domestically produced materials; process, refine, transport, and store such materials; and expand and modify government-owned industrial facilities and upgrade private facilities. A later amendment, approved June 30, 1953, points up the success of these expanded powers. The President was permitted to transfer to the existing National Stockpile those materials that had been accumulated under the Defense Production Act. Indeed, the act had created a "stockpile" of its own.

Besides these existing Stockpiles, legislation enacted by Congress in 1954 resulted in another "Stockpile." To increase foreign consumption of surplus US agricultural products, Congress authorized the sale of these products for foreign currencies that could be used to purchase strategic and critical materials for a "supplemental stockpile." In effect, the United States was able to get rid of bulky, short-lived farm products that were costly to store and receive in return needed strategic materials that were longer-lasting and less expensive to store. Releases from this supplemental Stockpile were restricted, as were releases from the National Stockpile.

This Act also gave rise to the Barter Program. Under the Act, the Commodity Credit Corporation (CCC) was permitted to barter agricultural commodities that it owned for strategic materials, creating, in essence, a fourth major Stockpile. Actually, the Barter Program was not primarily intended for acquiring strategic and critical materials; it was intended to reduce costs and storage of short-lived surplus agricultural commodities by exchanging them for more durable metals and minerals.

Between 1950 and 1954, barter was used when possible to acquire necessary materials for the Korean conflict. Beginning in 1954, however, materials obtained through barter were placed in the CCC inventory and, when Public Law 480 was modified in 1956, transferred to the supplemental stockpile.

2.6.2.4 Accelerating the Stockpile Program

Unquestionably, the Defense Production Act was a major factor in expanding the Stockpile program. Prior to the original stockpiling legislation, Congress, in 1939, had allocated $70 million in appropriations and advance contract authorizations. Between 1939 and 1950, Congress had appropriated $1,535 million (including contract authorizations) for stockpiling, but only $895 million had actually been spent. The rest of the money was tied up in long-term contracts for future deliveries. Within a 14-month period during 1950 and 1951, Congress increased authorizations for the program, totaling nearly $3.5 billion.

By the end of fiscal year 1955, major appropriations for Stockpile procurement had ended, though some Stockpile objectives remained unfulfilled. Within the first five years after its enactment in 1950, the Defense Production Act had become a major source of materials for filling National Stockpile objectives.

Particularly noteworthy in this regard were the expansion programs created under the Defense Production Act. Through various procurement and financing techniques, as well as other incentives, these programs created new industries and expanded existing industries. Some of the programs instituted were

1. *Contracts, loans, and grants.* Agencies of the government entered into over $8 billion in contracts, grants, and loans for defense production. Over 90% of the commitments were made by GSA.
2. *"Put" contracts.* These contracts gave the contractor the option to deliver to the government that part of his production that he could not sell on the open market at or above a stipulated option price.
3. *Reopening old facilities.* Government-owned mining, smelting, and refining facilities originally built during World War II and later deactivated were rehabilitated and put into operation.
4. *Purchase regulation programs.* Basically, these programs were government commitments to purchase from any qualified producer who delivered specified material to a designated receiving point, on a first-come, first-serve basis, until the program limit was fulfilled or the expiration date was reached.

Many suppliers saw these programs as an important commercial outlet. Previously nonexistent domestic industries flourished under them, only to vanish when the programs were curtailed or finally completed by 1962.

2.6.2.5 From Shortage to Surplus

The Stockpile expansion programs that were begun in response to the Korean War created an unprecedented buildup of stockpiled materials. These programs continued long after the war ended in 1953. Within the same time period, the expansion of the nuclear age, with its completely different theories and strategies of warfare, changed the philosophy of stockpiling materials. These two factors laid the foundation for an era of Stockpile surplus.

Certainly, discord over Stockpile policy was not new. As we have seen, the program was plagued since its inception by disagreements over how stockpiling should be conducted. In August 1955, the Assistant Secretary of Defense, Supply and Logistics, sent a letter to the Director, Office of Defense Mobilization (ODM), in which he pointed out that the Joint Chiefs of Staff had recently approved a revised objectives plan based on a three-year war instead of the five-year war. Nothing came of his concern at the time, but the issue resurfaced in October 1957. The new Director of Defense Mobilization appointed a Special Stockpile Advisory Committee to advise him of the adequacy of Stockpile policy. On Jan. 28, 1958, the committee submitted its report to the Director, ODM. Among their most important recommendations were the following:

1. That the current government Stockpiles virtually eliminated the possibility of raw material shortages.

2. That the correct government inventory plus production in accessible areas could support an expanded defense industry for several war years.

3. That a nuclear war would require fewer materials than a limited war.

4. That instead of raw materials, the Stockpile should contain more finished items and supplies for survival, relief, and rehabilitation.

5. That Stockpile planning should be revised to meet shortages for a three-year emergency; provisions should also be made for major military and atomic energy programs.

6. That the Director, Office of Defense Mobilization, should have more authority to dispose of surplus materials.

On June 11, 1958, the Director, Office of Defense Mobilization, ratified most of the advisory committee's findings when he signed Defense Mobilization Order (DMO) V-7. The order became effective on June 30, 1958.

Major policies delineated in the order were

1. Strategic Stockpile objectives should be reduced from a five-year war to a three-year war; Stockpile objectives should be adequate for a limited or general war.

2. Commitments for deliveries beyond the maximum Stockpile objectives should be cancelled when possible.

3. Disposals of excesses should be made only if they do not disrupt US domestic markets or US international relations.

In effect, the order reduced current Stockpile objectives by 40%, thereby creating surpluses in a number of commodities.

In response to DMO V-7, terminations of or reductions in outstanding contracts for defense materials were implemented, and obligations declined progressively year by year. Although these contracts did not generally contain termination clauses for government convenience, they could usually be cancelled by negotiation. And by the close of fiscal year 1962, natural attrition had further reduced outstanding contracts. Commodities included agar (seaweed gelatin), hog bristles, castor oil, coconut oil, cryolite, opium poppy seed, pyrethrum, quinine, titanium, and zirconium ore.

Disposals, on the other hand, did not proceed quite so smoothly. The original V-7 order provided for retention of materials to meet not only current objectives but also anticipated increases in objectives. A revised order issued in 1959 further restricted disposals by adding a requirement for review of any proposed disposal of excess materials by the Departments of Interior, Commerce, State, Agriculture, and Defense, as well as any other concerned Federal agencies. Inevitably the necessary review and approval process greatly slowed disposals. Fear of disrupting foreign or domestic markets, based on the original V-7 order, was often cited as the basis for vetoing proposed disposals.

The total value of disposals under this order was about $550 million, divided about equally between the National Stockpile and the Defense Production Act Stockpile. Disposal items from the National Stockpile totaled approximately $276.5 million and included hog bristles, coconut oil, rubber, and tin. Disposals from the Defense Production Act Stockpile totaled approximately $273.7 million and included aluminum, copper, and nickel.

Perhaps the greatest impetus to such disposals was the shake-up of the Stockpile program that occurred after Jan. 30, 1962, when President Kennedy announced at a news conference his assessment of the Stockpile program. Kennedy stated that, ". . . this excessive storage of costly materials [is] a questionable burden on public funds, and in addition a potential source of excessive and unconscionable profits."

In February 1962, the President created an Executive Stockpile Committee under the chairmanship of the Director, Office of Emergency Planning. Members were drawn from Cabinet-level government agencies, namely the Departments of State, Defense, Interior, Commerce, and Labor, and the General Services Administration. The President's committee recognized the magnitude and importance of the Stockpile disposal problem. In this regard, they made a series of recommendations, including the following:

1. Develop a program to provide the public with appropriate information about Stockpile surpluses.

2. Present maximum Stockpile objectives should be used to determine the surplus.

3. Defer disposals when they might inadvertently upset the economies of foreign countries and thus cut the US off from needed imports.

4. Give priority to cash sales of surpluses to commercial sources.

5. Reuse surpluses for other government functions only when those surpluses could not be disposed of.

6. Pass legislation to speed up disposal of Stockpile surpluses.

The rationale behind some of those recommendations is clear. The first recommendation took into account the disastrous effect of rumors upon the national economy. Rather than let the public speculate about Stockpile surpluses, partly because such information had been classified in the past, the committee chose to inform them. The second recommendation sprang from the committee's belief that the present Stockpile objectives were adequate, despite recurring suggestions to develop new objectives to meet the needs of a nuclear or a conventional war.

The final recommendation sought to correct what the committee saw as defects in existing legislation. The committee felt that the DPA had too many restrictions on Stockpile sales and on trading surplus items to upgrade Stockpile materials. The Strategic and Critical Materials Act, on the other hand, was considered too restrictive about disposals. What the committee preferred was the Agricultural Trade Development and Assistance Act of 1954, which allowed supplemental Stockpile materials to be sold or exchanged with National Stockpile commodities. It also recommended more flexibility in the interchange of materials among the different "Stockpiles" to make disposals easier and improve Stockpile management.

In addition, the National Stockpile and Naval Petroleum Reserves Subcommittee, under the chairmanship of Senator Stuart Symington, initiated a series of hearings on the Stockpile that began on Mar. 28, 1962, and concluded on Jan. 30, 1963. In October 1963, the committee published the reports of the hearings.

Chief among the subcommittee's findings was the conviction that the assumption of a three-year conventional war was unrealistic because it would be useless in fighting a nuclear war. On the other hand, survival and rehabilitation items for such a war should be stockpiled, and many of these items were lacking. The surpluses created by this proposed change in objectives should be disposed of, but maximum caution should be used to avoid disrupting the usual markets and US employment. The committee's final recommendation, however, was the most salient one—that all inventories of strategic and critical materials be consolidated so they could be uniformly acquired, stored, and disposed of.

2.6.2.6 Changing the Face of the Stockpile

The reevaluation of the Stockpile during the early 1960s set the stage for changes in Stockpile policy. In March 1973, the

National Security Council analyzed all aspects of the Stockpile. Their analysis was the basis for new guidance regarding Stockpile objectives for individual materials, issued by President Nixon on Apr. 16, 1973. The guidance assumed that

1. The new Stockpile would be used for defense purposes only; the planning period would be for the first year of an emergency.
2. Theaters of war would exist simultaneously in Europe and Asia.
3. Imports of supplies would be available for all years of an emergency from all noncommunist countries outside the war zone.

Obviously, these assumptions reflected confidence that strategic and critical materials would not be inaccessible in wartime. In his April 16th message on the Stockpile, the President had stated that, ". . . [because the US] economy and technology are dynamic, our capability to find substitutes for scarce materials is far greater today than in the past . . . [The United States is] now able to meet defense requirements for materials during possible major conflicts without imposing an excessive burden on the economy or relying on an enormous stockpile, as was once necessary."

Clearly, the policy trend was toward reduction of the Stockpile, a complete reversal of the acceleration policy promulgated in the early 1950s. The July-December 1973 edition of the Stockpile Report to Congress reported that, "As of December 31, 1973, total quantities of the Stockpile grade materials on hand in all Government-owned inventories with the exception of jewel bearings, were in excess of the Stockpile objectives for all of the materials now in the National Stockpile."

In fact, from July to December 1973, disposal sales of excess strategic and critical materials from all government inventories totaled $734.7 million, the largest sales effort since 1966. Of this total, $646.2 million represented sales from the national and supplemental Stockpiles, $87.2 million came from sales from the Defense Production Act inventory, and other sales accounted for the remaining $1.3 million.

That same year, control of the Stockpile planning policy changed hands. On July 1, 1973, a Reorganization Plan abolished the Office of Emergency Preparedness that ran the Stockpile. Its functions were transferred to the President and later reassigned to other agencies. The General Services Administration was made responsible for emergency resources planning, resources allocation, resource crisis management, and Stockpile materials policy. The Federal Preparedness Agency was created in GSA and made responsible for those functions.

In August 1976, the President issued new Stockpile policy guidance, based on the results of an interagency study of stockpiling policies and procedures. The new policy required a stockpile that could support US defense requirements

1. During a major war over a three-year period.
2. Assuming large-scale industrial mobilization and increased materials demands.
3. Providing for a wide range of basic civilian economic needs to ensure a healthy wartime economy.

The new policy also contained stipulations regarding Stockpile planning, including provisions to

1. Maintain current data and planning factors.
2. Develop an "Annual Materials Plan," a plan that will respond to changes in national security, budgetary considerations, domestic markets, and international events.

The new policy became effective Aug. 23, 1976; new goals were announced on Oct. 1, 1976.

The Stockpile program underwent another change in 1979 when President Carter signed the Strategic and Critical Materials Stock Piling Revision Act on July 30. This Act was the second major revision since the 1946 Act, which rewrote the original 1939 Stock Piling law.

One important feature of the revised Act was that it shifted Stockpile administration again. On July 15, 1979, the Federal Emergency Management Agency (FEMA) was officially created. The Stockpile policy functions previously delegated to GSA were transferred, with the entire Federal Preparedness Agency, to FEMA. Management functions, such as purchases and sales of materials, storage, security, maintenance, rotation, and refinement and processing of materials, remained with GSA.

In addition to changing the manager, the revised Act also accomplished the following:

1. Established the National Defense Stockpile Transaction Fund to receive money from the disposal of excess Stockpile materials. Monies from the fund, if authorized by Congress, were to be used only to buy new materials or transport newly acquired materials.
2. Encouraged the uses of barter to acquire and dispose of Stockpile materials.
3. Reaffirmed the statement that the Stockpile is to be used only for defense purposes.
4. Specified that the Stockpile must be sufficient to sustain the United States for a period of not less than three years in the event of a national emergency.

The revision of 1979 eventually set the wheels of change in motion. On Mar. 13, 1981, President Reagan announced a new major purchase program for the National Defense Stockpile. Reagan stated that, "It is now widely recognized that our nation is vulnerable to sudden shortages in basic raw materials that are necessary to our defense production base."

At the time, the Stockpile contained 61 family groups and individual materials. These materials could be categorized as follows:

1. Twenty-four group and individual materials with inventories equal to or greater than Stockpile goals.
2. Thirty-seven group and individual materials with inventories less than the goals. Of these materials, 14 goals were over 50% filled.

To fill these gaps, the administration set up priorities for purchasing materials to ensure that urgently needed ones with high defense uses and large goals were purchased first. Priority materials included refractory grade bauxite, cobalt, medicinals, nickel, platinum group metals, rubber, tantalum, titanium, and vanadium. Congress appropriated an initial $100 million from the Transaction Fund for fiscal year (FY) 1981 purchases. In fact, purchases of cobalt, bauxite, iridium, tantalum, and quinidine sulfate were made during FY 1981.

Disposals were also an important issue. On Aug. 13, 1981, President Reagan signed the Omnibus Budget Reconciliation Act (Public Law 97-35). The Act authorized disposal of antimony, asbestos (amosite and chrysotile), diamond stones, diamond industrial crushing bort, iodine, mica (muscovite splittings), phlogopite splittings, muscovite film first and second qualities, muscovite block (stained and lower), mercuric oxide, mercury, silver, and vegetable tannin wattle to be sold over a three-year period from 1982 to 1984.

Also included in the Budget Reconciliation Act were several amendments to the Strategic and Critical Materials Stock Piling Act. Most significant was the new Section 11, which required Congress to submit an Annual Materials Plan (AMP) each year with the President's budget. The AMP, developed through an interagency effort, was supposed to include planned expenditures for acquisitions and anticipated receipts from sales. The plan was required to cover the next fiscal year and the assumptions for the succeeding four fiscal years.

The Annual Materials Plan process was conducted within the framework of the AMP Steering Committee, an advisory group to the Director of the Federal Emergency Management Agency. The AMP Steering Committee was chaired by FEMA and included as designated member representatives of the Departments of Agriculture, State, Commerce, Defense, Energy, Interior, and Treasury, and GSA, OMB, and NSC. The AMP was developed in a way that balanced National Defense Stockpile requirements against the need to avoid undue market disruption and to conform to budget limitations.

The AMP process began within the Office of Resources Preparedness of FEMA, which provided a list of goals, shortfalls, excesses, and priorities to GSA. After an evaluation of the market outlook, the Office of Stockpile Transactions of GSA proposed quantities of commodities for acquisition or disposal. After FEMA review, the proposals were sent to the Strategic Implications Subcommittee and the Market and International Political Impacts Subcommittee, which furnished suggested revisions to the full AMP Steering Committee.

The Strategic Implications Subcommittee was chaired by the Department of Defense and included, as other designated members, the CIA, the Department of Energy, and FEMA. The subcommittee's primary function was to determine whether any of the materials proposed for the AMP would be affected by anticipated changes in defense requirements.

The Market and International Political Impacts Subcommittee was cochaired by the Departments of Commerce and State with the other designated members being the Departments of Defense, the Treasury, and the Interior, and FEMA, and GSA. This subcommittee evaluated, case-by-case, the effects that Stockpile acquisitions and disposals would have on domestic and international markets. The subcommittee examined significant trends and areas in which Stockpile activities could lead to undue market impacts and affect the normal market participants, as cited in the Stock Piling Act. The subcommittee also determined the international economic and political impacts of Stockpile acquisitions and disposals. In particular, the subcommittee determined the impact that Stockpile activities could have on earnings of international producers and producer countries, on international trade patterns, and on international agreements.

The results of these analyses were presented to the AMP Steering Committee, which developed a recommended AMP. The AMP was submitted to the Director of FEMA for approval and then to the NSC for review. At the same time, an information copy was provided to the Office of Management and Budget. Any revisions to the initial AMP each year were made jointly by the NSC and FEMA. The AMP was then submitted to the Committees on Armed Services of the Senate and the House of Representatives.

In 1982, two other important policies were created, each one affecting the Stockpile. On Apr. 5, 1982, President Reagan transmitted to Congress a report related to the National Materials and Minerals Policy, Research and Development Act of 1980. The President stated that, "It is the policy of this administration to decrease America's minerals vulnerability by taking positive action that will promote our national security, help ensure a healthy and vigorous economy, create American jobs, and protect America's natural resources and environment."

To put this policy into action, the President endorsed the following actions:

1. Continuing to inventory federal land to determine its mineral availability and potential. The President wanted to open appropriate public lands for mining and to develop deep seabed mineral resources as well.

2. Using tax incentives in the Economic Recovery Act to encourage private research with wide general application to materials problems and increased productivity.

3. Carefully disposing of excess Stockpile items while acquiring needed items. To enhance acquisitions and disposals, the President encouraged the use of exchange and barter.

The other important policy was separately created in November 1982, when President Reagan issued a directive stating that, over a 10-year period, chromite and manganese ores in the Stockpile were to be upgraded into high-carbon ferrochromium and high-carbon ferromanganese. The intent of the directive was to help sustain the processing capability necessary to national defense. The idea was that maintaining this capacity would reduce the time required to convert Stockpile materials to ferroalloys in an emergency.

Careful management of the Stockpile continued to be an important issue in subsequent years. On Nov. 14, 1986, the National Defense Authorization Act for FY 1987 (PL 99-661) was enacted. Among its provisions were the following:

1. No action may be taken before Oct. 1, 1987, to implement or administer any change in a Stockpile goal in effect on Oct. 1, 1984, that results in a reduction in the quality or quantity of any strategic and critical materials to be acquired for the National Defense Stockpile.

2. Uses of the Stockpile Transaction Fund are extended to include transportation of commodities, storage, development of specifications, upgrading expenses, testing and quality studies of Stockpile materials, material and mobilization studies, and other reasonable requirements for management of the Stockpile.

3. A seven-year ferroalloy upgrading program is required to convert Stockpile chromite and manganese ore to 374,000 st (339 kt) of high-carbon ferrochromium and 472,000 st (428 kt) of high-carbon ferromanganese, effectively putting the President's 1982 directive into law.

In addition, the Act required the Secretary of Defense to submit to the Congress a report describing, inter alia, the war emergency situation that should serve as the basis for planning and management of the National Defense Stockpile. This requirement reflected the Senate Committee on Armed Services' concern that a 1985 Stockpile goal study performed by the National Security Council had been inadequate. Evidence provided by the General Accounting Office, the various Administration agencies, and private individuals indicated that the National Security Council had used "an optimistic wartime scenario."

Based on the National Security Council Study, a proposed modernization program of the National Defense Stockpile of Strategic Materials was sent to Congress. The plan called for the inclusion of two types of materials in the Stockpile, those that would be unavailable during a long war and a reserve of materials already in the Stockpile to provide extra protection against materials shortages during a war.

Congressional opposition and evidence presented before the Armed Services Committee caused the proposed plan to be heartily rejected. Instead, the committee members made recommendations of their own, including the following:

1. Amending Section 2 of the Strategic and Critical Materials Stock Piling Act by restating (a) "that the intent of Congress is that the National Defense Stockpile be used to serve the interest of national defense only and not be used for economic or budgetary purposes" and (b) "that the quantities of materials stockpiled under the Act should be sufficient to sustain the United States for a period of not less than three years during a war emergency that would necessitate total mobilization of the economy of the United States for a conventional global war of indefinite duration."

2. Transferring management of the Stockpile to the Secretary of Defense. The Secretary would be responsible for acquisition, storage, security, maintenance, disposal, refining, rotation, processing, and upgrading Stockpile materials; for budgeting for Stockpile operations; and for managing the National Defense Stockpile Transaction Fund.

3. Requiring the Secretary of Defense to submit an annual report of his recommended requirements for the National Defense Stockpile based on total mobilization of the US economy for a three-year conventional global war. The Secretary would also be required to provide the National Emergency Planning Assumptions used to determine the Stockpile requirements.

The proposal to return control of the Stockpile to DOD represented a departure from previous policy since control had been in the hands of civilian agencies, namely FEMA and its predecessors, the Office of Emergency Preparedness and the Office of Civil and Defense Mobilization, since the 1950s. At the time, the House adopted the recommendations as part of H.R. 1748, but the Senate did not concur. So more time was allotted to determine whether the Stockpile could be efficiently managed by FEMA and GSA.

In addition to provisions on stockpiling goals, the Transaction Fund, and ferroalloy upgrading, PL 99-661 actually added certain amendments to the existing Strategic and Critical Materials Stock Piling Act. A particularly important one was the new Section 6A, which required the President, by Feb. 15, 1987, to designate a single federal official to be the National Defense Stockpile Manager and perform the functions of the President under the Stock Piling Act. The designated official ". . . shall be an officer who holds a civilian position to which the person was appointed by the President, by and with the advice and consent of the Senate." The official was to be known as the "National Defense Stockpile Manager." To comply with Section 6A, President Reagan, on Feb. 13, 1987, named Julius W. Becton, Jr., Director of FEMA, National Defense Stockpile Manager.

Becton's appointment was short-lived, however. On Feb. 25, 1988, President Reagan issued Executive Order 12626, designating the Secretary of Defense as National Defense Stockpile Manager. Essentially, control of the Stockpile was being shifted from civilian to military hands, as had been suggested earlier in H.R. 1748. That same year, the National Defense Authorization Act for FY 1988 and 1989 expanded the powers of the Stockpile Manager by adding provisions to the new Section 6A of the Strategic and Critical Materials Stock Piling Act. The new provisions required the President to delegate most presidential functions to the National Defense Stockpile Manager. The Stockpile Manager, however, could not authorize special releases from the Stockpile, initiate research and development of Stockpile commodities, or determine whether certain imported commodities should be prohibited.

2.6.3 THE STOCK PILING ACT

The Stock Piling Act reflects current philosophy about the Stockpiling program. (For a complete text of the Act, see Appendix A.) Highlights of the implementation of the Stock Piling Act of 1979, as amended, are summarized in the following.

2.6.3.1 Materials to be Stockpiled

The National Defense Stockpile Manager, per delegation by the President, makes the ultimate determinations as to which materials are strategic and critical. He also determines the quality and quantity of each such material. Section 3 of the Stock Piling Act provides Congressional guidance as to how these determinations are to be made.

The Stockpile is not an inventory of finished products but is rather comprised of materials necessary to meet military, industrial, and essential civilian needs during a crisis. The form of the material stockpiled should reflect (1) handling and processing capabilities of US industry and (2) defense requirements during a national emergency.

The classification of materials as strategic and critical changes over time as underlying assumptions and new technology dictate the obsolescence of materials such as silk and talc and the emergence of new materials such as germanium. The Stockpile is generally comprised of materials that are raw or refined rather than finished. (For a list of materials currently in the Stockpile, see Appendix B.)

2.6.3.2 Quantities of Materials to be Stockpiled

The Stockpile's Annual Materials Plan (AMP) contains an operations forecast of the Stockpile during the fiscal year and the succeeding four fiscal years. The Stock Piling Act requires the report to include "details of planned expenditures . . . and of anticipated receipts." Estimates are provided for both disposals and acquisitions.

Each year, by February 15, the President should submit a new AMP to the Congress, as required by Section 1 l(b) of the Act.

2.6.3.3 Method of Acquisition and Disposal

Stockpile goals indicate the magnitude of Stockpile requirements for given materials. Section 6(b)(2) provides that efforts shall be made to avoid undue market disruption to the "maximum extent feasible" for both acquisitions and disposals.

Section 6(b) of the Stock Piling Act stipulates that Stockpile acquisitions and disposals must, to the maximum extent feasible, utilize formal advertising or competitive negotiation procedures. The Stockpile's contract managers must draft their solicitations for acquisitions in accordance with the established federal procurement practices and the agency regulations.

2.6.3.4 Barter and Other Stockpile Transactions

Section 6(c) of the Act permits and encourages the use of barter for accomplishing Stockpile transactions. In recent years, excess Stockpile materials that are authorized for disposal have been transferred at fair market value as payment to contractors who are upgrading ferroalloys. Industrial diamonds, iodine, tin, and vegetable tannins, for example, have been used as payment.

Should the President determine that he cannot acquire or dispose of Stockpile materials in accordance with established federal procurement practices, he may elect to waive the applicability of any provision that controls or limits an acquisition or disposal. This waiver provision requires the Stockpile Manager to provide the Armed Services Committees of the Senate and House of Representatives with a written notice at least 30 days before proceeding with the transaction. The notice must include the reasons for not following established procedures.

2.6.3.5 National Defense Stockpile Transaction Fund

The Transaction Fund, or T-Fund, was established as of 1979 in the US Treasury as a separate fund under Section 9 of the Act. Moneys received from the sales of Stockpile materials are deposited into the T-Fund in the Treasury where they remain until expended. By earmarking the money in the Transaction

Fund, Congress helps ensure that funds will be available for restructuring and upgrading the Stockpile.

Moneys in the Transaction Fund are restricted in the Act to a limited number of specific uses. These uses include the acquisition of strategic and critical materials, transportation related to such acquisitions, storage, development of current specifications, the upgrading of existing Stockpile materials to meet current specifications, testing and quality studies of Stockpile materials, and studying future materials and mobilization requirements for the Stockpile.

2.6.3.6 Operational Expenditures for the Stockpile

National Defense Stockpile operating expenses are authorized by Section 5(c) of the Act. This section provides for appropriation of moneys to provide for the transportation, processing, refining, storage, security, maintenance, rotation, and disposal of materials. Now that the Stockpile is part of the Department of Defense, operating expenditures are a portion of the agency's appropriation for Operations and Maintenance.

2.6.3.7 Congressional Oversight of the Stockpile

The Committees on Armed Services of the Senate and House of Representatives oversee the National Defense Stockpile. The President and the Stockpile Manager must notify these committees in writing if the Stockpile Manager does any of the following: (1) releases materials for national defense purposes; (2) waives competitive procedures for acquisition or disposal actions; (3) waives the disposals for domestic use requirement; (4) revises the quantity of any Stockpile material goal by more than 10 percent. The Act specifies additional reporting requirements concerning: (1) information on foreign and domestic purchases of materials; (2) information on acquisition and disposal actions by barter; and (3) the financial status of the National Defense Stockpile Transaction Fund and anticipated acquisition appropriations for the following fiscal year.

Congress must legislate any disposals from the Stockpile. Section 5(b) of the Stock Piling Act requires that all disposals, whether by direct sale or by barter, be authorized to include the commodity's name and quantity. The only exceptions are for rotational sales, for sales of excess materials with a limited shelf life that might deteriorate if the disposal were delayed seeking authorizing legislation, and for releases ordered by the President in time of national emergency or for the national defense.

2.6.3.8 Specialized Leasing Authority

The Act provides a special leasing authority for use in Stockpile operations. Subsection 6(e) permits the President to "acquire leasehold interests in property, for periods not in excess of 20 years, for storage, security, and maintenance of materials in the Stockpile."

2.6.3.9 Special Disposal Authority of the President

In addition to the normal disposals authorized by Congress for excess materials, the President, under Section 7 of the Act, has authority to release materials from the Stockpile for use, sale, or other disposition. The President may order release of materials whenever he determines such release is necessary for the national defense.

This provisional authority has been utilized sparingly in response to situations that have not been national emergencies but are of concern or impact on national security.

The President has ordered releases pursuant to Section 7 for mercury for atomic weapons production when mercury was not available through commercial sources in 1956 and also in 1959. In 1973, quinine was released for sale to pharmaceutical manufacturers with unfilled defense orders so they could meet Vietnam requirements. As recently as 1979, chrysotile asbestos was released to meet specific defense contract deficiencies caused by international supply problems.

2.6.4 FUNCTIONS AND ORGANIZATION OF THE DEFENSE NATIONAL STOCKPILE PROGRAM

Operation of the Defense National Stockpile Program is conducted through the Defense National Stockpile Center (DNSC). Under the policy guidance of the Assistant Secretary of Defense for Production and Logistics, Operations, management is provided by the Directorate of the Defense Logistics Agency. The DNSC is composed of two major separate entities, the Primary Level Field Activity (PLFA) and the Secondary Level Field Activities (SLFA). (See organization charts, Appendix C.)

2.6.4.1 Primary Level Field Activity

The Primary Level Field Activity has its headquarters in Washington, DC, and is staffed by approximately 100 people. The PLFA includes the Office of the Administrator; Office of Counsel; Office of Management and Systems Support; Office of Planning and Market Research; and the Directorates of Stockpile Contracts; Quality Assurance and Technical Services; and Stockpile Operations.

The Administrator, who heads the PLFA, reports to the Director, DLA. The Administrator oversees all functions of the PLFA and the Secondary Level Field Activities. He ensures that all programs are operating effectively to achieve program objectives. He is assisted by the Deputy Administrator, who acts for the Administrator whenever necessary.

The Office of Counsel provides legal advice and services to the Administrator and all staff elements on matters involving or affecting the DNSC. Such services include analyzing the legal impact of proposed and new legislation, helping to formulate acquisition policy and procedures, and providing advice on the legal aspects of establishing internal controls.

The Office of Management and Systems Support keeps accounting and inventory records for the Stockpile Transaction Fund and for the quantity of materials sold and purchased by the contracting officers. The staff provides property accountability; supplies and services; mail, telecommunications and audit-report response; space management; management studies; and personnel services.

This office also maintains and periodically updates an international mailing list of persons or firms interested in selling strategic and critical materials to or buying excess commodities from the National Defense Stockpile inventories located at various depots around the country.

The Office of Planning and Market Research plays a major role in determining the market limitations on buying and selling materials. It evaluates proposed and current purchases and disposals of Stockpile materials and considers whether these actions are feasible and what effect they will have on the worldwide markets. The Office also recommends, evaluates, and monitors potential and actual domestic mining and processing projects approved for funding by Defense Production Act (DPA) appropriations.

The Office has a legislative affairs group that monitors and comments on proposed legislation affecting the Stockpile and reports the Stockpile program's progress semiannually in the Stockpile Report to Congress.

The Directorate of Stockpile Contracts acquires Stockpile materials and contracts for services needed for upgrading existing materials and for repairing and maintaining storage facilities. When authorized, the Directorate also disposes of excess Stockpile materials to producers, consumers, trading firms, or other government agencies.

One important program is the initiative to sustain a domestic ferroalloy furnace and processing capability that is vital to national security. Calendar year contracts provide for the upgrading of the Stockpile's chromite and manganese ore at a combined cost of more than $60 million/year.

This Directorate also manages the William Langer Jewel Bearing Plant at Rolla, ND. This plant is the only facility of its kind in the Western Hemisphere. Its purpose is to maintain a US capability for the manufacture of jewel bearings and related items to ensure a domestic source of supply in a national emergency. The plant is a government-owned facility, operated under contract by the Bulova Corp. Skilled personnel, 60% of whom are of Chippewa Indian descent, shape, polish, and drill synthetic sapphire and ruby raw materials into jewel bearings for Stockpile requirements and for sale to defense contractors. Each year, from 500,000 to l,000,000 jewel bearings, valued at more than $l,000,000, are purchased for the Stockpile.

Two of the metals that the division has purchased recently for the Stockpile are beryllium and germanium. Beryllium, a lightweight, high-strength metal vital in nuclear and aerospace technology, is produced by a domestic firm. Germanium is a hard, glassy, brittle metal closely related to silicon. It is used in lenses for night-vision optics and has been obtained from domestic and foreign firms.

The Directorate of Stockpile Contracts administers legislatively mandated grants to construct facilities for scientific research on strategic materials. It also sells excess silver to the US Treasury to be minted into bullion and commemorative coins. In recent years, the Treasury has used more than $100 million worth of Stockpile silver and lowered inventory holdings by more than 30 million oz (933 Mg).

The Directorate of Quality Assurance and Technical Services ensures the quality of all materials acquired for the Stockpile by sampling and testing these materials for conformance with the purchase specifications and contract requirements. The Directorate also develops sampling, testing, and analyses procedures and acceptance/rejection criteria for use by quality assurance personnel in the secondary level field activities. These procedures are used for materials accepted into the Stockpile by acquisition or upgrade.

The Directorate also conducts semiannual inspections of materials in the Stockpile to ensure their readiness. These inspections are especially important for the more sensitive materials or those materials subject to deterioration, such as quinidine sulfate and natural rubber. Some materials are also selected for special quality assessment tests and analyses. This testing determines the effect of long-term storage on the quality and usability of the commodities.

This Directorate, through its secondary level field activities, acts as the contracting officer's representative for most field operations involving commodities.

The Directorate of Stockpile Operations Management manages the Stockpile storage space, assuring safe storage, security, and maintenance of Stockpile commodities. This Directorate takes care of and secures all storage depots and sites. It develops and administers an equipment preventive maintenance system and maintains a nationwide equipment usage and inventory system. It also develops and implements plans for moving Stockpile materials and for maintaining the quality of commodities in the inventory.

For the secondary level field activities, the Directorate develops storage, handling, and packaging methods and publishes instructions for receiving, safeguarding, and shipping various materials. It also offers technical advice and assistance on storage and handling methods to other government agencies and private industry.

Environmental protection is an important concern. The Directorate of Stockpile Operations Management has overseen the removal of radioactive wastes from several storage sites to the Nuclear Regulatory Commission (NRC)-licensed burial sites. The soil and nearby waterways were thoroughly tested for radioactive thorium and were certified as safe by the NRC.

The Directorate determines the long-term environmental effect of materials stored outdoors. Tests are devised to discover the extent to which certain materials can leach into groundwater and surface water and to find out whether acid rain can cause some substances, such as lead, to leach into the surrounding soil.

Since the Directorate is concerned about the health of depot workers, it maintains high industrial hygiene standards. It also administers the pilot Baseline and Annual Medical Surveillance Program to guard employees' health. The program requires employees to be medically and physically examined before they begin working with the Stockpile and to be reexamined annually to be certain they are continually capable of performing the job safely while wearing any necessary protective clothing or devices.

2.6.4.2 Secondary Level Field Activities

The Secondary Level Field Activities consist of three zones and their respective storage depots. Each zone supports the NDS mission for a specific geographical area within the United States. The Defense National Stockpile Center maintains 94 storage sites around the country, staffed by approximately 200 people.

Each zone has an *Administrator* who guides and supports all Stockpile activities within that zone. The Administrator formulates and controls zone funding allotments and carries out the zone contracting program.

The *Quality Assurance Division* of each zone inspects, samples, and tests Stockpile materials within the zone, following procedures and instructions provided by the PLFA. The *Storage Operations Division* manages the operations of the depot properties and is responsible for commodity shipments, receipts, control, and proper storage.

2.6.5 THE FUTURE OF THE STOCKPILE

What will be the future of the Stockpile? This question is especially pertinent as the NDS enters a new stage as part of the Department of Defense (DOD). Current projections for the Stockpile involve changing priorities to make the Stockpile more responsive to defense needs. Program changes and advancing technology will continue to affect the content of the Stockpile as well as its funding.

2.6.5.1 Modernization

A primary objective is to modernize the Stockpile to make sure it can support both the military and civilian requirements in case of a national emergency. The modernization program is to be conducted in two phases. The first phase will be directed at making the Stockpile useful for accelerating and sustaining military production during a national emergency. This phase will be guided by the recommendations contained in the Secretary of Defense's annual report to the Congress on Stockpile requirements.

The second phase of modernization is expected to begin in FY 1992 and to be carried out over the next five to ten years. This phase will aim at changing the character of the Stockpile to meet the needs at that time. Changes will reflect refinements in military requirements and consideration of civilian requirements. The purpose is to support national security efforts in a future national emergency.

2.6.5.2 Consultations

A second program priority will be to consult more with Congress and private industry to receive expert suggestions on policy and program direction. DOD will work closely with the House Armed Services Subcommittee on Seapower and Strategic and Critical Materials to determine long-term objectives for the NDS program. DOD also plans to consult with industrial experts regarding technical information on special low-volume, highly critical advanced materials.

2.6.5.3 Research and Analysis

DOD has begun a series of research projects to determine future Stockpile requirements and thus appropriate objectives for its modernization program. Among the projects are

1. Analyses of possible conflict scenarios to determine the full range of potential threats to national defense.
2. Analyses to determine essential civilian requirements during a defense emergency.
3. Updating databases containing material consumption ratios, emergency operating capacities, and domestic and foreign production capacities for strategic and critical materials, a task that will be made difficult by budgeting constraints.

2.6.5.4 Proposed Change in Financial Policy

Currently, Section 5(b) of the Stock Piling Act prohibits selling excess Stockpile materials when the balance in the Transaction Fund exceeds $100 million. This cap on disposals was intended to encourage acquiring or upgrading Stockpile materials whenever sales of excess materials brought in additional money. The National Defense Stockpile Manager has proposed that the Section 5(b) cap be rescinded.

Rescinding the Section 5(b) cap would allow the National Defense Stockpile Manager to keep a balance in the Transaction Fund that could be used to build up the Stockpile in case of threats to the national defense. In response to warnings of an impending emergency, the Stockpile Manager could amend the Annual Materials Plan under Section 5(a)(2) of the Stock Piling Act. After notifying the Senate and House Committees on Armed Services, the Stockpile Manager would be allowed to use the existing balance in the T-Fund to acquire or upgrade materials to make up for any shortfalls in the Stockpile.

2.6.5.5 Meeting the Challenges Ahead

Of course, the future goals and policies of the National Defense Stockpile are speculative. Decisions made by the Congress and by DOD will determine its future direction. One thing, however, is certain. The Stockpile cannot remain static but must continually change to meet changing needs.

As we have seen, new trends in technology and in military strategies can render existing policies and stores of materials obsolete. If the Stockpile is to serve as an essential resource to America's defense system, it must adapt accordingly. The major objective of the National Defense Stockpile, then, is not stability but adaptability. With careful foresight, flexibility, and good management, the Stockpile will serve this country well in the coming decades, a symbol of America's strength and readiness.

For reports on the program status and questions regarding commodity/material specific activities involving contracts for acquisition, disposal, upgrading, storage, or testing, the reader should contact the Administrator, Defense National Stockpile Center, Defense Logistics Agency, 1745 Jefferson Davis Highway, Crystal Square 4, Ste. 100, Arlington, VA 22202.

APPENDIX A.
STRATEGIC AND CRITICAL MATERIALS STOCK PILING ACT

(50 U.S.C. 98 *et seq.)*
As amended by the National Defense Authorization Act
for Fiscal Year 1991 (P.L. 101-510)

SEC. 1. This Act may be cited as the "Strategic and Critical Materials Stock Piling Act."

FINDINGS AND PURPOSE

SEC. 2. (a) The Congress finds that the natural resources of the United States in certain strategic and critical materials are deficient or insufficiently developed to supply the military, industrial, and essential civilian needs of the United States for national defense.

(b) It is the purpose of this Act to provide for the acquisition and retention of stocks of certain strategic and critical materials and to encourage the conservation and development of sources of such materials within the United States and thereby to decrease and to preclude, when possible, a dangerous and costly dependence by the United States upon foreign sources for supplies of such materials in times of national emergency.

(c) In providing for the National Defense Stockpile under this Act, Congress establishes the following principles:

(1) The purpose of the National Defense Stockpile is to serve the interest of the national defense only. The National Defense Stockpile is not to be used for economic or budgetary purposes.

(2) The quantities of materials to be stockpiled under this Act should be sufficient to sustain the United States for a period of not less than three years during a national emergency situation that would necessitate total mobilization of the economy of the United States for a sustained conventional global war of indefinite duration.

MATERIALS TO BE ACQUIRED: PRESIDENTIAL AUTHORITY AND GUIDELINES

SEC. 3. (a) Subject to subsection (c), the President shall determine from time to time (1) which materials are strategic and critical materials for the purposes of this Act, and (2) the quality and quantity of each such material to be acquired for the purposes of this Act and the form in which each such material shall be acquired and stored. Such materials when acquired, together with the other materials described in section 4 of this Act, shall constitute and be collectively known as the National Defense Stockpile (hereinafter in this Act referred to as the "stockpile").

(b) The President shall make the determinations required to be made under subsection (a) on the basis of the principles stated in section 2(c).

(c) (1) The quantity of any material to be stockpiled under this Act, as in effect on Sept. 30, 1987, may be changed only as provided in this subsection or as otherwise provided by law enacted after December 4, 1987.

(2) If the President proposes to change the quantity of any material to be stockpiled under this Act, the President shall include a full explana-

tion and justification for the change in the next annual material plan submitted to Congress under section 11(b).

(3) If the proposed change in the case of any material would result in a new requirement for the quantity of such material different from the requirement for that material in effect on Sept. 30, 1987, by less than l0%, the change may be made by the President effective on or after the first day of the first fiscal year beginning after the explanation and justification for the proposed change is submitted pursuant to paragraph (2).

(4) In the case of a proposed change not covered by paragraph (3), the proposed change may be made only to the extent expressly authorized by law.

(5) If in any year the reports required by sections 11(b) and 14 are not submitted to Congress as required by law (including the time for such submission), then during the next fiscal year no change under paragraph (3) may be made in the quantity of any material to be stockpiled under this Act.

MATERIALS CONSTITUTING THE NATIONAL DEFENSE STOCKPILE

SEC. 4. (a) The stockpile consists of the following materials:

(1) Materials acquired under this Act and contained in the national stockpile on July 29, 1979.

(2) Materials acquired under this Act after July 29, 1979.

(3) Materials in the supplemental stockpile established by section 104(b) of the Agricultural Trade Development and Assistance Act of 1954 (as in effect from Sept. 21, 1959, through Dec. 31, 1966) on July 29, 1979.

(4) Materials acquired by the United States under the provisions of section 303 of the Defense Production Act of 1950 (50 U.S.C. App. 2093) and transferred to the stockpile by the President pursuant to subsection (f) of such section.

(5) Materials transferred to the United States under section 663 of the Foreign Assistance Act of 1961 (22 U.S.C. 2423) that have been determined to be strategic and critical materials for the purposes of this Act and that are allocated by the President under subsection (b) of such section for stockpiling in the stockpile.

(6) Materials acquired by the Commodity Credit Corporation and transferred to the stockpile under section 4(h) of the Commodity Credit Corporation Charter Act (15 U.S.C. 714b(h)).

(7) Materials acquired by the Commodity Credit Corporation under paragraph (2) of the section 103(a) of the Act entitled “An Act to provide for greater stability in agriculture; to augment the marketing and disposal of agricultural products and for other purposes” approved Aug. 28, 1954 (7 U.S.C. 1743(a)), and transferred to the stockpile under the third sentence of such section.

(8) Materials transferred to the stockpile by the President under paragraph (4) of section 103(a) of such Act of August 28, 1954.

(9) Materials transferred to the stockpile under subsection (b).

(b) Notwithstanding any other provision of law, any material that (1) is under the control of any department or agency of the United States, (2) is determined by the head of such department or agency to be excess to its needs and responsibilities, and (3) is required for the stockpile shall be transferred to the stockpile. Any such transfer shall be made without reimbursement to such department or agency, but all costs required to effect such transfer shall be paid or reimbursed from funds appropriated to carry out this Act.

AUTHORITY FOR STOCKPILE OPERATIONS

SEC. 5. (a)(l) Except for acquisitions made under the authority of paragraph (3) or (4) of section 6(a), no funds may be obligated or appropriated for acquisition of any material under this Act unless funds for such acquisition have been authorized by law. Funds appropriated for such acquisition (and for transportation and other incidental expenses related to such acquisition) shall remain available until expended, unless otherwise provided in appropriation Acts.

(2) If for any fiscal year the President proposes certain stockpile transactions in the annual materials plan submitted to Congress for that year under section 11(b) and after that plan is submitted the President proposes (or Congress requires) a significant change in any such transaction, or a significant transaction not included in such plan, no amount may be obligated or expended for such transaction during such year until the President has submitted a full statement of the proposed transaction to the appropriate committees of Congress and a period of 30 days has passed from the date of the receipt of such statement by such committees. In computing any 30-day period for the purpose of the preceding sentence, there shall be excluded any day on which either House of Congress is not in session because of an adjournment of more than three days to a day certain.

(b) Except for disposals made under the authority of paragraph (3), (4) or (5) of section 6(a) or under section 7(a), no disposal may be made from the stockpile (1) unless such disposal, including the quantity of the material to be disposed of, has been specifically authorized by law, or (2) if the disposal would result in there being an unobligated balance in the National Defense Stockpile Transaction Fund in excess of $100,000,000.

(c) There is authorized to be appropriated such sums as may be necessary to provide for the transportation, processing, refining, storage, security, rotation, and disposal of materials contained in or acquired for the stockpile. Funds appropriated for such purposes shall remain available to carry out the purposes for which appropriated for a period of two fiscal years, if so provided in appropriations Acts.

STOCKPILE MANAGEMENT

SEC. 6. (a) The President shall-

(1) acquire the materials determined under section 3(a) to be strategic and critical materials;

(2) provide for the proper storage, security, and maintenance of materials in the stockpile;

(3) provide for the upgrading, refining, or processing of any material in the stockpile (notwithstanding any intermediate stockpile quantity established for such material) when necessary to convert such material into a form more suitable for storage, subsequent disposition, and immediate use in a national emergency;

(4) provide for the rotation of any material in the stockpile when necessary to prevent deterioration of such material by replacement of such material with an equivalent quantity of substantially the same material;

(5) subject to the notification required by subsection (d)(2), provide for the timely disposal of materials in the stockpile that (A) are excess to stockpile requirements, and (B) may cause a loss to the Government if allowed to deteriorate; and

(6) subject to the provisions of section 5(b), dispose of materials in the stockpile the disposal of which is specifically authorized by law.

(b) Except as provided in subsections (c) and (d), acquisition of strategic and critical materials under this Act shall be made in accordance with established Federal procurement practices, and, except as provided in subsections (c) and (d) and in section 7(a), disposal of materials from the stockpile shall be made by formal advertising or competitive negotiation procedures. To the maximum extent feasible—

(1) competitive procedures shall be used in the acquisition and disposal of such materials; and

(2) efforts shall be made in the acquisition and disposal of such materials to avoid undue disruption of the usual markets or producers, processors, and consumers of such materials and to protect the United States against avoidable loss; and

(c)(l) The President shall encourage the use of barter in the acquisition under subsection (a)(l) of strategic and critical materials for, and the disposal under subsection (a)(5) or (a)(6) of materials from, the stockpile when acquisition or disposal by barter is authorized by law and is practical and in the best interest of the United States.

(2) Materials in the stockpile (the disposition of which is authorized by paragraph (3) to finance the upgrading, refining, or processing of a material in the stockpile, or is otherwise authorized by law); shall be available for transfer at fair market value as payment for expenses (including transportation and other incidental expenses) of acquisition of materials, or of upgrading, refining, processing, or rotating materials, under this Act.

(3) Notwithstanding section 3(c) or any other provision of law, whenever the President provides under subsection (a)(3) for the upgrading,

refining, or processing of a material in the stockpile to convert that material into a form more suitable for storage, subsequent disposition, and immediate use in a national emergency, the President may barter a portion of the same material (or any other material in the stockpile that is authorized for disposal) to finance that upgrading, refining, or processing.

(4) To the extent otherwise authorized by law, property owned by the United States may be bartered for materials needed for the stockpile.

(d)(1) The President may waive the applicability of any provision of the first sentence of subsection (b) to any acquisition of material for, or disposal of material from, the stockpile. Whenever the President waives any such provision with respect to any such acquisition or disposal, or whenever the President determines that the application of paragraph (1) or (2) of such subsection to a particular acquisition or disposal is not feasible, the President shall notify the Committees on Armed Services of the Senate and House of Representatives in writing of the proposed acquisition or disposal at least thirty days before any obligation of the United States is incurred in connection with such acquisition or disposal and shall include in such notification the reasons for not complying with any provision of such subsection.

(2) Materials in the stockpile may be disposed of under subsection (a)(5) only if the Committees on Armed Services of the Senate and House of Representatives are notified in writing of the proposed disposal at least thirty days before any obligation of the United States is incurred in connection with such disposal.

(3) The President may acquire leasehold interests in property, for periods not in excess of 20 years, for storage, security, and maintenance of materials in the stockpile.

SPECIAL DISPOSAL AUTHORITY OF THE PRESIDENT

SEC. 7. (a) Materials in the stockpile may be released for use, sale, or other disposition-

(1) on the order of the President, at any time the President determines the release of such materials is required for purposes of the national defense; and

(2) in time of war declared by the Congress or during a national emergency, on the order of any officer or employee of the United States designated by the President to have authority to issue disposal orders under this subsection, if such officer or employee determines that the release of such materials is required for purposes of the national defense.

(b) Any order issued under subsection (a) shall be promptly reported by the President, or by the Officer or employee issuing such order, in writing, to the Committees on Armed Services of the Senate and House of Representatives.

MATERIALS DEVELOPMENT AND RESEARCH

SEC. 8. (a)(l) The President shall make scientific, technologic, and economic investigations concerning the development, mining, preparation, treatment, and utilization of ores and other mineral substances that (A) are found in the United States, or in its territories or possessions, (B) are essential to the national defense, industrial, and essential civilian needs of the United States, and (C) are found in known domestic sources in inadequate quantities or grades.

(2) Such investigations shall be carried out in order to-

(A) determine and develop new domestic sources of supply of such ores and mineral substances;

(B) devise new methods for the treatment and utilization of lower grade reserves of such ores and mineral substances; and

(C) develop substitutes for such essential ores and mineral products.

(3) Investigations under paragraph (1) may be carried out on public lands and, with the consent of the owner, on privately owned lands for the purpose of exploring and determining the extent and quality of deposits of such minerals, the most suitable methods of mining and beneficiating such minerals, and the cost at which the minerals or metals may be produced.

(b) The President shall make scientific, technologic, and economic investigations of the feasibility of developing domestic sources of supplies of any agricultural material or for using agricultural commodities for the manufacture of any material determined pursuant to section 3(a) of this Act to be a strategic and critical material or substitutes therefor.

(c) The President shall make scientific, technologic, and economic investigations concerning the feasibility of—

(1) developing domestic sources of supply of materials (other than materials referred to in subsections (a) and (b) determined pursuant to section 3(a) to be strategic and critical materials; and

(2) developing or using alternative methods for the refining or processing of a material in the stockpile so as to convert such material into a form more suitable for use during an emergency or for storage.

(d) The President shall encourage the conservation of domestic sources of any material determined pursuant to section 3(a) to be a strategic and critical material by making grants or awarding contracts for research regarding the development of—

(1) substitutes for such materials; or

(2) more efficient methods of production or use of such material.

NATIONAL DEFENSE STOCKPILE TRANSACTION FUND

SEC. 9. (a) There is established in the Treasury of the United States a separate fund to be known as the National Defense Stockpile Transaction Fund (hereinafter in this section referred to as the "fund").

(b)(1) All moneys received from the sale of materials in the stockpile under paragraphs (5) and (6) of section 6(a) shall be covered into the fund.

(2) Subject to section 5(a)(l), moneys covered into the fund under paragraph (1) are hereby made available (subject to such limitations as may be provided in appropriations Acts) for the following purposes:

(A) The acquisition of strategic and critical materials under section 6(a)(1).

(B) Transportation, storage, and other incidental expenses related to such acquisition.

(C) Development of current specifications of stockpile materials and the upgrading of existing stockpile materials to meet current specifications (including transportation, when economical, related to such upgrading).

(D) Testing and quality studies of stockpile materials.

(E) Studying future material and mobilization requirements for the stockpile.

(F) Activities authorized under section 15.

(3) Moneys in the fund shall remain available until expended.

(c) All moneys received from the sale of materials being rotated under the provisions of section 6(a)(4) or disposed of under section 7(a) shall be covered into the fund and shall be available only for the acquisition of replacement materials.

(d) If, during a fiscal year, the National Defense Stockpile Manager barters materials in the stockpile for the purpose of acquiring, upgrading, refining, or processing other materials (or for services directly related to that purpose), the contract value of the materials so bartered shall—

(1) be applied toward the total value of materials that are authorized to be disposed of from the stockpile during that fiscal year;

(2) be treated as an acquisition for purposes of satisfying any requirement imposed on the National Defense Stockpile Manager to enter into obligations during that fiscal year under subsection (b)(2); and

(3) not increase or decrease the balance in the fund.

ADVISORY COMMITTEES

SEC. 10. (a) The President may appoint advisory committees composed of individuals with expertise relating to materials in the stockpile or with expertise in stockpile management to advise the President with respect to the acquisition, transportation, processing, refining, storage, security, maintenance, rotation, and disposal of such materials under this Act.

(b) Each member of an advisory committee established under subsection (a) while serving on the business of the advisory committee away from such member's home or regular place of business shall be allowed travel expenses, including per diem in lieu of subsistence, as authorized by section 5703 of title 5, United States Code, for persons intermittently employed in the Government service.

REPORTS TO CONGRESS

SEC. 11. (a) The President shall submit to the Congress every six months a written report detailing operations under this Act. Each such report shall include—

(1) information with respect to foreign and domestic purchases of materials during the preceding 6-month period;

(2) information with respect to the acquisition and disposal of materials under this Act by barter, as provided for in section 6(c) of this Act, during such period;

(3) information with respect to the activities by the Stockpile Manager to encourage the conservation, substitution, and development of strategic and critical materials within the United States;

(4) information with respect to the research and development activities conducted under sections 2 and 8;

(5) a statement and explanation of the financial status of the National Defense Stockpile Transaction Fund and the anticipated appropriations to be made to the fund, and obligations to be made from the fund, during the next fiscal year; and

(6) such other pertinent information on the administration of this Act as will enable the Congress to evaluate the effectiveness of the program provided for under this Act and to determine the need for additional legislation.

(b) Not later than February 15 of each year, the President shall submit to the appropriate committees of the Congress a report containing an annual materials plan for the operation of the stockpile during the next fiscal year and the succeeding four fiscal years. Each such report shall include details of all planned expenditures from the National Defense Stockpile Transaction Fund during such period (including expenditures to be made from appropriations from the general fund of the Treasury) and of anticipated receipts from proposed disposals of stockpile materials during such period. Any proposed expenditure or disposal detailed in the annual materials plan for any such fiscal year, and any expenditure or disposal proposed in connection with any transaction submitted for such fiscal year to the appropriate committees of Congress pursuant to section 5(a)(2), that is not obligated or executed in that fiscal year may not be obligated or executed until such proposed expenditure or disposal is resubmitted in a subsequent annual materials plan or is resubmitted to the appropriate committees of Congress in accordance with section 5(a)(2), as appropriate.

DEFINITIONS

SEC. 12. For the purposes of this Act:

(a) The term “strategic and critical materials” means materials that (1) would be needed to supply the military, industrial, and essential civilian needs of the United States during a national emergency, and (2) are not found or produced in the United States in sufficient quantities to meet such need.

(b) the term “national emergency” means a general declaration of emergency with respect to the national defense made by the President or by the Congress.

IMPORTATION OF STRATEGIC AND CRITICAL MATERIALS

SEC. 13. The President may not prohibit or regulate the importation into the United States of any material determined to be strategic and critical pursuant to the provisions of this Act, if such material is the product of any foreign country or area not listed as a Communist-dominated country or area in general headnote 3(d) of the Harmonized Tariff Schedule of the United States (19 U.S.C. 1202), for so long as the importation into the United States of material of that kind which is the product of such Communist-dominated countries or areas is not prohibited by any provision of law.

ANNUAL REPORT ON STOCKPILE REQUIREMENTS

SEC. 14. (a) The Secretary of Defense shall submit to Congress an annual report on stockpile requirements. Each such report shall be submitted with the annual report submitted under section 11(b) and shall include—

(1) the Secretary’s recommendations with respect to stockpile requirements; and

(2) the matters required under subsection (b).

(b) Each report under this section shall set forth the national emergency planning assumptions used in determining the stockpile requirements recommended by the Secretary, based upon total mobilization of the economy of the United States for a sustained conventional global war for a period of not less than three years. Assumptions to be set forth include assumptions relating to each of the following:

(1) Length and intensity of the assumed emergency.

(2) The military force structure to be mobilized.

(3) Losses from enemy action.

(4) Military, industrial, and essential civilian requirements to support the national emergency.

(5) Budget authority necessary to meet the requirements of total mobilization for the military, industrial, and essential civilian sectors.

(6) The availability of supplies of strategic and critical materials from foreign sources, taking into consideration possible shipping losses.

(7) Domestic production of strategic and critical materials.

(8) Civilian austerity measures.

(c) The President shall submit with each report under this section a statement of the plans of the President for meeting the recommendations of the Secretary set forth in the report.

DEVELOPMENT OF DOMESTIC RESOURCES

SEC. 15. (a) Subject to subsection (c) and to the extent the President determines such action is required for the national defense, the President shall encourage the development of domestic sources for materials determined pursuant to section 3(a) to be strategic and critical materials-

(1) by purchasing, or making a commitment to purchase, strategic and critical materials of domestic origin when such materials are needed for the stockpile; and

(2) by contracting with domestic facilities, or making a commitment to contract with domestic facilities, for the processing or refining of strategic and critical materials in the stockpile when processing or refining is necessary to convert such materials into a form more suitable for storage and subsequent disposition.

(b) A contract or commitment made under subsection (a) may not exceed five years from the date of the contract or commitment. Such purchases and commitments to purchase may be made for such quantities and on such terms and conditions, including advance payments, as the President considers to be necessary.

(c)(1) Descriptions of proposed transactions under subsection (a) shall be included in the appropriate annual materials plan submitted to Congress under section 1l(b). Changes to any such transaction, or the addition of a transaction not included in such plan, shall be made in the manner provided by section 5(a)(2).

(2) The authority of the President to enter into obligations under this section is effective for any fiscal year only to the extent that funds in the National Defense Stockpile Transaction Fund are adequate to meet such obligations. Payments required to be as a result of obligations incurred under this section shall be made from amounts in the fund.

(d) The authority of the President under subsection (a) includes the authority to pay—

(1) the expenses of transporting materials, and

(2) other incidental expenses related to carrying out such subsection.

(e) The President shall include in the reports required under section 11(a) information with respect to activities conducted under this section.

NATIONAL DEFENSE STOCKPILE MANAGER

SEC. 16. (a) The President shall designate a single Federal office to have responsibility for performing the functions of the President under this Act, other than under sections 7 and 13. The office designated shall be one to which appointment is made by the President, by and with the advice and consent of the Senate.

(b) The individual holding the office designated by the President under subsection (a) shall be known for purposes of functions under this Act as the “National Defense Stockpile Manager.”

(c) The President may delegate functions of the President under this Act (other than under sections 7 and 13) only to the National Defense Stockpile Manager. Any such delegation made by the President shall remain in effect until specifically revoked by law or Executive Order. The President may not delegate functions of the President under sections 7 and 13.

(d) During any period during which there is no officer appointed by the President, by and with the advice and consent of the Senate, serving in the position designated by the President under subsection (a) or during which the authority of the President under this Act (other than under sections 7 and 13) has not been delegated to that position, no action may be taken under section 6(a)(b).

APPENDIX B

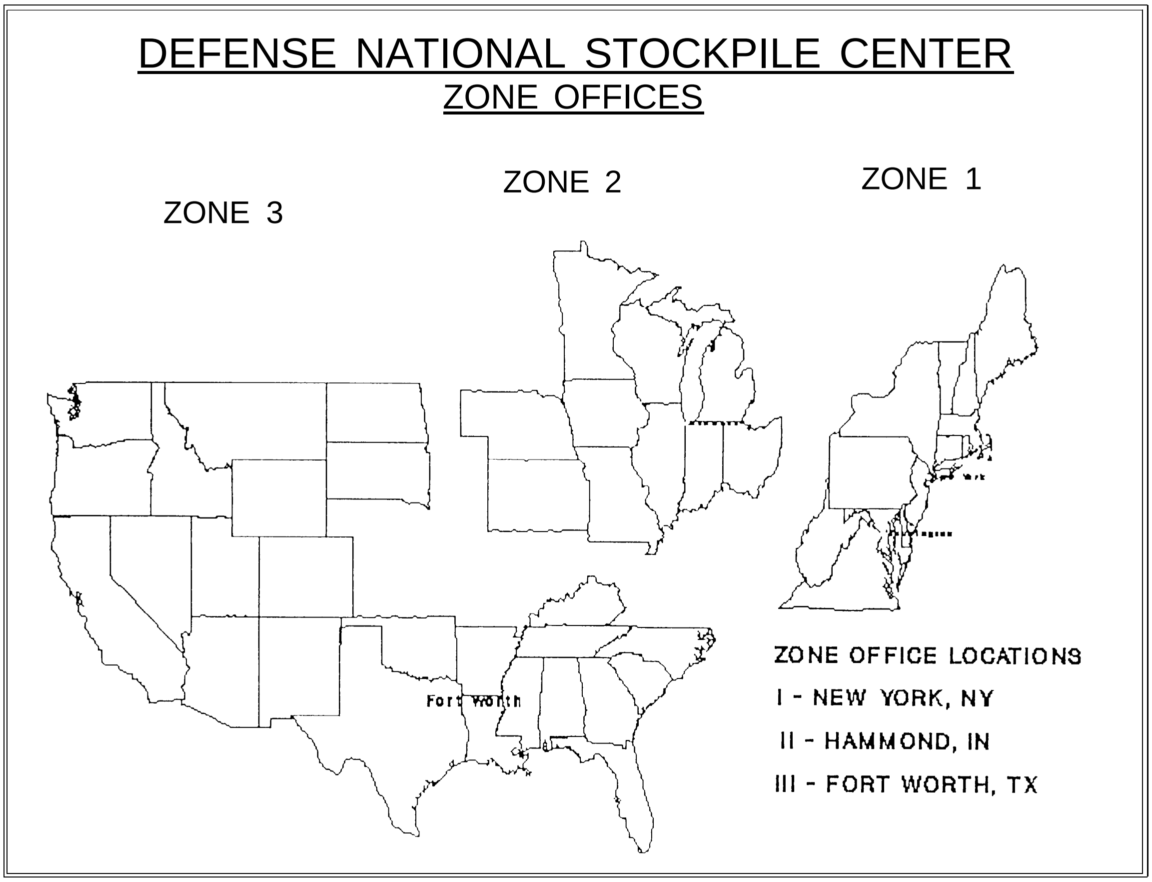
Materials with Stockpile Requirements
As of September 1989

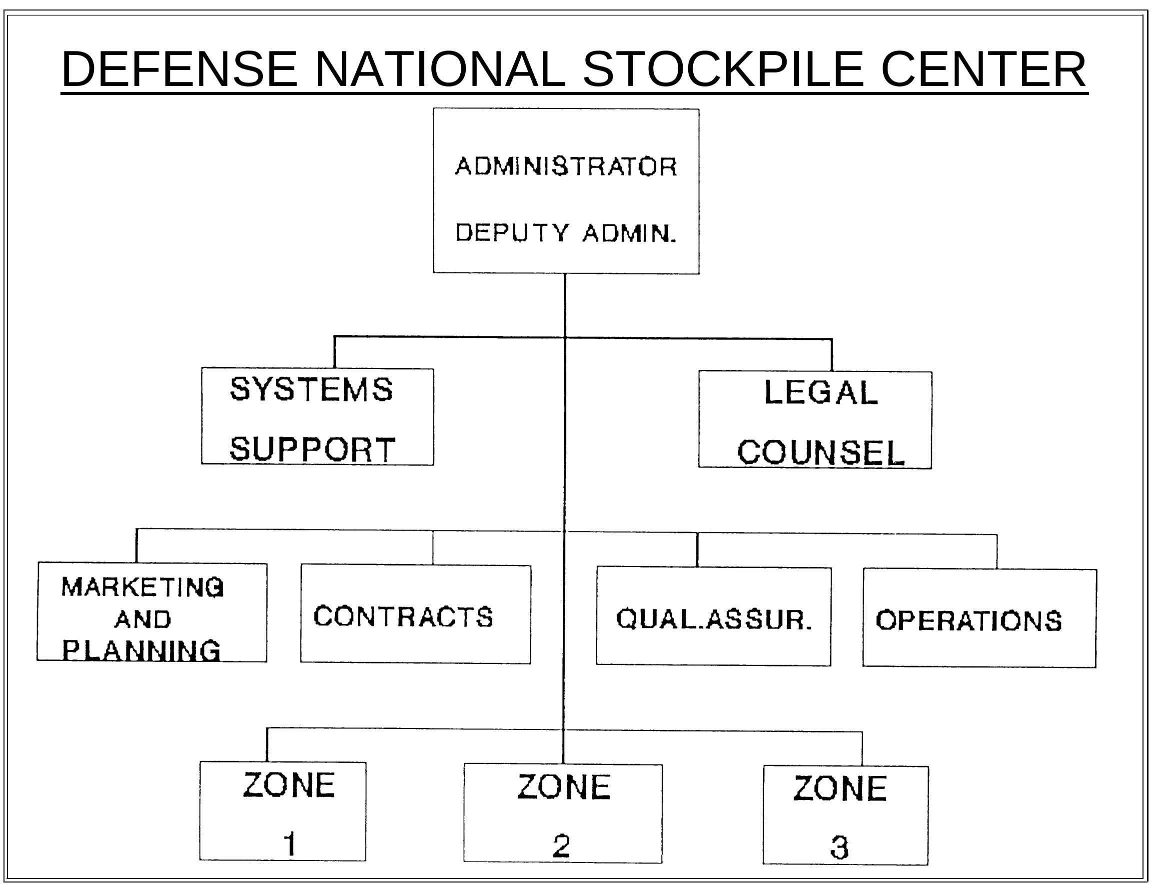
Aluminum Metal Group
Aluminum Oxide
Antimony
Asbestos, Chrysotile
Bauxite, Refractory
Beryllium Group
Bismuth
Cadmium
Chromite, Refractory
Chromium Group
Cobalt
Columbium Group
Copper
Cordage Fibers
Diamond Group
Fluorspar, Acid Grade
Fluorspar, Metal Grade
Germanium Metal
Graphite Group
Indium
Iodine
Jewel Bearings
Lead
Manganese Dioxide
Manganese Group
Mercury
Mica Group
Morphine
Natural Insulation Fibers
Nickel
Platinum Group Metals
-Iridium
-Palladium
-Platinum
-Rhodium
-Ruthenium
Pyrethrum
Quartz Crystals
Quinidine
Quinine
Rayon Aerospace Fiber
Ricinoleic/Sebacic Acid
Rubber
Rutile
Sapphire and Ruby
Silicon Carbide
Silver
Talc Block and Lump
Tantalum Group
Thorium Nitrate
Tin
Titanium
Tungsten Group
Vanadium
Vegetable Tannins Group
Zinc

Other Non-Stockpile Materials in Inventory

Asbestos Crocidolite
Celestite
Kyanite
Mica (stained and lower)
Rare Earth
Talc (ground)

APPENDIX C
Organization Charts





Chapter 2.7
RAW MATERIAL TRADE: MINE TO MARKET

WILLIAM G. HOLROYD

2.7.1 INTRODUCTION

The mining and production of industrial raw materials—be it as a mineral, an ore, or a fuel—is only the beginning of the process to deliver the product to the marketplace. While a sale represents an agreement between the seller and buyer, there are many steps leading to the finalization of the sale which, if performed well, result in a satisfactory conclusion for both parties.

A producer anxious to increase market share must sooner or later address the export market where different terms and conditions apply. Although the responsibility for sales may fall under the responsibility of an experienced member of the company's marketing and sales department, or to a mining engineer who has recently moved from the field to management, those responsible for the marketing and sales development of raw materials should be aware of the rules of trade for domestic and international markets.

The following is a check list that will serve to identify some of the important items relative to the aspects of raw material trade: what should be understood under the terms of the sale, what must be identified, how delivery can be effected, and where further information is available. For discussions of other topics related to trading, see other chapters of Section 2, especially 2.2 and 2.3, and also Chapter 25.2.

2.7.2 SOURCING

Anytime a new industrial raw material is to be brought into a market, a feasibility study will define what products are presently being sourced for that market. Given the trend towards basing a finished product on multiple-sourced raw materials, producers will find eager consumers awaiting news of newly mined products and happy to provide details of how the product is expected to be produced as well as delivery expectations.

Producers should not be surprised to find their products competitive on another continent, even though they may have difficulty being competitive within their own. Geographical locations with proximity to water will greatly assist this situation.

2.7.3 TRANSPORTATION — DOMESTIC US

The ability of any industrial raw material to compete in a marketplace is often a function of inexpensive transportation. This reflects on the ability of the producer to take advantage of various modes of transport to offset any geographical disadvantages, in order to reach their domestic markets.

The geographical statistics of the United States are impressive: slightly over 3.5 million mi^2 (9.0 M km^2) is packed into an area that runs almost 3000 mi (4800 km) from the farthest coastal points east to west. Fortunately for raw material suppliers and consumers alike, the country is split down the middle by the third largest river system in the world: the confluence of the Mississippi, Missouri, and Ohio Rivers.

This natural water roadway acts as a massive feeding and distribution system to/from New Orleans, one of the busiest ports in the United States, and provides access to a total of 14,500 mi (23,300 km) of navigable waterways, which includes an inland waterway system that links Cincinnati and Pittsburgh in the northeast, Minneapolis/St. Paul and Chicago in the north-central region, and Tulsa to the west, as well as the Intercoastal Waterway System between Brownsville, TX, and St. Marks, FL.

Transportation of raw materials within the United States is accomplished through the selection or combination of truck, rail, or water transport. In discussing bulk raw materials, it is generally accepted that highway truck transport, for distances over 1000 mi (800 km), will not compete with rail.

Truck transport is traditionally used to ship construction type materials whose sources are usually close to the population areas they supply. Truck transport offers the advantages of flexibility, not only of prompt loading but also direct delivery and discharge to whatever area the buyer desires.

The statistics on the rail transportation network are as impressive as the geography of the United States: 18 major class 1 rail companies (those having revenues of over $50 million/yr) plus slightly over 300 medium-sized and short rail-line companies provide almost 233,000 mi (375,000 km) of rail mile arteries. Some 140,000 of these miles (225,000 km) are owned by the 18 major rail companies.

In 1980, the Staggers Rail Act allowed rail carriers pricing flexibility on railroad transportation, and, more importantly, the rail carrier did not have to allow access to other railroads to use their rail lines. The outcome of this was a reduction in rates for large point-to-point volume bulk commodities and an increase in rates for lesser volume traffic.

The newer and smaller mineral producer was then unjustly compensating for the lower-revenue-higher-volume bulk commodities. Upon appeal, the Interstate Commerce Commission (ICC) can grant trackage rights to one carrier over another carrier's lines if it is determined that shippers on that line are in an adverse position. This allows the raw material producer more freedom to negotiate with two or more carriers.

Deregulation of the trucking industry, which took place in 1982, did much to bring rail rates down to compete with trucks in the 500- to l000-mi (800- to 1600-km) radius. Although the railroads have a trailer-on-flat-car (TOFC) rate that competes very favorably with truck rates below 1000 mi (1600 km), these rates are usually only competitive for packaged goods.

Barge transport is an extremely economical way of moving bulk commodities. For instance, the cost in 1990 to deliver a full barge containing approximately 1350 lt (1370 t), loaded in New Orleans to a 9-ft (2.7-m) draft, would cost approximately $7.50/lt ($7.38/t) to the Pittsburgh area and approximately $5.50/lt ($5.41/t) to the Chicago area.

While rail rates are generally fixed for a tonnage volume or for a period of time, barge rates are susceptible to market conditions: for example, the US government may decide to store grain in barges and pay three times the generally accepted storage rate. This has the combined effect of not only removing barges from the waterway network but also immediately increasing barge rates. Alternatively, rather than return empty, barges may accept a back haul at reduced rates.

In recent years, there has been a trend toward rail lines attempting to buy out barge lines in order to act upon the concept of offering a "full transportation service." This has come about also because of the Staggers Rail Act, which for the first time allowed companies to own more than one transportation mode.

The following table, reflecting costs in effect for different transportation methods during the year 1990, clearly illustrates how the use of marine transport, sometimes combined with that of rail, will provide a producer with greater access to a market area.

Transportation Method	Transportation: Unit Cost, ¢/ton-mile (¢/t-km)	Total Cost, $/ton ($/t)	Effective Market Area mi (km)
Truck	12 (8.2)	$7.00 (7.72)	60 (97)
Rail	5 (3.4)	$7.00 (7.72)	140 (225)
Marine (barge)	1 (0.7)	$7.00 (7.72)	1000 (1609)
Marine (ship)	< 1 (< 0.7)	$7.00 (7.72)	6000 (9656)

2.7.4 TRANSPORTATION-EXPORT

Any producer desirous of increasing sales through wider market penetration will need to approach the export markets for their products. This will involve a different type of transportation system than that utilized for domestic shipments: the use of ocean vessels to deliver the goods.

The rental of a vessel, or a portion of the vessel space, is known as a charter party, which can take several forms:

1. "Voyage charter," covering one cargo at a time moved between one or more ports.
2. "Time charter," covering the hire or use of a vessel for specific periods of time.
3. "Contract of affreightment," covering the hire of the vessel for more than one cargo, usually with the cargoes being carried between the same ports or defined geographic areas.

Fortunately, decisions related to when and how to charter are usually handled by an ocean freight "broker," who acts either on behalf of the charterer or the vessel owner. Qualified ocean freight brokers are registered with the Association of Ship Brokers.

2.7.5 STATISTICAL PROCESS CONTROL

SPC (statistical process control)—which became the "buzzword" in the 1980s—will be an inherent part of every production system in the 1990s and beyond. An American professor, W. Edwards Deming, exported the use of statistics in production to Japan in 1950. Since that time, the use of statistical controls has enhanced the consistency and uniformity of many of the Japanese products that we have come to know so well.

Deming refers to SPC as "quality control at the source," where each worker acts as his own inspector in the production process, by being provided the statistical tools to monitor a particular activity, in order to prevent the manufacture of bad parts or waste material.

The concept of SPC can be applied to any production process. For raw materials, this could mean the gathering and plotting of data that are normally available in the quality control department, covering the testing of samples taken at specific intervals during the product process. A pattern is defined that will point out any trends or cycles of quality. Any definition of points or trends above the norm indicates problems in the processing system.

While variation is a simple fact of nature and particularly true for raw materials, random influence variation must be eliminated in order to guarantee the supply of a quality product to the market.

The implementation of an SPC system within any raw material production process will assure customers that the producer is working diligently towards controlling variation in order to prevent the production of an off-grade product.

Producers should not be surprised that a consumer requests to view and audit their SPC system: this is a positive step and indicates a desire of the consumer to enter into a relationship of longer term and to become a "partner" in guaranteeing quality.

2.7.6 CONTRACTS

Once seller and buyer have agreed that a raw material will be able to meet the quality required and will be delivered at an acceptable price, then the general terms and conditions should be recorded in contract form. This serves not only to protect the interests of all parties but, more importantly, sets down what has been agreed.

A contract in raw material trade is usually made up of four parts:

1. *Description of the Material.*

This usually incorporates a definition of the material and grade, which is producer-oriented, as well as a material specification, which is the buyer's requirement for product quality. A proper description should include any desired minimum/maximum chemical and physical characteristics, plus any ancillary data (specific gravity, loss on ignition, moisture, etc).

2. *Quantity to be Shipped.*

This is defined as an exact tonnage number, with a percentage variance at either the seller's or the buyer's option. Shipment of precise tonnages are difficult due to differences in methods of transport, and variances provide comfort to both parties.

This section also defines the date when a shipment is to be made and/or when it is to be delivered.

3. *Price and Payment Terms.*

In addition to describing the cost of the raw material in a defined currency, the price should also give a definition as to where the transfer of title takes place. This is simply done through the use of either one of the terms FOB/CFR/CIF (named port), which are INCOTERMS (described in detail at the end of this segment).

This part of the contract also defines in what form the material is to be shipped: in bulk or in bags (with exact weight described as well as construction of the bags) or in any other mode of packaging.

Payment terms for domestic shipments usually stipulate a procedure for initiating payment to a bank, or directly to the seller, by cheque or telegraphic transfer. Export shipments are normally channeled through banks, where the buyer establishes a "letter of credit" between its bank and the seller's bank, and payment is provided for once the seller has met the terms and conditions defined in the contract.

4. *Quality and Quantity Assurance.*

This final section of the contract outlines the procedures employed to assure quality and quantity at load port and at port of destination. Since the transport involves third parties whose only connection to the transaction is the carriage of goods, an independent company is hired at the port of loading and unloading to sample and analyze and report on the quality of the material. The quantity loaded and discharged is also independently verified.

Costs for the quality and quantity assurance are normally shared between seller and buyer.

There are several well-known US and international companies who act as independent "superintendent" quality and quantity verifiers:

Andrew S. McCreath & Co.
Commercial Testing and Eng. Co.
Ste Generale de Surveillance (SGS)
Inspectorate Griffith
Alfred H. Knight

No discussion on contracts would be complete without a definition of INCOTERMS. International commercial transactions can be risky business: misunderstandings, disputes, and litigation are time consuming and costly. In 1936, the International Chamber of Commerce organized a reliable set of international rules for the interpretation of trade terms. Through their use, any uncertainties due to different interpretations of the terms used in various countries can be avoided or considerably reduced.

The current amendment, INCOTERMS 1990, incorporates changes in transportation techniques and renders them compatible to electronic data interchange (EDI). They are presented in a format which allows the seller and the buyer to follow a step-by-step process to determine their respective obligations.

The terms are grouped in four different categories:

Group E Departure	Seller makes the goods available to the buyers at the seller's own premises.	
	EXW	Ex Works
Group F Main Carriage Unpaid	Seller is called upon to deliver the goods to a carrier appointed by the buyer.	
	FCA	Free Carrier
	FAS	Free Alongside Ship
	FOB	Free On Board
Group C Main Carriage Paid	Seller has to contract for carriage, but without assuming the risk of loss of or damage to the goods or additional costs due to events occurring after shipment and dispatch.	
	CFR	Cost and Freight
	CIF	Cost, Insurance, and Freight
	CPT	Carriage Paid To
	CIP	Carriage and Insurance Paid To
Group D Arrival	Seller has to bear all costs and risks needed to bring the goods to the country of destination.	
	DAF	Delivered at Frontier
	DES	Delivered Ex Ship
	DEQ	Delivered Ex Quay
	DDU	Delivered Duty Unpaid
	DDP	Delivered Duty Paid

Any party desiring to use these rules should specify that their contracts will be governed by "INCOTERMS 1990."

2.7.7 CURRENCY

As a general rule, the US dollar is the common currency in which most industrial raw materials are denominated. Barring an unlikely devaluation of the dollar by the United States government, this eliminates the need for the US-based producer to be concerned about receiving a lower value for his commodity. However, since exchange rates do have a tendency to fluctuate based on a country's economic and competitive position during the course of a contract or a shipment, it is possible that either the buyer or seller will make a gain (or a loss) on the value of their shipment in terms of their domestic currency.

2.7.8 COUNTERTRADE

In Colonial times, barter on the United States frontier represented commercial exchange for furs and wheat in compensation for rum and whale oil. Today companies specializing in barter execute both multitrader and international transactions for goods or services, where no money is exchanged and deliveries are covered in one contract.

Given the complexities of today's trade of raw materials, *counterpurchase* has emerged as the most popular form of countertrade since barter relies upon a coincidence of identical needs. Counterpurchase includes as a condition of the sale, where the seller agrees to purchase goods or services from the buyer. The cost of the transaction is tied to the value of the raw materials sold.

Compensation, otherwise understood as buy-back, is similar to counterpurchase except that the products received in the transaction are resultant or connected, instead of their being unrelated to those sold.

A recent innovation has been the establishment of barter-exchange companies where reciprocal trade is established through trade dollars, which are units of account denoting the right to receive, or obligation to pay, in goods or services available from the participating members of an exchange.

What this means is that a mining operation requiring a new jaw crusher could transact with a barter exchange company that in turn would arrange to receive a specific tonnage of raw mineral from the mining company, including some cash, in exchange for the desired equipment. This type of transaction is gaining popularity but requires time and patience to satisfy all parties involved.

Any countertrade operation requires a thorough understanding by both the seller and the buyer of what they are getting into.

2.7.9 ROLE OF TRADING COMPANIES

One of the important questions that any producer must answer early in developing a marketing plan is "who will and how will the marketing be handled?" One method is through the use of professional marketing organizations, sometimes known as *traders* who act on behalf of either the producer, the consumer, or for their own account to deliver a wide variety of raw materials to the marketplace.

Since the first initiatives of commerce, producers have utilized the services of a middleman to market their goods or have appointed an agent to act on their behalf to introduce and market their goods to a particular market. Whatever the choice, the overall purpose is the same: to supply the product at an equitable price, within the shortest time, for the economic benefit of both parties.

Once a producer has decided that its goals will be accomplished by working through a trader, what relationship with that trader does it choose?

If the producer decides that it wants someone to work alongside to introduce its raw material to the market, then it has chosen a long-term approach where the trader acts as agent.

In the long run, this usually yields a higher return, as well as providing two other important benefits:

1. The preservation of the identity of the producer's product.
2. The promotion of the producer's good name.

In deciding on an agency relationship, the mineral producer must also accept that the mineral trader will be working as its extended arm in the marketplace, and the producer must be prepared for not only joint decisions regarding the marketing strategy but also joint visits to key accounts.

There are other advantages that an agency relationship brings to the producer of minerals, and these are related to the overall concept of a mineral trading organization, which is that of a fully integrated team taking a product from the mining center and delivering it all the way to the consumer market, sometimes via an interim processor, at a competitive price.

To accomplish this, the producer can expect the trading organization to possess:

1. A professional marketing and sales personnel team.
2. Frequent contacts with consumers at both the technical and commercial levels.
3. Intimate technical and commercial knowledge of the products being sold.
4. Established success in all segments of the industry served.
5. The strength of a diverse customer base.

There are circumstances under which the industrial mineral producer may find it to its advantage to sell its mineral directly to a mineral trader and, by consequence, have little involvement in the market development.

For instance, during the early 1970s, the majority of mineral sand producers in Australia benefited from the practice of selling off their zircon or rutile mineral sands stockpiles to traders, thereby improving their year-end financial position. From that point on, the mineral trader acted as principal to bring those minerals to the marketplace. In the process, the mineral sand lost its identity and, more often than not, the producer later competed with its own product in the market.

Today the major mineral sand producers are represented worldwide by mineral trading organizations, and both the consumer and the producer now benefit from a stable pricing and fully integrated distribution system, tied in with regular tonnage allocations. In fact, during the rapid increase in spot prices for zircon mineral sand during the last two years, traders assisted the consumer markets by entering into the trade of spot tonnages and prevented further price escalations, which might have pushed consumers into looking at substitute materials.

Another example of the mineral trader functioning as principal is in the area of industrial raw materials originating from the People's Republic of China. Although the Chinese, through the independence of their provinces, have attempted to align themselves through so-called "exclusive contracts" with trading companies or with their own foreign-based offices, few of these arrangements have worked in practice. This may be due to the fact that China has far too many official selling agencies: sometimes as many as six agencies compete with each other to sell the same mineral from one Chinese province.

In other cases, consumers have experienced difficulties when dealing directly with the Chinese due to contract interpretations and unexpected delivery delays and quality questions. Mineral traders have served this market well by being able to bridge many of these differences and offsetting some of the problems against other contracts.

In other countries, there are circumstances under which a producer may have financial considerations for selling his minerals directly to a mineral trader. As in the case of a chromite mine in the Philippines, the trader performs the important role of acting as a party to financing the operations of the mine by establishing a letter of credit for a defined tonnage. This letter of credit is then taken to a Philippine bank, and monies are advanced to meet production tonnage.

There are many trading companies involved in the marketing and distribution of raw materials. Producers should fully investigate a trading company's expertise in the markets they serve, with particular emphasis on knowledge of materials and markets competitive with the producer's raw materials.

2.7.10 INFORMATION SOURCES

There are many excellent publications that cover the subject of minerals, ores, fuels, and other industrial raw materials. In addition to covering specifics relative to new mine or market developments, most are issued on a monthly basis and contain valuable up-to-date market price information. Government agencies are also good sources of information on raw materials but may be limited to specific import and export data.

Industrial Minerals (published by Industrial Minerals Div. of Metal Bulletin Inc. of UK) 220 Fifth Ave. New York, NY 10001 Tel: 212-213-6202	Considered by many to be the monthly "textbook" on nonmetallic industrial minerals, their worldwide producers and markets.
Engineering and Mining Journal 29 North Wacker Dr. Chicago, IL 60606 Tel: 312-726-2802	Monthly magazine providing world coverage of all types of mining projects and mineral prices.
Skillings' Mining Review First Bank Place Duluth, MN 55802 Tel: 218-722-2310	Weekly magazine devoted to iron ore mining and markets.
Pit & Quarry 750 Old Oak Blvd. Cleveland, OH 44130 Tel: 216-243-8100	Monthly magazine devoted to the rock and quarry stone industries.
Mining Engineering 8307 Shaffer Parkway Littleton, CO 80127 Tel: 303-973-9550	Monthly magazine published for members of the Society for Mining, Metallurgy, and Exploration, Inc.
Ceramic Industry 5900 Harper Rd. Solon, OH 44139 Tel: 216-498-9214	Published monthly and devoted to more traditional refractory and advanced ceramics.
Ceramic Bulletin 757 Brooksedge Plaza Dr. Westerville, OH 43081 Tel: 614-890-4700	Monthly magazine published for members of the American Ceramic Society.
US Bureau of Mines Minerals Information Office Washington, DC 20241 Tel: 202-634-1001	Published bimonthly periodical *Minerals Today* as well as import commodity summaries.
Arundale Mineral Intelligence Letter Arundale & Associates 351 E. Thomas Rd. Phoenix, AZ 85012	Distributed without charge to those who are interested in, or involved with, the broad field of "minerals intelligence."
INCOTERMS 1990 ICC Publishing Corp. 156 Fifth Ave. New York, NY 10010 Tel: 212-206-1150	International Chamber of Commerce guide explains the respective obligations of buyer and seller.
International Bulk Journal Ranmore House Ranmore Rd. Dorking, Surrey RH4-H4E, UK Tel: 44-306-887433	Published monthly journal devoted to the shipping and port handling of a variety of raw materials.

CIM Bulletin 1210-3400 De Maisonneuve Blvd. W. Montreal, PQ, Canada H3Z 3B8 Tel: 514-939-2710	Monthly magazine published for members of the Canadian Institute of Mining, Metallurgy and Petroleum.
Mining Journal 60 Worship St. London EC2A 2HD, UK Tel: 44-71-377-2020	Weekly newsletter providing timely information on mine development, production, technology, equipment contracts, and company financial news.
Coal 29 North Wacker Dr. Chicago, IL 60606 Tel: 312-726-2802	Monthly magazine devoted to coal mining, processing, and markets.
Minerals Engineering Pergamon Press Inc. Maxwell House Fairview Park Elmsford, NY 10523	Monthly international journal covering developments in mineral processing and extractive metallurgy.

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Section 3 Government Role and Influence in Mining

JAN M. MUTMANSKY, ASSOCIATE EDITOR
THOMAS V. FALKIE, SECTION COORDINATOR

Chapter 3.0
INTRODUCTION

THOMAS V. FALKIE

In bygone days, the mineral engineer spent most of his/her time on the technical aspects of designing, building, and managing mineral operations. In the last few decades, mostly since the 1960s, increasing amounts of engineering manpower and time are spent dealing with federal, state, and local government agencies, laws, and regulations. It is not possible to plan, design, build, or operate mines, mills, smelters, or even office buildings without detailed consideration of government regulations and permitting requirements dealing with environment, worker health and safety, etc.

The four chapters of this section provide an overview of the social, legal, political, environmental, and health and safety aspects of mineral enterprise. The section sets the stage for the working engineer/geologist in industry, government, consulting, or academe and points the direction for more detailed study in each of these areas.

Chapter 3.1, Social-Legal-Political-Economic Impacts, describes how government works in a general way and how it applies to mining. It also discusses the socioeconomic and political aspects of mineral policy and how and why there are many constituencies to be considered, not only in mineral policy but in the actual process of planning mines.

Government plays a strong role in every facet of mineral development from exploration through mining, processing, and consumption. While there is no coherent US mineral policy and probably never will be, the mineral industry is profoundly affected by laws, regulations, and other government actions on land use, worker health and safety, environment, taxes, fiscal policy, and many other policy areas.

The chapter also discusses the importance of mining to our country's well-being and to our standard of living. Virtually everything we eat, every metallic object, most of our building materials, the cars we drive are all made from minerals or use metallic or nonmetallic or energy minerals in some way. This is taken for granted by large segments of the public. Even worse, some people believe that mineral production can be curtailed without affecting our standard of living.

The practicing engineer must deal with government and the public in every facet of his or her career. Not only must the engineer understand how government functions, but he or she should also get involved in the legislative and policy process.

Chapter 3.2, Mining Law, describes in detail provisions of the Mining Law of 1872 and other statutes and regulations dealing with mining on the public lands. The odds are very high that the mineral professional will be involved with the public lands sometime in his/her career because about one-third of the area of the United States (about 730×10^{12} acres) is owned or controlled by the federal government. In addition, the states and

Indian tribes control large amounts of land and mineral rights. Significant amounts of minerals are produced from the public lands, and these lands include much of the most geologically promising areas of our country. It is also true that increasing amounts of public lands are becoming "off-limits" to mineral exploration and development by various land-withdrawal actions of government. Perhaps two-thirds or more of the public lands is unavailable or severely restricted. About one-quarter is slightly to moderately restricted. The land withdrawal problem is not discussed in this chapter but is mentioned in Chapter 3.1.

Chapter 3.3, Health and Safety Standards, shows why and how mine health and safety laws and regulations were developed, how they are structured, and how the enforcement system works. The discussion is general, and the reader will find specifics in other chapters dealing with design and engineering. Over the past century, no subject has brought more negative attention and more controversy to the mining industry. The health and safety record of the industry has improved, but there is need for even more diligence. In fact, health and safety considerations affect virtually every design or operating decision connected with mining. Every step of the mining process, whether it be materials handling, ventilation, equipment operation, roof control, or other activity, is regulated by very detailed federal and state laws, and these laws are enforced by equally detailed procedures. This chapter provides an introduction to these standards.

Chapter 3.4, Environmental Consequences, introduces the reader to some of the major federal environmental laws dealing with clean air, clean water, land reclamation, waste disposal, preservation of animal species and antiquities, and other aspects of environmental protection. Every environmental law has some effect on mining, although the Surface Mining Control and Reclamation Act of 1977 was the first major federal regulatory law aimed specifically at mining. That law, incidentally, applies to underground and surface mining. Every one of these laws has spawned many regulations and requires extensive engineering efforts in order to obtain permits from various governmental agencies. Once again, details on many of these will be found throughout the design and engineering chapters of this *Handbook.*

The status of these laws and regulations is under constant change. Therefore, the reader should use this chapter as a guide and should always refer to the *Federal Register* for updated information.

This section will help mineral professionals at all levels and furthermore should be required study material for university students in all mineral-related fields.

Chapter 3.1
SOCIAL-LEGAL-POLITICAL-ECONOMIC IMPACTS

JOHN J. SCHANZ, JR.

3.1.1 INTRODUCTION

The impact of government permeates all facets of the operational, financial, and managerial life of a mine or a company. As a consequence, there are few topics found in this *Handbook* that are free of its imprint. This chapter is totally devoted to examining in general the breadth of government impact. The following chapters present in detail three major domains of government involvement in mining—federal control over the public lands, health and safety, and environmental standards and regulations.

Any individual or an employee of a private firm annoyed by this intrusion into his, her, or its, internal affairs can be excused for occasionally wondering, "Why do we need government to start with? It's nothing but trouble!" A moment's reflection usually recalls to mind that we do need government, and in fact the public wants government to exist. Government is not merely a nuisance and a nonproductive element in our society, despite irritation with regulation, taxes, or police powers. But these activities are the unavoidable accompaniments of providing what are known as *public goods.*

A classic example of a public good is national defense. It is not usually practical for the individual, the town, or even a state to provide for its own security and defense against a foreign enemy. As a consequence, society relies upon some form of central government, in our case the federal establishment. Numerous other examples of public goods, provided by both local and national government, come to mind: the formation or construction of courts, schools, parks, roads, and so on. In some cases, there is disagreement as to whether a good or service should be provided by public or private means. Private vs. public power production has been debated for decades. But an industrial society, unlike a rural or agrarian one, does not function cheaply and efficiently in the absence of public goods.

This chapter will first concern itself with how government functions. Without an awareness of this, one cannot really understand the policy and decision process that ultimately impacts the operation of a mine in both positive and negative ways. Some of the broader public policies with respect to mineral resources and mining are then recapitulated.

There is an array of broad social concerns of government to be considered. For example, there are needs relating to the work force directly employed by the mining company—health and safety are prime examples. When government deals with the national budget, monetary or fiscal policy, domestic and foreign trade, aid to domestic industry, or foreign policy, it will unavoidably at the same time do things of consequence to the mineral industries. Many of these economic issues are placed in a somewhat broader public policy context than in the seven chapters of Section 2. There, economic matters are explored primarily from the perspective of the individual firm or the mining industry at large.

The regulatory role of government is important. Mining on the public lands, health and safety, and environmental matters deservedly draw considerable attention. So they are examined in the following separate chapters of this section. But there are also the regulations and controls on how land is used in general, including private lands. Water laws and water allocation are extremely important to mining—and both federal and state governments are involved. Government also provides a whole array of support functions from information gathering to pilot plant construction using experimental technologies.

Finally, the question must be addressed as to what this means to the engineer in performing his tasks throughout his career. Obviously, the visibility of governmental impacts tends to expand as the engineer moves from workplace to the executive suite. But for many decisions from beginning to end, he or she will have to learn how to factor in regulations, added costs, and other forms of government involvement into the engineering process. To do this, however, involves constantly facing an element of uncertainty over time.

The task of the mining engineer is an international one. Thus it would be ideal if this chapter could address the many forms government involvement can assume in foreign lands. However, the limits of space preclude this, so the United States is the focus of attention.

3.1.2 STRUCTURE OF GOVERNMENT

Federal, state, and local governments through statutes (1) create institutions and administer them, (2) establish regulations relating to the conduct of individuals and organizations and enforce them, and (3) collect taxes and allocate the receipts. Much of this is described in general terms in the following subsections. The federal government is the primary focus of attention. However, it should be recalled that similar, but not identical, organizations and processes are encountered in the lower jurisdictions. Obviously, too much space would be required to cover each state, and much would be redundant. However, in those cases where the state or local government is the primary actor, this will be incorporated into the discussion.

3.1.2.1 Legislatures

The statutes and the budgetary support to accomplish the various designated tasks originate in the legislature. This, of course, must be accompanied by sufficient tax revenues to pay for the costs of performing them. But it is quite possible for the Congress of the United States to pass legislation and not immediately provide for it in its budget or not appropriate tax monies to pay for it. Many water projects have been authorized but not (or never) built for years for lack of an appropriation. In this fashion, Congress receives credit for approving of something, but then shows an unwillingness to divert scarce funds to the cause. This may happen when the administration and the legislature are of different parties. The executive branch has the leverage to get something proposed and approved, but then watches its program languish for the want of adequate money.

The Process: All legislative initiatives, whether proposed by members on their own in the public interest, for their constituencies, or on behalf of the current administration, have to be converted into a bill for submission. The drafting of this document is a critical step because it establishes the objective and dimensions of the intended statute. Once it enters the legislative process, it is subjected to constant proposals for modifications both large and small. The best chances for success are for legislation in which the member has particular expertise and where the bill will pass through committees where he has some leverage.

The beginning of the legislative drafting is frequently an inconspicuous event. The member of Congress may not necessarily want publicity because it invites more attention than desirable from those wishing to influence the content of the bill. Competing or opposing members may be tempted to take the initiative away from him or her. Staff doing the ground work will call upon those, both inside and outside of the Congressional family, who they know have expertise in the area or political savvy. This is an advantageous time for interested parties to be involved.

Bills customarily originate in the Senate or the House according to the nature of the bill; for example, money bills start in the House. But bills on the same issue may be drafted and submitted by members of both bodies. The bills will be assigned to the committee(s) having jurisdiction over these matters. This gives the legislative leaders some leeway as well as leading to competition among committees.

The committee leadership has important leverage in deciding when the hearings will be held on a bill, how much time will be devoted, and what happens after the hearings. Who will appear and the topics to be discussed are carefully programmed. Anyone can request time to appear, but the time available is limited. So many have to be satisfied with making a written submission for the record.

While hearings can influence the content of proposed legislation, in most cases this is not the primary goal. Rather the committee is intent on “making a record” on the various viewpoints. The committee members, as well as other members, are careful to get their position on the record by statements and questions of witnesses. The transcript will be reviewed and edited carefully by all involved. This document provides information to those who did not attend. Most importantly, if the bill becomes law, the hearings documents will help show the “intent of Congress.” This may not always be explicit in the language of the bill itself.

If the bill is to become law, it must be voted upon by the subcommittee(s) or committee(s) having jurisdiction. Once again, the chairman has leverage in deciding whether the bill has merit and has a real opportunity to become law. A member can display his responsiveness to his constituency’s concerns by drafting a bill. He may even obtain a hearing on the legislation. But only a small proportion of the bills ever justify the time needed to get them to a final vote.

A crucial step is the “mark-up” sessions held by the committee. This is the moment when the other members have an opportunity to alter the bill as originally drafted. At this point, the lobbyists representing various interest groups will apply their persuasiveness to modify or to get votes for or against. The final version is likely to be not only a compromise of interests but considerably modified from its original construct. Substitute bills may even appear.

If the bill is finally passed, it moves on to the floor of the House or Senate. If passed by a subcommittee, the bill must survive another review by the parent committee, as well as surveillance by the main committee chairman, before moving along. Once the bill arrives at the floor, it is subject to more debate and possible amendment as well as a number of parliamentary procedures that will determine whether a vote will be taken at all.

Once passed, the bill must then move to the other chamber of the Congress. That chamber then has an opportunity to review the legislation, debate it, amend it, and approve or reject it. Quite commonly, a bill from one chamber will encounter a similar but not necessarily identical bill that has survived the entire process in the other chamber. The bill received from the other chamber may replace its own legislation and be voted upon. Or the chamber may approve its own version and the two versions are referred to a conference committee of representatives from both chambers. This group then seeks to arrive at a version acceptable to both sides, which is then referred back to both chambers for a final vote.

The final hurdle is the president. He may sign it, let it become law without his approval, prevent it from becoming law by not signing before the session ends, or veto it and return it to Congress. In the latter case, both chambers must override his veto with a two-thirds majority for it to become law.

Beyond Legislation: The role of the legislature does not end with the passage of a statute. Oversight of the implementation of a program or enforcement of a regulatory regime is common. If budgets and appropriations are involved, then the oversight is to determine the manner in which the funds are being expended.

The legislature may also engage in hearings and investigations on matters of interest. This may be prompted by a concern that laws are being violated by individuals, firms, or members of administrative agencies. Or it may be exploratory in nature to determine if action is needed or legislation should be formulated.

Congress is a part of the policy process but is constrained on how far it can go. As a nonoperational unit, it cannot execute its own wishes; this is done by an administrative or management agency. It is also impossible for Congress to draft a document that can stand alone as *the national policy*. Any national policy is in essence the composite of the acts of Congress and the implementation of those pieces of legislation by the responsible agencies.

One of the most challenging aspects of legislation is how far the legislature can, must, or should go in specifying how the statute is to be implemented. Broadly stated objectives may lead to little action or so much discretion that the intent of the legislature is not achieved. Overspecification allows the responsible agency little flexibility in adjusting to circumstances and may require them to be concerned more with procedure than substance. And in most cases, the legislature may not have the technical ability to formulate regulatory detail or to foresee what will be faced by an administrator on a daily basis. How to address the many variables in the reclamation of surface mined lands had to be faced in formulating the language of the Surface Mine Control and Reclamation Act (SMCRA). Even in its final wording, Congress recognized that it was most relevant to coal mining but not to all the other forms of surface mining.

Once the legislature has acted, this places limits upon the future actions of the administrative and regulatory agencies. Executives and regulators frequently have considerable discretion in the conduct of their offices. But it is important for them to recognize how far that discretion can reach without coming into conflict with the letter of the law.

The engineer in design and in operations encounters regulations and procedures that are reflective of prior actions by legislatures. In many cases, these are in response to actual laws, and conformance is a legal requirement. But there may also be some freedom, official discretion, or appeal permitted under the law. So familiarity with the details of legislation can be useful to the engineer.

The mine executive comes into broader contact with the consequences of legislation, from tax burdens to constraints on how the firm or the mine can be operated. Here familiarity with the law and legislative process is important. Background information and reasoned personal arguments placed in the hands of legislators and their staff can be important when they are timely. Testimony and submissions to committee hearings are useful, but their impact may be limited. When actual voting begins, political sensitivity peaks, and the time for more data is passed.

There is a tendency among those not familiar with Washington to feel that legislators would perhaps perform "better" if they were "more informed about the real world." However, each member of Congress has available personal staff who are assigned areas in which to keep informed; numerous people attached to the committees to which they belong, among them technically trained professionals; and four well-manned support agencies created to respond to requests for help from any member or committee from both the majority and minority sides. In addition, there are all those individuals from the outside world eager to inform them of the "truth." In reality, the legislator may frequently be a victim of more knowledge than is possible to digest.

The committees that are more frequently interested in matters relating to mining in particular are the Senate Committees for Energy and Natural Resources and for Environment and Public Works, and the House Committees on Energy and Commerce, Interior and Insular Affairs, and Science, Space, and Technology. Within these main committees are subcommittees that focus on more specialized matters. The subcommittees are particularly important in the House with its larger membership.

3.1.2.2 Executive Branch Agencies

The administrative agencies are in the executive branch of government created for the purpose of administering, managing, and regulating. They perform not only the functions of government designated by the Constitution but also those subsequently made a part of our public laws by the Congress. Only the president and vice president are elected. All others are either appointive positions filled with the advice and consent of the Congress or employees hired from funds appropriated by the Congress.

Budgets: The Congress for certain activities includes specific funding as part of its legislation. But for most of the continuing activities of the administration, annual funding legislation is required. This funding must not only provide for specific activities mandated under existing laws (e.g., payment of social security to those eligible), but also for the various functions of the administrative agencies where there is some discretion as to what will be done (e.g., how to conduct research to keep domestic mining viable).

The budget proposal to be made to the Congress each year by the Executive Office of the President is coordinated through the Office of Management and Budget (OMB). This consists of the budgets proposed by each segment of the administration's organization to perform their obligatory programs and also monies to engage in new activities supported by the Executive. This budget is coupled with estimates of the federal income during that period from various sources including existing or proposed tax revenues. For new programs or changes in old programs, as well as the necessary changes in taxation, the Executive Branch will propose the enabling legislation and will appear before Congress to support their proposals.

The Congress, in turn, designs its own budget and makes its own estimates of income. Since the Congress's priorities do not necessarily match those of the president, the various amounts do not necessarily match those coming from the Executive Branch. The budget committees proposals are reviewed, modified, and adopted by Congress to provide guidelines for all of the Congressional committees as to the monies likely to be available. By September 30, the Congress attempts to pass bills originating from the appropriation committees to provide money to the Administration for the coming fiscal year. If the deadline cannot be met, stopgap funding is provided through the use of a "continuing resolution." In recent years, this has been relied upon frequently, in effect, short circuiting the customary authorization and appropriation process.

Each branch of government then has available to it an actual amount of money. How closely this matches what was originally requested can vary considerably. Congress may have added things to be done, not funded others, and/or altered the total. But much of the budget is assigned to broad categories of activity, allowing the agency considerable freedom on exactly what is done.

Where not constrained by Congress, a large department may also make determinations as to which bureaus or offices will receive more or less than previously and how these funds will be allocated or left to the discretion of that agency's manager. Some funds may not be immediately allocated by a departmental secretary or comparable official but kept under his control for later disbursement.

The OMB reenters the picture at this point. Once the appropriations are received, then the OMB becomes an overseer as to how the funds are spent. This provides on one hand, the need to keep overall expenditures in line with the amounts appropriated. But on the other hand, it also allows the Executive Branch to control to considerable extent how its various agencies operate and keep in tune with the overall objectives of the Executive. Historically, the importance of the OMB has varied as administrations have changed. At one extreme, its functions have been straightforward monitoring and auditing of the expenditures made for various purposes, that is, management *of the* budget. But the OMB also can be used as a potent policy management tool by the president, that is, dealing in management *and* budget. In recent years, OMB has been an influential part of how the federal government actually functions.

Policy Making: What is the policy-making process for the administrative side of the government? Even a rather small government agency will refer to its "policies." This usually involves procedural matters relating to how that office performs its assigned functions.

But as one moves up through the successive layers of government, the term "policy" changes meaning. The US Bureau of Mines may adopt a policy relating to the research it will undertake using the funds made available to it. This has a real impact on the research that is conducted. The Secretary of Interior, at the same time, may be making even broader policy decisions on the overall emphasis of his departments on the matters at hand. These have less to do with procedure and daily management but can be consequential. Above all of this, there still remains the overarching national policy objectives of the president and his staff advisors.

An awareness of the many facets of the total policy formulation process underscores how complex the federal policy fabric may become. This is why identifying the origins of "a policy" may be difficult. One finds many who claim they do not make policy but merely implement or enforce it. In fact, policy evolves from many places—from those below making suggestions and recommendations to their superiors as well as by senior officials setting general guidelines to be followed by those below them.

Regulation: One of the major roles of central government is regulatory. Many people prefer and one can find many nations that have taken the stance of "the less the better." But there are many societies where a strong hand by government is applied as well as welcomed by the people. Regulation is likely to be considered to be beneficial if individuals or organizations are not able to, or will not, control their actions in a manner that respects the interests of the populace at large.

Thus we find that the inspection of food or instituting other forms of health regulation is generally accepted as being a proper role for government. But as the regulatory regimes expand their domains, public resistance is likely to grow apace and to demonstrate the merit of regulatory rulings is more difficult. For example, an antitrust ruling may be more debatable than defining potable water qualities.

For the most part, it has been considered prudent to separate regulatory activities from the other functions of government. In some cases regulatory bodies may be a part of departments with other functions, but they are kept operationally separate from the other agencies. For example, the Federal Energy Regulatory Commission is a separate entity within the Department of Energy. A few examples of visible, independent regulatory offices are the Consumer Product Safety Commission, Federal Communications Commission, Interstate Commerce Commission, Occupational Safety and Health Administration, Public Health Service, and the Securities and Exchange Commission.

The various parts of the administrative or Executive Branch of the federal government are the ones with which people and firms are most familiar. They serve and regulate in a variety of ways which provide almost daily contact, both welcome and unwelcome, admired or ridiculed. The individual, while free to complain to the "proper authority" usually feels that there is little that he can do. Businesses, however, do seem to have better access and may very well have more influence on the manner and effectiveness of the functioning of federal and local government.

Administrative or regulatory agencies do not simply decide how they will operate away from public view. A significant part of their official activity is made a part of the public record. Many meetings must be held in a manner which provides public access. The more important procedures and rules they establish frequently require advance notification, public hearings, or periods for written comment to be considered before they can go into affect. Individuals and firms should be alert to use this access to its fullest extent in their own self-interest.

The most visible branch of government to the mining engineer is the Department of Interior and its various departments such as the Bureau of Land Management, Bureau of Mines, Geological Survey, Mineral Management Service, and the Office of Surface Mining. Other Executive Branch agencies with which mining companies have frequent contact include, among others, the Department of Defense, Department of Energy, Environmental Protection Agency, Federal Energy Regulatory Commission, Federal Trade Commission, International Trade Commission, Mine Safety and Health Administration, and the Securities and Exchange Commission.

3.1.2.3 The Judiciary

The Judiciary is the branch designated to interpret the Constitution and the laws provided by legislators, to ascertain if the other two branches are functioning in accord with these documents, to resolve conflicts, and to assign proper penalties or relief when the law is violated or inequities are identified. There are many layers to the judiciary, from local traffic courts to the US Supreme Court. The functions also vary from original hearing through subsequent appeal and as to what body of law for which they are responsible.

Given its place in the structure of our government, one does not think of the judiciary as part of the policy process. But that does not suggest it is solely a reactive body. To the contrary, the judiciary in many instances has a very powerful impact on the way in which things are accomplished.

With the respect to the legislatures and the executive branches, the judiciary is charged with ascertaining if these bodies are functioning properly within the provisions of the appropriate constitutions as passed or subsequently amended by the representatives of the people. This establishes the basic ground rules as to authority and procedure for all parts of government. If the judiciary finds a violation, then that action can be reversed by the court. If the legislature has taken the improper action, then it must find another way to accomplish its goals that is acceptable to the court or take the more difficult route of a constitutional amendment.

Administrative agencies must not only perform in accord with the Constitution but also be in conformance with any legislation passed by the legislatures that relates to them. This means that responsible officials of the government may be subject to suits filed against them charging violations of the law or nonconformance to the procedures they have established to execute that which the law charges them to do.

In some cases, the impact of the courts on the engineer may be quite significant. If the courts find that a government agency or a firm is not in accordance with the law, a regulation, or a procedure, the court may specify what needs to be done by government or industry to come into conformance. For example, under the requirement that an Environmental Impact Statement (EIS) should consider various alternatives for the development of the nation's natural resources, the court may decide that not all alternatives have been adequately considered or that the technical details are insufficient to make a judgment among them.

In some cases, the law is not sufficiently specific or clear. Then the court may decide what was the intent of the Congress or what the meaning of the certain language should be. The implementation of the Mining Law of 1872 has been refined by over a century of legal effort dedicated to interpreting legal "words of art." There has also been a need to develop practical ways to enforce the law, such as how to determine whether or not "valuable" minerals are present within a mining claim.

3.1.2.4 Interrelationship of Mining and Government

In the early stages of the emergence of an industrial economy, the natural resource industries are likely to be a major focus of attention in national affairs. In this respect, the public and government are particularly solicitous of the agriculture sector—perhaps as a reflection of a basic concern about adequate supplies of essential food and clothing items. In the US government, we find a separate Department of Agriculture, separate Congressional committees for agriculture, and a complex fabric of farm policy developed over many decades.

This is not to suggest that the mining sector has not experienced its own mix of supportive government activity, but it is of a different magnitude. We find that there are some specialized bureaus and subcommittees. The kinds of policies and actions they generate is reviewed in the next section.

Importance of Mining: The minerals industry has sought over the years to publicize its fundamental importance to the nation's industrial economy. This is probably appreciated to some degree by most of the public. But the extractive industries lack visibility in that their output is not usually an end product as perceived by the final consumer. People drive automobiles, not the separate materials in the assembly; turn on electrical appliances, not burn coal; and wear a garment, not the machine that made it. As a consequence, except in periods of shortage or wartime, public concern about mining and processing tends to be localized rather than having a national focus of concern. Moreover, as an industrial economy matures, manufacturing, commerce, services, and other economic activities expand while segments of the primary resource industries, despite growing in absolute terms, may diminish in relative size. As this occurs, mineral enterprises may tend to become "just another part of industry."

It is important to be aware of the modest size of the extractive industries—fuels, metals, and nonmetallics—compared to the rest of the economy. Collectively, using Bureau of Mines historical data, they have provided a fairly predictable proportion of the nation's national product since 1920, varying from 2 to 9% between the top and the bottom of the business cycle. This cyclic behavior reflects the fact that minerals are closely tied to the consumption of capital goods by basic industry. But in 1989 despite a healthy economy, the value of US fuel and nonfuel mineral production is estimated to have been approximately $128 billion or 2.4% of the Gross National Product (GNP). This was at the lower limit of the long-term average situation because the mineral commodities were experiencing price declines in the decade of the 1980s. Also the structure of the US economy had been changing over the previous two decades.

It is also informative to separate out the value of the three subsectors of mineral production. The role of domestic metals output has shrunk relative to the national economy as well as compared to total mineral output. Despite the considerable attention to energy conservation by the public and industry, the fossil fuels still command a high unit value and remain the dominant member of the mineral industries. Finally, the nonmetallic group has in recent years moved ahead of the metals in terms of gross value of annual output. In 1989, the estimated individual contributions to GNP were fuels 1.8%, nonmetallics 0.4% and metals 0.2%. The total work force in extractive industries in 1988 was about 753,000.

As noted previously, there tends to be a public perception of national importance concerning the agriculture sector. However, in 1989, gross farm income contributed 3.6% to GNP, about 1% higher than minerals. There were 3.0 million workers on the nation's farms out of a total US civilian labor force of about 122 million in 1988. These numbers are somewhat larger than those for mining, but do not necessarily suggest great political leverage. But one must recognize that the various activities of agriculture are widespread and found in almost every county of the country. Mining operations, in contrast, are found in a more limited number of locales, and they are not relatively important in many states. Thus mining's immediate impact has always been highly regional or localized.

The decline in some parts of the mining sector since 1974 has been particularly devastating. From a high of 167,200 in 1974, the metal mining and smelting work force in 1988 was only 49,500. This reduced its proportion of the US nonagricultural work force to less than 1.0%. Congress has expressed its concern about how this has an adverse economic impact on many local communities and in some cases entire states. But its response in the way of actual remedial measures tends to be tempered by consideration of any such actions on the work force in other industrial sectors as well as on the consuming industries and public.

Legislative proposals to assist mining are judged within the context of overall industrial or economic policy. The two dimensions are not always complimentary; for example, assistance to mining may mean that raw material prices and consumer products will become more expensive. The political resolution of this kind of conflict may not always be the one sought by the mining industry.

3.1.2.5 Other Jurisdictions

The engineer's or mine executive's attention should not or cannot be totally absorbed by consideration of the federal government's actions. Administrative activity, support, regulation, and involvement in mining by government extends over many jurisdictions. Thus we have local or municipal, county, state, federal, and international institutions. To mine minerals and sell them triggers interaction at many political levels. Security by local police, road maintenance by counties, state taxation, or international agreements on trade are just examples of the many kinds of government involvement that go beyond the federal establishment. To function, an engineer may have to obtain a local permit, conform to state labor laws, deal with a regional federal environmental office, and/or ship to a foreign destination that will apply a tariff to his product at the port of entry. Obviously, to be unaware of or to ignore any of the particulars of how these jurisdictions function and interact is a serious oversight on the part of the engineer that can negate all of the value to be found in the technical competence of his work.

Previously, it was described how the mine and the entire mineral industry must compete for attention with all other sectors of the economy in dealing with the federal government. It is noted that this does not necessarily work to mining's advantage. However, the economic importance of mining can escalate as one moves down to the smaller jurisdictions. Local governmental actions taken concerning taxation, land regulation, fiscal policy, and other domains may recognize more fully the particular needs or problems of mining. The consequences of a mine opening, declining employment, or a closing are important, can get attention, and will generate support. To pursue these opportunities requires the same familiarity with process and a sensitivity to good timing that is suggested for company or individual involvement with the federal government. But the specifics are quite different and vary widely from one operating location to another. This suggests considerable preparation is in order for the engineer, and that may have to be repeated throughout his or her career.

3.1.3 CREATION AND IMPLEMENTATION OF POLICY

The fact that various kinds of policies are created, interpreted, and altered at all branches and levels of government is discussed in the previous section. Now we turn to the policy formulation process itself as it relates to the mining industry in broad national terms. The mineral policy of the United States is the cumulative set of actions that the various branches of government have taken over time and the manner in which these have been subsequently interpreted and implemented.

Occasionally government may set for itself a specific or clear-cut objective. The Mining and Minerals Policy Act of 1970 is an example. The key paragraph states in essence that the United States will develop a sound and stable minerals industry, it will pursue orderly and economic development of its mineral resources, it will foster mineral research, and it will seek to lessen the impact of mining upon the physical environment. The actions of the government a few years later with respect to energy, while not quite as simple and direct as with the Mineral Policy Act, were rather uniformly directed toward reducing the national dependency on imported petroleum. But time has demonstrated that despite the clarity of the goal set for the nation, achieving it is a different matter.

It is perhaps better to visualize the national mineral policy to be a complex matrix of legislation, orders, and regulations developed gradually by incremental steps. In many cases the individual elements may even be in conflict with one another. Nor does this matrix stay fixed over time-new objectives are added and the manner in which goals are pursued is changed.

The following chapters in Section 3 discuss major components of current national mineral policy. These are the character of our current implementation of the Mining Law of 1872, how we now seek to enhance mine health and safety standards, and how mining is being impacted by national efforts to mitigate environmental consequences of mineral extraction and processing. None of these various elements that constitute our national policy has remained unchanged over the years since each was first introduced into the policy matrix.

3.1.3.1 Serving Many Publics

Whenever a bill is drafted, an executive order circulated, or a regulation proposed it immediately becomes apparent how many "publics" are impacted. Thus there is a national interest, a regional interest, a state interest, and a local interest to be served. Coupled with that there are the various participants in industrial enterprise: stockholders, managers, workers, suppliers, and consumers. Nor are the enterprises themselves monolithic in that they can be large or small as well as domestic or international in character. The members of these various publics will want policies to enhance their situation.

It is quite apparent that, in some fashion, a democratic government needs to arrive as often as possible at policies that benefit more publics than those experiencing extensive negative impacts. It is not often that the Congress will experience the comfortable situation of a strong public mandate and considerable support among its entire membership. Rather, most legislation that emerges must deal with the totality of society and the many interactions that may be involved. From the drafting of legislation designed to address the problems of a constituency or a particular public to having a law signed by the president is a process of resolving conflicts, compromising, and building a consensus that will provide a majority affirmative vote.

The Congress in its efforts tends to gravitate toward not what any one individual would view as the "best" answer to a national need but rather to what might be described as the "least painful" alternative. This is most likely to occur when many members find no serious objection to what is being voted upon, and those who are most directly concerned find more advantages than disadvantages in the final version being proposed for enactment. If this is not the case, Congress may simply choose not to act at all.

While Congress may often be the first to have to deal with the many publics in arriving at national policies, the executive and judicial branch are also similarly affected. Either in serving the intent of Congress or performing their assigned administrative roles, the departments and agencies of government will submit their orders and regulations to public scrutiny before putting them into effect. The various publics will once again react from their own economic or personal viewpoints. While directed or restrained by the statutes which prompt the actions taken, the officials involved will attempt to respond to the extent possible. This is essential if their subsequent efforts to implement the policies are to be both successful and efficient in practice.

The administrative agency with which the mining industry has had the closest involvement over the years is the Department of the Interior. It itself is a microcosm of the many interests that go to make up the national interest. In addition to the Geological Survey and the Bureau of Mines, well known to all mineral specialists, there is a diverse group of other agencies involved in both technical and land management matters (e.g., the Bureau of Land Management, Bureau of Reclamation, Fish and Wildlife Service, National Park Service, Bureau of Indian Affairs, and Mineral Management Service).

Each one of Interior's agencies is involved in a different set of existing or potential uses for the lands and resources of the United States. Various kinds of land uses can be compatible, but more frequently there are likely to be conflicts and competition among them for exclusive use. To illustrate, a proposal for a new dam and reservoir may remove land from use for mining, hunting, parks, grazing or farming, and wildlife protection. Even among the consumers of water, there may be controversy because the past patterns of water supply will be altered.

Not only are the various parts of Interior charged with being responsible for different user groups and representing their viewpoints, they may also come under the jurisdiction of different committees of Congress. Obviously, an oversight hearing before a House Subcommittee on actions taken by the Secretary of Interior will have quite a different thrust depending upon whether it is the Subcommittee on Energy and Power, Mining and Natural Resources, National Parks and Public Lands, or Water and Power Resources.

Despite the efforts of the Congress and the Administration to arrive at policies and their implementation that minimize the conflict between the various publics involved, a challenge to the legality or the equity of the law itself or how its being employed may still be made. The courts are the final haven for expressions of opposition. At this point, the law itself may be struck down, its manner of implementation may be altered, or inequities in its impact may be resolved. As a consequence, national mineral policies are not fully defined until the various elements have either avoided challenge in the courts or have been further refined by judicial decisions.

3.1.3.2 Industrial and Economic Policy in General

In this chapter, we are particularly interested in the mineral policies of the United States. But it is important to recall that the mining industry, in addition to when it is intentionally considered as a separate sector, is also impacted by the broader economic and industrial policies of the nation. During the late 1970s and early 1980s, many of the difficulties experienced by domestic mining were in part a reflection of the fiscal and monetary policies of the United States. These policies, and changes in them, were reflections of general concern about the economic and industrial welfare of the United States. While it was recognized that the mining sector might be affected in a variety of ways, the policies were actually neutral with respect to mining.

During the extended battle against inflation and high interest rates from 1977 through 1982, a strong US dollar emerged as result of the various fiscal and monetary strategies used. Not solely, but to an important degree, this made the output of US mines more expensive than foreign output because the exchange rate for other currencies lowered the comparative cost of their output. In their own self-interest, US and foreign raw-material consuming industries bought less of US origin. Subsequently, inflation eased, the dollar became cheaper in currency exchanges, and US mines introduced cost-reducing measures. The mining industry found itself in an improved condition. But the policies pursued to protect the national economy, while of real significance to mining, never were influenced to any major degree by what was good or bad for mining per se.

National industrial policy, as was described for mineral policy itself, is the composite of many laws, executive decisions, and regulations. Taxation and trade policy are of perennial importance to industry. When it first became apparent that US industry seemed to be falling behind other nations, such as Japan, in our ability to do as well as we had in the past in international competition, our aging plant and low investment rate were blamed. The Accelerated Cost Recovery System (ACRS) was introduced in 1981 to correct the situation. ACRS allowed industry to depreciate equipment faster for tax purposes than previously. This would act as a stimulant. There was also introduced a 10% investment tax credit (ITC) for the purchase of certain machines and equipment. It is believed that capital investment in industry and in mining probably did show a positive response.

But Congress may also be motivated to remove incentives that it had previously approved. The ACRS and ITC were removed in the changes that were made in the Tax Reform Act of 1986. This Act was designed to restructure the tax system with respect to rates, incentives, penalties, complexity, and other features that resulted from the product of many years of Congressional tinkering with the tax laws on an item-by-item basis. On balance it was to be "tax neutral" and not change the total revenues to be received in the future by the Treasury. Efforts were made to replace any lost tax incomes with increases in taxes elsewhere. Generally speaking, while industry benefited from the lower rates it had to pay, the removal of tax "shelters" and industry tax incentives tended to be negative from the viewpoint of industry. However, incentives targeted for the mining industry in particular were left largely untouched.

Energy Policy as an Example: The energy policy considerations and actions of the 1970s and 1980s are also an excellent example of how policy can become multifaceted. Prior to 1970, most individuals would have viewed the national energy policy as being for the largest part government actions directed toward assuring a reliable supply of domestic energy over time. But a United States, which no longer had a surplus capacity to produce petroleum, that faced a series of interruptions in supply in the 1970s and a tenfold increase in the price of crude oil, had to make decisions on a much broader front.

It was quite apparent that future energy policy had to concern itself with demand as well as supply. The questions to be answered were abundant. In the event of interruptions in the flow of petroleum products, should government resolve the shortages with allocations or let the market and the consumer adjust to the higher prices on their own? Should taxes on fuels be used to influence consumers' energy-buying habits? Should the poorer segments of the population be insulated from these economic shocks? How could more energy-efficient equipment be made available and then actually bought by energy users? Should tax incentives be employed to encourage this to happen? If so, what kinds of energy savings and capital investments should be eligible? Should standards be set for energy consumption, for example, miles per gallon for vehicles or controlling the design of buildings?

A similar array of questions surfaced on the supply side. Since the immediate problem was the national reliance on petroleum imports, should policy focus on this in particular or on energy supply in general? With respect to petroleum itself, should there be an emergency stockpile of oil? If so, how should it be created, how large should it be, and how should it be used? Should there be new incentives for the exploration, development, and enhanced recovery of domestic oil? Or would it be better to begin to direct the national effort toward reducing dependency on crude oil in general? If so, of the more abundant energy forms that should be encouraged, should the other familiar fossil fuels to be used directly or converted into synthetic hydrocarbons? Or would it be better to tackle the more difficult task of shifting toward the renewable forms of energy created from current solar influx? In all of this, how should the government be involved-basic research, applied research, development, demonstration, or commercialization? Did the oil supply threat justify subsidizing these new supply initiatives?

As a mineral or energy policy matrix becomes more complex, it spills over into other areas of social concern. Had past policies concerning encouraging home ownership or improving national transport unintentionally accentuated our energy problem? Should we change these policies in the interest of energy? Should energy pricing which had been rewarding more consumption be inverted so that those who used less would pay less per unit? Improved air and water quality were relatively new national goals and are closely tied to energy use. Should new, more stringent standards be reexamined in the interest of easing the energy supply problem? There had been a growing effort to set aside public lands in a variety of ways that reduced their availability for extractive purposes. Should access to the tidelands, continental shelves, and onshore public lands be improved so that any remaining energy resources could be located and developed?

3.1.3.3 Current Mining and Mineral Policies

World War I demonstrated clearly that industrial capacity and the raw material supplies to support it were the "sinews of war." Many authors in the next 30 years devoted attention to what nations or combinations of nations had the capability of entering into a major, worldwide conflict. World War II confirmed this, and the combatants relied upon strategic plans that included assuring supplies for themselves while denying them to the opposition. Despite the richness of its resource base, the United States recognized its vulnerability in certain minerals not produced domestically in adequate amounts. These are known as *strategic and critical materials.*

As a consequence, one of the primary focuses of national mineral policy for a half century has been to stockpile these materials that were viewed to be of critical importance to have on hand in a national emergency (see Chapter 2.6). The strategic goals of the stockpile, the materials stocked, and the quantities stocked have been modified frequently since the first Stockpiling Act was passed in 1946. Management was handled for many years by the General Services Administration but is now in the Department of Defense. As of Mar. 31, 1990, 66 commodities were in the stockpile, and the inventory was valued at $9.7 billion.

A companion element in assuring national security is the Defense Production Act of 1950. This authorizes materials allocation in an emergency and provides for loans and guarantees to companies to expand capacity and facilitate production. In addition, the government itself could become involved in purchasing and resale of material supplies.

However, anticipation of emergencies is not sufficient in the absence of a functioning raw-materials supply industry. Four pieces of legislation stand out in this respect: the Mining Law of 1872, discussed in detail in Chapter 3.2; the special tax provisions for mining discussed in Chapter 2.4; the Mineral Policy Act of 1970; and the National Materials and Minerals Policy, Research, and Development Act of 1980. Each of these is directed specifically to encourage the mining sector and is not a part of the broader economic and industrial policies of the nation. The Mining Law provides access to the public lands for mineral exploration and development at modest cost, and the holders of claims can eventually obtain fee title to the land. Certain mineral resources, in particular the fossil fuels found in large sedimentary basins, were removed from this "location" system by the Mineral Leasing Act of 1920.

In the mid-1920s, the Congress addressed the problem of the taxation of mineral deposits. A minable deposit can be either discovered by a mining firm and developed or purchased after discovery by someone else. The value of a given discovery can exceed by many times the directly related expenditures of the successful explorationist, but not necessarily his total expenditures, or money spent by those who are less successful. Even after discovery, the actual size and ultimate market value of the deposit is not known with accuracy. The Congress devised the concept of permitting either cost or percentage depletion for arriving at the taxes to be paid by the mine operator. This not only avoided the problems of estimating a property value but provided an encouragement for further exploration to replace depleted deposits. The tax schedules and the mineral commodities eligible have been changed over the years, but this still remains a part of the federal tax laws.

The Mining and Minerals Policy Act of 1970 quite clearly expressed the general concern of Congress that the domestic mining industry should not be allowed to atrophy. Its Section 2 is quite specific:

> Sec.2. The Congress declares that it is the continuing policy of the Federal Government in the national interest to foster and encourage private enterprise in (1) the development of economically sound and stable domestic mining, minerals, metal and mineral reclamation industries, (2) the orderly and economic development of domestic mineral resources, reserves, and reclamation of metals and minerals to help assure satisfaction of industrial, security and environmental needs, (3) mining, mineral and metallurgical research, including the use of our natural and reclaimable mineral resources, and (4) the study and development of methods for the disposal, control, and reclamation of mineral waste products, and the reclamation of mined land, so as to lessen any adverse impact of mineral extraction and processing upon the physical environment that may result from mining or mineral activities.

In the 1970s, the world experienced a general period of economic growth that had not been anticipated by the international mining industry. Demand for raw materials exceeded the capability of existing mines and plants to meet it. The resulting shortages, hoarding, and higher prices generated widespread concern that the world was facing persistent shortages. However, new capacity was developed and the growth in demand slowed from its frenetic pace.

However, the United States continued to be concerned that it was not paying sufficient attention to what is needed to maintain its competitive edge in producing materials but also to participate in the development of new materials. The National Materials and Minerals Policy, Research, and Development Act of 1980 was a reflection of this concern. Unlike the Mineral Policy Act, this legislation for the first time had a strong emphasis on research and development. The Act involves all materials used by industry, the military, and individuals, not just those that result from mining per se. It attempts to provide a framework for materials policy decisions by the executive and encourages government support for materials research and development. By the end of the Reagan administration, the effectiveness of the Act had not been demonstrated.

The foregoing presents some of the broader aspects of governmental policy formulation. It also provides illustrations of policies that are broad in character or those specifically designed for the mining industry. But the mining engineer's daily work is probably more influenced by a much larger array of laws and specific governmental activities. These are reviewed in the following discussion.

3.1.4 SPECIFIC AREAS OF GOVERNMENT INVOLVEMENT

In this part, we demonstrate the sweep of government influence on the mining process from exploration to abandonment, and how every facet of engineering, supervision, and management is touched to some extent by government. The subject is far too extensive to be presented in a comprehensive fashion here. But it is important that the engineer be alert to the diversity of laws, procedures, and regulations that are relevant to his tasks. Government may be involved in some fashion in matters as diverse as what information may be requested from a prospective employee to the disposal of used lubricants. This section provides some general guidance as to the many ways in which government may impact mining. But more specific references should be consulted as appropriate for detailed information.

3.1.4.1 Work Force

A century ago, an employer was fairly free to operate his mine, plant, or business as he wished. Today companies, particularly the larger ones, are much more constrained in what they can do and in some cases are directed as to what they must do. Federal regulations, frequently coupled with state requirements, are encountered from hiring to retirement. Some of the more important ones and their major purpose are summarized in the following.

Labor/Management Relations: This aspect of employment was the earliest to receive attention. The Clayton Act of 1914 dealt with strikes and picketing. In 1926, the subject of collective bargaining appeared in the Railway Labor Act of 1926.

As the union movement grew, the Norris-LaGuardia Act of 1932 sought to define and protect what were lawful union activities. The National Labor Relations Act of 1934 established the National Labor Relations Board and dealt with the conduct of labor unions. Union practices and collective bargaining were dealt with further in the Taft-Hartley Act of 1947. The Labor-Management Reporting and Disclosure Act of 1959 established a "Bill of Rights" for union members.

Civil Rights: Explicit concern by government about civil rights first appeared with the passage of the Civil Rights Act of 1964. This was subsequently followed by the Equal Opportunity Act of 1982. In addition to this, there are Presidential Executive orders, such as the one relating to affirmative action. State laws are an integral part of these government efforts to assure that employment is not influenced by religion, race, age, color, or sex. Affirmative action goes one step further in directing that efforts be made to correct for past discrimination in hiring.

Wages and Hours: An early piece of legislation in this category was the Fair Labor Standards Act of 1938 and its subsequent amendments. The minimum wage laws, both state and federal, are constantly being revised to reflect the changing cost of living and the nature of the work force. In addition, statutes define the maximum permissible hours that can be worked and the schedule of compensation for various types of overtime.

Other Compensation: Employees also receive compensation beyond that related to actual hours on the job. Originally, compensation to a worker injured on the job was obtained, as in other injury claims, through reliance on the courts. This is now handled through reliance on state laws that provide for appropriate compensation and associated hearings and appeals.

When an employee loses his job with a firm, the states now provide temporary compensation while he seeks other employment. In support of this, the federal government permits 90% of amounts paid into state unemployment funds to be deducted from their federal unemployment tax obligation.

Income is now provided the individual after retirement from full-time employment. The Social Security System was first established in 1935. It has become the foundation stone for most workers' retirement planning. But its benefits are not restricted solely to retirement due to age but may also may be paid in connection with death or disability at an earlier age.

Pension plans established by companies have grown more numerous and comprehensive over the years. This, however, has prompted considerable attention by government in recent years and some legislation. How pension funds are to be managed, the protections to be offered the worker, the equity held by the employee, and the disposition of retirement funds when a company fails or is merged with another firm are all important issues. These will almost certainly be further explored and defined in the years to come.

Training: During periods of economic stress in general or in particular economic sectors, the federal or state governments frequently attempt to ease the stress of job relocation through offering special training, for example, the Job Training Partnership Act.

Safety: In the general area of industrial safety, the passage of the Occupational Safety and Health Act in 1970 was important. But more specific federal involvement in mine safety has a long history. At one time this was an important facet of the work of the Bureau of Mines. But today, mine safety is handled by a separate agency, and the key document is the Mine Safety and Health Act. This topic is discussed in depth in Chapter 3.3.

3.1.4.2 Economic

The most significant things that the federal government does on the economic front are those related to the economy in general. How broad-based fiscal or monetary decisions can impact mining has been illustrated previously. But the federal and state governments can also take actions that separate out the mining industry for special attention. In the area of taxation and property evaluation, this may be quite important to the profitability or even the survival of a mine. A favorable position on depletion allowances is an economic plus while a new or higher severance tax is negative.

Direct Assistance: Most governments, including the United States, may direct special attention toward "infant" or depressed industries and small firms in general. In its more mature state, the United States has less need to be concerned about "infant" industries. But much of the economic encouragement, or subsidization, that still is evident in mining originated in the last century when the mining sector was in its infancy. In this century, the light metals emerged, and the federal government was very much involved in how aluminum, magnesium, and beryllium evolved into mature industries. The post-World War II expansion of uranium mining and processing was also dominated by the government, but this was more motivated by national security concerns than economic.

When a particular extractive industry gets into difficulty, it has not been unusual for the federal government to step in with assistance, quotas, or tariffs. Domestic oil producers and independent oil refiners have been insulated from the impacts of imports in recent decades. In 1958, the United States became concerned about the viability of domestic lead and zinc mining. Imports of lead and zinc were limited by quotas, and other measures, such as assistance to exploration, were introduced. The program continued until 1965 and then was discontinued as the economy strengthened.

Marketing and Trade Practices: Federal and state governments also concern themselves with the manner in which goods are priced and sold. The initial legislation in this area was the Federal Trade Commission Act of 1914 to prevent unfair or deceptive practices. There are companion state laws with respect to fair trade in 46 states, and much of the burden now falls upon them. Most of the mining industry is not affected on a daily basis by these laws and regulations because products are transferred within firms or sold as intermediate goods to other industries for further processing or manufacture. But some large-volume industries and those that sell directly to final consumers may receive particular attention to their pricing and marketing practices. Steel, cement, and petroleum products are examples.

Monopoly and Antitrust: Closely related to trade practices is the body of law that deals with monopoly conditions in any industry or the concentration of market power among a few firms. The first of these was the Interstate Commerce Act of 1887 to control the railroads. This did not relate to mining itself but was to impact the cost of transporting mined products in the years following. Broader legislation to provide public protection against monopolies and conspiracies, including their dissolution, appeared in the Sherman Antitrust Act of 1890. This was further defined by the Clayton Act of 1914 that dealt with such matters as price discrimination and fixing, concerted action by firms, exclusive contracts, mergers and acquisitions, and interlocking directorates. The Robinson-Patman Act of 1936 also deals with discrimination.

At first, and still to a major extent, the US Justice Department and the Federal Trade Commission are the agencies most concerned about monopolies or other restraints on competition. However, in more recent years, the states have also become active in enforcement.

International Trade: We have noted that the United States, in common with other industrial nations, tries to optimize the use of its indigenous natural resources. In pursuing this goal, industrial nations may institute tax preferences, provide various forms of subsidies that will reduce production costs, and may even introduce trade barriers that lower domestic access to materials of foreign origin to the advantage of domestic producers.

There can be advantages to low-cost raw materials from other nations. Material consumers benefit from receiving what they need at lowest cost. Moreover, no western industrial nation is able to satisfy totally all of its needs. Stockpiling can help meet emergency needs of critical materials, but this does not help in meeting peacetime requirements. So the national interest is also served by seeking to have diverse and competing sources of that which we cannot produce as well ourselves.

However, our viewpoint as an exporter can be quite different. Mining companies seeking to market abroad do not welcome the tax preferences, subsidies, and market barriers used by potential consumer countries to protect their own producers. These are viewed as unfair and discriminatory. Obviously, it is most difficult to interfere in the policy affairs of other nations. But retaliatory acts are taken, and trade wars erupt.

There are no international institutions that can control international matters as directly as we are able to do internally. But an effort to arrive at least some measure of control has been attempted by the General Agreement on Tariffs and Trade (GATT) signed in 1947 and the creation of an International Trade Organization (ITO).

This kind of intergovernmental cooperation must be created from the top down. First, there is establishment of the general objectives, then definition of the kinds of trade restrictions of concern, and finally consideration of the particulars as they relate to individual commodities and circumstances. The need to reach agreement naturally tends to narrow the definitions of what constitutes trade restrictions and what remedies are considered appropriate. GATT is subjected to review at infrequent intervals called "Rounds." It is important to international mineral trade, but it functions primarily through persuasion.

The formation of "common markets" by governments has also become important. These reflect close trading relations among groups of countries and are an attempt to lower barriers and to enhance the collective economic well-being. The European Economic Community (EEC) is the most important of these. In 1992, the EEC has agreed to have its common market begin to function as though it is a single market. That effort by these important materials-consuming nations plus the free trade agreement between Canada and the United States, two major mineral producers, will be events of considerable significance in the future of mineral trade.

3.1.4.3 Regulatory

Many of our laws in effect prohibit certain activities by individuals or firms. But even then, this requires further monitoring and enforcement. Other statutes permit activities but restrict them or control how they can be accomplished. This process of regulation of the activities of industry has become of increasing importance to mining.

The mining industry now recognizes that its activities will be monitored much more closely in the future than in the past. Violations of that which is prohibited by law are likely to incur penalties. But more importantly, regulation has brought with it a whole new framework of seeking official approval or permits for each aspect of the mining process beginning with exploration and persisting to abandonment. The steps involved in this process before federal and state government requirements are satisfied customarily number in the dozens.

Not only are numerous permits or other forms of approval required, but in many cases there are administrative procedures to be followed as work progresses. Performance standards may be specified as to what is done by the firm. Onsite inspection to assure conformance is also a common part of the regulatory process. How pervasive regulation has become is readily apparent in many of the *Handbook* chapters such as those concerning public land operations, health and safety, and environmental control.

An enumeration of what is now involved in environmental protection is informative. The modern era of environmental control began with the passage of the National Environmental Protection Act in 1970. Not only did this impact industry, but it also instituted the requirement that government agencies themselves would have to prepare environmental impacts statements with respect to each of their proposed programs or activities. In the same year, the Environmental Protection Agency (EPA) was formed to coordinate the public control of private actions on a national and regional basis.

The authority for governmental involvement in environmental protection is drawn largely from the following federal statutes:

Clean Air Act
Clean Water Act
Noise Control Act
Solid Waste Disposal Act
Toxic Substances Act
National Resource Conservation and Recovery Act
Comprehensive Environmental Response, Compensation and Liability Act (Superfund)
Endangered Species Act
Federal Lands Policy and Management Act
Antiquities Act
Wilderness Act
Surface Mining Control and Reclamation Act
Coal Leasing Amendments Act

One needs to remain aware that there are also state environmental protection acts of a general nature as well as more specific laws. Chapter 3.4 provides a more detailed discussion of environmental regulations and mining.

3.1.4.4 Government in Support of Industry

In reviewing the various impacts of government on mining, it may appear that they tend to be mostly negative or disruptive in some fashion. While it is generally agreed that government in pursuing its assigned tasks or legal obligations can create problems for individuals and firms, we should also recognize that industry obtains considerable support from government.

A mining firm operating in a developing country or in the frontier area of any country soon recognizes what government does provide elsewhere. This is known as the *infrastructure*—the roads, the transport, the utilities, the security, the education, and the public amenities we take for granted in developed areas. These must be provided by the company itself in isolated regions.

In addition, a developed area has an established commercial sector. Housing, food, services, and other needs are provided by other firms and people. Not only does this remove a burden from the firm, but these profit-making enterprises, plus the public at large, share in the tax burden of paying for the public goods provided by local and national government.

Government may also be capable of providing specialized technical services or problem-solving agencies. Advanced education or vocational training may also be developed by government entities to meet local needs.

Another area of importance to government is its involvement in research of all kinds. This can be either doing the research in-house or the providing of financial support to industry, research organizations, or educational institutions. An important issue in this respect is at what point in the research process is the government most effective, how can it be assured that it achieves a net addition to the research effort, and how does it transfer results to the ultimate user. In some cases, government has even created quasi-corporations to function on an interim basis until private firms can take over a permanent role.

A sometimes overlooked aspect of government support, and perhaps an undervalued one, is the information services that government provides. At the federal level, the Bureau of Mines, the Geological Survey, the Department of Energy, the Bureau of Standards, and the National Aeronautic and Space Administration are all recognized organizations for research activity and financial support. But they are also prolific sources of specialized or general information such as maps, characteristics of materials, new technology reports, or industry statistics, all at little or no direct cost to the mining industry.

3.1.4.5 Land and Water Resources Allocation

Much of the early history of American mining involved the private lands of the eastern part of the United States. The property owner had title to both the surface and the minerals below. An arrangement for the recovery of mineral values was a contract between the property owner and the miner. In addition, the landowner possessed riparian rights to the water flowing through his land. He could use the water, the stream, and the bed in whatever way suited his needs. It has only been in recent years that these freedoms have been somewhat restrained because of concerns about air and water pollution and reclamation of the surface after mining.

As American mining grew and spread westward, and as the older mining regions of the east declined in more recent years, the industry has had to deal more and more with the subsurface ownership of mineral rights and surface ownership by the federal government. Coupled with that, obtaining the rights to scarce water supplies in the west is entirely different than in the east. In the west, water rights are a reflection of use rather than of property ownership. Those who use and become dependent upon water from a stream establish a legal right to that quantity of stream flow in the future. Consumers along the entire water course must respect these rights no matter where they are located. Moreover, the prior possessor of a water right may transfer or sell all or a portion of it to others.

All of industry, including mining and processing operations, is dependent upon water. Thus any new western mining operation must include an adequate water supply in its planning. This means dealing with the established governmental and legal institutions for water allocation found in each state. Generally speaking, tradition has favored agricultural uses of water, municipal needs have ranked next, and industrial requirements have not receive any special advantages. To some extent, this reflects a fear that in an open market for water, industry could outbid other users for water supplies. This does not suggest that mining has been excessively constrained by a lack of water, but it does underscore the fact that the engineer must be prepared to cope with the established systems of water allocation.

In dealing with public lands, the prospector a century ago was offered a preferred position to entice him to explore. Today, as the competition for the use of land has increased, the private owner still retains for the most part the option of selling or leasing to mining interests because of the higher dollar value that may represent. But the variety of land use demands now being made upon the public lands are not necessarily resolved by what is the highest present value per acre. Rather a judgment is made by public land managers as to where the greatest social return may be realized. One criterion might be to maximize the income to government. But the aggregate social return is not always readily measured solely in terms of dollars and may have to reflect qualitative measures as well. In most decisions, there is also in the final analysis significant reliance on the political process. That is, regardless of the motive behind voter preferences, in the opinion of their elected representatives, what do a majority of the public want?

A part of the public land allocation process has been the dedication of blocks of land for particular purposes. An effort is sometimes made to appraise the mineral potential of such lands. But what might be discovered in unexplored areas is difficult to substantiate. However, the more visible value of land as a pristine wilderness, its attraction as a park, or its demonstrable value for agriculture, wildlife preservation, or military purposes give these uses an advantage in these land use decisions.

Even the extraction of minerals on private lands is not totally free of government involvement. The police powers of government extend into matters relating to conflicts in land use. Fortunately for mining, most of its operations tend to be in relatively isolated locations. But when mining is near population centers, or as human settlement encroaches upon mineralized lands, conflict often appears. This means that mining has to contend with such legal issues as what constitutes a public nuisance, zoning decisions which may preclude mining operations, or condemnation of land for public use. Originally, these conflicts tended to arise between adjacent uses of the surface of the land, but now less specific aesthetic and environmental concerns may also enter into the legal actions taken.

3.1.4.6 Socioeconomic Consequences of Mining Operations

Both industry and government try to anticipate and prepare for the social, economic, and environmental effects of the development of new mining and processing operations. The new economic benefits are for the most part welcomed. But there is also the downside when mining operations eventually cease. In addition to the positive and negative economic and environmental consequences encountered during the life of a mine, there can also be considerable social strain placed on a community in many phases. Governments are fearful of the costs to be faced and uncertain of their revenues. Industry, on the other hand, is trying to make a new venture successful and does not welcome unexpected financial burdens or public restraints. Both mine and local government managers must anticipate these public vs. private stresses that accompany mining and plan how to cope with them independently and cooperatively.

Life Profile of a Mine: Each of the four stages in the life of a mine and its closure has different socioeconomic impacts. They are

1. Exploration is first and the least intrusive. It involves only a small work force for limited periods of time.
2. The period of mine development and of plants for associated surface activities (if any) is perhaps the most disruptive event. It lasts only a few years, but at its peak on a monthly or quarterly basis, it can involve the largest number of people. The work force is to a considerable degree transient in character.
3. The period of mine and plant operation is the longest, often two decades or more, and most likely to be a relatively stable period. It usually provides the greatest economic rewards to the community.
4. The final stage is abandonment for any of a variety of reasons. Obviously, this is a negative event on both economic and social counts.

Key Factors in Planning for New Mine Development: The opening of a "greenfields" mine and plant in the United States today is a relatively uncommon event. Mine extensions and plant expansions are more frequent. As a consequence, there is limited recent experience upon which to plan for greenfields operations. However, in the early 1970s, attention was directed toward the vast coal resources of the west.

For the western communities involved, the prospect of new coal mines, and perhaps associated power plants or synthetic fuel plants, was initially welcomed by those eager for economic expansion. Others preferred their lives to remain largely unchanged. Some communities with populations of a few thousand near several new mine and power plant projects were engulfed by thousands of construction workers. Much of the anticipation turned to anxiety about these events.

Now that these events have largely run their course, it is possible to provide some guidelines about what is likely to occur when a new mine or major expansion occurs. The following discussion draws to a considerable extent on unpublished reports prepared for regional clients during those years by Browne, Bortz, and Coddington, a Denver consulting firm.

First, each event is unique and complex, and what will occur cannot be anticipated precisely. The severity of the impact to be experienced is controlled to a large extent by the amount of surrounding development that already exists. A mature, economic region can absorb a new mining operation with little dislocation. But for an isolated community, the consequences can for a time seem overwhelming. If no prior settlement exists, then the entire burden may fall upon the company unless the state or federal government steps in to assist.

For public administrators, with the help of company engineers and managers, it is important to scope the size and timing of the events that are about to happen. The size of the mine, the type of mine, the mineral commodity to be extracted, and the associated surface processing facilities are important variables as to the number and type of workers that will be required, how long they will stay, and the associated needs and services beyond the operation itself. Much of this will seem mundane to company engineers, but city planners may not be aware that it does make a difference if it is a copper open pit, a coal stripping operation, or an underground mine. If there will also be associated surface activity, planning must reflect whether it is a mill, preparation plant, smelter, power plant, or a synthetic fuel plant.

For areas where infrastructure is not fully developed, as in parts of the Rocky Mountain west, the following are offered as useful parameters.

1. For some mining plus surface plant developments, the construction work force may number in the thousands and needs to be estimated by the company(ies). For a mine alone, it will be much smaller. This means a large temporary increase in local employment, with associated demands on housing and all types of public services. The group of workers will change somewhat throughout the period because the type of workers needed shifts as development progresses.

2. Local hiring can be important for many of the less-specialized jobs. This may not reduce the need for inward migration because the jobs they leave need to be filled. However, a previously active area may develop an indigenous work force for activities of this type.

3. Construction workers will tend to live in mobile homes, recreation vehicles, or rental houses, or to commute from other locations.

4. Depending of the size of the community, there may be a need for approximately 0.2 to 0.6 additional service workers plus family to support each construction employee.

5. The permanent operating work force for a mine and plant is much smaller and usually numbers in the hundreds. Again a company estimate to fit the specific situation is needed.

6. Operating personnel intend to stay and will tend to buy or rent permanent housing close to work. But this depends upon the quality of the community and the cost. More will be married, and they tend to have slightly smaller families.

7. Permanent personnel have a greater impact on the economy because they may require from 0.4 to 1.0 service workers and their families, depending upon the services already present.

8. Retail and service establishments must increase but not as much as might be expected. The pressure created by the higher paid jobs at the new installation will tend to raise the local wage level, the cost of housing, and property taxes.

For the local government officials of a small community, the foregoing situation is a difficult one. The construction force will need housing, but only of a certain type and for a limited period. An unknown number may commute. They might overload the roads, the schools, and public and private services. But overreaction and too much expansion is dangerous because these numbers will then diminish rapidly.

Once the operating era begins, the situation changes. Needs will be larger per person and more stable. But it is tempting to overestimate the magnitude of secondary effects on employment and services. The local purchases by the company of services and supplies may be disappointing. Corporate headquarters will likely be elsewhere, and purchasing may be done for more than one location. Some or all of the profits from operations may also flow out of the community. Government revenues will increase from the additional taxes generated by business and a larger population. But in some cases, the boundaries of local jurisdictions separate the public incomes from where public expenditures are needed. Also many expenditures may need to be made early based on anticipated needs and incomes before revenues are actually in hand.

The Magnitude of Change: Fairly sophisticated economic models can be constructed for mature local economies and can approximate the complex interactions that will result from a new mine and plant. For smaller communities, simpler projections are usually sufficient to translate company and community planners raw data into population increases, the multiplier effect on total employment, and the acceleration of incomes resulting from the new dollars spent in the local economy. But in both cases, past experience suggests that the resulting projections tend to be too high and unanticipated events or conditions are the rule rather than the exception.

In retrospect, western communities that were affected by the coal expansion are likely to assess the net economic result as positive. The negative facets, such as coping with higher rents and higher labor costs, were unexpected and the gains were less than hoped for. The immediate turmoil created by the inflow of new workers did subside, and the long-term socioeconomic effects eventually proved manageable.

In addition to the general parameters presented, some quantitative orders of magnitude connected with new mines are useful. Based on actual experience during that period, a typical new Rocky Mountain underground coal mine capable of producing 1.5 million tons (1.4 Mt) annually involved a four-year development period requiring over 500 people. As the beginning of mining approached, the approximately 450 permanent workers would begin to arrive, with the peak employment being reached in the third year when both development and operating people were present.

The company might invest $200 million in the mine and transport facilities. The community then had to find the means to accommodate the mine workers and their families, approximately 1750 individuals. Their presence generated an additional 225 jobs and families. New housing requirements amounted to about 550 units, mostly single-family. School enrollment jumped from 700 to 1100. Retail trade increased l0%, personal incomes 12%, property values 12%, and total county population by 13%.

This is a single mine in a small community. To place it into a larger context, the experience of Gillette and Campbell County in Wyoming is an informative case study.

As of 1981, there were 11 employment centers in Campbell county related to old and relatively new coal mines. These consisted of seven coal mines, two mining company headquarters, a power plant, and the local operations of the Burlington Northern Railroad. Coal mine employment was approximately 2400.

The statistical comparisons below capture how much had happened to Campbell County and Gillette, its major city, in the preceding six years.

	1976	1981
Mining Employment[a]	1,676	4,795
Construction Employment	1,782	3,413
Manufacturing Employment	102	160
Transport and Utility Employment	481	870
Trade Employment	1,597	2,922
Financial Services, etc., Employment	182	379
Services Employment	835	2,671
Public Administration Employment	NA	466
Total Employment	6,655	15,676
Occupied Housing Units	5,810	10,000
Retail Sales (Millions $)	50	148
Bank Deposits (Millions $)	105	256
Property Values (Millions $)	305	1,000
Building Permits	475	807
School Enrollments	3,342	6,492

Source: Browne, Bortz, and Coddington, Inc., Denver, CO
[a]Combined oil and gas, coal, and uranium.

Along with all of these fundamental economic activities of the county, there are many other things related to living that must be accommodated. The number of hospital admissions went from 1418 in 1977 to 2685 in 1980, increasing the occupancy rate from 51 to 83%. Cases filed in county court went from 840 in 1974 to 7008 in 1981. Almost 90% of those in 1981 were noncivil offenses.

Those planning for how the 1980s would affect Campbell County anticipated that growth would continue but at a less frantic pace. But they did not anticipate the dramatic decline in the world oil price in 1986 and the sudden demise of the boom in western energy: an effective demonstration of the fragility of projections. This suggests that all parties at interest, both private and public, must be flexible enough to withstand the unanticipated.

The joint company and community projections and planning that precede new projects must be monitored, at least annually, by tracking the events that follow the mine opening. The projected snapshots of the future become obsolete rapidly. Slowing of expected growth or unfavorable signs of trouble must activate actions to mitigate the effect of these new trends both on the mine and the community. Certainly, any company wants to be optimistic when planning a new project so as to generate community support. However, this does not negate the need to invest in backup plans as insurance against any difficulties if events take a turn for the worse.

Abandonment Phase: Despite its inevitableness, the abandonment of a mine is not a precisely predictable event. Mining may persist in one location for over a century, but it can also end in a matter of a few years. As technology changes, what is economic to produce and what the consumer wants can alter the viability of an operation. Newer mines appear elsewhere, and established mines face increasing costs and the obsolescence of equipment and methods. Even depletion of the minable reserves is uncertain because detailed examination of a deposit is usually made as needed to extend the mine plan.

A mine closing can be expected to be a stressful situation for most of the employees. For those in the community who were dependent upon providing goods and services to the mine and the employee families, there is a similar impact. As is the case in the opening phase, how significant this is for the county, region, and state depends upon the size and maturity of the total economy. A closing of a zinc mine in New Jersey or an iron mine in Pennsylvania is a local event, while a single molybdenum mine is more important to the whole state of Colorado.

Sometimes the decline of a given extractive industry can have major impact on a whole region. The sharp reduction in the mining of anthracite over a period of two decades involved northeastern Pennsylvania's major industry. The economic impact and social disruption this caused was felt for many years. More recently, the decline in iron ore mining in Minnesota, uranium mining in New Mexico, and energy production in several areas has been of concern to entire states.

Because mining is often found in less well-developed locations, an abandonment may leave a population stranded without sufficient economic activity. A century ago, this created the ghost towns that have become tourist attractions today. But in modern times, government takes a more active role in cushioning the impact of the decline or termination of mining and processing. Unemployment compensation, retraining, relocation, and enticements for new businesses to locate locally are all part of this strategy.

Companies today are also more involved than in the past with displaced workers. A mining company is reluctant to close a mine when there are still recoverable resources present. Both the resources and the past investment may be totally lost. Operations may even continue for some time at a loss. But eventually the economic drain and future outlook dictate a closing. Companies may keep employees on the payroll as long as possible, provide severance pay, or offer relocation counselling.

However, this can be a difficult time in government/company/labor relations. The decision may appear abrupt to local government and the workers. They would naturally prefer to see the operation continue. From the company viewpoint, it is reluctant to postulate abandonment of a mine prematurely because of the impact it will have. A closing is usually a last resort and rarely a reversible situation or one that the company wishes to see prolonged. How well management and individual engineers are able to clarify the circumstances that lead to abandonment is important. This may provide a better environment for cooperative efforts to mitigate the economic damage and shorten its duration.

The economic impact that occurs may at least be anticipated and addressed. But the sociological aspects are more subtle and difficult to handle. Some of the skills employed in mining and processing are not readily transferable to other jobs. This means that moving to another job means a reduction in pay. But on the other hand, many workers from open pit or stripping operations can move rather readily to a variety of construction operations. Also, many tasks in mineral processing plants are sufficiently similar to those in the chemical or other manufacturing industries that retraining and relocation are not too difficult.

Experience has shown that many workers once established in a community are reluctant to move for a variety of reasons. This may mean that job opportunities elsewhere are refused, and the family remains where it is. The wife may become the major source of income, and the family resources tend to remain less than they had been. The consequences affect the entire family in many ways, and the community begins to experience the resulting health and social problems and costs this creates. There are no ready solutions to this problem, and they persist for many years.

Technical aspects of mine closures are considered in Chapter 8.7.

3.1.5 IN CONCLUSION: GOVERNMENT AND THE ENGINEER

Mining has traditionally been characterized as a high-risk venture. The mining engineer has been trained to deal with the uncertainties and hazards of mining. No matter how carefully a property has been examined prior to mining, the deposit's geology will not be known in minute detail, and the quality of deposition will vary spatially. Mining conditions will not be uniform, and adjustments will have to be made as operations progress. Efforts are made to factor these variables into the economic feasibility studies made prior to mining. This may be done statistically, through using conservative assumptions, or in other ways. Nonetheless, many mines may successfully extract minerals from the ground but may not be financial successes.

The corporate decision makers are also aware that unpredictable economic conditions external to the mine are an unavoidable business risk. These can also be addressed in a variety of ways. Risk is introduced into the property evaluation prior to investment. The financing used can be fitted to variable income levels, break-even points and cutoff grades can be designed to accommodate possible changes in output, and conservative assumptions can be made about minimum and average price conditions. Larger corporations can seek refuge in the protection of large numbers. As Herbert Hoover noted in his book on mining in 1909, the prudent mine executive recognizes the risk of failure in one mine but would expect to be successful in the aggregate if he/she has ten.

3.1.5.1 Government as a Source of Uncertainty for Mine Management

Despite the capability of the mining executive or the individual engineer to recognize and deal with the risks and hazards that are a part of mining, they may find the uncertainties that originate in the behavior of governments a less familiar situation. As is apparent in the various discussions in this chapter, every aspect of mining is touched by government policies, regulations, or prohibitions. Relations with the labor force can be influenced, new costs may appear, or both foreign and domestic markets may be affected. At one moment, government actions may be supportive of domestic mining, but the next day they may be advantageous to competitors at home or abroad.

Perhaps the most frustrating aspect of government rules and policies is that they do not remain static. A corporate decision tends to be made on the basis of policies and regulations at the time the decision is made. Once done, it may become deeply imbedded in how the mine is operated, the ore or mineral is processed, or to what market output is directed. Yet the policies and regulations may change continuously over the several decades that the mine operates. Adjustment, if not actually impossible, is difficult and costs money. Unfortunately, the resulting objections raised by industry often tend to appear dogmatic and unresponsive to public needs. Although their actions may appear capricious, thoughtful politicians would like to feel that their actions are durable and dependable. But the pressures of social and political change often preclude that being the case.

This situation demands that management as well as engineers involved in planning future operations need to weave government as a variable into their work and their decisions. Initial plans are best drafted employing current laws, regulations, and policy. But these plans should then be reviewed using assessments on how the government posture could possibly change within reasonable bounds over time. Such scenarios can provide rough measures of sensitivity to the effect of future government actions and plans. Projects can then be adjusted to minimize this impact. This should be just as much a part of engineering design and management decisions as the careful examination of technology, costs, and prices. Finally, approval or disapproval of a project at the board of directors level may once again assess these tangible and intangible political elements before a decision is made.

Investment decisions naturally tend to be somewhat influenced by current economic conditions. If prices have been dropping, then pessimism colors profit expectations and extra caution appears. Rising prices will have the opposite effect. This occurs despite the fact that all businessmen know prices behave cyclically and do not continue to rise or fall indefinitely. To a degree, the same is true on the world political scene. Leaderships change, and the policy pendulum swings between relatively conservative and liberal viewpoints. But there are bounds to these swings that are not likely to be exceeded, and reversals should be anticipated.

This does not suggest that persistent trends or basic changes should be ignored. For example, it is not likely that the regulatory presence of the federal government will revert back to what it was decades ago. The environmental controls that have appeared in the last 20 years will likely continue to change and may become more responsive to cost considerations, but they will not disappear. The mineral policies when we were a pioneer nation, or even those of mid-20th century America, will not likely have the same public appeal in the 21st century.

3.1.5.2 The Engineer's Role

Most mining engineers may have once been inclined to view their professional careers as being largely apolitical. There are always a few individuals who opt at some point to become active in political party affairs or to be a spokesperson for the mineral industry's interests before government bodies. But a general sense of political detachment may no longer be wise for engineers. Government is a pervasive force in the affairs of industry, and the daily existence of mining and all of its employees is affected. The engineer must be sensitive to law, regulation, and policy. This is different than suggesting that the engineer must now become a political advocate or activist, but rather that he must be capable of assessing governmental and political behavior in the context of his job.

The individual engineer can also be a participant in the legislative and policy process itself. As a professional, he has competence and experience to offer to the politician and the government official. As a citizen and as an engineer, he should not be reluctant to communicate his views to his elected representatives through personal communication. Members of Congress have attested to the fact that personal communications are effective. It should be noted that this is apart from sending form letters used in campaigns by organized groups or when an engineer is acting as a representative of a company or an industry.

Providing professional advice to legislative assistants as they draft bills can be extremely helpful if the opportunity presents itself. The earlier the technical input by engineers, the better. But it is important that the information should be provided in a form relevant to current legislative or policy purposes. Offering technical and objective testimony in a digestible form at a hearing or in response to a proposed administrative order can also be useful. Participating in studies by scientific or engineering organizations requested by government is important. Those in government are particularly receptive to any timely inputs they feel are not tainted by political objectives. Such information is welcomed because it is rare. Those in government need to be adept in recognizing the motives of all those who are eager to offer guidance. Even then, as public servants, they listen but will react accordingly.

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Note: Books on the general topic of the relationships between government and mining are rare. Engineers interested in keeping abreast of the subject on a contemporary basis, or to find relevant statistics, need to be aware of useful government publications. In addition, certain professional or trade journals publish articles and regular features about the activities of governments and annual commodity Reviews. Some examples follow.

Congressional Record

Federal Register

US Department of Commerce
Statistical Abstract of the United States

US Department of the Interior, Bureau of Mines,
Minerals Yearbook (multiple volumes annually
Mineral Commodities Summary (annually)
Mineral Commodity Summaries (annually)
Minerals Today (bimonthly)

Congressional Quarterly, Washington

Engineering and Mining Journal, Chicago

Energy Policy, London

Mining Engineering, SME, Littleton, CO

Mining Journal, London

Resources Policy, London.

Chapter 3.2
MINING LAW*

CLAYTON J. PARR

Mining law is essentially a branch of the law of real property. It concerns the acquisition of property for extracting contained minerals, and the rights, privileges, and duties that fall upon the holder of the property once the rights are acquired. Accordingly, this chapter focuses on the procedures to be followed in acquiring state, federal, Indian, and private lands, and defines briefly the nature of the rights obtained.

This discussion is not intended to be the basis for legal decisions. Rather, its purpose is to provide the engineer with an understanding of basic principles and procedures. For a detailed treatise on mining law, reference can be made to the *American Law of Mining,* 2nd ed., a six-volume set edited by the Rocky Mountain Mineral Law Foundation.[1] Coverage is extended in this section to the United States and to US law only.

The law changes constantly, especially that based on federal and state statutes, and the reader is cautioned to be alert for subsequent modifications of statutes, regulations, and case law.

3.2.1 PRELIMINARY LAND-STATUS CHECK

After completion of area reconnaissance, or perhaps after receipt of a submittal, a specific and more localized target area will often become of interest. Usually, an intensive exploration program comprising detailed surface mapping, sampling, geophysical prospecting, and drilling will be required before the mineral potential of the ground can be properly evaluated. Before an investment of this magnitude is made, however, it is advisable to obtain rights to mine ore from the property. Otherwise, knowledge of the exploration activity might greatly inflate land values, or speculators or competing mining concerns might step in. The first step in a land acquisition program is a land-status check to determine whether the lands are available and, if so, from whom they can be obtained and by what means. Further discussion of this subject is provided in Chapter 7.1.

3.2.1.1 Bureau of Land Management Records

In those states that contain large areas of federal land, basically the western states, the primary information source is the records of the Bureau of Land Management (BLM). These may be examined at the land office having jurisdiction over federal lands in the area of interest. The records for Montana and North and South Dakota are kept at the Billings, MT, land office; Wyoming, Nebraska, and Kansas, at Cheyenne, WY; New Mexico, Oklahoma, and Texas, at Santa Fe, NM; northern Alaska, at Fairbanks, and southern Alaska, at Anchorage; Utah, at Salt Lake City; Nevada, at Reno; Arizona, at Phoenix; Oregon and Washington, at Portland; and Colorado, at Denver. There are two land offices in California, at Sacramento and Los Angeles. Records for other states are kept in Washington, DC, at the office of the director, Bureau of Land Management. At the land offices, most of the pertinent records are available for perusal by interested parties, but one unfamiliar with the system should seek assistance from land-office personnel.

Essentially, the examiner of the federal land records should ascertain what lands are owned by the federal government and their status. It also is advisable to note how lands no longer owned by the federal government were disposed of and when. Some federal lands have been conveyed out of federal ownership with a reservation of all or some of the minerals to the United States.

The first records that should be examined are the *master title plats* made up for each township that show all lands, minerals, and other rights owned by the federal government, all lands and rights that have been conveyed out of federal ownership, and reservations made by the United States in patents or other conveyances out of federal ownership. Withdrawals, rights-of-way, national forests, Indian reservations, military reservations, and acquired lands also are shown. Those lands affected by a particular action are outlined on the plat by lines of various weight and composition, and abbreviated symbols indicate the status of the parcels.

A second plat, the *use plat,* gives information regarding mineral leases, oil and gas leases, applications, and other current uses.

The master title plat book is accompanied by a *historical index* listing chronologically all past and present actions that affect title to the federal land. The index provides references that can be used to locate copies of the official documents.

The *control document index* contains microfilms of patents and other control documents. It is especially useful in determining if minerals were reserved by the government in a patent.

Other records available in the BLM land office are the *serial register* books, often accessible on computer display screens, that summarize the actions taken in connection with certain entries or classifications of federal land, and *case files,* that contain all correspondence and instruments relating to any particular serial number.

Information regarding unpatented mining claims may be found by checking the geographical index maintained for that purpose, that lists claims by township, range, section, and quarter section. More detailed information can then be obtained by referring to case files in which detailed information on such claims or group of claims is kept.

3.2.1.2 State Records

Often the check of BLM records will show that certain parcels in the area of interest have been conveyed to the state. For this reason, and because lands may have been otherwise acquired by the state, an examination of state land records must be made. They can be found at the office of the agency having jurisdiction over state lands, usually in the capitol city.

Most states have plat books similar to those of the BLM that show lands owned by the state and their status. Like the federal government, the state might have issued patents or other instruments conveying the land into private ownership, possibly with reservations of minerals. The plat books will usually indicate

*This chapter is a revised version of materials in the first edition prepared by Clayton J. Parr and Northcutt Ely. Grateful acknowledgment is given to Robert G. Holt and Daniel A. Jensen for valuable assistance in preparing these materials.

[1] References in this chapter are listed numerically.

whether mineral leases have been issued covering properties of interest. Inquiry should be made of state land-office personnel to learn how such information can be ascertained. It also should be determined whether any special-use permits or leases, such as grazing leases, affecting the surface are in effect.

3.2.1.3 County Records

Information regarding title to privately owned lands must be obtained from the records of the county in which the land is situated. To obtain a quick indication of apparent ownership, the records of the county treasurer or assessor, which show to whom ad valorem property taxes are being assessed, can be examined. County assessors keep abreast of title transfers and generally will be billing the current owner for the taxes. Even if the records do not reflect current ownership, however, discussion with those whose names are given generally will yield up-to-date information.

If time permits, a more accurate indication of ownership can be obtained from the county recorder's files. The recorder must keep an index of all instruments recorded. Usually the index is by name of the grantor and grantee. Except in sparsely populated counties, this index is not very useful in obtaining a rapid indication of land ownership in a large target area. Many states also require that a tract index be kept, which is keyed by legal subdivisions. Examination of the tract index will show chronologically all recorded transactions affecting title to a particular piece of property, and should reveal the current record titleholder and the existence of mineral interests. Some county recorders keep indexes of mining claims by area. These are very helpful, but are not to be relied upon to show all claims. County records will also reveal judgments or liens, overriding royalty conveyances, and other encumbrances.

3.2.1.4 Surface Check

The final step, but certainly not least in importance, is a field examination. If the land is public domain open to mining location, the ground must be examined to determine whether any claims have been located thereon. BLM and county indices are usually not current and they are often not complete or accurate. Also, notices recorded with the BLM or with the county recorder might not be discovered during a record search or, even if discovered, might contain inadequate or incorrect descriptions.

If claim markers are observed, the notices should be examined and the contents carefully noted. The field examiner also should check the claim markers to determine whether the boundaries are adequately marked and to plot the area covered by the claims. Any excavations, prospects, drillholes, or other evidence of geological or mining activities should be carefully examined and described. Any evidence of assessment work should be noted, as should the contents of any assessment affidavits in view.

If claims are found to have been made, it will be necessary to check county and BLM records to see if affidavits of annual assessment work have been filed. If within the time permitted by state law, no notice of performance of annual assessment work for the year expiring noon the last preceding Sept. 1 has been recorded, or if a copy of the recorded document has not been filed in the state office of the BLM by Dec. 30 of that same year, the lands might be open to location. All data acquired from the examination should be reviewed by an attorney to determine whether the claimant has complied with applicable laws regarding the location and preservation of the claims.

The field examiner also should make note of any other uses being made of the surface, such as industrial, residential, or agricultural.

3.2.2 FEDERAL LANDS

Federal lands can usually be placed in one of several categories depending on how the lands have been classified. These classifications are important in determining whether mining rights can be obtained, and if so, by what procedure. The basic categories are defined in the *American Law of Mining,* 2nd ed.:[2]

> "Public lands" means any land and interest in land owned by the United States within the several states and administered by the Secretary of the Interior through the Bureau of Land Management, without regard to how the United States acquired ownership, except: lands located on the Outer Continental Shelf, and lands held for the benefit of Indians, Aleuts, and Eskimos. "Public lands" includes "public domain lands" and "acquired lands.". . . . Generally, public lands do not include submerged lands.
>
> "Public domain," within the meaning of this title, denotes those lands which are or were subject to the public land laws of the United States. It includes lands initially acquired by the United States by cession, purchase, and treaty, as well as lands acquired by other methods where the latter have expressly been declared by Congress to be public lands or public domain.
>
> "Public domain lands" are defined by the US Department of the Interior to be "original public domain lands which have never left Federal ownership; also lands in federal ownership which were obtained by the Government in exchange for public domain lands or for timber on public domain lands.
>
> "Acquired lands" are defined by the Department of the Interior as "lands in Federal ownership which were obtained by the Government through purchase, condemnation, or gift, or by exchange for such purchased, condemned, or donated lands, or for timber on such lands. They are one category of public lands." Notwithstanding the characterization of acquired lands as a component of public lands by this definition, the mineral developer or prospector will find that the public land laws are generally inapplicable to acquired lands.
>
> The Department of the Interior defines "reserved lands" as "federal lands which are dedicated or set aside for a specific public purpose or program and which are, therefore, generally not subject to disposition under the operation of all the public land laws." Reservations are frequently mistaken for withdrawals because reserved lands are usually withdrawn.
>
> A withdrawal segregates particular lands from the operation of specified public land laws, making those laws inapplicable to those lands. Lands may be withdrawn from all or any part of the public land laws, including the mineral location and mineral leasing laws. Withdrawal is a frequently used technique of the government when its purpose is to preclude disposition or a particular use of a given tract of land. The breadth of the statutory definition, especially its use of the term "reserving," creates a temptation to equate withdrawals with reservations; however, the traditional distinction discussed in the text continues to be honored.

Severed Lands: The federal government has conveyed the surfaces of many lands, but has retained rights to all or certain named minerals. This is known as a severance of the mineral and surface estates, and such lands can be described as severed lands. Procedures for acquisition of the minerals are dependent upon the type of disposition that was made.

3.2.2.1 Mining Law of 1872

The Mining Law of 1872 is an embodiment of custom established by early western miners. The law is based fundamentally on two principles: (1) one can acquire mineral rights to land by making a discovery of valuable minerals thereon, and (2) a person who has made such an acquisition must continue to develop the minerals on the land to retain possession. Under this law, land is "claimed" for minerals by marking boundaries on the surface. Thus a claim is "located." The Mining Law of 1872 and subsequent modifications thereto often are commonly referred to as "the general mining laws."[3]

Lands Subject to Location: Federal lands that are locatable are unappropriated public domain and withdrawn or segregated lands that have by special provision not been segregated from application of the mining laws. Location can be made in Alaska, Arizona, Arkansas (except lead), California, Colorado, Florida, Idaho, Louisiana, Mississippi, Montana, Nebraska, Nevada, New Mexico, North Dakota, Oregon, South Dakota, Utah, Washington, and Wyoming.[4] Lands in national forests in these states are locatable, but not lands in national parks and national monuments, except where specifically authorized by law. Lands in Indian reservations, military reservations, and acquired lands, are not locatable.

Qualifications of Locators: Mining locations can be made by citizens of the United States and aliens who have declared their intention to become citizens.[5] Corporations organized under the laws of the United States or any state are considered citizens without regard to the nationality of shareholders and can locate and hold mining claims if empowered to do so under state law. By special statutory provision, certain employees of the Bureau of Land Management are disqualified,[6] but the Secretary of the Interior has, by administrative action, also excluded all full and part-time employees of the department, except members of advisory boards and councils.[7]

The proscription against location of claims by aliens is enforceable only at the instance of the United States. Hence, as a general rule, the fact that a mining claim is owned in whole or in part by an alien cannot be used by a private party as the basis for acquiring a dominant interest or for challenging validity.[8] Defective title of an alien is cured upon transfer to a citizen or upon a declaration of intent by the alien to become a citizen. The required declaration of intent usually is manifested by the filing of an application for citizenship.

Minerals Obtainable Under Location Law: The Mining Law of 1872 provides that "all valuable mineral deposits" in lands belonging to the United States shall be free and open to exploration and purchase.[9] There are two questions in determining what constitutes a "valuable mineral deposit" subject to appropriation under the general mining laws. First, is the particular mineral locatable and, second, is the mineral deposit valuable?

Theoretically, a particular *kind* of mineral is valuable and hence locatable if there is a market for it. A *valuable mineral* also has been defined as "any substance found in the earth having sufficient value to warrant the cost of its extraction."[10] Because of the unrestricted definition, the Location Law came to be applied to deposits of common nonmetallics, such as sand and gravel.[11] By special enactment at a later time, building stone and saline deposits were made locatable.[12]

The number of locatable minerals has been substantially reduced, however, by the Mineral Leasing Act of 1920 and the Common Varieties Act of 1955. The 1920 Act provides that oil, gas, coal, potassium, sodium, phosphate, oil shale, native asphalt, gilsonite, solid and semisolid bitumen, bituminous rocks, and deposits of sulfur in Louisiana and New Mexico can be acquired only by lease.[13] The 1955 Act foreclosed future locating of common varieties of sand, stone, gravel, pumice, pumicite, and cinders[14] ("stone" has been interpreted to encompass building stone and limestone). Excluded from the Act, however, are deposits of minerals with "some property" giving them a "distinct and special value." These are to remain locatable.

There is much uncertainty as to which characteristics are sufficient to give a deposit distinct and special value, and the issue has been the subject of many adjudications by the Department of the Interior (DOI). One writer summarizes by saying that "locatable common substances must have 'an exceptional nature' or be so unusual, exceptional or unique as to add to its value for uses over and above normal uses of the general run of such deposits."[15] Characteristics such as unique fracturing features, superior cleavability, unusual purity, unique use, and high price might be sufficient in a particular case, but as a practical matter, there is no way to be sure. As a result, substantial risk of loss of investment and possible liability for trespass exists whenever deposits of such materials are claimed and mining subsequently is undertaken.

Prediscovery Right—Pedis Possessio: The task of modern-day exploration geologists is complex. New mineral deposits most often are found far beneath the surface, and actual penetration of mineralized zones may occur only after long, multiphase exploration programs. There can be no appropriation under the mining laws until a discovery of valuable minerals has been made, but in most cases the prospector will find indications of mineralization, such as geophysical anomalies, long before the deposit can be proved out. It has long been recognized that a person actively exploring an area should have some protection against preemption by others of the land that he/she is exploring until he has had the necessary time to make a discovery of minerals in place. The doctrine of *pedis possessio* was formulated to provide such protection.

Under the doctrine, a qualified person who peaceably and in good faith enters vacant unappropriated public domain in search of valuable minerals may exclusively hold the place where he is working against others having no better right, as long as he remains in continuous exclusive occupancy and diligently and in good faith prosecutes work toward making a discovery.[16]

Because of *pedis possessio* rights, it is common for prospectors to set up claim monuments before making a discovery. It must be emphasized, however, that claims located before discoveries are made are simply convenient delineations of areas in which the locator claims the exclusive right of exploration. Each must be supported by a discovery of minerals in place before it becomes legally perfected under the mining laws.

Whether a prospector can successfully assert rights of *pedis possessio* will depend on the facts of the particular case, but in every case there must be (1) actual physical occupancy of the ground, (2) diligent bona fide work directed toward making a discovery, and (3) exclusion of others.[17] Although the requirements can be stated quite simply, it is difficult to prescribe with certainty what actions must be taken in the field to satisfy them.

It is common practice to locate a large block of claims over and around an area where deposits of valuable minerals are suspected. Locators of such blocks have attempted to maintain exclusive possession on the theory that work being done on some of the claims constitutes constructive possession of the remainder. This theory has met with little success, however, and as a result, little security is had under *pedis possessio* in the absence of actual physical possession of each claim.[18]

The occupancy must be more than mere presence. The occupant must be actively working toward making a discovery. Mere preparations for exploratory work may not be sufficient unless the activity directly precedes actual drilling or digging. The sufficiency of radiometric or geophysical testing is doubtful.[19]

It is difficult to state any hard and fast rule as to what conduct will be sufficient to establish that work is being directed toward making a discovery. If, however, the claimant is acting in good faith and is utilizing the most appropriate exploration method under the circumstances, his chances of overcoming a challenge by a third party should be good. Actual penetration of the earth in an attempt to uncover minerals in place provides the most security. It is most important, however, that exploration work be taking place on each claim.

The requirement of physical occupancy of the ground is usually satisfied as a result of the work in progress. Exclusion of others, however, requires positive action, because *pedis possessio* rights are lost if an adverse claimant is permitted to enter the property peacefully. *Pedis possessio* protects against "forcible" entry. Thus it is necessary that the intruding party be denied entry. If a confrontation occurs, and the party being denied entry resorts to force to gain such entry, he need not be successfully excluded. One simply yields to force, and then resorts to his legal remedy. No statements should be made, however, that indicate acquiescence.

It has been suggested that denial of access may be made by signs, written or verbal notice, or other reasonable means.[20] If no other means are available, such methods should be used for whatever protection they might offer. If it is possible, however, especially in a land-rush situation, the area should be patrolled and entry denied all trespassers. If problems are anticipated, legal counsel should be engaged beforehand so that necessary preparations can be made for obtaining a restraining order.

Pedis possessio provides only tenuous prediscovery protection. It is unclear in a given situation what conduct will satisfy the requirements and, as a result, much litigation has resulted.

Discovery: Even though the particular mineral of interest is subject to acquisition under the mining laws, it also is required that there be a valuable deposit of the mineral within the land desired to be located. That is, the geologic setting must be such that the deposit is of sufficient grade, quality, and amenability to render the ground valuable because of its mineral content. If so, there has been a "discovery" in the legal sense, and a valid mining claim can be located.

A discovery is essential to the acquisition of rights under the federal mining laws.[21] A temporary, but highly tenuous, right to exclusive possession can be obtained through *pedis possessio,* but there must be a discovery of valuable minerals before the locator will have perfected a legally valid mining claim.

The definition of "discovery" under the mining law has been and continues to be a subject of controversy. The basic legal standard for discovery is the so-called prudent-man test, which reads:

> (W)here minerals have been found and the evidence is of such a character that a person of ordinary prudence would be justified in the further expenditure of his labor and means, with a reasonable prospect of success in developing a valuable mine, the requirements of the statute have been met.[22]

Hence the test is not whether the individual claimant feels he is justified in further expenditure of labor and means, but whether the hypothetical reasonable man would be so justified.

Imperfect as the objective prudent-man test is, it became the universally accepted standard of determining discovery. In 1933, however, the DOI formulated a modified standard applicable to discovery determination by the United States, the so-called "marketability test:"

> (T)he mineral locator or applicant, to justify his possession, must show that by reason of accessibility, *bona fides* in development, proximity to market, existence of present demand, and other factors, the deposit is of such value that it can be mined, removed, and disposed of at a profit.[23]

Thus the marketability test focuses on the economic value of the deposit as a whole. In *Coleman v. US,*[24] which involved a deposit of building stone, the court upheld the marketability test on the ground that it is merely a facet of the prudent-man test.

The stringency of application of the discovery test varies according to the identity of the adversaries and the nature of the controversy. The standard is applied with much more strictness, for example, when the existence of a discovery is challenged by the BLM in connection with a patent application than when lack of discovery is alleged by a rival locator. A critical fundamental difference is that application of the marketability test per se is limited to "controversies between a mining claimant and the United States or between a mining claimant and someone other than a mining claimant, claiming under the United States."[25] Thus courts can reach different conclusions based on the same set of facts. The BLM has listed six categories in the order of decreasing discovery requirements: [26]

> A. Building stone and saline placer, where it has to be shown that the land is chiefly valuable for building stone or salines.
>
> B. Mineral patent applications.
>
> C. Claims, located on deposits of commonplace materials where the existence of an actual market has to be shown which were located prior to July 23, 1955 . . .
>
> D. Claims in conflict with nonmineral (agriculture) entries or located in areas where condemnation proceedings may be required.
>
> E. Mining claims located on lands involved in multiple use procedures under Public Laws 167, 357, 359 and 585.
>
> F. Disputes between rival mining claimants.

Even though the scope of applicability of the test is uncertain, as a result of these decisions, every locator should be prepared to defend his discovery under the standards imposed by the marketability test. Hence, the locator should begin building his case at the time he exposes minerals and should continue to do so for as long as the claim is held. He should gather and retain as much data as he can about the character and extent of the deposit and its economic value. Because of the variability of the test and the difficulty in applying the accepted definition to field conditions, one can rarely be confident that he has a discovery unless he is actually mining the deposit at a profit.

It is well established that discovery contemplates an actual physical exposure of mineral-bearing rock in place within the limits of a claim.[27] Discovery of float rock or other surface indications will not suffice. Geophysical and geologic data sufficient to establish a strong inference of mineralization are not sufficient standing alone. In several cases, however, it has been held that, as against a rival locator, radiometric readings plus corroboration based on sampling and assays of nearby exposures and a strong geologic inference of projection of the sampled zone, will be sufficient.[28] In *Ranchers Exploration & Dev. Co. v.*

Anaconda Co.,[29] the court held that discovery of ore in place on one claim can be sufficient to support a discovery on an adjacent claim if projection of the ore-bearing zone across the boundaries can be demonstrated. In determining this, the court weighed evidence of geologic inference based on the opinions of miners, geologists, and other experts. As against the United States, geologic inference alone is not enough to demonstrate a discovery, but it may help establish the further elements of a discovery with regard to an exposure of rock in place.[30]

There is an oft-repeated statement in mining contests brought by the government that information sufficient to justify additional exploration is not enough to show a reasonable probability of developing a paying mine. This was emphasized in the decision in *Converse v. Udall,*[31] where the court stated, "(T)he finding of some mineral, or even of a vein or lode, is not enough to constitute discovery—their extent and value are also to be considered."

From the standpoint of the field engineer, the objective should not be so much to understand the intricacies of the law relating to discovery, but more to appreciate the necessity of fortifying the claim by amassing as much information as possible, both geologic and economic, tending to show that the deposit is valuable. The existence of a discovery always is subject to challenge until a patent issues, and when a challenge is made, the success of a lawyer's defense of the claim will be dependent on the facts with which he or she has to work.

Choice Between Lode and Placer: The location law authorizes two types of claims, lode and placer, depending upon the character of the deposit. It requires that lode claims be made "upon veins or lodes of quartz or other rock in place bearing gold, silver, cinnabar, lead, tin, copper, or other valuable deposits."[32] However, "all forms of deposit, excepting veins of quartz, or other rock in place" must be located as placers.[33] Hence the locator must decide into which category his deposit falls before making his location.

Congress apparently was drawing a distinction between the traditional gold placer in unconsolidated alluvial material along stream beds, as opposed to the common vein, or lode, found in solid rock. Many deposits, however, do not clearly fall into either category, and consequently, the choice can be difficult at times.

The most frequently quoted definition of a lode is given in *Eureka Consol. Mining Co. v. Richmond Mining Co.,*[34] where the court stated: "The term lode, as used in the Acts of Congress, is applicable to any zone or belt of mineralized rock lying within the boundaries clearly separating it from the neighboring rock." Placers have been defined as "merely superficial deposits, occupying the beds of ancient rivers or valleys, washed down from some vein or lode,"[35] and as "deposit(s) of valuable minerals found in particles in alluvium or diluvium, on beds of streams."[36]

On the basis of these and similar definitions, which emphasize the form of the deposit rather than its mode of origin, it has been held that blanket-type deposits of nonmetallic minerals, such as phosphate and distinct stratigraphic units of intermittently mineralized limestone can be lodes.[37] Deposits of uranium contained in a horizontal sandstone have also been characterized as lodes.[38] It has been held by some courts, however, that badly distorted near-surface blanket deposits of minerals in place are placers.[39] Generally speaking, certain recurring factors are considered by the courts. Thus, if the rock is in place and bounded on either side by other rock in place, it is likely to be a lode. If the ore is on top of the ground, has broad lateral continuity, is weathered and has no cover except perhaps for a thin veneer of soil, it is likely to be a placer. Although in many instances the choice is difficult, some comfort can be derived from the knowledge that the courts have exhibited an inclination to uphold the choice of the first locator in the event of a dispute.[40]

A placer location does not establish rights to lodes within its boundaries. Hence, up to the time of patent, lodes within an unpatented placer claim are subject to appropriation either by the placer claimant or third parties. It is generally accepted that the locator of a placer claim may cover the same ground with lode claims without invalidating his prior claim. Most often the junior lode claims are located full size, but there is a mix of opinion as to whether lode claims located over valid placers are limited to a width of no greater than 50 ft (15.2 m) plus the width of the vein.[41] A lode location made by a third party over a valid placer claim without consent of the placer owner, however, may be invalid by reason of its being located in trespass.[42]

An included lode can be acquired at the time of patent, but a known lode that is not so acquired remains subject to separate location.[43] If no reference to an included lode is made in the patent, however, a strong presumption exists that the placer does not contain a known lode, and a subsequent locator must show by clear and convincing evidence that a valuable lode deposit was known to exist when the application for patent was made.[44]

Although the customary sequence is location first of placer claims and then location of lodes, there is authority supporting the proposition that placer claims can be located over prior valid lode claims, or over known lodes.[45] Such location would presumably be subject to the same consent requirements described previously if the junior location were made by a third party. A patent of the prior lode claim, however, would vitiate the placer claim because the lode claim will include all placers upon issuance of a patent.

Placer claims located over patented lode claims are invalid.

Size and Shape of Mining Claims: A lode location may not exceed 1500 ft (457.2 m) in length along the vein or lode (sideline), nor more than 300 ft (91.4 m) on each side of the vein (endline). A smaller claim may be located if desired. The endlines should be parallel to each other,[46] although they do not have to be perpendicular to the sidelines. It has been held, however, that nonparallel endlines do not invalidate a claim, although they do deprive the locator of extralateral rights if the endlines diverge in the direction of the dip of the vein.[47] The claims can overlap previously located claims as long as the discovery is made on unappropriated land. However, only the unappropriated ground is held over a senior locator whose claim is overlapped.

A placer claim can be no larger than 20 acres (8 ha) for an individual, but associations, or groups of persons, can locate up to 160 acres (64 ha) in a single claim, based on 20 acres (8 ha) per individual in the association.[48] A corporation is considered to be a single person. On surveyed land, placer locations must conform as nearly as practicable to rectangular legal subdivisions.[49] Legal subdivisions of 40 acres (16 ha) may not be subdivided smaller than into l0-acre (4-ha) tracts. If the legal subdivision contains more than 20 acres (8 ha) per claimant because of discrepancies in the public land survey, up to 5 acres (2 ha) of excess are permitted.[50]

Care must be taken to ensure that all individuals in an association have a true ownership interest in the venture, because the interest held by a so-called "dummy locator" can be invalidated.[51]

Location Procedures: The federal law requires only that there be a discovery of a vein or lode within the limits of a lode claim, and that the claim be marked on the ground so that its boundaries can be readily traced.[52] Other requirements for location are to be established by regulations made by the miners of each mining district that are not in conflict with the laws of the United States or of the state or territory in which the mining district is situated.[53] State governments have supplanted local mining districts with regard to the enactment of location requirements.[54] Hence the laws of each state must be examined to determine requirements concerning the manner of posting and recording notices of location and the manner of marking the boundaries of the claim.

State law generally requires that there be (1) posting of a location notice, (2) marking of the boundaries of the claim, (3) performance of the required discovery work, and (4) recording of the location notices or a location certificate. Once the location notice is posted, the remaining order in which these steps are performed is not important as long as they are all performed within the time permitted.[55]

The federal mining laws do not require posting of a notice of location, but the Secretary of the Interior has, by regulation, provided posting procedures and has described the contents of the notice and the place and manner of posting.[56] This regulation suggests that the discovery notice ties in with some permanent landmark, such as stone monuments, blazed trees, etc.; the regulation also states:

> (T)he claimant should state the name of adjoining claims or, if none adjoin, the relative positions of the nearest claims, should drive a post or erect a monument of stones at each corner of his surface ground, and at the point of discovery or discovery shafts should fix a post, stake or board, upon which should be designated the name of the lode, the name or names of the locators, the number of feet claimed, and in which direction from the point of discovery, it being essential that the location notice filed for record, in addition to the foregoing description, should state whether the entire claim of 1500 ft is taken on one side of the point of discovery, or whether it is partly upon one and partly upon the other side thereof, and in the latter case, how many feet are claimed upon each side of such discovery point.

Under the laws of some states, the notice posted on the claim must simply contain the name of the claim, the name of the locator, and the date of discovery. More detailed information may be required on a recorded location certificate. In other states, the recorded notice is a copy of the posted location notice, and both must contain detailed information regarding the location.[57] Once the location notice is posted, the land is withdrawn from location by others as long as the locator performs the required acts of location within the prescribed time and has a valid discovery.

The boundaries of a claim are established by markings on the ground, and these markings control over descriptions in the location notices or deeds. Federal law requires that a location be distinctly marked on the ground so that its boundaries can be readily traced.[58] In each state, the minimum requirements for marking the boundaries are set out by statute. These statutes should be checked to determine what type of monument must be posted and how extensively the boundaries must be marked. Usually, as a minimum, a monument must be placed at each corner of the claim. Once a discovery is made and the location monument is erected, most states allow a set period of time for completion of location requirements. The notice of location, or a separate certificate of location, usually must be recorded within the same period of time.

Discovery work differs from discovery of valuable minerals. The former is a diligence requirement having as its purpose the discouraging of non bona fide locators by requiring a minimum amount of work on the claim. Performance of the work does not constitute a discovery of valuable minerals.[59] With the exception of Washington, which requires discovery work in the form of labor to develop placer claims, all states have now eliminated the requirement that physical discovery work be performed, although some allow it as optional and others require the filing of maps in lieu of physical work.[60]

Requirements imposed by state law usually are very detailed and sometimes seemingly without rationale, but nevertheless, they should be complied with as nearly as possible. It must be kept in mind that any procedural defect can be seized upon by a competing locator as a means of obtaining priority.

State laws requiring recordation of notices or certificates of location must be complied with. In addition, a copy of the notice or certificate of location, in the exact form of that recorded if the state requires recordation and otherwise in form required by BLM regulations, must be filed with the appropriate state BLM office within 90 days after the date of location.[61] Failure to make the BLM filing causes the claim to be void.[62]

Once the boundaries are marked and the location monuments erected, the claimant need not maintain or restore the monuments, if they are obliterated without his fault, to preserve his possession as against rival claimants.[63] If the monuments are obliterated after patent issues, the patent description controls. Prior to patenting, the location markers should be maintained because the boundaries of the claim are determined by the markings on the ground.

Amendment: If a location notice contains defects, a corrective amended notice can be filed without making changes on the ground.[64] If the notice is so defective that it is void, an amendment will not defeat the priority of an intervening locator (junior locator). Boundary changes can be made to include new unappropriated ground by filing an amended location notice and changing the boundary markers. Prior to the posting of location markers, a locator can swing his claim around the point of discovery to better orient the claim along the vein.

Rights of the Locator: The owner of a valid mining claim has the right of exclusive possession of the claim as against other locations as long as he complies with the acts of Congress and local statutes. Claims are real property and cannot be taken away by the United States without just compensation.[65] Except in situations where a purely legal issue is involved, such as where required filings have not been made, claims cannot be declared invalid without a hearing.[66] Actual fee title to a claim remains in the United States until patent issues.

The owner of an unpatented lode claim is entitled to all veins or lodes within the limits of his claim, not subject to extralateral rights of another, and he can proceed to mine them for his own exclusive use and benefit. He can use the surface as required for his mining operations. The owner of a placer claim has similar rights to any placers within the boundaries of his claim.

Extralateral Rights: The locator of a valid lode mining claim acquires the right to "all veins, lodes and ledges throughout the entire depth the top or apex of which lies inside of such surface lines extended downward vertically, although such veins, lodes or ledges may so far depart from a perpendicular in their courses downward as to extend outside the vertical sidelines of such locations.[67] The apex of a vein is its top (or the highest point), the terminal edge from which the vein extends downward in the direction of the dip.[68] The rights of a claimant to follow a vein or lode beyond the boundaries of the claim are known as extralatera1 rights. Such rights are confined to the area between two hypothetical vertical planes that extend through the parallel endlines.

Extralateral rights are acquired only if the location is a lode claim. The vein or lode must be rock in place, and it must be essentially continuous in its downward course. Some offset along faults or thinning of ore zones is permissible, but if the vein disappears or loses its identity so that it can no longer be traced, the rights cease. If the deposit is horizontal, no extralateral rights are acquired.

An entire stratigraphic unit, such as a limestone bed, can be a lode to which extralateral rights will attach if it is mineralized throughout. Such a unit is referred to as a “broad lode.”[69]

Ideally, a lode claim would be located along a vein outcropping at the surface in nearly a straight line. In most instances, however, the course of a vein is difficult to define on the surface and, as a result, the locator often does not, or cannot, orient his claim so that the apex intersects both endlines. By judicial decision, however, the rule has been adapted to cover a variety of situations.

If the apex of a vein passes through both sidelines of a location, and the sidelines are parallel to each other, the sidelines become endlines for the purpose of determining extralateral rights to the vein.[70]

No extralateral rights are acquired if the endlines diverge in the direction of the dip of the vein. Some courts have held that extralateral rights are acquired if the endlines converge, but only within the converging endlines to the point of intersection.[71]

If the apex of the vein crosses one side and one endline, the extralateral rights are bounded by a plane extending perpendicularly downward through the endline through which the vein passes, and a plane parallel therewith dropped downward perpendicularly through the point where the vein crosses the sideline.[72]

If the vein enters and leaves the claim through the same boundary line, it has been held that extralateral rights are obtained within vertical planes drawn parallel to the endlines at the points where the vein crosses the sideline.[73]

If the vein crosses only one boundary line, or is contained wholly within the boundaries of a claim, extralateral rights apparently will be bounded by vertical planes parallel to the endlines drawn at the point or points where the vein terminates.[74]

If the apex of the vein on its course is split or divided by a boundary line of a location, or if the vein is wider than the location itself, the senior locator is entitled to extralateral rights to the entire vein.[75]

Extralateral rights are not restricted merely to the discovery vein; the locator also acquires extralateral rights to any hidden or secondary veins, the apexes of which lie within the claim’s boundaries. It is the discovery vein, however, that determines which lines are endlines and which are sidelines. Otherwise, the extent of extralateral rights to the secondary vein are determined in the same manner as are those extending to the discovery vein.[76]

The mining law provides that “where two or more veins intersect or cross each other, priority of title shall govern, and such prior location shall be entitled to all ore or mineral contained within the space of intersection; but the subsequent location shall have the right-of-way through the space of intersection for the purpose of the convenient working of the mine. . . .”[77] If two veins unite, the prior locator obtains rights to the vein below the point of union and to the space of intersection.

At one time, lawsuits concerning extralateral rights were common, but now they occur rarely. Such litigation generally was bitter, prolonged, and exceedingly costly. A good practice is to enter into vertical sideline agreements with adjoining claim owners limiting each party to the mining of ore directly beneath the surface of his claims.

Tunnel Sites: The location law provides for acquisition of minerals encountered in a tunnel run for the development of the vein or lode for the discovery of mines.[78] The owners of such a tunnel have the rights of possession of all veins or lodes within 3000 ft (914.4 m) from the face of the tunnel on the line thereof not previously known to exist. The rights acquired are the same as if the vein had been discovered from the surface, so the tunnel claimant’s right to a vein is limited by the maximum dimensions of a lode claim: 1500 ft (457.2 m) along its length and 300 ft (91.4 m) on each side of the vein. If a vein is encountered, the tunnel claimant can delineate the boundaries of his rights by locating on the surface.[79]

In effect, a tunnel locator segregates a 3000-ft (914.4-m) square measured from the face of his tunnel for so long as he is diligently working on the tunnel. His rights are not dependent upon an initial discovery, and they have priority over surface locations made by others after commencement of the tunnel. Failure to prosecute the work on the tunnel for six months constitutes an abandonment of the tunnel locator’s rights.

Procedures for the initiation of the tunnel rights are defined in regulations promulgated by the Secretary of the Interior.[80] A monument must be placed at the face of the tunnel upon which a notice must be posted giving the names of the parties or company claiming the tunnel right, the actual proposed course or direction of the tunnel, the height and width thereof, and the course and distance from such face or point of commencement to some permanent well-known object in the vicinity. The tunnel claimant must establish the boundary lines of the tunnel by stakes or monuments placed along its lines for a length of 3000 ft (914.4 m) from the face. In addition, a copy of the notice must be filed with the recorder for the district in which the claim is located and, to this, must be attached a sworn statement stating the amount expended by the tunnel claimants, the extent of work performed, and the bona fide intent to prosecute work for the development of the vein or lode or for the discovery of mines.

Tunnel sites are rare.

Mill Sites: The mining law permits acquisition of land for nonmining purposes in plots up to 5 acres (2 ha) that are referred to generally as mill sites. Three type of mill sites are recognized: (1) nonmineral land not contiguous to a vein or lode can be occupied by the claimant for mining or milling purposes, and the mill site can be patented with the lode claim; (2) nonmineral land not contiguous to a vein or lode claim can be appropriated for a mill site by the owner of a quartz mill or reduction works, and separate patent can be obtained; and (3) nonmineral land can be acquired by the proprietor of a placer mining claim for mining, milling, processing, beneficiation, or other operations in connection with the placer claim, and patent can be obtained at the time patent is acquired for the placer claim.[81]

By custom, mill sites are staked in the manner provided for lode claims. The mill site does not have to be any required shape, as long as its area does not exceed 5 acres (2 ha). It may consist of one or several pieces of ground.[82]

Land obtained for a mill site must be nonmineral in character up to the time of patent.[83]

A mill site must be used for mining and milling purposes. The statute has been interpreted broadly, however, and apparently requires only that some step in or directly connected with the process of mining or some feature of milling be performed upon, or that some recognized agency of operative mining or milling, occupy the mill site.[84] Thus it has been recognized that the construction of substantial improvements for the conveyance and utilization of water from or to land in connection with mining operations, the construction of storage or shop buildings required for mining operations, ore-storage facilities, and, of course, an actual mill, are proper uses.[85] It is likely, although not certain, that a mill site can be used for waste dumps.[86] The DOI has disapproved of a mill site solely for the use of transporting water.[87]

Land which has been entered for a mill site cannot be kept indefinitely without being used for its intended purpose. Moreover, unless steps toward putting it to use are clearly under way, a patent will not issue.[88]

Although the statute is unclear on the issue the DOI apparently does not impose a numerical limit to the number of dependent mill sites that can be acquired for a single mining claim. Conversely, a group of mining claims will not necessarily justify the acquisition of an equal number of mill sites. An applicant must show a sufficient and satisfactory need for more than one mill site in an area and must show need for the entire 5 acres (2 ha) in any individual mill site.[89]

Patents generally are applied for at the time patent is sought for the supporting lode or placer claim. It may be obtained by an independent application, however, if good faith is manifest in its use or occupation in connection with the lode, and no adverse claim exists.[90]

Annual Assessment Work: The rights acquired by locating a valid mining claim are terminable by others unless the claim is maintained. The federal mining law requires that not less than $100 worth of labor be performed or improvements made each year to keep the land encompassed by the claim closed to acquisition by other claimants.[91] The requirement applies to both lode and placer claims.[92] The law also provides that local mining districts (i.e., the states) can make regulations not in conflict with the laws of the United States regarding the amount of work necessary to hold possession of a mining claim.[93]

The assessment work year commences as 12 o'clock noon local time, Sept. 1. Assessment work need not be performed during the assessment work year in which the claim is located.

There is substantial uncertainty with regard to what type of work will satisfy the annual labor requirement. Basically, the work must be performed in good faith, and it must tend to develop the claim and facilitate the extraction of minerals from it.[94] Hence the work should be part of a development plan for a known deposit, as distinguished from exploration work done for the discovery of a new deposit.

It is the reasonable value of the labor that is considered and not necessarily the amount paid for it. The amount paid, however, is an important factor in determining the value of improvements.[95]

Construction of mining excavations, such as surface pits, tunnels, shafts, or other workings, generally will satisfy the requirement. Construction of buildings and replacement of machinery and equipment usually will suffice if the facilities are close to the claim and are directly related to the mining of ore. Construction and maintenance of roads for access to and from claims is a popular mode of improvement but it, too, must be related to the development of the claim. Milling and processing facilities generally will not suffice. Some courts have held that the expense of a watchman or caretaker can be used if the mine is only temporarily idle and mining equipment and materials must be protected from loss or damage, but other courts have held that such expenditures do not apply.[96] The diversion of water for use in mining generally will be satisfactory.[97]

Drilling is perhaps the most widely used form of annual labor. Such drilling should be part of a development program and not simply exploratory.

In 1958, Congress passed legislation that specifically permitted geologic, geochemical, and geophysical surveys in satisfaction of annual labor requirements.[98] Such surveys must be conducted by qualified experts, and they must be verified by a detailed report filed in the county office in which the claim is located. The report must set forth fully (1) the location of the work performed in relation to the point of discovery and boundaries of the claim; (2) the nature, extent, and cost thereof; (3) the basic findings therefrom; and (4) the name, address, and professional background of the person or persons conducting the work. Such surveys may not be used for more than two consecutive years or for more than a total of five years for any one mining claim, and the surveys must not be repetitive. The surveys should be performed on the ground. Doubt has been expressed whether radiometric testing, standing alone, is authorized.[99]

The required report can be submitted with the affidavit of labor to the county recorder, either as an appendage or as an integral part of the text of the affidavit.

The federal statute says that where claims are held in common, the annual expenditure may be made upon any one claim. Because of a statement of judicial dictum by the court in *Chambers v. Harrington,*[100] it has become common belief that the claims must be contiguous before work done on one will inure to the benefit of the others, and, in fact, BLM regulations so state.[101] Certainly the case is much stronger if the claims are contiguous, but it seems also to be well established that work done off the claims or upon a claim not contiguous to the others can satisfy the requirement as long as the work benefits the noncontiguous claims.[102] To avoid uncertainty, however, separate work should be done on each group of contiguous claims if possible.

Group assessment work should be made as part of a plan tending to develop all claims in the group, and the value of the work done must be at least $100 for each claim in the group.

Assessment work may be performed by the owner or his agent, by someone in privity with him, by someone who holds an equitable or beneficial interest in the property, or by someone who intends that the work inure to the benefit of the owner.[103] A co-owner who fails to contribute his portion of the expenditures forfeits his interest to the other owners if they follow a statutory procedure for giving notice.[104]

Each of the western states has a statute prescribing the information that must be contained in the affidavit, or "proof," of annual labor and the time within which it must be filed.[105] Six states require that the affidavit be filed within a specified time after the closing of the assessment year. The time periods are 90 days in Alaska and Montana; 30 days in California, Utah, and Washington; and 60 days in Idaho. Two states require that the affidavit be filed within a specified time after completion of the work. Twenty days are allowed in Oregon and 60 in Wyoming. In Nevada, the deadline is Nov. 1, and in Arizona, Colorado, and New Mexico, on or before Dec. 30 of the year in which the time expires for doing the work.

The owner of the claim must also file with the BLM a notice of intention to hold, an affidavit of assessment work, or a copy of a detailed report concerning the performance of geologic, geochemical, or geophysical surveys.[106] The filing must be made at the office where the certificate or notice of location is recorded as well as at the BLM state office. The BLM filing must be made on or before Dec. 30 of each year following the *calendar* year in which the claim is located. Thus, even though assessment work is not required in the ensuing calendar year for a claim located after Sept. 1 and before Dec. 31 in a given year, the BLM filing must nonetheless be made. Copies of the same document must be filed in both the recorder's and the BLM office, and the notice must set forth the BLM serial number assigned to the claim or provide other information constituting a description of the location of the mining claim sufficient to locate the claimed lands or the ground.[107]

A notice of intention to hold is ordinarily filed only when assessment work is not required for the assessment work year ending on Sept. 1 of the year in question, either because the claim was located during the prior calendar year after Sept. 1, or because a deferment of assessment work has been allowed by the BLM.[108] The notice can be in the form of an instrument, generally in the form of a letter, signed by the owner, a reference to the BLM deferment decision, or a reference to a pending decision for deferment.[109]

Failure to make required BLM assessment work filings on time and consistent with statutory requirements, including the requirements that the exact same form of instrument be filed with the county recorder and that the BLM file number for the claim be shown, will cause the claim to be conclusively deemed to have been abandoned, or in effect become void. [110]

Although not required by statute, BLM regulations require the filing of notices of intention to hold mill sites and tunnel sites on or before Dec. 30 of each year.[111] Forfeiture does not occur, however, unless the delinquent filing is not made within 30 days after notice from BLM of failure to make a timely filing is received.

In most states, the recorded affidavit constitutes prima facie evidence of performance of the work or making of the improvements described therein. It must be emphasized that the mere filing of an affidavit does not give protection if the work has not been done, and the signer of a false affidavit could be charged with perjury. Even if an affidavit is not filed, however, a locator can prove the performance of the required annual work by other evidence. The affidavit also can be supplemented by other evidence that work was done. [112]

Annual assessment work should be planned well ahead so that a determination can be made whether the contemplated work program will fulfill the requirements. Complete detailed records should be kept of all work done.

Failure to perform annual assessment work (as opposed to failure to make the required annual filings) does not result automatically in loss of the claim. The effect is to open the claim to relocation by others. If, however, the original locators, their heirs, assigns, or legal representatives resume work upon the claim before the relocation is made, the claim is no longer subject to relocation as long as the requirements are thereafter met. The claimant need not make up delinquent work for years past.

A locator can abandon a claim by voluntarily leaving it and concurrently intending not to preserve or reestablish his rights. A locator who has abandoned his claim cannot later revive his rights by resuming assessment work.[113] Abandonment, however, is very difficult to prove. A federal regulation provides that the BLM may take action to cancel a claim if a mining claimant fails to comply substantially with the requirement of an annual expenditure of $100 in labor or improvements on a claim. [114]

Patent Applications: If a discovery has been made, the claim has been properly located, and $500 worth of labor has been expended or improvements made upon the claim by the owner of his predecessors, the claim owner who meets citizenship requirements is eligible to apply for a patent.[115] With respect to a placer claim, there must be a discovery on each 10-acre (4-ha) tract within the claim.[116] Detailed procedural requirements for obtaining a patent are prescribed in Title 43, *Code of Federal Regulations,* Pt. 3860 (1987). These procedures have been summarized as follows: [117]

(1) The claimant must file an application in the proper office under oath, showing compliance with the law, together with a plat and the field notes of the claim or claims, showing the boundaries, which must be distinctly marked on the ground.

(2) Prior to filing the application, the claimant must post a copy of the plat, with a notice of his intended application, in a conspicuous place on the land embraced in the plat, and must file an affidavit of at least two persons that such notice has been duly posted, together with a copy of the notice, in the proper office.

(3) When the application, plat, field notes, notice and affidavits have been filed, the manager of the land office is required to publish a notice of the application for a period of 60 days in a newspaper to be designated by him as being published nearest the claim, and to post the notice in his office for the same period.

(4) At the time of filing his application, or at any time thereafter within 60 days, the applicant is required to file a certificate of the office cadastral engineer that $500 worth of labor has been performed or improvements made upon the claim by the applicant or his grantors, that the plat is correct, with such further description, by reference to natural objects or permanent monuments, as will identify the claim. The applicant must also furnish an accurate description of the claim to be incorporated in the patent.

(5) At the expiration of 60 days, the claimant is required to file his affidavit showing that the plat and notice have been posted in a conspicuous place on the claim during the period of publication.

A patent applicant is advised to engage the services of an attorney to process his application. The procedures are very detailed, and many delays can result from errors occurring because of unfamiliarity with the rules and regulations.

Before the applicant submits his application for patent to the land-office manager, he must obtain a correct survey, unless the claim is a placer claim on surveyed land. Application for survey is submitted to the state director of the BLM for the state in which the claim is situated, who after an initial review will then issue an order for a survey to the qualified mineral surveyor who has been designated by the applicant. [118] The surveys are conducted under authority of the cadastral engineer. A nominal fee of $5/acre ($12.50/ha) or fractional part of an acre of a lode claim and $2.50/acre ($6.25/ha) or fractional part of a placer claim must be paid as a purchase price for the patent. [119]

For many years after enactment of the Mining Law of 1872, the issuance of mineral patents was fairly routine and commonplace. Because of ever-increasing vigilance on the part of federal land-administration officials, however, it is now difficult to obtain a patent. They are issued only in cases where clearly there is full compliance with the law. The most important requirement is that of showing a valid discovery. The stringency of the test for discovery varies in different settings, but it is applied most strictly in connection with a patent application. If the BLM field engineer concludes that no discovery exists, contest proceedings can be initiated to invalidate the claim. Hence, an application for patent should be made only after careful consideration of the circumstances.

Within 60 days after commencement of publication of notice of application for patent, an adverse claim can be asserted by a person alleging a right to all or a portion of the surface ground of a mining claim sought to be patented. These are referred to simply as "adverses." An adverse might be filed, for example, by another claimant whose boundaries overlap those of the patent applicant. Within 30 days after an adverse claim is filed, the party must commence proceedings in a court of competent jurisdiction to determine the question of the right of possession.[120]

At any time prior to the issuance of patent, a protest may be filed against the patenting of a mining claim based upon any ground tending to show that the applicant has failed to comply with the law on any matter essential to a valid entry.[121] The protestant cannot acquire title to the land in question, but he can defeat the efforts of the applicant. The disposition of a protest is discretionary with the BLM through the local office in which the claim is located. The land-office manager can either dismiss the patent or order a hearing to determine the character of the claim and whether there has been due compliance with the mining law.[122]

Contest Proceedings: BLM can contest the validity of a mining claim at any time prior to patent "for any cause affecting the legality or validity of any entry or settlement or mining claim."[123] The usual basis for a contest is lack of discovery. A contest can be initiated at the instance of the BLM or after request by another governmental agency. Such requests most often are made by the US Forest Service. Contests may be initiated when the land has been requested by a federal agency for another use, after investigation in connection with a patent application, upon discovery that minerals properly removable only by lease are being mined, where there is conflict with a nonmineral user (for example, a grazing permittee), or where the claim is being used for a nonmining purpose. Discovery may be challenged under the Common Varieties Act,[124] but an adverse determination does not invalidate the object claims. After the contest is initiated, a hearing is held before an administrative law judge. The decision of the administrative law judge can be appealed to the Interior Board of Land Appeals.[125]

3.2.2.2 Mineral Leasing Act of 1920

Leasable Minerals: Under the Mineral Leasing Act of 1920, [126] deposits of oil, gas, coal, potassium, phosphate, sodium, oil shale, gilsonite, native asphalt, solid and semisolid bitumen, bituminous rock, and sulfur in New Mexico and Louisiana have been made subject to disposition through lease. The 1920 Act has had its most extensive application in connection with leases for oil and gas. Federal oil and gas law is a subject beyond the scope of this section, and the Leasing Act is discussed only as it relates to minerals traditionally extracted by mining methods. The reader is referred to *The Law of Federal Oil and Gas Leases*, 2nd ed., edited by the Rocky Mountain Mineral Law Foundation, for a thorough treatment of that subject.

Lands Available for Leasing:[127] With certain special exceptions, lands withdrawn by Executive Order for use as military bases, bombing ranges, and similar purposes; lands within other classified areas; lands within naval-petroleum and oil-shale reserves; Indian lands; lands in incorporated cities, towns, and villages; submerged lands; and lands in national parks and monuments are unavailable for leasing under the Act. Other federal lands not previously disposed of under any other public-land laws are available for leasing. Public domain lands administered by agencies other than the Interior Department can only be leased after consultation with the agency.[128]

Qualifications to Lease: Leases may be held only by adult, native-born, or naturalized citizens of the United States, domestic corporations and associations of citizens. [129] A US corporation whose shares are owned in whole or in part by aliens can hold leases if their country grants similar or like privileges to citizens or corporations of this country. [130] The regulations require that corporations reveal the names and citizenship of shareholders owning or controlling more than 10% of its stock.[131] Associations of adult US citizens can hold leases.[132] A partnership as such cannot take and hold leases, but an application submitted by a partnership is considered and treated as an application or offer by an association. [133] Municipalities are eligible to lease coal, oil, oil shale, or gas. [134]

Procedures: Authority to issue prospecting permits and leases has been delegated to the land-office managers of the BLM. Applications for leases or prospecting permits are filed in the land office having jurisdiction. Detailed and precise information must be contained on applications. A person not familiar with procedures should obtain forms from the land office and should consult with land-office personnel or a lawyer regarding procedures.

Acreage Limitations: [135] To prevent excess holdings of mineral lands under lease, limitations have been imposed on the total acreage that can be held by a person, a business association, or a corporation under the Leasing Act. The same acreage limitations apply to leases issued under the Acquired Lands Act, but acreage held under the two acts is considered separately.[136]

A person, association, or corporation may not hold in any one state more than the following acreages under lease or permit: coal, 46,080 acres (18,650 ha) per state, 100,000 acres (40,500 ha) total; sodium, 5120 acres (2070 ha) per state; phosphate, 20,480 acres (8290 ha) total; sulfur, 3 permits or leases per state, 640 acres per lease; potassium, 52,100 acres (21,050 ha) under permits and 25,600 acres (10,360 ha) total; oil shale, 5120 acres (2070 ha) per lease, one lease maximum; gilsonite, 7680 acres (3100 ha) per state; and tar sands, 246,080 acres (99,590 ha) per state.

Both direct and indirect holdings are included in computing acreage holdings. A person holding indirectly as a member of an association or corporation will be charged with his proportionate part of the corporation's or association's accountable acreage, unless he holds less than 10% of the stock or other instruments of ownership or control. [137] A party owning an undivided interest is chargeable with his proportionate part of total lease and permit acreage. An assignee remains chargeable with acreage pending approval of an assignment. [138] Acreage held, owned, or controlled in common by two or more parties may not exceed in the aggregate an amount equivalent to the maximum number of acres permitted for one lessee or permittee.[139] If a contract is entered into for the development and operation of leased land, both parties are charged with their proportionate interests in the total acreage.

The chargeability of options is discussed specifically in reference to oil and gas leases,[140] but the statutes and regulations are not clear as to whether the rules regarding options apply to leases for minerals other than oil and gas. Since options generally are described as "interests" constituting chargeable acreage holdings, [141] it is likely that the same rules would apply. Thus option acreage is charged to both the optionor and the optionee until exercised, whereupon the charge is only to the optionee. If an optionee holds an option to only a portion of the optionor's holdings, he nevertheless is chargeable with the full acreage if he has the power under the option to prevent further alienation of the balance of the optionor's interest during the option period or to otherwise effectively tie up the optionor's interest.

Interests creating acreage chargeability also include record title interests, overriding royalties, working interests, operating rights, any agreement covering such interests, and any claim or any prospective advantage or benefit from a lease, and any participation or share in production or profits which may derive from the lease.[142]

If it is determined that a party's holdings exceed the applicable acreage limitation, an application for or assignment of an amount in excess will be denied, and any interest held in violation of the acreage limitations may be cancelled or forfeited.[143]

Prospecting Permits. The Secretary of the Interior is authorized to issue prospecting permits to unclaimed and undeveloped lands for phosphate, sodium, sulfur, potassium, and gilsonite. A permit gives the holder the exclusive right to prospect on and explore the lands involved. They are issued to qualified applicants on the basis of priority in time of filing for a term of two years. With the exception of sodium and sulfur, permits can be extended an additional two years (four years in the case of phosphate). Permittees may not remove commercial amounts of minerals.

Preference Right Leases: A permittee who makes a discovery of a valuable deposit of minerals is entitled to a preference-right lease of all or part of the lands in the permit in reasonably compact form. The hard-rock prudent-person discovery test has been adopted in the regulations as the determinant of whether there has been a discovery of a valuable deposit.[144] Before a sodium, sulfur, or potassium preference right lease is issued, the land must be shown to be more valuable by reason of the presence of these minerals than it is for other uses.[145]

Competitive Leases: Lands available for leasing where prospecting and exploratory work is unnecessary to determine the existence or workability of a deposit are leased through competitive sale. Sales are arranged upon receipt of a proper application from a qualified party. After notice, sales are by sealed bids or at public auction, which may be preceded by sealed bids.[146]

Areas: Individual phosphate, sodium, and potassium leases and permits cannot exceed 2560 acres (1035 ha). No sulfur lease or permit can exceed 640 acres (260 ha). Leasing tracts for coal are of such size as the Secretary of the Interior finds appropriate and in the public interest.[147] Lands in a lease or permit must be in a reasonably compact form.[148]

Royalties and Rentals: By statute, royalties cannot be less than 12½% of the value of the coal removed from a surface mine, but they can vary as determined by the Secretary on coal removed by underground mining.[149] Current regulations provide for an 8% royalty on coal removed from underground mines.[150] Rentals are not less than $3.00/acre ($7.50/ha) per year.[151] Overriding royalties on coal leases cannot exceed 50% of the federal royalty except under certain special circumstances.[152]

The yearly rental for phosphate leases may not be less than 25¢/acre (62$/ha) for the first year, 50¢/acre ($1.25/ha) for the second and third, and $l/acre ($2.50/ha) for each year thereafter.[153] Royalties are fixed by the Secretary of Interior in advance of offering the lease, at not less than 5% of the gross value of the output of phosphate.[154]

Each lease for minerals other than coal and hard-rock minerals will contain appropriate conditions fixing a minimum annual production of the lease deposits beginning with the sixth year from date of the lease or payment of a minimum royalty of $3.00/acre ($7.50/ha) per year in lieu thereof.[155]

Sodium and potassium royalties are set by the Secretary at an amount not less than 2% of the quantity or gross value of the output at the point of shipment to market. Rental payable in advance is set at 25¢/acre (62¢/ha) for the first calendar year or fraction thereof; 50¢/acre ($1.25/ha) for the second, third, fourth, and fifth, and $l/acre ($2.50/ha) per annum thereafter during the period of the lease.[156]

A yearly rental of 50¢/acre ($1.25/ha) or fraction thereof is prescribed for sulfur leases. A royalty of 5% of the quantity or gross value of the output of sulfur at the point of shipment to market is provided for in preference right leases.[157]

Duration of Leases: Both preference right and competitive leases for phosphate, potassium, and gilsonite are issued for indeterminate periods, subject to readjustment on renewal at the end of the first 20-year period.[158] The holder of a sodium or a sulfur lease has a preferential right to renew for successive periods of 10 years.[159]

Bonds: A surety or personal bond is required prior to the issuance of a prospecting permit or lease. The amount of the bond is set on a case-by-case basis subject to minimums set in the regulations.[160]

Special Provisions Related to Coal: A complex procedure is followed before coal leases are issued. Regional coal teams comprising representatives of federal, state, and local agencies make recommendations for regional coal-leasing targets and participate in lease tract delineation, ranking, and scheduling.[161] All new coal leases are issued by competitive bidding either after a regional lease tract designation process in completed or after receipt and review of applications submitted by private parties under special circumstances. A lease sale may not be held unless the lands containing the coal deposits have been included in a comprehensive land-use plan or land-use analysis, and unless the sale is compatible with and subject to any relevant stipulations, guidelines, and standards set out in that plan or analysis.[162] After areas for potential leasing are identified, numerous reviews and consultations with governmental officials and others are held before tracts are approved for leasing under the regional tract designation process.[163] Leasing or application may be approved if the federal coal is needed to maintain an existing mining operation, to supply coal for contracts signed prior to July 19, 1979, or to prevent bypassing of federal coal.[164]

For coal leases issued after Aug. 4, 1976, diligent development, defined as the production of 1% of the removable coal reserves in a lease, must be achieved by the end of the 10th lease year. Leases issued prior to that date are burdened with a similar requirement where they are readjusted. If the leases are included in a *logical mining unit* (LMU), a unit compromising properties logically developed or part of a single mining operation, the 10-year period begins to run (1) from the effective date of the LMU, if the LMU contains a federal coal lease issued prior to LMU approval, but not readjusted after Aug. 4, 1976, and prior to LMU approval, (a "non-readjusted pre-1976 lease"); or (2) from the effective date of the most recent federal lease issuance or readjustment prior to LMU approval, for an LMU that does not contain a nonreadjusted pre-1976 lease.[165] As to an LMU, the quantity of coal that must be mined within the 10-year period is 1% of the estimated federal and nonfederal recoverable coal reserves in the LMU. Once production is begun, it must be continued through the production of 1% of the recoverable coal reserves in the lease or land during each of the first two years of production and a rolling average of that amount over continuous three-year periods.[166] The authorized officer may suspend the continuous operation requirement in certain circumstances and may also upon application suspend the requirement upon the payment of advance royalty in lieu thereof.[167]

3.2.2.3 Mineral Deposits Within Acquired Lands

Leasing Act Minerals: In 1947, Congress enacted the Mineral Leasing Act for Acquired Lands.[168] Acquired lands have been defined as (1) those in federal ownership that have never been "public domain," and (2) those lands that once were public lands but have been disposed of and subsequently reacquired by the United States by purchase, condemnation, or donation.[169] The Act authorized and established procedures for leasing deposits of coal, phosphate, oil, oil shale, gas, sodium, potassium, and sulfur on lands acquired by the United States, except those acquired for the development of mineral deposits or reported as surplus pursuant to the provisions of the Surplus Property Act of 1944, or lands within incorporated cities, towns, and villages, national parks or monuments, lands set apart for military or naval purposes, or tidelands, or submerged lands.[170] Acquired lands are not subject to the Mining Location Law.

Leases on acquired lands are issued under the same conditions as contained in the Mineral Leasing Act of 1920. The head of the executive department, independent establishment, or instrumentality having jurisdiction over the lands containing such deposits must consent to the lease, and he or she can prescribe conditions in the lease felt necessary to ensure the adequate utilization of the lands for the primary purpose for which they have been acquired or are being administered.[171]

Nonleasing Act Minerals: Section 402 of Reorganization Plan No. 3, 1946,[172] transferred the functions of the Secretary of Agriculture relative to leasing and other disposals of nonleasable minerals in certain acquired lands to the Secretary of Interior. The Act of Sep. 1, 1949, authorized the Secretary to lease non-leasing act minerals in certain lands added to Shasta National Forest. [173] The authority provided by the Acts has been broadly construed as being applicable as well to nonleasing-act minerals in all lands acquired by agencies of the DOI.[174] Pursuant to this authority, and other authority related to other designated areas, [175] the Secretary has promulgated regulations establishing procedures for leasing minerals not subject to the Mineral Leasing Act of 1920 in the subject acquired lands. [176]

Nonleasable minerals in acquired lands subject to the jurisdiction of federal agencies other than the Interior or Agriculture departments are leasable only as specifically authorized by statute.

Prospecting permits including not more than 2560 acres (1030 ha) are issued to the first qualified applicant for a period of two years, and may be extended for a period not to exceed four years. [177] Upon discovery of a valuable deposit of minerals, the permittee is entitled to a preference-right lease.[178]

No applicant may hold more than 20,480 acres (8290 ha) under prospecting permit and lease, of which not more than 10,240 acres (4140 ha) may be held under lease. The Secretary may authorize an additional 10,240 acres (4140 ha) upon an adequate showing of need. A lessee cannot hold in excess of 10,240 acres (4140 ha) of leased land for the mining of any dominant single mineral, nor shall any person at any one time hold more than 20,480 acres (8290 ha) under permit and lease in any one state.[179]

Except for preference right leases, lands containing valuable mineral deposits will be leased only to the qualified person offering the highest bonus in competitive bidding. Rentals are $l.00/acre ($2.50/ha) per year. Other terms and conditions of the leases, including the royalty rates, are established on an individual-case basis. [180] The lease will be issued for a period not exceeding 20 years, with right of renewal for successive periods, not exceeding 10 years each, at the discretion of the Secretary.[181]

3.2.2.4 Multiple Mineral Development Act of 1954—Public Law 585

Prior to Aug. 13, 1954, mining claims could not be located on land subject to prior dispositions under the mineral leasing laws. Public Law 585, commonly referred to as the Multiple Mineral Development Act, which became effective on that date, authorized the location of mining claims on lands that at the time of location are included in a permit or lease issued under the mineral leasing laws or an application for permit or lease and on lands known to be valuable for leasable minerals. [182] Conflicts may still exist between mining claims and leases in existence prior to that time. [183]

The Act attempts to provide for multiple use of land subject to both mining claims and mineral leases. A holder of a mining claim must conduct his operations, so far as reasonably practicable, in a manner that will avoid damaging any known deposit of any leasing-act mineral, and in such a manner as will not endanger or materially interfere with any existing surface or underground improvements, working or facilities that may have been made for the purpose of leasing-act operations.[184] Operations conducted under a lease on land covered by an unpatented mining claim or mill site shall likewise avoid damage to any known deposit or any mineral not reserved from the mining claim or mill site, and to any existing mining facilities.

If a leaseholder or mining claimant feels that his operations cannot be carried on without endangering or materially interfering with another operation, petition can be made to a court. If the court finds that one of the operations cannot be conducted without such interference, it can permit the operation to continue upon payment of fair compensation for the injury or damage.[185]

The Act also provides for a determination of rights to leasable minerals under preexisting mining claims.[186] A Mineral-Leasing-Act applicant, offeror, permittee, or lessee can obtain a determination of validity of claims to leasable minerals, asserted by the owners of unpatented mining claims located prior to Aug. 13, 1954. After proper service of notice, the mining claimant's rights are determined at a hearing.

3.2.2.5 Undersea Resources

The threshold question that must be answered before rights to undersea resources can be acquired is whether the area of interest is under state or federal jurisdiction. The Submerged Lands Act of 1953, the result of conflicting claims by state and federal governments to the mineral wealth of offshore lands, gave the coastal states ownership of all lands lying within three geographical miles (4.84-km) of their coastlines. [187] Historic boundaries to the extent of three marine leagues (9 miles or 14.5 km) were permitted in the Gulf of Mexico. Only Texas and Florida have been able to take advantage of the three-league boundary provision. [188]

The seaward limits of state jurisdiction are to be measured from the "coast line," defined as the "line of ordinary low water along that portion of the coast which is in direct contact with the open sea and the line marking the seaward limit of inland waters."[189] Since no definition of the "the seaward limit of inland waters" was given, it was left to the courts to define the baseline with more certainty. In 1965, the United States Supreme Court adopted the boundary definitions set forth in the 1958 Convention on the Territorial Sea and the Contiguous Zone.[190] Since that time, there has been litigation in Alaska, California, Louisiana, the Atlantic Coast states, and elsewhere regarding specific boundaries with respect to islands, coves, bays, and the like.[191]

Thus lands from the point of mean high tide to the three-mile (4.84-km) limit (three-league limit in the Gulf of Mexico) are subject to the jurisdiction of the coastal states, and reference must be made to state statutory materials to determine the method by which minerals in such lands can be acquired.

The Outer Continental Shelf Lands Act of 1953, a companion measure to the Submerged Lands Act, authorized the Secretary of the Interior to lease offshore lands owned by the federal government for the purpose of mineral development.[192] The Secretary is authorized to grant competitive oil, gas, sulfur, and other mineral leases covering any area of the outer continental shelf owned by the United States and not already under lease or withdrawn. The outer continental shelf is defined in the Act as the subsoil and seabed of that part of the continental shelf that belongs to the United States and is subject to its jurisdiction, control, and power of disposition, which, of course, excludes the coastal lands owned by the states. Thus the inland boundary of the federal offshore land subject to the Act is governed by state ownership, while the seaward boundary is a matter of international law.

The seaward boundary, as presently defined, was established at the 1958 United Nations Conference on the Law of the Sea at Geneva, Switzerland, and was subsequently ratified by the United States. It defines the continental shelf as "the seabed and subsoil of the submarine areas adjacent to the coast [and islands] but outside the area of the territorial sea, to a depth of 200 m (660 ft) or, beyond that limit, to where the depth of superjacent waters admits of the exploitation of the natural resources of the said areas."[193] Thus the boundary varies according to the limit to which exploitation of undersea resources can be achieved.

This, of course, is a matter of technology, commodity prices, and world economics.

It should be noted that the legal definition of the continental shelf has little, if any, relationship to the geologic definition. The legal definition may include areas that would be known to geologists as continental slope, continental rise, and continental borderlands. In any event, the definition covers a vast area of land. As applied to the United States, the legal definition of continental shelf, using the 200-m (660-ft) cutoff, encompasses some 850,000 mi^2 (1,360,000 km^2), an area equal to roughly one-fourth of the United States.[194]

Pursuant to authority given by the Act, the Secretary has promulgated regulations providing for the issuance of oil and gas and sulfur leases after competitive bidding. These are contained in Part 200 of Title 30, *Code of Federal Regulations,* Part 200.

Sulfur leases are issued in much the same manner as oil and gas leases. Leases are awarded on the basis of sealed competitive bonus bids. The size of the lease is determined by the Secretary, and the term is for not more than 10 years and so long thereafter as sulfur may be produced from the area in paying quantities, or drilling, well reworking, plant construction, or other operations for the production of sulfur are being carried out. Royalties and rentals are determined by the Secretary, but the royalty can be no less than 5% of the gross production value.[195]

The Secretary is also authorized to grant leases on the outer continental shelf to qualified persons offering the highest cash bonus bid for any mineral other than oil, gas, and sulfur. Any person may submit a request to the director of the Minerals Management Service that such offshore minerals as gold, chromium, tin, titanium, salt, phosphate, and sand and gravel be offered for lease. The Secretary may also voluntarily offer such leases. The primary lease term is 10 years for sand and gravel and not less than 20 years for all other minerals. The lease can be continued thereafter for as long as production continues. Leases are subject to such royalty, rental, acreage, and other terms and conditions as the Secretary may prescribe.[196]

Finally, the Act provides that any person authorized by the Secretary may conduct geological and geophysical exploration in the outer continental shelf so long as it does not interfere with operations under an existing lease and is not unduly harmful to aquatic life.[197]

For years, the nations of the world have struggled over how to regulate appropriation of the deep seabed and the minerals contained therein. These minerals represent a vast resource, particularly for the mineral needs of the future (see Chapter 22.8, Marine Mining). For example, 10% of the entire floor of the Pacific Ocean is thought to be covered by manganese nodules, potato-sized oval rocks generally found at depths between 10,560 and 19,800 ft (3200 and 6000 m). These nodules contain over 20 elements, with an average metal content of 24.4% manganese, 14.0% iron, 1.9% nickel, 0.5% copper, and 0.35% cobalt. These strategic metals are particularly important to the United States, which imports almost all of its manganese and cobalt, most of its nickel, and much of its copper requirements.[198] The cost, however, to develop a single mine site to exploit these deep-sea minerals has been estimated at $2.5 billion.[199]

Since the 1960s, the United Nations has attempted to enact an international Law of the Sea Treaty, based on the principle that the mineral resources of the deep seabed are the common heritage of mankind. The treaty would allocate jurisdiction of deep seabed mineral resources to the coastal nations that would develop the minerals in accordance with treaty standards and pay over to an international fund, for assistance to the less-developed countries, a portion of the revenues derived from mineral development. To date, a vast majority of countries has adopted and signed the Law of the Sea Treaty, but the United States, for various reasons unrelated to mining, has not.[200] Instead, the United States enacted the Deep Seabed Hard Mineral Resources Act of 1980.[201]

The Act is designed as an interim measure to allow deep seabed exploration and mining until such time as an international Law of the Sea Treaty is ratified by the United States. If such a treaty is ratified, the Act will be superseded; if not, the Act will permit regulated deep seabed mining on a permanent basis. By its terms the Act pertains to the "deep seabed," which is defined as the seabed (and the underlying 33 ft or 10 m of subsoil) lying outside the "continental shelf' as defined earlier under the Outer Continental Shelf Lands Act.[202] The Act prohibits exploration for or commercial recovery of deep seabed minerals unless licensed to do so by the federal government or by a foreign government that grants reciprocal rights to United States citizens under a similar regulatory scheme. This prohibition does not apply to scientific research, mapping, random sampling, or the design, construction, or testing of exploration or commercial recovery equipment and facilities.[203]

Exploration licenses and commercial recovery permits may be issued by the Administrator of the National Oceanic and Atmospheric Administration, the agency charged with regulating deep seabed mining. Subject to environmental considerations, the applicant chooses the size and location of the area for exploration or mining. Exploration licenses have a primary term of 10 years, with discretionary extensions.[204] Detailed procedures and rules regarding exploration licenses are set forth in administrative regulations.[205] The application fee for an exploration license is presently $100,000.[206]

Any person holding a valid exploration license can apply to the Administrator for a commercial recovery permit covering all or part of the exploration area. A commercial permit authorizes the holder to recover, own, transport, use, and sell deep seabed minerals.[207] In order to allow a time period for ratification of a suitable international treaty, the 1980 Act prohibited the issuance of any commercial recovery permits before Jan. 1, 1988. Commercial permits have a term of 20 years and so long thereafter as minerals are recovered in commercial quantities.[208] Detailed procedures and rules regarding commercial permits and operations have been set forth in administrative regulations.[209] The present fee for a commercial recovery permit is $100,000.[210]

Applicants for an exploration license must file a detailed exploration plan, while applicants for a commercial permit must file a recovery plan describing the technology and environmental protection measures to be employed. All applicants must show they have the requisite financial responsibility and technical capability to operate according to the submitted plan. All leases must include requirements to assure diligent development, including a requirement that commercial recovery be underway within 10 years from the permit date unless an extension is granted by the Administrator for good cause.[211]

The Act contains provisions designed to protect deep-sea miners' investments should the United States ratify a treaty having terms inconsistent with the Act.[212] But the Act expressly disclaims any financial responsibility by the government in the event an investment is impaired through the ratification of such a treaty.[213] These facts, along with the extremely high cost of deep-sea operations, political instability in many parts of the world, low-metal prices, and, in the absence of a treaty, the legal uncertainty of mining rights in international waters, indicate that deep seabed mining is not likely to occur soon.[214] There are also concerns over how to conduct large-scale seabed mining without wreaking havoc on the economies of the land-based producers of the same metals.[215] The ability to conduct deep seabed mining has been proven from a technological standpoint.

Sooner or later, such mining will surely occur, with the first mines likely to be located beneath the deep waters of the north Pacific between Hawaii and the west coast of the United States.[216] Most authorities agree, however, that despite earlier expectations, deep seabed mining will not likely occur before the next century.[217]

3.2.2.5 Materials Act of 1947

The Materials Act of 1947 authorizes the Secretaries of Interior and Agriculture to dispose of mineral materials, specifically including, but not limited to, common varieties of sand, stone, gravel, pumice, pumicite, cinders, clay, and petrified wood, in lands under their respective jurisdictions, if such disposal is not otherwise authorized by law.[218] The Act also authorizes disposal of minerals found on lands which have been withdrawn in aid of the function of a federal department or agency other than the DOI, or a state, county, municipality, water district, or other local-government subdivision or agency, with the consent of such federal department or agency or such state or local government unit.[219] Regulations promulgated by the Secretary of Interior under the Act are found in Title 43, *Code of Federal Regulations,* Group 3600 (rev. June 13, 1970); and by the Secretary of Agriculture in Title 36, *Code of Federal Regulations,* Secs. 251.4 and 251.4a.

The Act does not authorize a Secretary to sell minerals found on lands in any national park or national monument or on lands held for the use or benefit of Indians.[220] Disposal from BLM lands cannot be made from lands on which there are valid existing mining claims.[221] Sales from national forest lands can be made from lands subject to a claim located before and after July 23, 1955, if the disposal will not materially interfere with the rights of the mining claimant.[222]

Materials can be sold upon the request of any interested party or upon request of an authorized officer. Mineral materials not exceeding 100,000 yd^3 (76,460 m^3), or weight equivalents can be acquired by negotiated sale with the authorized officer, if it is impracticable to obtain competition, and if a negotiated sale is in the public interest.[223] If the materials are to be used in connection with the development of federal lands or in connection with a public-works improvement program on behalf of the federal, state, or local government agency, and the public exigency will not permit the delay incident to advertising, negotiated sales can be made for a sum not exceeding $10,000.[224] Sales of up to 200,000 tpy (180 kt/a) may be made to lessees under a US-issued mineral 1ease.[225] Otherwise, the sale is by competitive bidding.

The materials to be sold are designated in the notice of competitive bidding, which must be advertised in a local newspaper and in the office where the bids are to be submitted. The materials are appraised, and the sale will not be complete unless the bid equals or exceeds the appraised value.

All contracts are subject to the continuing rights of the United States to use the surface and to issue leases, permits, and licenses involving the use and occupancy of the surface, provided that such subsequently authorized use shall not endanger or materially interfere with the purchaser's production or removal of the materials under contract. Performance and reclamation bonds must be provided, and lands mined under the Materials Sales Act must be restored after completion of the mining operation.[226]

Deposits of the listed minerals having some property giving them distinct and special value remain locatable under the mining laws. It is extremely difficult to determine whether a particular deposit of minerals has those characteristics. If it does, the deposit technically is not subject to disposal under the provisions of the Materials Act. Nevertheless, because of the BLM's policy of interpreting the distinct and special-value exception strictly, more security will be gained in most instances by acquiring the minerals under the Materials Act rather than under the Mining Location Law.[227] Certain materials not considered to be common varieties because of their distinct economic or commercial value are block pumice, cement grade limestone, metallurgical and chemical grade limestone, and gypsum.[228]

A mining claimant risks prosecution for trespass and is liable for damages to the United States if he removes materials from a claim later held to be invalid. To be weighed against these considerations, however, is the possibility that a person who discovers a deposit of substantial value might not be the high bidder at a competitive sale.

3.2.2.6 Indian Lands

If, during a preliminary land-status check, it is determined that some of the lands of interest are Indian lands, great care should be taken to obtain as much information as possible regarding the specific status of the lands so designated. The superintendent of the particular Indian agency having jurisdiction has custody of pertinent land records. These include all leases, deeds, patents, assignments, and related documents. Duplicates of all instruments affecting the title to Indian lands should be available at the Bureau of Indian Affairs (BIA) Office, Washington, DC.[229]

Lands which are classified as Indian lands by the US government fall generally into one of two categories: tribal lands and allotted lands.

Allotted lands are those allotted to individual tribal members under the general Allotment Act of 1887.[230] Pursuant to the Act, certificates of allotment containing restrictions against alienation were given to individual Indians. These gave the holder the right to exclusive possession and equitable title to the land.[231] Title to allotted lands was to be held in trust for the benefit of the allottee by the United States for a period of 25 years. This period has been extended, however, and much of the allotted lands still are held in trust. Because of the trust arrangements, any dealings with regard to allotted lands must be carried on with both the allottee and the BIA. Allotments were no longer made after passage of the Indian Reorganization Act of June 18, 1934.[232]

Tribal lands are defined as those within the boundaries of a reservation that have not been allotted and in which no vested rights exist.[233] They are administered for the use and benefit of all Indians constituting the tribe. Dealings regarding tribal lands must be carried on with both the tribe and the BIA.

Indian reservations are parts of the public domain that have been set aside by proper authority for use and occupation by a group of Indians.[234] The boundaries were determined by treaty, executive order, or act of Congress. Hence title is held by the United States, and the Indians have the right of use and occupancy. Governing control rests in the tribal councils, subject to much supervision by the BIA.

Indian tribes have periodically claimed aboriginal property rights to lands historically occupied by the tribes. In the case of *Tee-Hit-Ton Indians v. US,*[235] the Supreme Court held that original or aboriginal Indian title can be extinguished at the will of Congress without compensation, provided no recognition has previously been given by the federal government to such right of occupancy.[236] The tribes enjoy full use and occupancy of such lands. These rights are enforceable against all but the federal government.[237]

Neither the general mining laws nor the federal leasing laws are applicable to reservations.[238] Hence, unless the United States reserved mineral rights in the grant of the reservation, special procedures must be followed to obtain mineral rights. If minerals were reserved by the United States, they are subject to acquisition under the general mining laws.[239]

Legislation enacted by Congress in 1934 and 1938 greatly simplified the law relating to acquisition of mineral rights on Indian lands. Section 17 of Indian Reorganization Act of 1934[240] provided that the Secretary of Interior could issue a charter of incorporation to any tribe that made application, which charter could convey to the tribe comprehensive powers of management and disposition of its property. After a trial period, chartered tribes could lease lands for mineral purposes, subject to Secretary approval. The issuance of minerals by chartered tribes is governed by provisions of the charter.

Congress subsequently enacted the Act of May 11, 1938,[241] which permitted the leasing of unallotted land for mining purposes by authority of the tribal council or other authorized spokesmen for the Indians with Secretary of the Interior approval. The Act excepts from its coverage the Crow Reservation in Montana, the ceded lands of the Shoshone Reservation in Wyoming, the Osage Reservation in Oklahoma, and the coal and asphalt lands of the Choctaw and Chickasaw tribes in Oklahoma.

Authorization for the leasing of tribal lands must come from the tribal council or other authorized representative of the tribe. However, the local superintendent of the BIA has the responsibility for handling formal procedures. Regulations governing the leasing of minerals in Indian lands are contained in the *Code of Federal Regulations.*[242] Leases may be issued not to exceed 10 years and as long thereafter as minerals are produced in paying quantities.[243] Leases for oil and gas must be offered for sale at a public auction or upon sealed bids, but leases for other minerals can be negotiated and approved without advertisement and sale.[244]

The Superintendent of Indian Affairs may, upon the consent of tribal authorities, issue prospecting permits for minerals other than oil and gas. These permits, however, do not give the permittee any preference right to a lease, unless specifically provided.[245]

Leasing of allotted lands was authorized by the Act of Mar. 3, 1909.[246] Under this Act, the allottee can lease the land allotted to him for mining purposes subject to approval of the Secretary of the Interior. Rules and regulations implemented pursuant to the Act of Mar. 3, 1909, are found in the *Code of Federal Regulations.*[247]

Leases on allotted lands for minerals other than oil and gas can be negotiated. Permission must first be obtained from the superintendent to negotiate with the allottee, or his heirs or devisees, as the case may be. The superintendent then decides whether it is in the best interests of the tribe and the allottee to negotiate privately. If his conclusion is negative, competitive bidding procedures will be followed. Otherwise, contact then can be made with the allottee with whom negotiations are carried on. The agreed terms are subject to approval by the superintendent and to pertinent regulations. Leases for oil and gas must be offered for sale at public auction.

Congress has enacted a series of statutes, each applicable only to a single tribe or tribal group, providing for a simultaneous termination of federal supervision over the trust and restricted property, both of the tribe and its individual members, and cessation of the federal services theretofore furnished to such Indians. Generally, the effect of this legislation is to give the tribe complete responsibility for the sale or encumbrance of all property, real and personal, held in trust by the United States for the tribe. Some of the statutes call for the transfer of title to a trustee, who can then dispose of such property and distribute the proceeds among the members of the tribe.[248]

Congress has recently enacted legislation that specifically authorizes the tribes to enter, with the approval of the Secretary of Interior, joint venture or operating type agreements.[249] The significance of this act, known as the Indian Mineral Development Act of 1982, lies in the fact that mineral developers may now use two separate and independent alternatives for development of Indian land. They can deal with the BIA using the traditional leasing methods described previously or they may enter into a "joint venture, operating, production sharing, service, managerial, lease, or other agreement" directly with the Indian tribe, subject to BIA approva1.[250]

When approving or disapproving a privately negotiated agreement for mineral development of Indian lands, the Secretary of Interior is required by the Act to determine whether the agreement is in the best interest of the Indian tribe. In making this determination, the Secretary must consider the potential economic return to the tribe; the potential environmental, social and cultural effects on the tribe; and the provisions for resolving disputes that may arise between the parties to the agreement.[251] According to the US 9th Circuit Court of Appeals, a tribe may rescind an agreement at any time prior to secretarial approval.[252] Therefore, all parties to a mineral development agreement should be aware that there is no agreement until Secretarial approval is received.

3.2.2.7 Alaska Native Lands

Special laws govern the acquisition of mineral rights from native groups in Alaska. Most of the lands owned by the Alaska Natives were granted to them as shareholders in Native corporations under the authority of the Alaska Native Claims Settlement Act (ANCSA).[253] Unlike most other lands reserved or granted to Native Americans, the lands conveyed to the Native Corporations pursuant to ANCSA are not subject to restrictions against alienation. The decision of a Native corporation to enter into an exploration agreement or to lease or sell any of its lands does not require the review or approval of the DOI. These lands are, in essence, private lands over which the Secretary of the Interior exercises no control.

Four different types of Native corporations were created to receive lands pursuant to ANCSA. They are regional corporations, village corporations, group corporations, and urban corporations. Village, group, and urban corporations receive only the surface estate in lands conveyed to them. The subsurface estate in those lands is acquired by the regional corporation for the region in which the village, group, or urban corporation is located.[254] Regional corporations acquire both the surface and the subsurface estates to lands conveyed to them.[255] Where the regional corporations are precluded from acquiring the subsurface estates to corporation lands, they are entitled to acquire the subsurface estate to certain in lieu lands, the surface owner of which may be the federal government, the state government, or some third party.[256]

Prior to the enactment of ANCSA, a significant number of Native reserves existed in Alaska. Pursuant to ANCSA, village corporations for villages located within these former reserves were granted the option to acquire lands within their reserve.[257] These village corporations are granted surface and subsurface rights to the lands within their former reserves.[258]

Because Native corporations exercise complete freedom to enter into agreements respecting their lands, an exploration and development agreement between a mining company and a Native corporation can theoretically be structured as any other agreement with a private landowner. Therefore, the opportunity exists for a mining company to obtain exploration and development rights to Native lands in Alaska, unencumbered to a large extent by the governmental restrictions attached to other Native lands.

3.2.2.8 Conflict with Surface Rights

During the preliminary land-status check, it should be determined, if possible, whether there has been a severance of the surface and the mineral estates of the subject property. Although private lands can be affected by severances, they most often occur when the states or the federal government dispose of the surface and retain the minerals. In the event a severance has been made, the rights of the surface owner must be considered before the lands are entered for minerals exploration and development. Where the minerals have been reserved by the United States, the rights of the surface owner are governed largely by statute. Thus, whenever a land status check reveals such a severance, further inquiry should be made to determine the statutory authority under which it occurred.

The Stockraising Homestead Act of 1916, which is the most common governing statute, provides for nonmineral entry and patent of stock-raising lands, provided that the patent contains a reservation to the United States of all minerals in said lands and the right to prospect for, mine, and remove the same.[259] That Act also establishes the rights of others to enter upon the potential land to develop the minerals.[260] Entry for prospecting purposes or to locate claims can be made without obtaining prior approval from the surface owner.[261] The right is specifically given to enter the land and prospect for coal and other minerals, providing that the prospector does not injure, damage, or destroy the permanent improvements of the stock-raising homestead entryman or patentee. The mineral prospector must pay for any damages to crops that occur as a result of his prospecting activities. To the extent permitted by the confidential nature of the prospecting activities, it is advisable to make an agreement with the surface owner as soon as possible. His interest in the property and improvements thereon should always be respected.[262]

Once a party acquires from the United States the coal or other mineral deposits in any such land, or the right to mine and remove the same, he is given the right to reenter and occupy so much land as may be required for mining. Before such activities are undertaken, however, the miner must (1) secure written consent or waiver of the homestead entryman or patentee, (2) make an agreement for payment of damages to crops or other tangible improvements with the entryman or owner, or (3) execute a bond or undertaking to the United States for the benefit of the homestead entryman or owner.[263] Any person who prospects for, mines, or removes by strip or open-pit mining methods, any minerals from any land included in a stock-raising or other homestead entry or patent must also compensate the entryman for any damage caused to the value of the land for grazing.[264]

The DOI has concluded that common varieties of sand, stone, pumice, pumicite, and cinders in lands patented under the Stockraising Homestead Act are subject to the Materials Disposal Act of 1947, with the exception of sand, gravel, and ordinary rock, which have no value separate from the soil itself.[265]

Multiple Surface Use Act of 1955: In 1955, Congress enacted the Multiple Surface Use Act[266] to curtail nonmining use of the surface of mining claims. The Act provides that any mining claim located after July 23, 1955, "shall not be used, prior to issuance of patent, for any purposes other than prospecting, mining or processing operations and uses reasonably incident thereto," and that rights under any mining claim thereafter located will be subject, prior to the issuance of patent therefore, "to the right of the United States to manage and dispose of the vegetative surface resources thereof and to manage other surface resources thereof (except mineral deposits subject to location under the mining laws of the United States)."[267]

The Act also provides that any such mining claims will be "subject, prior to issuance of patent therefore, to the right of the United States to use so much of the surface as may be necessary for such purposes or for access to adjacent land. " Any use of the surface of the mining claim by the United States, however, must not "endanger or materially interfere with prospecting, mining, or processing operations or uses reasonably incident thereto."[268] A mining claimant is authorized to cut and use timber from the claim and to use sand and gravel for mining purposes.[269]

As a result of the legislation, the owner of mining claims located subsequent to July 23, 1955, is entitled to use the surface only as necessary for his operation, and his claim is subject to surface-management activities of the federal government until patent.

A procedure is established in the Act for determination of surface rights on claims located before July 23, 1955.[270] After notice and response, a hearing is held to determine the validity and effectiveness of the claimant's interest in the mining claim. At that hearing, the claim can be contested on various grounds, including lack of discovery, but the effect of a determination adverse to the claimholder is simply to restrict his surface rights as specified in the statute. The claim is not invalidated.

Holders of claims located prior to July 23, 1955, although not necessarily subject to surface management by government agencies, cannot use the surface for nonmining purposes.[271]

3.2.3 STATE LANDS

States have acquired ownership of lands within their boundaries by various means, which are important because they may impact a particular state's ability to grant mineral rights, procedures to be followed in obtaining such grants, and conditions and limitations to them.[272] Most state lands that are of interest to mineral developers were acquired by grants from the United States. Other important categories of state lands are those comprising the beds of navigable streams and offshore submerged lands three geographical miles (4.8 km) seaward from the ordinary low waterline. Large areas in many of the western states are state owned through grants of specific numbered sections of the public land survey.

Often an examination of records in a state land office occurs by reason of notations of BLM records of transfers by the United States to the state. The state records may show that the state in turn conveyed ownership to a private citizen. Such conveyances might have reserved minerals in the state.

State land office records, of course, vary from state to state, but many have plats showing state-owned lands, including lands in which the state owns minerals only. If a particular tract of interest has passed from state ownership, the document by which it was conveyed and the statutory authority for the conveyance should be reviewed to determine the nature of the estate that passed and any reserved rights in the state.

Authority to grant mineral rights in state lands stems from state constitutions and statutes.[273] To determine the procedures that govern the granting of mineral rights, inquiry should be made of the agency that administers state lands. Most states have a leasing system for minerals, although Alaska, Arizona, Colorado, Idaho, Maine, and Oregon utilize some form of location as a part of the process of obtaining a 1ease.[274]

The regulations of the particular state involved must be examined to determine eligibility requirements. Corporations, for example, might have to be qualified formally to do business in the state before being eligible to lease state lands. Whether an alien or a corporation whose stockholders are aliens can hold state leases must also be ascertained. Acreage limitations might also exist.

Some states issue leases directly, others go through a prospecting permit phase with a preference right lease being issued only if minerals are discovered by the permittee. Leases other than preference right leases might go to the first qualified applicant, or they might be issued upon competitive bidding. When leases are relinquished or terminated, it is common to put them up for competitive bid, and if there are no bidders to make them available to the first qualified applicant.

Minerals are generally classified into groups such as metalliferous minerals, coal, uranium, industrial minerals, etc., with lease terms designed accordingly. Like all leases, the key provisions are the royalty, rental, and lease term clause.

State leases often require approval of mine plans by the administering agency. Performance and reclamation bonds are also commonly required.

3.2.4 PRIVATE LANDS

3.2.4.1 Geologic and Geophysical Trespass

Before entering privately owned land to conduct geologic and geophysical investigations, permission of the surface owner and the minerals owner should be obtained, both to establish good relations and to avoid possible liability for trespass. If unauthorized geologic or geophysical tests are made, the prospector might be liable to the landowner under the theory of geophysical trespass. In connection primarily with oil and gas exploration, geophysical information is acquired over very large areas. When entry upon the land is made for these purposes, without the permission of the owner, or where the permission given by the owner is exceeded, the courts have in some cases held the prospector liable for trespassatory damages. The landowner has the right to have the mineral potential of his land remain unknown unless he is paid a fee or gives permission to someone to enter and gather such information, and if the right is violated he can obtain damages for the reasonable value of the information that was obtained. In addition, if a geophysical trespasser publishes information to a third person that is untrue, the value of the land is disparaged, and the landowner can recover an amount equal to the reduction in value of the 1and.[275]

The theory of geophysical trespass could apply as well to hard-mineral exploration. For this reason, the consent of the owner should always be obtained before land is entered for prospecting.

3.2.4.2 Ownership of Minerals

Questions can arise as to who owns a particular type of mineral in fee lands. There are many situations where there has been a conveyance of the land in question with reservation of minerals using wording that is not necessarily all inclusive. There has been much litigation, for example, over the issue of whether a reservation of "oil and gas and other minerals" includes such minerals as sand and gravel, geothermal resources, or minerals that were not recognized to be valuable at the time the conveyance was made.[276] In one line of Texas cases, the issue was whether minerals that could only be extracted through destruction of the surface were encompassed in a prior reservation of minerals.[277] If the mineral involved is near surface and of widespread occurrences and there has been a severance of the mineral and surface estate on the land involved, state law should be examined to determine whether the surface owner has any claim to ownership of the mineral in question.

Questions can also arise whether the owner or lessee of a particular mineral in lands also has rights to associated minerals. A good example of such a problem is provided by claims of the owner of oil and gas right to methane gas found in coa1.[278] In *U.S. Steel v. Hoge,*[279] the Pennsylvania Supreme Court held that methane gas within a coal seam is owned by the owner of the coal, but the issue is still open in other jurisdictions.

3.2.4.3 Agreements

Agreements pertaining to private mineral lands differ from those involving state and federal lands in that the terms are extremely variable (see Chapters 6.4 and 6.6). Terms of agreements relating to governmental lands generally are established by statute and regulation, whereas terms relating to private lands are established by negotiation. The same principles that apply to agreements relating to private lands, however, apply to agreements with owners of mining claims and to a lesser extent to agreements with the holders of mineral leases. Transfers of mineral interests in private lands can be made in many different ways, and a variety of terms can be included in any particular agreement. Nevertheless, both the vendor and the vendee have certain basic objectives that they should seek to obtain through the agreement terms. These transactions can be structured with respect not only to lands that have passed from public ownership ("fee lands"), but also to mining claims and governmental leases that are held by private citizens or business entities. All such interests are described in this chapter as private lands. Further discussion of this subject is available in Chapter 7.1.

Most transactions fall into one of several basic categories. These are an outright purchase, a purchase with the reservation of royalty, a lease, and an option to lease or purchase. If the property owner is in the mining business, the deal may be structured as a "farmout agreement," where the new party earns an interest by spending money on development and then participates as a joint venture partner with the owner.

Private lands usually become of interest because preliminary studies indicate that an area has mineral potential. The immediate objective of the mining company, then, is to gain access to the land so that detailed exploration can be carried on. Because such activities might inflate the value of the land, however, the company usually desires to obtain mineral rights beforehand to as much land as possible in and around the area of interest.

Because of the high cost of exploration, the company desires that the initial outlay for land acquisition be as small as possible. It is desirable, therefore, to defer payments and to make them contingent on production. For the same reasons, it is in the company's interest to be able to terminate future payment obligations after having had sufficient time to evaluate the property.

Perhaps the best agreement for achieving these objectives is the option—an agreement by a vendor to give the vendee the exclusive right to accept the terms of the vendor's offer for a set period of time. The option can be to purchase or to lease. The terms of the sale or lease are established at the time the option is granted, and if the optionee exercises the option, he binds himself to the preestablished terms and conditions. An option must be coupled with an agreement giving the optionee the right to enter and explore the land during the option period.

The advantage of an option is that it permits the mining concern to examine the property without undue expense and without fear of losing it to another. If the findings are negative, the option can simply be released or permitted to expire. The disadvantage to the mining concern is that the terms of the lease or sale are negotiated on the assumption that favorable indications of mineralization will have been found in the event the option is exercised. The economic terms may accordingly be less favorable to the company.

The main advantage of an option to the landowner is that it forces the optionee to conduct exploration without delay before the option expires. The landowner may not gain this benefit, however, if he options the ground to a speculator, who might simply hold the land hoping for its value to increase. The landowner should receive reasonable consideration for the option, and he should restrict the option period to an amount of time sufficient for the company to evaluate the land thoroughly.

Outright purchases of land are rare because they require an advance outlay of capital by the mining company without sufficient knowledge of the land's value. It might occasionally be advantageous to the vendor, however, to seek an outright sale with a reservation in himself of a royalty interest. This permits him to get an immediate return with the possibility of substantial royalty income later. He will not, however, have the power to force rapid development of the property. It is that same lack of control by the owner that provides an incentive to the company to make an outright purchase.

Perhaps the most common type of mineral transaction is the mineral lease. The terms of such a lease are extremely flexible, and they can be tailored to the interests of both the lessor and the lessee.

The lessor can usually demand a bonus payment for a lease, especially if other companies are interested in the property.

The lessor wants to ensure that the lessee does not simply tie up the land without benefiting it, and so he should attempt to insert some provision that has the effect of pressuring the lessee to take steps toward development. There are several approaches. The lease can provide for the payment of delay rentals if there is no production from the property within a specified period of time. The same effect can be achieved by providing for periodic increases in rental payments. Another common practice is to require the lessee to make yearly minimum expenditures for exploration, or to perform a set amount of work, such as a minimum number of feet (meters) of core drilling.

The lessor also can assure himself of some return before mining operations are profitable by requiring the periodic payment of a minimum advance royalty during the term of the lease. Minimum royalty payments can be credited cumulatively against future royalty payments, but often are credited only in the year of payment.

The lessor also can require an express-development covenant, which obligates the lessee to be reasonably diligent in commencing mining operations and in continuing thereafter. Breach of such a covenant by the lessee will permit cancellation of the lease or possibly recovery of damages. If the property has known ore reserves, the express-development covenant can be made more specific. For example, the lessee can be required to produce a certain number of tons per month. In the absence of an express-development covenant, the courts have recognized that the lessee has an implied obligation to use reasonable diligence in commencing mining operations.[280] A delay-rental or minimum-royalty provision might negate the obligations of the lessee to carry out diligent development, because the lessor in effect receives a substitute for production.

It is important to the lessee, however, to keep initial payments small so that the money can be put into exploration. Deferred increased payments usually are acceptable because, by the time they become due, enough information will have been obtained either to justify the outlay or to cause termination.

The lessee also desires a long primary term so that there will be sufficient time to evaluate the property and get it into production if minerals are encountered. On the other hand, the lessor will often set a shorter primary term, 5 to 10 years, with extensions to be contingent upon production.

The heart of any mining-lease agreement is the royalty provision. It is by receiving a share of the minerals produced that the lessor fully realizes the value of his property. The wording of the royalty provision will be governed by the type of mineral or minerals involved, and the royalty percentage, although arrived at by negotiation, generally will be influenced strongly by the royalty schedule in common use in the area at the time.

Because of the variety of products that can be sold from a mining operation, the term "net smelter returns" is currently less favored. It is common to see a reference to the royalty as being a percentage of "net proceeds," meaning the proceeds received from a buyer of product produced from the mine, no matter what its form. Products may be in the form of ores, concentrates, leachates, precipitates, or in the case of gold and silver, nearly pure dore' bars. The purchase may be a gold refinery rather than a smelter. Clarity should be provided in stating the point at which the valuation is made, and the costs that may be deducted. If gold dore' bar is produced at an onsite mill, the royalty would ordinarily be based on the value of that product rather than on the value of the concentrate or other gold-bearing material prior to its entering the final stage furnace. Deductible costs then would be those incurred in shipping and further treating that product into refined gold.

A good rule of thumb in this regard is to consider the salable product to be that which is produced at a milling or other beneficiating facility constructed as part of the mining operation. Deductible costs then would be those incurred in shipping, treating, and selling that product to third parties. Clearly, that rule will not work in all cases because the operator might send the product before sale for further treatment on a contract or "tolling" arrangement, or it might perform upgrading or refining at its own facility offsite. Careful drafting of the royalty provision in light of the particular mineral involved can reduce the likelihood of problems arising from such factors. (Mineral sales are discussed in Chapters 2.3 and 25.5.)

Another common form of royalty is the net profits royalty where, instead of a percentage of proceeds, the owner is given a percentage of profits. Ideally, the lessee simply subtracts all of his costs from revenues and gives the owner a percentage of the profits realized when and if they ever occur. In this way, the owner shares the risk that the operation will not be profitable. Because returns are deferred and are keyed to recovery of costs by the lessee, the net profits percentage paid to the owner is much higher than what a net smelter or net proceeds royalty would be. It is common, for example, for a lessee to offer a 15% net profits royalty in place of a 5% net smelter returns royalty.

The timing of net profits revenues to the owner is dependent upon whether preproduction costs can be recovered in full prior to the realization of net profits or whether they are to be recovered on an annualized amortization basis. Other important factors to consider in a net profits royalty provision are the degree to which the lessee can charge offsite overhead and the basis for charging personnel and equipment that are used in part for other operations. The lessee will usually want to charge an interest factor to reflect the costs of invested funds. Net profits royalties are often the basis for disputes, so they should be drawn with great care.

Revenues received by the lessee may be those paid by the purchaser, such as a smelter or refinery, that are based upon current published market prices, or they can be based upon various contractual provisions that might vary considerably from prevailing market prices. Product may be sold, for example, months before it is actually produced. The owner has the choice then of tying his royalty revenues to whatever the lessee negotiates, or he can tie his revenues to market price at the time of production by so specifying in the lease. There are obviously risks either way, both to owner and lessee.

If the desired property is already held by a mining company, the owner might prefer to have the ability to participate actively in a mining operation rather than just to be a royalty owner. The parties might therefore form a "joint venture," a special type of partnership formed for a single purpose, or the interested party might first be given the opportunity to "earn in" by spending a set amount to achieve ownership of a proportionate interest in the property. The earn-in agreement is just a modified form of the exploration and option agreement described earlier. The earning party can drop out at any time, thus giving up the option, or he can spend the required amount and thereby earn a vested interest in the property. The vested interest can be a 50% interest, or it can be over or under that amount.

Once the earn-in is completed and the acquired interest vested, then the parties will usually form a joint venture pursuant to a joint venture or joint operating agreement. One party is designated as the operator with responsibility for preparing and carrying out plans and budgets and the other, the nonoperator, pays his share of approved budgets. A management committee is usually formed of representatives of the operator and nonoperator. The party with a majority interest usually has voting control.

A key provision in any joint venture agreement is that which states the consequence of a party's election not to fund his share of an approved budget. The nonparticipating party can be diluted on a proportionate basis to a minimum interest of say l0%, or he can forfeit his participating interest. It is common in either case to convert the nonparticipating party's interest to a carried net profits interest.

A vital part of any joint operating agreement is the accounting procedures that govern how the operator will charge his costs. Those dealing with offsite overhead and shared equipment, facilities, and labor are key provisions in the accounting procedures.

Royalties for large-volume, low-unit-value commodities, such as sand and gravel, are often based upon tonnage or cubic yards (cubic meters) removed. Royalties on most other commodities are generally based upon the amount received from sale, less certain expenses incurred prior to the time of sale. Defining the point at which such deductible costs arise is critical in preparing a royalty provision. It can be set at the mine mouth, thereby allowing deduction of transportation and treatment costs from that point on, or more commonly, at the outlet of a mill or other treatment facility. This arrangement is the traditional net smelter returns royalty where the royalty is based upon the amount paid for concentrates of a smelter less smelter treatment charges and deductions, and transportation costs incurred by the operator in moving the ore from the mill to the smelter. If the ore is sold to a mill, the same procedure applies except that transportation costs are those incurred between mine and mill.

It is important to the lessor that the lease provide that charges be competitive with those of other comparable mills or smelters if the lessee sells to a mill or smelter in which the lessor owns an interest (a non-arm's-length transaction). The same is true if the transportation is performed by the lessee.

In some instances, particularly as to precious metals, the lessor may wish to take a share of production in kind, rather than a percentage of proceeds.

If a large-scale mining operation is begun, it is desirable for the company to have exclusive title and not to be obligated to make indefinite payments out of production. For this reason, it is common practice to negotiate an end price for the lessee's entire interest. Often, if a sizable ore body is uncovered, the lessee simply will pay off the lessor before production begins.

The lease should contain a *force majeure* clause so that the lessee will be excused from certain obligations under the lease if he is prevented from performing because of events beyond his control.

The lessor will desire provisions requiring the lessee to comply with all applicable environmental protection and other laws and regulations, to indemnify the lessor for any claims by third parties, and to have adequate insurance coverage.

A termination clause also should be inserted permitting the lessee to terminate all obligations under the lease upon the giving of notice, usually 15 or 30 days. Conversely, the lessor should be given the right to terminate the lease upon breach by the lessee within a specified time after giving notice to the lessee requiring him either to comply with the contract or face termination.

Provisions allowing for commingling of ores and concentrates and for use of the leased lands in connection with mining on adjoining or nearby lands are desirable.

If the consideration is substantial, the lessor should be required to warrant his title. The warranty should extend to compliance with location, assessment work, and BLM filing requirements with respect to unpatented mining claims. In all cases, the lease should be reviewed for its tax implications and tailored to the needs of the parties involved.

Since the lessor is assessing the terms of the lease in light of the regulation and resources of the particular lessee, the lessor will desire that any assessment by the lessee be conditional upon the lessor's consent, not to be unreasonably withheld.

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193. *See* R. Krueger, *supra* at 472-73.
194. *Id.* at 451.
195. *See* 43 U.S.C. §§ 1337(i)-1337(j) (1982).
196. *See* 43 U.S.C. § 1337(k) (1982); 54 Fed. Reg. 2049-56 (to be codified at 30 C.F.R. pt. 281).
197. 43 U.S.C. § 1340 (1982).
198. *See* Comment, 1978 "The Deep Seabed Hard Mineral Resources Act and The Third United Nations Conference on the Law of the Sea: Can the Conference Meet the Mandate Embodied in the Act," *San Diego Law Review,* Vol. 18, pp. 509, 509 n.3; Gillis, R., 1981, "Exploration for and Exploitation of Deep Seabed Hard Mineral Resources: A Deep Seabed Mining Industry Perspective 44–47" (United Nations Ass'n of Conn., Proceedings of Conference on Deep Seabed Mining and Freedom of the Seas, 1981); Peters, W., 1978, *Exploration and Mining Geology,* p. 207.
199. *See* Larson, D., 1986 "Deep Seabed Mining: A Definition of the Problem," *Ocean Development and International Law,* Vol. 17, pp. 271, 277–78.
200. *Id.* at 272–74, 297–303; Biblowit, C., 1984 "Deep Seabed Mining: The United States and the United Nations Convention on the Law of the Sea," *St. John's Law Review,* Vol. 58, pp. 267, 268.
201. 30 U.S.C. §§ 1401-1473 (1982).
202. *See Id.* § 1403(4).
203. *Id.* § 1411.
204. *See id.* §§ 1413, 1417.
205. *See* 15 C.F.R. Part 970 (1988).
206. *Id.* § 970.208(b).
207. *Id.* § 970.102(c).
208. *See* 30 U.S.C. §§ 1412, 1417 (1982).
209. *See* 54 Fed. Reg. 514-48 (to be codified at 15 C.F.R. Part 971).
210. 54 Fed. Reg. 530 (to be codified at 15 C.F.R. § 971.208(b)).
211. *See* 30 U.S.C. §§ 1413, 1418 (1982); 54 Fed. Reg. 535, 537-38 (to be codified at 15 C.F.R. §§ 971.418, 971.503).
212. *See* 30 U.S.C. §§ 1441-1443 (1982).
213. *See* 30 U.S.C. § 1444 (1982). "This express disclaimer of any obligation to pay compensation varies greatly from earlier draft legislation which had provided for some sort of compensation or insurance in the event that the United States chose to ratify a Law of the Sea treaty which had a negative impact on U.S. seabed miners." Collins,

H., 1981, "Deep Seabed Hard Mineral Resources Act—Matrix for United States Deep Seabed Mining," *Natural Resources Lawyer,* Vol. 13, pp. 571, 577.

214. *See* R. Gillis, *supra,* at 46; C. Biblowit, *supra,* at 270, 303-05 (discussing whether the World Court would hold non-treaty mining in international waters legal or illegal).
215. *See* Comment, *supra,* at 529-30.
216. *See* R. Gillis, *supra,* at 46 n.7, 54.
217. *See* D. Larson, *supra,* at 271, 294; Kimball, L., 1986, "Turning Points in the Future of Deep Seabed Mining," *Ocean Development and International Law,* Vol. 17, pp. 367, 367-68, 376-77.
218. 30 U.S.C. §§ 601-04 (1982).
219. *Id.* §§ 601-04.
220. 43 C.F.R. § 3600.0-3(3) (1987).
221. *Id.* § 3601. l-l(a).
222. 36 C.F.R. § 228.41(b)(3) (1988).
223. 43 C.F.R. § 3610.2-1(a) (1987).
224. *Id.* § 3610-2-1(b).
225. *Id.* § 3610. 2(3). Materials not exceeding 200,000 tons (or weight equivalent).
226. *Id.* § 3610.1-5.
227. *See generally* Lonegran, J.B., 1969, "The Materials Act as a Solution to the Common Varieties Problem," *Rocky Mountain Mineral Law Institute,* Vol. 15, p. 51.
228. *American Law of Mining,* Vol. 1, § 21.02.
229. *See generally* Rees-Jones, T., 1962, "Problems in the Development of Mineral Resources on Indian Lands," *Rocky Mountain Mineral Law Institute,* Vol. 7, pp. 674-75; Gibbons, F. M., 1964, "Examination of Indian Mineral Titles," *Rocky Mountain Mineral Law Institute,* Vol. 10, p. 73; Burley, C.L., 1982, "Indian Lands—An Industry Dilemma," *Rocky Mountain Mineral Law Institute,* Vol. 27B, p. 1605; Moore, L.R., 1983 "Mineral Development on Indian Lands—Cooperation and Conflict," *Rocky Mountain Mineral Law Institute,* Vol. 28, p. 1.
230. 24 Stat. 391 (1887), 25 U.S.C. §§ 331 *et seq.* (1982).
231. Berger, E.B., 1968, Indian Mineral InterestsÑPotential for Economic Advancement," *Univ. of Arizona Law Review,* Vol. 10, p. 628.
232. 48 Stat. 984 (1934), 25 U.S.C. §§ 461 *et seq.* (1982).
233. Dick, R. A., 1955, "Indian Lands," *Rocky Mountain Mineral Law Institute,* Vol. 1, p. 65.
234. 58 Int. Dec. 331, 343 (1943).
235. 348 U.S. 272 (1955).
236. These principles were recently affirmed in United Indian Nation v. County of Oneida, 414 U.S. 661 (1974).
237. *American Law of Mining,* Vol. 3, § 67.02[2].
238. 34 Op. Atty. Gen. 18 (1924).
239. Berger, *supra,* p. 680.
240. 48 Stat. 984 (1934), *codified as* 25 U.S.C. §§ 461-86, as amended (1982).
241. 25 U.S.C. §§ 396a-f (1982).
242. 25 C.F.R. Part 211 (1988).
243. *Id* . §§ 21l.l0, 211.12.
244. *Id.* § 211.2, 211.3.
245. *Id.* § 211.27a.
246. 35 Stat. 783 (1909), codified as 25 U.S.C. § 396 (1982), as amended.
247. 25 C.F.R. Part 212 (1988).
248. *See generally* 1960, *American Law of Mining,* 1st ed., Vol. 1, Matthew Bender, New York, § 2.40, p. 280, Note 13.
249. Pub. L. No. 97-382, 96 Stat. 1938 (codified at 25 U.S.C. §§ 2101-2108 (1982).
250. *Id*. § 2102. *See generally American Law of Mining,* Vol. 3, § 67.05[2], p. 67-19.
251. *Id.* § 2103(b).
252. Quantum Exploration, Inc. v. Clark, 780 F.2d 1457 (9th Cir. 1986).
253. 43 U.S.C. §§1601 *et seq.* (1982).
254. ANCSA §§ 14(a), 14(b), 14(f), 43 U.S.C. §§ 1613(a), 1613(b), 1613(f) (1982). But see ANCSA § 19(b), 43 U.S.C. § 1618(b) (1982) (concerning conveyance of land within former reserves).
255. ANCSA § 14(e), 43 U.S.C. § 1613(e) (1982).
256. ANSCA §§ 12(a)91), 14(h)(9), 43 U.S.C. §§ 1611(a)(l), 1613(h)(9) (1982).
257. ANCSA § 19(b), 43 U.S.C. § 1618(b) (1976).
258. *Id.*
259. 43 U.S.C. § 291 (1982).
260. *Id.* § 299 (1982).
261. Visintainer Sheep Co. v. Centennial Gold Corp., 748 P.2d 358 (Co. Ct. App. 1987).
262. *See generally* Ary, T.S., and Morgan, P.J., 1969, "Problems of Access to Public Domain State and Fee Lands, From Shotgun to the Courthouse," *Rocky Mountain Mineral Law Institute,* Vol. 15, p. 481.
263. 43 C.F.R. § 3814(c) (1987).
264. 30 U.S.C. § 54 (1982).
265. Opinion of the Acting Solicitor, M-36417 (Feb. 15, 1957).
266. 30 U.S.C. § 611-15 (1982).
267. *Id.* § 612.
268. *Id.*
269. *Id .; American Law of Mining,* Vol. 1, § 21.03[2].
270. 30 U.S.C. § 613 (1982).
271. United States v. Etcheverry, 230 F.2d 193 (10th Cir. 1956); Teller v. United States, 113 F. 280 (8th Cir. 1901).
272. *American Law of Mining,* Vol. 3, § 60.01.
273. *Id.* § *62.02.*
274. *Id.* § 63.01.
275. *See generally* Brown, E.A., Jr., 1957 "Geophysical Trespass, " *Rocky Mountain Mineral Law Institute,* Vol. 3, p. 57.
276. *See generally American Law of Mining,* Vol. 3, Ch. 84.
277. *Id.* § 84.02[4].
278. *See* Craig, E., and Myers, M., 1978, "Ownership of Methane Gas in Coalbeds," *Rocky Mountain Mineral Law Institute,* Vol. 24, pp. 767-810; Cohen, H., 1984, "Developing and Producing Coalbed Gas; Ownership, Regulation, and Environmental Concerns," *Pace Environmental Law Review,* Vol. 2, pp. l–24; Comment, 1985, "Coalbed Gas Ownership in Pennsylvania—A Tenuous First Step with U.S. *Steel v. Hoge," Duchesne Law Review,* Vol. 23, pp. 735-51.
279. 503 Pa. 140, 468 A.2d 1380 (1983).
280. *See American Law of Mining,* Vol. 6, § 203.01[1].
281. A thorough review of joint operating agreements and suggested documentation is provided in *American Law of Mining,* Vol. 4, Ch. 141.

Chapter 3.3
HEALTH AND SAFETY STANDARDS

KENNETH P. KATEN

3.3.1 INTRODUCTION

Throughout the centuries of documented mining activities, perhaps the subject of most concern to the public pertains to safety aspects. However, in recent years, concerns for the environmental effects of mining and the health effects of industrial processes on the work force have equaled the long-standing concern for safety. These three facets, and the manner in which the mining community addresses each, constitute the primary sources from which the public derives its perception and image of the industry. Consequently, it is incumbent upon every member of the mining community, and particularly directors and managers of mining companies, to undertake their responsibilities in such a manner that will at all times include appropriate concern for these elements in their decisions and actions.

The purposes for discussing health and safety standards in a mining engineer's handbook are (1) to provide some historical perspective to the developments in these areas, (2) to outline the intent and requirements of applicable federal legislation, and (3) to provide some guidance in meeting the obligations and responsibilities that this legislation confers upon mining personnel. While focusing on those pieces of legislation that apply on a national level, it is beyond the scope of this work to address the many similar developments in each state in which mining takes place. However, adherence to the standards promulgated in these various states is equally important to compliance with the federal standards.

For further discussion of the subject of health and safety programs, see Chapters 11.0 and 11.1.

3.3.2 BACKGROUND OF HEALTH AND SAFETY DEVELOPMENTS

3.3.2.1 Statistics

Purpose: Although much has been written about mine health and safety, only in recent decades has there been a structured effort to accumulate meaningful data on accidents and injuries. Documents relating to this subject are numerous and varied in their approach to the topic. Most, however, share the common thread of analysis of safety statistics in order to demonstrate the necessity of understanding the scope of the safety situation before attempting to ascertain cause or source of accidents.

Safety experts recognize the value of accurate information in measuring the safety performance of a mine and the manner in which it relates to the occupations and operations therein. In addition to providing a basis for measuring safety performance, statistical information is valuable in isolating elements that are causative factors in accidents. This type of information, along with accident analyses, provides the basis for several approaches in furthering the reduction of accidents. For instance, with this type of data, engineers, supervisors, and workers can develop an awareness of hazards and develop the means to eliminate, reduce, or avoid them. Governmental agencies make use of statistical information to focus on areas in which research may offer a solution, or to develop an inspection strategy that will provide the most assistance in addressing identified hazards. Mining equipment manufacturers can design equipment to eliminate a particular hazard that has become apparent through statistical analysis of accidents related to their products. Application of this type of information is valuable in planning stages; for example, in the layout of the mine and facilities, design engineers can increase safety as well as economy by planning layout or location to remove hazards wherever possible rather than add protective safety equipment at some later time. Finally, training instructors can use statistical data to develop training strategies to address hazards or accident-causing procedures that are unique to or prevalent at a particular site.

Review: From late in the 19th century, records relate the most tragic accidents that occurred and the number of associated fatalities. However, it was not until the creation of the US Bureau of Mines in 1910 that consistent and comprehensive records became available. It is reasonable to assume that, prior to 1910, even without substantiation from accurate records, mine accidents probably claimed thousands of lives each year. Fig. 3.3.1 indicates the magnitude of the situation in the earlier part of the 20th century as well as the progress that has been made in the intervening years. The data in Figs. 3.3.2 through 3.3.5 delineate more precisely developments in the last 16 years based on injury rates that compare the fatalities and all injuries in the coal and metal/nonmetal sectors as a function of the number of exposure hours of miners.

Injury rates are generally expressed in two categories to provide an indication of the rate at which accidents occur and how serious in nature they may be. The frequency of occurrence, or *incidence rate* (IR), represents the number of injuries sustained in any particular category per 200,000 hr of exposure, that is,

$$IR = \text{number of cases} \times 200{,}000/\text{hr of exposure} \qquad (3.3.1)$$

The 200,000-hr exposure represents the number of hours that 100 miners would experience in an average work year.

Although most attention centers on incidence rates, the second category, *severity measure* (SM), provides an indication of the relative seriousness of injuries sustained at a site. This statistic derives from the number of days lost from work and days of restricted work activity, adjusted for the same amount of exposure as indicated above. Thus,

$$SM = \text{sum of days} \times 200{,}000/\text{hr of exposure} \qquad (3.3.2)$$

In the absence of a definitive number of work days lost due to a particular accident, especially in the case of a fatality or permanently disabling injury, the most widely accepted standard is the schedule provided by the American National Standards Institute (ANSI). In this standard, a fatal or permanent disabling injury carries an assignment of 6000 lost workdays, while other injuries that result in loss of limb or body parts carry a specified number of days (Table 3.3.1) to be charged regardless of the number of days that may have actually been lost. It is important to note that, in many cases, the actual days lost may vary significantly

Table 3.3.1. Tabulation of Scheduled Charges of Lost Workdays Caused by Injuries

A. For Loss of Member-Traumatic or Surgical

Fingers, Thumb and Hand

Amputation involving all or part of bone	Thumb	Fingers: Index	Fingers: Middle	Fingers: Ring	Fingers: Little
Distal phalange	300	100	75	60	50
Middle phalange	—	200	150	120	100
Proximal phalange	600	400	300	240	200
Metacarpal	900	600	500	450	400
Hand at wrist, 3000					

Toe, Foot, and Ankle

Amputation Involving All or Part of Bone	Great Toe	Each of Other Toes
Distal phalange	150	35
Middle phalange	—	75
Proximal phalange	300	150
Metatarsal	600	350
Foot at ankle, 2400		

Arm

Any point above elbow, including shoulder joint	4500
Any point above wrist and at or below elbow	3600

Leg

Any point above knee	4500
Any point above ankle and at or below knee	3000

B. Impairment of Function

One eye (loss of sight), whether or not there is sight in the other eye	1800
Both eyes (loss of sight), in one accident	6000
One ear (complete industrial loss of hearing), whether or not there is hearing in the other ear	600
Both ears (complete industrial loss of hearing), in one accident	3000
Unrepaired hernia	50

Source: American National Standards Institute (ANSI Z16.1 1967, R1973)

from this standard, but the standard enables a comparison of the severity of accidents from site to site.

The trend in the number and rate of fatal accidents indicated in Fig. 3.3.1 is fairly clear upon analysis and, though there is some minor year-to-year deviation, shows consistent improvement. The same can be said for accidents that result in disabling injuries, although advances in medicine and treatment may claim some credit for this reduction. Due to the nature of these types of accidents, these figures tend to be more reliable than the number and rate of nonfatal and nondisabling accidents, the recording of which varies with interpretation of reporting requirements.

3.3.2.2 Health and Safety Laws

Intent: As with the passage of any statute, the theoretical intent of statutes relating to mine health and safety is remedial in nature. Public concern for the well-being of the nation's miners has prompted Congress to act on several occasions during the course of this century, beginning in 1910 with the formation of the Bureau of Mines (Public Law or P.L. 61-89). This legislation formed the first federal governmental body to become directly involved with the safety of miners. It also included provisions for investigation of accidents, research in the areas of mineral extraction and safety, teaching accident prevention, and training for first aid and mine rescue. Public Law 77-49 in 1941 provided for safety inspections of mines and gave federal inspectors the right of entry in order to conduct these inspections. In 1947, Public Law 80-328 created safety standards for bituminous coal mines, but had no enforcement provisions and expired after one year.

The year 1952 saw enactment of the first Federal Coal Mine Safety Act, which encompassed all underground coal mines employing more than 15 people. This act provided for enforcement of federal standards under a state plan and mandated annual inspections, delineated safety standards, and provided for issuance of orders of withdrawal in instances of imminent danger. The passage in 1961 of P.L. 87-300 authorized study of the causes and prevention of injuries and health hazards in metal and nonmetal (M/NM) mines and included provisions for access to information by federal officials. Public Law 89-376 extended the coverage of the 1952 law to small mines and included language for the withdrawal of miners in cases of repeated and unwarrantable failures to comply with safety standards. The Federal Metal and Nonmetallic Mine Safety Act of 1966 established annual inspections for this sector of the industry and established a procedure for the development of additional standards. Shortly following its passage came the Federal Coal Mine

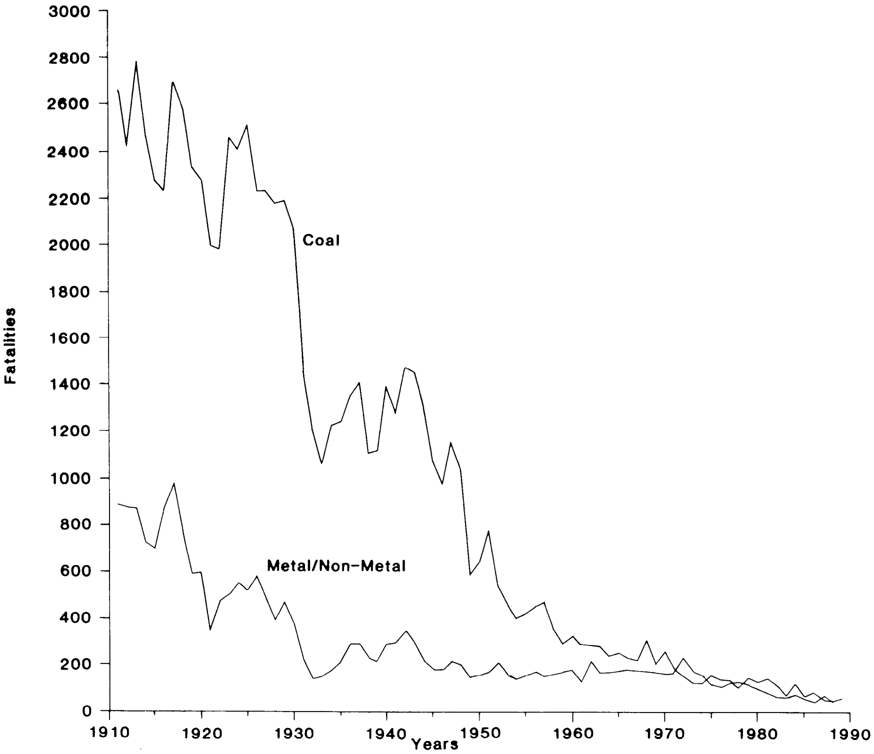

Fig. 3.3.1. Fatal injuries in the US mining industry, 1911-1988. *(Source:* Mine Safety and Health Administration.)

Fig. 3.3.2. Coal fatality rates, 1973-1988. *(Source:* Mine Safety and Health Administration.)

Health and Safety Act of 1969 (Anon., 1969) that included surface mines in its coverage and eliminated state-plan enforcement of standards. Noteworthy in this piece of legislation were mandated annual inspections, civil penalties for violations, and criminal penalties for knowing and willful violations. This act was then amended in 1977 (P.L. 95-164) to place coal, metal, and nonmetal mines under a single law, to eliminate advisory standards in the metal/nonmetal sector, and to mandate miner training.

Provisions: Viewing the intent of any legislation as curative or remedial, one can derive the sense or purpose of the various provisions as elements that, taken in sum, will provide guidance

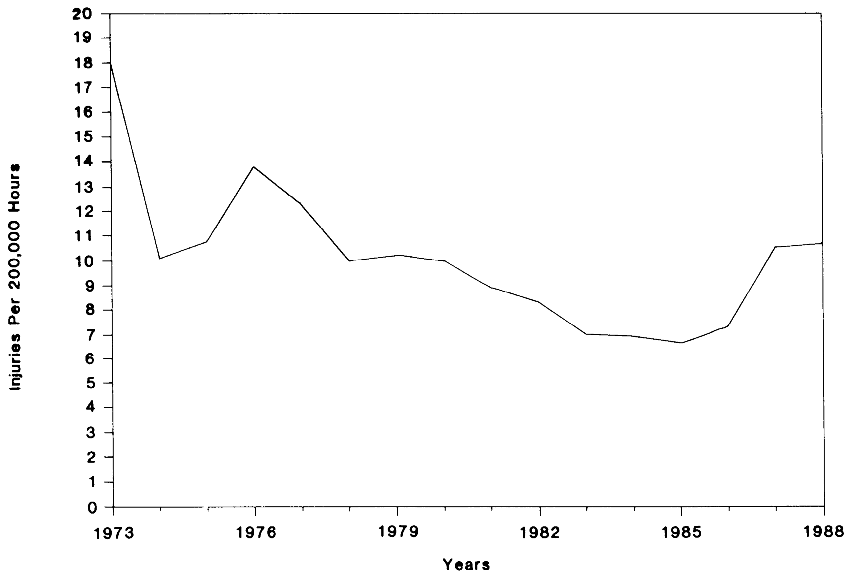

Fig. 3.3.3. Coal all-injury rates, 1973-1988. *(Source:* Mine Safety and Health Administration.

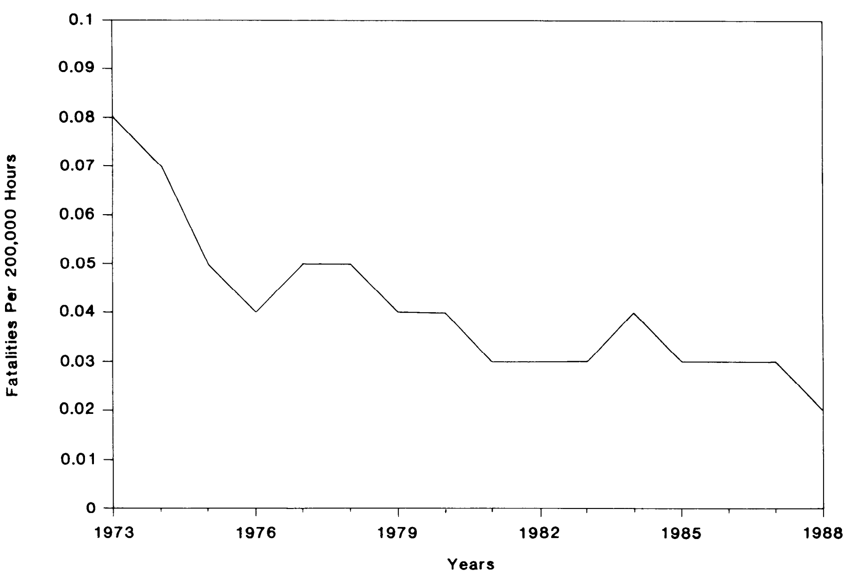

Fig. 3.3.4. Metal/nonmetal mine fatality rates, 1973-1988. *(Source:* Mine Safety and Health Administration.)

as to how we as a society feel that we should address a given situation. It is in this light that we view the provisions of the predominant piece of health and safety legislation, namely, the Federal Mine Safety and Health Amendments Act of 1977 (Anon., 1977), hereinafter referred to as "the Act."

Developing health and safety standards plays a central role in addressing the respective aspects of mining. In the statement of findings and purpose of this Act, Congress specifically stated in Section 2(g) that one of the purposes of the Act was to establish interim health and safety standards, to direct the Secretaries of

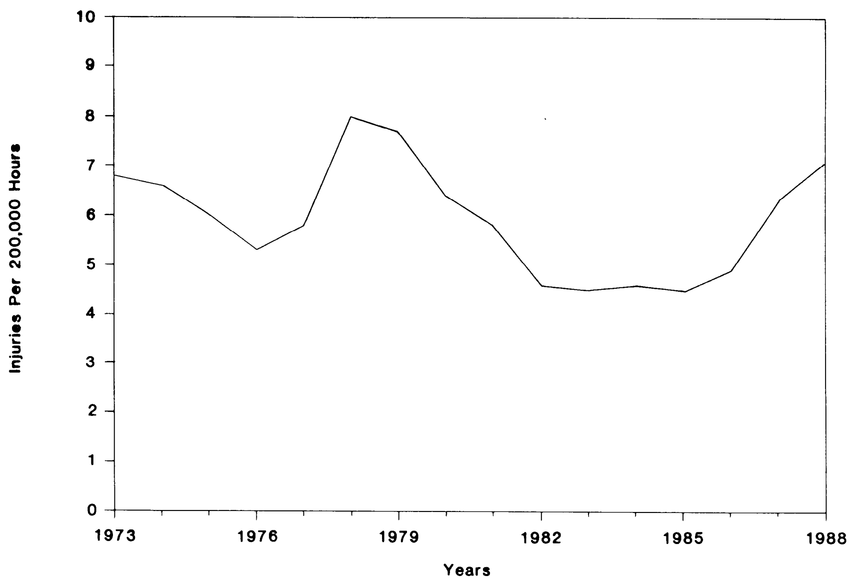

Fig. 3.3.5. Metal/nonmetal mine all-injury rates, 1973-1988. (Source: Mine Safety and Health Administration.)

Labor and of Health, Education and Welfare to develop improved standards, to cooperate with the states in development and enforcement of other mine health and safety programs, and to improve and expand research and development as well as training programs.

In Title I of the aforementioned legislation are provisions for development, promulgation, or revision of the interim standards for the protection of life and prevention of injuries in coal or other mines. Since a reading of this portion of the Act provides the details and procedures for these actions, it would be redundant to review them in this text. In general, these provisions charge the Mine Safety and Health Administration (MSHA), as agent for the Secretary of Labor, to develop improved standards in matters of mine health and safety and invest it with the rule-making authority to do so.

Title II of the Act contains Interim Mandatory Health Standards for all underground coal mines. Regulations contained in Title 30 of the *Code of Federal Regulations,* 30 CFR (Anon., 1988), implement this portion of the Act and contain these interim standards as well as those which have been since promulgated for surface coal mines and all metal/nonmetal mines. Although similar in many respects, health regulations for coal mines and metal/nonmetal mines are separate, as indicated in Table 3.3.2. This table lists the various parts of Title 30 and the respective area that each part addresses.

Interim mandatory safety standards for underground coal mines are contained in Title III of the Act, and Part 75 of 30 CFR implements these statutory provisions. Regulations promulgated since passage of the 1969 Act dealing with surface coal mines and surface areas of underground mines comprise Part 77 of the Code. Parts 56 and 57 of this same code contain the standards for metal/nonmetal surface and underground mines, respectively. In addition to these mandatory health and safety standards, Title 30 contains regulations that address the many related health and safetv topics, as indicated in Table 3.3.2.

3.3.2.3 Regulations and Standards

Types:

Mandatory—Many of the regulations contained in the aforementioned parts of 30 CFR are mandatory and provide no latitude in their observance, except as specified by paragraph 101(c) of Title I of the Act, with procedures outlined in 30 CFR, Part 44, Rules of Practice for Petitions for Modification of Mandatory Safety Standards.

Although there has been significant discussion in recent years regarding the concept of performance-oriented standards, most of the present standards are specific in their wording. Standards that are performance-oriented state the goal to be achieved by the standard and allow mine operators to use whatever techniques or means of compliance that best suit the circumstances of mining situations. An example of such a standard would be that of a coal mine operator maintaining the airborne dust at a level not to exceed a concentration of 2.0 mg/m^3 in order to prevent the development of coal workers' pneumoconiosis (CWP). However, present standards include this requirement, in addition to the specifically stated requirements for minimum air quantities and velocities, line brattices, distances of line brattices from the working face, cross-sectional areas available for air passage on the return-air side of the brattices, requirement of a specified water pressure, number of water sprays, etc.

Performance-oriented and specifically worded standards each have advantages and disadvantages. An advantage of performance-oriented language is that mine operators have greater latitude to employ the techniques that best suit the particular operation and are most economical. The disadvantage is that it may be difficult to determine whether any such method does, in fact, achieve the goal of the regulation at all times.

Specifically worded standards have the disadvantage of limiting all mining operations to the same approach regardless of

Table 3.3.2. Title 30—*Code of Federal Regulations* Health and Safety Topics

Part(s)	Subject
11	Respiratory protective devices
15-17	Explosives and related articles
18-29	Electrical equipment, lamps, methane detectors
31-36	Mechanical equipment for mines
40	Representative of miners
41	Notification of legal identity
43	Procedures for processing hazardous conditions complaints
44	Rules of practice for petitions for modification of mandatory safety standards
45	Independent contractors
46	State grants for advancement of health and safety in coal and other mines
47	National Mine Health and Safety Academy
48	Training and retraining of miners
49	Mine rescue teams
50	Notification, investigation, reports and records of accidents, injuries, illnesses, employment, and coal production in mines
56	Safety and health standards, surface metal and nonmetal mines
57	Safety and health standards, underground metal and nonmetal mines
70	Mandatory health standards, underground coal mines
71	Mandatory health standards, surface coal mines and surface work areas of underground coal mines
74	Coal mine dust personal sampler units
75	Mandatory safety standards, underground coal mines
77	Mandatory safety standards, surface coal mines and surface work areas of underground coal mines
90	Mandatory health standards, coal miners who have evidence of the development of pneumoconiosis
100	Criteria and procedures for proposed assessment of civil penalties

Source: Code of Federal Regulations, Title 30, 1988.

the circumstances that exist at the various sites. This approach tends to limit technological developments and provides little latitude to mine operators. On the other hand, standards of this nature are usually easily measured or observed and have the benefit of being "tried and proven" in most situations.

Many of the existing regulations, in addition to being specifically worded, are based on the ability to measure some parameter(s) in order to ascertain the performance of the mining company in complying with the standard. This approach relies on measurement-based standards to inform mine operators of compliance responsibilities in exact terms. An example of this type of standard is the requirement that electric transformers installed on the surface be totally enclosed, or be at least 8 ft (2.4 m) aboveground, or be surrounded by a fence at least 6 ft (1.8 m) high and at least 3 ft (0.9 m) from any energized parts. This type of standard not only has the advantages of providing mine operators with guidance as to what constitutes a safe installation, but frequently provides alternative means of achieving the safety goal. Safety inspectors find that this type of standard is most easily overseen because of the absolute nature of the measurement that can determine compliance or lack thereof.

Although measurement-based standards are becoming more prevalent and may be desirable in some cases, there are many situations that require the use of experience or judgment in determining what constitutes not only compliance with a particular regulation, but, more importantly, what will provide the proper measure of safety to personnel. Equipment-guarding standards are examples of experience or judgment-based regulations. For example, wording of these standards generally requires guarding of exposed moving parts that may be contacted by persons and that may cause injury.

Advisory—The bulk of regulations for the mining industry is mandatory in nature, but there are some that can be categorized as advisory and are generally labeled as criteria or are worded in such a manner that the verb "should" replaces the verb "shall" in the wording of the standard. The purpose of this type of regulation is to provide guidance or options for mine operators in determining the means of compliance to achieve the desired safety goal.

Development of Regulations and Standards: Existing regulations for the metal/nonmetal and coal sectors have developed from several sources, but in most cases derive their origin from interim standards. In the case of the regulations governing metal/nonmetal mines, advisory committees composed initial standards that were later published as proposed rules and finalized after public hearings and comment, as prescribed in the Administrative Procedures Act of 1946 (5 U.S.C., 1976) (Anon., 1976). The coal regulations, having evolved over a greater period of time, were part of the interim standards in the 1969 and 1977 Acts and continue to be revised with advances in technology and experience with their use and administration.

The Administrative Procedures Act outlines the framework for formal rule-making as it applies to the development of all regulations. In addition to this, Title I of the 1977 Mine Health and Safety Act provides language and direction regarding regulations that pertain specifically to the area of mine health and safety.

The procedures in each of these statutes include, as a minimum, publishing the proposed rule in the *Federal Register,* a public comment period, opportunity for any member of the public to request a public hearing, and publication of the final rule in the *Federal Register*. In the 1980s, MSHA added an earlier stage by publishing an advance notice of proposed rule-making (ANPRM) to provide for more participation from the mining community prior to the development of a proposed rule.

Elements that comprise standards in each regulation are derived from diverse sources, but most have an engineering origin, especially those standards that are measurement-based. Accident statistics, known or accepted safety and engineering standards, epidemiological data, experience, and data from research or specific studies are all used as sources in developing standards that are incorporated in proposed rules.

In most cases, safety regulations are structured in such a manner as to provide several "lines of defense" for a particular hazard in order to prevent a catastrophic development should one facet or line of defense fail. Viewing the Act and implementing regulations in this manner, we can ascertain the tiers of safety for any one of several situations. For example, in preventing methane explosions in underground coal mines, the first and most obvious means of prevention includes provisions for separation of intake and return airways, minimum air quantities for adequate dilution, and specified levels of tolerance for methane to prevent the accumulation of explosive amounts. A second line of defense, accompanying these methods of prevention, constitutes eliminating man-made sources of ignition, such as prohibition of smoking and requiring electrical equipment to be approved by MSHA as permissible for use in gassy atmospheres. Thirdly, training and certification requirements for supervisory, inspection, and electrical maintenance personnel (certified persons) ensure that those persons entrusted with the inspection of conditions and maintenance of electrical equipment are knowledgeable about detecting hazards and proper means of prevention. Regular instruction for machine operators (qualified persons) who check for methane during the course of the performance of their work provides further protection against dangerous accumulations of methane.

Should there be a set of circumstances in which a face ignition occurs, additional "lines of defense" serve to eliminate or mitigate personal injuries. For example, rock-dusting requirements reduce the potential of initiating or propagating an ignition beyond the immediate area of the face. Regulations governing the designation and maintenance of escapeways provide a means of egress in the event of a development that exceeds the capability of the personnel in any area of the mine.

Mine Plans—Recognizing the uniqueness of individual mining operations, Congress made provision in the interim mandatory safety standards of the Act for mine operators to develop safety plans for various situations. MSHA has generally continued this approach in regulations that it has subsequently developed.

The mine-plan concept allows personnel at individual mines to develop safety plans for the design, construction, and operation of safety systems and procedures that take into consideration the mining conditions and equipment that are peculiar to the site. Once an operator has developed a plan and MSHA has approved it, that plan becomes the established regulation for the facet of the operation that the plan addresses.

There is great merit in and reliance placed upon the mine-plan approach to safety regulation, particularly in the coal sector. Since this approach allows mine personnel to develop the safety plan which is most appropriate for the mining conditions and equipment at a site, the plan can better address site-specific hazards and better protect the miners exposed to those hazards. In recent years, however, there has been a trend to make the plans more generic or broad-based and less unique to each operation. Though this development may have the benefit of providing an easier means by which to oversee compliance, it has the disadvantage of imposing requirements that may not be as appropriate as those that would pertain to site-specific hazards.

Examples of plans that are commonly compiled for coal mines include slope and shaft sinking, ground control, roof and rib control, ventilation system and methane and dust control, training, fire fighting and escape and evacuation, smoking articles search, self-rescuer storage, mine rescue, and emergency assistance and transportation. In each of these cases one can see the need for adaptation to local or peculiar situations that a generalized mandatory standard would find difficult to address.

Although the metal/nonmetal sector does not have the breadth of involvement in developing these sorts of plans, there is involvement in the development of training plans that can address the site-specific nature of hazards.

Variances—As mentioned earlier, Congress provided means by which mine operators could request exception to or variance from a mandatory safety standard. This sole exception allows the Secretary of Labor, that is, MSHA, to modify the application of any mandatory safety standard upon petition by a mine operator, if one of two tests can be met. (Noteworthy is the fact that there are no provisions for variances from mandatory health standards.) The first test is that the alternative or proposed method of achieving the safety result will at all times guarantee no less than the same measure of protection afforded miners by the existing standard. This approach evidently recognizes the fact that emerging technology could outpace the development of safety regulations and that, in some cases, alternatives will provide at least the same degree of, if not greater, safety to the miners.

The second criterion or test allows alternative means of complying with mandatory standards if application of a particular mandatory standard would result in a diminution of safety to the miners. Here, again, it appears that Congress recognized the fact that a standard that generally promotes safety may result in a lessening of safety to miners when applied in some situations.

In developing a petition for modification of a safety standard, an operator can receive guidance from Part 44 of 30 CFR that outlines the procedures for this approach. When considering filing a petition for modification, mine operators should consider the fact that they bear the burden of showing that the proposed alternative meets one of the two tests. Petitioning for modification of a safety standard is historically a lengthy process and involves a thorough study of the situation and proposed alternative by MSHA before issuance of a decision.

Enforcement: Regulatory oversight in the mining sector ranks among the most intensive in modern society. In passing the 1977 Act, Congress specifically charged the Secretary of Labor, that is, MSHA, to "make inspections of each underground coal or other mine in its entirety at least four times a year and of each surface coal or other mine at least two times a year." These inspections are supplemented by many others that depend on the type and condition of a mine, safety or health complaints that may be filed by miners, allegations of discrimination, accidents, previous citations, and several others.

Enforcement actions stemming from an inspection vary and may range from issuance of a single civil penalty (104-a of the Act) citation to closure orders for imminent-danger situations (107-a) or for unwarrantably failing to comply with regulatory provisions (104-d). Regardless of the type of enforcement action, the violative condition must be abated within the period of time specified in the citation. Failure to abate a violative condition within the timeframe specified in a 104-a citation may result in a 104-b closure order that will remain in effect until the situation is rectified.

Citations are civil actions that require correction of the violative condition within the abatement period specified in the citation and payment of a monetary penalty, as outlined in the 30 CFR Part 100 criteria.

Orders, however, require cessation of work activities in the affected area and immediate abatement of the condition. Furthermore, these enforcement actions usually result in greater civil penalty amounts. In the case of either a citation or order, should the violative condition result from knowing or willful violations of safety or health standards, managers may be subject to individual civil or criminal penalties in addition to the aforementioned corporate civil penalties.

Should a citation or order be issued as a result of an inspection or investigation, mine operators have the option of discussing the enforcement action with MSHA officials in an informal conference. Further options include paying the amount of the civil penalty or contesting the citation or amount of the proposed penalty. These formal contests begin with a hearing before an administrative law judge and can proceed, upon appeal by either party, as far as the Supreme Court.

3.3.3 HEALTH AND SAFETY ORGANIZATIONS/ AGENCIES

3.3.3.1 Governmental

Although MSHA and the various state agencies are perhaps the highest-profile organizations involved with health and safety, they are by no means the only ones in existence. On the national level, in addition to MSHA, are the Occupational Safety and Health Administration (OSHA), the US Bureau of Mines (USBM), the National Institute for Occupational Safety and Health (NIOSH), and the National Institutes of Health (NIH).

Although not all these organizations conduct inspections or enforce regulations per se, those that do not are active in conducting research and surveys that serve as underlying bases for development of new or revised standards and regulations.

State mining agencies play various roles, depending on the respective statutory requirements. Their roles generally include, but are not limited to, inspections, training, mine rescue, and mine accident analysis. In general, despite some overlap, the national and state governmental agencies complement each other's activities to the general benefit of all miners.

3.3.3.2 Nongovernmental

In addition to the federal and state health and safety inspectorates, there are numerous other entities whose primary interest is the health and safety of mining personnel. These organizations, many voluntary, contribute significantly to the health and safety efforts on a grass-roots level and are of extreme value in the public comment stage of the rule-making process. Many mining companies have safety departments to oversee safety systems and procedures and, frequently, conduct inspections in the same manner as governmental inspectors. Insurance companies that underwrite coverage for mining activities employ safety inspectors or consultants to ascertain conditions and practices at a prospective client's operations and regularly inspect the insured operations to determine whether those operations observe appropriate safety measures on a regular basis.

In addition to organizations whose activities pertain to inspections, there are a number of associations whose primary or subsidiary purpose is the health and safety of miners. The National Safety Council, the American Conference of Governmental Industrial Hygienists, the Society for Mining, Metallurgy, and Exploration, Inc., the American Society of Safety Engineers, the Mine Inspectors Institute of America, the Holmes Safety Association, and the Smokeaters Association are perhaps the best known, but there are many other state, local, academic, and manufacturing organizations that are equally dedicated to the health and safety of mining personnel.

3.3.4 ECONOMICS OF HEALTH AND SAFETY STANDARDS

Recent data indicate that the cost of accidents is a major factor in the economics of many mining operations. Cost elements directly included in or attributable to lost-time accidents include the immediate loss of production due to the accident itself and rendering of first aid, wage compensation, medical treatment and hospital care, benefits for injuries resulting in death or disability, investigations of fatal or serious accidents, and loss of production associated with these investigations. These costs for fatal, disabling, and lost-time accidents are included in Figs. 3.3.6 through 3.3.9.

Other costs associated with safety may not be attributable to accidents, nor readily apparent or calculable. Among these are civil penalty assessments, lost revenue associated with closure orders, the cost of legal appeals and tort liability litigation, loss or damage of equipment, and post-accident reduction of productivity levels due to replacement of injured miners with less-experienced personnel.

It is equally important to recognize that the employer is not the only economic victim as a result of an accident. If a miner is injured, in most cases there is decreased primary income accompanied in some cases by physical impairment and attendant inability to enjoy normal activities. Additionally, there is the possibility of decreased future primary earning power as well as possible secondary financial losses.

Correlation of accidents, injuries, and property damage indicates that for each fatal, disabling, or lost-time injury, there are 10 accidents resulting in minor injuries, 30 cases of property damage, and 600 near misses. This information underscores the fact that, in addition to the concern for the well-being of personnel, it is in the best interest of all parties to conduct operations safely to help ensure the economic viability of the producing company and its employees.

3.3.5 ASPECTS OF HEALTH AND SAFETY STANDARDS

3.3.5.1 Engineering

In the industrial sector, and particularly in mining, the uncompromising need for a safe workplace cannot be overstated. As mentioned earlier, redundancies of safety systems are routinely used in order to prevent catastrophe in the event of malfunction of any one system or line of defense.

In the construction of safety-system requirements and procedures, perhaps the most prevalent aspect is engineering. Most of the standards that comprise the regulations derive their origins from some aspect of the engineering sciences. For example, understanding ventilation requires knowledge of hydraulics and fluid flow; roof control, the properties of materials, and beam theory; electrical safety, the basics of electrical power generation, distribution, and protective circuitry; and fire prevention and fighting, the elements of combustion theory. Indeed, the modern engineer has the challenge, opportunity, and responsibility to participate fully in establishing safety regulations that are based on sound engineering theory. Furthermore, engineering validation of many standards can help ensure their credibility and subsequent compliance by all members of the mining community.

3.3.5.2 Training

In addition to the need for a safe workplace, studies indicate that training has a tremendous opportunity to produce safety dividends. These same studies indicate that more than 80% of accidents and injuries are the result of unsafe acts or omissions on the part of personnel. This in no way should be seen as an indictment of any segment of the industry, but rather as evidence of the need for and value of timely and relevant training.

Present mining regulations, as articulated in 30 CFR Part 48, require initial safety training of miners before they assume their duties as well as annual retraining in general safety practices and procedures. Additionally, before miners embrace a new task or operate a new piece of equipment, the employer must instruct each such employee in the various safety and operational aspects of the soon-to-be-assumed responsibility.

A common industry practice that provides regular reminders consists of daily or weekly "tool-box" training. Although not required by regulation, this form of training usually takes place at the beginning of a work shift. Using this approach, a supervisor, member of the safety department, or member of the work crew reviews safety regulations or practices that apply to the particular work site. Presentations may vary and may include such approaches as a group discussion of circumstances relevant to the work site, a review of accident information and discussion of preventative measures, and a demonstration of procedures for particular work tasks or dealing with unusual geologic conditions.

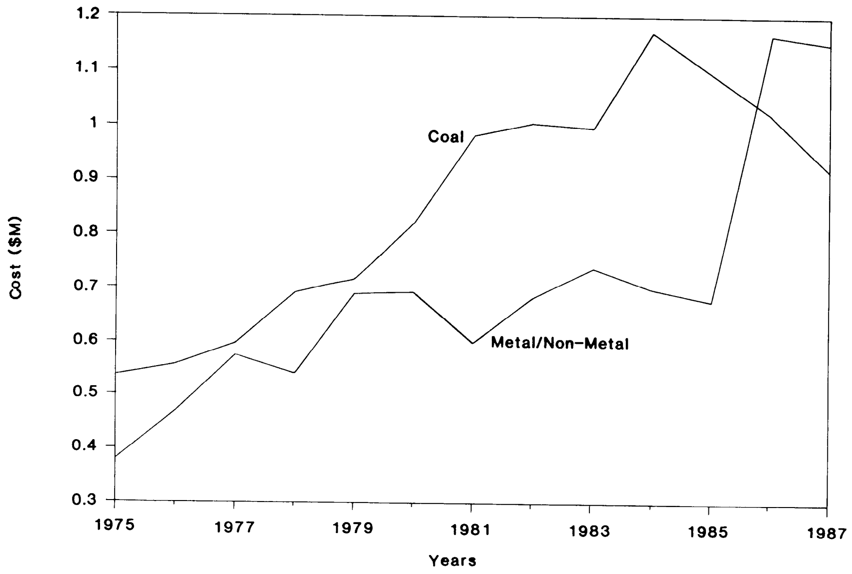

Fig. 3.3.6. Average cost per fatality in underground mines. *(Source:* US Bureau of Mines.)

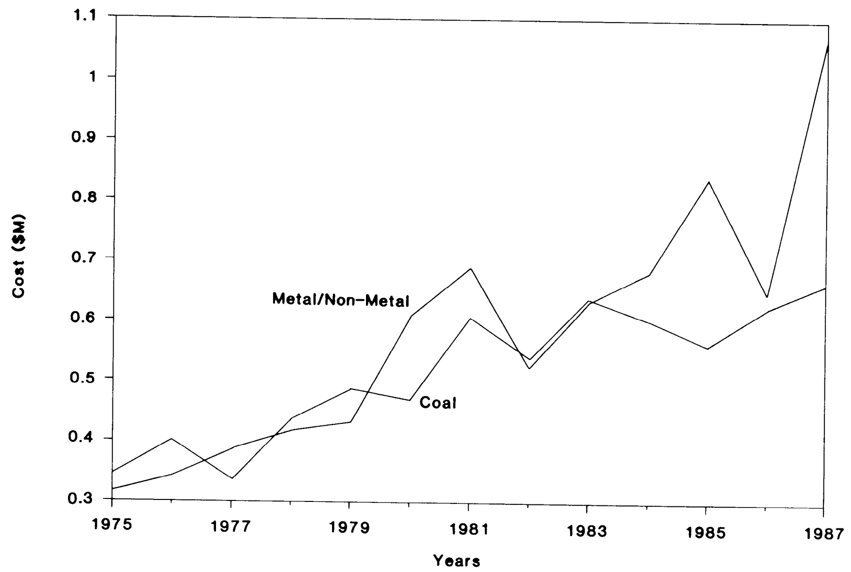

Fig. 3.3.7. Average cost per fatality in surface mines. *(Source:* US Bureau of Mines.)

The dialogue and feedback that result from interactive training are most beneficial to the overall safety effort. Not only will training sessions of this nature educe greater interest on the part of the participants, but will promote greater safety awareness and improve morale long after the class has ended.

3.3.5.3 Maintaining Safe and Healthful Conditions

Systems installed to provide a safety or health benefit must occasionally be inspected, calibrated, or maintained in some manner to ensure their proper function. Inspections to ensure

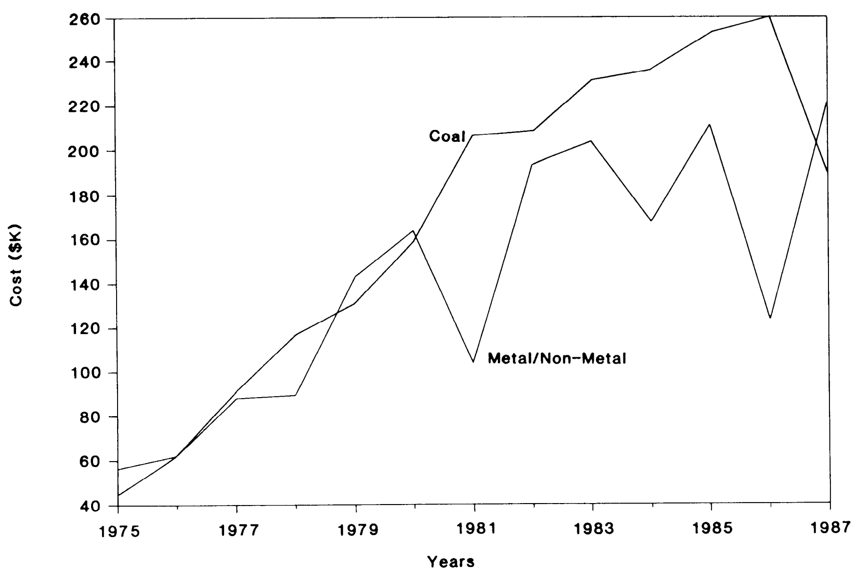

Fig. 3.3.8. Average cost of permanent disability in underground mines. *(Source:* US Bureau of Mines.)

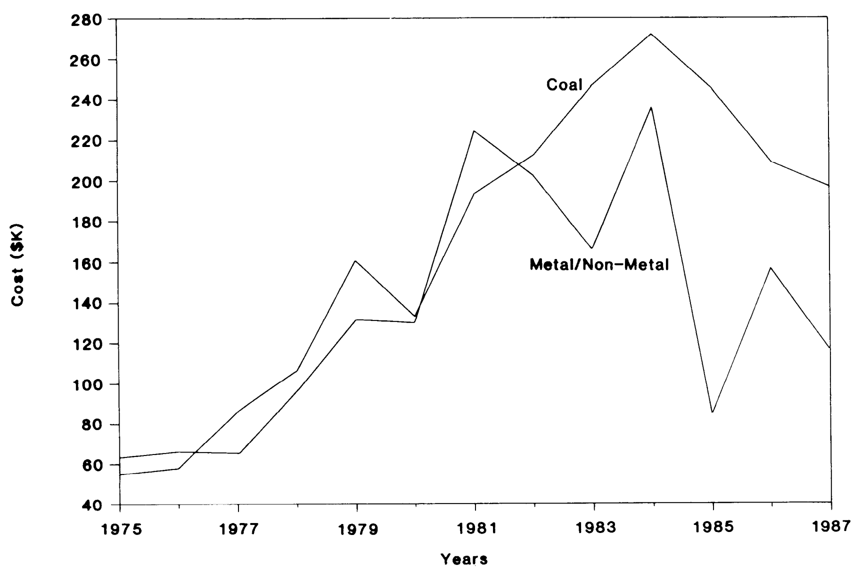

Fig. 3.3.9. Average cost of permanent disability in surface mines. *(Source:* US Bureau of Mines.)

safe working conditions and equipment are basic elements in safety programs. The fact that inspections comprise such an important role in maintaining a safe working environment is good reason for their inclusion in regulations as mandatory requirements. These types of requirements include examinations of pre- and on-shift conditions, escapeways, ventilation equipment, fire-fighting and suppression equipment, electrical equipment, hoisting equipment, etc. Where examination and maintenance of equipment is involved, equipment manufacturers often recommend the intervals at which these functions are to be performed, but the majority are outlined in mandatory health and safety standards.

The basic construction of these types of regulations provides for examination, testing or calibration, and subsequent reporting or record-keeping on the part of the person responsible for conducting the activity. Along with the activity itself, development of a record can allow personnel to establish or discern trends that will provide information for preventative or remedial action. So important are some of these records that failure to report or record, or falsifying a document provides the basis for a civil penalty citation and, possibly, criminal indictment of the responsible person(s).

Along with the examination and reporting of a condition comes the requirement and, indeed, responsibility to take the appropriate corrective actions to remedy any detected unsafe or unhealthful situation. Remembering the intent of the safety legislation as being remedial in nature, it is incumbent upon mine operators to take whatever action is necessary and appropriate to ensure the health and safety of their employees.

3.3.6 CONTROL OF HAZARDS

3.3.6.1 Primary Goal of Standards

Inasmuch as the stated purpose of safety legislation is "to protect the health and safety of the Nation's coal or other miners" (P.L. 95-164, 1977) (Anon., 1977), one must view the goal of the resultant regulations as fulfilling this intent. Consistent with this philosophy, in its statement of purpose, the Act also requires the Secretary of Labor "to develop and promulgate improved mandatory health or safety standards" (Anon., 1977) to achieve the goal.

Due to the nature of the rule-making process, development of such standards is not only a lengthy process but also one filled with controversy. For instance, in many cases there is sincere disagreement about what constitutes an "improved" standard. Further, does improving a standard necessarily involve converting from experience- or judgment-based to measurement-based standards to remove subjectivity from the compliance determination decision? In order for the rule-making process to have accurate, documented, and applicable information, it requires input from responsible elements in the community. What better source of information than the mining engineers who comprise the technical/engineering aspect of mining and who are trained to understand and recognize the many implications of health and safety requirements?

3.3.6.2 Means of Control

Whether a hazard be a safety or health concern, mine operators are ultimately responsible for its correction or abatement. In many situations, curative action is outlined in the regulations and thus leaves little room for disagreement as to whether any action was, in fact, sufficient to correct or abate the condition. Regardless of the number and detail of published regulations, however, the sheer number of hazardous situations that could arise renders the construction of prescribed actions impossible. Therefore, it is left to the mine personnel to assess any such situation and determine the appropriate course of action to pursue in its remedy.

The means of control that could be utilized to effect abatement of unsafe or unhealthful conditions consist of three primary categories: engineering controls, administrative or procedural controls, and personal protective devices.

Where engineering controls are involved, the mining engineer obviously has the greatest opportunity to participate in, and perhaps influence, the decision-making process about how best to proceed. For example, in providing proper ventilation, diluting and removing methane or other unwanted gases, and controlling respirable dusts, selection of equipment and configuration of the delivery system are paramount considerations. Although regulations require powered fans, prescribe minimum quantities of air, and establish threshold levels of action for various unwanted gases, it is left to the mining company to determine the type and size of equipment as well as the design of the ventilation system. Further, once having selected and installed safety equipment, it falls to the mine operator to effectively maintain it in such a manner that it will fulfill its function. Similar situations exist in the areas involving ground control, electrical systems, materials handling, fire prevention, use of explosives, etc.

Procedural controls are categorized with administrative actions in that they constitute some type of action on the part of the miner or supervisor. In this sense, they consist of learned actions or responses that are a function of training. With the great variation in types and kinds of mining equipment that are used in the industry from region to region, mine to mine, and even within the same mine, it is imperative that personnel be instructed in the proper and safe use of that which is applicable to the particular work site.

The third means of control that can provide safety and/or health protection consists of personal protective devices. In contrast to engineering controls that require no or little action or choice on the part of the individual, these devices, as indicated by their classification, require the individual to wear or make use of the device in the proper manner in order to derive the intended safety or health benefit. Although many safety experts prefer engineering controls, there are situations in which personal protective devices provide additional or necessary protection, especially in cases where technology does not yet provide an engineering solution or where exposures are of short term or on infrequent bases.

3.3.7 FIRST-AID AND MINE-RESCUE STANDARDS

Recent decades have held numerous accounts of mine disasters around the world that focused on the rescue efforts for survivors or recovery efforts of victims and the mine itself. In each of these cases, volunteers who comprise the rescue teams conduct these rescue efforts, with oversight or coordination from company and governmental personnel.

Perhaps less heralded than the mine-rescue teams, but equally important at every mine site, are the first-aid personnel who perform not only in disaster situations but also on a more routine basis. Although transportation and communication facilities are more capable of expeditiously rendering formal medical attention than in previous decades, the ability to provide first aid remains a vital requirement at every mine.

In order to ensure that these capabilities are available and current at each mine, MSHA has developed regulations to address these items. The regulations governing mine-rescue team availability, equipment, and training constitute Part 49 of 30 CFR, while Part 48 contains the first-aid training requirements. In some states, mining laws require the presence of an emergency medical technician (EMT) at the mine site to ensure first-aid capability at all times.

As a result of these regulations, we in the United States have several means of providing first-aid and mine-rescue capability. Individual mining companies remain the most prevalent source of trained mine-rescue teams, but several states' regulations require that the state itself maintain teams at various locations where they will be within an hour's traveling time of the mines in that particular state. Private companies, whose primary role consists of mining-related health and safety efforts, also constitute a source of trained rescue teams. These types of companies usually contract their mine-rescue services to mining companies in their locale.

3.3.8 CONCLUSION

Through developments in technology, the evolution of health and safety legislation and regulations, and a greater awareness of the need for health and safety considerations, mine safety improvements in the United States in the 20th century have been appreciable. These improvements are most noteworthy in the reduction of personal injuries and the achievement of more healthful workplaces.

Recognizing these developments is an important step in providing the philosophical approach to a constantly improving level of achievement. However, it is only with the commitment of all participating entities to maintain safe and healthful workplaces and to carry out activities in accordance with trained procedures that we will continue to improve our performance and safety record and prevent adverse occupational health exposures.

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Chapter 3.4
ENVIRONMENTAL CONSEQUENCES

L. MICHAEL KAAS AND CLAYTON J. PARR

3.4.1 INTRODUCTION

L. MICHAEL KAAS

The development, operation, and ultimately the closure of any mining and mineral processing facility are profoundly influenced by the environmental laws and regulations promulgated by government organizations at the federal, state, and local levels. The purpose of this chapter is to provide a review of the evolution of these laws and regulations at the federal level that has occurred in recent years and its effect on the mineral industry.

The body of environmental laws and regulations is large and complex. For example, the Office of Surface Mining, Reclamation, and Enforcement (OSMRE) regulations implementing the Surface Mining, Reclamation, and Control Act of 1977 (SMCRA) comprise nearly 700 pages in the *Code of Federal Regulations* (CFR). Existing environmental laws are frequently amended and significant new laws are passed by the Congress. Regulations are constantly changing as witnessed by the hundreds of pages of new or revised regulations that are proposed or finalized weekly by regulatory agencies. These regulations are published daily in the *Federal Register* (FR). Finally, the environmental laws and regulations are subject to legal interpretation and influenced by thousands of court decisions.

While this chapter provides a useful introduction to environmental law for the minerals engineer, it is not intended to be a substitute for competent legal counsel. The old adage that a physician who treats himself has a fool for a patient may have a direct parallel in the field of environmental law. The engineer who tries to be his own environmental lawyer may have a fool for a client.

References to environmental protective measures may be found elsewhere in this *Handbook* (e.g., Chapters 7.3, 12.1, 12.2, and 12.3 and Section 11).

3.4.1.1 Rise of Environmental Awareness

Few aspects of minerals resource development have experienced changes as drastic and rapid as those in the field of environmental regulation. These changes reflect the increased public awareness of environmental issues during the 1960s. That awareness emerged in response to many and varied influences such as the earlier conservation movement, the publication of Rachel Carson's *Silent Spring,* and the euphoria that followed the successes in the US space program. This awareness ultimately led to the first Earth Day, Apr. 22, 1970.

The 1960s have been called the "Decade of the Environment." We became convinced as a nation that we could develop whatever technology might be necessary to protect and restore the environment and that we could afford whatever the costs might be. Indeed, we simply could not afford not to take whatever precautions that might be necessary to protect the environment on Spaceship Earth for generations to come.

The crowning achievement of the 1960s was the passage of the National Environmental Policy Act of 1969 (NEPA). This act stated that it was national policy "to create and maintain conditions under which man and nature can exist in productive harmony, and fulfill the social, economic, and other requirements of present and future generations of Americans."[1] NEPA laid the groundwork for the decade of the 1970s, which could be called the "Decade of Environmental Law." During the decade, numerous landmark federal environmental statutes were passed by Congress and transformed into regulations by their implementing agencies. The most prominent of these laws were the Clean Air Act of 1970, Federal Water Pollution Control Act of 1972 (known as the Clean Water Act), Resource Conservation and Recovery Act of 1976, Surface Mining Control and Reclamation Act of 1977, Federal Land Policy and Management Act of 1976, Endangered Species Act of 1973, Toxic Substances Control Act of 1976, Clean Air and Water Act Amendments of 1977, Uranium Mill Tailings Radiation Control Act of 1978, and Archaeological Resources Protection Act of 1979. Numerous other environmental statutes were passed by Congress and state and local governments.

The growth of environmental laws and regulations continued throughout the 1980s reflecting the continued broadbased political support for the national commitment to protection of human health and the environment. Major statutes passed during the 1980s included the Comprehensive Environmental Response, Compensation, and Liability Act of 1980 (also called Superfund), Superfund Amendments and Reauthorization Act of 1986, and Water Quality Act of 1987 (amendments to the Clean Water Act).

The strength and support for the environmental movement was underestimated by some early skeptics. Today there is no doubt about the high level of national commitment to a cleaner and healthier environment. The tremendous nationwide response on the 20th anniversary of Earth Day, Apr. 22, 1990, demonstrated the strength of that commitment. That concern has reached global proportions with the realization that protection of air and water quality frequently transcends political boundaries.

3.4.1.2 Impact of Mineral Development on the Environment

There is no question that the development of mineral resources does impact the environment. With proper precautions, these impacts can usually be eliminated or can be minimized in terms of severity or duration. However, for nearly two centuries, this nation developed its mineral resources with few environmental controls. Many of the mineral industry's harshest critics have delighted in pointing out the legacy of environmental disturbances created by past mining and mineral processing activities. In Appalachia, over 5000 miles (8000 km) of streams are polluted by acid drainage from abandoned coal mines (Fig. 3.4.1.1). Thousands of unreclaimed coal strip mines dot the landscape in coal country (Fig. 3.4.1.2). Hundreds of thousands of acres (hectares) of land are underlain by abandoned mine workings and face subsidence hazards. Over 50 billion tons (45 Gt) of solid waste have been produced from hard-rock mining, primarily in the western states (Fig. 3.4.1.3). Dozens of identified sites containing hazardous waste from metal mining and processing

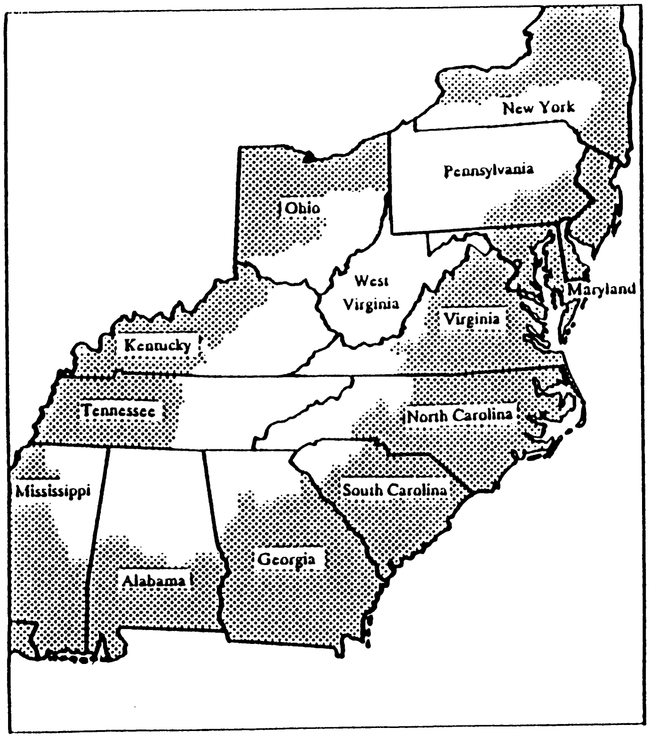

Fig. 3.4.1.1. Coal mine drainage contaminates streams in an extensive area of Appalachia.

operations pose myriad problems ranging from soil and water contamination (Fig. 3.4.1.4) to airborne particulate and toxic gas emissions. Cleanup of many sites with the most onerous of these problems is being conducted under the Environmental Protection Agency's Superfund program and OSMRE's Abandoned Mine Land Reclamation Program.

It is easy for critics of mineral resource development to forget that most of these adverse impacts predate the current laws and regulations. Today's modern mines and processing plants have adopted new technologies, sometimes at great expense, that will help insure that future generations will not have to face many of these same environmental problems. However, it is also important to realize that much still needs to be learned about the potential for long-term adverse environmental impacts such as those associated with mineral waste materials or about shorter-term impacts of some newer production methods. For example, the rapid expansion of heap and dump leaching methods utilizing large quantities of acid or sodium cyanide for recovery of copper and precious metals has led to public concern for the prevention of groundwater contamination and wildlife kills. The increased use of full-extraction coal mining methods such as longwall mining has triggered other concerns relating to land disturbance and aquifer damage caused by subsidence. These concerns and others are the targets of emerging regulatory and legal developments that will place new controls on disposal of mining and beneficiation wastes, give added protection to groundwater resources, and control airborne toxic substances.

And yet, as the world population grows and standards of living improve, the worldwide demand for minerals will continue to increase. The nation, and indeed the world, will continue to deplete high-grade, shallow mineral resources. Deeper and lower-grade mineral deposits will be mined. The new mining methods and new processing techniques required for exploitation of these resources could pose new types of environmental disturbances. Improved environmental technology will be needed to insure that potential environmental hazards can be avoided and that the goals of producing necessary minerals and maintaining environmental quality can be achieved.

3.4.2 MAJOR FEDERAL ENVIRONMENTAL LAWS AND REGULATIONS

L. MICHAEL KAAS

3.4.2.1 National Environmental Policy Act of 1969 (Public Law 91-190)

The National Environmental Policy Act of 1970 serves as the fundamental national charter for environmental protection. The purposes of the Act are "to declare a national policy which will encourage productive and enjoyable harmony between man and his environment . . . and to establish a Council of Environmental Quality."[2] Although short in length when compared with other major environmental statutes, NEPA has had a profound impact on the governmental decision-making process. The Act's importance stems from its requirement that all agencies of the federal government must "include in every recommendation or report on proposals for legislation and other major Federal actions significantly affecting the quality of the human environment, a detailed statement by the responsible official on the environmental impact of the proposed action."[3] Although mineral resource development is traditionally the domain of the private sector, the need for permits from federal agencies or the involvement of federal lands means that it is unlikely that many mining projects will escape the requirement for preparation of an *environmental impact statement* (EIS). One major exception to this is the area of mineral exploration, which does not require an EIS. The regulations that have been promulgated under NEPA begin in Part 1500 of Title 40 of the *Code of Federal Regulations* (40 CFR § 1500).

Title I declares that it is national environmental policy for the federal government "to use all practicable means and measures, including financial and technical assistance, in a manner calculated to foster and promote the general welfare, to create and maintain conditions under which man and nature can exist in productive harmony."[4] The following requirements are placed on all federal agencies:

1. Utilize a systematic and interdisciplinary approach in planning and in decision making that may have have an effect on the environment.
2. Develop methods and procedures that insure that environmental values are considered in the decision-making process.
3. Prepare environmental impact statements of proposed federal government actions.
4. Develop alternatives to recommended courses of action in any proposal involving unresolved conflicts.

Title II of the Act creates the Council for Environmental Quality (CEQ) in the Executive Office of the President. The principal functions of CEQ are to advise the President on government programs and policies that affect the environment and to issue regulations for implementation of NEPA by the federal agencies. The most significant of these regulations pertain to the EIS process.

Environmental Impact Statement Process: Because of the likelihood of most new mining projects having "significant ef-

Fig. 3.4.1.2. Unreclaimed strip mines pose environmental and public safety hazards.

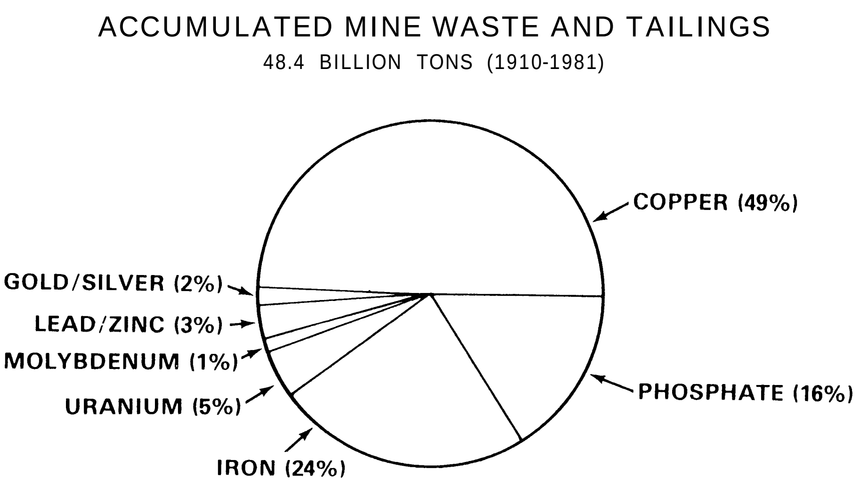

Fig. 3.4.1.3. Over 50 billion tons (45 Gt) of hard-rock mining and beneficiation waste has accumulated during this century.

fects" on the quality of the environment and involving some type of "major Federal action," it is important to understand the EIS process.[5] It is equally important to begin planning at an early point in the life cycle of the project for eventual EIS preparation by the federal agencies involved. Failure to do so may result in a longer and/or more costly approval process.

NEPA states that the EIS is to be prepared by the agency, not the applicant for a permit. To facilitate compliance with the EIS requirement, most agencies have specialized staffs in place to prepare the EIS and administer the process. When an agency is in doubt about whether or not an EIS is required, it may elect to have the applicant prepare, at his expense, an *environmental*

Fig. 3.4.1.4. The Yak Tunnel, part of the California Gulch Superfund Site, drains acid, metal-laden water from the Leadville, CO, mining district.

assessment (EA).[6] The EA usually involves a very comprehensive data-gathering effort including baseline environmental analyses. The list of data types is extensive, including geologic, hydrologic, botanical, zoological, archaeological, historical, cultural, sociological, and economic. If the agency ultimately decides that an EIS will be required, it will utilize data collected during the EA in its preparation.

It is not uncommon for several federal agencies to be involved in the approval of a mining project. In such cases, the agencies can agree jointly to prepare the EIS. In this case, one of the agencies may be designated as the lead agency with supervisory responsibility for EIS preparation. At the request of a lead agency, any other federal agency with special expertise may be asked to be a cooperating agency. A scoping process is used to insure that the significant issues of concern to the participating agencies are identified and their scope determined.

When a number of major actions are expected to occur in a given geographic area, such as during the opening of a new mining district, the agency or agencies involved may choose to prepare a regional EIS. This approach enables the cumulative impact of the expected actions to be considered and may result in a more efficient process overall. As a result, it is possible that some actions included in the region may not require a separate EIS unless they involve a project or site-specific impact not considered in the overall regional statement.

The format of the EIS is prescribed in the CEQ regulations. A brief description of the format follows:

1. *Cover Sheet.* A list of the responsible agencies including the lead agency and cooperating agencies. Title of the proposed action. A one-paragraph abstract. Designation of the statement as a draft, final, draft supplement, or final supplement. Name, address, and telephone number of a contact for additional information. Date by which comments must be received.

2. *Summary.* An accurate summary of the EIS including major conclusions, areas of controversy, and issues to be resolved.

3. *Table of Contents.*

4. *Purpose and Need.* A brief discussion of the purpose and need to which the agency is responding by preparation of the EIS.

5. *Alternatives Including the Proposed Action.* The main section of the EIS. Presents the environmental impacts and alternatives in detail and in a comparative form. Includes a no-action alternative. Sharply defines the issues and provides a clear choice among the options. Identifies the agency's preferred alternative.

6. *Affected Environment.* Describes the environment of the affected area at a level of detail necessary to understand the alternatives. Presents data and analyses commensurate with the magnitude of the impacts.

7. *Environmental Consequences.* Describes the environmental impacts of the alternative courses of action, including the proposed action. Provides the scientific and analytic basis for comparisons of alternatives. Describes the relationship between short-term uses and longer-term productivity. Discusses irreversible or irretrievable commitments of resources involved in the proposal.

8. *List of Preparers.*

9. *List of Agencies, Organizations, and Persons to Whom Copies of the Statement Are Sent.*

10. *Index.*

11. *Appendices.* Supplementary material prepared in conjunction with the EIS (optional).

After preparation of the draft EIS has been completed and before the final EIS is prepared, the agency must obtain comments from any federal agency that has jurisdiction by law, has special expertise in any environmental impact, or is authorized to develop and enforce environmental standards. Comments must also be sought from any state or local agency authorized to develop environmental standards. When the effects of the proposed action may be on an Indian reservation, the tribe's comments must be obtained. At least 45 days must be allowed for comments. The length of the comment period may be extended by the agency.

Diligent efforts must also be made to involve the public in the EIS process[7] and to obtain public comment. Public notices must be provided for hearings, public meetings, and the availability of documents such as the draft EIS. When a proposed action is of local concern, notices are sent to state and area-wide clearinghouses, community organizations, and interested persons, and published in local newspapers and through other local media. Agencies must hold public hearings or meetings whenever appropriate. These hearings are a particularly important part of the EIS process if the proposed action is—as most mining projects are—controversial.

Company strategies for public involvement vary from its active solicitation to its discouragement. However, there appears to be growing sentiment favoring a proactive position with the public. Successful permitting of a number of highly controversial minerals projects has been attributed to the companies actively seeking early public involvement so that citizens' concerns are identified. Once the public's concerns are identified, they frequently can be satisfactorily addressed in the EIS process and through public education activities.

Completed draft and final EISs are filed with the Environmental Protection Agency (EPA) which publishes a weekly notice of their availability in the *Federal Register* (FR). No decision on a proposed action can take place until 90 days after publication of the notice for a draft EIS or 30 days for a final EIS. These deadlines can run concurrently.

Once comments have been received, the draft EIS is modified to address the comments. The agency must respond to each comment. The comments and the agency's response are included in an appendix to the final EIS.

When the permitting agency has reached its decision, it must prepare a concise public Record of Decision (ROD).[8] The ROD must clearly state the decision. It must also identify the alternatives considered, stating which one(s) were considered environmentally preferable. Finally, it must state whether all practicable means have been taken to minimize environmental harm.

More detailed discussion of the EIS process is contained in Chapter 7.3.

3.4.2.2 Surface Mining Control and Reclamation Act of 1977 (Public Law 95-87)

The Surface Mining Control and Reclamation Act of 1977 was the first piece of legislation specifically aimed at elimination of environmental problems associated with mining. These problems included "destroying and diminishing the utility of land for commercial, industrial, residential, recreational, agricultural, and forestry purposes, by causing erosion and landslides, by contributing to floods, by polluting the water, by destroying fish and wildlife habitats, by impairing natural beauty, by damaging the property of citizens, by creating hazards dangerous to life and property, by degrading the quality of life in local communities, and by counteracting governmental programs and efforts to conserve soil, water, and other natural resources."[9]

After nearly a decade of legislative conflict among industrial, environmental, and governmental organizations, SMCRA was signed into law on Aug. 3, 1977. Since its passage, the Act has subsequently been amended many times by other pieces of legislation. It created a nationwide framework for regulating active surface coal mines and the surface effects of underground coal mining (Title V).[10] It includes detailed environmental protection standards and reclamation requirements, establishes a state or federal permitting and enforcement system, and restricts mining in certain areas or under certain circumstances. SMCRA provides a means for reclaiming previously mined areas where little or no reclamation had been carried out prior to the Act (Title IV).[11] To pay for the cost of this reclamation effort, the Abandoned Mine Reclamation Fund was established, and taxes on both surface and underground coal production were created to generate needed revenues. The Office of Surface Mining, Reclamation, and Enforcement, a unit of the Department of the Interior, was created to implement the Act (Title II).[12] OSMRE was to assist the coal-producing states in the development of approved regulatory programs and to provide oversight on those programs once established. The regulations that have been promulgated under SMCRA begin in Part 700 of Title 30 of the *Code of Federal Regulations* (30 CFR § 700).

SMCRA was passed at the very time the nation was recovering from a serious energy crisis. There was widespread recognition that mining of the extensive US coal reserves would be necessary to meet a significant portion of future energy requirements. The concern of the environmental community as reflected in the Act was that future coal production would be conducted with ". . . appropriate standards to minimize damage to the environment the productivity of the land and to protect the health and safety of the public."[13]

Since its passage, implementation of SMCRA has been fraught with seemingly endless court challenges, Congressional hearings, and protests by environmental interest groups. Much of this conflict has stemmed from the rigid requirements that force the use of specific reclamation practices without regard to regional differences and that allow only limited flexibility for innovative reclamation approaches and subsequent land use.

State Programs: Any state in which there are or may be coal exploration or surface coal mining and reclamation activities on nonfederal and non-Indian lands may assume exclusive jurisdiction or "primacy" over their regulation.[14] To gain this authority, the state must have an approved regulatory program. If a state chooses not to conduct such a program or if its program is not approved or is withdrawn, OSMRE must promulgate and implement a federal regulatory program for that state. Currently, 23 states have approved programs, and 11 have federal programs.[15] Similarly, under Title IV, the states must also submit a state reclamation plan for administering an abandoned mine land reclamation program. A synopsis of each state program or the federal program applicable to that state is given in the *Code of Federal Regulations.*

Federal Lands: Title V requires that the Secretary of the Interior promulgate and implement a regulatory program for federal lands except Indian lands.[16] When federal lands are located in a state with an approved state program, the federal program is, at a minimum, to include all of the requirements of the approved state program. States with approved state programs may enter into cooperative agreements with OSMRE to assume regulatory authority over surface coal mining on federal lands within the state.

Indian Lands: Title VII requires the Secretary of the Interior to study the regulation of surface mining on Indian lands so that all surface coal mining on these lands complies with the requirements of the Act.[17] A federal program has been established for the Indian lands. OSMRE, the regulatory authority, consults with the Indian tribes, the Bureau of Indian Affairs, and the Bureau of Land Management on various aspects of the program.

Lands Unsuitable for Mining: Title IV requires that each state establish a planning process that designates state and private land areas as unsuitable for mining.[18] Similarly, federal lands may be designated unsuitable. Individuals also have the right to petition the appropriate regulatory authority to have an area designated as unsuitable. Once an area has been designated as unsuitable, OSMRE cannot issue a mining permit involving that area.

States must designate lands as unsuitable if postmining reclamation is not "technologically and economically feasible."[19] In addition, lands may be determined unsuitable if mining would "be incompatible with existing state or local land use plans or programs;"[20] damage "important historic, cultural, scientific, and esthetic values;"[21] "result in a substantial loss or reduction of long-range productivity of water supply or of food or fiber products;"[22] or "could substantially endanger life or property."[23]

The Act states that federal lands may be designated unsuitable for mining as a result of a land review. Federal lands within the National Park System, National Wildlife Refuge Systems, National System of Trails, National Wilderness Preservation System, Wild and Scenic Rivers System, and National Recreation Areas are designated as unsuitable. With some exceptions, mining is also prohibited in national forests. Mining is prohibited if it would adversely effect any publicly owned park or place included in the National Register of Historic Sites. In addition, it is not permitted to mine within 100 ft (30 m) of the outside of the right-of-way of any public road or cemetery or within 300 ft (91 m) of any occupied dwelling or public building, school, church, or community or institutional building.

Coal mining can be permitted within land designated as unsuitable if the mine operator has valid existing rights such as where "substantial legal and financial commitments"[24] in such an operation were in existence prior to Jan. 4, 1977. The concept of valid existing rights is to prevent taking of a person's property that would entitle the person to just compensation under the 5th and 14th Amendments to the Constitution of the United States.

Permitting: Title V of the Act establishes a permitting system for all surface coal mining operations.[25] Permits are issued by the regulatory authority for five years with the right of successive renewal. Longer-term permits may be issued under certain circumstances. Permits terminate after three years if mining has not been started.

A permit application along with the application fee are submitted to the appropriate state or federal program office. The application requires extensive information identifying the applicant, operator, and any property owners or lease holders. Owners of adjacent properties are also identified. The applicant's prior, existing, and pending mining permits must be included, and any suspensions, revocations, or bond forfeitures during the last five years must be explained. The applicant must also file a listing of any violations of the Act and any air and water quality environmental protection violations associated with coal mining. Additional information includes details of the mining plans, a description of the probable hydrologic consequences, climatological data, maps, cross sections, and drill logs of the coal deposit, and a soil survey if mining will involve prime farmland. The applicant is also required to place a public notice of his application in a local newspaper for four weeks. The advertisement must clearly state the location of the proposed mining operation and the location where the application is available for public inspection.

A reclamation plan, blasting plan, and certification of public liability insurance must also be submitted by the applicant for a mining permit. The reclamation plan identifies the location of the land proposed for mining, the uses of the land at the time of the application, any previous mining history, the capability and productivity of the land to support other uses, and the proposed use of the land after mining and reclamation. The detailed reclamation procedures, cost estimate, and timetable must be provided. Steps taken to protect surface water and groundwater systems and to comply with air and water quality and health and safety laws and regulations must be stated.

Performance Standards: Title IV sets forth a list of minimum environmental protection performance standards that are applicable to all surface coal mining and reclamation operations.[26] General standards require such operations to:

1. Maximize the utilization and conservation of the coal resource so as to minimize future land disturbance.
2. Restore the land to a condition capable of supporting premining uses or higher or better uses.
3. Restore the approximate original contour of the land.
4. Stabilize surface areas and spoil piles to control erosion and air and water pollution.
5. Remove, segregate, protect, and replace topsoil or other strata shown to be more suitable for supporting vegetation.
6. Restore the topsoil or the best available subsoil for supporting vegetation.
7. Utilize special handling for the various soil horizons present in prime farmland.
8. Create approved water impoundments that are stable and that protect water quality.
9. Conduct augering to maximize coal recovery, sealing auger holes to prevent drainage.
10. Minimize disturbances to the prevailing hydrologic balance on and off the mine site.
11. Construct stable and revegetated surface disposal areas for mine wastes, tailings, and coal processing wastes.
12. Refrain from mining within 500 ft (152 m) of active or abandoned underground mines to prevent breakthroughs.
13. Design, operate, maintain, and abandon coal waste disposal areas in accordance with standards and criteria.
14. Treat, bury, or dispose of acid-forming, toxic, or combustible waste materials to prevent water contamination or spontaneous combustion, and develop contingency plans to prevent sustained combustion,
15. Use explosives in accordance with existing state and federal laws and regulations.
16. Proceed with reclamation efforts in an environmentally sound manner and as contemporaneously as practicable with mining operations.
17. Construct and maintain access roads so that erosion, siltation, water pollution, damage to fish and wildlife or their habitat, or damage to public or private property will be prevented.
18. Refrain from road construction that will seriously alter the flow of streams or drainage channels.
19. Revegetate disturbed lands with diverse, effective, and permanent species native to the area.
20. Assume responsibility for successful revegetation for a period of 5 full years after revegetation work is completed and for 10 full years in areas or regions with 26 in. (660 mm) or less annual average precipitation.
21. Protect offsite areas from slides or damage occurring during the mining and reclamation operations.

22. Place excess spoil material in such a manner to assure stability and proper drainage and prevent erosion.

23. Meet other criteria to achieve reclamation, taking into consideration physical, climatological, and other site characteristics.

24. Use the best technology currently available to minimize disturbances and adverse impacts on fish, wildlife, and environmental values.

25. Provide a natural barrier to slides and erosion.

Title IV also establishes minimum performance standards for underground coal mining operations.[27] Many of the performance standards are identical to those just described for surface mining. The Act does provide that regulations should consider the distinct differences between surface and underground coal mining. The general standards that are unique to underground mining are summarized:

1. Prevent subsidence causing material damage to the extent technologically and economically feasible, except where the mining technology used requires planned subsidence in a predictable and controlled manner.

2. Seal openings between the surface and underground workings when they are no longer needed for operations.

3. Fill or seal exploratory holes and return, to the maximum extent technologically and economically feasible, mine and processing waste to the mine workings.

4. Protect offsite areas from damages resulting from mining operations.

5. Eliminate fire hazards and other hazards to public health and safety.

6. Locate openings of drift mines in acid- or iron-producing coal seams so that they prevent gravity discharge of mine water.

The regulations implementing the performance standards set forth in the Act are not included here because of their length and detail. The reader is directed to the appropriate sections of the *Code of Federal Regulations* for the specific standards that apply to his or her situation. Permanent performance standards are provided for coal exploration activities,[28] surface coal mining and reclamation activities,[29] and underground mining activities.[30] The regulations also provide special performance standards for auger mining,[31] anthracite mines in Pennsylvania,[32] operations in alluvial valley floors,[33] operations on prime farmland,[34] mountaintop removal,[35] special bituminous coal mines in Wyoming,[36] coal preparation plants not located within the permit area of a mine,[37] and in situ processing.[38]

Performance Bonds: Under Title V, once an application has been approved but before the permit has been issued, the applicant must provide a performance bond.[39] The bond provides an incentive for the applicant to complete the required reclamation. The amount of the bond is determined by the regulatory authority and is based on the reclamation requirements. It must be sufficient to assure completion of the reclamation plan if the work had to be performed by the regulatory authority in the event of forfeiture. The minimum amount of the bond for the area under one permit is $10,000.

Release of the performance bond requires that reclamation be completed along with the period of time for which the mine operator is responsible for successful revegetation as specified in the performance standards.[40] To obtain release of a bond, the operator must file a bond-release application and advertise the application and the location of the mining operation in a local newspaper for four weeks. He must also send letters of notification to adjacent landowners and local authorities. Copies of these letters and the advertisement are included with the application for release. The regulatory authority will then inspect the reclamation work and determine if the performance bond or any part of it is to be released.

Enforcement: Inspections and fines for violations are the major enforcement tools provided by the Act. Inspections of the mining operation are conducted on an irregular schedule with a partial inspection occurring monthly and a full inspection occurring quarterly.[41] In addition to inspections, the operator is required to keep appropriate records, submit monthly reports to the regulatory authority, and make the records available to inspectors and the residents of the mining area. If an inspection identifies a violation of the Act that poses imminent danger to the health and safety of the public or could cause significant and imminent environmental harm, a cessation order will be issued. A notice of violation will be issued if the violation does not pose an imminent danger to the public or the environment. Both cessation orders and notices of violation must describe the violation, the remedial action required, and the time limit for correction of the violation.

Failure to comply with the terms of the mining permit or violations of any other provision of Title V of the Act results in fines not to exceed $5000 for each violation. Each day of continued violation may be considered a separate violation. The operator may request a hearing prior to the assessment of the fine. Any person who willfully and knowingly violates a condition of a permit or fails or refuses to comply with any cessation order can be punished by a fine of not more than $10,000, or by imprisonment for not more than one year, or both. A pattern of repeated violations or failure to comply with the requirements of the permit may result in either suspension or revocation of the permit.

3.4.2.3 Resource Conservation and Recovery Act of 1976 (Public Law 94-580)

Management of solid wastes, including those generated by the minerals industry, is regulated under the authority of the Resource Conservation and Recovery Act of 1976 (RCRA). RCRA was designed to provide "cradle-to-grave" management of land disposal of solid waste to protect health and the environment. Along with the Clean Air Act and the Clean Water Act, Congress viewed RCRA as the third critical environmental law needed for protection of the air, land, and water. The regulations that have been promulgated for solid waste management under RCRA begin in Part 240 of Title 40 of the *Code of Federal Regulations* (40 CFR § 240).

RCRA uses a very broad definition of *solid waste* that includes "any garbage, refuse, sludge from a waste treatment plant, water treatment plant, or air pollution control facility and other discarded material, including solid, liquid, semisolid, or contained gaseous material, resulting from industrial, commercial, mining, and agricultural operations, and from community activities."[42] Regulations further define solid waste to include any "discarded material" that is not excluded by regulation or by a specific variance granted by the EPA or an authorized state.[43] A "discarded material" is defined as any material that is abandoned, recycled or "inherently wastelike."[44] A material is "abandoned" if it is disposed of, burned, or incinerated, or accumulated, stored, or treated prior to or in lieu of abandonment.[45] A material is a solid waste if it is "recycled" in a manner constituting disposal, by burning for energy recovery, by reclamation, or by speculative accumulation.[46] The only type of recycled materials that are clearly not solid wastes are those directly "used or reused as ingredients in an industrial process to make a product" or "used or reused as effective substitutes for commercial products," or those recycled in a closed-loop manner "to the original process from which they are generated without first being reclaimed."[47] A material is "inherently wastelike" if the EPA so defines it by regulation.[48] These interlocking definitions of solid waste result in the EPA regulating a universe of materials, some of which may not be commonly understood to be wastes in a particular industry or company.

Solid waste management facilities are described under an equally broad definition and include "(A) any resource recovery system or component thereof, (B) any system, program, or facility for resource conservation, and (C) any facility for the collection, source separation, storage, transportation, transfer, processing, treatment or disposal of solid wastes including hazardous wastes, whether such facility is associated with facilities generating such wastes."[49]

The result of these broad definitions is that essentially all mining, minerals processing, and materials recycling operations fall under the jurisdiction of the Act. Some waste materials commonly found in minerals production operations such as used oils, solvents, and machine shop wastes are already regulated under RCRA. Waste streams associated with mining and beneficiation operations such as mine overburden, mine waste rock, leaching residues, and mill tailings are not currently regulated. A regulatory program for these wastes is under development by the EPA with implementation expected in the mid-1990s.

The EPA has categorized those production stages that take place after beneficiation as *processing.* Typical mineral processing operations include smelting and refining. Most wastes generated during mineral processing are subject to regulation as hazardous wastes. However, the final regulatory status of 20 large-volume mineral processing wastes is expected to be determined in 1991.[a]

Two major parts of RCRA are particularly important. Subtitle C (Subchapter III) deals with hazardous waste management. Subtitle D (Subchapter IV) deals with nonhazardous waste management. Each of these subtitles is described in the following.

Hazardous Waste Regulation: A *hazardous waste* is defined by Subtitle C of RCRA as "a solid waste, or combination of solid wastes, which because of its quantity, concentration, or physical, chemical, or infectious characteristics may (A) cause, or significantly contribute to an increase in mortality or an increase in serious irreversible, or incapacitating reversible, illness; or (B) pose a substantial present or potential hazard to human health or the environment when improperly treated, stored, transported, or disposed of, or otherwise managed."[50]

The regulations include lists of specific hazardous wastes.[51] A solid waste is also defined as hazardous if it exhibits one or more characteristics including ignitability, corrosivity, reactivity, or EP toxicity.[52] The latter characteristic is determined by the Extraction Procedure (EP) Toxicity Test, which is described in the regulations.[53] The EPA has proposed a modification for the EP Toxicity Test, the Toxicity Characteristic Leaching Procedure (TCLP).[54]

Mixtures of solid wastes and listed hazardous wastes are also considered hazardous wastes unless the listed hazardous waste was listed "solely because it exhibits one or more of the [four] characteristics of hazardous waste" and "the resultant mixture no longer exhibits any characteristic of hazardous waste."[55]

Certain mining wastes are specifically designated as not being hazardous. These include "mining overburden returned to the mine site"[56] and "materials subjected to in-situ mining techniques which are not removed from the ground as part of the extraction process."[57] Other waste "from the extraction, beneficiation and processing of ores and minerals (including coal), including phosphate rock, and overburden from mining of uranium ore" were temporarily excluded.[58] More will be said about this exclusion later in this discussion.

Standards are included in the regulations for hazardous waste generators,[59] transporters,[60] and owners and operators of hazardous waste treatment, storage, and disposal facilities? In general, those participating in any of these types of activities are required to obtain an EPA identification number. A uniform manifest system is to be used for shipment, transport, and receipt of any hazardous waste. The extensive regulations for owners and operators of hazardous waste treatment, storage, and disposal facilities cover design, operation, and maintenance of the facility; contingency plans and emergency procedures; detection and correction of releases from the facility; closure and post-closure requirements; and financial requirements, to name just a few of the provisions.

Nonhazardous Waste Regulation: The management of *nonhazardous solid waste* is covered under Subtitle D of RCRA. The Act required the EPA to provide technical and financial assistance to state, regional, and local authorities for the development of solid waste management plans. The primary target of this subtitle was municipal waste. The EPA was required to develop criteria for sanitary landfills. All solid waste was to be disposed of in sanitary landfills. Open dumps were prohibited.

Guidelines for solid waste management plans were to be developed by the EPA and periodically reviewed. Rather than a rigid set of federal regulations as exist for hazardous waste, these guidelines were designed to provide flexibility. They were to consider "the varying regional, geologic, hydrologic, climatic, and other circumstances under which different solid waste management practices are required in order to insure the reasonable protection of the quality of the ground and surface waters from leachate contamination, the reasonable protection of the quality of the surface waters from surface runoff contamination, and the reasonable protection of ambient air quality."[62] They were also required to consider the "characteristics and conditions of collection, storage, processing, and disposal operating methods, techniques and practices, and location of facilities where such operating methods, techniques, and practices are conducted, taking into account the nature of the material to be disposed."[63]

At this writing, the future of the EPA's regulatory program under Subtitle D is less than clear. Many feel that the provisions of Subtitle D need to be strengthened and made more specific to individual waste types. However, the flexibility provided by Subtitle D is particularly important because the EPA has announced its intention to regulate mining and beneficiation wastes under this Subtitle.[64] The current status of the development of these regulations is described in the following.

Potential Changes in Mining Waste Regulation: Mining, beneficiation, and mineral processing wastes constitute nearly 40% of the nation's solid wastes generated annually by municipalities, industries, and agriculture. However, RCRA recognized the special characteristics of most of these wastes, namely that they are large in volume and generally low in toxicity. The Act required the EPA to study these wastes before developing regulations. Specifically, the Act required that "the Administrator [of the EPA], in consultation with the Secretary of the Interior, shall conduct a detailed and comprehensive study on the adverse effects of solid wastes from active and abandoned surface and underground mines on the environment."[65] In addition, it required that "the Administrator [of the EPA] shall conduct a detailed and comprehensive study on the adverse effects on human health and the environment, if any, of the disposal and utilization of solid waste from the extraction, beneficiation, and processing of ores and minerals, including phosphate rock, and overburden from uranium mining."[66] In 1980, Congress

[a] At this writing, the actual regulatory status of solid wastes in the minerals industry is complex and in a state of flux. The reader is cautioned to check carefully the current status of mining, beneficiation, and large-volume mineral processing waste regulations to see what new regulations may apply.

amended RCRA. One section of these amendments, the Bevill Amendment,[67] specifically excluded "solid waste from the extraction, beneficiation, and processing of ores and minerals" from regulation under Subtitle C, pending completion of the studies mentioned previously.

Because of the complex problems associated with the disposal of wastes such as mine overburden, mine waste rock, leach residues, mill tailings, and smelter slags, the development of regulations moved very slowly; however, the pace of regulatory developments has accelerated in recent years. In 1985, the EPA submitted a report to Congress entitled *Wastes from the Extraction and Beneficiation of Metallic Ores, Phosphate Rock, Asbestos, Overburden from Uranium, and Oil Shale.* This report partially met the requirements for the studies of mining and mineral processing wastes required by RCRA. In July 1986, EPA issued a regulatory determination in which it found that regulation of mining and mineral processing wastes as hazardous wastes under Subtitle C of RCRA was not warranted and that it would develop a regulatory program under Subtitle D. This determination was viewed as an important victory by the mineral industry. Although EPA originally intended that the regulatory determination would apply to all mining and mineral processing wastes, subsequent court decisions have caused EPA to narrow its scope to only mining and beneficiation wastes. It now appears likely that these wastes will be regulated by EPA and the states in the mid-1990s.

A 1988 court rendered an opinion that Congress had not intended that the mining waste exclusion under the Bevill Amendment should include all wastes from primary smelting and refining. It also decided that the term "solid waste from the . . . processing of ores and minerals" in that Bevill Amendment should include only those wastes from processing ores and minerals that meet undefined criteria for large volume and low hazard. The court directed EPA to immediately list six waste streams from aluminum, copper, lead, and ferroalloy smelting as hazardous. In September 1989, EPA published its final rule reinterpreting the Bevill exclusion for "solid waste from the . . . processing of ores and minerals."[68] In this ruling, EPA also established the criteria for designating wastes as large volume and low hazard.

In January 1990, EPA released another final rule which applied the revised interpretation and resulted in only 20 large-volume, mineral processing wastes being retained within the Bevill exclusion.[69] Included in this group were such wastes as lead smelter slags and phosphogypsum from phosphoric acid production. These wastes will be subject to study as required in RCRA. The study could lead to the final regulatory determination for mineral processing wastes within a year. As a result of these actions, all other mineral processing wastes are now subject to regulation under RCRA's Subtitle C as hazardous wastes.

REFERENCES
(Parts 3.4.1 and 3.4.2)

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3. *Id.* § 102(C).
4. *Id.* § 101(a).
5. 40 C.F.R. § 1502 (1989).
6. *Id.* § 1506.5(b).
7. *Id.* § 1506.6.
8. *Id.* § 1505.2.

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9. SMCRA § 101(c) (1988).
10. *Id.* § 501.
11. *Id.* § 401.
12. *Id.* § 201.
13. *Id.* § 101(d).
14. SMCRA § 503 (1988) and 30 C.F.R. §§ 730-736 (1989).
15. 30 C.F.R. §§ 900-950 (1989).
16. SMCRA § 523 (1988) and 30 C.F.R. §§ 740-746 (1989).
17. SMCRA § 710 (1988) and 30 C.F.R. §§ 750-756 (1989).
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19. SMCRA § 522(a)(2) (1988).
20. *Id.* § 522(a)(3)(A).
21. *Id.* § 522(a)(3)(B).
22. *Id.* § 522(a)(3)(C).
23. *Id.* § 522(a)(3)(D).
24. *Id.* § 522(a)(6).
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26. SMCRA § 515 (1989).
27. *Id.* § 516.
28. 30 C.F.R. § 815 (1989).
29. *Id.* § 816.
30. *Id.* § 817.
31. *Id.* § 819.
32. *Id.* § 820.
33. *Id.* § 822.
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36. *Id.* § 825.
37. *Id.* § 827.
38. *Id.* § 828.
39. SMCRA § 509 (1988).
40. *Id.* § 519.
41. SMCRA § 517 (1988) and 30 C.F.R. §§ 840-846 (1989).

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42. RCRA § 1004(27) (1986).
43. 40 C.F.R. § 261.2(a)(l) (1989).
44. *Id.* § 261.2(a)(2).
45. *Id.* § 261.2(b).
46. *Id.* § 261.2(c).
47. *Id.* § 261.2(e).
48. *Id.* § 261.2(d).
49. RCRA § 1004(29) (1986).
50. *Id.* § 1004(5).
51. 40 C.F.R. § 261.30 (1989).
52. *Id.* § 261.20.
53. *Id.* § 261 Appendix II.
54. 51 Fed. Reg. 1602 (1986).
55. 40 C.F.R. §§ 261.3(a)(2)(iii)-(iv) (1989).
56. *Id.* § 261.4(b)(3).
57. *Id.* § 261.4(a)(5).
58. *Id.* § 261.4(b)(7).
59. *Id.* § 262.
60. *Id.* § 263.
61. *Id.* § 264.
62. RCRA § 4002(c)(1) (1986).
63. *Id.* § 4002(c)(2).
64. 51 Fed. Reg. 24496 (1986).
65. RCRA § 8002(f) (1986).
66. *Id.* § 8002(p).
67. *Id.* § 3001(b)(3)(A)(ii).
68. 54 Fed. Reg. 36592 (1989).
69. 55 Fed. Reg. 2322 (1990).

3.4.3 MAJOR FEDERAL ENVIRONMENTAL LAWS AND REGULATIONS (cont.)

CLAYTON J. PARR

3.4.3.1. Comprehensive Environmental Response, Compensation, and Liability Act of 1980 (Superfund)

The Comprehensive Environmental Response, Compensation, and Liability Act (commonly referred to as "CERCLA" or "Super-fund") focuses upon discharges of hazardous substances.[1] Such hazardous substances can exist in mine water, leachate (including natural leachate from waste piles), mill tailings, and chemicals used in various stages of mining, mineral processing, and mine-related activities.[2]

Ongoing mine- and mill-related discharges of hazardous solid wastes are covered by the Resource Conservation and Recovery Act. CERCLA applies to discharges of "hazardous substances" at inactive sites occurring through the effects of natural processes on discarded materials.

CERCLA also applies to accidental spills or releases of certain hazardous substances that are not subject to reporting and cleanup requirements of other laws.

CERCLA is enforced by the US Environmental Protection Agency, but it also involves state agencies.

Inactive Sites: Mine operators may encounter CERCLA-governed inactive sites on owned property or property that is being acquired. Old tailings disposal areas in particular are high risk.

The definition of *hazardous substance* is extremely broad, covering generally materials that present a substantial danger to the environment or public health. The term includes certain hazardous substances and toxic pollutants designated under the Clean Water Act, Solid Waste Disposal Act, Clean Air Act, and Toxic Substances Control Act. A list of substances designated as hazardous by EPA is published in Title 40, Code of Federal Regulations (CFR), Part 302.

The most common situation encountered at mine sites involves deposits of wastes, such as mill tailings, with hazardous substance constituents that through action of the elements can pollute surface water or groundwater. Some near-surface sites can emit dangerous gases or particles that can enter the body through respiration. A few materials, such as PCBs, are dangerous to the touch. Petroleum, including crude oil and any fraction thereof that is not otherwise specifically listed or designated as a hazardous substance, natural gas, natural gas liquids, liquefied natural gas, or synthetic gas usable for fuel, are excluded.[3]

CERCLA requires that persons in charge give notification of any facility where hazardous wastes have been treated, stored, or disposed of that could threaten the environment, unless the site has already been identified through the RCRA notification procedures. Sites are also identified by EPA through its own investigations.[4]

Once a site is identified by EPA as a possible source of discharges of hazardous substances, further evaluation procedures are governed by the National Contingency Plan (NCP)[5] The NCP also establishes the framework for site investigations and for removal and remediation. EPA has the authority to request information concerning use, transportation, or generation of hazardous substances.[6] In a final rule promulgated on Mar. 8, 1990, EPA adopted changes to the NCP (1) to implement regulatory changes necessitated by the Superfund Amendments and Reauthorization Act of 1986; (2) to clarify existing NCP language; and (3) to reorganize the NCP to coincide more accurately with the sequence of response actions.[7]

A preliminary assessment is performed to determine whether a serious problem exists. Through a mathematical evaluation methodology, known as the Hazard Ranking System (HRS), sources, pathways, and receptors are assessed.[8] If the score and resulting ranking are sufficiently high, the site is placed on the National Priorities List (NPL), a listing of approximately 1200 sites throughout the United States that have been evaluated as needing to be cleaned up.[9] A remedial investigation/feasibility study (RI/FS) is then conducted to determine appropriate remedial alternatives and objectives. After a remedial action is selected, which can involve stabilization or removal, work begins.

CERCLA is designed to put the obligation to pay for all of this expensive work on those who were responsible for putting the hazardous substances at the site in the first place or who now own the land upon which the site is located. If no responsible party can be found, or if the cleanup must proceed before the responsible parties can be tagged with the liability, funds can be used from the federal Hazardous Substance Superfund.[10] Reimbursement is then sought from the responsible parties. Governmental entities and private parties can also obtain reimbursement from the fund or from responsible parties.[11] Claims are not recoverable where the release of a hazardous substance is federally permitted, which can include releases authorized under state permits issued pursuant to delegated federal programs.[12]

Potentially Responsible Parties (PRPs) may include present or former owners or operators of the facility or the property upon which the hazardous wastes have been stored, treated, or disposed of; the generators of the hazardous waste; and parties who accepted the hazardous waste for transport and selected the facility.[13] The unique, and most frightening aspects of CERCLA, involve the scope of the liability of the PRPs. Because the liability is retroactive, a PRP can have enormous financial responsibility for activities that took place decades ago when the hazardous waste generating activity did not conflict with any then applicable laws. Although the law provides for contribution among responsible parties, liability is joint and several, thereby exposing any one of the responsible parties to an obligation to pay for the entire remedial action even though only a small fraction of the problem can be directly attributed to that party. A strict liability standard is applied, meaning that no negligence or willful misconduct need be shown.[14]

Completion of remediation cleanup of the site that removes the threat does not end the matter. CERCLA also provides for liability of responsible persons for damages for, injury to, destruction of, or loss of natural resources, including the reasonable costs of assessing such injury, destruction, or loss resulting from a release or threatened release of a hazardous substance.[15] This element of the law addresses more long-range effects of the discharge on such things as land, fish, wildlife, biota, air, water, groundwater, drinking water supplies, and the like that are controlled by the United States or any state or local government.[16] Natural resources damages cannot be recovered with respect to damages and releases that occurred wholly prior to enactment of CERCLA.[17]

Because of the immense liability that can be imposed upon an owner of a property under CERCLA, an examination for possible sites containing hazardous substances should be made before an acquisition is completed.[18] Under the "innocent landowner defense," liability can be avoided if it can be shown by a landowner that, at the time the facility was acquired, the purchaser did not know and had no reason to know that any hazardous substance which is the subject of a release or threatened release was disposed of on, in, or at the facility.[19] This requires that an appropriate inquiry has been undertaken at the time of acquisition into the previous ownership and uses of the property consistent with good commercial or customary practice in an effort to minimize liability. A thorough examination by trained professionals is needed to show such an appropriate inquiry.

Accidental Spills: Upon the occurrence of a spill or other actual release of a reportable quantity of a hazardous substance, the person in charge must immediately notify the National Response Center (NRC) at US Coast Guard headquarters in Washington, DC. The reportable quantity, as prescribed in Section 302 of Title 40, CFR, may be as little as 1 lb (0.5 kg) or as much as several thousand pounds (kilograms) per 24-hr period. The person in charge is the individual or entity with power to direct the activities of persons who control the mechanisms causing the pollution.[20] A responsible person who fails to notify under CERCLA is liable for civil and criminal penalties.[21]

Certain substances and types of releases are exempted from reporting requirements. Among these are petroleum, crude oil, natural gas, natural gas liquids, and releases exempted from reporting under RCRA. The petroleum exclusion includes refined petroleum products.

A release includes "spilling, leaking, pumping, pouring, emitting, emptying, discharging, injecting, escaping, leaching, dumping or disposing into the environment (including the abandonment or discarding of barrels, containers, and other closed receptacles containing any hazardous substances" or pollutant or contaminant)."[22] Since it is only releases "into the environment" that are covered, a hazardous substance spill onto a concrete floor of an enclosed building or structure need be reported only if the substance escapes the building structure in a reportable quantity.[23]

The owner or operator of the facility from which the release has been made has the responsibility to clean up. If that does not happen, EPA may issue an administrative order or seek an injunction ordering a responsible party to mitigate or clean up the release.[24] EPA may also go ahead with the cleanup (a remedial action) and then seek reimbursement of costs from the responsible party.[25] Cleanup is performed pursuant to the National Contingency Plan.[26] Such "remedial actions" are designed "to prevent or minimize the release of hazardous substances" or pollutants or contaminants so they do not migrate to cause substantial damage to present or future public health or welfare or to the environment.[27]

The owner/operator with responsibility for such costs includes the entity with legal possession of and operating control over the facility, and other individuals or entities such as parent corporations that actively control the daily operations of the subsidiary; lenders or other secured creditors who take possession and participate in management of the facility; passive owners, such as lessors, even if they do not participate in facility management or in the disposal of hazardous substances; and individual officers and controlling shareholders who actively participate in management of the facility.[28]

SARA/EPCRA: The Superfund Amendments and Reauthorization Act of 1986 ("SARA") enacted the Emergency Planning and Community Right-to-Know Act ("EPCRA" or "SARA Title III") as a free-standing law. EPCRA's purpose is to encourage state and local emergency planning regarding chemical releases, and to develop a bank of information for residents and local governments on potential chemical hazards in their communities. That bank of information is collected from owners and operators of subject facilities pursuant to the reporting requirements of EPCRA. EPCRA's reporting requirements include (1) emergency release notification under Section 304;[29] (2) community right-to-know reporting under Sections 311 and 312;[30] and (3) toxic chemical release reporting under Section 313.[31]

The Environmental Protection Agency has developed a list of "extremely hazardous substances" under EPCRA. The list is codified at Title 40, CFR, Part 355, Appendix A. Under Section 304, facilities must immediately notify state and local planning committees of the release of a reportable quantity of either a CERCLA "hazardous substance" or an EPCRA "extremely hazardous substance." This requirement is closely related to but slightly different from the accidental release reporting requirements under CERCLA. Currently, the major difference in reporting requirements is that releases of "extremely hazardous substances" need be reported only to state and local officials, while releases of "hazardous substances" must be reported to the National Response Center (to satisfy CERCLA reporting requirements) *and* to state and local officials (to satisfy EPCRA emergency release notification requirements). EPA is currently involved in rulemaking that would eliminate the confusion caused by these different notification requirements.[32]

Owners and operators otherwise required (under the provisions of the Occupational Safety and Health Act, or OSHA) to prepare or have available Material Safety Data Sheets (MSDS) for "hazardous chemicals" under OSHA (codified at Title 29, CFR, Section 1910.1200) must annually submit either the MSDS or a list of such chemicals present at a facility above specified threshold levels to state and local committees and to the local fire department. In addition, subject facilities must submit annual inventory forms to the same governmental entities providing aggregate information on the maximum and average amounts and general location of hazardous chemicals present at a facility above specified threshold levels.[33]

Subject facilities must submit to EPA and designated state officials a toxic chemical release inventory form ("Form R") annually on July 1.[34] Form R covers all types of releases to all environmental media. The list of toxic chemicals is codified at Title 40, CFR, Part 372 (Subpart D). Data provided to EPA under the provisions of Section 313 is available to the general public on a computerized database. Facilities subject to Form R reporting are, among other things, manufacturing facilities under Standard Industrial Classification Codes 20-39. Although the mining and milling of natural resources falls outside of those Codes (i.e., Standard Industrial Classification Codes 10 through 14), many mining operations, such as metal mining, also include manufacturing operations (e.g., smelting and refining), which brings them within the scope of this reporting requirement.

3.4.3.2 Federal Water Pollution Control Requirements

The initial inquiry as to environmental regulations concerning water pollution must be directed to federal law, and particularly to the Water Pollution Control Act Amendments of 1972[35] as amended by the Clean Water Act of 1977[36] (collectively referred to here as the Act). This long and very complex piece of legislation establishes the basic procedural and technical requirements applicable to the discharge of water as a consequence of any mining or milling activity.

The basic thrust of the Act is to improve water quality by regulation of discharges of pollutants into navigable waters at the source. Thus the addition of any pollutant to navigable waters from any point source except in compliance with the Act is unlawful.[37]

The term *point source* is defined as follows:

> [A]ny discernible, confined and discrete conveyance, including but not limited to any pipe, ditch, channel, tunnel, conduit, well, discrete fissure, container, rolling stock, concentrated animal feeding operation, or vessel or other floating craft, from which pollutants are or may be discharged.[38]

The term *navigable waters* is so broadly defined in the Act and in the regulations that any discharge from a point source, whether intermittent or continuous, that enters surface waters, or tributaries of surface waters (even if not flowing), should be presumed to be covered.[39]

The Act also provides for the adoption of standards applicable to the discharge of pollutants into publicly owned treatment works.[40]

Responsibility for administering the Act initially rests with EPA. As described in the following, individual states can take over certain administrative and enforcement responsibilities under the Act.

Effluent Limitations: Regulations applicable to discharges from point sources at mining operations are designed to protect water quality by limiting the concentrations of certain pollutants and by limiting the level of acidity in whatever quantities of water that are discharged from point sources. Those levels are determined by evaluating the level to which the quantities of pollutants can be limited through the application of available technology.[41]

Existing sources have been permitted under requirements that have become more restrictive over time.

Since the most modern technology can be incorporated into new construction, the restrictions for discharges from new sources, as would be expected, are the most stringent. New sources are thus to be governed by effluent limitations based upon standards reflecting application of "the best available demonstrated control technology, processes, operating methods, or other alternatives."[42] Where these "new source performance standards" have yet to be developed, as is the case with some of the mining discharge categories described in the following, the most stringent limitations that have been developed are applicable to new sources, with the subsequently developed standards to be imposed over time as permits are renewed.

Standards of performance for new sources located in a particular state may be applied and enforced by the state, if the procedures and the laws of the state require the application and enforcement of standards of performance to at least the same extent as is required by the federal law.[43]

Ore Mining and Dressing Point-Source Category: Effluent limitations promulgated by EPA for the ore mining and dressing point-source category are set forth in Part 440, of Title 40, CFR.[44] Separate regulations are set forth for each of 12 subcategories of ore. These are (1) iron, (2) aluminum, (3) uranium, radium, and vanadium, (4) mercury, (5) titanium, (6) tungsten, (7) nickel, (8) vanadium (mined alone), (9) antimony, (10) copper, lead, zinc, gold, silver, and molybdenum, (11) platinum, and (22) gold placers.

The regulations are applicable to mines and mills producing and processing ores in the various subcategories. The term *mine* is defined in the regulations as an

> active mining area, including all land and property placed under, or above the surface of such land, used in or resulting from the work of extracting metal ore or minerals from their natural deposits by any means or method, including secondary recovery of metal ore from refuse or other storage piles, wastes, or rock dumps and mill tailings derived from the mining, cleaning, or concentration of metal ores.[45]

The regulations are directed to waste water discharged from mines as mine drainage. The term *mine drainage* means any water drained, pumped, or siphoned from a mine.[46]

If drainage from mines or discharges from mills are polluted, treatment will be required to reduce the concentration to permitted levels before it can be discharged to navigable waters. Because the quantity of mine water is dependent upon many factors beyond the control of the mine operator and is unrelated or only indirectly related to mine production, discharge restrictions are expressed in terms of concentration (with the exception of pH units) rather than units of production.

The term *mill* is defined as a

> preparation facility within which the metal ore is cleaned, concentrated, or otherwise processed before it is shipped to the customer, refiner, smelter, or manufacturer. A mill includes all ancillary operations and structures necessary to clean, concentrate, or otherwise process metal ore, such as ore and gangue storage areas and loading facilities.[47]

Mill water that may have become polluted is that water used in ore and product washing, dust suppression, grinding and classification, heavy-media separation, flotation, and equipment and floor washing. In addition, boiler blowdown and noncontact cooling water, such as bearing cooling water, have been identified as possible sources of stream pollutants.

Mineral Mining and Processing Point-Source Category: Effluent limitations promulgated by EPA for the mineral mining and processing point-source category, which includes minerals generally classified as industrial minerals, are set forth in Part 436 of Title 40, CFR.[48] Separate regulations are given for each of 21 subcategories, and EPA contemplates that standards eventually will be promulgated for 38 subcategories of mineral types or classes. Minerals for which effluent limitations have been published are crushed stone, construction sand and gravel, industrial sand, phosphate rock, gypsum, asphaltic minerals, asbestos and wollastonite, barite, fluorspar, salines from brine lakes, borax, potash, sodium sulfate, Frasch sulfur, bentonite, magnesite, diatomite, jade, novaculite, tripoli, and graphite.

The regulations are directed toward limiting the pollutants from mine drainage, transportation of the ore to the processing plants, and waste waters from the processing plant itself, including transport water, ore and product washing water, dust-suppression water, classification water, heavy-media separation water, flotation water, solution water, air-emissions control-equipment water, and equipment and floor washdown water.[49]

Coal-Mining Point-Source Category:[50] Effluent limitations promulgated by EPA for the coal-mining point-source category are set forth in Part 434, Title 40, CFR. Separate regulations are provided for each of three subcategories:

1. The Coal Preparation Plant and Coal Preparation Plant Associated Areas Subcategory is applicable to facilities where coal is cleaned, concentrated, or otherwise processed or prepared in order to separate coal from its impurities and is loaded for transfer to a consuming facility and associated areas such as coal preparation plant yards, immediate access roads, coal refuse piles, and coal storage piles and facilities.

2. The Acid or Ferruginous Mine Drainage Subcategory is applicable to mine drainage resulting from the mining of coal of any rank that, before any treatment, has either a pH of less than 6.0 or a total iron concentration of more than 10 mg/L.

3. The Alkaline Mine Drainage Subcategory is applicable to mine drainage resulting from the mining of coal of any rank that, before any treatment, has a pH of more than 6.0 and a total iron concentration of less than 10 mg/L.

4. The Postmining Areas Subcategory is applicable to discharges from reclamation areas until the performance bond issued to the facility under the Coal Surface Mining and Reclamation Act has been released by the appropriate authority.

The regulations define *active mining area* as:

> the area, on and beneath land, used or disturbed in activity related to the extraction, removal, or recovery of coal from

its natural deposits. This term excludes coal preparation plants, coal preparation plant associated areas and post-mining areas.[51]

Postmining area is defined as: (1) a reclamation area or (2) the underground workings of an underground coal mine after the extraction, removal, or recovery of coal from its natural deposit has ceased and prior to bond release.[52]

The term *mine drainage* means any water drained, pumped, or siphoned from an active mining area or a postmining area.[53]

Parameters are established for four pollutants typically discharged into streams from coal mining operations. These are total iron, total manganese, total suspended solids, and pH.

State Areawide Waste Treatment Management Plans: Under Section 208 of the Act,[54] the governor of each state, in compliance with regulations published by the Administrator, must identify each area within the state that has substantial water-quality control problems. A continuing areawide waste-treatment management planning process is to be prepared for each such area and will be applicable to all wastes generated within the area involved. Areas within a state, including federal lands, that are not so designated by the governor will, by operation of the provisions of Section 208, be ultimately designated and made subject to areawide waste-treatment plans.[55]

The waste-treatment planning process must provide for control or treatment of all point and nonpoint sources of pollution, including in-place or accumulative pollution sources, to protect both groundwater and surface-water quality.[56] Each plan is (1) to include a process to identify mine-related sources of pollution including new, current, and abandoned surface and underground mine runoff, and (2) to set forth procedures and methods (including land-use requirements) to control such sources to the extent feasible.[57] So-called "best management practices" will be established to control nonpoint-source pollution. Best management practices are those selected by an agency to meet its nonpoint source-control needs. They include structured and nonstructured controls and operation and maintenance procedures, and they can be applied before, during, and after pollution producing activities.[58]

Section 208 plans have significance for mining operations because no permit under Section 402 of the Act will be issued for any point source that is in conflict with an areawide waste treatment plan.[59] In addition, control over nonpoint-source pollution, such as would occur from storm runoff or from leach dumps, is provided for by the Act, even though the Section 402 permit requirements directly relate only to point sources.[60]

State Water Quality Standards: Under Section 303 of the Act,[61] each state also must adopt interstate and intrastate water quality standards applicable to water in the state. Water quality standards are essentially water-use classifications generally based upon the quality criteria for water developed by EPA. Typical water quality programs assign each body of water one or more classifications, based upon existing quality and uses, such as public water supply, recreational, aquatic life, industrial, and agricultural.[62] For such standards to be achieved or maintained, it is necessary to regulate effluents entering those waters from industrial operations. Therefore, a discharge of pollutants from a mining operation is evaluated to determine whether this discharge will allow achievement or maintenance of the state water quality standards.

Section 303(d)[63] requires that each state further identify the waters within its boundaries for which the effluent limitations required under Section 301 of the Act are not stringent enough to implement any water quality standard applicable to any such waters. Maximum daily load limits are to be established for those pollutants for which the effluent limitations are inadequate. Consequently, a mining operation located on water so classified may have to meet more stringent effluent limitations than otherwise would be imposed.[64]

Permit Requirements: As is indicated from the foregoing discussions, the degree of water pollution control required at any given property is determined both by the water quality standards to be achieved in the area and by the effluent limitations that are imposed by regulation. Permit conditions are designed to assure that the discharge meets such requirements. When necessary to meet water quality standards, treatment standards, or schedules of compliance established pursuant to state law or regulation, more stringent limitations may be imposed than are required by the Act.

These substantive aspects of the Act are imposed upon mining operations by means of a permit system; a permit is required for any discharge of pollutants subject to the Act, which establishes the National Pollutant Discharge Elimination System (NPDES).[65]

NPDES permits must be acquired for all existing and new sources. A permit is not required for discharges made to a publicly owned treatment works, but, in such instances, the discharger must comply with applicable pretreatment standards.[66]

The Environmental Protection Agency has recently promulgated NPDES permit application procedures for discharge of storm water "associated with industrial activity" pursuant to Section 402(p) of the Act.[66a] The term "associated with industrial activity" includes storm water discharges from "[f]acilities classified as Standard Industrial Classifications 10 through 14 (mineral industry) including active or inactive mining operations."[66b] If storm water from these facilities is "contaminated by contact with . . . any overburden, raw material, intermediate products, finished product, byproduct or waste products located on the site of such operations," then storm water discharges from the facilities must be permitted.[66c] Under these new regulations, facilities may choose to apply for general, group, or individual permits.[66d]

Permits are issued on such conditions as the Administrator determines are necessary to carry out the provisions of the Act, and they can be extremely stringent. Specifically, the permits must comply with published state and federal effluent limitation requirements. Federal requirements are based upon determinations of the minimum concentrations of certain pollutants that can be achieved through the application of varying levels of pollution control technology. Effluent limits under permits may also be influenced by limitations imposed to improve or maintain water quality in the drainage system involved. It is through this means that water pollution is controlled, and the overall objectives of the Act are ultimately achieved. The nature of such requirements is discussed in the following paragraphs.

The permit program established under Section 402 of the Act may be administered by the state for discharges into navigable waters within its jurisdiction, if a proposed state plan is submitted to and approved by the Administrator of the EPA in accordance with the standards established by the Act.[67] The Administrator, however, retains the power to review permit applications and to object to the proposed issuance of a permit.[68] In such an event, the state may resubmit the permit, revised to meet the objection, or request a hearing. If neither step is taken, the Administrator may issue the permit with such modifications as he deems necessary to effect compliance with the Act.[69]

A separate permit application must be submitted for each facility that is discharging separately. Therefore, more than one permit may be required for a particular mining operation. One NPDES permit, however, may cover several discharge points from the same facility or operation.

Current procedures for obtaining NPDES permits (applicable in states not having an approved plan for state administration) are set forth in Part 124, Title 40, CFR. State programs are patterned after the federal regulations.[70] Details of permit contents are presented in Part 122 of Title 40, CFR.

A new discharger must apply for a permit at least 180 days before discharge begins, unless the Regional EPA Administrator approves a later application date.[71] The application must be submitted on an appropriate form.

No action will be taken on an application to the EPA until the applicant provides the Regional Administrator with a certification from the state or interstate water pollution control agency having jurisdiction over the discharge point of origin that the discharge will comply with any applicable effluent limitations or water quality standards. Any certification must set forth any applicable effluent limitations, standards, and monitoring requirements needed to insure compliance with state regulations.[72]

Public notice is given of each application, and a draft permit is prepared for comment.[73] If a sufficient degree of public interest is found or if a request is made, a hearing will be held.[74] EPA has 90 days to comment on a state-issued permit.[75] If EPA objections are not met, EPA can take over and issue the permit.[76]

An environmental impact statement will not be required prior to issuance of a permit for an existing source, but it may be required in connection with permits for new sources.[77] Preparation of an EIS will expand the time required for obtaining a permit.

An adjudicatory hearing will be held if a request is made and if the Regional Administrator determines that the request sets forth material issues of fact or that a hearing is necessary or appropriate.[78]

Permits must contain any special conditions deemed necessary to assure compliance with applicable effluent limitations or other water quality requirements, including schedules of compliance, treatment standards, record keeping, and monitoring requirements (including those set forth in the state certification).[79]

Permits are issued for a fixed term not exceeding five years. Short-term permits of less than five years may be issued where there are special circumstances.[80]

If the operator of a point source fails to obtain a permit or violates the permit conditions, EPA may issue a compliance order or bring a civil action against the violator.[81] Civil penalties of up to $25,000 per day of violation may be imposed for discharges without a permit or in violation of a permit.[82] Any person negligently violating the Act or any condition or permit issued by EPA or by a state is subject to criminal sanctions and may be punished by fine of not less than $2500 nor more than $25,000 per day of violation, by imprisonment for not more than one year, or both. Willful violations trigger penalties of not less than $5000 per day of violation, or imprisonment not to exceed three years, or both.[83] Conviction of subsequent violations is punishable by a fine of not more than $50,000 per day and/or two years' imprisonment.[84]

Violation of the Act, of any conditions or limitations set forth in a state or federal permit, or of any compliance order, which violation is not willful or the result of negligence, is subject to civil penalties assessed administratively by EPA.[85]

Hazardous Substances: Section 311 of the Act[86] provides for the designation of hazardous substances other than oil by the Administrator of EPA, and the imposition of penalties for the unauthorized discharge of such substances. Hazardous substances are elements or compounds other than oil which, when discharged in any quantity into or upon navigable waters of the United States, present an imminent and substantial danger to the public health or welfare, including, but not limited to, fish, shellfish, wildlife, shorelines, and beaches.[87]

The Administrator must make a determination whether any substance designated as a hazardous substance can be removed after discharge.[88] The unauthorized discharge of any hazardous substance determined not to be removable creates liability for monetary penalties based on the quantity of such hazardous substance that is released.[89]

The discharge of any hazardous substance into navigable waters in harmful quantities is prohibited.[90] The quantity of a substance that constitutes a harmful quantity is determined by the Administrator of EPA and shown in Part 117 of Title 40, CFR. Any such discharge must be reported to the EPA, National Response Center in Washington, DC, by the person in charge, failing which, a penalty of a fine or imprisonment can be imposed. Penalties of not more than $5000 will be assessed for each discharge.[91] The penalties are based upon the toxicity, degradability, and dispersal characteristics of the substance or the number of units of measurement discharged. The owner causing such discharge is also liable for the costs of removal of the hazardous substance.[92]

Regulations designating hazardous substances promulgated by EPA are contained in Part 116, Title 40, CFR. The list of substances designated as hazardous includes many chemicals common to mining and milling operations. Consequently, unauthorized discharge of such substances through direct release or through breach of a tailings pond could result in liability for the penalties provided in the Act.

Regulations may also be promulgated by the Administrator, establishing best management practices for hazardous pollutants to control plant-site runoff, spillage or leaks, sludge or waste disposal, and drainage from raw material storage.[93] Such controls must be required in any NPDES permit.

Discharges of Oil: Discharges of oil in harmful quantities are prohibited.[94] Harmful quantities are those that "cause a film or sheen upon or discoloration of the surface of the water or adjoining shorelines or cause a sludge or emulsion to be deposited beneath the surface of the water or upon adjoining shorelines."[95]

Any prohibited discharge must be reported to the Natural Response Center in Washington, DC.[96] Civil penalties are prescribed for violation of oil pollution regulations.[97]

A spill prevention, control, and countermeasure (SPCC) plan is required of any facility that has a buried oil-storage tank with a capacity of greater than 42,000 gal (159 kL), a nonburied oil-storage unit of more than 1320 gal (5 kL), or any single container greater than 660 gal (2.5 kL).[98] The plan must be certified by a registered professional engineer and must contain the procedures, methods, and equipment that will prevent the discharge of oil spills to navigable waters. No plan need be submitted to the EPA, but it must be available for inspection at any time.[99]

The plan is required of any onshore facilities not related to transportation. Facilities which, due to their location, cannot reasonably expect to discharge oil onto navigable waters are exempt from preparing an SPCC plan. The regulations establish penalties for oil spills.[100]

Toxic Pollutants: Section 307(a) of the Act establishes a list of 65 pollutants considered to be toxic.[101] Included are the following that could be encountered in mining and milling operations:

1. Antimony and compounds.
2. Arsenic and compounds.
3. Asbestos.
4. Beryllium and compounds.
5. Cadmium and compounds.
6. Chromium and compounds.
7. Copper and compounds.
8. Lead and compounds.

9. Mercury and compounds.
10. Nickel and compounds.
11. Selenium and compounds.
12. Silver and compounds.
13. Zinc and compounds.

The Administrator is authorized to add pollutants to or remove pollutants from the list from time to time. Effluent standards or prohibitions for toxic pollutants set forth in EPA regulations may be incorporated in an NPDES permit.[102]

At the time of this writing, EPA has established special toxic effluent standards for only six pollutants, which do not include any of those listed previously.[103]

Regulations also may be promulgated by the Administrator, establishing best management practices for toxic pollutants to control plant-site runoff, spillage or leaks, sludge or waste disposal, and drainage from raw material storage.[104] Even though such sources are more in the nature of nonpoint sources, effluent controls requiring best management practices will be included in NPDES permits.

3.4.3.3 Dredge and Fill Permits

Section 404 of the Water Pollution Control Act[105] authorizes the Secretary of the Army, acting through the Chief of the Corps of Engineers, to issue permits for the discharge of dredged or fill material into the navigable waters of the United States at specified disposal sites. Guidelines for the selection of disposal sites are to be developed by the Administrator of EPA in conjunction with the Secretary of the Army.[106]

Regulations prescribing the policy, practice, and procedure to be followed by the Corps of Engineers in connection with applications for permits for the discharge of dredged or fill material into navigable waters are contained in Chapter II, Title 33, CFR. Guidelines prepared by the Administrator of the EPA for the selection of disposal sites are available in Part 230, Chapter I, Title 40, CFR.

Although Section 404 refers to navigable waters, regulation of the discharge of dredged or fill material under Section 404 has been applied by the Secretary of the Army to all waters affected by the Water Pollution Control Act, referred to as "waters of the United States." This term is defined in the regulations as including

> (a) The term "waters of the United States" means
>
> (1) All waters which are currently used, or were used in the past, or may be susceptible to use in interstate or foreign commerce, including all waters which are subject to the ebb and flow of the tide;
>
> (2) All interstate waters including interstate wetlands;
>
> (3) All other waters such as intrastate lakes, rivers, streams (including intermittent streams), mudflats, sandflats, wetlands, sloughs, prairie potholes, wet meadows, playa lakes, or natural ponds, the use, degradation or destruction of which could affect interstate or foreign commerce including any such waters:
>
> (i) Which are or could be used by interstate or foreign travelers for recreational or other purposes; or
>
> (ii) From which fish or shellfish are or could be taken and sold in interstate or foreign commerce; or
>
> (iii) Which are used or could be used for industrial purpose by industries in interstate commerce;
>
> (4) All impoundments of water otherwise defined as waters of the United States under the definition;
>
> (5) Tributaries of waters identified in paragraphs (a) (1) through (4) of this section;
>
> (6) The territorial seas;
>
> (7) Wetlands adjacent to waters (other than waters that are themselves wetlands) identified in paragraphs (a) (1) through (6) of this section.
>
> Waste treatment systems, including treatment ponds or lagoons designed to meet the requirements of CWA (other than cooling ponds as defined in 40 CFR 123.1 l(m) which also meet the criteria of this definition) are not waters of the United States.[107]

As a result of this broad definition of waters of the United States, activities conducted throughout all of the interior of the United States are subject to the jurisdiction of the Corps of Engineers under Section 404, if such activities involve the discharge of dredged or fill material into virtually any standing or moving body of water or their tributaries, whether or not they are perennial and adjacent wetlands. Moreover, intermittent streams and isolated standing bodies of water that are not a part of a tributary system to interstate waters or to navigable waters, are included. Most importantly, wetlands, wherever occurring, come within the definition. Consequently, a mining operation may be subject to permit requirements under Section 404 if it deposits any material into a drainage channel or depression where water exists or may periodically accumulate or flow.

The most pervasive impact of Section 404 on mining operations occur in connection with wetlands, which are defined as follows:

> (b) The term "wetlands" means those areas that are inundated or saturated by surface or groundwater at a frequency and duration sufficient to support, and that under normal circumstances do support, a prevalence of vegetation typically adapted for life in saturated soil conditions. Wetlands generally include swamps, marshes, bogs, and similar areas.[108]

Wetlands have become vital areas of concern because of the broad range of wildlife that they support and because of the key role they play in the natural ecosystem. Their protection constitutes a significant, but important, extension of Corps jurisdiction beyond matters related to commerce on navigable bodies of water.

Discharge of Dredged or Fill Material: Section 404 refers to the discharge of dredged or fill material. *Dredged material* is any material excavated or dredged from waters of the United States.[109] A *discharge of dredged material* occurs when there is addition of dredged material to a specified discharge site located in waters of the United States and the runoff or overflow from a contained land or water disposal area.[110] The term does not include plowing, cultivation, seeding, or harvesting for the production of food, fiber, and forest products.

Fill material is any material used for the primary purpose of replacing an aquatic area with dry land or of changing the bottom elevation of a water body.[111] It does not include any pollutant discharged into water primarily to dispose of waste, as that activity is regulated under Section 402 of the Water Pollution Control Act.

A discharge of fill material occurs when fill material is added into waters of the United States, including the placement of fill that is necessary to the construction of any structure in a water of the United States; the building of any structure or impoundment requiring rock, sand, dirt, or other material for its construction; site-development fills for recreational, industrial, commercial, residential, and other uses; causeways or road fills; dams and dikes; artificial islands; property protection and/or reclamation devices such as riprap, groins, seawalls, breakwaters, and revetments; levees; fill for structures such as sewage treatment facilities, intake and outfall pipes associated with power plants and subaqueous utility lines; and artificial reefs.[112] The term does not include plowing, cultivating, seeding, and harvesting for the production of food, fiber, and forest products.

The list of activities for which a permit is required illustrates that there are many surface activities related to the operation of mines located near waters of the United States as defined in the Act that could trigger the requirement for a permit under Section 404. Such activities would include the construction of bridges, the placement of culverts, the alteration of natural stream channels, the laying of utility pipelines, the construction of diversion dams, the construction of bulkheads along stream cuts, the placement of waste dumps, the excavation of open pit or strip mines, and most importantly, the disturbance of any wetlands.

General and Individual Permits: Recognizing the difficulty of issuing separate permits for all such discharges, the Secretary of the Army has established a system that provides for the issuance of individual and general permits. Specific statutory authorization to issue general permits on a state, regional, or nationwide basis for any category of activities involving discharges of dredged or fill material has been given to the Corps of Engineers.[113]

A general permit has been issued for discharges of dredged or fill materials into certain kinds of waters and watercourses.

General permits, described in Part 330 of Title 33, CFR, have been issued as of the date of this writing for 26 specified authorized activities.[114] Some of these are important to the mining industry.

One permitted activity directly related to mining pertains to structures, works, and discharges associated with surface coal mining activities permitted under the Surface Mining Control and Reclamation Act of 1977, provided that the appropriate district engineer is given the opportunity to review the permit application and the district or division engineer makes a determination that the individual and cumulative adverse effects on the environment from the structures, work, or discharges are minimal.[115]

Another permitted activity important to mining covers discharges causing the loss or substantial modification of less than 10 acres (4 ha) of waters of the United States, including wetlands, into (1) nontidal rivers, streams, and their lakes and impoundments, including adjacent wetlands, that are located above the headwaters, and (2) other nontidal waters, including adjacent wetlands, that are not part of a surface tributary system to interstate waters or navigable waters of the United States.[116] Prenotification of the district engineer is nonetheless required for such discharges that cause the loss or substantial adverse modification of 1 to 10 acres (0.4 to 4 ha) of such waters, including wetlands.[117] The term *headwaters* means the point on a nontidal stream above which the average annual flow is less than 5.0 ft^3/sec (0.039 m^3/s), using the mean annual area precipitation, area drainage basin maps, and the average runoff coefficient, or by similar means.[118] For streams that are dry during long periods of the year, district engineers may establish the headwater point as that point on the stream where a flow of 5.0 ft^3/sec (0.039 m^3/s) is equalled or exceeded 50% of the time. The surface area of a lake is determined at the ordinary high-water mark.

To rely on the general permit for these remote activities, a number of specific conditions must be met including requirements that no significant disruption of the movement of aquatic life indigenous to the waterbody occur, that the structure of fill be properly maintained, and that management practices listed in the regulations[119] be followed. The general permit does not remove the necessity of complying with applicable local or state laws.[120] Other categories of discharges covered by a general permit include (1) survey activities including core sampling, seismic exploratory operations, and plugging of seismic shot holes and other exploratory-type boreholes; (2) outfalls and associated intake structures subject to NPDES permits; (3) dredged or fill material placed as backfill or bedding for utility-line crossings; (4) material discharged for minor bank stabilization; (5) minor road-crossing fills, including all attendant features, both temporary and permanent, that are part of a single and complete crossing of a nontidal water body; (6) fill placed incidental to the construction of bridges, except approach fills and causeways; (7) the repair, rehabilitation, or replacement of any previously authorized currently serviceable fill, or of any currently serviceable fill discharged prior to the requirement for authorization; and (8) discharges into waters other than wetlands that do not exceed 10 yd^3 (7.6 m^3) as part of a single project provided the material is not placed for the purpose of stream diversion. Once again, any such discharges must be performed in accordance with conditions and limitations set forth in the regulations application to the general permit.[121]

In addition to the general permits granted under the regulations, the District Engineer of the Corps of Engineers may issue general permits for certain clearly described categories of structures of work requiring permits, including discharges of dredged or fill material.[122] After such a local general permit has been issued, individual activities falling within those categories will not require individual permit processing, unless the District Engineer determines, on a case-by-case basis, that the public interest requires individual review. Appropriate conditions will be included in such general permits.

The Water Pollution Control Act requires that such general permits cover activities that will cause only minimal adverse environmental effects when performed separately and will have only minimal cumulative adverse effects on the environment.[123] Thus, if the cumulative effect of discharges made under a general permit begin to have a severe adverse effect on the aquatic environment, further restrictions may be required.[124]

The division engineer has discretion to modify, add regional conditions, or modify general permits on a case-by-case basis based upon concerns for the aquatic environment as expressed in the guidelines published by EPA.[125] This authority may be significant in cases where the proposed activity is controversial.

Six principal exemptions from permit requirements are provided for in Section 404. Of importance to the mining industry is an exception from Section 404 regulation for construction and maintenance of temporary roads for moving mining equipment.[126] In such road construction, best management practices must be followed to insure that flow and circulation patterns and chemical and biological characteristics of the navigable waters are not impaired, that the reach of the navigable waters is not reduced, and that any adverse effects on the aquatic environment will be otherwise minimized.

Exceptions also exist for discharges in connection with the maintenance, including emergency reconstruction of recently damaged parts, of currently serviceable structures such as dikes, dams, etc., and discharges for the purpose of construction of a temporary sedimentation basin on a construction site that does not include placement of fill material into navigable waters.[127]

Procedures: Procedures for processing Section 404 permits are established under Part 325 of Chapter II, Title 33, CFR. Unless an approved state plan is in effect, applications for permits are made to the District Engineer of the Corps of Engineers in charge of the district where the proposed activity is to be performed.[128] Forms are provided. Public notice will be made of the application.[129]

An important consequence of a requirement that an individual permit be obtained is that NEPA review is triggered. An environmental impact analysis can be expected in nearly all instances. If the district engineer believes that the proposed activity would significantly affect the quality of the human environment, he must either prepare an environmental impact statement or cooperate with other agencies that may act as the lead agency in preparing the EIS for the project.[130]

If water quality certification for the proposed activity is necessary under Section 401 of the Water Pollution Control Act, certification must be given by the state or appropriate interstate water pollution control agency prior to the issuance of a permit to the effect that any discharge will comply with applicable effluent limitations and water quality standards.[131]

Applications for permits for the discharge of dredged or fill material at specific disposal sites are reviewed in accordance with guidelines promulgated by the Administrator of the EPA.[132] Permits can still be issued, however, notwithstanding that they would be denied if the guidelines alone were applied. The EPA and the Department of the Army have recently entered into a Memorandum of Agreement (MOA)[133] which provides guidance to agency field personnel on the implementation of certain guideline requirements. The MOA primarily focuses on standard individual permits under Title 33, CFR § 325.5(b)(l). EPA and the Corps must adhere to the guidance provided in the MOA when considering mitigation requirements for standard permit applications. The MOA will be used by the agencies in their review of standard individual permit applications received on or after Feb. 7, 1990, for compliance with EPA guidelines. The Administrator of the EPA has the authority to prohibit the specification of the defined areas as a disposal site, if he determines that the proposed discharge will have an unacceptable adverse effect on the municipal water supply, fishing areas, wildlife, or recreational areas.[134]

After review of the application in accordance with the regulations and the holding of a public hearing, the District Engineer decides whether or not the permit should be issued and what conditions are to be attached.[135]

Permits may authorize both the work and the resulting use.[136] Authorizations continue in effect until they automatically expire or are modified, suspended, or revoked. Authorization for the existence of the structure or other form of alteration of the waterways is usually for an indefinite duration; authorizations for construction work or activity, however, will specify time limits for accomplishing the work or activity.[137]

The District Engineer has authority to modify, suspend, or revoke a permit "as may be made necessary by considerations of the public interest."[138]

Administration by States: Administration of individual and general permit programs except as to waters used or reasonably susceptible to use to transport interstate or foreign commerce, may be undertaken by the states upon obtaining approval of a state plan meeting specified requirements.[139] States with approved programs, nevertheless, must transmit a copy of each permit application and a copy of each proposed general permit to the Administrator of the EPA for comment, except for certain categories excepted by the Administrator.[140] If the Administrator objects to the issuance of a permit, a public hearing will be held on request of the state. Within 30 days after the public hearing or 90 days after the objections are made and no public hearing is requested, if the state does not resubmit the permit revised to meet the objections, the Chief of the Corps of Engineers may issue the permit revised in accordance with the objections.[141]

Control Through Section 208 of Areawide Waste-Treatment Plans: Control of the discharge or placement of dredged or fill material into navigable waters may also be accomplished through state areawide waste-treatment plans promulgated under Section 208.[142] If such a program is approved and the state is also administering an approved program under Section 404, an operator conducting activities involving the placement or discharge of dredged or fill material into navigable waters in accordance with approved best management practices need not obtain an individual permit or comply with a general permit.[143]

3.4.3.4 Safe Drinking Water Act

The Safe Drinking Water Act[144] requires that national health standards be established for water quality in public water systems.[145] Under the Act, primary drinking-water standards and standards applicable to public water systems are imposed, maximum contaminant levels are established, and monitoring and sampling requirements are dictated. States have the primary enforcement responsibility if they adopt regulations and set up enforcement procedures. A mine facility would have to comply with the Act if it provides water where there are more than 15 service connections, or if the system regularly services an average of at least 25 individuals daily (including miners) 60 days out of the year.[146]

The Act also contains provisions relating to the protection of underground drinking water through control of underground injections, which may endanger underground drinking-water sources.[147] EPA has promulgated regulations that establish minimum requirements for state underground injection-control programs. The regulations apply to "well injections," which includes drilled, bored, or driven wells and dug wells having depths greater than the largest horizontal dimension.[148]

Five separate classes of wells are described in the regulations.[149] Class III wells cover wells into which fluids are injected to extract minerals. Examples are underground in situ leaching wells and Frasch process sulfur wells.[150] Permits are required for Class III wells from the state permitting agency.[151] Various conditions designed to protect groundwater, such as preventing leakage into aquifers and ensuring careful plugging and abandonment, are incorporated into the permit.[152] An entire project may be covered by a single permit.[153]

Class I wells are those that dispose of hazardous or nonhazardous wastes beneath the lower-most formation containing an underground source of drinking water within a one-quarter-mile radius of the wellbore.[154] Class I wells, like Class III wells, must be permitted.[155]

Miscellaneous types of wells, some of which are particular to mining operations, are included in the general Class V category. Examples include sand backfill and other backfill wells used to inject sand, mill tailings, or other solids into mined-out portions of underground mines; wells used for solution mining of conventional mines such as stope leaching; injection wells used in experimental technologies; and injection wells used for in situ recovery of lignite, coal, for sands, and oil shale.[156] Class V wells are subject to regulatory requirements, but they are not necessarily required to be permitted, depending upon state law.[157] Owners and operators of such wells are required to provide notice to the appropriate authority that they exist and to provide certain information about them.[158]

Wells being used at a mining operation for the subsurface emplacement of fluids, such as solution-mining wells, would be covered under the regulations. Excavations such as tailings ponds and wastewater settling pits from which contaminated water can percolate into underground water systems are not directly regulated under the Act.

The primary enforcement responsibility under the Safe Drinking Water Act rests with states. The Act provides that states prohibit underground injection, not authorized by a permit or rule, and that no underground injection be permitted that will endanger drinking-water sources.[159]

3.4.3.5 Air Pollution Control

Emissions from facilities such as conveyor belts, washing plants, crushers, and concentrators, as well as fugitive dust from mining and materials handling activities, cause mining operations to be subject to certain requirements under federal and state air pollution control laws.

Federal controls over air pollution have been established under the Clean Air Act of 1970, as amended by the Clean Air Act Amendments of 1977 and 1990. The Clean Air Act Amendments of 1990 ("1990 Amendments") added a comprehensive program to regulate emissions of toxic air pollutants (Title III), created a new permit program (Title V), significantly revised nonattainment provisions (Title I), and strengthened the existing enforcement provisions. This Act, in turn, has resulted in the adoption by many states of extensive air pollution control legislation under the state delegation provisions of the federal law. The four aspects of the Act having the most significant impact on mining operations are the provisions relating to (1) implementation plans for national primary and secondary ambient air quality standards, (2) permit requirements, (3) standards of performance for new stationary sources, (4) emission standards for hazardous air pollutants, (5) air toxics standards, and (6) restrictions preventing significant deterioration of clean air areas.

Implementation Plans for National Primary and Secondary Ambient Air-Quality Standards: The program for adopting implementation plans for national primary and secondary ambient air quality standards has several stages. The first is the designation of air quality control regions in interstate or major intrastate areas for the attainment and maintenance of ambient air quality standards (NAAQS).[160] Achievement of improved air quality under the Act is measured on the basis of conditions in the designated air quality control regions.

For the purpose of establishing national primary and secondary ambient air quality standards, the Administrator of EPA publishes a list that includes each air pollutant that has an adverse effect on public health or welfare and the presence of which in the ambient air results from numerous or diverse mobile or stationary sources.[161] The Administrator then issues air quality criteria for each air pollutant on the list, including information on conditions and factors relating to the impact on public health or welfare.[162] The Administrator also publishes information on air pollution control techniques, including data on available technology, costs of installation and operation, energy requirements, emission-reduction benefits, environmental impact of the emission-control technology, and alternative methods of prevention and control of air pollution for each pollutant.[163]

National primary ambient air quality standards and national secondary ambient air quality standards are established by EPA for each air pollutant for which air quality criteria are issued.[164] Primary ambient air quality standards establish minimum permissible concentrations of a pollutant that are necessary to protect the public health. Secondary ambient air quality standards establish a minimum level of air quality that must be obtained in order to protect the public welfare, including such considerations as conditions of vegetation, wildlife, odor, and other factors which, although not constituting dangers to public health, do adversely affect the general quality of life. Essentially, the standards set the maximum amount of each pollutant that will be permitted in the atmosphere.

Primary and secondary air quality standards have been established for particulate matter, sulfur dioxide (SO_2), nitrogen oxides (No_x), ozone (O_3), carbon monoxide (CO), and lead (Pb).[165] Mining operations are most concerned with particulate matter because that category covers fugitive dust.[166] The primary and secondary NAAQS for particulate matter now applies to particles equal to or less than 10 µm in aerodynamic diameter referred to as PM-l0.[167]

The states determine how national air pollution objectives are to be reached. After the promulgation of a national primary or secondary ambient air quality standard for any air pollutant, each state must adopt and submit to EPA a plan that provides for implementation, maintenance, and enforcement of such primary or secondary standards in each air quality control region (or portion thereof) within the state.[168] These are referred to as state implementation plans, or SIPs. If such a plan is approved by EPA after application of the rigid standards set forth in the Act, the state is given responsibility for the enforcement of those provisions of the plan that will insure that the air quality standards are attained and maintained. If the state's plan is not approved, EPA will adopt regulations setting forth an implementation plan for the state. Such implementation plans may require installation of extensive pollution-control equipment at existing sources, so that emissions will be reduced to the extent necessary for the attainment of the national air quality standards.

Few existing mining operations are affected by the air quality standards achievement programs because they are generally located outside nonattainment areas, and those that are, by this time, are at least generally aware of the nature of implementation programs and their impact. State programs for the achievement of air quality standards do, however, have an impact on the construction and operation of new sources.

Permit Program: Title V of the 1990 Amendments adds a new uniform federal permitting program. Sources must obtain permits if they emit 100 tons per year ("TPY") or more of any air pollutant or 10 TPY of any hazardous air pollutant (or 25 TPY of any combination of any hazardous air pollutant). Moreover, sources which are located in certain nonattainment areas may need permits if they emit as little as 10 TPY of a criteria pollutant.

Under Title V, sources are required to prepare and submit permit applications by November 1995. States which have received primacy from the EPA to operate the permit program have up to 18 months to review the permit applications and act on the applications. The permit application must include a schedule for compliance with the emission standards, including periodic compliance certifications. Permits are to be issued for a fixed period, not to exceed five years.

The permit program includes many unique aspects including permit "shield" provisions. The states may draft their permit programs so that compliance with the permit constitutes compliance with other Clean Air Act provisions. The permit program also requires the imposition of permit fees sufficient to recover direct and indirect costs of the states permit program. Under proposed regulations, the EPA presumes that the appropriate permit fee should be at least $25 per ton of regulated pollutant (which is defined very broadly). The fees are to be imposed on the first 4000 tons of regulated pollutant (e.g., up to $100,000); however, the fees may be larger if the source emits more than one regulated pollutant or the state chooses not to restrict fees to the 4000-ton limit.

Standards of Performance for New Stationary Sources: Of more direct concern to the mine operator are the requirements that affect new operations. One or more of the facilities at such an operation may be considered to be a new stationary source, the emissions from which must meet federal standards of performance (NSPS).

A new source is any building, structure, facility, or installation that emits or may emit an air pollutant, the construction or modification of which is commenced after the publication of regulations or proposed regulations establishing an emission standard applicable to such a source.[169] An existing source can be considered to be a new source subject to applicable performance standards if it is modified to the extent that the methods of operation are changed so as to result in an emission increase or the emission of an air pollutant not previously emitted.[170] New source performance standards are to be established for new sources that contribute significantly to air pollution that causes or contributes to the endangerment of the public health or welfare.

The Administrator of EPA is required to publish a list of categories of stationary sources and to publish regulations establishing federal standards of performance for new sources within each such category.[171] A standard of performance establishes allowable emission limitations for categories of stationary sources.[172] The standards are based on the degree of emission reduction achievable through the best technology the Administrator determines has been adequately demonstrated. If the Administrator concludes that it is not feasible to prescribe or enforce a standard of performance, he may instead promulgate a design, equipment, work practice, operational standard, or combination thereof, that reflects the best technological system of continuous emission reduction that the Administrator determines has been adequately demonstrated.[173]

Of the categories of new stationary sources for which standards of performance have been promulgated as of the date of this writing, the only categories of obvious direct concern to underground mining operators are those for portland cement plants, coal preparation plants, metallic mineral processing plants, phosphate rock plants, and nonmetallic mineral processing plants, contained in subparts F, Y, LL, NN, and OOO of Part 60, Title 40, CFR.

The Act provides that states may implement and enforce standards of performance for new sources if the state develops a plan found to be adequate by the Administrator of EPA.[174] However, EPA still retains authority to enforce any applicable standard of performance.

The Clean Air Act itself does not require specifically that permits be obtained for new sources. The owner or operator has the legal responsibility for compliance with an applicable standard, and he must comply with regulations that contain extensive requirements relating to reporting and data gathering. A permit for a new source, however, will be required under state new-source permitting authority or under state programs for the implementation of primary or secondary ambient-air standards for all new or modified sources with the potential (design capacity) to emit 100 TPY or more of a given pollutant.

If the affected facility is located in a state not having an approved new-source review program, an owner or operator subject to the performance standards must notify EPA of the anticipated startup date of an affected facility not more than 60 days or less than 30 days prior to such date.[175] The EPA must also be notified of the actual date of initial start-up within 15 days after such date.[176] Extensive records must be kept concerning the emissions and the effectiveness of emission-control equipment.[177]

Within 60 days after the maximum production rate is achieved, but not later than 180 days after initial start-up, performance tests for the affected facilities must be conducted and the results thereof sent to EPA.[178] Compliance with the new-source performance standards is to be determined by the application of performance tests specifically set forth in the regulations.[179]

In nonattainment areas (areas wherein any air pollutant exceeds any national ambient air quality standard for such pollutant), new major emitting facilities cannot be built unless SIP requirements are met and there first is secured an offset (greater than one to one) to existing emissions.[180] Under the 1990 Amendments, offset requirements may range from at least one to one to as high as two to one, depending upon the extent to which pollutant levels exceed the standard in the area.[181] The sources must also be controlled to a level reflecting the lowest achievable emission rate (LAER), which is the most stringent emission standard in any SIP (unless it is shown to be unachievable) or the lowest emissions any source in the same category has achieved in production.

Emission Standards for Hazardous Air Pollutants: Section 11a of the Clean Air Act gives authority to EPA to set national emission standards for hazardous air pollutants.[182] Hazardous air pollutants are those to which no ambient air quality standard is applicable and which, in the judgment of the Administrator of EPA, may cause or contribute to an increase in mortality or an increase in serious irreversible, or incapacitating reversible, illness.[183] In contrast to the provision authorizing establishment of ambient air quality standards, this section of the Clean Air Act does not require that the pollutant come "from numerous or diverse mobile or stationary sources;"[184] Only stationary sources must comply.[185]

The Administrator is directed to prescribe emission standards or design, equipment, work practice, or operational standards for each hazardous air pollutant that he identifies on a list, which standard establishes the level that provides an ample margin of safety to protect the public health from such hazardous air pollutant.[186] They are referred to as NESHAPs (National Emission Standards for Hazardous Air Pollutants). After the effective date of any such emission standard, no person may construct any new source or modify any existing source that will emit a hazardous air pollutant in violation of the standards.[187] Existing sources must comply with the standard 90 days after the effective date, unless waivers or exemptions are obtained in accordance with the Act.[188]

Regulations of EPA prescribe that permission of EPA must be given prior to the construction or modification of a stationary source that emits hazardous air pollutants.[189] Within 60 days of receipt, EPA must notify the owner or operator of its approval or its intent to deny approval of construction or modification.[190]

States may be given authority over implementation and enforcement of standards for hazardous air pollutants upon submission and approval by the Administrator of an adequate plan. However, the Administrator is still empowered to enforce applicable federal emission standards.[191]

Emission standards for hazardous air pollutants have been established for radon 222 emissions from underground uranium mines and from licensed mill tailings, asbestos, beryllium, and mercury.[192] Emission standards have also been established for such hazardous air pollutants as arsenic, benzene, vinyl chloride, and radionuclides.

Air Toxics: Title III of the 1990 Amendments completely revised the NESHAP provisions of Section 112. Under the air toxics provisions, the Clean Air Act establishes a list of 189 toxic air pollutants that are to be primarily regulated through technology-based standards.[193] Sources subject to regulation include "major" and "area" sources. Major sources are those sources that have the potential to emit 10 tons or more of any hazardous pollutant (or alternatively any source that emits 25 TPY of any combination of any hazardous pollutants). Area sources include any stationary sources that are not major sources, other than vehicles.

The EPA is required to publish a list of sources it proposes to develop standards for and regulate by Nov. 15, 1991. These sources are to be subject to standards that represent the maximum available control technology (MACT).[194] The EPA is to prepare a standards promulgation schedule and the first round standards must be promulgated by Nov. 15, 1992. All standards are to be promulgated by November 2000.

New sources must generally comply with the new standards immediately. If construction of the new source began after the standards were proposed then the source is allowed up to three years to comply if the final standard is more stringent than the proposed standard. Generally, existing sources have three years after promulgation to comply with the standard. Certain sources, including sources of mining waste, may obtain a compliance extention of up to three additional years.

In setting emission standards, the EPA is allowed to develop less stringent standards for existing sources. For existing sources the emission standard must be at least as stringent as the emission control that is achieved in practice by the best controlled similar sources. The 1990 Amendments include a savings clause that preserves any standards developed under the NESHAP program prior to November 15, 1990. Like the NESHAP program, states may develop programs under Section 112 to obtain enforcement authority for the air toxics program. Since the radionuclide rule has been the subject of judicial challenge, the 1990 Amendments retain the radionuclide standard for some sources and suspend the standard for other sources.

Enforcement: Violation of provisions of the Clean Air Act may result in the issuance of a permanent or temporary injunction or imposition of a fine or imprisonment.[195]

The Administrator or the state, upon the approval of the Administrator, may require the owner or operator of any emission source to monitor and sample emissions, to maintain and provide records thereof, and to permit inspection of the premises in which an emission source is located.[196]

Prevention of Significant Deterioration of Existing Air Quality (PSD): Special provisions have been enacted to prevent significant deterioration of the air quality in areas where the quality of the air is better than required under national ambient air quality standards.[197] Such zones may be classified by the states into areas based on degrees of air quality deterioration, measured by concentrations of sulfur dioxides and particulates, to be permitted therein. EPA has also promulgated allowable amounts for nitrogen oxides based on the same classification scheme.[198] In other areas, referred to as nonattainment areas, the objective is to reduce pollution to levels consistent with national ambient air quality standards.

Areas are to be classed as I, II, or III depending upon the maximum allowable increase of sulfur dioxide, particulate matter, and nitrogen oxides over the baseline concentration of such pollutants to be permitted in the area.[199] Generally, class I areas include National Parks, National Wilderness Areas, National Monuments, and similar regions of unique beauty. Most areas are class II, where limited growth can be permitted. Class III areas are those where industrial development can occur. All state air quality control regions not designated as class I initially have been designated as class II. All areas redesignated as class I under regulations in effect prior to passage of the Clean Air Act Amendments of 1977 retain that status.[200] An area which as of Aug. 7, 1977, exceeded 10,000 acres (4047 ha) in size and was a national monument, a national primitive area, a national preserve, a national recreational area, a national wild and scenic river, a national wildlife refuge, a national lakeshore or seashore, and a national park or national wilderness established after Aug. 7, 1977, which exceeds 10,000 acres (4047 ha) in size may be redesignated only as class I or II. An area may not be designated as class III if such redesignation will cause or contribute to concentrations of any pollutant that exceed any maximum allowable increase or maximum allowable concentration permitted under the classification of any adjoining area.[201] Thus areas adjoining class I areas are not likely to be redesignated as class III.

Once an area has been designated, no "major emitting facility" may be constructed therein without a permit from the state, or from EPA if the state does not have an approved PSD plan.[202] To obtain a permit, the owner or operator of the facility must demonstrate that emissions from the construction or operation of such a facility will not cause or contribute to air pollution in excess of any maximum allowable increase or maximum allowable concentration of sulfur oxides, particulates, or nitrogen oxides more than one time per year and that the proposed facility is subject to the best available control technology for each pollutant subject to regulation under the Act.[203]

Best available control technology (BACT) requires the achievement of an emission limitation based upon the maximum degree of reduction of each pollutant subject to regulation under the Act emitted from a major emitting facility that the permitting authority determines, on a case-by-case basis, is achievable for such a facility through the application of production processes and available methods, systems, and techniques.[204]

Major emitting facilities are specifically designated categories of stationary sources of air pollutants that emit, or have the potential to emit, 100 TPY or more of any air pollutant and any other source with the potential to emit 250 TPY or more of any air pollutant.[205] The designated categories include coal-cleaning plants with thermal dryers, portland cement plants, phosphate rock-processing plants, and taconite ore-processing facilities. Although not specifically named in any of the 28 categories, mining operations contiguous or adjacent to such facilities may be included.[206] Both new operations and major modifications of existing operations can be considered major emitting facilities.[207]

Whether a facility is a major emitting facility is determined by its capacity to emit a pollutant under its physical and operational design, which includes air pollution control equipment and restrictions on hours of operation or on the type of material combusted, stored, or processed.[208] Thus, even though uncontrolled emissions may be more than the threshold limits for categorizing a facility as a major emitting facility, it will be considered to be a major emitting facility only if its controlled emissions will exceed those limits.

Unless a major emitting facility is exempted, the owner or operator must satisfy rigorous air quality review standards and conduct ambient air quality monitoring and additional impact analyses as part of the permitting process. Such requirements, as described in the regulations,[209] can constitute a significant burden. Generally, a full year of monitoring of all listed pollutants that a source would have the potential to emit in significant amounts, or as to major modifications in amounts constituting a significant net increase, is required.[210]

Of importance to the mining industry is an exemption applicable to fugitive dust. Fugitive dust is not considered in determining whether a source is a major emitting facility, unless the source is one of the 28 listed sources or is being regulated by a new source performance standard or national emission standard for hazardous air pollutants in effect as of Aug. 7, 1980.[211] Fugitive emissions are those emissions that could not reasonably pass through a stack, chimney, vent, or other functionally equivalent opening.[212] Fugitive dust may include emissions from haulage roads, wind erosion of exposed soil surfaces and soil storage piles, and other activities in which the soil is removed, stored, transported, or redistributed.

Copies of permit applications received by a state are transmitted to the Administrator of EPA for review by federal officials to insure that the proposed emitting facility will not have an adverse effect on air quality in class I areas. A permit will not be issued unless the owner or operator demonstrates that emissions of particulate matter, sulfur dioxide, and nitrogen oxides from the permitted facility and secondary sources established in the same area as a result of the facility will not cause concentrations exceeding maximum allowable increases.[213] A variance may be obtained if emissions of sulfur dioxide from the source exceed the allowable increments within specified limits for a period of exposure of 24 hr or less on not more than 18 days during a year.[214]

Once the permissible increment for increases in pollutant concentration over the baseline concentration in a PSD classified area are utilized, including contributions by unpermitted minor sources and mobile sources, then no additional discharges will be permitted.[215] Thus many PSD areas ultimately will be closed to any additional industrial activities that cannot be carried on without the discharge of sulfur dioxide or particulates.

3.4.3.6 Federal Surface Management Regulations Concerning Unpatented Mining Claims

Under federal mining laws (see chapter 3.2), a person has a statutory right to enter upon and use available public lands for mineral prospecting, exploration, development, and extraction. This statutory right, however, carries with it a responsibility to conduct such activities in a way that will prevent undue degradation of the land and will ensure that reasonable reclamation takes place. The federal government has adopted various regulations to accomplish these objectives. The pertinent regulations are contained in Titles 36 and 43, CFR. These regulations do not preempt, but are supplemental to, state laws and regulations relating to mining operations and reclamation,[216] so the operator must also be familiar and comply with applicable state and local regulations, including applicable zoning provisions.[217]

The geographic location of the unpatented mining claim determines which group of federal regulations must be followed. Separate regulatory programs have been established for claims located on land administered by the Bureau of Land Management (BLM) and land administered by the Forest Service. Other regulations may apply if the land is within a designated wilderness study area. The regulations apply not only to unpatented mining claims but to tunnel claims and unpatented millsite claims as we11.[218]

BLM-Administered Land: If the claim is located on BLM-administered land and is not within a designated wilderness study area, exploration and mining activities are governed by regulations contained in Title 43, CFR subpart 3809. Increasingly stringent regulations are imposed on the operator according to the extent of anticipated surface disturbance. The applicable regulations can be divided into three rather distinct categories.

Casual Use—If the intended activity amounts only to "casual use," no notification to or approval by the BLM is required. The operator is free to conduct casual-use activities at will and without notice, so long as they are carried out in a way that will prevent unnecessary degradation of the land.[219] "Casual use" is defined as activities ordinarily resulting in only negligible disturbance of the surface of the land. Activities are generally considered casual use if they do not involve the use of explosives, mechanized earthmoving equipment, or motorized vehicles in areas designated as closed to off-road vehicles.[220]

Surface Disturbance of Five Acres or Less—If the annual cumulative surface disturbance from activities on the claim will exceed casual use but will be kept to 5 acres (2 ha) or less (including disturbance associated with establishing access to the claim), the operator is required to notify the BLM in writing at least 15 days prior to the commencement of operations. Approval by the BLM of the proposed activities is not required, although consultation with the BLM may be required if the activities include the construction of access roads.[221] The written notice, which may be in the form of a letter, must include the name and address of the claimant and the operator; the names and BLM serial numbers of the mining claims on which disturbance will occur; a statement describing the activities that will take place, including their location, the approximate commencement date, the location of access routes to be constructed, and the type of equipment to be used; a statement that reasonable measures will be taken to prevent unnecessary degradation of the land during operations; and a statement that appropriate reclamation will be completed as set forth in the regulations.[222] No reclamation bond to recover the cost of reclaiming the land is required, but once reclamation has been completed the operator is required to notify the BLM so that an inspection of the area can be made.[223]

Generally, the 5-acre (2 ha) provision will allow most exploration activities and perhaps limited extraction activities to take place without having to file anything more than a written notice with the BLM, provided the disturbed area is reclaimed in the manner required by the regulations.[224]

Surface Disturbance of More Than Five Acres—When more than 5 acres (2 ha) of surface will be disturbed by the contemplated activities, a "plan of operations" must be submitted to and approved by the BLM prior to commencement.[225] While no special form is required, the plan of operations must include the name and address of the claimant and operator; the names and BLM serial numbers of the mining claims involved; a topographic map showing existing and proposed access routes; a description of the proposed operations, describing the timeframe and manner in which they will be conducted; a listing of measures to be taken to prevent undue degradation of the environment and to accomplish reclamation of the land as required in the regulations; and a statement of measures to be taken to maintain the area during extended periods of nonuse.[226]

Once the plan is submitted, the BLM has 30 days to review and act on it. The BLM can respond to the plan in any of several ways. It can (1) approve the plan, (2) require additions to or modifications of the plan, (3) determine that an environmental impact statement is required and delay action on the plan until 30 days after completion of a final EIS, (4) delay acting on the plan while the BLM complies with provisions of the Endangered Species Act or the National Historic Preservation Act, or (5) take an additional period of time (up to 60 days) to complete the review.[227] The same process must be followed if the operator decides to significantly modify the plan after its initial approval.[228] It should be noted that all information and data submitted by the operator will be available for public examination in accordance with the Freedom of Information Act unless the information is specifically identified by the operator as containing trade secrets or confidential commercial or financial information.[229]

The regulations require that while final approval is pending, the BLM approve on an interim basis any operations that may be necessary for timely compliance with state and federal laws, subject only to conditions needed to prevent undue degradation of the land. Such state and federal laws would include those relating to assessment work, discovery work, claim staking, and the like.[230]

Before a plan of operations encompassing land not previously covered by an approved plan can be approved, the BLM must conduct an environmental assessment of the proposed operation to identify the environmental impacts of the proposed activities, determine whether an EIS is required, and assess the adequacy of the operator's proposed reclamation measures.[231] This assessment will be performed during the 30-day review period. If the plan is approved the BLM may, at its discretion, require the operator to furnish a bond in an amount determined by the BLM to cover the estimated cost of reclaiming the land. No bond is required if the operations will cause only minimal disturbance to the land.[232]

All exploration and mining operations, whether casual, under notice, or by plan of operations, must be carried out so as to prevent unnecessary or undue degradation[233] of the land and must be completed in compliance with specific regulations relating to air and water quality, solid wastes, fisheries, wildlife and plant habitat, cultural resources, and survey monuments.[234] The BLM may monitor and inspect any such operations to ensure compliance with the regulations. If the operator is found to be in noncompliance with the regulations or with the provisions of an approved plan, the BLM can issue a notice of noncompliance. Failure to remedy the problem as specified in the notice of noncompliance may subject the operator to a court order prohibiting further operations and to monetary damages.[235]

Finally, any operator adversely affected by a decision of the BLM has the right to appeal the decision. An appeal is made first to the BLM State Director of the applicable state and thereafter, if necessary, to the Interior Board of Land Appeals. An appeal will not be considered unless it is made within 30 days after receipt of the adverse decision.[236]

Requirements for Claims Within BLM Wilderness Study Areas—A "wilderness study area" is an area of public land designated by the BLM as having wilderness characteristics, making it a candidate for inclusion in the federal government's wilderness preservation system.[237] Wilderness study areas are managed in a manner that will not impair their suitability for preservation as wilderness. The applicable regulations are designed to achieve this management objective[238] and are very similar to those established for activities requiring a plan of operations as discussed previously.

Subject to certain exceptions, before any exploration or mining operation on claims located within a wilderness study area begins, a plan of operations must be submitted to and approved by the BLM. The regulations expressly require submission and approval of a plan of operations if the proposed activity will involve the construction of access facilities, the destruction of trees greater than 2 in. (50 mm) in diameter at the base, the use of mechanized earthmoving equipment, the use of motorized vehicles in certain areas closed to such use, the placing of portable or fixed structures on the land for more than 30 days, the use of explosives, or the changing of a water course. On the other hand, no plan of operations is required for activities involving limited geophysical testing, the occasional removal of mineral samples, the use of motorized vehicles in permitted areas, or the making of minor improvements to existing access routes.[239] Additionally, no plan is required for ongoing operations that were in existence on Oct. 21, 1976, unless and until the operation is enlarged or is found by the BLM to be causing unnecessary degradation to the environment.[240]

In reviewing the plan of operations, the key issue is whether the proposed activities will impair the suitability of the area for preservation as wilderness. In analyzing that issue, the BLM can approve or reject the plan or require modifications or additions to preserve the wilderness character of the land.[241] The required contents and approval procedure for the plan of operations are substantially the same as those established for claims not within a wilderness study area. The same is true with respect to interim approval to conduct assessment work and the like, plan modifications, bond requirements, environmental assessments, treatment of confidential information, monitoring and inspection procedures, noncompliance remedies, and appeal rights.[242] The reclamation requirements for operations conducted within a wilderness study area are similar to but substantially more stringent than those pertaining to operations conducted outside of wilderness study areas.[243] Additionally, the operator must notify the BLM if operations are suspended for more than 30 days (except for seasonal suspensions) and must reclaim the site within one year after operations cease (unless additional time is allowed for good cause).[244]

Forest Service Administered Land: If the unpatented claim is located within a National Forest, exploration and mining activities are governed by regulations found in Title 36, CFR, Part 228, subpart A. The purpose of the regulations is to provide for exploration and mining activities while minimizing adverse environmental impacts on the forest resources.[245] The Forest Service's regulatory scheme is similar to that established for BLM nonwilderness areas.

Insignificant Surface Disturbance—Forest Service regulations do not use the term "casual use," but they do similarly provide that activities that will result in only insignificant surface disturbance can be conducted without notice to or approval from the Forest Service. Insignificant disturbance activities include such acts as using vehicles on existing roads, removing small mineral samples on an occasional basis, prospecting to the extent that it does not significantly disturb surface resources, marking and monumenting mining claims, and conducting subsurface operations with insignificant surface effects. Operations that do not involve the use of mechanized earthmoving equipment or the cutting of trees are also considered insignificant and are exempt from notice requirements.[246]

Notice and Plan of Operations—Prior to commencement of any action that might cause "significant disturbance of surface resources" (a term not defined in the regulations), the operator is required to notify the Forest Service.[247] The notice must identify the area involved, the nature of the proposed operations, the access route, and the method of transportation. The Forest Service then has 15 days to determine whether a plan of operations will be required. If the Forest Service decides that the proposed activity will likely result in significant surface disturbance, an approved plan of operations will be required.[248]

The plan of operations, if required, must include the name and address of the claimant and operator; a map showing the proposed area of operations, the existing and proposed access routes, and the approximate location of anticipated surface resource disturbance; and a description explaining the proposed operations and how they will be conducted, the type of existing and proposed access routes, the means of transportation to be used, the time period during which the proposed activities will take place, and the environmental protection measures to be taken.[249]

After receipt, the Forest Service has 30 days to consider the plan of operations and notify the operator of its decision. The Forest Service can (1) approve the plan, (2) require changes in or additions to the plan, (3) delay any decision for an additional period of time (up to 60 days), (4) delay any decision until a final EIS, if necessary, has been completed, or (5) conclude that no plan is required for the proposed operations.[250]

All mineral operations, whether conducted pursuant to a plan or otherwise, must be conducted so as to minimize adverse environmental impacts on forest resources and must comply with applicable state and federal environmental standards. Activities must also comply with specific environmental provisions relating to air and water quality, solid wastes, scenic values, fisheries and wildlife habitat, roads, and reclamation procedures as set forth in the regulations.[251]

The Forest Service follows provisions similar to the BLM's with regard to completion of an environmental analysis during the approval process, interim approval to conduct assessment work and the like, procedures for modifying an approved plan, bond requirements, treatment of confidential information, inspection procedures, noncompliance remedies, and appeal rights.[252]

Unlike the BLM, the Forest Service does not have a separate regulatory scheme for lands under wilderness review. Operations within Forest Service wilderness study areas are subject to the same requirements as operations outside of such areas, but wilderness study area operations must be conducted in a manner that will not impair the wilderness character of the land.[253] It can be expected that the Forest Service will carefully consider the nonimpairment requirement in any review of activities to be conducted on wilderness study lands.[254]

3.4.3.7 Federal Lease Operating Regulations (Other than Coal)

Operations for the extraction of hard minerals other than coal, conducted on federal lands under lease, permit, or contract from the federal government (which excludes mining claims), are governed by operating regulations set forth in Part 23 of Title 43, CFR.

Regulations[255] provide for the protection and conservation of nonmineral resources during operations for the discovery, development, surface mining, and onsite processing of minerals under permits, leases, or contracts issued pursuant to the Mineral Leasing Act of 1920[256] and the Mineral Leasing Act for acquired Lands.[257] No definition of surface mining is given in the regulations, but it is assumed for purposes of this discussion that surface activities incident to underground mining operations are subject thereto.

These regulations also establish the procedures to be followed by BLM in reviewing applications for permits or leases under the Mineral Leasing Act.[258] When such an application is submitted, a technical examination is made of the prospective effects of the proposed exploration or surface mining operations.[259] As part of the technical examination, a determination is made whether special measures are needed (1) to preserve and protect nonmineral resources, including recreational, scenic, historic, and ecological values; (2) to prevent air and water pollution; (3) to isolate toxic materials; or (4) to accomplish surface reclamation.[260] After the technical examination is completed, special provisions are drawn up setting forth such special measures that need to be taken. These requirements are made known to the applicant before the permit, lease, or contract is issued, and, if accepted, they are incorporated therein.[261] If the lands are under the jurisdiction of an agency other than the Department of the Interior, environmental protection requirements proposed by that agency generally will be attached to the lease, permit, or contract.[262]

These special requirements are referred to as stipulations, and they become part of the lease, permit, or contract. They may set forth detailed requirements pertaining to all aspects of environmental protection and surface reclamation.

Compliance with general objectives is achieved primarily through review by the mining supervisor of a mining plan, which must be approved before operations can be commenced. After approval, mining and milling must be performed in accordance with the plan. A mining plan must show, in detail, the proposed mining operations to be conducted. A mining plan must include the following information:

1. A statement of proposed methods of operating, including a description of the surface or underground mining methods.
2. A description of proposed roads or vehicular trails.
3. The size and location of structures and facilities to be built.
4. An estimate of the quantity of water to be used.
5. Identification of pollutants that are expected to enter any receiving water.
6. A design for the necessary impoundment, treatment or control of all runoff water and drainage workings so as to reduce soil erosion and sedimentation and to prevent the pollution of receiving waters.
7. A description of measures to be taken to prevent or control fire, soil erosion, pollution of surface water and groundwater, damage to fish and wildlife, and hazards to public health and safety.
8. A statement of the proposed manner and time of performance of work to reclaim areas disturbed by the operations.[263]

In most instances, provisions of the permit or lease will require revegetation of areas affected by the operations.[264]

A performance bond must be provided to assure that required environmental protection and reclamation activities are performed.[265]

After approval of the mining plan and commencement of operations, a report is required each year from the operator, containing a description of the operations that have been performed and a description of the reclamation work that has been done, including grading, backfilling, and planting.[266]

Failure of the lessee to take action in response to a notice of noncompliance with the terms of the lease or with the mining plan can result in suspension of operations and/or cancellation of the lease.[267]

As far as day-to-day planning and operation of a mining facility is concerned, at least when such operation is conducted on lands held under federal lease, the environmental protection requirements imposed by the lease terms and by the mining supervisor are the most significant. Close attention should be paid to requirements that are imposed, and close coordination should be maintained with the mining supervisor at all times.

3.4.3.8 Preservation of Historical and Archeological Data

Prior to the commencement of construction on any project licensed by the federal government, a cultural resources inventory may be required. Such inventories are authorized or required under the National Historic Preservation Act of 1966,[268] the Antiquities Act of 1906,[269] the Act of May 24, 1974,[270] and Executive Order No. 11593 issued May 13, 1971. If the study reveals that the project may cause irreparable loss or destruction of significant scientific, prehistorical, historical, or archeological data, a mining project could be delayed while additional studies are performed. Discovery of important historical or archeological data could preclude permitting or licensing of the project altogether.

3.4.3.9 Endangered Species Act of 1973

Section 7 of the Endangered Species Act of 1973[271] provides that all federal departments and agencies shall take such action as is necessary "to ensure that actions authorized, funded or carried out by them do not jeopardize the continued existence of . . . endangered species and threatened species or result in the destruction or modification of habitat of such species which is determined . . . to be critica1."[272] The Secretary of the Interior identifies endangered or threatened species through rulemaking.[273]

Every federal agency must assure that any action it takes will not jeopardize the continued existence of an endangered or threatened species or result in the destruction or modification of habitat of an endangered or threatened species if the habitat is determined by the Secretary to be critical. They must consult with the Fish and Wildlife Service of the Department of the Interior and the Department of Commerce in order to assure that actions do not cause such jeopardy or destruction.[274]

If in the course of preparation of an environmental impact analysis, it is determined that a threat to an endangered or threatened species might result from the contemplated project, special protective measures, including prohibition of the proposed action, if necessary, must be adopted. This becomes more and more of a possibility as additional species of plants, fish, and wildlife are classified as endangered or threatened. An exemption is possible under extraordinary circumstances upon action taken by a National Review Board and the Endangered Species Committee.[275]

Failure of a federal agency to take such action as is necessary to protect the endangered or threatened species does not mean that the proposed action can take place without restrictions. The law permits lawsuits to be brought by any citizen to enjoin any person who is alleged to be in violation of any provision of the Endangered Species Act or any regulation issued under the authority thereof or to compel the Secretary of the Interior or the secretary of Commerce to apply the protective regulations adopted pursuant to the Act.[276]

3.4.4 STATE AND LOCAL GOVERNMENT ENVIRONMENTAL REQUIREMENTS

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To date, this chapter has focused primarily on federal laws and regulations pertaining to environmental protection at mine sites because of their general applicability. There is, however, another body of law peculiar to each state that will govern the conduct of mining operations within its boundaries. In addition, counties or municipalities also can adopt laws and ordinances affecting mining.

At the earliest possible stage in mine development, it is most important to learn of the environmental protection requirements that may exist in the state wherein an operation is contemplated. Ideally, this inquiry should begin at the earliest exploration stage. Of particular importance are permits that must be acquired before exploration and/or production operations can commence. General information regarding such matters usually can be obtained through inquiry to the state agency having jurisdiction over mining operations. Often there will be some sort of office of industrial promotion in a state that can give helpful assistance in this regard. Detailed questions as to compatibility between state and federal laws, procedures for variances, and other such subjects often are best directed to an attorney.

The following paragraphs are a brief checklist of the types of laws and regulations that may be in effect in a particular state.

3.4.4.1 Environmental Impact Assessment Procedures

Some states have adopted legislation similar to NEPA that requires an environmental impact assessment prior to the taking of action by a state agency on an operation that may have a significant effect on the environment. California, for example, requires that an environmental impact report be prepared by state agencies prior to approval of an activity involving the issuance of a lease, permit, license, or certificate that may have a significant effect on the environment.[277] An EIS prepared in accordance with NEPA often will satisfy the state requirements.[278]

3.4.4.2 Water Pollution Control

Most states have enacted legislation prohibiting the pollution of waters in the state and requiring that discharges that might cause pollution not be made in the absence of a valid permit issued by a state agency or board.[279] Such laws often will govern discharges not subject to the Federal Water Pollution Control Act, such as discharges that do not enter surface waters and nonpoint source discharges. Separate legislative schemes may apply to discharges affecting groundwaters.[280] Separate state water discharge requirements may exist even if the state administers the program related to NPDES permits. Since the NPDES program, when administered by EPA, requires state certification of the quality of water being discharged, requirements in excess of those under the federal law may be imposed.

In those states where water rights are obtained upon application to the State Engineer, the State Engineer may impose environmental protection requirements with respect to the construction and operation of diversion and storage facilities.

3.4.4.3 Air Pollution Control

Bodies charged with the responsibility for air pollution control have been established in all states to comply with the Clean Air Act. The emission requirements imposed by the state can be more stringent than those imposed under the federal law and can apply to types of emissions not covered under federal law. State laws and regulations often dictate that no new source of air pollution be operated without a permit. The permit will set maximum discharge levels and generally will require that an emission monitoring system be set up.

3.4.4.4 Solid Waste Control

Disposal of solid wastes such as garbage or construction debris often require a permit.[281] Few states specifically deal with tailings disposal and waste dumps through solid waste control. However, more activity in that area can be anticipated, particularly in light of special studies that are to be conducted on the adverse effects of solid wastes from active and abandoned surface and underground mines on the environment under the Resource Conservation and Recovery Act of 1976.

3.4.4.5 Reclamation

Surface reclamation requirements have been adopted by many states, although some relate only to surface-mining operations.[282] State coal mining regulations must conform to the Coal Surface Mining and Reclamation Act of 1977. Such regulations generally require that a reclamation plan be submitted and approved prior to the commencement of mining operations, that reclamation meet certain general standards, and that a bond be provided to insure that the reclamation is performed when mining is completed. The conduct of all phases of mine operations can be influenced on a day-to-day basis by such surface-reclamation requirements.

3.4.4.6 State Lease Requirements

Mining conducted on state lands generally will be subject to environmental protection requirements spelled out in the lease terms. Such leases usually require prior approval of a mining plan before the commencement of operations and compliance with state environmental protection laws and regulations.

3.4.4.7 Noise Abatement

Some states have adopted legislation for noise control. Under such laws, noise-emission standards may be adopted to limit the noise that can be emitted by particular products or categories of noise-emission sources. Because of noise emitted by crushing and milling operations, a permit or variance may be needed under such a law before operations can be commenced.

3.4.4.8 Construction Permit

Approval of construction permits can be conditioned upon compliance with environmental protection requirements. A permit for a large installation, such as a concentrator, might impose numerous requirements with respect to dust control, ventilation, water discharge, noise abatement, and aesthetics.

3.4.4.9 Zoning

It is quite likely that the zoning laws of a county or municipality, within the boundaries of which an operation is sited, will have a bearing thereon. Some states have enacted comprehensive land-use plans.[283] If the lands have been zoned for a use other than mining, it will be necessary to seek rezoning. Even if mining is permitted, it usually will need to be conducted in accordance with zoning regulations that provide for environmental protection.

REFERENCES
(Parts 3.4.3 and 3.4.4)

Comprehensive Environmental Response, Compensation, and Liability Act (Superfund)

1. 42 U.S.C. § 9601-75 (1982), as amended by Superfund Amendments and Reauthorization Act of 1986, Public Law 99-499. For additional information on the subject, see Jacus, J., and Root, T., 1989, "The Law of Mine Waste: A Primer—Mine Waste from Agricola to CERCLA and Beyond," *Rocky Mountain Mineral Law Institute*, Vol. 35; Baird, J., 1987, "Identification, Evaluation, and Limitation of Environmental Liabilities During Exploration Activities: An Ounce of Prevention," *Rocky Mountain Mineral Law Institute*, Vol. 33, Ch. 21; Graham, D., and Lopatto, J., 1986, "Hazardous and Solid Waste Laws and Regulations; Effects on the Mining of Coal and Other Minerals," *West Virginia Law Review*, Vol. 88, No. 3, p. 587; Keppler, P., and Marucheau, L., 1986, "Mining Wastes at the Crossroads: Application of RCRA and CERCLA," *Rocky Mountain Mineral Law Institute*, Vol. 32, Ch. 8; Alkire, M., 1984, "CERCLA Liability for Mining and Milling Operations," *Rocky Mountain Mineral Law Institute*, Vol. 30, Ch. 7; Rocky Mountain Mineral Law Foundation, 1989, *American Law of Mining*, 2d ed., Matthew Bender, New York, Vol. 5, Ch. 171 (hereafter referred to as *American Law of Mining)*.
2. CERCLA also covers under separate procedures the release or substantial threat of a release of a "pollutant or contaminant" which may present an imminent and substantial danger to the public health and welfare. 42 U.S.C. § 9604(a)(l)(B) (1982). For a discussion of this subject see Keppler and Marucheau, *supra*, Sec. 8.07.
3. 42 U.S.C.A. § 9601(14) (1989).
4. *Id.* § 9604(b).
5. *Id.* § 9605; 40 C.F.R. Part 300 (1989).
6. 42 U.S.C.A. § 9604(2) (1989).
7. 55 Fed. Reg. 8666 (Mar. 8, 1990). The published rule is effective as of Apr. 9, 1990, and replaces Title 40, Code of Federal Regulations, Part 300, in its entirety, with the exception of Appendices A, B, and C, and Section 300.440.
8. 40 C.F.R. 300, App. A (1989).
9. 42 U.S.C.A. § 9605(a)(8)(B) (1989); 40 C.F.R. Part 300, App. B (1989).
10. 42 U.S.C.A. § 9604 (c)(l) (1989).
11. *Id.* §§ 9608, 9611.
12. *Id.* §§ 9601(10), 9607(j).
13. *Id.* § 9607(a).
14. See Alkire, *supra*, at 7-41 through 7-46; *American Law of Mining*, § 171.05.
15. 42 U.S.C.A. § 9607(a) (1989).
16. *Id.* § 9601(16).
17. *Id.* §§ 9607(f)(l), 9607(j).
18. For a discussion of this subject, see Baird, *supra*.
19. 42 U.S.C.A. §§ 9601(35)(A), 9607(b)(3) (1989).
20. United States v. Mobil Oil Corp., 464 F.2d 1124, 1127 (5th Cir. 1972).
21. 42 U.S.C.A. § 9603(b)(3) (1989).
22. *Id.* § 9601(22).
23. 50 Fed. Reg. 13456, 13462 (Apr. 4, 1985) (preamble).
24. 42 U.S.C.A. § 9606(a) (1989).
25. *Id.* § 9607(a).
26. 40 C.F.R. Part 300.400 (1990).
27. 40 C.F.R. Part 300.5 (1990).
28. 33 U.S.C.A. § 1321(a)(6) (1989); 42 U.S.C.A. § 9601(2) (1989).

Federal Water Pollution Control Requirements

29. 42 U.S.C.A. § 11004 (1989).
30. *Id.* §§ 11021-11022.
31. *Id.* § 11023.
32. 54 Fed. Reg. 3388 (Jan. 23, 1989) (proposed rule designating all EPCRA "extremely hazardous substances" as CERCLA "hazardous substances").
33. 42 U.S.C.A. § 11022 (1989).
34. *Id.* § 11023.
35. 33 U.S.C. §§ 1251-1376 (1982). For a thorough discussion of the impact of this legislation on mining, see Keppler, P., 1975 "Mining and the Federal Water Pollution Control Act Amendments of 1972," *Rocky Mountain MineraI Law Institute*, Vol. 20, p. 501. References to the Act with the numerical designations originally adopted will be identified herein as "FWPCA § ____."
36. Pub.L. 95-217, 91 Stat. 1567 (1977).
37. FWPCA §§ 301(a), 502(12), 33 U.S.C.A. §§ 1311(a), 1362(12) (1989).
38. FWPCA § 502(14), 33 U.S.C.A § 1362(14) (1989).
39. FWPCA § 502(7), 33 U.S.C.A. § 1362(7) (1989); 40 C.F.R. § 122.2 (1989); *American Law of Mining*, § 169.02[2][a][D].
40. FWPCA §§ 307(b), (c), 33 U.S.C.A. §§ 1317(b), (c) (1982); 40 C.F.R. Part 403 (1989).
41. *See American Law of Mining*, Vol. 5, § 169.02[2][c][ii][A].
42. FWPCA § 306(a), 33 U.S.C.A. § 1316(a)(l) (1989).
43. FWPCA § 306(c), 33 U.S.C.A. § 1316(c) (1989).
44. For a complete discussion of the background to the ore mining and dressing effluent guidelines, see Walpole, J., 1978, "Ore Mining and Dressing Effluent Guidelines—To BPT or Not to BPT?" *Rocky Mountain Mineral Law Institute*, Vol. 23, p. 353.
45. 40 C.F.R. § 440.132(g) (1989).
46. *Id.* § 440.132(h).
47. *Id.* § 440.132(f).
48. *Id.* Part 436.
49. 40 Fed. Reg. 48,653 (1975).
50. For a thorough discussion of water pollution regulations pertaining to coal mines, see Begley, J.T., and Williams, J.P., 1976, "Coal Mine Water Pollution: An Acid Problem With Murkey Solutions," *Kentucky Law Journal*, Vol. 64 p. 507b.
51. 40 C.F.R. § 434.11(b) (1989).
52. *Id.* § 434.1l(k).
53. *Id.* § 434.1l(h).
54. 33 U.S.C.A. § 1288(a)(2) (1989).
55. *See* Nat. Res. Def. Council v. Train, 396 F.Supp. 1286 (D.D.C. 1975); 40 Fed. Reg. 55334-49 (1975).
56. FWPCA §§ 208(a), (b), 201(c), 33 U.S.C. §§ 1288 (a), (b), 1281(c) (1982).
57. FWPCA § 208(b)(2)(G), 33 U.S.C.A. § 1288(b)(2)(G) (1989).

58. 40 C.F.R. § 130.2(M) (1989).
59. FWPCA § 208(e), 33 U.S.C.A. § 1288(e) (1982).
60. Ipsen, H., 1978, "Water Quality Management Plans and Their Impact on Mining Operations," *Rocky Mountain Mineral Law Institute*, Vol. 23, p. 551.
61. 33 U.S.C.A. § 1313 (1989).
62. Begley, J.T., and Williams, J.P., 1976, "Coal Mine Water Pollution: An Acid Problem With Murky Solutions," *Kentucky Law Journal*, Vol. 64, pp. 507, 518.
63. 33 U.S.C.A. § 1313(d) (1989).
64. *See* Keppler, *supra*, note 1, at p. 511; Ipsen, *supra*, note 26, at pp. 554-66.
65. FWPCA § 402, 33 U.S.C.A. § 1342 (1989).
66. FWPCA §§ 301(b)(l)(A), (2)(A), 33 U.S.C. §§ 131l(b)(l)(A), (2)(A) (1982).
66a. 55 Fed. Reg. 47990 (November 16, 1990).
66b. *Id.* at 48065 (to be codified at 40 C.F.R. §122.26(b)(14)(iii)).
66c. *Id.* at 48063 (to be codified at 40 C.F.R. § 122.26(a)(2)).
66d. *Id.* at 48003. *See also*, 56 Fed. Reg. 12098 (March 21, 1991) (extension of application deadline for Part I of group permit), and 56 Fed. Reg. 12101 (March 21, 1991) (proposed extension of application deadline for individual permit).
67. FWPCA § 402(b), 33 U.S.C.A. § 1342(b) (1989).
68. FWPCA § 402(d), 33 U.S.C.A. § 1342(d) (1989).
69. *Id.* § 1342(e).
70. *See* generally, 40 C.F.R. Part 123 (1989).
71. 40 C.F.R. § 122.21(c)(l) (1989).
72. *Id.* § 124.53.
73. *Id.* § 124.57.
74. *Id.* 124.75.
75. 33 U.S.C.A. § 1342(d)(2) (1982).
76. *Id.* § 1342(d)(4).
77. 40 C.F.R. 124.61 (1989). Regulations for the preparation of environmental impact statements in connection with new source NPDES permits are provided in 40 C.F.R. § 6.805 (1989).
78. *Id.* § 124.114(b).
79. *Id.* § 122.44.
80. *Id.* § 122.46.
81. 33 U.S.C.A. § 1319(a)(3) (1989).
82. *Id.* § 1319(d).
83. *Id.* § 1319(c)(1)-(2).
84. *Id.* § 1319(c)(l).
85. *Id.* § 1319(g).
86. 33 U.S.C.A. § 1321 (1989).
87. FWPCA § 31l(b)(2)(A), 33 U.S.C.A. § 1321(b)(2)(A) (1989).
88. FWPCA § 311(b)(2)(B)(i), 33 U.S.C.A. § 1321(b)(2)(B)(i) (1989).
89. FWPCA § 31l(b)(2)(B)(iii), 33 U.S.C.A. § 1321(b)(2)(B)(iii) (1989).
90. FWPCA § 31l(b)(3), 33 U.S.C.A. 1321(b)(3) (1982).
91. 33 U.S.C.A. § 1321(b)(6)(A) (1982); 40 C.F.R. § 117.22 (1989).
92. FWPCA § 311(f), 33 U.S.C.A. § 1321(f) (1989).
93. FWPCA § 304(e), 33 U.S.C.A. § 1314(e) (1982).
94. 33 U.S.C.A. § 1321(b)(3) (1989).
95. 40 C.F.R. § 110.3 (1989).
96. *Id.* § 110.10.
97. *Id.* Part 114.
98. 40 C.F.R. § 112.1 (1989). The regulations were adopted by authority of FWPCA § 311, 33 U.S.C. § 1321 (1982).
99. 40 C.F.R. § 112.3(e) (1989).
100. *Id.* § 112.6.
101. 33 U.S.C.A. § 1317(a) (1982).
102. 40 C.F.R. § 129.1(b) (1989).
103. *Id.* §§ 129.100-105.
104. FWPCA § 304(e), 33 U.S.C.A. § 1314(e) (1989).

Dredge and Fill Requirements

105. 33 U.S.C.A. § 1344(a) (1989). For a more thorough discussion of this subject see Fereday, J.C., 1989, "Wading Through the Dredge and Fill Permit Process: A Practitioner's Guide to the Clean Water Act," *Rocky Mountain Mineral Law Institute*, Vol. 34, p. 4-l.
106. FWPCA § 404(b), 33 U.S.C.A. § 1344(b) (1982).
107. 33 C.F.R. § 328.3(a) (1989).
108. *Id.* § 328.3(b).
109. *Id.* § 323.2(c).
110. *Id.* § 323.3(d).
111. *Id.* § 323.2(e).
112. *Id.* § 323.2(f).
113. 33 U.S.C.A. § 1344(e) (1982).
114. 33 C.F.R. § 330.5(a) (1989).
115. *Id.* § 330.5(a)(21).
116. *Id.* § 330.5(26).
117. *Id.* §§ 330.5(26), 330.7.
118. *Id.* § 330.2(b).
119. *Id.* §§ 330.6.
120. *Id.* §§ 330.5(b)(11), 330.9.
121. *Id.* § 330.5(b).
122. *Id.* § 325.2(e)(2).
123. FWPCA § 404(e), 33 U.S.C.A. § 1344(e) (1989).
124. 33 C.F.R. § 330.7(d) (1989).
125. 33 C.F.R. § 330.8 (1989).
126. FWPCA § 404(f)(l)(E), 33 U.S.C.A. § 1344(f)(l)(E) (1989).
127. FWPCA §§ 404(f)(l)(B), (D), 33 U.S.C.A. § 1344(f)(l)(b) (1989).
128. 33 C.F.R. § 325.1(c) (1989).
129. *Id.* §§ 325.2(a), 325.3.
130. *Id.* Part 230; § 325.2(a)(4); § 325 App. B.
131. *Id.* 325.2(b)(l); see FWPCA § 401, 33 U.S.C.A. § 1341 (1982).
132. FWPCA § 404(b), 33 U.S.C.A. § 1344(b) (1982); 33 C.F.R. 325.2(a)(6) (1989); 40 C.F.R. Part 230 (1989).
133. Memorandum of Agreement between the Environmental Protection Agency and the Department of the Army Concerning the Determination of Mitigation Under the Clean Water Act Section 404(b)(1) Guidelines, February 6, 1990.
134. FWPCA § 404(c), 33 U.S.C.A. § 1344(c) (1982).
135. 33 C.F.R. § 325.2(a)(6) (1989).
136. *Id.* § 325.6(a).
137. *Id.* §§ 325.6(a), (b)(6).
138. *Id.* § 325.7(a).
139. FWPCA §§ 404(g)-(i), 33 U.S.C.A. §§ 1344(g)-(i) (1982), 40 C.F.R. Part 233 (1989), 33 C.F.R. § 323.5 (1989).
140. FWPCA §§ 404(j), (k), (l), 33 U.S.C.A. §§ 1344(i), (k), (1) (1989).
141. FWPCA § 404(j), 33 U.S.C.A. § 1344(j) (1989).
142. FWPCA § 208(b)(4)(B), 33 U.S.C.A. § 1288(b)(4)(B) (1989).
143. FWPCA § 404(f)(l)(F), 33 U.S.C.A. § 1344(f)(l)(F) (1982).

Safe Drinking Water Act

144. 42 U.S.C.A. §§ 300f-j (1982).
145. *Id.* § 300g.
146. *Id.* § 300f(4); 40 C.F.R. § 143.2(m) (1989).
147. 42 U.S.C.A. § 300h (1982). For a more complete discussion of this subject, refer to Sanderson, J., 1978, "The Effects of the Federal Safe Drinking Water Act on Oil, Gas and Mining Operations: Bittersweet or Unpalatable?" *Rocky Mountain Mineral Law Institute*, Vol. 23, p. 941; Renwick, E., 1978, "The Effects of the Federal Safe Drinking Water Act on Oil, Gas and Mining Operations: An Oil and Gas Lawyer's view," *Rocky Mountain Mineral Law Institute*, Vol. 23, p. 975.
148. 40 C.F.R. § 144.3 (1989).
149. *Id.* § 146.5; *see American Law of Mining*, Vol. 5, § 169.03[2][a].
150. 40 C.F.R. § 146.5(c) (1989).
151. *Id.* § 144.11.
152. *Id.* §§ 144.5l-52.
153. *Id.* § 144.33.
154. *Id.* § 146.5(a).
155. *Id.* § 144.36.
156. *Id.* § 146.5(e).
157. *Id.* § 144.24.
158. *Id.* § 144.26.
159. 42 U.S.C.A. § 300h(b)(1) (1982).

Air Pollution Control

160. 42 U.S.C.A. § 7407 (1989).
161. *Id.* § 7408(a)(l)(A), (B).
162. *Id.* § 7408(a)(2).

163. *Id.* § 7408(b).
164. *Id.* § 7409.
165. 40 C.F.R. §§ 50.4-50.12 (1989).
166. *American Law of Mining,* Vol. 5, § 168.02.
167. 52 Fed. Reg. 24,634-750.
168. 42 U.S.C.A. § 7410 (1982).
169. *Id.* §§ 741l(a)(2), (3).
170. *Id.* § 741l(a)(4).
171. *Id.* § 7411(b).
172. *Id.* § 7411(a)(l).
173. *Id.* § 7411(h)(l).
174. 42 U.S.C.A. § 741l(c) (1982).
175. 40 C.F.R. § 60.7(a)(2) (1989).
176. *Id.* § 60.7(a)(3).
177. *Id.* §§ 60.7(b), (c).
178. *Id.* § 60.8.
179. *Id.* §§ 60.1l(a), (d).
180. 42 U.S.C.A. §§ 7410(a)(2)(I), 7503 (1989).
181. *See, e.g.,* Clean Air Act §§ 182(b)(5), 182(c)(10).
182. 42 U.S.C.A. § 7412(b) (1982).
183. *Id.* § 7412(a)(l).
184. 42 U.S.C.A. § 7408(a)(l)(B).
185. 42 U.S.C.A. § 7412(a)(2), (c)(l).
186. *Id.* § 7412(b)(l)(B).
187. *Id.* § 7412(c)(l)(A).
188. *Id.* § 7412(c)(l)(B).
189. 40 C.F.R. § 61.07 (1989).
190. *Id.* § 61.08.
191. 42 U.S.C.A. § 7412(d)(2) (1989).
192. 40 C.F.R. Part 61, Subparts B, C, E, M, W (1989).
193. 42 U.S.C.A. § 7412(b)(1).
194. 42 U.S.C.A. § 7412(d).
195. 42 U.S.C.A. § 7413.
196. *Id.* § 7414.
197. *Id.* §§ 7470-79; *see* 40 C.F.R. § 52.21 (1988).
198. 53 Fed. Reg. 40,656 (Oct. 17, 1988).
199. 42 U.S.C.A. § 7473 (1989).
200. *Id.* § 7472.
201. *Id.* § 7474(a)(2)(B).
202. *Id.* § 7475(a)(l).
203. *Id.* §§ 7475(a)(3), (4).
204. *Id.* § 7479(3), 40 C.F.R. § 52.21(b)(12) (1988).
205. 42 U.S.C.A. § 7479(l) (1989); 40 C.F.R. 52.21(b)(l) (1988).
206. *American Law of Mining,* Vol. 5, § 168.05[3][d][i].
207. 40 C.F.R. § 52.21(b)(2)(i) (1988).
208. *Id.* § 52.21(b)(4).
209. *Id.* §§ 52.21(l)(m)(o).
210. *Id.* § 52.21(m).
211. *American Law of Mining*, Vol. 5, § 168.01.
212. 40 C.F.R. § 52.21(b)(20) (1989).
213. 42 U.S.C.A. §§ 7475(a), (d)(2)(C) (1989); 40 C.F.R. § 52.2l(b)(18) (1988); (k), *American Law of Mining,* Vol. 5, § 168.05[3][d][iii][c].
214. 42 U.S.C.A. § 7475(d)(2)(D) (1989).
215. *Id.* § 7475(a)(3); *see American Law of Mining,* Vol. 5, § 168.05[3][?].

Federal Surface Management Regulations Concerning Unpatented Mining Claims

216. *See* 43 C.F.R. § 3809.3-1(a) (1988).
217. A good overview of the various state regulations can be found in *American Law of Mining,* Vol. 5, § 173.03. For a discussion of zoning and other land use controls as they relate to mining, see § 174.
218. 43 C.F.R. §§ 3802.0-5(e), 3809.0-5(d) (1988). The regulations concerning operations on Forest Service lands do not specifically include tunnel claims. *See* 36 C.F.R. § 288.3(d) (1989).
219. 43 C.F.R. §§ 3809.1-2, 3809.2-2 (1988).
220. *Id.* § 3809.0-5(c). As an example of the legal interpretation of "casual use," a federal court has held that an operator who lived on his claim, cut trees, violated health codes, littered, discharged wastewater, raised animals and a garden, and excluded the public by fencing off the access road and posting "no trespassing" signs was clearly not making casual use of the land as contemplated in the regulations. *See* Bales v. Ruch, 522 F. Supp. 150, 155-57 (E.D. Cal. 1981).
221. 43 C.F.R. § 3809.1-3(a) and (b) (1988).
222. *Id.* § 3809.1-3(c).
223. *Id.* §§ 3809.1-9, 3809.1-3(d)(5).
224. *See* Kimball, D., III, 1983, "Impact of BLM Surface Management Regulations on Exploration and Mining Operations," 28 *Rocky Mountain Mineral Law Institute*, pp. 509, 526. A plan of operations may be required if the activities will take place in certain specified environmentally sensitive areas. 43 C.F.R.
225. 43 C.F.R. § 3809.1-4(a) (1988). Unless the operations constitute "casual use," a plan of operations is also required if the claims are located in certain specially protected environmental areas such as lands within or proposed for inclusion within the wild and scenic rivers system, certain lands within the California Desert Conservation Area, wilderness areas, areas closed to off-road vehicle use, and designated areas of critical environmental concern. *See id.* § 3809.1-4(b).
226. *Id.* § 3809.1-5.
227. *Id.* § 3809.1-6.
228. *Id.* § 3809.1-7(b). Procedures also exist for the BLM to request at any time that the operator modify the approved plan of operations. *See id.* § 3809.1-7.
229. *Id.* § 3809.5(a). Examples of information that may be specified by the operator as confidential (if provided at all) would include such items as the outline, location and extent of the mineral deposit, and the location of exploration pits and drill holes. *See* 36 C.F.R. § 228.6 (1989) (discussing confidentiality with respect to claims on Forest Service land).
230. 43 C.F.R. § 3809.1-6(d) (1988); Kimball, *supra,* at 530.
231. 43 C.F.R. § 3809.2-l (1988).
232. *Id.* § 3809.1-9.
233. "Unnecessary or undue degradation" is specifically defined in the regulations. See *id.* § 3809.0-5(k).
234. *See id.* § 3809.2-2.
235. *See id.* §§ 3809.1-1, 3809.1-4(e), 3809.3-2, 3809.3-6.
236. See *id.* § 3809.4. For further information regarding BLM surface management regulations governing nonwilderness areas, see 43 C.F.R. subpart 3809 (1988), *American Law of Mining,* Vol. 5, § 173.05[4][a][ii], and Kimball, *supra,* at 520-37.
237. See 43 C.F.R. § 3802.0-5(c) (1988) and *American Law of Mining,* Vol. 1, ch. 15 (discussing wilderness areas and the wilderness system).
238. See 43 C.F.R. § 3802.0-2(a) (1988).
239. *Id.* §§ 3802.l-l, 3802.1-2.
240. *Id.* § 3802.1-3. "Undue and unnecessary degradation" is defined in the regulations. *See id.* § 3802.0-5(l).
241. *See id.* § 3802.1-5. Ongoing operations that existed on Oct. 21, 1976, are subject to a lower level of scrutiny. Such operations do not generally necessitate a plan of operations and need only prevent undue or unnecessary degradation of the environment. Post-1976 operations generally require a plan of operations, must prevent unnecessary degradation, and must also prevent any impairment of the area's suitability for wilderness. *Id.* § 3802.0-6.
242. *See id.* §§ 3802.1-4 to 3802.6.
243. *Compare id. §* 3809.0-5(j) *with id.* § 3802.0-5(a).
244. *Id,* §§ 3802.4-7, 3802.4-8. For further information regarding BLM surface management regulations governing wilderness study areas, see 43 C.F.R. subpart 3802 (1988), *American Law of Mining,* Vol. 5, § 173.05[4][a][iii], and Kimball, *supra,* at 538-53.
245. 36 C.F.R. § 228.1 (1989).
246. *Id.* §§ 228.4(a)(l) and (2). As an example of how the courts have interpreted "significant disturbance of surface resources," a federal appeals court has ruled that the presence of a cabin, an uncompleted mill building, a tool shed, a shower, and an outhouse on an unpatented mill-site claim located within a National Forest constituted significant surface disturbance within the meaning of the Forest Service regulations. *See* U.S. v. Brunskill, 792 F.2d 938, 939, 941 (9th Cir. 1986).
247. 36 C.F.R. § 228.4(a) (1989).
248. *Id.* § 228.4(a) and (a)(2).

249. *Id*. § 228.4(c).
250. *Id*. § 228.5(a).
251. *Id*. § 228.8.
252. *See id.* §§ 228.4(f),228.5(b),228.5(c),228.13,228.6,228.7,228.14.
253. *Id*. § 228.15(b).
254. For further information regarding Forest Service surface management regulations, see 36 C.F.R. part 228, subpart A (1989) and *American Law of Mining,* Vol. 5, § 173.05[4][b].

Federal Lease Operating Regulations (Other Than Coal)

255. 43 C.F.R. § 23.2 (1988).
256. 30 U.S.C. § 18l-209 (1982).
257. 30 U.S.C. §§ 351-59 (1982).
258. 43 C.F.R. § 23.5 (1988).
259. *Id*.
260. *Id*.
261. *Id*. § 23.5(b).
262. *Id*.
263. *Id*. § 23.8.
264. *Id*. § 23.10(d).
265. 43 C.F.R. § 23.9 (1976).
266. *Id*. § 23.10.
267. *Id*. § 23.1l(d).

Preservation of Historical and Archeological Data

268. 16 U.S.C. §§ 470-470h (1982). For thorough discussions of this topic, refer to Glazier, G.E., 1983, "Cultural Resource Preservation: A Consideration Before Mineral Development," *Rocky Mountain Mineral Law Institute,* Vol. 28, p. 635.
269. 16 U.S.C. § 431-33 (1982).
270. 16 U.S.C.A. § 469a-1 to 469b (1982).
271. 16 U.S.C.A. § 153l-43 (1988). For thorough discussions, see Lundberg, C.K., 1978, "Birds, Bunnies and the Furbish Lousewart—Wildlife and Mining of the Federal Lender," *Rocky Mountain Mineral Law Institute,* Vol. 24, p. 93.
272. 16 U.S.C.A. § 1536 (1985).
273. The listing is in 50 C.R.R. Part 17 (1988).
274. *Id.* (1989).
275. 16 U.S.C.A. § 1536 (1985).
276. *Id.* § 1540(g)(1).
277. Deerings California Code Ann., Pub. R. §§ 21000-176 (1976).
278. *Id.* § 21083.5.
279. *E.g., Utah Water PoIlution Control Act, 73-14-1, -13 Utah Code Ann. 1953; Colo. Rev. Stat. 1973, §§ 25-8-101, -104.*
280. Gilbert, A.J., 1986, "Groundwater Contamination: Pollutants, Priorities, and the Pursuit of Sensible Regulation," *Rocky Mountain Mineral Law Institute,* Vol. 32, p. 2-l.
281. For a thorough discussion by individual state of laws pertaining to dumps and tailings, see Reeves, G.E., and Alfers, S.D., "Dumps and Tailings," *Rocky Mountain Mineral Law Institute,* Vol. 23, p. 419 (1978).
282. Dietrich, C.C., "Mined Land Reclamation in the Western United States," *Rocky Mountain Mineral Law Institute,* Vol. 16, p. 143 (1971).
283. Kidd, D.T., "The Effect of Zoning and Land Use Control on Mineral Operations," *Rocky Mountain Mineral Law Institute,* Vol. 19, p. 277 (1974); Dietrich, *supra,* note 355, at pp. 173-76; Raymond, G.L., "Zoning and Land Use Planning as They Effect Coal Mining," *Rocky Mountain Mineral Law Institute,* Vol. 23, p. 173 (1978).

Part II
Stages of Mining

Section 4 Mineral Prospecting and Exploration

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Chapter 4.0
INTRODUCTION

WILLARD C. LACY

A mining operation begins with prospecting and exploration-stages with long periods of investment and high risk of failure. However, success in exploration ultimately determines survival of the mining industry.

Prospecting and exploration may discover evidence of a mineral occurrence and outline its size and character, but ore deposits that support mining are "made" through the collective efforts of project geologists, geophysicists, geochemists, metallurgists, engineers, chemists, lawyers, and even politicians (Joralemon, 1975). Some deposits may go through multiple cycles of rejection and recommendation, discovery and development, decline and abandonment, rediscovery and development, etc., as economic, technological, or political conditions change or geologic understanding is improved. Peters (1987, p. 233) graphically illustrates the normal life cycle of a mine and the position of exploration in the cycle (Fig. 4.0.1).

During the past fifty years, mineral exploration has evolved from comparing evidence of mineralization of prospects with that of known deposits to a more quantitative science employing many facets of geochemistry, geophysics, multispectral sensing from space, computer data storage, analysis, modeling, and most importantly, a better understanding of ore genesis. This modern approach to exploration began in the late 1940s and continues to develop. Exploration efforts peaked in the 1960s with emphasis upon the search for porphyry copper deposits, Mississippi Valley lead-zinc deposits, and volcanogenic massive sulfide copper-lead-zinc deposits. It peaked again in the middle and late 1980s with emphasis upon micron-size disseminated gold and epithermal and stratiform precious metal deposits.

Revival and augmentation of the concepts of continental drift and its effects on continental plates, sea floor spreading, subduction zones and related fault systems, observation of actual sea floor metal sulfide deposition, and precious metal deposits forming in hot spring environments have suggested potential target areas and revealed deposit characteristics that enabled discovery of new varieties of deposits in newly recognized environments.

The object of mineral commodity search shifts with development of geological concepts and search techniques, changing markets, price fluctuations, and developments in mining and

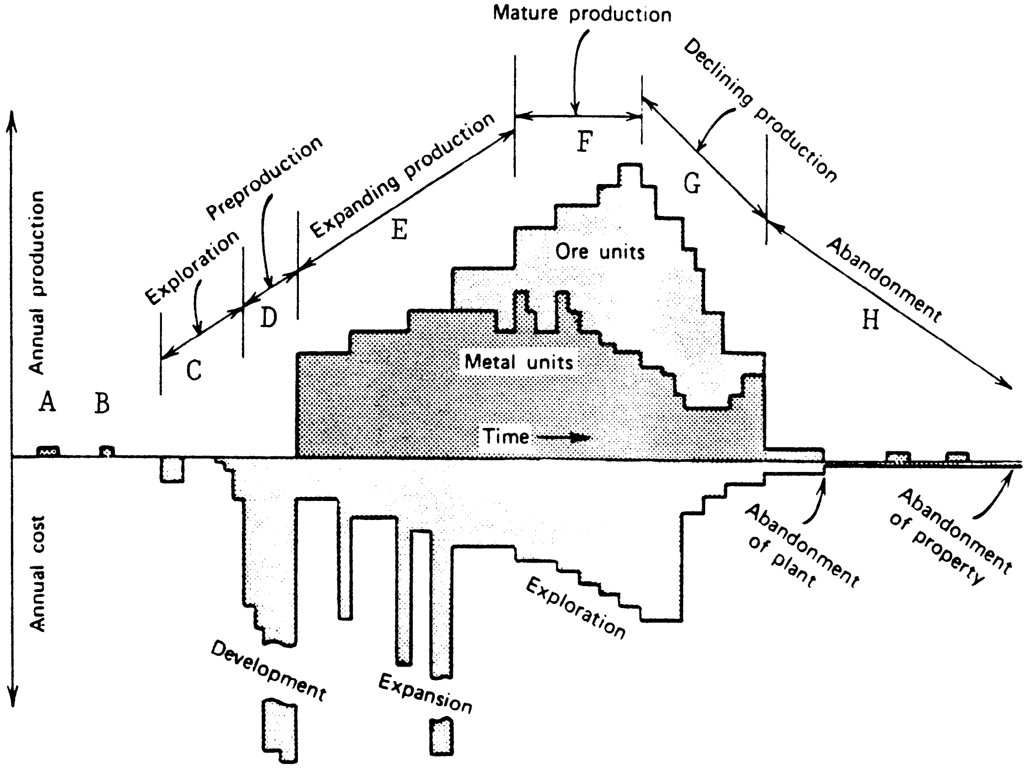

Fig. 4.0.1. The life cycle of a mine. A. Discovery of the district. B. Repeated examination by geologists and engineers. C. Recognition of potential of a major ore body. D. The preproduction interval: preliminary estimates, financing, testing, development, staffing. E. Expanding (youthful) production: payment of dividends, repayment of debt, vertical and horizontal growth. F. Mature production: innovation, verification of limits of ore, local exploration, cost reduction. G. Declining (old-age) production: sale or lease of assets, "outside" exploration, mining of pillars, cost reduction. H. Mine abandonment: salvage of machinery, custom milling, treatment of dumps, lessee operations, land restoration. (Peters, 1987. By permission from John Wiley & Sons, New York.)

metallurgical technology. This is well demonstrated by the commodity gold (Fig. 4.0.2). Exploration for gold in the United States lacked enthusiasm until discovery of the Carlin micron-size disseminated gold deposit in 1962, combined with freeing of gold price control in 1971, and development of heap-leach and carbon extraction techniques for gold recovery. These developments sparked the "gold rush" of the 1980s. In contrast, copper exploration suffered from an oversupplied market, depressed prices, and restrictive environmental legislation. Exploration and discoveries declined.

While changing economic factors, technological developments, and new market demands have created new target areas and revived old ones, restrictive legislation continues to reduce areas available for search and development and to increase governmental controls.

4.0.1 DEFINITIONS

In this section, the term "ore" is used in its broadest sense to include economic metallic mineral deposits, industrial mineral deposits, and energy mineral deposits.

In North America, the terms "prospecting" and "exploration" are often considered equivalent, but there is the implication that prospecting is less technical and usually carried out by an individual. Company departmental groups engaged in the search for mineral deposits are referred to as "exploration departments," which may on occasion employ or finance practical "country-wise" individuals to search given areas, "prospecting" for surface evidence of mineralization.

In this presentation, *prospecting* is considered as a part of exploration–the direct search for surface indications of ore mineralization. *Exploration* includes all activities involved in the discovery and evaluation of a mineral deposit, establishing the size, grade, initial flowsheet, and annual output of the new extractive operation (Bailly and Still, 1973, p.5-2). At an operating mine, the search for additional ore bodies, discussed in Section 5, may also be referred to as exploration.

"Discovery" of valuable mineral is the foundation of the US Mining Law of 1872. It is a prerequisite to mining claim validity. Initially, "discovery" was legally defined as the finding of sufficient quality and quantity of mineralization that a person of ordinary prudence, with a reasonable hope of success, would be justified in further expenditure of labor and means-the "prudent man" test. However, at present the Department of the Interior and the courts have modified their definition of discovery to a more stringent test, a "marketability" test. In order to establish discovery, the locator must demonstrate the existence of a mineral deposit that can currently be mined, treated, and marketed at a profit-an "economic discovery." The US Revenue Act of Feb. 9, 1919, uses discovery as practically synonymous with "development." This leads to a "Catch 22" situation where tenure may be required before capital for development can be acquired, yet development is required for tenure.

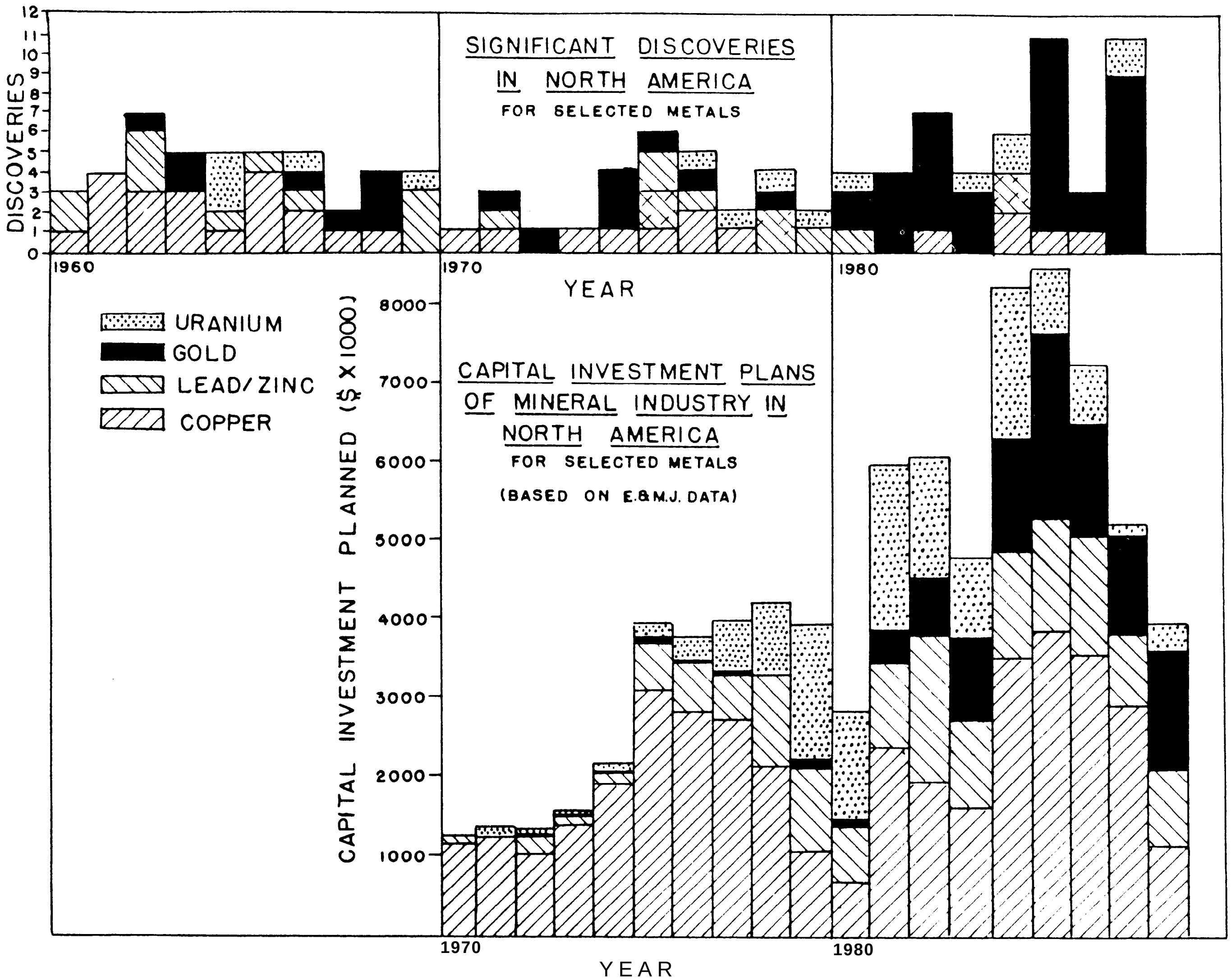

Fig. 4.0.2. Significant ore deposit discoveries in North America for selected metals, and capital investment plans projected by North American industry (source of data: *Engineering and Mining Journal, Mining Engineering,* Cook, 1986).

Lack of agreement among persons within the mineral industry as to the definition of "discovery" results from different objectives. What may be considered as a successful discovery to one company may not be acceptable to another.

MacKenzie and Bilodeau (1984) state, "A discovery is considered to be made when the deposit which ultimately forms the basis for development is first encountered by drilling (or other means). . . . An economic discovery is one which realizes a total revenue of at least $20 million and a rate of return of at least 10%."

Miller (1976, p.840) is much more restrictive, stating, "Within the modern corporate system an ore discovery is a mineral deposit which can be brought into production within a five-year period. A technical success is the discovery of a mineral deposit which is subeconomic into the indefinite future . . . exploration geologists can be judged successful only on their rate of ore discovery, not on the basis of technical successes."

Cook (1986, p. 87) recognizes "significant discoveries" as deposits of a size and grade that would yield more than $500 million in gross revenue, using average metal prices for the past five years. "World class discoveries" consist of those deposits that are continuously operated and consistently return a profit even during periods of low metal prices, that are large enough to significantly affect a medium-sized corporation's profits, and that rank in the lower one-third cost per unit metal produced for all similar deposits.

In this discussion, *discovery* refers to the first encounter with mineralization that can be developed into a profitable mine under normal economic conditions.

An exploration program may find a "prospect"—interesting surface indications of mineralization that bear investigation. This may develop into a "geological," "geochemical," or "geophysical" discovery when sufficient data are gained to locate the probable presence of an economic mineral deposit. An "economic discovery" can be achieved when (1) capital for development can be raised within a reasonable period of time, (2) tenure and ownership will be respected (the mining claims will qualify for patent), (3) a reasonable profit margin can be projected, (4) technology for mining and treatment exists or can be developed within a reasonable period of time, and (5) there is social and political acceptance of the mining activity. An economic discovery results from the collective efforts that go into the making of a mine.

4.0.2 ANALYSIS OF SIGNIFICANT MINERAL DISCOVERIES: 1950 to 1990

4.0.2.1 Performance by Commodity

During the 1950s and 1960s, base metal search and discovery dominated mineral exploration. Successes stemmed from refined geological and geophysical models, in particular—relating to fluid flow through geologic formations, recognition of alteration halo patterns, positioning of deposits relative to geotectonic plates, and refinement of search techniques.

Models of porphyry copper deposits were developed by exploration and research geologists of several major mining companies and consultants, and the distribution of deposits was recognized as being associated with subduction at plate margins and associated with crustal breaks. This encouraged extension of search and resulted in many new discoveries. Refinement of induced polarization geophysical techniques enabled recognition and evaluation of disseminated sulfides under alluvial and/or post-mineralization cover.

Localization of volcanogenic massive sulfide copper-lead-zinc deposits along interplate environments led to discoveries along the cratonic margin in southeastern Canada and northeastern United States from Michigan to Maine. Exploration for these deposits was assisted by refinements in aeroelectromagnetic methods.

Mississippi Valley type lead-zinc deposits were related to fluid-flow models, to basin margins, and to reef distribution in limestone units. This led to a rash of discoveries and recognition of the Viburnum Trend in southeastern Missouri.

The roll-front model of sandstone uranium deposits resulted in many discoveries in the 1960s and 1970s on the Colorado Plateau and adjacent areas, and the energy crisis of the 1970s stimulated search for uranium in Canada that was rewarded by large high-grade discoveries along the Proterozoic unconformity of the Canadian Shield.

Search for precious metals was stimulated by the French "run" on gold reserves of the US Treasury, discovery of the Carlin micron-size disseminated gold deposit in 1962, freeing of gold price controls in 1971, and development of low-cost extract techniques. Genetic models relating deposits to hot spring and epithermal environments, detachment fault zones and peraluminous intrusives of the western United States, and to stratiform Proterozoic environments in eastern Canada have led to many discoveries that dominated success statistics during the 1980s (Fig. 4.0.2).

Lag time between discovery and development of deposits is reflected in capital investment plans, which may be seen in Fig. 4.0.2. Between the time of initial detection of base metal deposits and planned major capital expenditures for development is an average period of about ten years. Uranium and gold deposits, on an average, require shorter preproduction periods and lower capital investment. Albers (1977, p.73) suggests an average of seven years' preproduction period from geological, geophysical, or geochemical discovery for all mine types, ten years for copper deposits, five years for uranium deposits, and three years for gold deposits. These figures lack sufficient numbers to give statistical validity, but they are indicative.

Table 4.0.1 Principal Techniques Responsible for Ore Body Detection in the United States and Canada, 1940-1976

	USA[1]	Canada[2]
Discoveries	63	122
Geological inference	73%	32%
Conventional prospecting	4%	37%
Geophysical	3%	25%
Geochemical	10%	4%
Combined	10%	2%

Source: [1]Albers, 1977, [2]Deny, 1969, Deny and Booth, 1978.

Table 4.0.2 Estimated Cost per Discovery in 1980 US Dollars (Millions)

	USA[1,3]*	Canada [1,2,3]*	Non-Communist World[3]*	Australia[4]**
1950-1960	55	60	60	
1960-1970	60	100	75	Sn/W: 14
1970-1980	100	150	140	Ni: 10
1980-1990	70	100	100	Total: 19

*Using criteria of $500 million gross metal value minimum.
**Using criteria of $20 million gross metal value minimum.
Source: [1]American Bureau of Metal Statistics; [2]Deny and Booth, 1978: [3]Cook, 1986; [4]MacKenzie and Bilodeau, 1984.

4.0.2.2 Performance by Search Technique

A comparison of ore deposit detection according to methods responsible for their detection is shown on Table 4.0.1

Direct observation of surface evidence or ore mineralization—conventional prospecting—is declining in importance except for some precious metal deposits that have very subtle surface indications, search for deposits in remote or underdeveloped portions of the world, and search for deposits of those elements or minerals that have had limited past search history. The prospector has become much more sophisticated in his recognition of evidence of mineralization and has become aware of current theory and search techniques.

It has become increasingly difficult to assign any discovery to a single exploration technique. The successful programs utilize search by all appropriate techniques.

4.0.2.3 Discovery Costs

Various estimates have been made as to the average costs for making discovery of an ore deposit (Cook, 1986; Albers, 1977; Derry and Booth, 1978; and MacKenzie and Bilodeau, 1984). Such approximations lack general agreement as to what constitutes a discovery and involve uncertainties as to expenditures. They do, however, present an order of magnitude and permit comparison between areas of search and target size. Figures need to be composited over ten-year periods to accommodate preproduction periods. Table 4.0.2 presents estimates of average discovery costs for finds with anticipated gross metal value in excess of $500 million in the United States, Canada, and the entire Non-Communist World, and in excess of $20 million in Australia.

A comparison of these statistics on a basis of estimated minimum acceptable gross revenue vs. cost of discovery (Fig.

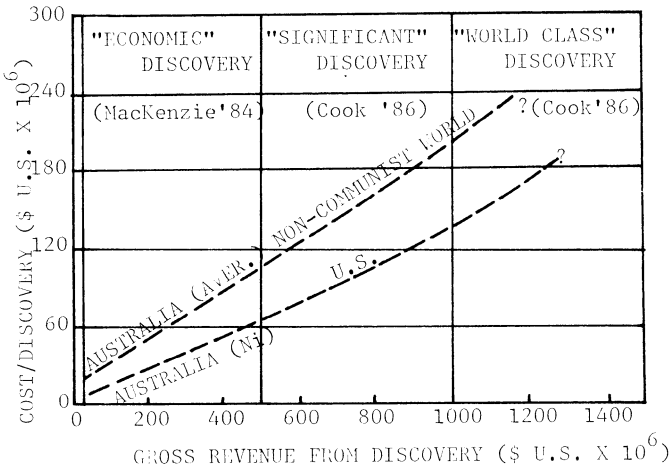

Fig. 4.0.3. Average discovery costs relative to minimum acceptable gross revenue of ore deposit discoveries (Cook, 1986; MacKenzie and Bilodeau, 1984.)

Table 4.0.3 Distribution of Exploration Costs in United States and Canada, 1980-1983

	USA	Canada
Geological inference	15.6%	20.3%
Geochemistry and conventional prospecting	18.5	14.3
Geophysics	4.6	12.2
Drilling	30.6	33.6
Cost per staff (excluding summer students) in 1980 U.S. dollars	130,000	140,000

Source: Barber and Muessig, 1984, 1985.

4.0.3) suggests that discoveries cost between 14 and 20% of minimum acceptable gross revenue from the deposit. Some company exploration success costs consistently beat these average costs, while others seldom realize success despite substantial expenditures. This appears to be a function of management of exploration programs (Chapter 4.6).

Distribution of costs in exploration programs in the United States and Canada for the period 1980 to 1983 is shown in Table 4.0.3.

4.0.3 FORECAST

Madigan (1984), paraphrasing Wallace Platt, wrote:

> "Where mineral is first found, in the final analysis, is in the minds of men. . . . When man no longer believes more minerals are left to be found, no more minerals will be found, but so long as mineral seekers remain with a mental vision of a new field to cherish along with freedom and incentive to explore, so long will mineral fields continue to be discovered."

Continued research on ore genesis will reveal new environments of mineral and metal concentrations, and new search and production technologies or escalation of prices may allow extraction of previously uneconomic concentrations.

The search for specific mineral and metal commodities will change as markets are modified and different technologies develop. For example, miniaturization and satellite communications have allowed less metal to do more, but, at the same time decrease in metal demand in some parts of the world may be compensated by increased Third World demand. New public transportation systems, local, national, and international, surface and underground, may require large quantities of steel and alloy metals—far in excess of the "railroad era." New mineral and metal commodities may become critical in supply—development of super-conductors may demand large quantities of elements yet to be determined. High-temperature engines using hydrogen fuels may require special ceramic materials replacing metal parts.

Limiting parameters for development and search include supplies of energy, environmental effects, restrictive legislation, excessive taxation, and social pressures. The latter may force ore search to less populated areas.

Cook (1986) foresees that exploration objectives for major natural resource companies will be directed toward (1) world-class discoveries and/or small, high-unit value, low capital-cost deposits that can be profitable even under most adverse market conditions, and (2) polymetallic deposits, particularly those amenable to open pit extraction.

With a willingness to seek and accept new mineral occurrence patterns, and with the aid of improved technology and support from management and government, we should be able to extract the most from our traditional ore hunting grounds; but it may be time also to reexamine the refractory ore deposits, to examine the more remote areas, and to review the more esoteric metal sources.

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Chapter 4.1
GENERAL PRINCIPLES

WILLARD C. LACY

4.1.1 INTRODUCTION

Design and conduct of successful exploration programs involve forecasting, defining of corporate and business objectives, determination of strategy and tactics, scheduling, and budgeting—all carried out within limitations of corporate, local, state, and national policies and procedures. Advanced planning and assessment of budget and personnel needs and financial and technical capabilities save wasted efforts. Instructions such as, "Go find us a mine, then we will decide what to do with it," makes the job of the explorationist difficult and fails to focus search. On the other hand, requirements that are too restrictive may condemn the program.

Some companies have established policies of acquisition of deposits through purchase, thus avoiding uncertainty and risk of exploration. Such a policy is shortsighted in terms of long-term survival of the mining industry. All new mineral concentrations are found through prospecting and/or exploration and made into mines through man's creativity.

4.1.2 OBJECTIVES

The principal objective of mineral exploration is to find an economic mineral deposit that will appreciably increase the value of the company's stock to the stockholder on a continuing basis or enrich the explorer. Bailly (1972, p.31) suggests that objectives in acquisition or discovery of new ore reserves and mineral resources are to maintain and increase profitability of production assets of mine/mill operations and supply raw material to the treatment plant, and to create new assets and profit centers by product and/or geographic diversification.

Each organization involved in exploration defines its own objectives in terms of mineral commodities and geographic locations, acceptable size, life and profitability, and acceptable risk and expenditure. The exploration program should be designed to find and acquire a maximum number of acceptable mineral deposits at a minimum cost and within a minimum time period.

Exploration is an economic activity—a form of investment for the future or insurance. Companies undertake the cost of searching for new deposits with hope of finding deposits that can either be sold or developed at a substantial profit. If they are successful, they should realize a rate of return on their exploration investment that more than compensates for their cost of capital and the high risk associated with exploration activities.

4.1.3 STRATEGIES

Prospecting and exploration strategies vary widely, dependent upon the mineral commodity sought, the geologic and climatic environment, political and social restrictions, and available resources. Bailly (1972) outlines possible strategies for acquisition of mineral deposits: (1) acquire a producer, (2) acquire developed reserves, (3) develop a known deposit, (4) explore known deposits, and (5) explore for new deposits—(a) near known deposits, (b) in a mining district, (c) in a mineral belt, or (d) in a favorable virgin area. All of these approaches have been successful.

4.1.4 TACTICS

Most exploration programs focus progressively on areas of decreasing size, using methods increasing in cost per unit area, with declining risk of failure. These include (1) "conventional prospecting" consisting of the search for directly observable natural features commonly associated with ore mineralization, or literature and geologic research with selection of geologically favorable localities; (2) multistage coverage of the area selected involving detailed geologic mapping, geochemical and/or geophysical coverage, and/or use of special techniques; and finally (3) a drilling program and/or underground exploration by shafts, drifts, and crosscuts. Emphasis upon each stage or technique depends upon the character of the deposit under investigation, the commodity sought, the human and economic resources available, previous experience of the explorer, and results obtained in earlier stages in the exploration.

Bailly and Still (1973) outline stages of a regional exploration program and detection methods in Figs. 4.1.1 and 4.1.2.

Analysis of satellite data and computer modeling are not generally direct ore-finding tools, but may be integrated with geological, geophysical, and geochemical data to improve the efficiency of the exploration program. They can lead to a better understanding of the relationships and controls of mineralization and increase the probability of making a discovery.

Ore deposits are detected by individuals and the importance of the human resource cannot be overemphasized. Local knowledge, detection methods, and time and money are of little value if the explorationist fails to recognize or misinterprets favorable indications, or management lacks courage or technical resources to proceed.

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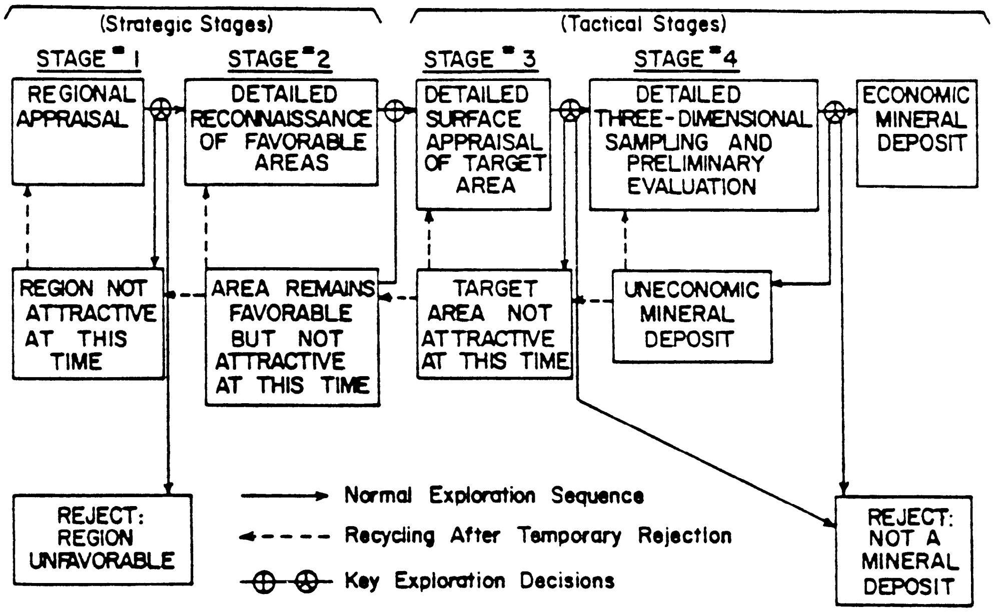

Fig. 4.1.1. Four main stages of "full-sequence" regional exploration (Bailly and Still, 1973).

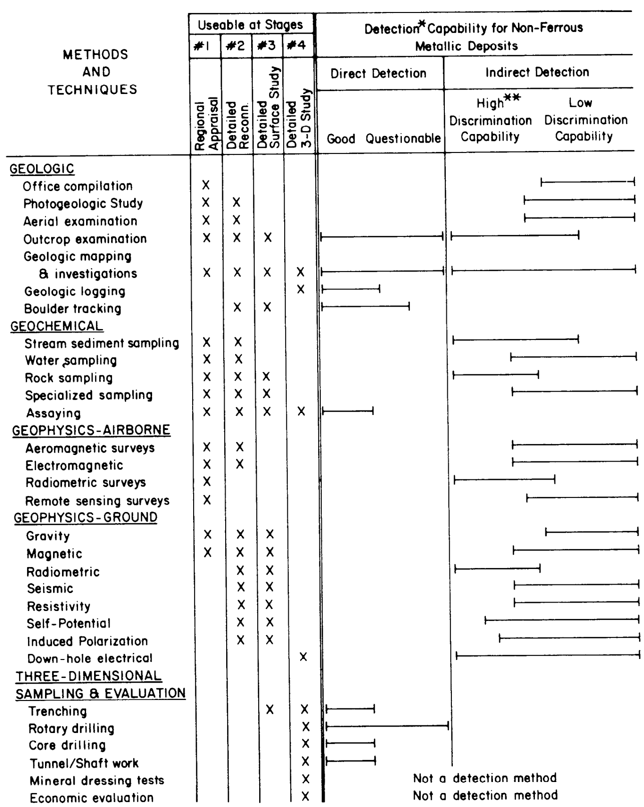

Fig. 4.1.2. Main exploration methods and techniques (scientific aids) (Bailly and Still, 1973).

* Detection refers to the ability to detect a deposit if is there Indirect detection refers to a geological, chemical or physical response showing a deposit may be the cause of the response; this is in opposition to direct evidence of the presence of a deposit.

** Discrimination with regard to indirect methods refers to the ability to determine if a certain response (anomaly) is due to a deposit or to another cause.

Chapter 4.2
FORMATION AND CLASSIFICATION OF MINERAL DEPOSITS

SPENCER R. TITLEY

4.2.1 INTRODUCTION

The knowledge of where and how ores form has been sought by geologists for many hundreds of years since the earliest of the "modern" expositions on the subject by Agricola in the 16th century, and for many centuries before that time, by "natural philosophers." The growth of knowledge about formation of mineral deposits through this period of time has resulted in continuous attempts at classification of ores, each successive attempt building upon accumulated knowledge, as ore bodies continue to be discovered and as detailed knowledge of geologic environments and processes has grown from research in the field and laboratory. Thus knowledge about the formation of ore bodies and their classification have moved on parallel tracks, hypotheses related to formation contributing to modification of existing schemes of classification. In the late 20th century, knowledge of the genesis of ores continues to expand at an ever accelerating pace, and corresponding modifications of classification schemes continue to be proposed.

At a very fundamental level, mineral deposits represent the results of processes that act to concentrate elements in mineral form or in anomalous concentration in other minerals. In those places where the concentration is sufficiently great that such rocks or minerals can be extracted from the earth at a profit, these deposits are called *ores.* The concentrations represent geochemical anomalies within the earth's crust, and their study in this context has led and continues to lead to the development of fundamental scientific information concerning evolution and modification of our planet.

In the study and classification of ores, we search for common, intrinsic, evolutionary, or environmental properties of occurrence of groups of these special rocks and minerals that allow definition and establishment of occurrence of classes or like-kinds of mineral deposits. It is a demonstrated fact that anomalous occurrences of minerals and metals, as well as rocks, *can* be classified in certain important and practical ways. However, discovery and recognition of like properties of genesis and environment of formation of ores, leading to hypotheses of classification, are not merely intellectual exercises. Ore classifications that relate environments and processes (as well as geological time) have immense practical and scientific value for they serve to direct and guide both continuing scientific research and the exploration for undiscovered ore bodies.

4.2.2 OVERVIEW OF ORE GENESIS

Modern knowledge concerning where and how ores form has evolved to its present state as a result of studies of mines, ore bodies, and regional geology during the past century. Although many of our current ideas stem from concepts of much older vintage, the expansion of knowledge about ores dates from the early part of the 20th century with the last half of the century witness to an explosion of knowledge. This growth of understanding is due to many things. The numbers of geologists studying ores during this century has increased from, perhaps, a few hundred to thousands. Further, only since about 1950 has our capability to measure many chemical and isotopic properties of ore systems and rocks been realized and the results of such studies, focused on ore deposits, have tested many early ideas and revealed clearly many hitherto unknown phenomena. And only during the late decades of the century has new knowledge allowed us to ascribe a meaningful association of ore formation with planetary and crustal evolution.

Mineral deposits represent geochemical anomalies in the sense that they are sites of, usually, significant concentration of elements that are ordinarily dispersed in crustal rocks. This concentration takes place in one of two fundamental environments, at the earth's surface, or within the crust, as an end result of one of three primary processes.

The most widespread and common of the processes is the dissolution of chemical components of rocks, their transport in solution and the precipitation of those components as ore minerals at some specific site (trap) of deposition, usually controlled by a chemical contrast or change in chemical environment. It is this localized site of chemical contrast, together with the channeling of metal-bearing fluids into it that results in metal or mineral concentrations. The process takes place at both air- or water-rock interfaces, for example the earth's subaerial surface or the ocean floor, and at deeper levels in the crust. The process of solution-dominated ore genesis is fundamental and makes no distinction as to kind and source of solution, nor of metal or mineral-component source, nor of specifications concerning traps of ore minerals. In view of the complex nature of the crust and its many environments, together with varied sources of fluids and metals, there are clearly many complex combinations of process and environment; the characteristics of such ores reveal a great variety of associations of rocks and different metals.

The second primary process involves the transport of particulate matter at the earth's surface. Certain valuable minerals, which occur in rocks, such as gold, platinum, and gems, are dense, or hard, or chemically inert and survive chemical weathering to be transported in hydrologic systems and reconcentrated. Both water under the force of gravity, and the density contrasts between the mineral in water, combine to move such minerals to sites where localized physical features of containment of the hydrologic system bring about concentration by changing hydraulic competence. The process results in placer deposits that concentrate at contemporary surfaces. Modern streams and wave action are currently producing placers; the process, however, has taken place since water washed over rocks, and there are important placer deposits of gold of great geological antiquity that survived by being buried after formation.

The third important process involves the formation of certain igneous or metamorphic rocks. Chemical and mineralogical evolution of such rocks is attended by the formation and segregation of special groups of ore minerals (e.g., platinum, chromite, and magnetite), which are concentrated at specific places and at specific stages of the processes. Resulting concentrations of these minerals occupy predictable parts of layers of igneous rocks or occur within certain facies of metamorphic rocks. In the case of igneous rocks, the mineral concentrations occur as distinctive flows in volcanic successions, or as layers within magmas that cooled at deep crustal levels. Metamorphism results in chemical changes that move water, resulting in metal concentrations identifiable with certain levels of intensity of the metamorphic processes.

4.2.3 SOME CLASSIFICATION SCHEMES OF THE EARLY AND MID-20TH CENTURY

The understanding of ore genesis is an evolving phenomenon and can be traced with some accuracy in reviewing the nature and rationale of various classification schemes of the past century. For, as concepts change, so does the classification. In order to appreciate the relevance of the direction in which studies of ores has progressed, it is important to know of the early ideas of ore genesis to understand where we have been.

As a knowledge of where metal ores occurred was developed through the 18th and 19th centuries, largely as a result of the industrial revolution, the westward movement of European cultures, and the search for wealth, the first serious attempts at classifying ores ensued. Such attempts were integral parts of broader theories of evolution of the earth and represented polarization of ideas of those who believed, on the one hand, that geological phenomena had their sources within the earth (plutonists) and those who held to terrestrial evolution controlled by events in a primeval ocean (neptunists). It is a curious fact that changing ideas based upon geologic evidence developed during the past century have brought us to a current point of view that both of these early ideas, albeit extensively modified and extended, may be uniquely applied to specific kinds of ores. Without regard to geologic time, it appears to us now that ores formed as a consequence of both, processes acting *within* the earth and processes acting at submarine or subaerial *surfaces.*

Many recent (late 19th and early 20th century) ore deposit classifications stem from Posepny (1898, 1910), whose ideas were extended and modified by Lindgren (1913, 1933) into a classification that remained broadly applicable to many mines and districts in the western hemisphere in the late 20th century. However, the classification has not endured without challenge, and many of its parts have required modification as knowledge of the occurrence of ores has advanced. As is true of the development and use of many classification schemes, it may not be the classification that is at fault; rather it may be its unreserved and unquestioned acceptance, and the way in which it is applied and used by the clientele. Adherents to any classification scheme dealing with natural phenomena are invariably faced with situations in which no category exists to exactly place or identify a phenomenon, or in which detail about a known feature only partially satisfies criteria for a particular class. With only few changes of substance, but in some cases many changes and augmentations in detail, Lindgren's classification has been modified through the years by various authors to accommodate newly recognized characteristics of certain kinds of ore bodies. Graton (1933) and Buddington (1935) split and expanded Lindgren's fundamental depth zones of his classification of hydrothermal deposits to accommodate some otherwise intractable ore styles; Ridge (1972), Park and McDiarmid (1974), and, most recently, Guilbert and Park (1986) have further detailed and expanded the Lindgren classification internally without modification of its fundamental structure. Although Lindgren's classification is conventionally viewed as based upon environments of formation (and he detailed the nature of those environments in a way that had not been previously done), it is actually premised upon processes. Survival of the basic framework of the classification as a fundamental statement about the occurrence of ores is testimony to Lindgren's perceptive mind; the scheme is outlined in the following.

A Classification of Ores Based Upon Processes

I. Deposits produced by mechanical processes of concentration.
II. Deposits produced by chemical processes of concentration.
  A. In bodies of surface waters.
  B. In bodies of rocks.
    1. By concentration of substances contained in the geological body itself (e.g., metamorphism, weathering).
    2. Concentration effected by introduction of substances foreign to the rock (e.g., circulating waters, igneous emanations, directly or indirectly).
  C. In magmas by processes of differentiation.

Source: Modified from Lindgren, 1933.

Lindgren's classification represents but one of many of his time that attempted to categorize ores and that reflected views of different workers. (For an excellent discussion of this subject, see Stanton, 1972, pp.7-35.) Among other significant contributions were those of Niggli (1941) whose framework of igneous rock types encompassed ores based on mineral-metal associations, and of Schneiderhohn (1941) whose classification is premised primarily on processes and secondarily on ore associations. It may be noted that in Schneiderhohn's classification, outlined below, the framework is based upon processes either related to specific geological environments (intrusive rocks and magmas) or to specific processes themselves (i.e., "pneumatolytic"). Further, the mineral-metal associations are those of generally compatible geochemical associations.

A Classification Based Upon Processes and Associations

I. Intrusive and liquid-magmatic deposits (Process).
II. Pneumatolytic deposits (Process).
III. Hydrothermal deposits (Process).
  A. Gold and gold-silver associations.
  B. Pyrite and copper associations.
  C. Lead-silver-zinc associations.
  D. Silver-cobalt-nickel-bismuth-uranium associations.
  E. Tin-silver-tungsten-bismuth associations.
  F. Antimony-mercury-arsenic-selenium associations.
  G. Iron-free associations.
IV. Exhalation deposits.

Source: Schneiderhohn, 1941, modified from Noble, 1955.

Development and application of classifications tend to be strongly provincial, reflecting the particular region or peculiar environment in which geologists work and gain experience. As Lindgren's classification evolved from wide experience in cordillera of the western hemisphere and was widely applied during a time when exploration of the cordillera was active, it is no surprise that strongly held views of processes related to magmatism, as a fundamental driving force of ore formation, came into ascendancy in this region in the first half of the century. Notwithstanding that fact, opposing views were expressed and caution counseled by some workers, notably Van Hise (1903), against uncritical adoption of all aspects of the magmatic theories. Van Hise noted that evidence for metal and solution source was not convincing in any genetic context and that magmatism was not by any means necessarily the sole agent of ore concentration. Testimony to the strength of prevailing opinions at that time is that Van Hise's work may be viewed as having been virtually

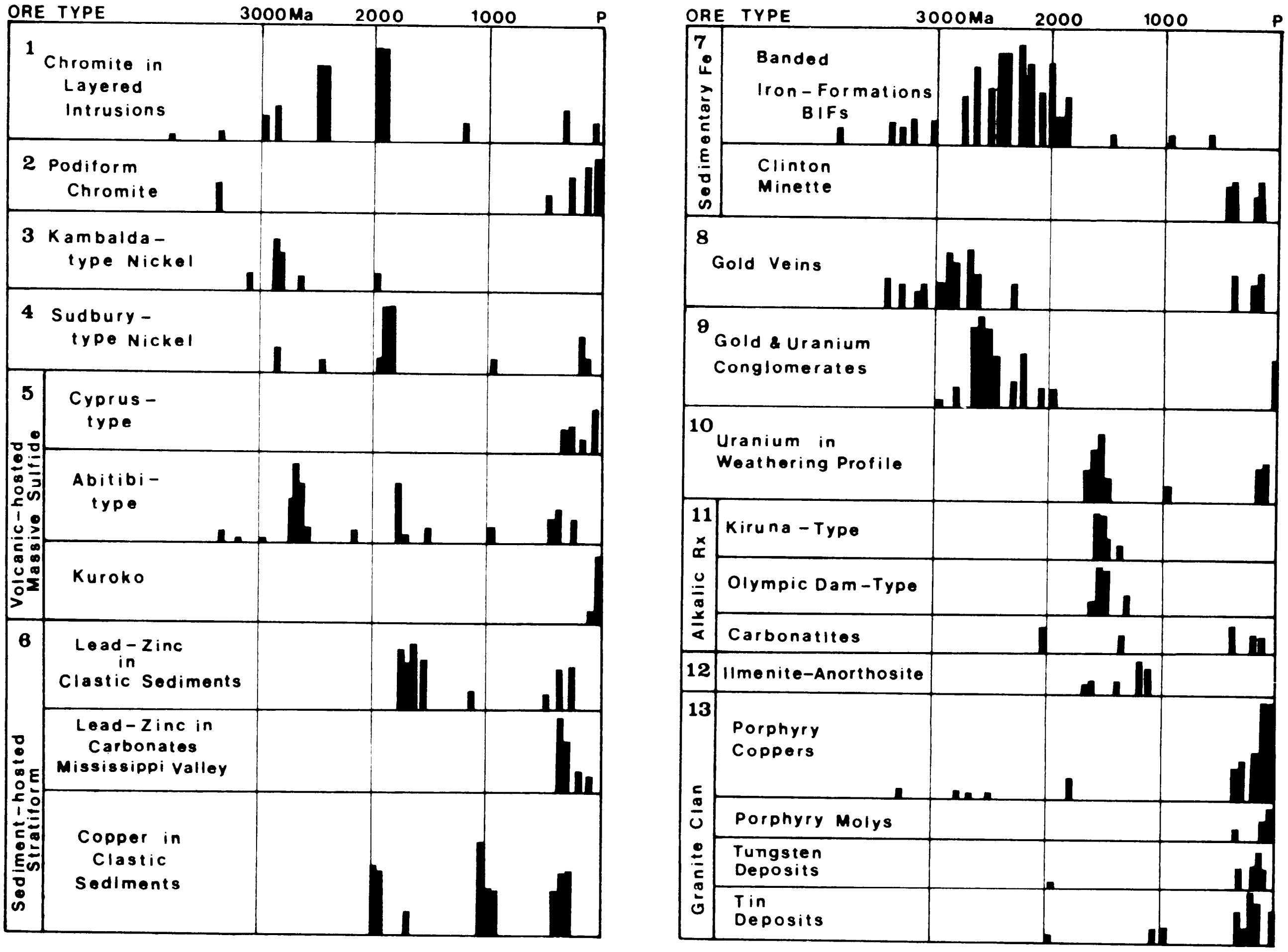

Fig. 4.2.1. Distribution of ore metal deposit types and metal production in geologic time. "The width of each vertical bar represents an interval of approximately 50 million years. The length of each bar is a geological estimate of the percentage of ore formed during that 50 m.y. as related to the total estimated tonnage—the sum of all of the bars *for that type of deposit*—for all geologic time. (From Meyer, 1988; reproduced, with permission, from the *Annual Review of Earth and Planetary Sciences,* Vol. 16. © 1988 by Annual Reviews Inc.)

sunk in a sea of igneous hydrothermalist notions. However, close review of Lindgren's (1933) classification and writings reveal that although he appears to have held strongly to magmatic concepts of formation of metallic ores, he expressed many reservations concerning the origin of metals and waters that formed them, and he clearly set apart certain industrial rocks and minerals.

4.2.4. SIGNIFICANT ADVANCES IN KNOWLEDGE OF THE LAST HALF OF THE 20TH CENTURY

Several important advances and discoveries of the past 40 years constitute bases for a reconsideration and modification of some earlier classification schemes. First, early schemes evolved in the absence of quantitative methods of dating ores; only since about 1955 have radiometric age dates of ores and rocks entered the literature in significant numbers to view *temporal* aspects of ores. A second point stems from this work. Meyer's (1981, 1988) treatises on variations of ore occurrence through geologic time discuss temporal patterns of metal concentration that bear importantly upon genetic classifications and environment (Fig. 4.2.1). A third important factor is the (now) widespread acceptance of the notion that metal sulfides form by various hydrothermal processes at and near the surface of the ocean floor (Routhier, 1963; Derry, 1973). This view, that ores *could* form at an ocean floor surface, was a fundamental working hypothesis applied by many geologists in Europe, Japan, Canada, and Australia for several decades before it was widely accepted in the magmatic-hydrothermal-dominated perspectives of many ore geologists in the United States. Nevertheless, following general and widespread acceptance of the kinds of ideas expounded by Routhier (1963) and Derry (1973), the important sea-floor environment and surficial hydrothermal and related processes still remained detached or isolated from existing and evolving ore classifications.

The period of intellectual evolution of the 1970s was a time when the geology of Precambrian shields was first expounded in a modern geochemical and petrochemical sense (Anon., 1969). There shortly followed significant papers reviewing the place of ores in such settings in Africa (Annhaeuser, 1974, 1981). Uniqueness of the greenstone environments was established as had been the environments of formation of Precambrian iron ores (James, 1954). King and Thompson (1953) suggested the Proterozoic, stratigraphically conformable massive sulfide ores at Broken Hill had been metamorphosed *after* metal deposition rather than formed by metamorphism, and Kinkel (1962) related some massive sulfide ores to sea floor volcanism. In North America, the importance of submarine volcanic processes in greenstone metallization in formation of greenstone ores had been developed with examples by Goodwin (1965), and the environment of concentration of many of these ores was at a contemporary surface. Although Gilmour (1971) proposed a classification of the massive stratabound sulfide ores, the stratabound ores of many geological environments remained isolated from the "mainstream" as classifiable phenomena in "traditional" overviews of ore occurrence.

In the 1970s, many workers attempted to group ores on the basis of their tectonic settings (Sillitoe, 1976), as revealed through the new knowledge of plate tectonics. These new insights of crustal evolution lent themselves to the appreciation of ore genesis in the context of global tectonics (Mitchell and Garson, 1981; Sawkins, 1984). Consequently, the long-recognized *geographic* distribution of mines and districts was given sense and significance from the perspective of *tectonic* environment; for example, the styles of mineral deposits on cratons became more fully understood in a *regional* and *evolutionary* sense as were metal deposits along active continental margins and in island arcs. Hypotheses of processes and environment were advanced in the context of what were perceived to be the effects of subduction of oceanic crust beneath continents or collision of crustal blocks; the sense and importance of continental extensional or compressional tectonics achieved stature in the study of region-wide structural ore control, and integration of knowledge of evolution of Precambrian rocks and ores has now led to assessment of crustal tectonic processes as they relate to these old rocks. The sorting of ore deposits in the context of this tectonic framework serves a useful purpose as most of the environments are not time dependent; the search for ores that may be unique to a specific environment is based solely upon recognition of that environment in the geological record and the assessment as to whether specific metals were concentrated at that time. Such a classification is outlined in the following.

A Classification of Ores Based on Tectonic Settings

I. Deposits formed in continental hot spots, rifts, and aualacogens.
II. Deposits formed on passive contental margins and in interior basins.
III. Deposits formed in oceanic settings.
IV. Deposits of subduction-related settings.
V. Deposits of collision-related settings.
VI. Deposits related to transform faults and lineaments in continental crust.

Source: Modified from Mitchell and Garson, 1981.

4.2.5 A CLASSIFICATION BASED UPON ENVIRONMENT AND PROCESS

Classifications of ore deposits may be created to meet various purposes. In the foregoing, examples have been described and shown that focus largely upon genetic considerations and various lithologic, mineralogic, or tectonic associations. In the following discussion, an outline of ore occurrence will be set forth that is developed upon first, one or the other of two fundamental environments in which ores were originally concentrated (within bodies of rocks or the crust, or at or near a contemporary surface), and secondarily upon processes of concentration. At this writing, the nature of processes that formed some ores and the events that modified them remain uncertain in many instances, as do the environments in which they are presently interpreted as having evolved. Consequently, the placing of some ore bodies into such a classification remains problematical. However, the scheme of classification and the contrasting environments are fundamental and lend themselves to a system in which specific kinds of ore occurrence can be moved to different parts of the format, as different concepts of environment or processes are developed and proven, without changing its basic structure.

4.2.5.1 Environments of Ore Formation

The most easily observed and studied environment is that of the modern surface. This surficial environment, in many different forms and compositions, has existed since formation of the planet and simply constitutes that surface between rock and atmosphere and rock and water. It has been a significant environment of chemical change involving at the one extreme, chemical modification and erosion (oxidizing and acid), and at the other extreme deposition of particulate matter or chemical precipitates (mostly non-oxidizing and acid to alkaline). At the subaerial surface, processes postdate diagenesis; at the submarine surface, processes predate or are synchronous with diagenesis. Many important ore types and ores, essentially synsedimentary or synvolcanic and including some of the largest known concentrations of base and precious metal mineralization known, have been formed in this environment; if they have survived subsequent erosion, they have a common property. The ores occur at an unconformity or disconformity, or within specific strata in the geologic column, and they are grossly concordant with their footwalls. They will be, or have been, modified by the same tectonic or thermal processes that may subsequently modify their crustal environment. Appreciation of this style of occurrence has a high practical value in prospecting as it focuses search toward and within *strata*, or to discernment and prospecting of *unconformities.*

The other environment is that *within* rocks of the crust. Except for ores of magmatic segregation and perhaps pegmatites of granites, these ores are fundamentally epigenetic, that is, they were deposited *after* lithification of their host. The environment is not always specifically restricted to rocks at depth, and it may connect with the surface where, in some cases such as supergene chemical enrichment, processes may act through some feet to 100s of feet (meters to 100s of meters) below it. Whereas ores of this environment, within the exceptions noted, are mostly discordant, they may also be concordant where certain properties of the rocks of the column, such as permeability or "reactivity" of specific strata, are amenable to fluid flow or act as chemical traps for mineral precipitation; they may thus be considered as stratabound. Further, physical modification of rocks below the surface, by surface-related processes, into mineralization traps such as deep karsts and caves, reveal the complex and important interaction between geologic processes acting in both environments.

4.2.5.2 Processes of Mineral Concentration at the Earth's Surface

Fundamental natural processes act to concentrate minerals and certain metals at contemporary surfaces. The effects of these same processes may be observed to result in different mineralization styles because of the different environments in which they take place. For example, hot springs moving through oceanic rocks and surfacing in the thermally insulated environment of the deep ocean result in different products from those of hot springs acting in the subaerial environment. These processes are briefly described below.

Subaerial Weathering: This process acts continuously at contemporary surfaces, but is most important at the atmosphere-land interface. Fundamentally, it is the chemical process that changes mineral, and thereby, rock compositions. Three products evolve; relatively insoluble components of silicate rocks such as clay and bauxite, concentrates of insoluble minerals, and soluble components of rocks that are removed by circulating waters.

Subaerial Chemical Precipitation: Some metallic components released to solution during weathering are reprecipitated in mineral form at various deeper levels (i.e., 10s of feet or meters beneath the surface). Most noteworthy of such mineral groups that form at shallow levels are those minerals of deposits of supergene nickel. They form as a result of weathering of Ni-rich ocean crust and tend to be stratigraphically controlled by levels of soil formation. The supergene concentration of lead and silver at shallow levels is widely recognized. The supergene concentration of copper, gold, and some silver, however, takes place at many levels, some of which are deep (i.e., 100s of feet or meters); controls of this process are chemical and generally related to water-table position, rather than to stratigraphic positions.

Erosion, Transportation, and Deposition: These processes proceed from weathering of rocks and involve the transportation of generally dense, insoluble phases (minerals) of rocks into those concentrations controlled by hydraulic competency, the placer deposits. The erosion of precious metal minerals, together with other dense and resistant minerals such as magnetite, monazite, rutile, and zircon, have resulted in placer deposits in a variety of sedimentological settings; the chemically reducing atmosphere of the Archean is believed to have been an agent that precluded chemical destruction of uraninite, which is concentrated in some fossil placers with gold, and the principal resource of diamonds and other gems lies in placer concentrations developed from weathering of their primary sources. Settings of placer deposits range from streams, through beaches to alluvial fans and deltas. Although placer deposits are abundant and widespread in geologically young sediments, formation of placer deposits has been a time-enduring process, and many buried and indurated placers, such as the gold of the Witwatersrand of South Africa, are of major economic importance.

Water Circulation at the Ocean-Water, Atmosphere-Land Interfaces: This process, most importantly developed in the tidal-flat environment, has been proposed as important in ore formation. Processes acting in this, the *sabkha environment,* have been proposed, although not without controversy, to have resulted in precipitation of important bedded copper ores such as those of the Zambian copper belt (Renfro, 1974). The sabkha process, which takes place in regions of extreme aridity and little topographic relief, involves evaporative pumping of subsurface water through evaporites and decomposing algal mats. The mats are reducing and sulfur-rich, forming metal traps that concentrate metals as sulfide minerals. Biogenic processes acting *within* the algal mats of this environment have also been suggested to be important in concentrating gold in the Witwatersrand strata of South Africa (Pretorious, 1976).

Submarine Precipitation: This important process is believed responsible for the formation of major deposits of ferrous and base metals, as well as gold, across a spectrum of submarine environments. Included are the major iron deposits of the world and the largest deposits of stratiform lead-zinc-silver ores such as Mt. Isa and Broken Hill in Australia; large strataform deposits of copper such as the Kupferscheiffer of Europe may be included as well. Submarine precipitation is also responsible for formation of sea-floor "manganese" nodules. Source of metals is likely diverse and ranges from metals contained in ocean waters to metals introduced into the environment from basin or ocean-floor volcanic hot springs along deeply penetrating faults. The metal concentrating processes may be controlled solely by chemical contrasts in the ocean-floor environment, such as redox changes, or they may be controlled by the existence of chemical or biogenic traps such as the presence of carbonate or sulfide at or near the ocean floor.

Processes at the Crustal Surface Related to Thermal Activity: Localized thermal contrasts developed from emplacement of magmas in continental or ocean crust or from other unknown causes result in the flow of metal-bearing water. In both crustal settings, the water emerges at the surface as hot springs; in continental settings, the hot springs are important sites of deposition of precious metals, mercury, arsenic, and antimony. In oceanic settings, they give rise to concentrations of base and precious metals at rises and spreading oceanic ridges.

Sea-floor springs related to deeply penetrating normal or down-to-basin faults, such as those with rifts may be sources of metals in some reducing sediments such as black shales and some of the important shale-hosted Pb-Zn-Ag (Cu) ores. In such ore-forming environments, metal concentration from additions of hydrothermal fluids may accompany sedimentary diagenesis, and the process may act within the accreting sedimentary section and close to although beneath the contemporary surface. Thermal activity and associated hot springs related to sea-floor volcanism are agents of the process of formation of the important class of volcanogenic massive sulfide ores of Kuroko type in Japan and likely many similar deposits that have formed in similar oceanic setting since the Archean. Stockwork sulfide deposits of the Cyprus type, developed in ophiolite complexes are of problematical origin in the context of their location with respect to the plane of the sea floor, but appear to have had connection with ocean water. However, massive sulfide bodies enclosed in associated pillow lavas appear to be related to sea-floor volcanic activity.

4.2.5.3 Processes Acting Beneath the Earth's Surface

Within the crust, the process of metal concentration takes place from the flow of metal-bearing fluid and precipitation of minerals in some kind of trap, and from the results of metal-bearing waters excluded by crystallization of magmas or recrystallization of rocks. The crustal environments in which fluids are believed to be important agents of ore formation extend from near-surface regions to depths that are unknown, but possibly as deep as 6 mi (10 km); bodies of igneous rocks containing synmagmatic ore minerals form at unknown depths within the crust, possibly as deep as 12 mi (20 km), and metamorphism that results in recrystallization and generation of ore fluids is, likewise, thought to be a deep-seated phenomenon, although there are few criteria by which such depths may be reliably estimated.

Processes Involving Supergene Fluids: Apparently demonstrable supergene fluids; that is, fluids whose immediate origin was at the surface, have been agents of secondarily enriched metals, especially in copper deposits. In the oceanic or basinal environments, fluids that have carried and precipitated metals within oceanic crust may have had their immediate origin in waters of those basins and have been important in high temperature systems. It is not specifically clear that such waters should be classified as supergene, but in the context considered here, their immediate, not penultimate, origins have been from sources above the relevant surface.

Processes Involving Hypogene Fluids: The presence of hydrous minerals in ore deposits, the presence of fluid inclusions in ores, and the evidence from theoretical chemistry point unequivocally toward water-based solutions as a significant agent of metal transport and deposition in many intracrustal ore deposit environments. Hypogene fluids are of high temperature (hydrothermal solutions) and may be inferred from isotopic evidence to have had many different origins peculiar to specific deposit types or to terranes in which deposits occur. Thus fluids may have their sources in igneous plutons, in the included water of sediments in basins, from formation waters derived from sedimentary rocks, from water excluded by dehydration reactions during metamorphism, and from meteoric waters recirculated by thermal and gravity-related processes in shallow crustal environments. Many of these fluid sources (and by inference some metal sources) are distinctive in the context of tectonic environments. For example, basin brines are believed to be important in formation of epigenetic ores in craton settings; magmatic sources inferred to be important in pluton-centered ore systems of cordillera of subduction or collisional settings; and metamorphic fluids important in forming epigenetic ores in the old (Archean) greenstone terranes.

The results of hydrothermal processes in forming ores change across the thickness of crust in which the processes act. Recognition of these changes led Lindgren to outline his hydrothermal depth zone classification and others to modify it. Thus *telethermal* and *epithermal environments* were the shallow cave and open fault systems in which filling of open space by ore minerals was the dominant process of deposition, *mesothermal* the intermediate *environment* characterized by filling and replacement, and the *hypothermal* zone, the deep *environment* dominated by replacement. With the passage of time in which countless numbers of ore deposits have been studied with sophisticated techniques of measurement, the depth zone aspect of the classification of hydrothermal ores has become less distinct and certainly less definable in a rigorous sense than when proposed 60 years ago; many hydrothermal systems transgress the boundaries originally established by Lindgren and others. And we have yet to discover geobarometers that would reliably establish depths of ore formation over much of the proposed range of hydrothermal activity. Nonetheless, the criteria set forth by Lindgren still remain as observable and reliable phenomena to assert for the most part that mineralization in many ore bodies may belong to deep or shallow parts of hydrothermal ore systems.

Processes Related to Magmatic Segregation: Platinum Group Metals (PGM) are enriched in mafic (iron and magnesium-rich) volcanic rocks; in certain mafic intrusions, they have been concentrated by the processes of mineral segregation during crystallization of magma. Deposits of this group are related to Layered Mafic Intrusions, most noteworthy of which is the Bushveld Complex of South Africa. In this body, platinum occurs in such concentration that mines of the Complex are profitable solely because of their platinum content; the Complex is also a significant source of chromium and vanadium, also concentrated in minerals by the process of cooling and segregation of these unusual magmas.

Other metals occur as products of segregation in igneous rocks of other compositions. Magnetite and ilmenite occur as segregated products in gabbros. Granitic rocks, locally anomalously rich in tin, niobium, and tantalum, are sources of extensive tin placers in southern Africa, southeast Asia, and northeastern Australia; in a comparable way, nickel is enriched in some mafic and ultramafic rocks. In old terranes, these lithologies host primary concentrations of nickel sulfides and oceanic igneous rocks, which when weathered, give rise to secondarily enriched and economic deposits of nickel silicates.

Granitic Pegmatites, the Link between Hydrothermal and Magmatic Processes: The pegmatites of intermediate to felsic igneous rocks represent late stage differentiates of granitic magma, rich in alkalies, large lithospheric ions, water, and other volatiles, which cooled to form a specific genetic class of ores. The magmatic-hydrothermal processes are inferred from the presence of minerals of granitic rocks and the presence of hydrous silicate minerals in these bodies.

Metamorphism: The effects of heat and pressure on old rocks at deep crustal levels were becoming increasingly recognized as potentially important in ore genesis by the late 1980s. Formation of granulites and consequent dehydration of the protoliths are suspected to have contributed in some ways to solution of metals and their transportation from deep to shallow levels in the crust.

In addition to effecting solution of and transportation of metals, a further important result of metamorphism has been that of tectonic transport of ores formed as part of the protolith. Examples of this are seen in folded and distended concordant lenses of sulfides in many geologically old terranes, especially where dynamothermal processes have taken place.

4.2.6 SUMMARY: DEPOSITS OF METALLIC ORES

The foregoing has outlined characteristics of processes as they have operated to form some typical styles of ore bodies in different settings. The intent of presentation of this classification has been to provide an overview of ore formation based upon the gross dichotomy of environments. The outline of ores in the framework of contemporary surfaces is presented here in a first attempt at such a classification. The outline of ore occurrence within the crust is an extension of Lindgren’s proposals.

A fundamental aspect of ore search and development planning revolves around the question of *where* ores were formed and what is their style of occurrence. Importantly, what is the likely shape of an ore concentration, and how far is it likely to persist? Fundamentally, does one view a fault system and an igneous pluton as the basic criteria of ore search in a specific terrane, or does the exploration geologist search across strata and along unconformities? Viewed from this perspective, the process of ore formation may in many instances assume a secondary level of importance; hydrothermal processes may be important in each of the two environments considered, but the results of those processes are controlled by the environment in which they take place. The classification is shown in format style in Tables 4.2.1 and 4.2.2.

4.2.7 INDUSTRIAL MINERALS AND ROCKS

A great variety of rocks and minerals are valuable to man because of certain physical properties or compositions. They are considered here and distinguished from the foregoing discussion of metals and ores because of their general “nonmetallic” nature. And although it is a misnomer in some respect, these commodities have been loosely referred to in that context, as the “nonmetallics” (Bates, 1959). Perhaps the most important and consistent distinction between these commodities and the metallic ores is the (relatively) higher cost of processing of ores compared with that of most of the industrial rocks and minerals.

Although this class of commodities contains within it some that are scarce, for the most part the class comprises naturally occurring materials that are relatively more common than metallic ore minerals, and that may be used with little additional processing beyond transportation to their point of consumption. Viewed in a very general way, rocks and unconsolidated materials are mined in bulk and by virtue of this character are attended

Table 4.2.1. Environments and Ore-forming Processes at or Near Contemporary Surfaces

Processes / Environments	Chemical Weathering (Leaching)	Subaerial Chemical Precipitation	Erosion, Transportation, Deposition	Subaerial/ Submarine Precipitation	Submarine Precipitation	Volcanic Processes	Hydrothermal Processes
Subaerial Surfaces	Bauxite, Clay, Mn, Fe Laterites	Ni Laterites, U Calcretes				Sulfur, magnetite, hematite in volcanic flows	Hot Springs, Hg, As, Sb, Ag, Au
Alluvial, Fluvial Environments			Placer deposits, Precious metals, U, Gems, Resistates	Reduction in algal mats, Base metals, Au, Ag, U			
Lacustrine Environment, Playas				Brines, Borax, U			Hot springs, B, Sb, Hg(?), U(?)
Platform, Low-High Tide Sabkha Environ				Base metals, Au/Cu ores (Zambia?)	Cu ores, Pb/Zn ores (Kupferschiefer?)		
Shelf Environment					Carbonate facies Banded Iron Fm, Shale-hosted Pb/Zn	Algoman Banded Iron Formation	
Euxenic Environment of Basins, Rifts					Pyritic black shale Birmingham Iron, Shale-hosted Pb/Zn	Algoman Banded Iron Formation	Hot springs, Au, Fe, Ni, Ag, Cu, W, Sn
Cratonic Basins					Superior-type Banded Iron Formation		
Abyssal Plains > 4.5 km (2.7 mi) Depth					Deep-sea nodules Mn Fe, Ni, Cu, Co		
Marginal Basins Extensional Environment					Volcanogenic Massive Sulphides	Volcanogenic Massive Sulphides Greenstones, Kuroko	Sea-floor hot springs
Ocean Ridge, Spreading Center							Cyprus-like deposits, Cu

by transportation costs that compose a significant component of their value. Viewed from the perspective of their occurrence, these materials have more value as a consequence of their location than from their intrinsic properties. In contrast, the value of many of these commodities, specifically certain minerals such as diamonds, mica, asbestos, and clay, is in great part dependent upon the quality of the product as it occurs or as it is mined or found, and because their bulk is small compared with that of rocks, the site component of their value is less critical than that of rocks and unconsolidated materials. These characteristics of rocks and minerals have led to a useful and widely applied classification scheme developed by Bates (1959, 1960), who separated these materials into two groups as shown in Table 4.2.3.

In a very general way, group 1 of the classification comprises rocks and unconsolidated materials; group 2 is minerals. From this separation, Bates further selects unit and place values for classification of the industrial rocks and minerals, examples of which are contrasted in Table 4.2.4.

Notwithstanding the economic-geographic basis of this classification, it nonetheless provides a realistic geologic separation of materials allowing a sensible geologic approach to their formation, occurrence, prospecting, extraction, and occurrence. The generally low intrinsic value of rocks stems from their widespread occurrence, common habits, ease of extraction, and negligible processing costs; the higher value of minerals results from their more restricted geographic occurrence, their relative (to rocks) scarcity, and processing costs.

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Table 4.2.2. Subsurface Crustal Environments and Ore-forming Processes

Environments (depth, km)	Processes	Supergene Processes: Reaction and Precipitation	Hydrothermal Processes Meteoric H_2O → Increasingly Primitive → Juvenile H_2O (?) Temp. Cool (100°) ------ to ------ Hot (500°(?))				Magmatic Segregation
	Near surface environment	Secondary Supergene Enrichment; Cu,Pb, Zn, Ag, Au					
0.5	Shallow Crustal Environment (Telethermal*)	Sandstone U,Cu,Mo	Sandstone Uranium, Red Bed Copper, Miss. Valley Pb/Zn/F deposits				
1	Shallow Volcanic and Hot Spring Environment (Epithermal*)		Precious metals, Complex metal assemblage Veins and breccia filling textures				
2-5	Intermediate to Shallow Environment (Mesothermal*)			Base and precious metal ores in stockworks, veins and replacement bodes-mostly intrusion centered			
10-1 5	Deep to Intermediate Environment (Hypothermal*)				Au, Sn, Cu, W veins and replacement ores Batholithic and deep greenstone and crustal environment; metamorphic terrances?		
?? KM	Deep Crustal Environment						In granitic rocks, pegmatites (U,Nb,Ta,F,Sn, REE, Li,Be); in mafic rocks, layered intrusions (Pt,Cr,V,Fe) diamond pipes(?)

*Essentially, Lindgren's (1933) Hydrothermal Depth-Zone Classification as modified.

Table 4.2.3. Twofold Subdivision of Nonmetallics

Aspect	Group 1	Group 2
Bulk	Large	Small
Unit value	Low	High
Place value	High	Low
Imports and exports	Few	Many
Distribution	Widespread	Restricted
Geology	Simple	Complex
Processing	Simple	Complex

Source: Adapted from Bates, 1960, p. 17.

Table 4.2.4. Examples of Industrial Rocks and Minerals

	Commodity	Unit Value	Place Value
GROUP 1	Granite	Low	High
	Slate	Low-Intermediate	Intermediate-High
	Aggregate	Low	High
	Sandstone	Low-Intermediate	Intermediate-High
	Clay	Low	High
	Limestone	Low	High
GROUP 2	Mica	High	Low
	Beryl	High	Low
	Fluorspar	High	Low
	Graphite	High	Low
	Quartz Crystals	High	Low
	Diamonds	High	Low

Source: Bates, 1960.

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Chapter 4.3
GEOLOGIC PROSPECTING AND EXPLORATION

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4.3.1 INTRODUCTION

Geology provides the framework in which mineral exploration and the integrated procedures of remote sensing, geophysics, and geochemistry are planned and interpreted.

A systematic exploration program is directed toward a mineral discovery that will be evaluated as a *prospect.* "Prospect" may carry several connotations. A geophysicist or geochemist may apply the term to an encouraging anomaly that bears additional investigation. In this discussion, a prospect is a site of recognized and potentially economic mineralization that has not yet come into production as a mine.

The evaluation of a new prospect may be done by the discoverer's own organization, by joint venturers in mine development, and by purchasers of the mineral property. Prospect evaluation and the ensuing feasibility studies are heavily geologic, but they are directed toward the specific economic and engineering considerations of mine development and production. As such, they are discussed most fully in Sections 5 and 6.

4.3.2 GEOLOGY IN THE EXPLORATION SEQUENCE

An initial requirement in exploration is to have a mental image of the *target*—the deposit being sought. Throughout history, prospectors have used a knowledge of known deposits as guidelines in their search for new deposits; in modern exploration, this is identified as the building of an *exploration model.* The exploration model applies to a favorable geological environment and a target with an expected response to the methods and techniques of exploration. The model will necessarily be revised and redirected during exploration. An exploration objective that allows for a discovery among several kinds of deposits will make use of several models.

Exploration may be done as a comprehensive "grassroots" program in a large and geologically attractive region. It may be done within the narrower limits of a specific district having a particular need for a new ore body or a recognized potential for ore discovery.

4.3.2.1 Grassroots Exploration

Grassroots or regional exploration follows a sequence of *reconnaissance, target selection, target investigation,* and *prospect evaluation,* as generalized in Fig. 4.3.1. In a preliminary design or generative stage, a strategic plan is made in relation to an economically acceptable objective, a cost and time budget, one or more exploration models, and the geologic nature of a selected region. The region itself may be selected in regard to uninvestigated terrain within a discernible association between geology and existing mineral deposits. This can amount to a recognized metallogenic province such as the Colorado Mineral Belt (Fig. 4.3.2), the Carlin Trend of gold deposits in Nevada, or the Mother Lode system of quartz-gold veins in California. The region may also encompass a large area with few known deposits but with potentially favorable geologic conditions involving lithologic and structural sequences or even plate tectonics features.

The reconnaissance stages provide for a stepwise screening by geological, geophysical, and geochemical methods in which areas of little or no apparent promise are rejected and the remaining work is focused on progressively smaller and more significant areas of manageable size. Primary reconnaissance, the coarsest stage in screening, amounts to an examination of the entire region for areas with some combination of exposed or indicated geologic attributes that relate to the exploration model. These are selected as favorable areas for further reconnaissance. Secondary or detailed reconnaissance is a process of finer screening for target areas that fit, or have the capability to fit, the exploration model. The reconnaissance stages, beginning with a selected region of several thousand to more than 40,000 mi^2 (100,000 km^2), may reduce the terrain under consideration to something on the order of 40 mi^2 (100 km^2) in a favorable area. A favorable area may then be reduced to less than 20 mi^2 (50 km^2) in a selected target area.

In the target selection stage, detailed surface studies in the target areas provide for the identification of specific targets on the order of a few square miles (square kilometers) or less. During the subsequent target investigation stage, work converges on the target with the highest priority, and the target is explored in its subsurface extent by drilling and sampling. The aim is to discover a body of ore mineralization. In the prospect evaluation stage, the discovery is appraised as a potential ore body, and an acceptable prospect is then approved for feasibility studies, environmental baseline data collection, and eventual development into a mine.

Reality seldom mirrors this neat scheme. A grassroots exploration sequence may be telescoped in various ways. With a stratigraphy-associated exploration model or within a region having considerable postore cover, subsurface work with drillholes may begin much earlier, even as a part of reconnaissance. The discovery of a potential ore body, considered as taking place during the target investigation stage, may actually happen at some point along the way, perhaps during a target area investigation. In the idealized sequence, the region reduces to a single target. In actuality, no target of sufficient interest may be found. Conversely, a number of targets may be selected, and these are investigated in an order of priority based on their relative strength.

As shown in the "data" path in Fig. 4.3.1, data from rejected targets and areas amount to exploration experience; these are collected for use in planning subsequent exploration. New information that can affect the geologic assumptions and interpretations made in prior stages, or even in the exploration model itself, may also come to light; this, shown in the "revisions" path, is used in a reconsideration of earlier decisions that were made and areas that were selected or rejected.

4.3.2.2 District Exploration

In district exploration, as opposed to grassroots exploration, a more limited area of 400 mi^2 (1000 km^2) or less will have been preselected for detailed reconnaissance and a search for targets. There may be a need for a specific kind of new ore body in

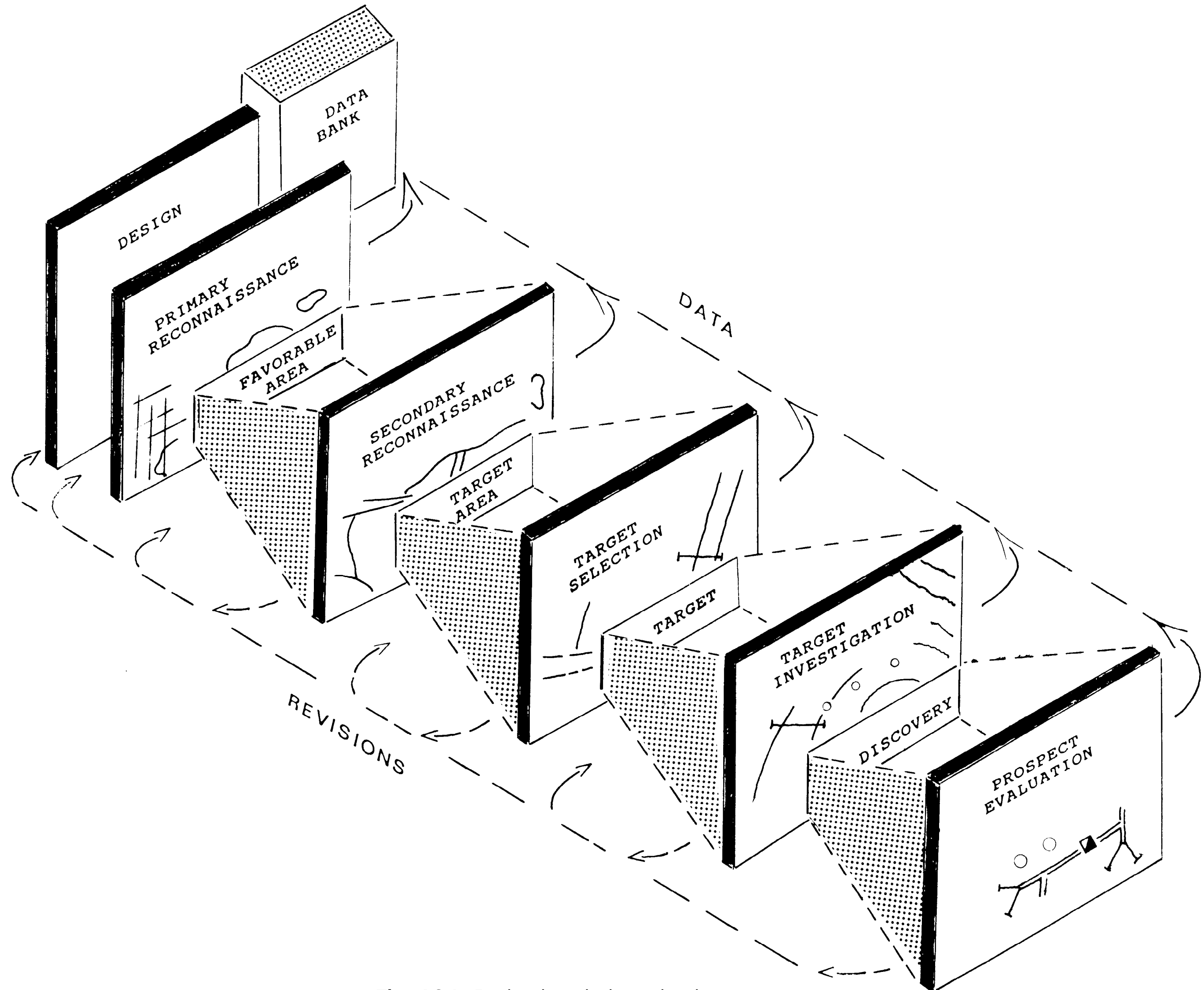

Fig. 4.3.1. Regional geologic exploration sequence.

relation to a mineral-using industry or a near-depleted mine. There may be a newly recognized potential for a significant discovery within an established mining district. This can derive from improved economic and technologic conditions for the mining of a particular commodity, from a perceived application for new exploration technology, or from the identification of important geologic ore controls in similar deposits and analogous terrain.

A district hitherto characterized as having small-scale mines may now be prospected once more in search of larger scale mineralization or for stratigraphically as well as structurally controlled ore bodies. The district may also be "elephant country" where major ore bodies have been mined. Here the search is directed to the discovery of less-apparent and "blind" ore bodies that may have formed at some depth beneath the present surface or that lie beneath unrelated postore lithologic units.

District exploration may take on an intense "gold rush" aspect when it follows the discovery of a new ore body—essentially a new exploration model—in the immediate area. In some instances, such exploration may be overly thorough. It then amounts to what has been termed "saturation prospecting" where an entire district is explored in detail as though it were actually a target area.

District exploration, discoveries, and the identification of potential ore bodies are commonly repeated by several groups and over long periods of time. With newer economic situations and geologic interpretations, information from the rejected targets, uneconomic discoveries, and mines of a prior cycle of exploration will then provide guidelines for the next cycle.

Geologic exploration, whether done in a grassroots or district sequence, relates field observations to the geologic conditions provided in the exploration model and to its expected signatures of geological, geophysical, and geochemical anomalies. Conditions and anomalies are tested in accordance with the model in successive areas and targets until one or more significant discoveries are made, or until the program is abandoned.

4.3.3 EXPLORATION MODELS

Mineral deposits can be expressed in terms of descriptive empirical models, genetic conceptual models, and more compre-

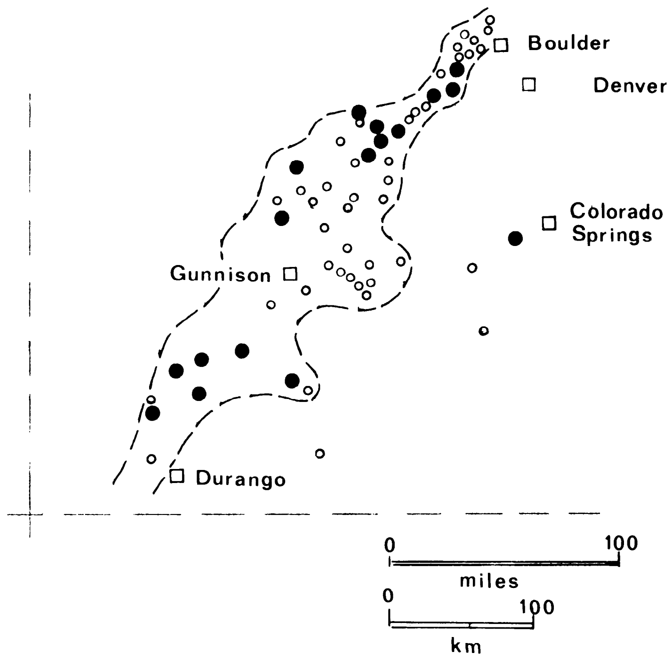

Fig. 4.3.2. Colorado Mineral Belt. Principal mines and mining districts associated with Laramide Tertiary intrusive porphyries and northeasterly shear zones.

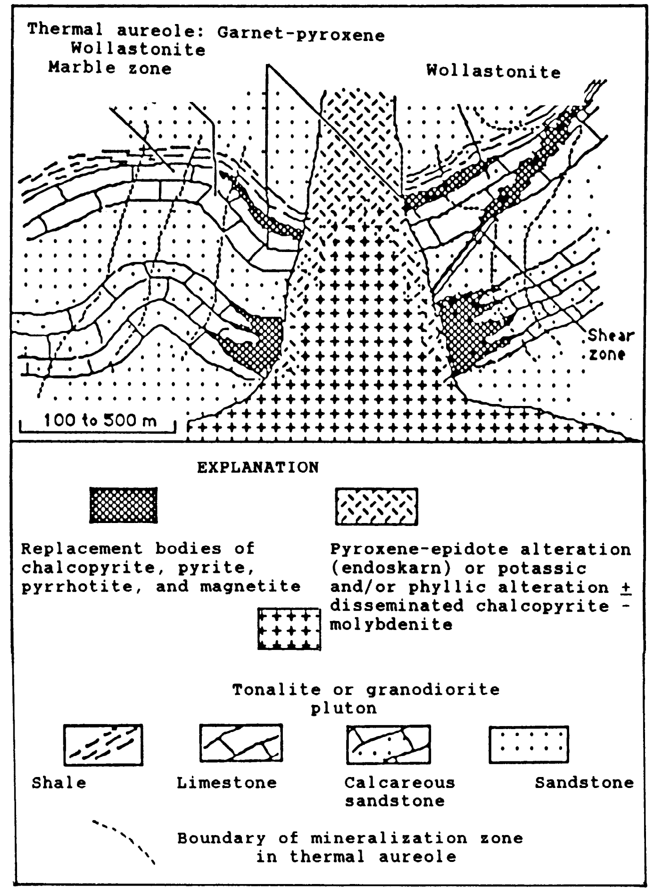

Fig. 4.3.3. Empirical model, illustration: copper skarn. (Cox and Singer, 1986, p. 87).

hensive exploration models. An exploration model is a body of geologic information taken from known deposits and used as target-recognition criteria in a particular program and a particular terrain.

Traditional prospectors have long used direct and indirect knowledge of known ore deposits as guides to new ore bodies. When formally assembled, this amounts to an empirical model describing the target ore body. An empirical model is a thorough compilation of data and observations taken from a group of similar mineral deposits. Empirical modeling relates the type of deposit to a range of geological characteristics, but it does not make specific use of genetic interpretations or concepts. The models are commonly named in relation to deposits such as an alluvial gold placer, a beryllium-lithium-bearing pegmatite, a polymetallic massive sulfide, a volcanic-hosted gold-silver ore body, or a copper skarn ore body (Fig. 4.3.3). Simplification and generalization are inherent; otherwise there would be an inordinate number of separate models, one for each of the individual deposits considered.

Summarized empirical and descriptive models for major types of mineral deposits have been compiled as guidelines to prospectors by the US Geological Survey (Cox and Singer, 1986) and the Geological Survey of Canada (Eckstrand, 1984). In these, examples are cited for each type of deposit, and each model is described in relation to its geological setting, characteristics, and typical conditions of tonnage and grade. The geologic setting is directed to the type, texture, and age of the host rock and associated rocks, to the tectonic situation, and to the association with other kinds of deposits. Deposit characteristics are given in reference to the form of the deposit, to its mineralogy, structural associations, attendant mineral zoning and wall rock alteration patterns, and to its weathering features and geochemical signature.

Taken with concepts on the genesis of a certain type of deposit, an empirical model evolves into a genetic or conceptual model in which the geologic processes associated with the deposit's attributes are identified. A conceptual model is designated according to a deposit prototype such as "Mississippi Valley lead-zinc" or according to group classifications such as "volcanogenic massive sulfide deposits" or "hot spring gold deposits" (Fig 4.3.4).

A conceptual model, accepted with the detailed empirical criteria for recognizing its attributes and associations, provides the fundamentals for an exploration model. An exploration model is formulated in respect to a specific target, or family of targets, that fit the overall geology of the region, an acceptable size and grade of deposit, and the capabilities of the organization. It relates the directly compelling and indirectly permissive characteristics of the expected target to the most appropriate geological, geophysical, and geochemical detection methods, and it provides a basis for designing a sequence of exploration steps and decisions.

Exploration models have an acknowledged limitation. They are drawn from ore bodies that have already been discovered rather than from those that are yet to be found in new terrain. Thus a model will be too restrictive unless it is open to modification as information is gathered during the exploration program. Some of the geological relationships in the model may depend on several alternative genetic concepts; the model must be flexible enough to accommodate the multiple working hypotheses. Assumptions about the area's geology and styles of ore mineralization will change as well. All of this requires feedback to the

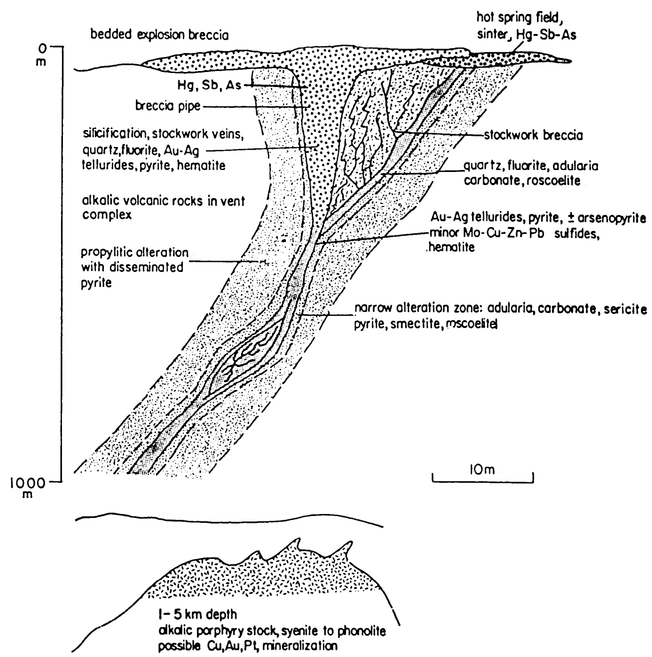

Fig. 4.3.4. Genetic model, schematic illustration: volcanic-hosted, alkalic gold-silver deposit (Bonham, 1988, p. 267). Conversion factor: 1 ft = 0.3048 m

model, with a provision for revising the choice of appropriate recognition criteria.

Mineralization found during the investigation of a specific target may be in an appropriate regional and local pattern but in a different geologic association from that expected on the basis of the guiding model. The remaining and more definitive exploration will then be guided by a model that has been changed to account for another or entirely new type of ore body. This was done in the exploration program that led to the discovery of the Olympic Dam copper-uranium-gold deposit in South Australia. The target investigation was based on an effective series of regional and area investigations, but the initial drillhole intersected significant mineralization associated with sedimentary breccias rather than with an anticipated basalt. Additional drilling and the discovery of economic mineralization in the principal ore body was then guided by a new model.

The format taken by exploration models does not follow a particular standard. It may involve descriptive text, tables and matrices of geologic characteristics, diagrams, and mathematical expressions of correlation between geologic character, tonnage, and grade. A deposit targeted for detection is also expressed in terms of its expected geophysical and geochemical signatures. As models in themselves, these are taken in geologic context and included in the exploration model.

The mass of data that must be processed and displayed in an exploration model is commonly handled by computer. In this regard, another family of models can also be used in selecting areas and designing programs for prospecting. These are computer-processed mathematical models that deal with the probability for certain types of ore deposit to occur as a geostatistical "mineral endowment" within a selected area (Harris, 1990) or to be recognized in a given search pattern (De Geoffroy and Wignall, 1985). One form of computerized exploration modeling, that of "Prospector" and related systems, uses the procedures of artificial intelligence. It accepts a series of actual and simulated data that would be obtained in prospecting, relates this to a network of exploration models for a chosen kind of deposit, and imitates the decision process of a geologist in evaluating the favorability for ore occurrence within a specific target area (Johnson and Keravnou, 1985).

In addition to the geologic models used in exploration, there are models for related purposes. A detailed geologic model of the specific ore body is fundamental to the evaluation and development of a mining project (Chapter 5.5), and models are basic to geostatistical ore reserve estimation (Chapter 5.6).

4.3.4 METHODS OF GEOLOGIC EXPLORATION

Methods of geologic exploration comprise the techniques and procedures of conventional prospecting, interpretation of airborne and satellite imagery, field geologic mapping, and subsurface investigation. Geochemical and geophysical exploration methods and laboratory analytical support are integrated with geologic methods throughout an exploration program.

Exploration is first a matter of using broad-scale geologic reconnaissance to find areas that could reasonably contain a suitable target, then of using more detailed geology to find the target itself (if one exists) and finally a matter of zeroing-in on the bull's eye—an ore body. Whether in extensive grassroots exploration or in more restricted district exploration, the stepwise coarse-to-fine screening provides for the work to be concentrated in the specific areas with a geological identity that most closely fits the exploration model. An initial investigation of an entire region or district in uniform detail would be inordinately costly; the need for time and money increase markedly during subsequent target investigation and delineation.

Geologic maps and geologic mapping are fundamental to exploration. They provide the framework for conventional prospecting, the basis for recognizing geologic target conditions, and the context for designing and interpreting geophysical and geochemical surveys in the stepwise sequence from reconnaissance to detail.

4.3.4.1 Conventional Prospecting

The procedures of conventional prospecting—looking for direct indications of ore mineralization in outcrops and in sediments and soil—are those of the traditional prospector. They remain a part of systematic exploration, and they are now carried out with the support of newer field and laboratory techniques. Conventional prospecting has a spatial aspect that relates to mapping; thus much of the field traversing used in geologic mapping during exploration can also be categorized as "conventional prospecting."

Some of the most direct indications of ore mineralization are associated with prior prospecting and mining, as shown in the character and pattern of old workings. An array of pits and caved adits on a hillside can reflect the trend of an unexposed vein or a bedded zone of mineralization. The succession of material in an old mine dump can indicate the nature of deeper ore mineralization and its lithologic associations. Relative sizes and mineral compositions in a series of mine dumps can give evidence of an ore body's structural attitude, depth of oxidation, and zoning sequence. Old placer workings may have a distribution that relates to undiscovered primary mineralization.

Conventional prospectors have long been aware of the colors and textures shown at the surface by weathered ore mineralization. They have made use of gossan and capping, the characteristic red, brown, and yellow accumulations of cellular limonitic material derived from the leaching of sulfide-bearing veins and disseminated ore bodies. Other guides to ore mineralization in weathered outcrops relate to such colors as the greens and blues of oxidized copper minerals, the black of oxidized manganese minerals, and the yellows and greens of silver halides. These traditional guides to ore have an expanded modern aspect in the applications of geochemistry and laboratory mineralogy to the identification and interpretation of gossans and weathered outcrops.

Prospectors have been drawn to topographic "ledges," the ridges and knobs that may be associated with vein quartz and silicification, and to aligned depressions made by collapse and slumping during the oxidation of ore bodies. Coalbeds that have oxidized and burned in the zone of weathering will also be reflected in a thinner remnant exposure and a surficial zone of slumping. With aerial photography, these geomorphic features, however subtle, continue to be used in ore search.

The tracing of mineralized "float," fragments of ore, gossan, and ore-associated rock that have been eroded and transported from a source area, is a long-recognized conventional practice in prospecting. It is now done with the support of geochemical rock sampling, definitive mineral identification, and the color patterns apparent on aerial photographs. Geomorphic and paleogeographic studies can be involved. A dispersal train or fan of rock and mineral fragments is generally from a source "uphill," but it may also relate to the more recent erosion of an earlier accumulation in ancestral drainages and divides. Even patterns of human occupancy have a place in the tracing of mineralized float; stones in farm walls would have come from less than half way to the next wall. This served as a guide to the discovery of the Tynagh zinc-lead ore body beneath several feet (meters) of glacial drift in Ireland.

The tracing of ore boulders in glaciated terrain served as a guide to the discovery of the Outokumpu copper deposit in Finland, and it is a longstanding practice of conventional prospecting in Scandinavia and Canada. It can now be associated with geochemical prospecting in glacial till, the sampling of the till-bedrock zone in shallow drillholes, and the recognition of specific patterns in glacial drift.

Panning for gold and other heavy minerals in alluvium is a longstanding and still-valid way of collecting geological information and of tracing relatively stable ore minerals to a placer deposit or to an exposed source in a specific drainage basin (Chapter 15.1.1). In modern practice, the sites, such as stream junctions and former stream channels, are selected and mapped on aerial photographs. Heavy mineral panning is now supported by mechanical, electromagnetic, and electrostatic separation techniques and by microscopic examination and instrumental mineralogical analysis. In geochemical exploration, panning is a technique associated with stream sediment and heavy mineral sampling, as described in Chapter 4.5.

The age-old practice of digging pits and trenches into placer deposits and through soil and weathered zones to obtain geologic information and samples is as valid as ever, but it is now done with bulldozers, backhoes, and lightweight drilling machinery. Stream channel depths can be outlined in advance by geophysical methods, and the sites for trenching and drilling are sometimes determined by using hammer-activated seismic equipment to indicate shallow bedrock.

4.3.4.2 Logistics

Four-wheel-drive vehicles are the standard means of field access for conventional prospecting and systematic exploration in relatively open desert and brush country. In remote and less accessible terrain, air transportation is applicable. Even though the cost of operating fixed-wing aircraft and helicopters is high, the alternative of slow surface travel from a distant base camp would often be more costly and would lessen the time spent in actual field geologic work. A common procedure is to establish a base camp at a site that can be serviced by ground transport or by a fixed-wing aircraft and then use a helicopter for access to field sites, cleared landing areas, and temporary "fly camps." In an extended exploration program, a base camp can also be the site of a mobile geochemical laboratory and a microcomputer-equipped workstation in a network that provides field geologists with maps, sample data, and information in direct contact by telephone line with other computers in laboratories, data-handling services, and principal offices.

Helicopters serve conventional prospecting and systematic exploration by providing "fly-over" confirmation for photogeologic interpretation and by giving views of minor but significant terrain features, hydrothermal alteration zones, and old prospect workings in a varied perspective that most aerial photography could not match. Helicopter-borne methods of geophysical prospecting can be incorporated as well. Throughout an exploration program in all but the most densely wooded terrain, a helicopter pilot can place geologists and equipment on sites and near significant outcrops for sampling and field traversing.

4.3.4.3 Scale of Exploration Work

The scale chosen for the recording and study of conventional prospecting data, geologic information, geologic field mapping, and geophysical and geochemical work depends on the extent of the exploration model and its associated features. In reconnaissance, the scale must be large enough to show the relevant geology without too much exaggeration, and yet it must be small enough to give a synopsis of regional patterns in preference to the more intricate geologic detail that comes with larger-scale mapping in target areas.

Primary reconnaissance is done at 1:100,000 (USGS intermediate scale maps) or smaller. In North America, primary reconnaissance in a grassroots exploration program often begins with geologic maps, photogeologic studies, and field mapping at a scale of 1:250,000 (1 in. = 4 mi). A government topographic map sheet at this scale covers an area on the order of 7000 mi^2 (18,000 km^2). Field observation sites can be plotted within a spacing of 0.6 mi (1 km) and related to enough topographic and cultural detail to be located again.

After primary reconnaissance has reduced the area to smaller areas of further interest, the appropriate data for secondary reconnaissance are transferred to a more detailed set of topographic and geologic maps, aerial photographs, and overlay sheets at a scale of 1:24,000 (1 in. = 2000 ft), 1:25,000 or, in some instances, 1:50,000 or 1:62,500 (USGS 15-min quadrangles). The increase in scale from that of primary reconnaissance is on the order of five to ten times; a lesser increase in scale would not be likely to bring out more detail. At 1:25,000, additional observations can be recorded at intervals of 170 ft (50 m); traverse lines for geology, geochemistry, and geophysics can be shown; and individual mine workings, local stratigraphic changes, and local structure can be discriminated well enough to proceed from reconnaissance investigations into more detailed investigation of the target areas.

The scale of geologic work in target areas is likely to be in the range from 1:5000 or 1:6000 (1 in. = 500 ft) to 1:10,000 or 1:12,000 (1 in. = 1000 ft). At this level, prospect pits 30 to 60 ft (10 to 20 m) apart can be shown by separate symbols, and a dike or fault zone several feet (meters) in width can be shown without exaggeration as a single fine line. The succeeding step in mapping within a more closely defined target will often be at 1:1000 to 1:2400 (1 in. = 200 ft) or 1:2500; at this scale, the smaller significant features associated with individual deposits can be shown.

Once a specific site for drilling and trenching has been designated, it may be mapped at a still larger scale such as 1:500 or 1:600 (1 in. = 50 ft) so that the results of sampling as well as the stratigraphic and structural detail can be plotted in their exact locations on maps and cross sections. In the trenches and pits themselves, there may be a need to plot the sample data in even more detail at a scale such as 1:100.

4.3.4.4 Airborne and Satellite Imagery

Because reconnaissance work begins with a synoptic view, it can begin at the scale of satellite and airborne imagery. Imagery includes direct photography and the images derived from other remote sensing data obtained from spectroradiometric channels or bands in the visible and infrared spectral wavelengths and in the wavelengths of microwave and radar systems. Geologic methods in remote sensing are related to geophysical prospecting (Chapter 4.4.) The response is, however, more from the land surface than from depth. Remote sensing has an additional relation to geobotanical patterns, a consideration in geochemical prospecting (Chapter 4.5.) The geologic exploration procedures are essentially those of image analysis, interpretation, and photogeologic mapping.

Satellite Imagery: The most widely used sources of satellite remote sensing imagery have been from the Landsat series of the Earth Observation Satellite Co. (EOSAT) and from the French SPOT series of spacecraft. Both have worldwide coverage from which spectral radiance data in individual scenes can be obtained in computer-compatible tape (CCT) and photographic form.

In the Landsat series, multispectral scanner (MSS) and thematic mapper (TM) data are obtained for ground scenes of 115 × 115 mi (185 × 185 km). MSS provides imagery at a ground resolution of 260 ft (80 m) from two spectral bands in the visible spectrum and two bands with wavelengths that correspond to the infrared portion of the spectrum just beyond the visible zone (Fig. 4.3.5). The thematic mapper on Landsat 4 and 5 provides additional imagery at a ground resolution of 100 ft (30 m) and a higher sensitivity from reflectance in six spectral bands, four in the visible and near infrared range and two in the short wave or middle infrared range. The thematic mapper also provides imagery at a ground resolution of 400 ft (120 m) in a thermal infrared band. Further Landsat missions are planned to have a stereocoverage capability, with a ground resolution of 33 ft (10 m) in the visible and near infrared spectral region, and a ground resolution of 200 ft (60 m) in several thermal infrared bands.

In addition to providing a thorough view of landscape patterns reflecting geologic structure, multispectral scanner imagery displays variations in the radiance given by rocks, soils, and vegetation in each spectral band. By comparing the data from a number of bands at their different wavelengths, types of land surface cover and lithology can be recognized.

The difference in reflectance between two MSS wavelength bands can also indicate a strong spectral absorption associated with ferric iron oxide staining in hydrothermally altered areas. The thematic mapper has the increased capability of differentiating between carbonate and silicate rock assemblages and identifying hydrothermally altered areas characterized by the presence of hydroxl-bearing minerals such as kaolinite, sericite, montmorillonite, and alunite.

Landsat data can be previewed on microfiche cards and can be obtained from EOSAT as computer-compatible tapes or disks and as derived photographic images. Landsat tapes and disks have a higher resolution than the derived images because they are a direct record of the sensor data; they can be processed into images, enhanced to highlight specific data, and enlarged to scales of 1:50,000 for accurate map making or 1:24,000 for geologic field work. Photographic products are 1:1,000,000-scale black and white film negatives and positives from single bands and 1:1,000,000 and 1:500,000-scale film positives composited from several bands in "false color." Paper prints can also be obtained as enlargements at 1:500,000 and 1:250,000 scale.

SPOT is a commercial French satellite system that obtains data for ground scenes of 38 × 38 mi (60 × 60 km) with a capability of stereoscopic viewing. A high-resolution sensor operates in two modes. One mode provides a ground resolution of 66 ft (20 m) for three spectral bands in the visible and near infrared wavelengths. A panchromatic mode provides black-and-white imagery with a ground resolution of 33 ft (10 m) in a single broad band.

SPOT products are available as computer-compatible tapes and derived photographic images. Precision processing of the tape data can provide accurate topographic maps at a scale of 1:50,000 and a field mapping base at a scale of 1:12,000. The photographic products are black and white or color composite transparencies and prints at scales of 1:400,000, 1:200,000, and 1:100,000.

For geologic reconnaissance study, the most widely used Landsat and SPOT products have been black and white prints of specific band imagery and color (false color) composites of three bands. In most color composites, vegetation appears in shades of orange and red, water in black, and soil and rocks in shades of green, gray, red, orange, and brown.

Geologic features can be accented in several ways. Imagery with minimal cloud cover can be selected for times of year when partial snow or vegetation conditions bring out subtle differences in structure or lithology. The data from tapes and disks can also be processed by photographic darkroom techniques, by optical diffraction techniques, or by computer in order to emphasize certain features having geologic context. Computer-aided processing of Landsat and SPOT data makes use of smoothing, filtering, and sharpening techniques. The resulting enhanced images are provided by contrast stretch, by edge detection, and by ratioing–showing the difference in spectral reflectance values between two wavelength bands.

In additional techniques, high resolution black-and-white SPOT imagery is merged with thematic mapper imagery, and topographic or geophysical data are merged with satellite imagery. The display of processed data can range from a computer-drafted isoline map or a sheet of superimposed characters in various densities for each pixel (resolution-size picture element) to a color-photographic rendition of images created and modified in a video display. A certain amount of preprocessing involving geometric correction, scene shifting, and map gridding (geocoding) can be obtained from EOSAT and SPOT. Additional processing can be done by a user with an adequate computer facility or can be obtained from one of several dozen service companies that deal specifically with the custom handling and interpretation of remote sensing data for mineral exploration.

Although Landsat and SPOT multispectral data provide the most widely used satellite imagery for reconnaissance, there are other sources of useful satellite imagery. These involve remote sensing data from other American and foreign satellites and also photography from manned spacecraft.

Shuttle multispectral infrared radiometer (SMIRR) and shuttle imaging radar (SIR) have applications to geologic reconnaissance. Satellite-borne radar is able to penetrate the persistent cloud cover and forest cover of areas such as in the tropics, and it can obtain signals with 80 to 130 ft (25 to 40 m) ground resolution relating to bedrock and its structural patterns beneath several meters of desert sand cover. Limitations exist, however,

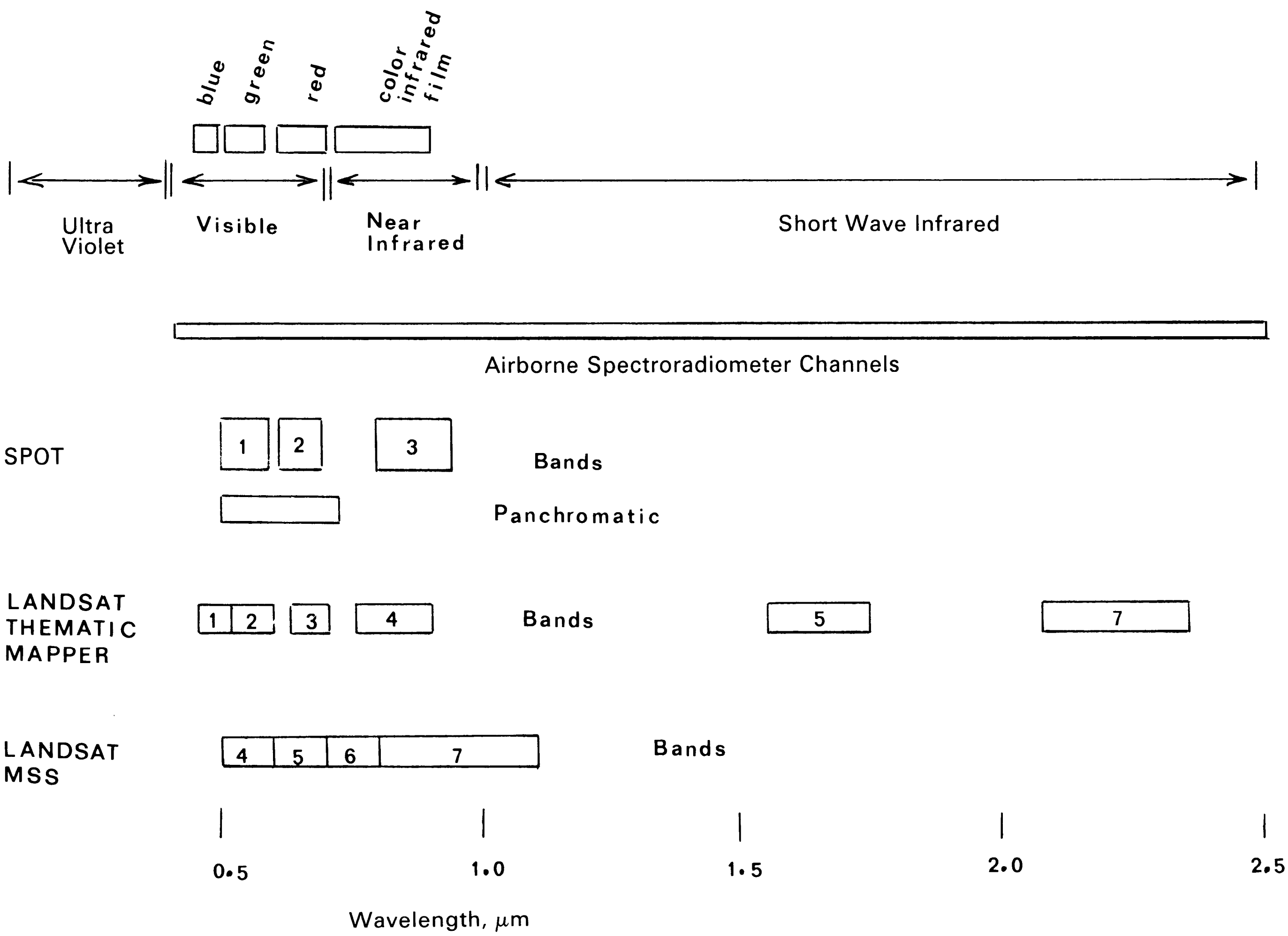

Fig. 4.3.5. Spectral zones and band designations, Landsat and SPOT. TM band 6 is at 10.4 to 12.5 μm in the thermal infrared zone.

in its discontinuous geographic coverage in relatively narrow swaths. Additional satellite-borne radar coverage is represented in Japan's marine observation satellite (MOS), which also furnishes stereo imagery in the visible wavelengths and imagery in the thermal infrared band. The ERS-1 satellite (1992) of the European Space Agency and Canada's Radarsat (1994) are designed with sensors to include radar imagery at 80 to 100 ft (25 to 30 m) ground resolution.

Geologic reconnaissance can make use of the spectacular and high-quality photography of some large areas on the earth provided by the Skylab and the later space shuttle missions. In these photographs, ground resolution is on the order of 30 ft (10 m) with black-and-white film and 60 ft (20 m) with color film.

Satellite imagery is of demonstrated value in reconnaissance work, but there are, as with all exploration tools, some limitations. Cloud cover and vegetation hide a great deal of terrain, and there are expanses of soil and sediment that mask geologic features. Still, vegetation responds in discernible ways to geochemical patterns, large structural features sometimes appear as "ghosts" through overburden, and satellite radar can penetrate some of the cloud cover. One of the most useful features of satellite imagery in comparison with aerial photography and aircraft-based remote sensing is its broader synoptic view of surface patterns and alignments in large areas—just what is needed in the early stages of reconnaissance.

The comparatively low resolution of satellite data, even in camera photography and in high-quality SPOT panchromatic images, prevents it from competing with middle to low altitude aerial photography and airborne remote sensing in detailed geologic mapping at a scale larger than that of reconnaissance. Vertical aerial photography in stereo coverage is the established "workhorse" of geologic mapping.

Aerial Photography: With aerial photography as with satellite imagery, we can obtain synoptic views that would not be possible on the ground. Photogeologic mapping is an established technique in reconnaissance; it provides for an economical and effective map in which geology is keyed to a certain amount of surface observation and "ground truth." Air photos also provide the base for more detailed field mapping. In target areas, geologic features are plotted on stereopairs of contact prints and on single photograph enlargements that are keyed to topographic maps.

There is hardly any upper limit to the scale at which aerial photographs can serve in geologic exploration. Government stereo pairs are provided in black-and-white contact prints and transparencies and to a lesser extent in color photographs at reconnaissance scales ranging from 1:120,000 to 1:20,000. For more detailed geologic mapping, enlargements of single photographs to several times the original scale can provide additional nonstereo prints without losing too much clarity for field work. Larger-scale photography, with stereopairs at a scale on the order of 1:6000 or 1:3000, is furnished for selected areas from new flying by air-photo contractors; these photographs can also be enlarged to provide high-resolution and corrected (orthophoto) prints at a scale on the order of 1:1000 or 1:500.

Whatever the scale of photography, there are certain places in which its use in geologic mapping is limited. Cloud cover may seldom, or never, permit good photography in some areas. Photography can be planned for the right season in areas of deciduous trees, but there are areas in which evergreen vegetation may be too dense in all seasons for an effective look at the ground. Topographic relief in some locales may be so great that the photographs are badly distorted for photogrammetric mapping unless taken from a very high altitude and therefore at a small scale.

Photogeologic work is done by interpreting contrasts and textures in tones and shades of gray and tints of color. Landforms are commonly quite distinct, and the stereoscopic view provided by adjacent photographs emphasizes even the most subtle variations in terrain. Quartzite, limestone, and felsic igneous rocks reflect a higher percentage of the incident light and have light tones; shale and slate show darker tones. A basalt flow or an amphibolite dike absorbs most of the incident light and appears very dark, as would be expected.

Geomorphic relationships, the shape, size, and distribution of topographic features, are the mainstay of photogeologic interpretation for exploration work. Linear and arclike alignments in stream courses, ridges, and vegetation can be related to known or suspected patterns of volcanic caldera features, intrusive boundaries, fault zones, folded beds, and unconformities. Most linear features are, however, labeled as “photo linears” until a field investigation can be made. A photo linear cutting across several tonal and textural zones may turn out to be a vein, a fault, a dike, the faint expression of a structural feature beneath alluvium or soil, or perhaps a fence line or an old road. A “hash” of linears in several directions may afford a significant geologic interpretation as a jointing pattern. It could also relate simply to livestock trails.

Most photogeology is done by direct study of the photographic prints. Aerial photography data can also be enhanced by some of the photographic, optical, and computer techniques mentioned for satellite imagery. These include selective filtering, contrast stretching, and edge enhancement. The final photogeologic map may also be integrated with other data such as topographic contours and geophysical and geochemical trends.

Color infrared (near infrared) aerial photographs provide additional coverage with considerable detail. They are widely available at 1:58,000 scale in the United States through the USGS National High Altitude Photography Program.

Airborne Scanning Systems: Side-looking airborne radar (SLAR) has poorer resolution than photography, but it is used in exploration, especially in areas where forest and cloud cover limit the effectiveness of aerial photography. Geologic features are not seen as such, but faint linear patterns may show up better at 30 to 60 ft (10 to 20 m) ground resolution in SLAR than with aerial photography. SLAR imagery of some one-third of the United States is available from the US Geological Survey as digital data and as 1:250,000 and 1:100,000 photographic imagery.

Airborne remote-sensing imagery has multispectral capabilities similar to those of the thematic mapper, but with data from additional spectroradiometric channels and at a closer resolution. Geographic coverage is still limited, but it is becoming available in mining regions. A system that has been tested over an extensive greenstone region in Western Australia scans a 2.5-mi (4-km) swath in the visible, near infrared, and thermal wavelengths to provide a 30-ft (10-m) ground resolution.

4.3.4.5 Geologic Field Mapping

Geologic mapping is fundamental to exploration, but the making of maps in uniform detail is not an objective. The level of geologic detail relates to the screening process in going from reconnaissance to target investigation. Areas of minimal interest, those that are to be rejected in screening, are mapped only in enough detail to support their designation as such. Areas mapped in considerably more detail are those with enough potential interest to warrant closer examination before deciding on their acceptance or rejection as target areas or targets.

Reconnaissance: The typical mapping base in reconnaissance field geology is a topographic map or an enlarged aerial photograph cut into sheets of manageable size. Working overlay sheets provide initial photogeologic information for “ground truth” checking in the field on apparent geologic unit boundaries, linear features, and known prospects and mines. Other overlay sheets may be used in the field to key-in the indications of available regional geophysical and geochemical data. Widely spaced traverse lines, commonly along ridges and drainages, are often plotted in such a way that information taken from actual outcrops can be differentiated from that which was projected or observed from a distance. Most of the geologic contact and boundary features in reconnaissance mapping will of necessity be designated as “approximate” or “inferred.”

In reconnaissance geologic mapping and in the more detailed mapping that follows reconnaissance, field notes with additional information and sample locations are taken in notebook or in a prepared computer-oriented form. Fig. 4.3.6 shows a portion of a reconnaissance field sheet with numbered locations of additional notes.

Target Areas: Temporary control surveys and base maps in target areas are generally made by the geologist, operating with a plane table or with a combination of compass triangulation and altimetry supported by traverses with tape, threaded “hip chain,” optical range finder, or pace. The traverses are related to patterns of outcrop or to a field grid. Field observations are recorded on enlarged topographic base maps with aerial photographs for reference or recorded directly on aerial photographs for transfer to a map.

In field mapping on aerial photographs, the data are transferred to topographic maps where they can be related to grid lines and survey monuments. Transfer is done with radial line plotting or with the use of camera lucida equipment in which the photo and map can be viewed together. Where the aerial photographs have been processed as orthophotographs to correct for topographic distortion, the geologic data can be transferred directly without further adjustment.

The ideal geologic mapping base would be a series of accurate and distinct contact prints of color aerial photographs in stereo pair with a supporting map at the same scale. In a more attainable situation, mapping is done on photographic enlargements cut to field sheet-holder size. Because enlargements cannot be viewed with a pocket stereoscope, a set of the original stereo pairs of photographs should also be taken into the field for concurrent study.

Mapping is sometimes done directly on the surface of the photo, with inked lines for geologic features and with needle holes through the photo for control stations. A more common practice is to draw the geologic lines and features on an overlay sheet of polyester drafting film that is keyed to reference points

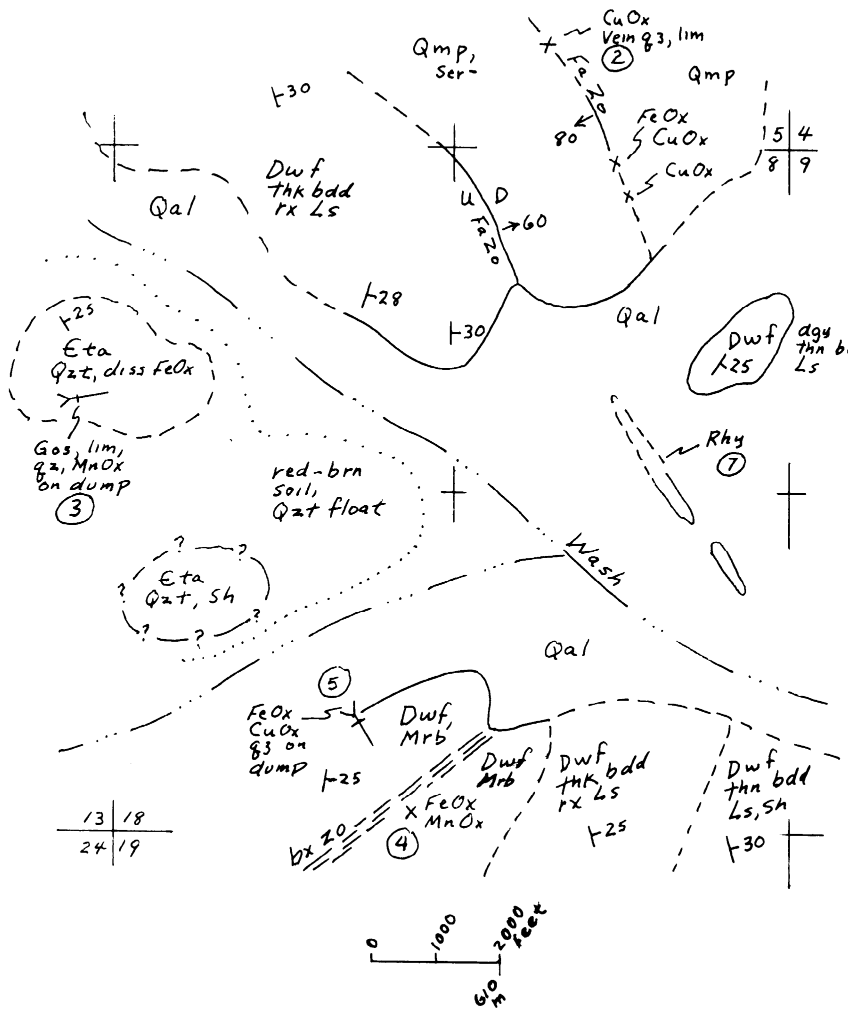

Fig. 4.3.6. Portion of a reconnaissance map field sheet.

on the photo. Additional overlay sheets are used to permit more information to be registered to the photo in the field.

In most practice with overlays, the photograph is left unmarked, and the geologic data are plotted on overlay sheets, as shown in Fig. 4.3.7.

Overlay 1. Outcrop boundaries; lithologic identifications; geologic contacts, notes on texture, color, and weathering; and the numbered location of additional notes.

Overlay 2. Structural data, including bedding attitudes, with notes on cleavage, schistosity, and rock fabric.

Overlay 3. Mineralization and alteration, with notes on geochemical sampling and with the locations of supporting field photographs of outcrops.

The boundaries of outcrops are drawn on the first overlay, and the rock types are identified by their field appearance rather than by their interpreted origin. This is a part of "outcrop mapping" or "multiple-exposure mapping," common practice in large-scale geologic work. Areas of abundant float or of a characteristic soil are outlined as well. Interpretation of the geologic conditions between outcrops is also done in the field, but it is distinguished from the directly observed outcrop data. The interpreted geology will evolve in critical areas with additional mapping, trenching, drilling, and sampling.

Geologic work provides the context for geophysical and geochemical exploration, but geophysical and geochemical surveys are often integrated with geologic field mapping as well. Lithologic units will commonly have a certain signature that can be interpreted and used for mapping in soil and alluvial cover between outcrops. A dike or an igneous contact may be traced by its magnetic or radiometric response in ground or airborne geophysics. A sedimentary formation with carbonaceous material or disseminated pyrite may have a characteristic response to induced polarization surveys. A key limestone bed may have a significantly higher resistivity than the surrounding beds. Soil samples taken across a projected lithologic zone will sometimes have a measurable higher trace-element background than those taken in surrounding terrain. Faintly visible areas of hydrothermal alteration may have a geochemical identity that can be incorporated into more detailed mapping.

4.3.4.6 Subsurface Methods

Pits, trenches, and shallow drillholes are common to conventional prospecting, geologic mapping, and geochemical exploration in areas of soil and alluvium. Deeper drillholes are ordinarily associated with the testing of indicated mineralization in the target investigation stage of exploration. Drilling is the characteristic step to discovery. Drillhole information is even more thoroughly associated with the delineation and sampling of ore bodies during prospect evaluation; as such, the specific methods of drilling and drillhole sampling are described in detail in Chapter 5.3.

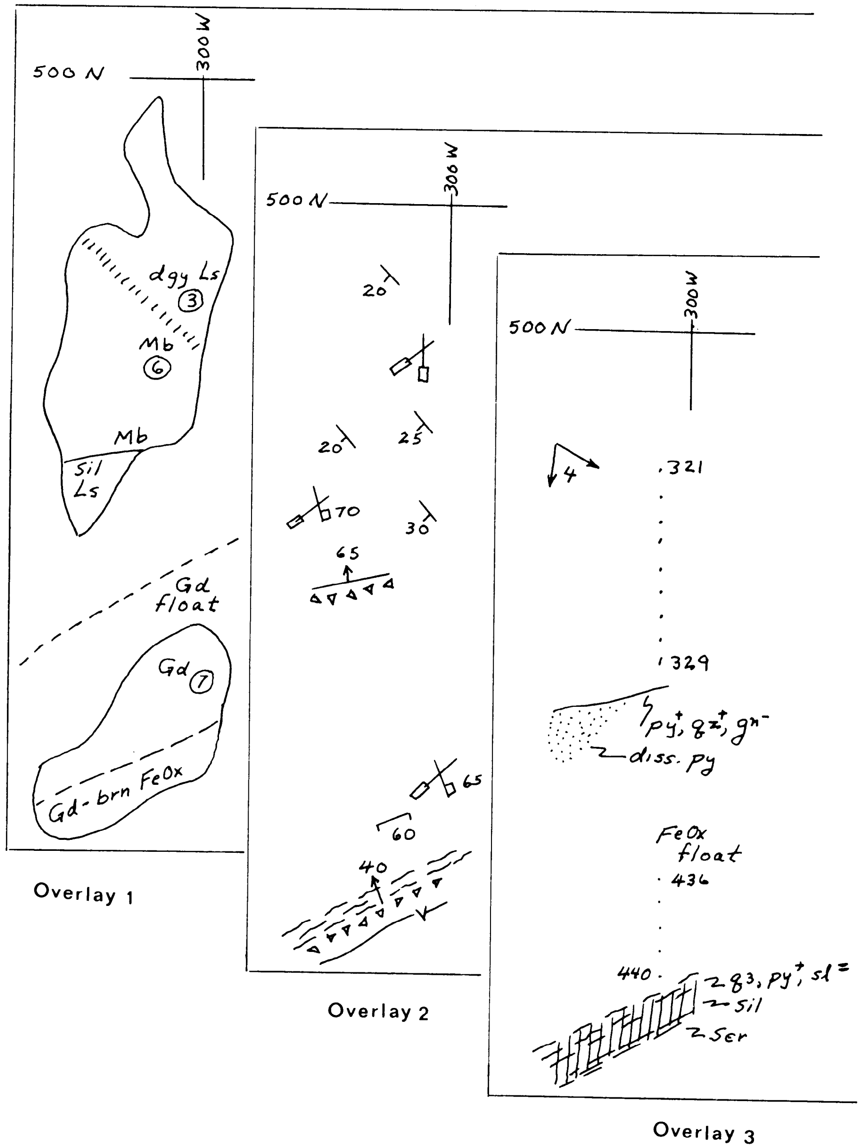

Fig. 4.3.7. Three overlay levels, geologic mapping field sheets. Lithology, alteration, and mineralization would be color-coded.

In systematic exploration programs, drillholes may also be appropriate to reconnaissance and to target area selection. In situations involving sediment-hosted mineralization concealed by younger rocks, "scout" drillholes at intervals of a few miles (several kilometers to tens of kilometers) are often a part of the search for indications of mineralization. In searching for Mississippi Valley-type deposits, for example, reconnaissance holes are drilled for stratigraphic information and for the recognition of significant facies changes and regional reef structures, as shown in Fig. 4.3.8.

4.3.4.7 Laboratory Support

Throughout an exploration program, laboratory work is integrated with the succession of field and interpretive geology. The analysis of geochemical samples (Chapter 4.5) and the assaying of ore mineralization (Chapter 5.4) are of such specific importance that they are treated in special chapters of this handbook. Mineralogical studies begin during reconnaissance, and they have an increasing role toward the stage of target investigation.

The field identification of minerals and rocks is done by hand lens at a magnification of 10 to 20 power and with a binocular microscope at magnifications on the order of 20 to 80 power. In this range, grain size, approximate mineral distribution, and texture can be determined. Laboratory study of mineral fragments and sections of rock and ore with the petrographic microscope provides for a closer determination of mineral associations and textures. Thin section petrography and polished section mineragraphy afford the determination of paragenetic sequences in ore mineralization and the identification of broader halos in mineralization and wall rock alteration. In these studies, microscopic characteristics are related to the geologic context of field observations—to the apparent sequence in lithogenesis, structural stages of folding and faulting, and stages of mineralization.

Laboratory petrographic work includes the study of fluid inclusions, bubbles or vacuoles of fluid, vapor, and solids in transparent and translucent minerals that can be observed in connection with a heating and freezing microscope stage. Geochemical (Chapter 4.5) and thermal studies of fluid inclusions

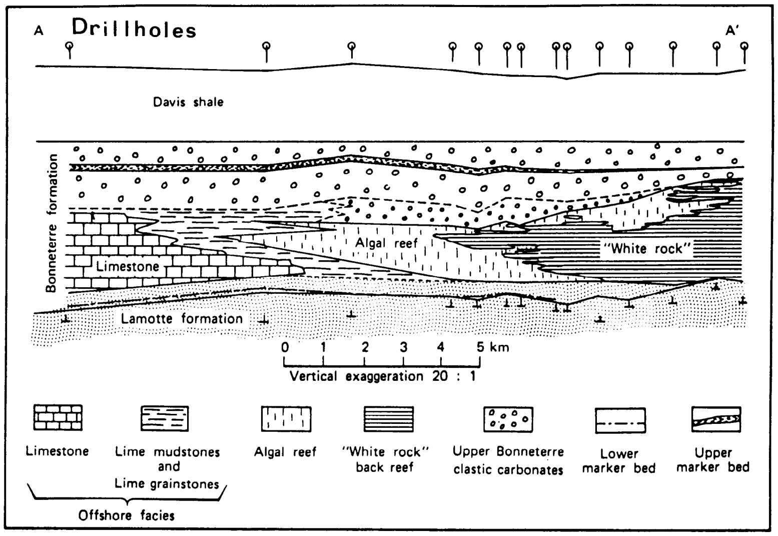

Fig. 4.3.8. Cross section with projected and interpreted data from reconnaissance drilling, Southeast Missouri (Gerdemann and Myers, 1972, p. 430). Conversion factor: 1 mi = 1.609 km.

are used as indicators of the source of ore mineralization and the relative temperature of ore deposition. In the exploration of a mining district, fluid inclusion studies in rock, soil, and gossan samples can provide vectors that point toward a thermal center from which the ore solutions migrated, differentiated, and deposited a sequence of ore minerals.

Combined mineralogical studies and instrumental analysis have a relation to field geology. Whole rock geochemical analyses are used in identifying the particular lithologic compositions associated with the targeted deposits and the accepted exploration model. X-ray diffraction analysis is especially applicable to the identification of fine-grained minerals in zones of alteration. Microprobe and scanning electron microscopy can be used to determine the composition and distribution of minute mineral grains and inclusions.

Some of the laboratory mineralogy and geochemistry relates to age dating and to the pattern of mineralization shown by radiogenic and stable isotopes. Radiogenic lead isotope patterns or signatures can be used to distinguish between a barren "ironstone" false gossan and the actual gossan derived from a sulfide-bearing ore deposit. Lead isotope studies can be used to determine whether the age and style of mineralization in a sample of bedrock or soil belongs to the principal target stage or to a less important stage. In the Coeur d'Alene District, for example, lead isotope signatures provide a basis for differentiating the principal ore mineralization of late Precambrian age from minor ore mineralization of Tertiary age.

4.3.5 EXPLORATION PROGRAM DESIGN

In the design stage of systematic exploration, a sequence of activities is set forth in relation to an exploration model and a designated region, both of which are taken from a compilation and study of existing geological information. The data and information involve regional and local studies and maps, airborne-satellite imagery, and the results of earlier exploration. Sources will range from company files-the in-house data bank-to the published literature and unpublished but available data.

The compilation of published geologic information in respect to the mineral deposits within a region begins with a search for titles and topics dealing entirely or in part with the specific geographic area, its type of geologic setting, and its expected association of mineral commodities. This job is commonly assisted by computerized information retrieval from extensive national and international databases devoted to geology and mining. By using a succession of geographical and topical key words, these can be accessed directly by personal computer, through libraries with search services, or by telephone and mail search centers such as the American Geological Institute's GeoRef Information Services and Research on Demand, Berkeley, CA, which has access to several databases. A reference book by Stark (1988) on sources of information for the mineral industries includes a listing of the principal geological bibliographies, indexes, and international databases.

Regional geologic information, databases, geologic maps, and metallogenic, geophysical, and geochemical maps are obtainable from national and state geological surveys and bureaus of mines. In addition to published information, these organizations can provide access to appropriate open-file reports, field investigation notes, and collections of drill core and cuttings.

Where an exploration program involves older and once-active mining districts, sources of geologic data-including some considerable detail on inaccessible workings-can be found in the archives of state historical societies. The American Heritage Center at the University of Wyoming, Laramie, provides access to an especially large collection of historic mining and geological archives; it holds a number of older mining company's exploration data banks, including the Anaconda Co.'s exploration collection of 1.8 million documents.

The compilation of geologic data for an extensive exploration program necessitates a painstaking evaluation of the material. Older maps may have been superseded by newer ones. Some data will involve projections and interpretations that are inconsistent with other data. Some geologic reports on earlier mined or "promoted" mineral deposits will be more subjective than factual. Faulty data that creep into the design stage can wreak havoc on an entire exploration program.

Aerial photography and remote sensing imagery can be obtained from governmental and commercial sources. In the United States, aerial photographs and NASA satellite and airborne imagery are primarily available from the US Geological Survey EROS Data Center in Sioux Falls, SD. Landsat spacecraft imagery is available from EOSAT, the Earth Observation Satellite Co. in Lanham, MD. Worldwide SPOT spacecraft imagery may be obtained from the SPOT Image Corp. in Reston, VA.

In designing a grassroots exploration program and preparing a format to be used in reconnaissance, the regional data will often be placed in a set of transparent overlay maps that can be studied in various combinations. A reconnaissance scale of 1:250,000 provides a suitable format in which topographic maps, satellite imagery, geologic maps, and a considerable number of government maps with gravity geophysics and airborne magnetic and radioactivity geophysics are available. Information from maps at other scales can then be photographically enlarged or reduced, and maps can also be digitized or obtained in digital form on tape and disk for computer-assisted drafting and plotting at the desired scale. The completed overlay maps can be arranged and studied as such. With maps that are stored and correlated in a computer, several levels of overlay data can also be displayed simultaneously on screen for study at various scales.

The set of maps will commonly have a topographic base sheet. Additional reference information may also be given on a property sheet showing existing mining claims and permit areas, and on an index sheet showing the location and scale of coverage available in more detailed geologic, geophysical, and geochemical surveys. Overlay sheets will include such information as:

1. Geology, simplified for clarity, with emphasis on metallogenic features such as structural zones, intrusions, and selected volcanic or sedimentary formations that can be related to the exploration model. Areas of "cover" by postore rocks and sediments may be shown on the same or another overlay sheet.

2. Satellite and airphoto information, showing significant linear features and remote sensing data on bleached and anomalous color zones.

3. Geophysical and geochemical information, with data points and interpretive isolines.

4. Mines, prospects, and ore mineral occurrences, with relative production and kind of ore.

Finally, a working overlay is used for developing interpretations and outlining areas for specific reconnaissance efforts. In the areas of further interest, the map sheets can then be enlarged for subsequent use in reconnaissance and follow-up detail.

The design stage in an exploration program serves its fundamental purpose of initiating a systematic sequence of tasks, but it carries an inherent load of geologic assumptions. The program and its guiding exploration model must be capable of revision as geologic information is gathered during reconnaissance, target identification, and target investigation.

The objective is a discovery of ore mineralization, a prospect that can be developed into a mine. Section 5 addresses the evaluation of prospects and the succeeding steps in feasibility studies and mine making.

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Chapter 4.4
GEOPHYSICAL PROSPECTING

JOHN S. SUMNER

4.4.1 INTRODUCTION

The term *geophysical prospecting* should be taken here in a broad sense, including not only geophysical surveying as directly employed for the purposes of mineral discovery but also for subsurface geologic mapping by use of geophysical methods. In fact, more often than not, the exploration geophysicist provides supplemental geologic information about subsurface structure and lithology rather than making a direct discovery. The reasons for this more indirect use of geophysics is that, in the "apparent" sense, ore deposit physical property contrasts and geometric features are strongly diluted and distorted when measured through the overburden and wall-rock environment. And an ore body is often a smaller target. Thus direct mineral deposit detection, while very desirable, is not readily achievable. But geophysical methods still can provide critical information about the ore deposit environment even without direct detection capabilities.

The interpretive skills of the geophysicist are greatly improved by information about the physical property contrasts of the materials being sought. Although values from handbooks (Carmichael, 1984; Clarke, 1966) and textbooks (Dobrin and Savit, 1988; Telford et al., 1976; Parasnis, 1983) are useful in the absence of other information, actual mineral deposit physical properties really help in solving the problems of geophysical interpretation. A knowledge of the different physical property contrasts in an ore environment enables the geophysicist to choose the optimal, most effective prospecting method. In general, the geophysical methods used in mineral exploration give ambiguous results that are only resolved by knowledge of physical property contrast and/or the size and shape of the anomalous body. Body geometry and also its specific geometric relationship with the geophysical excitation and detection systems are very important factors in data interpretation, but these problems are solvable only after physical property contrasts are known.

Often more than a single geophysical method should be applied to an exploration problem to improve the odds for success. The principle of synergism applies whereby the total interpretive benefit can exceed the sum of individual contributions. For example, in the resistivity and the induced-polarization (IP) methods, IP contributes information about subsurface mineralization and the resistivity method provides geologic data, while the metal factor ratio is yet a third data set.

A geophysical anomaly can have different meanings, depending on the method being employed and the geologic environment of the survey. An *anomaly* is the difference between an observed response and normal response, given by the equation

$$\text{anomaly} = \text{observed response} - \text{normal response} \tag{4.4.1}$$

Certain small or background geophysical anomalies are to be expected, and they may have little or no exploration significance. Some geologic materials produce unwanted anomalies, while other geophysical anomalies are due to "geologic" or "cultural" noise or errors resulting from equipment limitations or erroneous assumptions of subsurface conditions.

As in any engineering or scientific measurement, an estimate of errors in geophysical values is an important aspect of a survey. Frequently, an error source is overlooked, and if it is large the survey will be adversely affected. The geophysical methods discussed in this chapter together with the basic physical characteristics and the type of applied fields are shown in Table 4.4.1.

The final result of geophysical measurements usually is a map or profile portraying the variation of the measured field in space. Geophysical measurements are taken at points, or "stations," whose location must be known with sufficient accuracy (1) to indicate how the measured quantity changes with location, (2) to determine certain corrections, and (3) to enable anomalous areas to be relocated at any time for future exploration work. Stations may be arranged in a regular grid or an irregular network. Station spacing is determined by the problem at hand—the expected size and depth of the sought-after feature or condition. In some surveys, particularly airborne surveys, data are continuously recorded along profiles. The spacing between profiles is also determined by the nature of the problem at hand.

The planning of a geophysical survey is one of the most important aspects of the prospecting problem. First, one must ask for the most probable geologic model of the target zone in terms of depth, body geometry, and physical property contrasts. Other factors such as terrain, survey costs, land ownership, condition of the environment, and equipment availability may be important. Also the planning geophysicist should not only examine the regional and local maps but should also visit the area to gain insight into the exploration problems. Only then can geophysical feasibility models be hypothesized and cost estimates made. Indeed, it may turn out that blind drilling is more cost effective than geophysical prospecting. To be truly objective and effective, the exploration manager should try to avoid hearsay or preconceived notions in employing geophysics to improve his exploration odds.

4.4.2 REGIONAL GEOPHYSICS

Focusing in on an exploration target is a matter of reviewing a series of scaled maps, first noting the various regional features and their interrelationships. The exploration geologist should not overlook the available regional geophysical maps in planning his exploration program.

4.4.2.1 Remote Sensing

Satellite imagery is available, with varying costs and quality, everywhere on earth. Geophysics has contributed its technology to the image processing and enhancements of satellite images (Mather, 1987), but remote sensing is not considered to be a geophysical surveying method in the very shallow penetrating (down to 0.4 in. or 1 cm) wavelength band. The conventional methods of remote sensing are discussed elsewhere in this section in Chapters 4.3 and 4.5.

High-altitude radar-equipped aircraft can map the earth's topography by using imagery and also side-looking airborne radar (SLAR), which is similar to aerial photography. L-band (390 to 1550 MHz) and P-band (225 to 390 MHz) radar can have penetrations of a few meters in nonconductive rocks and has been found to penetrate thousands of feet (meters) of glacial

Table 4.4.1. Geophysical Methods

Method	Name of Method	Physical Property	Type of Field
Gravity	gravimetric	density	natural gravity
Magnetic	magnetometric	magnetic susceptibility natural remanent magnetism	natural magnetic
Electrical	telluric magnetotelluric self-potential applied potential resistivity induced-polarization electromagnetic controlled-source audio-frequency magnetotellurics (CSAMT) radar, microwave	electrical conductivity dielectric permittivity magnetic permeability	natural telluric currents natural electrochemical reactions artificially applied electric or electromagnetic
Seismic	reflection seismology refraction seismology	seismic wave velocity (elastic moduli and density	artificially created seismic waves
Radiometric	radiometric	radioactive decay	natural radioactive

ice. For example, the sands of the Sahara Desert have been scanned by radar to reveal ancient alluvial channels, incised when the climate was wetter.

4.4.2.2 Aeromagnetic Surveying

Most mineral deposits have some mappable magnetic expression, but few of these appear distinctly on a regional magnetic map. Nevertheless, the geologic trends related to districts and deposits should be examined for supplemental information. Most continental rocks are magnetized by induction, a fact that can be verified by careful examination of the magnetic map.

The earth's normal magnetic field varies from about 0.35 oersted in a horizontal direction at the magnetic equator to 0.65 oersted in a vertical direction at the magnetic poles. Rocks possessing ferromagnetic susceptibility become magnetized in the direction of the earth's field, and some rocks exhibit permanent or remanent magnetization that may be independent of the earth's present field. The earth's normal magnetic field will be disturbed near both of these kinds of rocks.

Magnetometers can measure the intensity of the earth's magnetic field to at least an accuracy of a gamma, or nanotesla (where the gamma is defined as 10^{-5} oersted), and this is quite adequate for regional exploration surveys. A map is prepared showing how the magnetic intensity varies over the area being surveyed, and this is the basis for interpreting the probable distribution of magnetic rocks in the earth's crust. Most magnetic surveys in large areas are conducted from an aircraft or a ship, and the relative cost is quite reasonable.

Magnetic Properties of Rocks: The magnetization of rocks is a complex subject, but for exploration purposes the most important and most common minerals to be considered are the magnetite-ilmenite series and, depending on its composition, pyrrhotite. The susceptibility of a rock to being magnetized in the earth's magnetic field varies almost directly with its magnetite content. In addition to the volume fraction of magnetite, susceptibility will vary somewhat with field strength, grain size, the presence of less common magnetic minerals, and the state of natural remanent magnetization. The relationship, in cgs units,

$$k = 300{,}000\ V_m \times 10^{-6} \tag{4.4.2}$$

is probably as satisfactory as any in relating rock susceptibility k to volume fraction magnetite V_m. Magnetite is a common accessory mineral in many igneous and some metamorphic rocks. Consequently, the magnetite content within one rock type may vary over a wide range, perhaps several percent, with attendant susceptibility variations. The "habit" of rock types in the survey region is empirically established by inspection of field data. The intensity of induced magnetization I_i of a rock due to susceptibility is given by

$$\vec{I}_i = k\vec{H}_o \tag{4.4.3}$$

where I_i is a vector quantity in the direction of the earth's magnetic field H_o. The total magnetization of a rock is the vector sum of the induced and remanent (permanent) magnetization:

$$\vec{I}_t = \vec{I}_i + \vec{I}_r \tag{4.4.4}$$

Unfortunately, I_r is seldom known, so that in practice a hybrid quantity is used that might be described as the "apparent induced magnetization." A thorough discussion of the influence of remanent magnetization on anomaly patterns is given by Hood (1963).

Processing Magnetic Data: The earth's magnetic field is not well ordered. It changes appreciably with time. Ordinarily, the time variations of magnetic intensity are gradual, especially at lower geomagnetic latitudes. Strong, abrupt variations in magnetic intensity sometimes occur and are caused by solar-related magnetic storms. At high geomagnetic latitudes, useful data seldom can be obtained during such events. The earth's magnetic field cannot be described simply, but it can be modeled, by using spherical harmonics, with a fair degree of accuracy. The earth's regional magnetic field can be removed from exploration data to create a more easily interpretable residual magnetic map. Topographic effects exist but seldom are evaluated because of the uncertainty in assigning effective susceptibilities for surface formations. Lightning strikes on topographic highs cause erratic remanent magnetic effects due to isothermal remanent magnetism. To create an interpretable map, the earth's total magnetic field is measured, variations with time and longer distances are removed, instrument readings converted to gammas, and the results are plotted.

Interpretation of Results: The objectives of a regional magnetic survey may be (1) the direct detection of magnetic minerals, (2) a means of extrapolating known geology beneath covered rocks, or (3) to give estimates of depths to basement rocks beneath a sedimentary basin. The interpretation procedure, therefore, will vary somewhat, depending on the objectives of the survey. The direct detection of magnetic minerals has important economic implications, because this category would include not only magnetic iron ore but also magnetite-chalcopyrite or pyrochlore occurrences in contact metamorphic deposits, magnetitic nickel ores, pyrrhotite in Cu-Zn-Ag and Ni-Cu massive sulfide deposits, and magnetite associated with gold in placers.

When used as an aid to geologic mapping, the magnetic map is often divided into "provinces" on the basis of the magnetic pattern. Broad areas of little relief often are underlain by sediments, a bird's-eye maple pattern is typical of flat-lying volcanics, and isolated equidimensional anomalies may represent intrusive plugs. Structural and lithologic trends frequently are reflected in the magnetic pattern of metamorphosed areas.

The success of this map interpretative procedure will depend on the geologic control available. The depth to basement and the structure in a sedimentary basin are of concern to exploration in such areas as the Blind River uranium district of Canada and the Southeast Missouri lead belt.

Once the foregoing type of qualitative interpretation has been accomplished, the isolated anomalies may be treated semi-quantitatively. The magnetic attraction for a wide variety of geometric shapes may be calculated and compared with observed data until an acceptable fit is achieved. The magnetic data can also be mathematically inverted to directly interpret the most probable body geometry for a given apparent susceptibility contrast. Magnetic data in itself cannot yield a unique solution for both the geometry and susceptibility contrast of the volume causing the anomaly without additional control, but the number of possibilities can be restricted by using the boundary conditions of geologic reasonableness. An older reference and atlas for use in interpreting total intensity magnetic anomalies has been prepared by Vacquier et al. (1951).

Further models of geologic conditions are given by Grant and West (1965), Nettleton (1976), and Dobrin and Savit (1988).

4.4.2.3 Regional Gravity Surveying

Gravity meters are capable of rapidly measuring the earth's gravity field to an accuracy of ±0.01 milligal, where the milligal is 10^{-3} cm/sec^2. After corrections to the observed data, a map or profile is prepared, which is the basis for interpreting the probable density distribution in the earth responsible for the gravity changes. Precision gravity surveys must be made from a position fixed with respect to the earth, but less accurate surveys can be made from boats, or even to within one milligal from aircraft (Hammer, 1983). Regional gravity surveys are those in which the station spacing is 0.3 mi (0.5 km) or greater.

Density of Earth Materials: The densities of rock types may vary with one or more of the following: compaction, porosity, saturation, texture, composition, age, fracturing, and burial depth. Hand-sample densities from outcrops or drillholes usually are the most practical source of density information. Sometimes the gravity data will permit an assignment of bulk density value.

Processing Gravity Data: Field gravity measurements require a number of corrections to remove nongeologic variations. First, it is necessary to remove real and apparent gravity variations with time, meter drift, and tidal effects. Variation in gravity due to station location and elevations are then taken care of.

The most important consideration in exploration surveys is to reduce the observed gravity in such a way that a high degree of relative, rather than absolute, accuracy is preserved. Exploration surveys are concerned with changes in geologic condition, so that changes in gravity rather than gravity itself are of primary interest. Regional surveys should be referenced to sea level. The reduction of observed gravity follows the formula

$$g_B = g_o - (g_t - kh - T - L) \qquad (4.4.5)$$

where g_B indicates that the reduced gravity is in the form of the complete Bouguer anomaly, g_o is observed gravity at a station with elevation h feet above sea level, g_t is theoretical sea-level gravity, k is elevation factor, T is terrain correction, and L is latitude correction referenced to the equator.

Drift and Tidal Corrections: Changes in gravity reading with time are removed by first observing a base station with a known gravity value. Unknown stations are then observed for a period of time, and finally either the original base or another station with a well-established gravity value is observed. The time interval between base station readings is sufficiently short that overall drift may be assumed to be linear with time. The station designation, time of observation, and meter readings are recorded. Each gravity meter has a scale constant C furnished by the manufacturer that expresses the value in milligal of each dial division on the meter. There are variations in calculation procedures, but effectively drift rate D in dial divisions per minute is computed from

$$D = \frac{(R_{b1} - R_{b2}) - (V_{b2} - V_{b1})/C}{t_{b2} - t_{b1}} \qquad (4.4.6)$$

where R is meter reading in dial divisions, V is accepted gravity value, t is time in minutes, and subscripts $b1$ and $b2$ refer to base station 1 and base station 2. If base stations 1 and 2 are the same, then $V_{b2} = V_{b1}$ and Eq. 4.4.6 is simplified. If they are different stations, then V_{b2} must be carefully established by replicate measurements (three or more) of the type next to be described. The gravity value at some unknown station n read during the interval between t_{b1} and t_{b2} is given by

$$V_n = V_{b1} - C\,[\,(R_n - R_{b2}) - D(t_n - t_{b1})\,] \qquad (4.4.7)$$

where V_n, R_n, and t_n are observed gravity, meter readings, and time of reading, respectively, at station n.

Theoretical gravity g_t, in gal, and the latitude correction L, which is the earth's normal gravity at sea level, are obtained by solving the Geodetic Reference System 1967 equations to obtain the Gravity Formula 1967, which is

$$g_t = (978.03185)\,(1 + 0.005278895\,\sin^2\phi) \qquad (4.4.8)$$

where ϕ is latitude. Eq. 4.4.8 may be differentiated to obtain the north-south change in gravity at any latitude. Gravity increases from the equator to the poles; therefore the latitude correction L is subtracted as one proceeds northward.

Elevation Correction: The elevation correction k, in milligal per foot, for a surface gravity station is given by

$$k = (0.09406 - 0.01276d) \qquad (4.4.9)$$

where d is the effective surface density in the survey area. For regional gravity surveys, the effective average density of the earth's crust above sea level is taken to be 2.67 g/cm^3. This yields an elevation correction factor of k = 0.05999 milligal/ft.

Terrain Correction: The terrain correction described previously assumes a plane for the earth's surface, but a terrain correction T may be required in hilly country. This correction is always positive because a hill is an excess mass above the station and a valley is a mass deficiency below. Both tend to reduce observed gravity. The correction is evaluated by approximating terrain with geometric shapes for which the gravity effect can be calculated. These may be the cylindrical and annular segments of Hammer (1983) or the vertical prisms and subdivisions thereof as described by Kane (1962) and which are amenable to the digital computer. In Kane's method, the mass within the prism is transferred to a vertical line element at the center of the prism for ease in computation.

Interpretation of Results: After the Bouguer gravity map or profile is prepared by applying the appropriate corrections in accordance with Eq. 4.4.5, variations may appear in the data due to very shallow or very deep density changes that are of no interest to the interpreter. Shorter wavelength anomalous variations are often referred to as "noise," and the longer wavelength variations are called "regional" effects. A discussion of techniques for removing or suppressing such variations is given by Grant and West (1965). When inconsequential variations are removed from a Bouguer gravity map, the result is generally known as a residual Bouguer gravity map, which shows the true shape and amplitude of local anomalies of interest.

The fact that there is no unique interpretation of gravity data has been adequately demonstrated by numerous investigations, but the situation becomes more acceptable if some geologic control or the concept of geologic plausibility is incorporated into the interpretive process. Simple geometric shapes can be postulated to approximate or model the inferred geologic conditions, then the gravity patterns of these shapes are calculated and compared to the residual (observed) gravity values. The model is readjusted until a reasonable fit to the residual gravity is achieved. A perfect fit of the observed and theoretical curves is no proof that a true shape has been computed. Appropriate references to this process include Nettleton (1976), Grant and West (1965), Talwani (1985), and Dobrin and Savit (1988). An alternative, more detailed interpretive method involves the mathematical inversion of the original residual gravity data set, as discussed by Parker (1975).

4.4.2.4 Aeroelectromagnetic (AEM) Surveying

Surveys of the electrical conductivity over broad portions of the earth have been very successful in revealing the presence of electrically conductive mineral deposits. Here the geologic environment is important: (1) rather flat or undulating terrain, (2) richly endowed (in mineral deposits), and (3) thin, concealing, but extensive covering overburden. Literally billions of dollars of massive sulfide ore bodies have been directly discovered in the Precambrian shield areas of the world by using the AEM reconnaissance method, and the ground follow-up geophysical surveying has been equally rewarding. Pemberton (1962) has reviewed AEM methods and equipment.

Fixed-wing aircraft are usually used for reconnaissance AEM surveying, employing either continuous wave frequency domain waveforms (Paterson, 1961) or pulsed time-domain (Palacky and West, 1973) techniques. The INPUT (induced pulse transient) method of A. R. Barringer, a Colorado consultant, has proved to be quite successful in regional exploration for massive sulfides, and Fraser's (1979) helicopter DIGHEM (digital helicopter electromagnetic) system gives useful near-surface resistivity information.

4.4.2.5 Airborne Radiometric Methods

Uranium, thorium, and potassium-40 occur naturally in earth materials, and because these elements are radioactive, their presence can be detected by radiometric surveys. Alpha, beta, or gamma particles may be emitted, but only gamma rays will penetrate a sufficient distance to be useful in airborne exploration. Gamma rays will penetrate several hundred feet (meters) through the atmosphere, but a few inches (centimeters) of earth are sufficient to attenuate natural radiation by half. Therefore, the radioactive elements or their radioactive daughter products must occur in outcrop to be detected. Gamma-ray detectors consist of a crystal that is activated to give off a minute flash of light (scintillate) upon being struck by an ionizing gamma particle. The crystal is coupled to a photomultiplier tube that generates an electrical pulse proportional to the intensity of each scintillation. "Total count" scintillometers give a readout proportional to the total number of flashes above a small threshold of intensity. Instead of a "total count" scintillometer, a pulse-height analyzer can sort out the flashes according to intensity and number. As a result, scintillation spectrometers are available to provide readouts of the source of radiation. The number of counts in each energy channel will be significantly less than the total count, so large crystals and slower, lower-flying aircraft are desirable. The uranium series presents a problem in that some of the daughter products have appreciable half-lives and are fairly mobile. Disequilibrium may result causing the indicated concentration of bismuth-214 to be out of proportion to the concentration of uranium-238 in the source material.

4.4.3 DETAILED GEOPHYSICAL SURVEYING

From closely-spaced measuring stations in a geophysical survey, exploration drilling can be planned. Detailed surveys generally have these stations spaced at 0.5 mi (0.9 km) or less, and ground-based rather than aerial surveying is required. Work can be done on the surface or in boreholes.

4.4.3.1 Surface Methods

The equipment requirements for surface geophysical surveying are similar to the airborne instruments, except for miniaturization. Power needs and power supplies must be small, and the device must be portable enough for one person to carry.

Gravity: The theory and practice of detailed gravity surveying is much the same as the regional work, except for the accuracy requirements. Whereas regional work can accept errors due to special and temporal variations that are ±0.2 milligal or less, detailed work will require errors to be less than ±0.02 milligal. This means that more precise location maps must be used and station elevations must be leveled to a precision of ±0.10 ft (30 mm). Also, closer drift loop control must be maintained and tidal variations watched closely.

New cryogenic and mechanical-optical gravimeters are in use that can measure gravity variation to within a microgal. These devices are proving useful in detecting small masses and subsurface voids such as caves or tunnels.

Some ores such as chromite, hematite, and barite have such a high density compared with the material that surrounds them that they can be located directly by detailed gravity surveys. Buried channels that might contain gold or uranium minerals can frequently be located by gravity or seismic surveys because the channel fill is less dense than the rock in which the channel has been cut.

Gravity surveys are employed in base-metal exploration for a number of purposes. In the southwestern United States, they have been used to find shallow bedrock structures such as the edges of pediments under alluvial cover. In the Canadian Shield, they have been carried out, along with shallow seismic surveying, to differentiate electromagnetic anomalies caused by massive sulfides from those caused by low-density, graphite-filled shear zones. Gravity surveys have also been useful for estimating tonnages in sulfide or iron deposits detected by drilling.

Interpretation of Detailed Gravity Surveys—Because residual gravity values can be stored in computer memory and operated on by digital filters, a number of different interpretational methods are available. The selection and use of these digital filters depend to a large degree on the type of geologic environment and the exploration problem at hand.

Two-dimensional Fourier transforms can assist in removing longer or shorter wavelength geologic effects from gravity data, as can wavelength filters. Directional filters can also select and quantify structural trends.

Upward continuation of gravity data tends to smooth the original data by attenuation of shorter wavelength anomalies relative to their longer wavelength counterparts, a regional consequence of the attenuation of anomaly amplitude with increasing distance from the source. Downward-continuation filtering tends to amplify the shorter wavelengths, so this method can be used for estimates of depth to the causative body. However, because the downward-continuation filters also amplify noise and errors, the method must be applied with caution.

Vertical and horizontal derivatives can also be readily computed for detailed gravity maps, so that certain sizes or shapes can be enhanced. Pseudomagnetic anomalies can be computed too, so that gravity and magnetic maps can be more closely compared. Most of these interpretive methods are discussed, with examples and references, in Dobrin and Savit (1988).

Magnetics: Thanks to electronic technology and related microprocessor instrumentation, truly portable, hand-held but completely automatic magnetometers are now a reality. Usually these instruments are of the proton-precession type, measuring the total magnetic flux density $|B|$ in a scalar sense. A particularly good explanation of field techniques and survey methods has been made available by Breiner (1973).

Because of their dipolar nature, magnetic anomalies and their interpretative methods are more complex than gravity methods, but magnetic resolution of subsurface bodies is much better. The improvement in resolution by magnetics is brought about by a concept in potential theory known as Poisson's relation (Garland, 1971) wherein it is shown that magnetic fields are higher in order than gravitational fields. Poisson's relation also allows magnetic models to be analytically calculated from gravity models. And conversely, a pseudo-gravity map can be computed from a magnetic map to allow a comparison with gravity data in the same area.

Detailed ground magnetic surveys are now quite simple to carry out and are therefore often supplemental to other ground survey methods. Usually the highest cost of a geophysical survey is logistics and transportation, and if these costs are borne by other survey methods, carrying an additional exploration tool such as the magnetometer is well justified.

Interpretation of Detailed Magnetic Data—In general, magnetic data can be interpreted either in a qualitative or a quantitative sense. Although it may seem that the qualitative approach would be simpler, this method is actually based on computed forward models so that the interpreter must have a "feel" for the data to be analyzed. In the qualitative method, one notes the half-width of anomalies because of the simple relationship that the half-width has to the depth to the body. Also magnetic gradients must be noticed and the relationship of the positive anomalies with their corresponding negative anomalies.

Talwani's (1985) method is probably the most widely used algorithm in potential field interpretation. It calculates the anomaly due to a prism or set of prisms of polygonal cross section and either an infinite (two-dimensional) or finite (three-dimensional) strike length. The three-dimensional Talwani method (Talwani, 1985) approximates the source by a set of thin laminae. Each lamina is, in turn, approximated by a polygon in plan view. The method is also applicable to topographic sources because the contours of topographic maps correspond to the boundaries of appropriate laminae. Plouff (1976) has modified Talwani's algorithm to model a stack of layers of finite thickness.

Table 4.4.2. Electrical Prospecting Methods by Source Field

Natural Field Source	Artificially Controlled Source
A. Magnetotellurics (MT)	A. Electrical resistivity (dc)
Telluric-MT (TMT)	B. Transient soundings (TEM)
Audio-frequency MT (AMT)	C. Induced polarization (IP)
Audio-frequency magnetic (AFMAG)	D. Electromagnetic (EM) induction
Telluric current	E. Ground-penetrating radar
B. Electromagnetic (EM) array profiling (EMAP)	F. Controlled-source audio-frequency magnetotelluric (CSAMT)
C. Self-potential (SP)	

Plouff's method overcomes the inaccuracy resulting from the inability to calculate the total field at the same levels as the source body.

Use of the inversion technique in interpreting data is probably more advanced in the magnetic method than in the gravity method today. The matrix method (Bott, 1967) divides the geometry into cells and seeks a numerical solution for a linear integral equation. Parker's (1975) inversion starts with a forward-modeled Fourier domain algorithm and derives an iterative scheme involving transforms to find the distribution of magnetization.

Several computational methods are useful in finding the depth to source, which is useful in finding basement structures. The Werner (Hartman et al., 1971) method uses deconvolution to solve the parametric unknowns of the source geometry. Power spectra of data can be compared with model spectra to find the key body parameters. Peters' (1949) half-slope method, although rather old, is quite simple and straightforward.

Electrical Survey Methods: Detailed electrical surveys have become very diverse over the past two decades, and the reader is referred to Sumner et al. (1967) for discussions of previously established methods. Some techniques (self-potential, telluric-current, and magnetotelluric) depend on naturally occurring electromagnetic (EM) fields. Other EM methods require electric currents or fields that are artificially introduced into the earth. The natural and artificial field methods are summarized in Table 4.4.2. Keller and Frischknecht (1966) reviewed the subject of EM methods very well.

In the broader sense, the electrical methods operate in segments of the electromagnetic spectrum shown in Fig. 4.4.1. Near-direct-current (dc) methods are on the left of the diagram, progressing successively to the gamma-ray region on the far right. Electrical and electromagnetic prospecting techniques generally operate in that part of the EM spectrum below 3×10^4 Hz (30 kc). In this range, radiation and propagation effects are small, and displacement currents are negligible. Thus the investigator is concerned principally with resistive and conductive coupling between circuits. The limiting factor is the skin depth, which is the depth in a medium at which the current density has diminished to $1/e$ (or about 0.368) of its surface value. No prospecting information is obtained from depths greater than the skin depth, and at these low frequencies, conductive bodies do not "reflect" electromagnetic radiation.

Electrical Properties of Rock—The specific resistance, or electrical resistivity, of earth materials depends strongly on the contained volume of water. Although the resistivity of minerals varies over many orders of magnitude, mixtures of minerals (rocks) in their natural environment are somewhat porous, and the pore spaces are occupied by water. Even rock and soil above the water table contains water that is bound to surface grains.

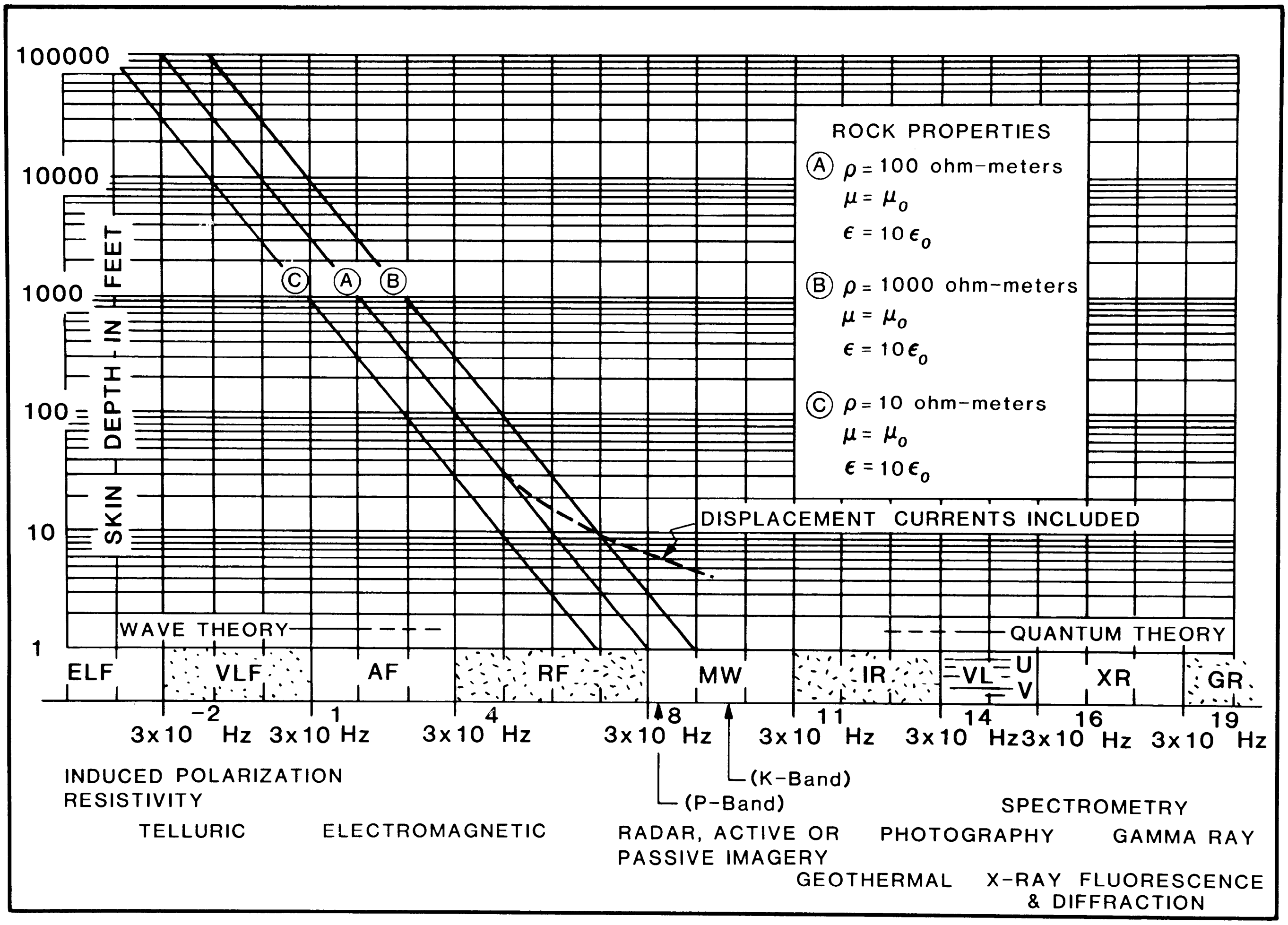

Fig. 4.4.1. The electromagnetic spectrum, skin depth, and the regions in which the different methods operate. Conversion factor: 1 ft = 0.3048 m.

The electrical conduction next to these particles is very high, and therefore it is difficult to detect the water table by resistivity methods alone.

The reciprocal of resistivity is conductivity, measured in mhos per meter. Metallic luster minerals, including magnetite and most base-metal sulfides, are very good conductors of electricity, particularly in the presence of groundwater.

Low-Frequency Electrical Methods—At frequencies below 10 Hz, the resistive coupling through the ground between the current and voltage circuits is dominant. It is necessary to have two voltage detection and two current insertion contacts with the ground for a low-frequency electrical measurement. The theory of low-frequency prospecting methods is discussed by Wait (1982). The electrode array configurations most commonly used for exploration are illustrated in Fig. 4.4.2.

If the values of *V, I,* and the electrode geometry and spacing are known, apparent resistivity of the earth is computed by the appropriate formulas, shown in Fig. 4.4.2. These formulas assume a homogeneous ground, hence the term "apparent." Variations in apparent resistivity plotted as the array is expanded are known as vertical electrical sounding (VES) curves, and depth to surfaces bounding contrasting resistivities can be computed by using a variety of different methods. An electrode array can also be used to traverse the target area, and the values of the resulting apparent resistivities can be plotted and contoured on a map. The dipole-dipole arrays can be used for both sounding and for profiling, and the resulting values can be plotted on an exploration cross section as a "pseudosection" of data (Fig. 4.4.2). The pseudosection is so named because it is not a true electrical cross section of the earth. A description of the electrical resistivity and the related field prospecting induced-polarization methods is given by Sumner (1976).

In the induced-polarization (IP) method, either a variable frequency or a square-wave pulse of current is applied to the earth. If polarizable minerals are present, the ground will become charged while current is flowing, and the resulting discharge can be measured by the voltage detection electrodes. In the frequency-domain induced-polarization method, the voltage differences between the low- (V_L) and high- (V_H) frequencies voltage determines the frequency effect *(FE)* such that

$$FE = (V_L - V_H)/V_H \qquad (4.4.10)$$

using one particular current value. Of course, a standard frequency difference such as one decade must be used for comparative purpose.

In the pulse (or time) method of induced polarization, the ratio of the secondary to the primary voltages is used to obtain the chargeability (M), as shown in Fig. 4.4.3b. Another common

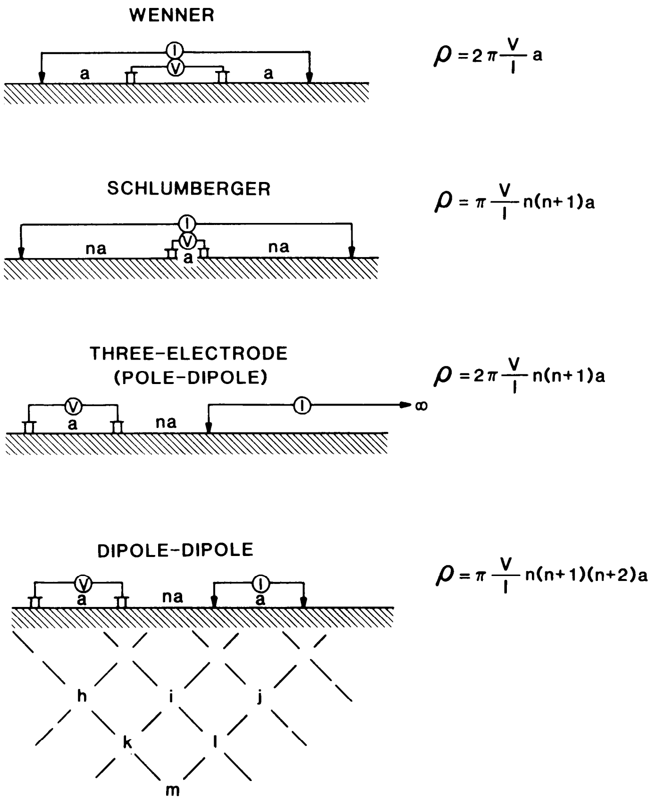

Fig. 4.4.2. Typical electrode arrays for resistivity and induced-polarization surveys (Sumner, 1976, p. 25).

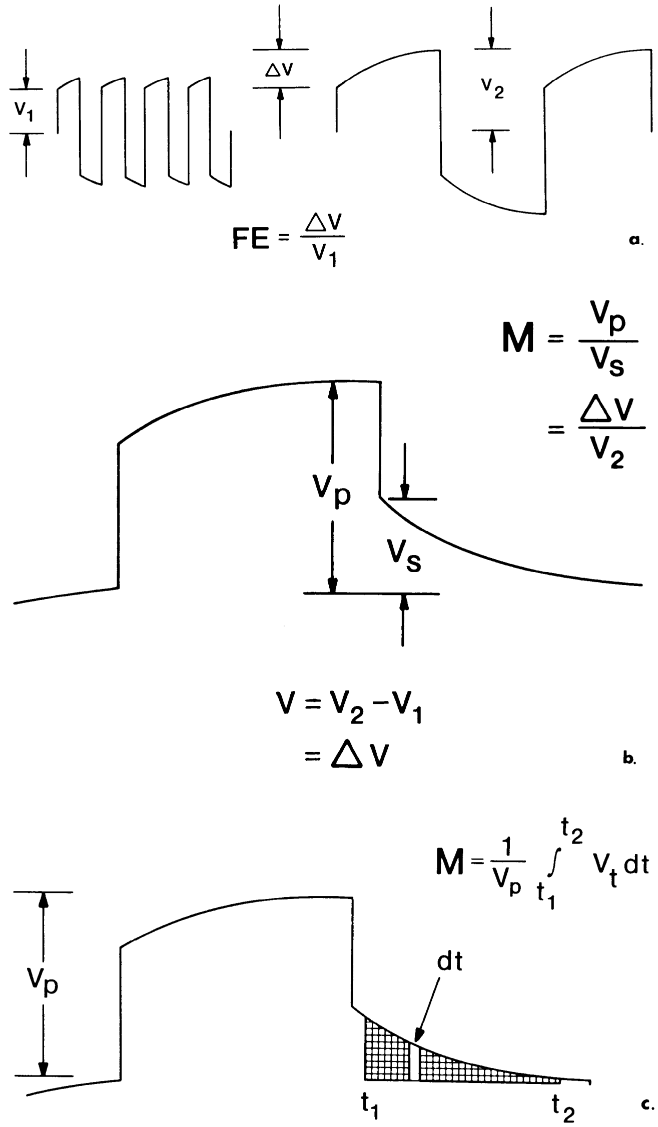

Fig. 4.4.3. Waveforms and parameters in the time and frequency methods of induced polarization. a. Frequency effect measurement. b. Time or pulse IP measurement: $M = V_S/V_P$. c. Chargeability M as an area under the discharge curve.

time-domain method is to measure the area under the decay curve that is also proportional to chargeability (Fig. 4.4.3c).

Complex Resistivity—Geophysicists have noted that the time and frequency induced-polarization methods are equivalent and differ mainly in field instrumentation. A third induced-polarization method measures the electrical phase shift between the transmitter and receiver to provide yet a third set of data. It has been pointed out (Zonge, 1975) that all three measurements can be combined to provide the most comprehensive electrical model of the subsurface.

Complex-resistivity measurements require rather sophisticated computerized instruments in the field, and it takes longer and thus costs more to make a reading. However, the interference by telluric noise is minimized, and it is possible to remove most of the unwanted electromagnetic effects that couple the transmitter and receiver systems. In fact, the electromagnetic response, which often obscures normal induced-polarization measurements, can be observed and interpreted to supplement the induced-polarization and resistivity interpretation.

Natural Source Electrical Methods—The signal source for natural field electromagnetic methods consists of naturally occurring fluctuations of the earth's magnetic field. This field is present everywhere, at any time, and contains a very wide range of frequencies. For exploration, the total range is from about 0.0001 to 10,000 Hz. Below 1 Hz, the field is due to solar-wind interactions with the earth's magnetic field and ionosphere, and above 1 Hz, the source is mainly terrestrial thunderstorm activity.

The self-potential (SP) method measures natural earth voltages existing at a nearly direct current level due to electrochemical action between minerals and solutions. The field survey method is very simple, requiring only a dc voltmeter and two ground controls as well as porous ceramic pots. The theory of Sato and Mooney (1960) is most widely accepted. The natural self-potential voltage of a shallow massive sulfide deposit is invariably negative and less than 1000 mV, although a 1700-mV potential anomaly was described over an alunite body by Kruger and Lacy (1949) at Cerro de Pasco in Peru.

The interpretation of natural-field electromagnetic methods involves recognition that electrical signal penetration into the earth is proportional to the electromagnetic skin depth. However, the magnetic component of the electromagnetic wave is virtually unaffected, so that their ratio gives a measure of apparent resistivity in much the way as making a dc resistivity sounding. A good treatment of natural-field electromagnetic methods is included in the volume by Nabighian (1988).

Controlled-Source Audio-Frequency Magnetotelluric Surveying—Starting initially as an outgrowth of the more academic magnetotelluric method, controlled-source audio-frequency magnetotelluric (CSAMT) surveying has become important in its own right. Basically, the long-wire transmitter is programmed to generate a series of increasingly lower-frequency signals. The portable receiver measures the electric field and the magnetic field at some distance from the transmitter. At a particular receiving station, the series of frequencies "sees" increasing depths of penetration, so these responses can be plotted vertically in serial fashion. Thus a line of receiver stations can yield a plotted parametric cross section of data that can be interpreted to give a geologic section. The controlled-source audio-frequency magnetotelluric method is fairly rapid to run in the field and thus is lower in cost. It has been applied to environmental problems as well as to mineral exploration. Most of the interpreted results appear to supplement knowledge of subsurface geologic structure and lithology rather than making a direct discovery of mineralization. The method is described rather completely by Zonge and Hughes (1988).

Prospecting with Inductive Electromagnetic Methods—All ground electromagnetic methods have two systems: one is the transmitter and one, the receiver. Beyond that similarity, comparisons cease because every imaginable configuration has been tried, and each has its advantage.

1. *The Slingram Method.* One of the most common systems is the Slingram or horizontal-loop method, the coils connected by means of conductive cable. The cable maintains a fixed distance, up to 600 ft (200 m), between the horizontal coils. The cable also allows phase measurements to be made between the transmitted signal and any secondary signals produced by a conductive body. The two coils are carried on traverses that are normal to conductive trends, and measurements are taken at about 75-ft (25-m) intervals. Interpretation is usually carried out by means of modeled type curves, and conductivities can be estimated by means of comparing phase ratios.

The Max-Min system is similar to the Slingram equipment but more sophisticated. The Max-Min data interpretation is well proven, and the equipment is capable of fairly deep (1000 ft or 330 m) exploration.

2. *The Dip-Angle Method.* In this method, sometimes called the vertical-loop method, the coils need not be electrically connected. The transmitter coil is held in a vertical plane pointed at the receiver. The receiver coil is then held nearly horizontal and rocked back and forth laterally to receive the dip angle (or more properly, the tilt angle) null. Two frequencies are used so that conductivity estimates can be made. Unlike the Slingram method, the dip angle (sometimes called the vertical loop method) is asymmetric, and a reversal of transmitter and receiver will give different readings.

In older dip-angle surveys, a large vertical loop was set in the center of a survey area, and one or two receivers covered a square area about 0.3 mi (0.5 km) on a side. This vertical loop transmitter coil configuration was also the basis for the early airborne electromagnetic survey methods, particularly the Inco and McPhar systems.

3. *The Crone "'Shootback" Method and Related Systems.* The dip-angle coil configuration was modified by Crone (1966) to switch the transmitter into a receiver and the receiver to a transmitter, so reversed readings could easily be taken. Also the transmitter plane could be made nearly horizontal with the plane of the coil pointed toward the receiver. Such a configuration removed topographic effects. To add to the versatility, a conductive cable could connect the coils together for phase comparisons.

4. *Turam Method.* With Turam, a long (0.3 mi or 0.5 km), grounded transmitting or large loop cable is used, and lines are surveyed adjacent to it. The two horizontal coils of the receiver are connected by a phase-comparison cable so that signal amplitude and in- and out-of-phase (quadrature) measurements can be made at the receiver site. It is also called the long-line system. An area of about 0.3 mi (0.5 km) on a side can be surveyed from one transmitter position, and two (or more) pairs of receiver coils can be used at one time.

5. *Helicopter Electromagnetic Systems.* Several helicopter-based electromagnetic systems have been used successfully for shallow exploration in the Precambrian Shield and in similar environments. On some the coils are fixed, on others the coils are "button on," and on yet others the helicopter dangles an elongated "bird" with the rigid coil system enclosed in an aerodynamic sheath. In the Dighem system (Fraser, 1979), the receiver coils are continuously electronically rotated, and signals from all of the transmitted directions are received and then unscrambled by onboard digital computers.

6. *Time-Domain Electromagnetic Systems.* A number of time-domain electromagnetic exploration systems similar to the airborne INPUT system have been employed over the past decade in the ground search for massive sulfides. Depending on affiliations of the original system designer, these have been called Utem, Sirotem, Zerotem, and so forth. The pulse electromagnetic system has the advantage over frequency-domain systems of "listening" to the received anomalous signal in virtually the absence of the transmitted signal and the geologic noise. There is also a claim by some that at longer time constants even an induced-polarization response is received.

7. Very Low Frequency Electromagnetic Systems. There are several very low frequency (VLF) radio transmitters operating continuously in countries over the earth. The signals are used for military purposes, mainly by submarines, and the frequency band is designed to have a refracted, ground-penetrating wave. On the 20-kHz band, the depth of penetration into earth material is such that exploration can be accomplished with only a finely tunable radio receiver. In some ways, the survey principle is similar to that of the dip-angle method in its simplicity, but additional electronic phase information is also obtained. Interpretation and contouring of data are sometimes done with reference to Fraser's (1969) article in *Geophysics.*

Although VLF frequencies are "very low" in the electromagnetic communication spectrum sense, they are very high as compared to frequencies normally used in geophysical exploration. As a result, VLF depth of exploration is small, because of the shallow skin depth (Fig. 4.4.1).

Seismic Methods: Reflection and refraction seismic methods depend on the velocities of acoustical energy in earth materials. Accordingly, these involve the creation and transmission of a short pulse of seismic energy and the broad-band recording of the arrival of the train of seismic pulses at distant locations. The time intervals after the initiating energy pulse instant must be determined with millisecond accuracy. Explosives or the impact of a mass furnishes the energy, and direction is accomplished by sensitive seismometers operating into electronic amplifiers and recording equipment. Wave (or optical) theory of travel time reflection, absorption, diffraction, and refraction are applicable to seismic interpretation. This involves quite different physical principles than with the potential methods discussed earlier.

Seismic methods have only been occasionally applicable to mining exploration problems. This is due to the vastly more complex mining geologic conditions, as well as costs. Thus there are few case-history examples of the application of seismic methods to mineral exploration. Most of the use by mining companies has been in engineering, environmental, and hydrologic applications.

Seismic exploration has been used to aid in the interpretation of other data by providing depth-to-bedrock information. For example, Bacon (1966) described basement profiling by refraction seismics as a method of correcting gravity exploration data. The buried alluvial channels of placer deposits can be traced by the refraction seismic method.

4.4.3.2 Borehole and Subsurface Methods

Surface geophysics has its subsurface counterpart in subsurface applications within the environment that is being searched. One of the important applications here is the in situ determination of physical property contrasts such as density, magnetic, electrical and electromagnetic, seismic and radiometric properties.

Gravity: In a compatible geological environment, underground gravity surveys can be quite successful (Sumner and Schnepfe, 1967). At Bisbee, AZ, the success ratio was about 0.8, using 14 diamond drills in the search for massive sulfides during the last years in the life of the mine.

Borehole gravity surveying is coming of age, and there are two or three instruments and as many contracting groups in the service industry. The borehole gravimeter is primarily a density-logging device, but it also "sees" anomalous masses near the borehole that can be overlooked in surface exploration. LaFehr (1983) has summarized borehole gravity exploration, applications, and interpretation in a succinct manner.

Borehole Electric Logging: Surface and underground electrical exploration methods have counterparts in borehole logging. However, there are major differences from the more well-known petroleum and water well logging systems. In mineral exploration, the holes are shallow in depth and smaller in diameter. Site access is often difficult, hole blockage is common, and proper equipment is not always available.

The types of logging that has been reported are

1. Resistivity logs.
2. Self-potential logs.
3. Induced polarization, resistivity, self-potential in combination, (a) single hole, multiple electrodes, (b) azimuth and radial surveys, (c) hole-to-hole tomography.
4. Magnetic vector and susceptibility logs.
6. Mise a la masse surveying.
7. Electromagnetic logging: (a) receiver coil down-hole, transmitter coil on surface; (b) both receiver and transmitter coils down-hole; (c) receiver and transmitter coils in adjacent holes; (d) down-hole Turam and gradient array surveys.

Uses of logs in mineral exploration are for physical-property measurements and for mineral exploration. To enlarge the sphere of detection, two or more holes can be used to separate the transmitting and receiving systems, in a method referred to as "tomography." The goal of tomographic surveys is to reconstitute the geologic section between drillholes by using all depths in each hole for transmitter and receiver.

Another means of gaining a larger sphere of detection from a drillhole is to insert current from a bottom hole electrode and to measure the electric potential along surface traverses radiating from the hole collar. This has been called the "azimuth array" or the "radial array" (Sumner, 1976).

4.4.4 COST OF GEOPHYSICAL WORK

The cost of performing geophysical surveys will vary over a wide range, depending on access, terrain, and the amount of detailed information required. Approximate relative costs per square mile (square kilometer) in units of 1980 US dollars are given in the tabulation for initial reconnaissance coverage of a "typical" area in Table 4.4.3.

Surveys in mining exploration are planned to suit the problem at hand and are not always used in a reconnaissance fashion.

Table 4.4.3. Costs of Geophysical Surveys

Method	Cost (in 1980 US$) per mi^2	per km^2
Radiometric	100- 200	50-100
Magnetic (ground)	100- 200	50-100
Gravity	200- 300	100-150
Electromagnetics	500-1000	250-500
Resistivity/induced polarization	700-1200	300-600

Table 4.4.4. Costs of Borehole Geophysical Surveys

Method	Cost (in 1980 US$) per ft	per m
Electromagnetic logging, discrete readings	1.00-1.50	3.00-5.00
Induced polarization, resistivity, discrete readings	1.00-2.00	3.00-5.00
Magnetic susceptibility, core log or cuttings	0.10-0.23	0.30-0.60
Radioactive logging, core or cuttings	0.10-0.25	0.30-0.65
Radioactive logging, downhole	0.50-1.00	1.50-3.00

Shallow seismic refraction surveys should be of a similar cost as resistivity and induced-polarization surveys. Reflection seismic surveys may require more elaborate equipment, especially for deep work. Costs of $150,000 per crew-month for reflection seismic surveys are not unusual.

Aeromagnetic surveys and aerial radiometric surveys are flown at costs of $20 to $30/line-mi ($15 to $20/line-km). Airborne EM surveys are almost always combined with aeromagnetic surveys at costs of at least $35 to $50/line-mi ($20 to $35/line-km). Helicopter costs would be much higher, especially for smaller surveys.

Drillhole logging costs vary with mobilization costs, hole depth, site access, condition of site, drill on or off site, hole condition, diameter, hole fluid, background physical properties, ambient noise, hole orientation relative to vertical cultural effects, facilities at hole, temperature and weather, method, contractor, type of accessory equipment, and number of holes to be logged. With careful planning, a reliable contractor and adequate equipment the data in Table 4.4.4 should apply.

Another approach to cost estimation is to allow $500 per day for operator and equipment, adding the cost of laborers and mobilization. On a per line-mile (line-kilometer) or per day basis, costs should be less but may be approximately of the same order of magnitude.

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Chapter 4.5
GEOCHEMICAL AND OTHER PROSPECTING TECHNIQUES

J. ALAN COOPE

4.5.1 GEOCHEMICAL SURVEYS

4.5.1.1 Definition

Exploration geochemistry, or geochemical prospecting, includes any method of mineral exploration based on the systematic measurement of one or more chemical or chemically influenced properties of a naturally occurring material. The property measured is most commonly the trace concentration of some chemical element or group of elements. It may also include molecular and isotopic compositions and bacterial counts. The naturally occurring material may be rock, soil, stream sediment, glacial sediment, surface water, groundwater, vegetation, microorganisms, animal tissues, particulates, or gases including air.

4.5.1.2 Geochemical Cycle

Chemical elements, including the ore metals, are unevenly dispersed throughout the lithosphere and are continuously being cycled and redistributed under the influence of the earth's dynamic geological processes. The *geochemical cycle* (Fig. 4.5.1) diagrammatically illustrates the complex physiochemical changes and the variety of pathways that earth materials and their contained elements follow in response to these processes. It conveniently defines the major *deep-seated* and *surficial* geochemical environments.

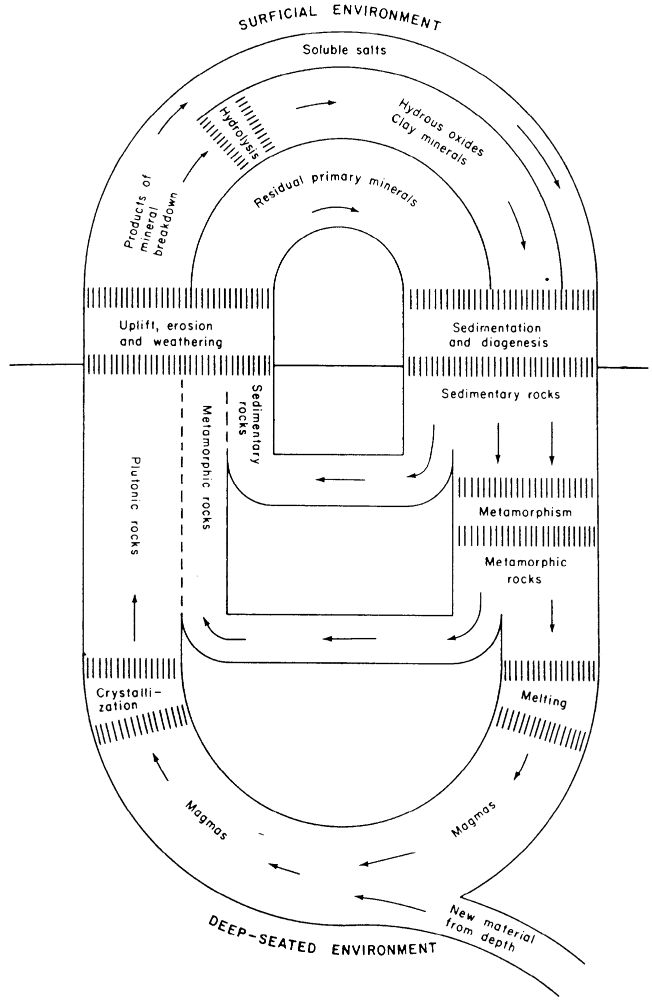


Fig. 4.5.1 Geochemical cycle (Rose et al., 1979).

4.5.1.3 History of Geochemical Prospecting

It is clear from historical records that the principles of geochemical exploration have been applied in prospecting over several thousands of years (Chapter 1.1). The prospector who panned for gold and traced the colors upstream to a bedrock source used mineralogical observations in a similar way to the modern geochemist, who utilizes sensitive chemical analyses to outline patterns of dispersion in the surficial environment. Boyle (1987) describes the working of placers for gold in the Preclassical Period. Geobotanical indicators were recognized as early as the 8th and 9th centuries (Boyle, 1967). The mid-16th century works of Biringuccio and Agricola describe the analysis of natural waters, springs, and their residues (Hawkes, 1957; Boyle, 1967).

Levinson (1974) records that modern methods of exploration geochemistry were first used in the early 1930s in the Soviet Union. Shortly thereafter, the methods were applied in the Scandinavian countries, particularly Sweden.

In North America, the earliest geochemical surveys were carried out between 1938 and 1940 by Lundberg in Newfoundland and in 1944 by Warren in British Columbia (Brummer et al., 1987). The first comprehensive geochemical exploration studies commenced at the US Geological Survey under the leadership of H. E. Hawkes in 1947 and at the Geological Survey of Canada with R. W. Boyle in the early 1950s. The Applied Geochemical Research Group was established at the Imperial College of Science and Technology in London in 1954 under the direction of J. S. Webb, and in France, research related to exploration geochemistry began in 1955.

The successful application and adaptation of geochemical exploration techniques in all parts of the world, the rapid development of analytical and computer technology, and improvements in field transportation have made geochemistry one of the more effective and widely applied exploration disciplines. Analytical capability is such that relatively rapid, sensitive analysis can be achieved for virtually all metals of economic interest. Continuing research is expanding our established capability to cost effectively detect and interpret dispersion patterns related to mineral deposits in a wide variety of often complex environments.

Through multielement analysis, geochemical data can reveal multiparameter signatures related to distinct geological units and processes. This capability, when applied to rock samples, permits geological correlation as well as the more precise delineation of otherwise invisible alteration features related to mineralization. When applied to soil and other types of samples, multielement data can help outline major geological units and identify

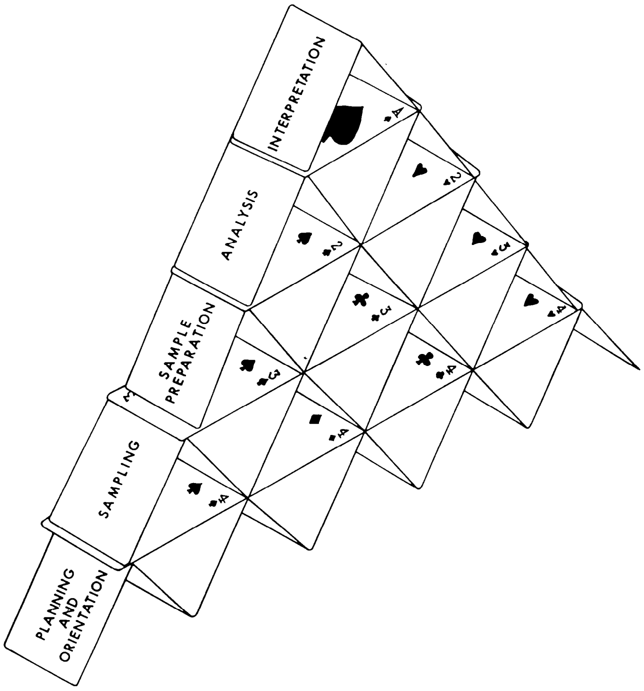


Fig. 4.5.2. The stages of exploration geochemistry related to a "house of cards" (modified after Lavin et al., 1986).

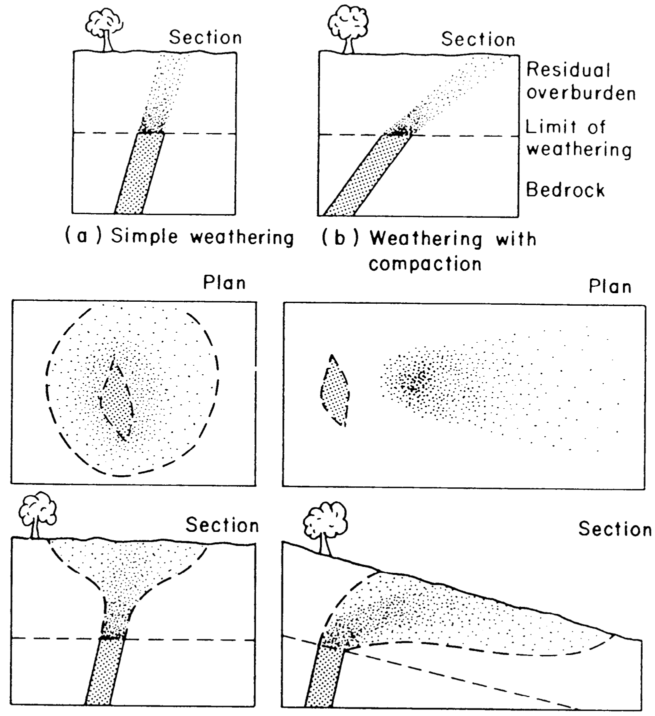


Fig. 4.5.3. Syngenetic (clastic) patterns in residual overburden (Rose et al., 1979).

the presence of mineralization buried under extensive cover (Hoffman, 1987).

4.5.1.4 Geochemical Prospecting Methods

The various components of a geochemical exploration program can best be understood by reference to the "house of cards" (Fig. 4.5.2). This model clearly indicates the importance and interdependence of the various phases of a geochemical survey. Good design and planning (including orientation) and well-conducted sampling programs constitute the foundation of all sound geochemical exploration work (Closs and Nichol, 1989). Inappropriate sample preparation prior to analysis can destroy the integrity of the well-chosen sample. It follows that no matter how accurate and precise the techniques used for analysis or how advanced the statistical treatments and computer programs used for data handling and interpretation, this cannot resurrect the lost quality and representativity and restore the decreased probability of exploration success caused by poor planning, improper field sampling, or faulty sample preparation.

It is convenient to categorize geochemical prospecting methods into those utilizing sample media from (1) the surficial environment and (2) the deep-seated environment.

4.5.1.5 The Surficial Environment

Reference to the geochemical cycle (Fig. 4.5.1) indicates that the surficial environment is characterized by the processes of weathering, erosion, transportation, sedimentation, and diagenesis. In geochemical exploration practice, the overwhelming majority of samples are collected from media in this environment.

Experience has confirmed that the success of a geochemical survey in the surficial environment depends largely on the correct assessment of all the natural factors that influence the mobility and dispersion of the metallic elements of interest. These natural factors, which are described more fully in textbooks by Rose et al. (1978) and Levinson (1974; 1980), are (1) the physical and chemical properties of the elements or parameters of interest, (2) the nature of the geology and mineralization, (3) the geomorphological history of the field area, (4) the vegetation, and (5) the topographic and climatic conditions.

These factors are completely interrelated and any change in one condition—such as a geologic change from an acidic to a basic environment, a local climatic change caused by elevation or a topographic change from rolling to mountainous terrain-can significantly influence mobilities and the extent of the dispersion from any mineralization type. Consequently, procedures for sampling, sample preparation, and analysis that proved to be satisfactory in one field area may be inadequate and unreliable in an adjacent region.

The processes of oxidation, weathering, erosion, transportation, sedimentation, and diagenesis that characterize the surficial environment are much too complex to describe in detail in this summary. The physiochemical conditions determine whether the weathering products of mineralization disperse either in solution in a *hydromorphic* form or in a solid or *clastic* form. The low pH, high Eh conditions of the acid weathering environment (e.g., oxidizing massive sulfides) promotes the solubilization of many base metal and other elements and their more widespread hydromorphic dispersion in surface and ground waters. Alkaline conditions developed over a weathering limestone or in a semi-arid climatic condition prohibit or arrest this hydromorphic dispersion. In these conditions, the insoluble metal particles disperse mechanically to form clastic patterns. Figs. 4.5.3 through 4.5.6, (Rose et al., 1979) illustrate how clastic and hydromorphic dis-

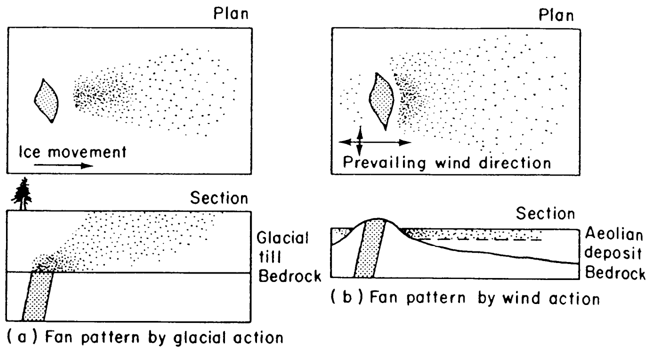


Fig. 4.5.4. Syngenetic (clastic) patterns in transported overburden (Rose et al., 1979).

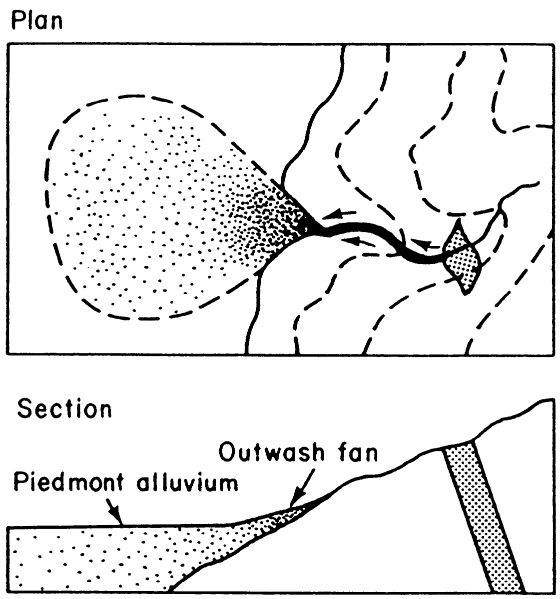


Fig. 4.5.5. Syngenetic (clastic) pattern in outwash fan and piedmont sheetwash alluvium (Rose et al., 1979).

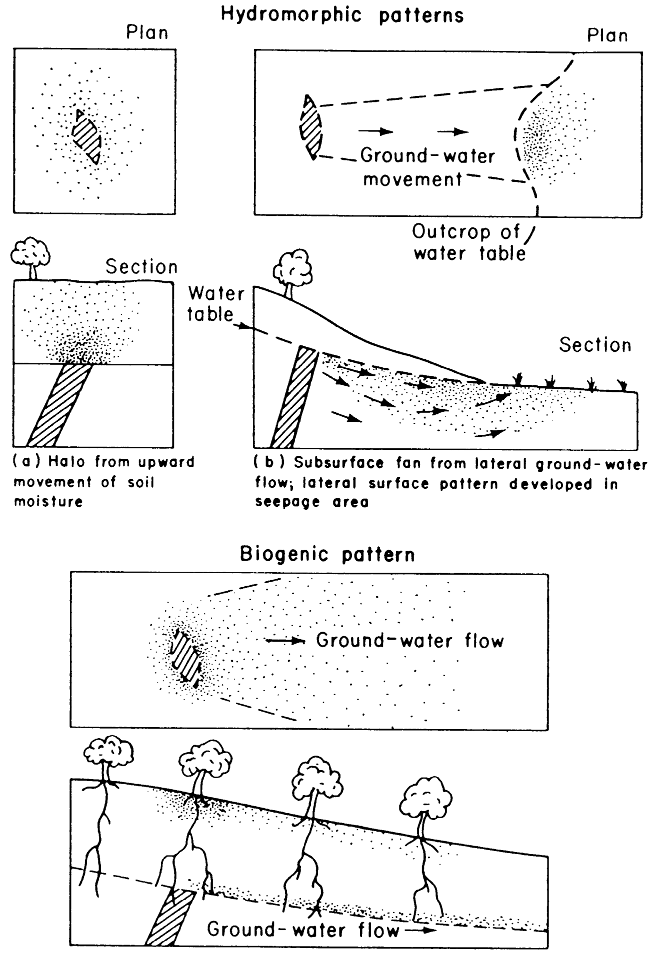


Fig. 4.5.6. Epigenetic patterns in transported overburden. Similar dispersion may also contribute to patterns in residual overburden (Rose et al., 1979).

persion can form syngenetic and epigenetic patterns in residual and transported overburden as well as providing one example of *biogenic* dispersion through the activity of vegetation. In some situations, *gaseous* dispersion can occur (e.g., radon escaping from a uranium deposit or SO, and other sulfur gases released during active oxidation of sulfides).

A general guide to the relative mobility of the common metallic elements under a variety of environmental conditions is shown in Table 4.5.1. A series of publications sponsored by the Association of Exploration Geochemists develops conceptual models based on conditions in Canada (Bradshaw, 1975), Scandinavia (Kauranne, 1976), the Basin and Range (Lovering and McCarthy, 1978), and Australia (Butt and Smith, 1980) that describe the principles and mechanisms of formation and configuration of anomalies and dispersion patterns revealed through geochemical exploration surveys in these regions.

The Orientation Survey: When contemplating a geochemical survey in a new region, the most reliable method of determining the extent and nature of dispersion patterns is to conduct an *orientation survey.* The objective of orientation sampling is to determine and outline the existence and characteristics of dispersion patterns or anomalies associated with mineralization and also background levels in similar environmental conditions. The specific sample media used are dependent on a knowledge of the field area, the prospecting problem, and, if available, previous experience, but may include any of the following: soils, stream sediments, surface waters, groundwaters, glacial sediments, lake sediments, rocks, vegetation or mull, gases or air, particulates, animal tissues, microorganisms.

The orientation survey commonly involves the collection of a number of relatively closely spaced samples over and in the vicinity of known, but preferably undisturbed, mineralization with the express purpose of outlining the dispersion patterns in the available sampling media. This information can then be used to select the most reliable sampling, sample preparation, and analytical techniques capable of detecting similar anomalies under similar environmental conditions. The detailed examination of the nature and shape of the dispersion patterns invariably yields information on the principal natural factors responsible for the observed distribution of anomalies. This is a significant aid in the development of interpretation procedures.

The accompanying table (Table 4.5.2) is a synthesis of the important parameters that can be derived from a properly planned and executed orientation survey. Based on this information, the optimum physical parameters (e.g., sample depth, sam-

Table 4.5.1. Relative Mobilities of the Elements in the Surficial Environment

Relative mobilities	Environmental conditions			
	Oxidizing	Acid	Neutral to alkaline	Reducing
Very high	Cl,I,Br S,B	Cl,I,Br S,B	Cl,I,Br S,B Mo,V,U,Se,Re	Cl,I,Br
High	Mo,V,U,Se,Re Ca,Na,Mg,F,Sr,Ra Zn	Mo,V,U,Se,Re Ca,Na,Mg,F,Sr,Ra Zn Cu,Co,Ni,Hg,Ag,Au	Ca,Na,Mg,F,Sr,Ra	Ca,Na,Mg,F,Sr,Ra
Medium	Cu,Co,Ni,Hg,Ag,Au As,Cd	As,Cd	As,Cd	
Low	Si,P,K Pb,Li,Rb,Ba,Be Bi,Sb,Ge,Cs,Tl	Si,P,K Pb,Li,Rb,Ba,Be Bi,Sb,Ge,Cs,Tl Fe,Mn	Si,P,K Pb,Li,Rb,Ba,Be Bi,Sb,Ge,Cs,Tl Fe,Mn	Si,P,K Fe,Mn
very low to immobile	Fe, Mn Al,Ti,Sn,Te,W Nb,Ta,Pt,Cr,Zr Th,Rare Earths	Al,Ti,Sn,Te,W Nb,Ta,Pt,Cr,Zr Th,Rare Earths	Al,Ti,Sn,Te,W Nb,Ta,Pt,Cr,Zr Th,Rare Earths Zn Cu,Co,Ni,Hg,Ag,Au	Al,Ti,Sn,Te,W Nb,Ta,Pt,Cr,Zr Th,Rare Earths S,B Mo,V,U,Se,Re Zn Co,Cu,Ni,Hg,Ag,Au As,Cd Pb,Li,Rb,Ba,Be Bi,Sb,Ge,Cs,Tl

Source: Modified after Andrews-Jones, 1968.

ple interval, analytical technique, etc.) for routine surveying can be chosen that necessarily take into account the defined dispersion characteristics as well as the physical, logistical, and economic conditions pertaining to the project. Large samples should be taken to provide sufficient material for the full evaluation of the parameters listed in Table 4.5.2 and representative samples should be collected from nonmineralized areas to adequately define background conditions. Recommended procedures for orientation sampling, sample preparation, and analysis are outlined by Bradshaw (1975).

A survey conducted over gold-silver vein mineralization at Mt. Nansen in the Yukon Territory (Coope, 1966) illustrates the application of the orientation approach. Detailed profile sampling of soil and overburden exposed in a trench across virgin mineralization produced patterns of lead, antimony, and zinc as illustrated in Fig. 4.5.7. It is apparent from these patterns that the dispersion behavior of lead and antimony is similar but quite contrasting with the patterns for zinc. All metals have been influenced by downslope movement in the overburden, but the zinc pattern is compatible with dispersion in solution along the bedrock surface. Examination of the patterns also indicates that near-surface sampling (0 to 12 in., or 0 to 300 mm) would not reliably indicate the mineralized vein. It was concluded that a sample depth of 18 to 24 in. (460 to 610 mm) and a sample interval of one-half the anomaly width at this depth (15 ft or 5 m) were the optimum parameters necessary for routine surveying (Fig. 4.5.8). Observation of these sampling criteria led to the discovery of several, previously unknown veins.

The soil profile interval in an orientation survey over more extensive types of mineralization will be proportionately larger than the intervals chosen for narrow-vein type mineralization.

A stream sediment orientation survey can define the appropriate sample interval for reconnaissance surveying. In the example illustrated in Fig. 4.5.9, closely spaced samples were collected

Table 4.5.2. Parameters to be Derived from a Properly Planned Orientation Survey

Rock	Soil	Sediment	Water	Biogeochemical	Gas
Shape, extent, reproducibility, homogeneity, and controls of geochemical patterns (Rock to Gas)					
Most suitable indicator elements and best analytical methods (Rock to Gas)					
Reproducibility of sampling and analysis (Rock to Gas)					
Sampling pattern and density (Rock to Gas)					
Amount of sample required (Rock to Gas)					
Required field observations (Rock to Gas)					
Possible contamination sources (Rock to Gas)					
	Effects of topography, drainage, vegetation, geology (Soil to Gas)				
	Seasonal or temporal variations (Soil to Gas)				
	Optimum sampling depth or horizon (Soil and Sediment)				
	Optimum sampling fraction (size, heavy minerals, organic fraction) (Soil and Sediment)				
Sample material (rock,vein,etc)	Nature of overburden (Residual vs transported) (Mechanism and direction)	Correlation with Fe-Mn oxides	Optimum material (ground or surface)	Optimum species (distribution, ease of anomaly recognition, ease of sampling, and depth of root system)	Effect of soil type Effect of soil depth
Type of sample (grab, channel, etc)	(Soil profile development) (Depth variation of indicator elements)	Correlation with organic matter	Control of water flow Difference in aquifers		Effect of soil moisture
Effects of weathering					
Effects of alteration			Relation to recharge areas	Part of plant to be sampled	
Applicability of mineral separations (sulfides, biotite, limonite, calcite, etc)			Variation with depth	Effect of aspect (sunlight, shading) (Biogeochemical and Gas)	
			Variation with type of lake		

Source: Modified after Coker, 1989.

both upstream and downstream of known molybdenum mineralization. With increasing distance from the deposit, the sample interval was progressively increased. Subsequent analytical results indicated the background level for molybdenum in the –80-mesh stream sediment fraction was 2 to 3 ppm Mo and that anomalous values extend downstream in the tributary sediments for approximately 1.2 mi (2.0 km) prior to being diluted by the background sediments in the main stream. In this area, therefore, sampling along the drainages at half this anomalous distance (0.6 mi or 1 km) with mandatory collection in each drainage immediately upstream of each tributary junction will satisfactorily indicate the presence of similar, or more extensive, molybdenum occurrences in a routine reconnaissance survey.

The sample intervals employed in stream sediment orientation surveys will vary according to the anticipated size of the target mineralization and the dispersion characteristics of the metals of interest. In areas where no previous experience exists, a short interval of 150 ft (50 m) over an initial downstream distance of 1000 ft (330 m) is recommended. This interval should then be progressively expanded with distance from the metal source to the limits of the known or anticipated dispersion pattern. Samples must also be collected from nonmineralized areas to establish the background range and sufficient material should be collected at each site to allow for the determination of optimum size fractions, analytical techniques, and other factors listed in Table 4.5.2.

Sample Collection and Handling: Effective sampling of all surficial media requires well-trained personnel capable of recognizing and describing the correct sample material and the sample site characteristics. Samplers should be able to recognize and, if possible, avoid situations where contamination from human activity or changes in the natural physiochemical conditions can

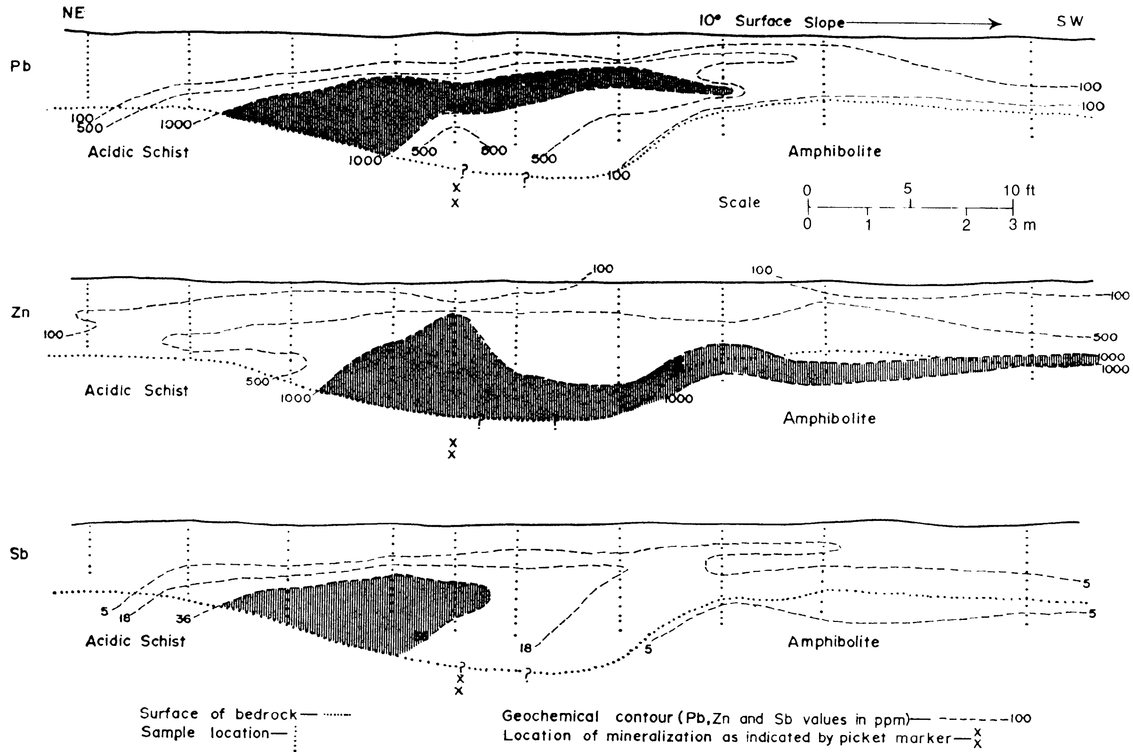


Fig. 4.5.7. Profile distribution of lead, zinc, and antimony over mineralized vein. Mount Nansen area, Yukon Territory (Coope, 1966).

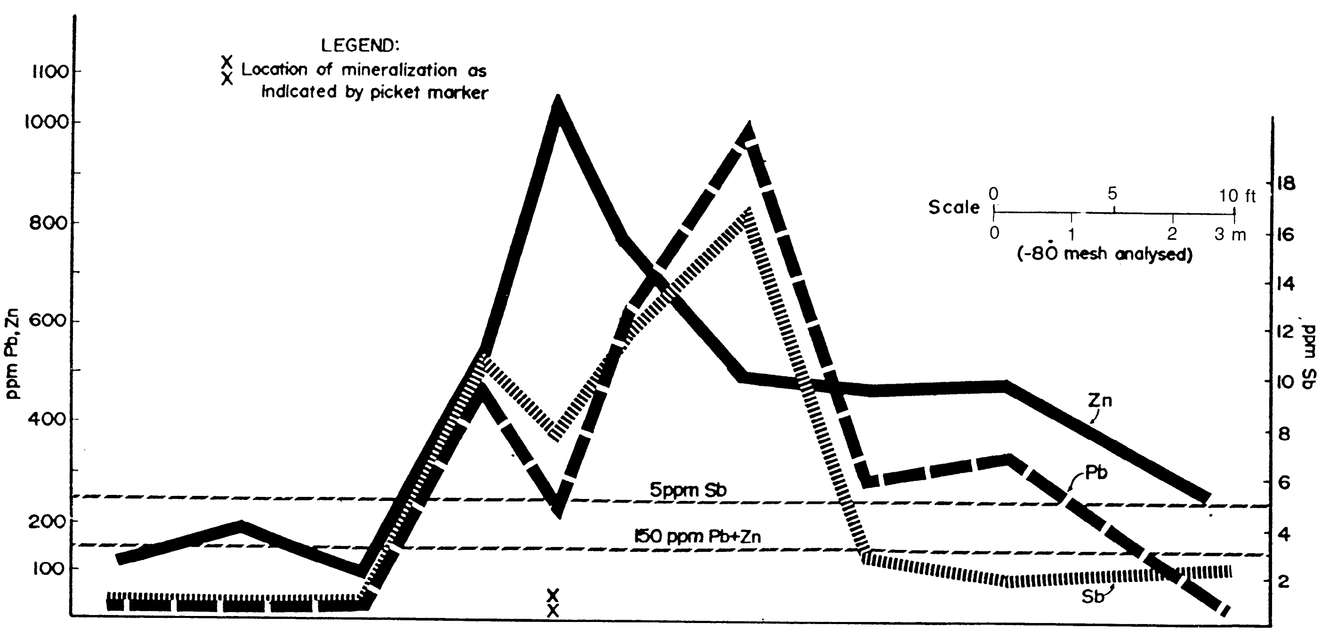


Fig. 4.5.8. Graphical representation of lead, zinc, and antimony over mineralized vein at 18- to 24-in. (460- to 610-mm) depth, Mount Nansen, Yukon Territory (Coope, 1966).

produce spurious or unusual results. In most situations, these sampling duties can be undertaken by trained technical personnel under the supervision of a geochemist or geologist with adequate geochemical exploration experience. In some surveys (e.g., where identification of the correct sample material is critical, as in biogeochemical or glacial till sampling programs), it is prudent

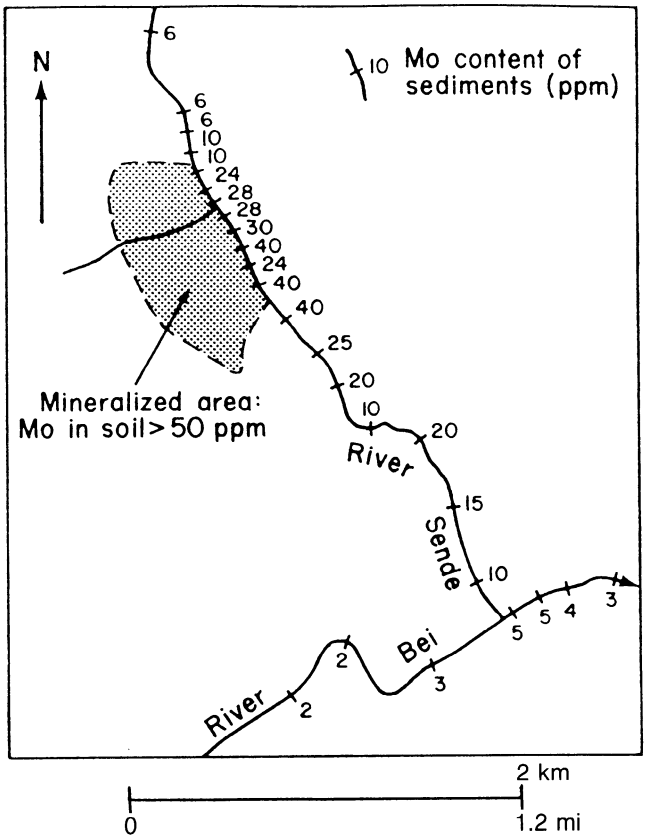


Fig. 4.5.9. Stream sediment survey of River Sende, Sierra Leone. Data on 80-mesh fraction (Rose et al., 1979).

to employ qualified specialists (e.g., botanists and Quaternary geologists) to conduct orientation surveys and instruct and supervise the sampling teams.

Sampling tools vary according to the medium and the field situation. Noncontaminating equipment is essential and care should be exercised in not only choosing noncontaminating steels for shovels, trowels, augers, etc., but in ensuring that lubricants and adhesives, weldings, and solders are metal-free too. This awareness of geochemical cleanliness extends to the dress of the sampler who should avoid wearing metal buckles, rings, etc., and handling coins which might lead to contamination by chipping or transfer of metal on fingers.

The same caution is necessary in the choice of sample containers. Kraft paper (with noncontaminating waterproof glue and closures), olefin, and plastic bag containers of appropriate size are frequently used. Kraft and olefin allow samples to be dried without transfer. Plastic bags are commonly used for larger samples. More rigid polypropylene and special glass bottles can be utilized in water sampling and a variety of sampling devices, many of them patented, are available for the sampling of gases and particulates.

Sample Media:

Soils—Soils vary considerably in composition and appearance according to their genetic, climatic, and geographic environment. Classified into residual and transported types according to their relationship to their substrate, soils are mixtures of mineral and biologic matter and may be distinctively differentiated into a series of soil horizons. Residual soils characteristically contain detectable dispersion patterns developed during the weathering of mineralization in the underlying bedrock, and these patterns are revealed by careful sampling of appropriate soil horizons. Transported soils present more difficult sampling problems, but meaningful sampling is possible in many areas once the genetic origins of the transported cover are understood.

With all but a few exceptions, soils are sampled along traverses or grids in the follow-up or detailed prospecting stage of a geochemical program. Orientation programs define criteria such as sample depth or soil horizon to be sampled, sample interval, and the size fraction for analysis. It is essential that these criteria be observed resolutely through the survey.

Stream Sediments and Waters—Stream sediment is one of the more commonly used media for geochemical reconnaissance surveys. The sediment at any point in a stream is a natural composite sample of erosional materials from upstream in the drainage basin and will include clastically, hydromorphically, and biogenically derived products from weathering mineralization. The length of anomalous dispersion trains will vary with the nature of the mineralization, source, and the physiochemical environment of the field area or drainage basin. In humid, actively oxidizing environments, dispersion trains from sulfide-rich base metal deposits may extend downstream for several miles.

Active stream sediment—that material constantly or most frequently washed by stream waters—is collected from the center of a drainage avoiding sites that may be contaminated or influenced by bank collapse. In most survey programs, approximately 500 to 1000 g (1.1 to 2.2 lb) of fine-grained material is collected from the upper few inches (tens of millimeters) of the sediment. Larger samples from deeper in the stream bed are collected from carefully selected sites if heavy minerals are to be examined. In all surveys in new areas, the critical parameters of sample interval, sediment size fraction, appropriate analytical procedures, significant anomaly contrasts, and background levels are determined through orientation surveys.

In the regional reconnaissance prospecting mode, stream sediment surveys can be designed to systematically cover areas up to several hundred square miles. Anomalous indications of mineralization can be followed up with more detailed sediment sampling to pinpoint source areas in more detail. Follow-up sampling of seepage areas is particularly effective in delimiting anomalous groundwater sources containing metal derived from oxidizing mineralization. If appropriate, soil sampling can be used to define suboutcropping mineralization in the anomalous source areas defined by the sediment survey.

Regional reconnaissance can also be achieved by sampling the waters of actively flowing streams where metal is dispersing in solution. A prospecting approach similar to the sampling of stream sediments is necessary, collecting waters in clean 500 or l000-mL (0.13 to 0.26-gal) polypropylene or glass bottles. Sampling of groundwater seepage sites is an integral part of stream water surveys. A few milliliters of acid is routinely added to the water samples after collection to keep the dissolved metals from adhering to the bottle walls. pH, conductivity and certain other measurements are commonly made at the sample site.

Lake Sediments and Waters—Lake sediments and lake water sampling have been developed into effective geochemical reconnaissance techniques in areas of Canada and Scandinavia where lakes are common, conditions are swampy, and/or where stream drainages are inaccessible or poorly developed. In low-relief regions, the lake sediment medium is dependent on the hydromorphic dispersion of metals into the lake environment through groundwaters and the adsorption of this metal onto hydrous oxides and the organic-rich muds being deposited on the lake bottoms. The sampling focuses on the collection of these organic muds using specially designed sampling devices. In more mountainous areas, fine-grained clastic dispersion into the lake sediment becomes a more important factor. In all areas satisfactory sample locations are found well away from lake shores and are reached using boats, float planes, or helicopters. Lake waters are commonly collected at the same sites as the lake sediments.

The lake sediment technique has successfully indicated the presence of uranium mineralization in Saskatchewan and base metal mineralization in the Northwest Territories, the Appalachians, and British Columbia (Coker et al., 1979).

Glacial Deposits—Extensive Quaternary glacial deposits occurring over most of Canada and the northern United States, northern Europe, and northern Asia have presented major challenges to exploration. As a better understanding of the origin and formation of these glacial sediments has grown, their blanketing presence has become progressively less formidable and effective exploration techniques have been developed.

Mineralized boulder tracing in glaciated regions is an established technique of the traditional prospector in Scandinavia and parts of Canada. In Scandinavia, dogs have been trained to assist the prospector by sensing SO_2 released from oxidizing sulfide boulders at shallow depths below the surface. In Finland, methods were developed for sampling tills in the 1950s, and this technique is now the preferred sampling method in most Finnish geochemical exploration programs. Esker sampling and till sampling for distinctive heavy mineral suites have been used for kimberlite and diamond prospecting in the Canadian Shield.

Approximately 70% of lodgement till is locally derived, and most of the early success with till sampling was in areas of shallow till cover (less than 30 ft or 10 m) where the sample medium is reasonably accessible. In the 1960s, lightweight percussion drills such as the Pionjar and Cobra models, were adapted to collect small samples of till from immediately above the suboutcropping bedrock to geochemically categorize anomalous geophysical features at depths of up to 70 to 80 ft (21 to 24 m) (Gleeson et al.,1971). Overburden drilling technology, particularly reverse circulation and sonic drilling, advanced rapidly with the utilization of larger drills in programs for uranium and gold deposits in glaciated areas. Because most types of gold deposits are not detectable by conventional geophysical methods, lodgement till sampling using overburden drills to depths of 330 ft (100 m) has been routinely used in prospecting for gold in the Canadian Shield through the 1980s. Large samples of till (approximately 20 lb or 10 kg) are recovered in these programs from which the heavy mineral fraction is separated and examined both visually and chemically for gold and other metals. The correct interpretation of these data is dependent on an understanding of the till stratigraphy and provenance of the transported material. The technique is expensive, with combined drilling, sample treatment, and analytical costs ranging from \$20 to \$30/ft (\$66 to \$98/m), but it is cost effective in this deep overburden-covered environment where other exploration methods have not been as successful in gold exploration. At least four significant gold deposits have been discovered in Canada by these overburden drilling methods.

In contrast with the heavy mineral sampling of tills practiced in Canadian exploration, Scandinavian explorers place a greater reliance on the 63-μm (–240-mesh) fraction of till. This fraction has successfully indicated the presence of several types of mineralization including gold.

Rocks—Rock sampling or lithogeochemical surveys in the surficial environment comprise systematic sampling of outcrops, trenches, drill cores, or cuttings within the zone of oxidation and weathering.

As with other types of geochemical surveys, the sampling procedures and the sample material collected in lithogeochemical work should be standardized as much as possible. However, considering the large number of variables that can be introduced by the processes of weathering and oxidation, the ideal of collecting similarly weathered material is sometimes impractical. Nevertheless, the geologist or the geochemist conducting the survey should ensure that individual samples at all sample sites are essentially comparable and that observed variations in weathering intensity are properly recorded for interpretation purposes.

Lithogeochemical sampling necessarily must take into account the geological environment and the type of mineral deposit of interest to the explorer. Very briefly, *syngenetic lithogeochemical patterns* can develop, for example, (1) in intrusives genetically associated with specific mineral deposit types, or (2) in volcanic stratigraphy where exhalative activity has dispersed detectable quantities of metals during the formation of volcanogenic massive sulfide deposits, or (3) in the vicinity of sedimentary deposits. The scale of sampling necessary for detection of these and other types of syngenetic genetic patterns will be determined by orientation surveys but may include the regional sampling of individual plutons or more detailed sampling of specific parts of an exposed stratigraphic section.

Epigenetic lithogeochemical patterns can develop in diffusion aureoles in the rocks hosting epigenetic mineral deposits, or as leakage aureoles along fractures and other structures that mark the pathways followed by hydrothermal or groundwater fluids prior to and subsequent to the deposition of the significant mineral deposits. Both these processes introduce the concept of mineral zoning of geochemical aureoles. The ratios of the different elements introduced during the mineralization period varies with distance from the principal deposit in accordance with the properties of these elements and the host rocks and the physiochemical conditions at the time of deposition. Leakage anomalies can extend for hundreds of feet or meters from a deposit whereas diffusion anomalies rarely exceed 100 ft (30 m). Several texts—Rose et al., (1979), Levinson (1974, 1980), and Govett (1983)—provide more detailed discussion and case histories of lithogeochemical surveys. The scale of dispersion, the presence of zoning patterns, and the sampling parameters for epigenetic geochemical patterns are determined by appropriate lithogeochemical orientation surveys.

It is clear that surveys designed to detect leakage anomalies will focus on a standardized collection of fault or fracture zones and, possibly, bedding structures. In contrast, the preferred lithogeochemical sample material for the detection of diffusion haloes is likely to be unfractured and the scale of sampling much more detailed. In all instances, geochemical analysis of lithogeochemical material has the potential of delimiting dispersion patterns associated with mineralization beyond visible alteration.

Vegetation and Humus—Early scientific observers dating from the 8th and 9th centuries recorded that the morphology and distribution of certain plants were affected by the presence of metals in the soils. Such visible variations in a plant species are referred to as *geobotanical indicators*. Many other plants, while not showing any visible variations, are capable of concentrating metals in their tissues and the presence of anomalous metals in the soils or groundwaters is often reflected in the metal content of leaves, twigs, or other plant organs. These invisible metal concentrations are known as *biogeochemical indicators* (Brooks, 1972).

The seasonal fall of leaves and needles transfers some of the accumulated metals to the surface soil where they are incorporated in the humus layer. Sampling of this *humus* (alternatively known as *mull, Ao,* or *Ah* material) by Scandinavians in the 1930s revealed its potential for geochemical prospecting. This is especially true in areas of transported material where the root penetration of the plant exceeds the thickness of this cover and

Table 4.5.3. Seasonal Variation in the Gold Content of Ashed Alder Twigs. Ash Yield from Dry Twigs Generally About 2%

	Gold concentration, ppb				
	1984			1985	
Site	Early June	Early August	Mid September	Mid April	Mid June
1a	2	7	23	250	12
1b	53	6	17	47	<5
1c	58	9	20	130	7
2	34	6	15	166	13
3a	29	8	10	37	20
3b	35	7	11	34	7
4a	23	6	13	57	13
4b	25	8	13	41	15
5	25	11	20	27	—
6	8	20	14	20	—
7	29	20	23	75	—
8	35	10	22	58	—
9	23	8	14	51	—
10	12	17	18	33	—
11	24	10	11	53	—
12	25	11	12	42	—
13	14	11	9	66	—
14	21	10	38	48	—
Mean[a]	28 (0.56)	10 (0.2)	17 (0.34)	69 (1.38)	

[a]Values in parentheses have been recalculated to a dry weight basis.
Source: Dunn, 1987.

obtains nutrients from the underlying mineralized bedrock and groundwater.

In addition to the direct recognition of geobotanical indicators, the most attractive feature of vegetation sampling is the ability of biogeochemical and mull prospecting to "see through" thicknesses of barren transported overburden. Consequently, these techniques have been applied with varying success in glaciated regions of North America, Europe, and Asia and in areas like the southwestern United States where pediment, colluvial, and alluvial cover is extensive (Boyle et al., 1969; Chaffee, 1977).

Plants are complex organisms and so is their metabolism. Different species respond differently to the same conditions and, consequently, some species are more effective biogeochemical indicators than others. Deep-rooted plants (e.g., the mesquite) are much more effective prospectors of the deeper groundwaters than the shallow-rooted flora of the southwestern US deserts and are, therefore, preferred species in biogeochemical work. Nevertheless, shallow-rooted plants growing in transported cover may reveal meaningful patterns in some desert regions. Evapotranspiration has been suggested as a mechanism for movement of metals into the nutrient depth of these plants.

Some species preferentially concentrate metals in specific tissues such as leaves, twigs, bark, or wood. It is therefore very important to establish the most favorable tissues for sampling once a useful species has been identified. This complexity is accentuated by the fact that metal uptake may vary with aspect and degree of uptake commonly varies with the seasons (Table 4.5.3). In temperate forest regions, accelerated uptake and higher concentration commonly occurs during the spring growth following a dormant winter season. In hot desert regions, following the exhaustion of available near-surface water during the dry season, deep-rooted plants will tap the deeper, more metal-rich groundwaters.

These variables make biogeochemical sampling a very specialized exercise. The expertise of an experienced geochemist or botanist is essential during orientation studies and the supervision of vegetation surveys. Because of the seasonal variations, biogeochemical surveys must be completed quickly, and sampling in the spring and fall is generally avoided.

The same complexities do not affect the humus or mull medium. Dead tissues are not subject to seasonal variations and annual accumulation has an integrating effect. Weathering, leaching, and bacterial decomposition will work to diminish the metal contents, but signatures in mull are generally preserved. Dunn (1987) has pointed out that, with some species, sampling of bark (a dead tissue) can be an effective technique that is not subject to seasonal metabolic variations.

Animal Tissues—Animal tissues have not been used extensively as a geochemical medium. Warren et al. (1971) analyzed 96 trout livers from locations in British Columbia and identified a general correlation between the zinc and copper contents of these livers and known mineral regions. As part of an environmental monitoring program, the government of Ontario has sampled fish tissues for their mercury content. Recent work has investigated the use of the trace element content of bee pollen as an exploration tool. Variations in concentrations were noted showing a general relationship with known mineralization, but follow-up is complicated by the territorial wandering of the insects.

Microorganisms—Parduhn and Watterson (1984) and Parduhn et al. (1985) have demonstrated that the population of the common microorganism, *Bacillus cereus*, increases with natural increases in the base and precious metal content of soils in the vicinity of known mineral deposits. It has also been noted that the increased antibiotic resistance of these bacteria correlates with increased metal concentrations in soils (Watterson et al., 1986).

Gases and Air—Under certain conditions, mineral deposits produce gaseous emanations that can be detected by specialized measurements. Radon, for example, is produced during the radioactive decay of uranium and radium. Survey techniques measuring the alpha particle emissions during radon decay have been used extensively in the search for uranium. Helium is another gas produced during radiogenic decay and is considered by many to be also of deep-seated origin. Mercury-bearing minerals, which can include sphalerite and other sulfides, often release mercury vapor during oxidation. This vapor can be measured in soil gas directly or adsorbed mercury can be released from conventional soil samples by heating to 210° to 390°F (100° to 200°C) (Landa, 1978).

The oxidation of sulfides leads to the generation of SO_2, H_2S, and CO_2. Because of the consumption of oxygen in the oxidation process, the atmospheric proportions of CO_2:O_2 change in the vicinity of oxidizing sulfides and these imbalances can be measured in the soil gas.

Biogenic activity in the soil can produce CH_4, H_2, H_2S, and CO_2. Therefore, near-surface detection of these gases can be suspect. Methane is found at depth in several types of mines, but a genetic association with mineralization is not always clear. Recent studies in Sweden (Malmqvist et al., 1986) have identified an upward fluxing of GEOGAS in the near-surface sections of the earth. This GEOGAS enters the groundwaters as dissolved air. The circulating meteoric waters and changing pressure conditions in the subsurface cause the GEOGAS to rise as small streaming bubbles. The bubbles contain other gases and can also collect metallic ions and particles that can be trapped in collectors set out in the surface soil. Anomalous patterns in GEOGAS have been recorded over mineralization buried under thicknesses up to 100 ft (30 m) of transported cover.

Particulates—Solid particles down to the size range of large molecules are present in the atmosphere. Weiss (1971) developed an airborne geochemical prospecting technique in arid terrain for the collection of dust particles suspended in the atmosphere. The AIRTRACE and SURTRACE techniques of Barringer Research Ltd. have been designed to sample a variety of particulates in the lower atmosphere including spores, pollen, dust, microorganisms, organometallics, and hydrocarbon complexes. The collected particulates are analyzed using a laser pulse/inductively coupled plasma spectrometry (ICP) procedure in the laboratory. The same airborne platform can be equipped with a mercury sensor for real-time measurements. AIRTRACE has been successful in sensing gas and oil resources and has produced positive results over mineral deposits. Reproducibility of the technique in the search for mineral deposits is hampered by variable weather conditions and temperature inversions. The SURTRACE method is designed to overcome these climatic problems by sampling particulate matter from the surface microlayer of the ground using a helicopter-based or manually transported system.

Sample Preparation: Inappropriate sample preparation can completely nullify the careful work of the sampler who has invested time and expertise in the selection and collection of representative material. Furthermore, it is impossible to restore the integrity of the poorly prepared sample by enterprising analytical treatments and interpretational procedures.

Drying—Prior to mechanical treatment, nonlithified clastic and rock geochemical samples have to be dried. This can be achieved by exposure to the sun in some climates, but most samples are dried in drying ovens. Heating these ovens to temperatures in excess of 160°F (70°C) can lead to the loss of volatile elements that may be of value to the exploration program. Organic samples, including humus, may be dried in a conventional drying oven or in a microwave oven at temperatures not to exceed 160°F (70°C).

Sieving and Crushing—After drying, nonlithified clastic samples should be agitated and pounded to achieve complete separation of component particles without crushing. This can be done with a pestle and mortar or other suitable noncontaminating mechanical device that can be thoroughly cleaned between samples. In earlier geochemical prospecting work, orientation studies on soils, stream sediments, and other clastic materials revealed that the separation of — 80-mesh material for analysis was appropriate in many surveys. With some elements, such as those commonly concentrated in residual minerals and other elements dispersed in weathering products in arid/semi-arid environments, fractions coarser than 80 mesh give superior geochemical patterns with better contrasts. Some surveys in glacial till environments benefit from the analysis of the –240 mesh (63-μm) or even finer fractions (Shelp and Nichol, 1987). Careful size fraction analysis during orientation will not only indicate the most appropriate fraction for routine work, but the metal distribution throughout the sample reveals invaluable information on metal behavior for interpretation purposes. Sieves must have a noncontaminating composition. Those most frequently used are made from stainless steel or nylon mesh.

Heavy Mineral Separations—Heavy mineral separations from stream sediment, glacial till, and rock samples commonly utilize the liquids tetrabromoethane (sp gr 2.9) and methylene iodide (sp gr 3.3). Solutions of sodium polytungstate, with specific gravities ranging from 1.0 to 3.1 according to dilution, can also be used. Usually these liquid separations are carried out on coarser sample size fractions (e.g., -30 + 80 mesh), but separations can be made on material down to 200 mesh if the metal of interest is present in the finer size fractions. If necessary, heavy mineral concentrates can be further subdivided by electromagnetic separation into magnetic, paramagnetic, and nonmagnetic fractions.

Vegetation Samples—Ideally, *vegetation* and *humus samples* should contain no clastic material. Dust on leaves and twigs can be removed by rinsing with demineralized water, but humus material is rarely 100% organic. The presence of excessive clastic material (1) may dilute or contaminate the metal content of the organic material, and (2) adversely interfere with lower detection limits possible on organic material using the neutron activation analytical technique.

After drying at temperatures of less than 160°F (70°C) in a standard drying or microwave oven, vegetation samples are macerated in a Wiley Laboratory Mill to a –2-mm mesh size. This material is compressed into either 8.0- or 30.0-g pellets for direct analysis by neutron activation or can be ashed in a muffle furnace at 842° to 878°F (450º to 470°C) for approximately 15 hr. The plant ash is then analyzed in a similar manner to clastic sample material.

Rock Samples—Some surveys call for the analysis of specific minerals in rock samples. Such minerals may include magnetite, biotites, feldspars, and sulfides that may be concentrated by magnetic separation or with heavy liquids from the most suitable size fraction, after appropriate sample treatment.

Most rock samples collected in geochemical programs are analyzed for their whole rock or trace element contents. This requires crushing and pulverizing and, if the contained metals of interest are heterogeneously distributed (e.g., as with coarse gold), the principles of sampling theory must be observed. The pulverized whole rock product that is commonly analyzed has a grain size of less than 150 to 200 mesh. The sample is rarely reduced to this fine grain size in its entirety, but, as a general rule, the finer the sample can be crushed or ground prior to sample splitting, the more representative the split is likely to be. The emphasis on gold exploration in recent years has focused attention on sample heterogeneity. This has stimulated some significant developments in the design and adaptation of crushing and grinding equipment for improved sample preparation. In addition to sample representativity, the geochemist must be concerned with sources of contamination in the sample preparation equipment. Grinding plates and blades of different composition are available so that the obvious contamination from chrome steel, tungsten carbide, and other materials can be avoided. Similarly, loss of sample representivity through "smearing" of native copper, native gold, molybdenite, or other soft material on pulverizer plates can be avoided in many instances by the selection of alternative equipment with a different comminuting action.

Since sample weights analyzed in geochemical work are small (0.1 to 50 g or 0.0002 to 0.1 lb) compared with the original sample size, sample homogeneity is very important. Improved mixing can be achieved, when necessary, by careful blending and the pulverization of coarser fractions to the –200-mesh size.

4.5.1.6 The Deep-Seated Environment

Geochemical sampling in the deep-seated environment as defined in Fig. 4.5.1 is, primarily, rock or lithogeochemical sampling beyond the influence of surficial processes. Sample material is necessarily obtained from deeper underground workings or from drill cores. Syngenetic and epigenetic patterns of primary dispersion (see discussion of rocks) are the targets of such sampling and the sample selection, analysis, and data interpretation is often directly related to the current understanding of the genesis of the mineral deposit type of interest.

Gases, particularly helium, and other fluids, circulate in the deep-seated environment. Several surveys measuring helium in the near-surface (Eremeev et al., 1973) have been interpreted as

Table 4.5.4. Comparison of Some Characteristics of Analytical Methods

Method[a]	Capital, $	Multielement	Determinations per day	Comments
Colorimetry	1×10^3	No	20-100	Very simple; adaptable to field
AAS	2×10^4	No	500	Easy to set up; several elements can be determined on same solution but not simultaneously
ICP	$1\text{-}3 \times 10^5$	Yes	> 2000	Needs skilled analyst to supervise; computer; sample in solution
XRF	$1\text{-}5 \times 10^5$	Yes	> 1000	Needs skilled analyst to supervise; computer; analyzes solid sample
INAA	$0.8\text{-}20 \times 10^6$	Yes	> 3500	Needs skilled analyst to supervise; computer; analyzes solid or liquid sample

[a]AAS = Atomic absorption spectrophotometry, UCO = Inductively coupled plasma spectrometry, XRF = X-ray fluorescence, INAA = Instrumental Neutron Activation Analysis.
Source: Modified after Fletcher, 1987.

Table 4.5.5. Variation in Extraction of Chromium from Different Lithologies with Five Digestion Procedures

Lithology	Chromium extracted, ppm				
	[a]A, %	B, %	C, %	D, %	E, %
Chromite	1100	930	600	140	nd
Pyroxenite	2900	1600	1550	480	nd
Chlorite schist	5800	1350	1400	390	nd
Serpentinite	2500	2200	2050	740	nd
Quartzite	270	115	42	18	nd

[a]A = Alkaline fusion; B = hot aqua regia; C = hot nitric acid; D = hot 0.5N hydrochloric acid; E = cold 0.25% EDTA; nd = not detected.
Source: Modified after Fletcher, 1987.

reflecting patterns related to the passage of the gas along deep-seated structural features in the earth's crust.

Widely spaced sampling of tills (one sample per 300 km^2 or 115 mi^2) in Finland has revealed patterns of elements believed related to deeper crustal features not immediately apparent from a study of the geological maps (Bjorklund, 1989). These Finnish surveys are illustrated in the segment 4.5.1.9. It is quite possible that these maps are highly significant in indicating how sampling of surficial materials can outline important structural patterns in the deeper crust. Such structural patterns very likely have influenced the migration of mineralized fluids and the crustal distribution of mineral deposits.

4.5.1.7 Analytical Techniques

A prime requirement for cost-effective geochemical exploration surveys is the availability of analytical procedures capable of high productivity, low detection limits, high precision, and acceptable accuracy. These criteria were met, in the 1950s, by a series of colorimetric analytical techniques with man-day productivities ranging from 20 to 100 samples per day. Technological developments have subsequently led to the introduction of atomic absorption spectrophotometry (AAS), inductively coupled plasma spectrometry (ICP), X-ray fluorescence (XRF), and instrumental neutron activation analysis (INAA) and a vastly increased productivity (Table 4.5.4). Along with the multielement capability of ICP, XRF, and INAA, these techniques provide accurate, precise determinations with detection limits of less than 5 ppb to 1 ppm for many of the elements commonly measured in exploration geochemical surveys. The inductively coupled plasma-mass spectrometer (ICP-MS) instrument is capable of even greater sensitivity, and other techniques including direct-current plasma (DCP), and ICP-atomic fluorescence analysis (ICP-AFS) are particularly well-suited for some element groups. The dc-arc emission spectrograph is still used for some geochemical work, but the method has lost ground to the other techniques that have been described.

In the minds of the inexperienced, the spectacular analytical capabilities of these instruments often overshadows the critical importance of the sample preparation and sample decomposition stages of sample treatment.

Table 4.5.6. Comparison of Six Stream-Sediment Analyses for Nickel

	Preparation		Digestion	Ni, ppm
	Crushing	Fraction		
Lab A	no	– 80	70% $HClO_4$	20
Lab B	no	– 80	1:3 HNO_3	60
Lab C	no	– 80	HNO_3/HCl	150
Lab D	no	– 80	1:1 HCl	320
Lab E	yes	–100	1:1 HCl	14
Lab F	no	–100	1:1 HCl	1120

Sample description: Stream sediment with 0.5% magnetite; magnetite contains an average of 0.28% Ni; 80% of magnetite is -80 mesh; 99% of sample is + 100 mesh and 96% is +80 mesh.
Source: Fletcher, 1987.

It was emphasized in the preceding section on sample preparation, that the most informative fraction of a geochemical sample may be the heavy mineral fraction, a relatively coarse fraction (– 30 + 80 mesh), a fine fraction (– 80 mesh), or a very fine fraction (– 240 mesh), depending on the material being analyzed or the particular property of interest in a sample. If an improperly chosen solid sample material is correctly analyzed by the XRF or INAA instruments, the results can be no better than mediocre in the context of the exploration program and may even be misleading.

Sample decomposition presents even greater complexities. The ICP and AAS methods require that elements of interest be introduced to the instruments in solution. This requirement can create both difficulties and opportunities.

The choice of method to take elements into solution requires careful consideration of the mineralogy of the material. Strong decompositions of a geochemical sample can be achieved through treatment with hot concentrated acids or by fusion. Some resistant minerals (e.g., chromite) may not be soluble in

Table 4.5.7. Summary of Decomposition Techniques

Decomposition	Reagents
Fusions	
acid fusions	$KHSO_4$, $K_2S_2O_7$
ammonium halide sublimations	NH_4I, NH_4Cl
alkaline fusions	Na_2CO_3, NaOH, $LiBO_2$, LiB_20_7
oxidative-alkali fusions	Na_2CO_3 or NaOH with KNO_3 or NA_2O_2
Acid digestions	
Hot strong mineral acids	$HF\text{-}HNO_3\text{-}HClO_4$, $HF\text{-}HClO_4$, $HF\text{-}HNO_3\text{-}HCl$
Partial decomposition	
Hot strong mineral acids	HF, $HCl0_4$, $HN0_3$, HCl, HBr, Aqua Regia
Partial digestions or leaches	
Nonselective extractions	
cold dilute inorganic acids:	0.1-1 *N* HCl
organic acids:	acetic acid (pH 5.5) oxalic acid
buffers:	NH_4-citrate NH_3OH (pH 2-8)
chelating agents:	0.05-0.25 *M* EDTA (pH 4-7) NH4-citrate-hydroxylamine hydrochloride (pH 8.5)
Selective extractions	
exchangeable metals:	NH_4-acetate; $MgCl_2$, $CaCl_2$, $BaCl_2$
organic matter:	H_2O_2; NaOCl; $Na_4P_2O_7$ or $K_4P_2O_7$
carbonates:	cold weak acetic acid
manganese oxides:	0.1 *M* hydroxylamine hydrochloride in 0.01 *M* $HN0_3$ (pH 2)
iron oxides:	
amorphous	0.175 *M* $(NH_4)_2\ C_2O_4$-0.10 *M* $H_2C_2O_4$ 0.25 *M* hydroxylamine hydrochloride
crystalline	0.02-1 *M* hydroxylamine hydrochloride -0.25 *M* acetic acid Na-dithionate Hydrazine chloride (pH 4.5) NH_4-oxalate 0.2 *M* oxalic acid in 0.1 *M* ascorbic acid
sulfides:	H_2O_2-ascorbic acid $KClO_3$/HCl followed by 4 *M* HNO_3

Source: Modified from Church et al., 1987.

hot concentrated acids but are broken down by fusion (Table 4.5.5). Table 4.5.6 indicates that nickeliferous magnetite is more readily dissolved by hydrochloric acid than perchloric acid, nitric acid, or aqua regia.

Hydrofluoric acid is the only acid decomposition medium that will dissolve silicate minerals in the typical clastic or lithogeochemical sample. The amount of metal extracted by the strong decompositions achieved with nitric, hydrochloric, perchloric acids, or their mixtures is not total and will vary with the mineralogy of the sample (Table 4.5.7). In addition, as emphasized by Fletcher (1981, 1987), even with similar acid mixtures, extraction efficiency will vary with the acid, sample to acid ratio, and the duration and the temperature of extraction.

Of the fusion techniques, potassium bisulfate treatment does not result in a total extraction, but the alkaline, fire assay, and sublimation techniques can be effective in attacking resistant minerals and extracting specific elements.

Comparison of the quantities of metals extracted by techniques capable of different degrees of decomposition can be very informative. Metals in silicate lattices of rock-forming minerals commonly constitute the background or threshold level of the geochemical sample material. The mineralization component of an anomalous sample is generally contained in sulfides, iron or manganese oxides, or in adsorbed positions on clay minerals. The metal on the clays, and that contained in the sulfides and hydroxides, is more easily extractable than the background/threshold component contained in the rock-forming minerals. Stronger decompositions, which break down the rock-forming minerals, will effectively dilute the anomalous metal components by releasing the lower concentrations of metal in the background/threshold component. Partial decomposition techniques utilizing cold or weaker acids and other reagents that do not break down the rock-forming silicates enhance the more readily extractable mineralization component, resulting in a much greater contrast between the anomalous and background values in the survey.

In numerous geochemical surveys, therefore, the data from partial decompositions, or partial decompositions ratioed against total decompositions, can be much more definitive in target delineation than data from strong or total decompositions. Examples of selective, partial decomposition techniques are listed in Table 4.5.7. Canney and Hawkins (1958) and Bloom (1955) developed cold extraction colorimetric tests for cold-extractable copper (CxCu) and cold extractable heavy metals (CxHM) which are readily adaptable for field use. More recently, Viets et al. (1984) have described a separation technique from solution extracts allowing more sensitive multielement analysis of rocks, minerals, soils, and stream sediments, and Church et al. (1987) have made a comprehensive study of multielement partial extractions.

QUALITY CONTROL. The "house of cards" analog (Fig. 4.5.2) illustrates the relative importance of the various phases of a geochemical program. In addition to observing the basic principles of orientation, sampling, sample preparation and analysis, adequate assurance of analytical quality, representativity, consistency, and precision are essential when all data are evaluated.

Quality controls can be introduced at several stages in the geochemical program. Duplicate sampling of the designated sample material at the sample site produces two samples (*field duplicates*) that will give a measure of the combined variance in the sample material and the sampling procedure. Division of a sample prior to the sample preparation stage (*sample duplicates*) provides a measure of the sample homogeneity. Splitting of the sample after the appropriate sample preparation (*analytical duplicates*) will provide information on analytical variance and also sample homogeneity.

Specially prepared *standard reference samples* having known metal contents provide essential material for checking the precision and accuracy of an analytical laboratory on a batch basis and also for monitoring analytical drift with time when numerous batches are forwarded to the same laboratory over the life of a project. It is very important to know that analytical data collected at all stages of a project are comparable.

The usual practice of the routine addition of one of a suite of standard reference samples with every 30 routine or batch samples and the inclusion of sample and analytical duplicates at a similar frequency will provide a 10% quality control volume. Routine scanning of results from these standards and duplicates will give an immediate indication of unsatisfactory precision and accuracy and sample inhomogeneities. Since it is the objective of the geochemical laboratory to produce good and reliable data and maintain an ongoing, profitable customer relationship, quality control is the initial concern of both the laboratory and the client.

If irregularities are noted, the laboratory should be contacted immediately and the discrepancies discussed. Every reputable laboratory will have its own standard reference samples and other controls and should rerun samples without charge if the quality of its output comes into question. If erratic analytical results are experienced with the duplicate and batch samples but not with the standard reference materials, sample inhomogeneity ("nugget" effect) is indicated. Experienced commercial laboratory personnel can assist with these problems by utilizing special techniques designed to overcome sample inhomogeneities.

4.5.1.8 Interpretation Principles

Geochemical interpretation does not begin only after all samples have been collected, prepared, and analyzed. Interpretive methodologies develop progressively throughout a geochemical project. The patterns observed during the orientation program are directly related to the dispersion characteristics of the metallic elements, the nature of the overburden, and the overall general geochemical environment. Such recognition contributes to the formulation of interpretational procedures and this developing understanding impacts directly on the sampling, sample preparation, and analytical procedures that are selected for the geochemical program.

The cumulative experience of geochemical behavior gained from orientation surveys and case histories in a broad spectrum of environments has enabled geochemists to compile a series of *conceptual interpretation models* that pictorially represent dispersion behavior in a wide variety of landscape configurations. The simplified example in Fig. 4.5.10 illustrates the formation of geochemical anomaly as patterns related to natural factors within the landscape. Models like these have been developed for a wide variety of environments including those in the Canadian Cordillera and the Canadian Shield (Bradshaw, 1975), in northern Europe (Kauranne, 1976), in the Basin and Range (Lovering and McCarthy, 1978), and in Australia (Butt and Smith, 1980). The initial framework for the conceptual models came from the portrayal of landscape geochemistry by the Soviets as early as the 1930s and was adapted to synthesize geochemical data by Fortescue (1975, 1980). All have certain common features (Hoffman and Thomson, 1987):

1. A body of mineralization or another source that may mimic mineralization.
2. The relative distribution of bedrock, overburden, soil, groundwater, surface water, vegetation, and other factors.
3. Highlighted dispersion pathways related to mineralization and anomaly formation.
4. Each model portrays dispersion as a series of patterns related to and controlled by a variety of identifiable natural factors.

Fundamentally, geochemical interpretation involves the recognition of these patterns, the identification of the factors causing them, and the extrapolation of the patterns back to a mineralized or other source.

With the gradual acceptance of these concepts, *geomorphological mapping* through ground and air-photograph observations is proving to be an invaluable tool in the interpretation of geochemical patterns particularly in areas of mature landscapes.

The importance of the preferred emphasis on patterns of dispersion rather than the magnitude of the geochemical values in parts per million or parts per billion units can be illustrated by reference to Fig. 4.5.10. Precipitation and accumulation of hydromorphically transported metal in the seepage anomaly area adjacent the stream channel can invariably result in concentrations markedly higher than in the surface horizons of a residual soil anomaly on a well-drained slope. In such a situation, reference only to the magnitude of values would result in first priority for follow-up being assigned to the seepage anomaly, whereas recognition of the location and shape of the seepage anomaly would immediately indicate its origin and direct the interpretation to the source areas upslope.

In parallel with pattern recognition, geochemical interpretation requires a knowledge of the *"anomalous," "threshold,"* and *"background"* values of the elements of interest in the survey. The fundamental observations leading to the identification of these values come from the orientation survey. In the simplest orientation scenario, background values are not influenced by the presence of mineralization and in Fig 4.5.11 are represented by the relatively homogeneous areas of low values at the extremities of the hypothetical soil traverse. Anomalous values are, in this example, the higher values peaking a short distance downslope of the suboutcropping vein zone. Weaker mineralization, disseminated in the rocks on either side of the vein zone, also gives rise to anomalous, but less spectacular, values which are also influenced by the topographic slope. The upper limit of the background population is referred to as the threshold and it is clear that contouring this data at the threshold level will outline an anomalous pattern related to the mineralization in the bedrock.

Unfortunately, too many explorationists apply this simple concept of anomaly definition far too rigidly in all their exploration programs. This reflects an erroneous conception of the geochemical environment and totally ignores the geological influences on metal distribution and the basic principles of dispersion illustrated by the geochemical models.

The hypothetical example described (Fig. 4.5.1l), assumes no variation in bedrock type or environmental factors. For the majority of field areas, the simple scenario portrayed in Fig. 4.5.11 is the exception rather than the rule, and the introduction of geological and geomorphological complexities such as those shown in Fig. 4.5.12 can produce spurious anomalies.

During mineralization processes, some elements (e.g., sodium) may be leached from the host rocks by the mineralizing fluids. In such a situation, a depletion in values below the regional background for the element will cause an "anomalous low" that is geochemically and economically significant.

The interpretation of geochemical data and the correct identification of significant anomalies therefore requires a fundamental awareness of the geochemical environment as presented in the geochemical model (Fig. 4.5.10), a knowledge of the geology, structure, and other characteristics of the type of deposit sought, and an underlying understanding of the geochemical behavior of the elements of interest. This procedure is clearly described and illustrated with numerous actual examples in a workbook format by Levinson et al. (1987).

Over the past several years, geochemists have adapted *statistical methods* of evaluation to assist in geochemical interpretation. This has not been a simple evolution. As noted by Sinclair, (1987), "to attempt to carry out a blind evaluation of data by submitting them to any one of an increasing number of packaged software systems and expect a computer to do our thinking for us is patently wrong." Effective application of statistical methods, whether univariate or multivariate, requires not only the same full appreciation of the geochemical environment, the geology, and the chemistry of the elements as described in the proceeding paragraphs but also an understanding of the statistical technique being employed. In addition, this effectiveness is also dependent on correct design, representativity and quality of the sampling and analytical phases of the program. It is not difficult to appreciate that statistical treatment of a data set lacking in representativity and quality will primarily highlight error, to the virtual exclusion of relationships of geochemical significance.

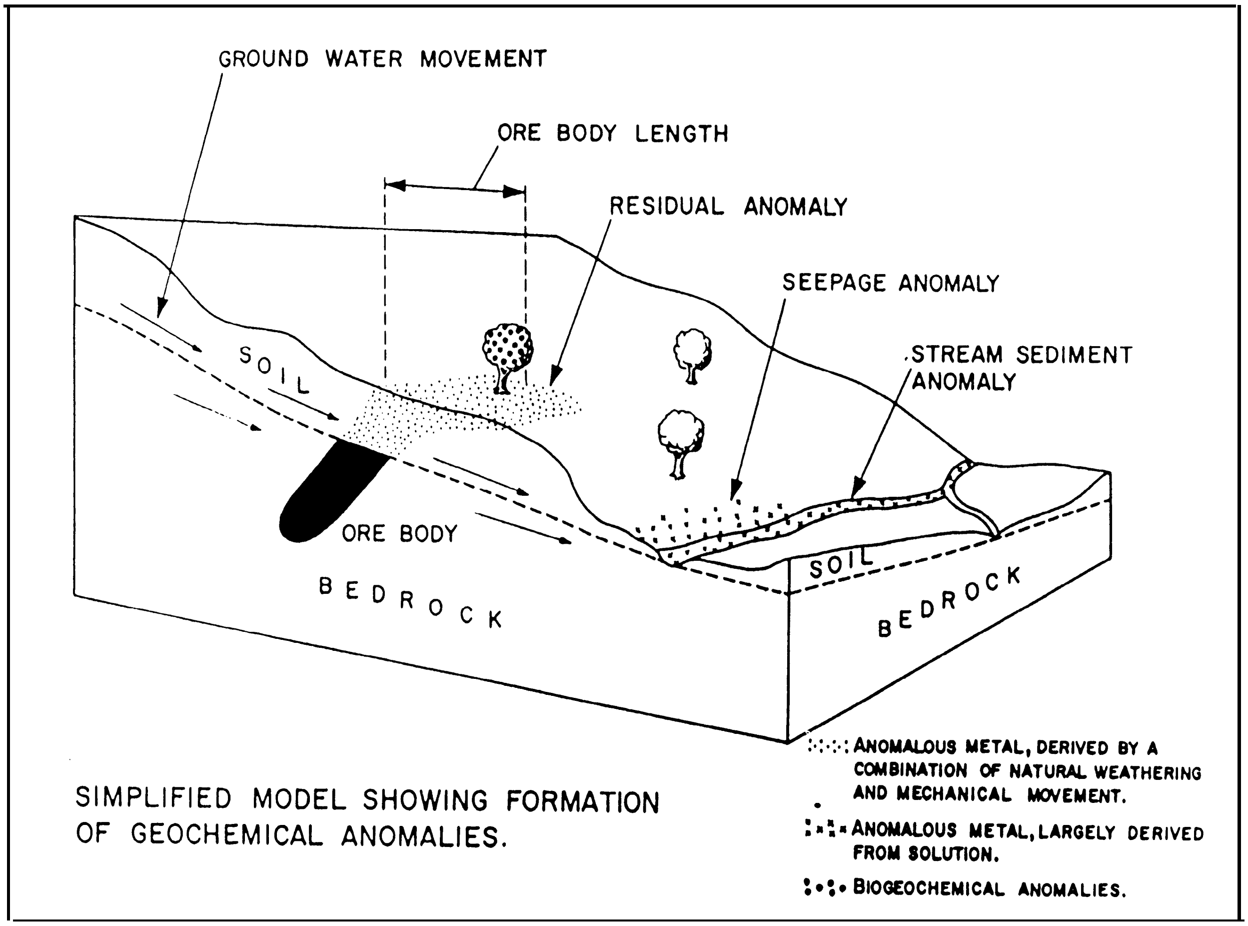


Fig. 4.5.10. Simplified model showing formation of geochemical anomalies (Hoffman and Thomson, 1987).

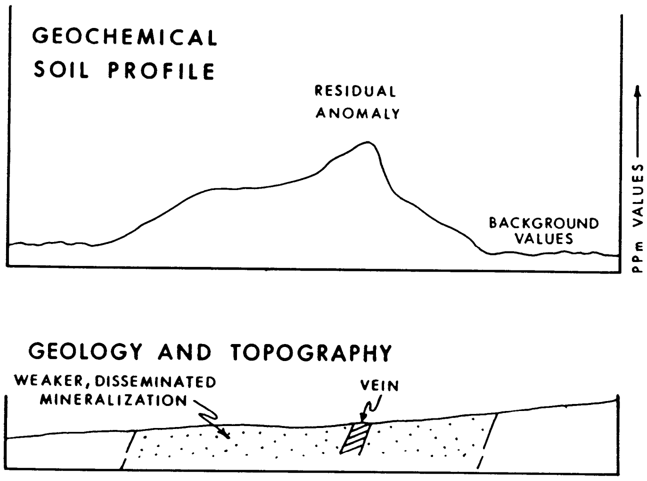


Fig. 4.5.11. Hypothetical soil profile across mineralized zone.

Given these essential understandings, statistical techniques provide useful and often powerful tools for geochemical data analysis. They can assist in explaining previously unrecognized characteristics in a data set and making significant patterns more consistent, larger, or otherwise more easily recognizable.

A basic assumption applied in the statistical treatment of geochemical data is that the data are unbiased and continuous. Most geochemical data are discrete, but fortunately, in practice these discrete values commonly are sufficiently abundant and close that they can be assumed to be continuous. Such is generally the case if the number of values is large and if the analytical uncertainty is small relative to the spread of the values. Because of this statistical assumption used in the analysis of geochemical prospecting data, other statistical approaches are continually being assessed (Kurzl, 1988).

A review of basic statistics is beyond the scope of this presentation, but a useful summary has been prepared by Sinclair (1987) and a more comprehensive treatment is covered by Howarth (1983).

Univariate statistics allow the geochemist to quickly become familiar with voluminous sets of data. Histograms and probability plots provide a means of displaying data and revealing internal complexity. A histogram (Fig. 4.5.13) has the obvious advantage of providing visual information on the total range of values, the existence of one or more populations, and possible thresholds separating background and anomalous values. Data grouped for purposes of constructing a histogram can be cumulated (from high to low values or vice versa) as for a cumulative histogram and plotted directly on probability graph paper (Fig. 4.5.14).

Probability plots, as described by Sinclair (1976), and the "PROBPLOT" computer program of Stanley (1987), have found

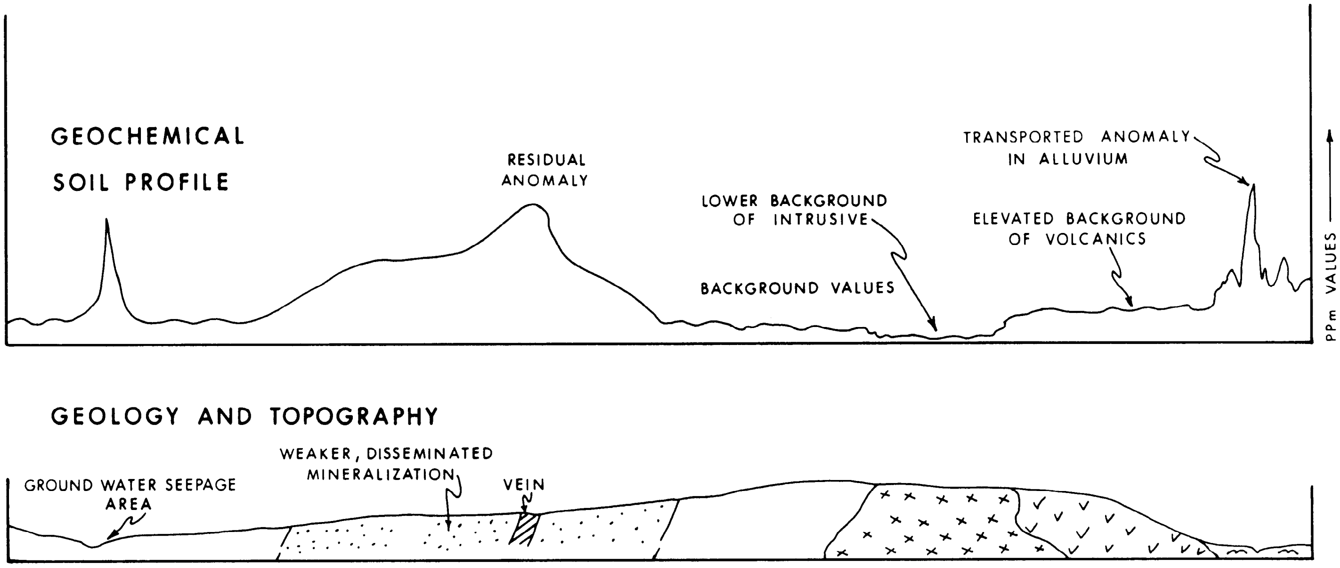


Fig. 4.5.12. Hypothetical soil profile illustrating spurious anomalies related to geological and geomorphological complexities.

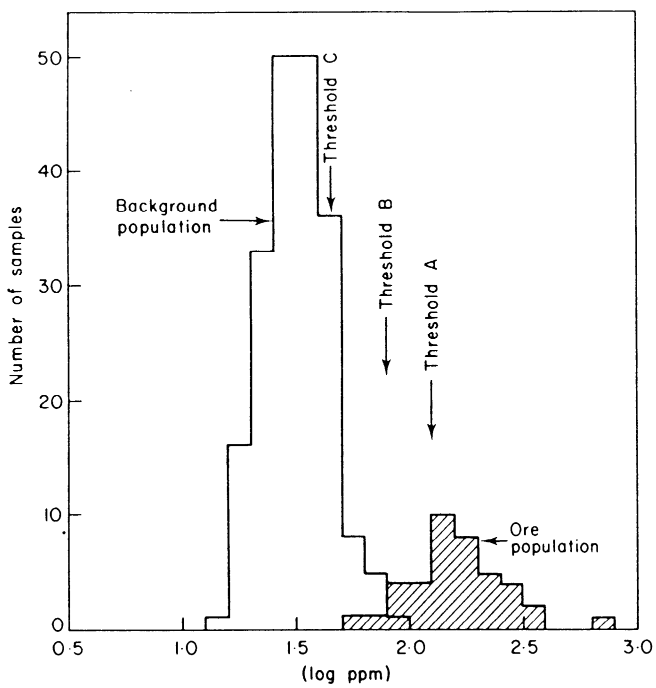


Fig. 4.5.13. Overlap of values in background and ore populations (derived as random samples from two log-normally distributed populations) (*Source*: Rose et al., 1979).

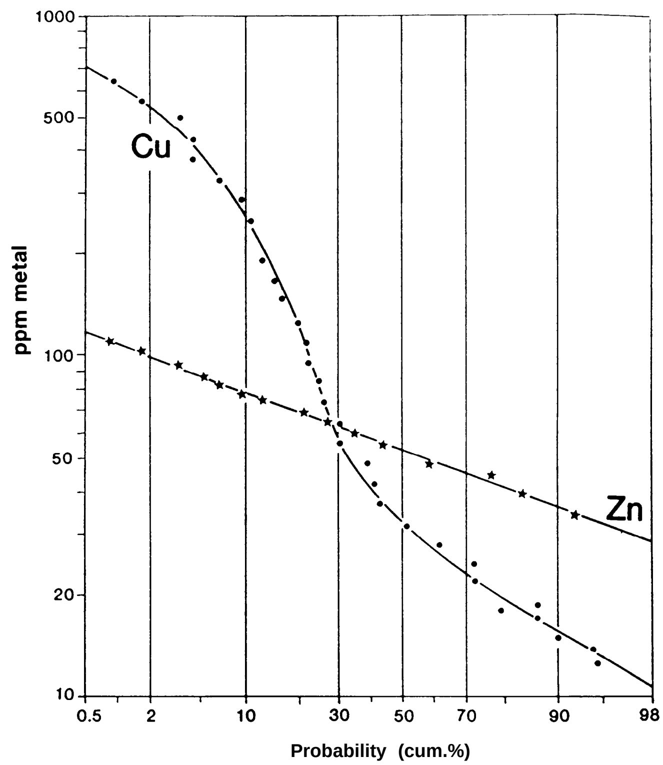


Fig. 4.5.14. Probability graphs (cumulative curves) for zinc and copper in B horizon soil samples over the Daisy Creek stratabound copper prospect, western Montana. Cumulated from high to low values (Sinclair, 1986).

widespread use in geochemistry and allow the partitioning of consistent populations and the estimation of threshold values. Depending on the statistical distribution of geochemical data, one ordinate of the probability graph paper is either arithmetic or logarithmic, and the other ordinate, the probability scale, is arranged such that a normal or log-normal distribution will plot as a straight line. This type of graph is a very sensitive method for recognizing the presence of multiple populations.

In this example shown in Fig. 4.5.14, the straight-line plot for zinc indicates a single log-normal population and the curved line for copper indicates the presence of more than one population. The inflection point of the copper curve can be interpreted from the probability plot to be at the 25 percentile (Fig. 4.5.15), indicating the curve to represent a mixture of 25% of the higher population A and 75% of the lower population B, both of which can be mathematically derived. The two populations overlap

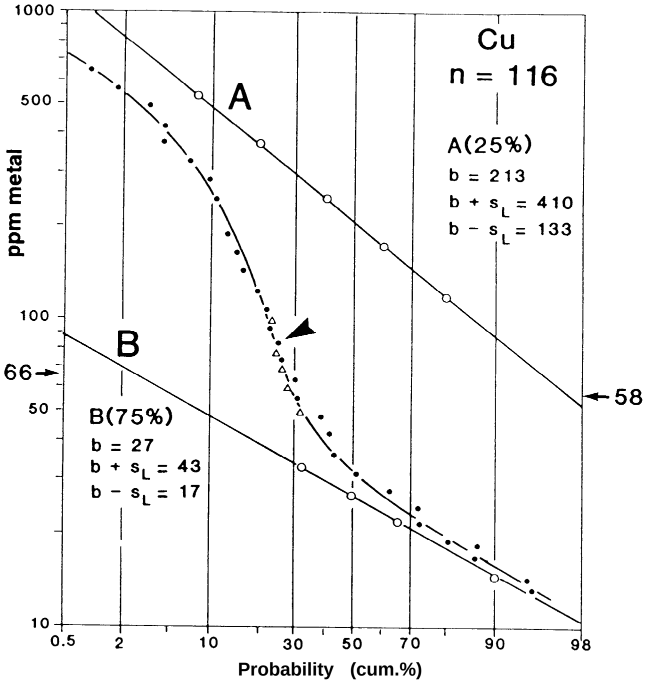


Fig. 4.5.15. Probability graph of copper in soils (from Fig. 4.5.14) partitioned into two ideal components, A and B. An arrowhead shows the interpreted position of an inflection point in the curve. Black dots are original data; open circles are construction points determined by the partitioning procedure; open triangles are check points determined by combining ideal populations A and B in their zone of overlap to compare with real data. Thresholds of 58 and 66 ppm are determined at the lower 2.5 percentile of A and the upper 2.5 percentile of B (Sinclair, 1986).

slightly, but it is clear that if populations can be defined in this way, an estimated or statistically calculated value can be identified that can be contoured or otherwise differentiated that will separate the populations for interpretation purposes.

Multivariate analysis is used to investigate complex interelement relationships and can include other types of exploration parameters based on geological, geophysical, or other data. A necessary prerequisite for more advanced multivariate analyses are correlation studies between pairs of variables (scatter diagrams, correlation coefficients) and between populations (analysis of variance). Based on this information, it is possible to investigate intra-element and between-sample relationships using a variety of techniques (principal component, factor analysis, cluster analysis, multiple regression, discriminate, and characterization analysis) (Howarth, 1983).

An example of the use of these multivariate techniques is the mineral resource assessment map produced as a result of joint collaboration of the geological surveys of Finland, Norway, and Sweden in the Nordkalott Project north of the 66th parallel. Discriminant and characterisation analysis methods were applied to integrate 20 geochemical, 5 geophysical, and 5 geological variables to produce a resource assessment map showing regional favorability for mineral deposits (Fig. 4.5.16). In addition, scores were calculated representing each of 13 metallogenic provinces, and the local favorability for the existence of each deposit type are shown as symbols in which size indicates favorability and color the most likely deposit model. The model areas are shown in Fig. 4.5.16 as irregular polygons.

4.5.1.9 Geochemical Atlases

Sampling of stream sediments, soils, and other media in geochemical exploration is usually conducted at relatively high densities. Early surveys at relatively wider spaced sample intervals (Holman, 1963) revealed geochemical patterns reflected, to a much greater degree, signatures related to the surface geological components with a relative diminution of the signal from anomalous mineralization sources. The publication of the Wolfson Geochemical Atlas covering England and Wales (Webb et al., 1979) vividly illustrated the influence of the geological component in a stream sediment survey at a density of one sample per square kilometer (per 0.4 mi^2). The Wolfson Atlas stimulated a number of additional large-scale geochemical mapping programs in Alaska, (Weaver et al., 1983), Germany, (Fauth et al., 1985), Newfoundland (Davenport, 1982), Great Britain (Plant and Slater, 1986), and Fennoscandia (Bolviken et al., 1986).

The Fennoscandinavian or Nordkalott Project involved the collection of several types of sampling media from the stream and glacial till environments at an average density of one per 25 km^2 (one sample per 9.65 mi^2). Data handling and presentation techniques developed as part of this program were very innovative (Fig. 4.5.17) allowing effective, broad-scale depiction of metal distributions that serve as a focus for more detailed geochemical prospecting surveys. In part, concurrent with the Nordkalott Program, the Geological Survey of Finland conducted a glacial till sampling survey covering the whole country at a scale of one sample per 300 km^2 (one sample per 116 mi^2). Again, using innovative data handling and data presentation techniques, these results revealed patterns that were not directly correlatable with mapped geological lithologies, but possibly reflecting deeper crustal structural patterns of metallogenic significance. Alignment of the geochemical patterns and significant mineral deposits (Fig. 4.5.18) supports this prediction (Bjorklund, 1989). The plotting of individual element results from the same sample database-one per 300 km^2 (one sample per 116 mi^2)— has also served as a focus for more detailed exploration (Fig. 4.5.19).

The low-density geochemical maps that have been described contain useful information for the agricultural, forestry, and environmental sciences as well as the geological sciences, and planning has begun for an International Geochemical Mapping Project which will include contributions from a large number of developed and less-developed nations (Darnley, 1988). It is clear that the project has the potential of generating invaluable information on the global distribution and relative concentrations of economically significant metals of fundamental interest to the mining industry.

4.5.2 OTHER PROSPECTING TECHNIQUES

4.5.2.1 Background

The remarkable advances in new technology over the past three decades have stimulated numerous avenues of research and the development of new methods that can be applied in mineral prospecting. Usually, each new method experiences a "breaking-in" period during which the technology is applied in a variety of situations leading to a better understanding of the properties being investigated and refinements in interpretation procedures.

Before these methods are described briefly, it is appropriate to note that certain well-established techniques of prospecting are still as effective today as they were in the earlier days of mining. Such techniques include the skillful use of *heavy mineral panning,* particularly in the search for gold, platinum, cassiterite, tungsten, barite, and other resistant metals and minerals of high specific gravity (Chapter 15.1.1). Although undocumented, panning may be responsible for the discovery of more ore and mineral occurrences than any other technique, and its application in the stream environment is well described in some older publications (e.g., MacKay, 1921, and Lindgren, 1911, in Boyle, 1987). The *ultraviolet light* or "blacklight" method is another established technique with specific application in the search for naturally fluorescent minerals such as scheelite, willemite, and fluorite.

4.5.2.2 Airborne and Satellite Techniques

Since the early 1970s, aerial photography has been supplemented by a variety of satellite- and aircraft-borne electronic imaging systems that can be used to map structural, lithologic, and alteration features. The era of satellite remote sensing for geology began with the launch of the ERTS-1 (later renamed Landsat 1) satellite on July 23, 1972. On board ERTS-1 was the Multispectral Scanner (MSS) that was the first satellite-borne electronic imaging system designed for the analysis of the earth's resources. MSS images provided a fairly coarse spatial resolution of 80 m in four regions covering visible and infrared light. Geologists quickly realized that they could use these data to map regional structures and, in some cases, limonite-stained rocks. Interest on the part of the earth-science community was great enough that government and industry geologists influenced the design of the Thematic Mapper that was first put in orbit aboard Landsat 4 in 1982. TM provides an improved spatial resolution of 30 m (98 ft) and coverage of longer wavelengths of infrared light. The longer wavelength infrared images collected by TM provide an ability to map hydrothermally altered areas by identifying high concentrations of clay in rock and soil. Nations other than the United States have also recognized the usefulness of satellite remote sensing. France recently launched SPOT-l (Systeme Probatoire d'Observation de la Terre) that provides spatial resolutions of 20 m (66 ft) for visible and near IR wavelengths and 10 m (33 ft) for a broad wavelength panchromatic band. SPOT has the additional capability of collecting stereo images.

Both governments and the private sector have produced airborne remote sensing systems which push the cutting edge of the technology forward. These systems are operated from aircraft much the same way as a conventional aerial camera. Systems such as the Jet Propulsion Laboratory (JPL)-developed Airborne Visible and Infrared Imaging Spectrometer (AVIRIS), and the Geophysical Environmental Research Inc.-developed Imaging Spectrometer (GERIS) are designed to map the mineralogy of the surface material. These systems produce images with spatial resolutions that vary from 5 to 30 m (16 to 98 ft) depending on the altitude of the aircraft. The Thermal Infrared Multispectral Scanner from JPL (TIMS) produces multichannel thermal (temperature) images that can be used to differentiate and map tectosilicates (particularly silica). Airborne remote-sensing systems provide data for both immediate use and means for determining which types of instruments will ultimately be mounted on satellites.

For more detailed information on remote sensing applications in geology and exploration, the reader is referred to the books by Drury (1987), Sabins (1987), and Siegal et al. (1980).

4.5.2.3 Fluid Inclusion Studies

Fluid inclusion studies have figured prominently in research in the composition of ore-forming fluids and the conditions of deposition of ore deposits. Spooner (1981) acknowledges that the studies, in combination with other techniques, have contributed very significantly to the better understanding of hydrothermal mineralizing systems.

A fluid inclusion in an ore or gangue mineral represents a fluid composition existing at some time during the history of a hydrothermal system. It can be anticipated that these compositions will vary with time and position as the hydrothermal system develops and matures (Roedder, 1967). Two types of fluid inclusion are recognized: (1) the *primary* inclusion entrapped during the growth of the host crystal and (2) the *secondary* inclusion in which fluids are entrapped after the growth of the host crystals and that commonly occur along microfractures. Secondary inclusions are the most numerous, and careful petrological observation is clearly a critical aspect of fluid inclusion studies.

Primary inclusions are frequently found in hydrothermal veins that have formed in open fissures. Their compositions vary considerably, but most fluids fall in the carbon-oxygen-hydrogen-sulfur plus sodium chloride (C-O-H-S + NaCl) system with the most prominent constituents being carbon dioxide (CO_2) and water (H_2O). Other species occurring in this system include carbon monoxide (CO), hydrogen (H_2), methane (CH_4), hydrogen sulfide (H_2S), and sulfur dioxide (SO_2). The salt composition ranges from zero to as much as 40% by weight consisting commonly of Na^+ but with variable but important amounts of K^+, Ca^{2+}, or Mg^{2+}.

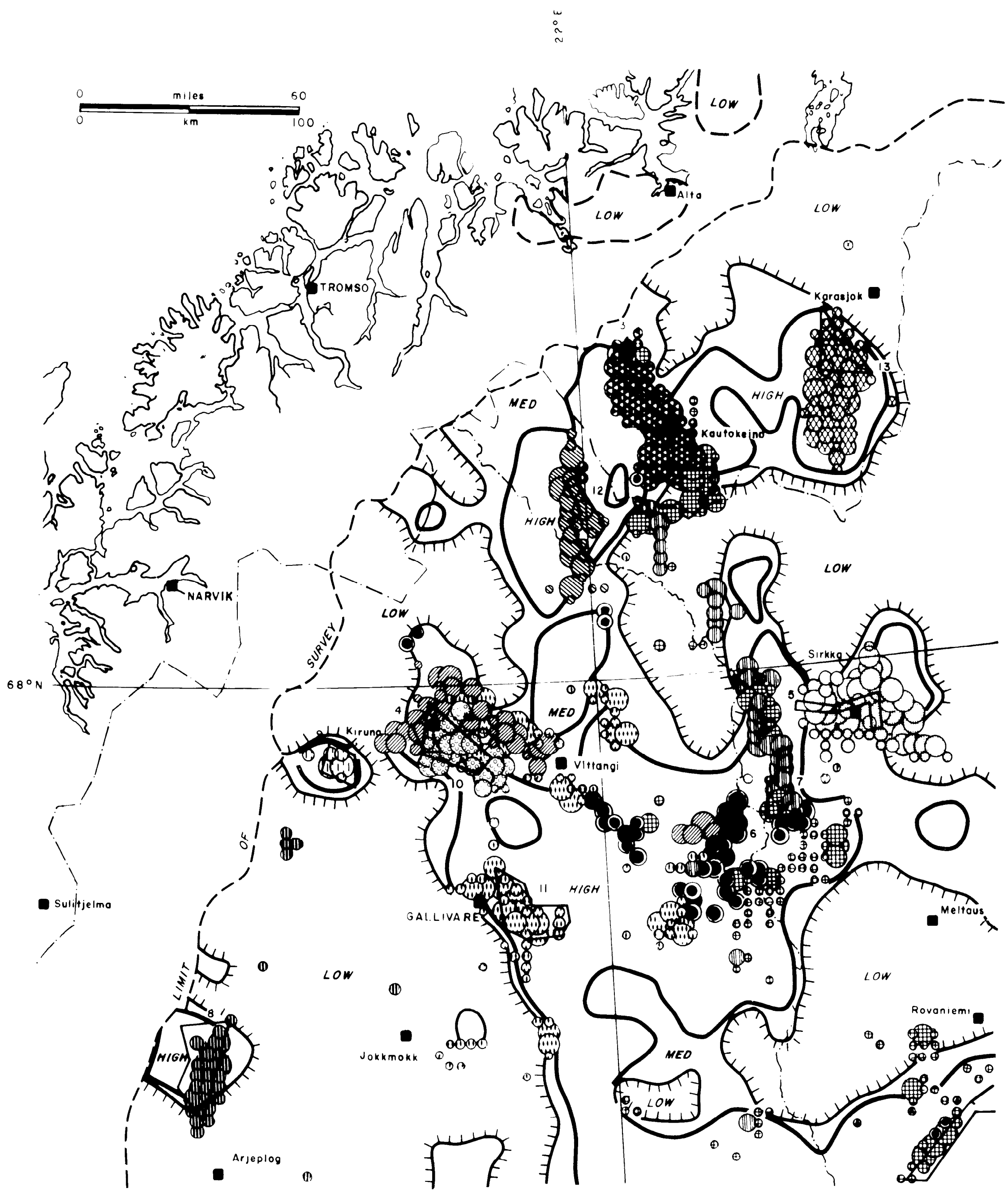
The fluid inclusion components are commonly distributed into immiscible gaseous and aqueous and, less frequently, solid phases. On heating, these phases homogenize and the temperature of homogenization correlates with the (minimum) temperature of the fluid at the time of entrapment. Consequently, by reference to phase diagrams for CO_2 and P-T diagrams for both CO_2 and H_2O, it is possible to determine the pressure and temperature of the ore-forming fluid by careful examination of primary inclusions. The study of inclusion compositions across a single vein or in a succession of veins can, therefore, indicate changes in composition (as well as temperature and pressure) of the parent fluid during the mineralizing period. Table 4.5.8 (Colvine et al., 1988) is a summary of fluid inclusion data for Archean gold deposits in the Canadian Shield and the Yilgarn Block of Western Australia. Homogenization temperatures range between 392° and 752°F (200° and 400°C) to cluster around 662°F (350°C), and calculated pressures range from 1.5 to 4.5 kb (equivalent to depths of 2.5 to 7 mi or 4 to 12 km). Compositionally, these fluids are of low salinity, aqueous with a moderate to high CO_2 density. Norman (1989) in studies of fluid inclusions from the Creede Mining District, has noted high H_2S concentrations associated with the precious metal deposits. Calculated temperatures of deposition at Creede and some other epithermal districts in the western USA are in the 482° to 536°F (250° to 280°C) range.

There is a huge bibliography of fluid inclusion studies, and the interested reader is referred to the publications of Roedder (1967, 1972, 1976) and Crawford (1981) for more detailed discussions of the technique, its application, and its contribution in mineral deposits geology.

4.5.2.4 Isotope Studies

In recent years, stable and radioisotope studies have expanded dramatically within mineral deposit geology and have found application in exploration. The object of these studies have been (Guilbert and Park, 1986)

1. To determine the composition and sources of ore-bearing fluids.
2. To determine the source of ore-forming components.
3. To determine the age of mineral deposits.



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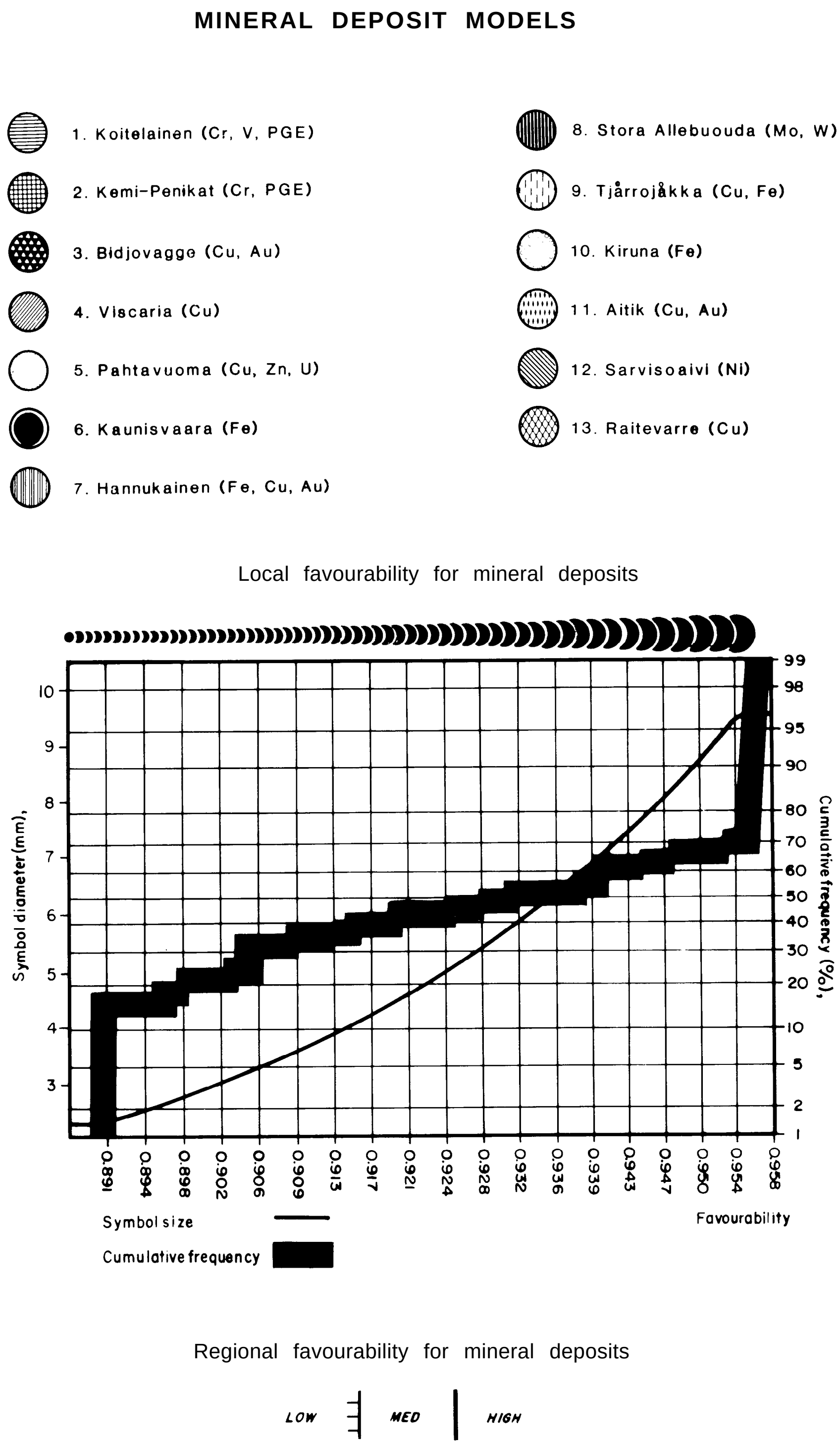

Fig. 4.5.16. Northern Fennoscandia mineral resource assessment map, northern Fennoscandia, regions and locations highly favored for mineral deposits *(Mineral Resource Assessment Map,* Northern Fennoscandia, Geological Surveys of Finland, Norway, and Sweden, 1986).

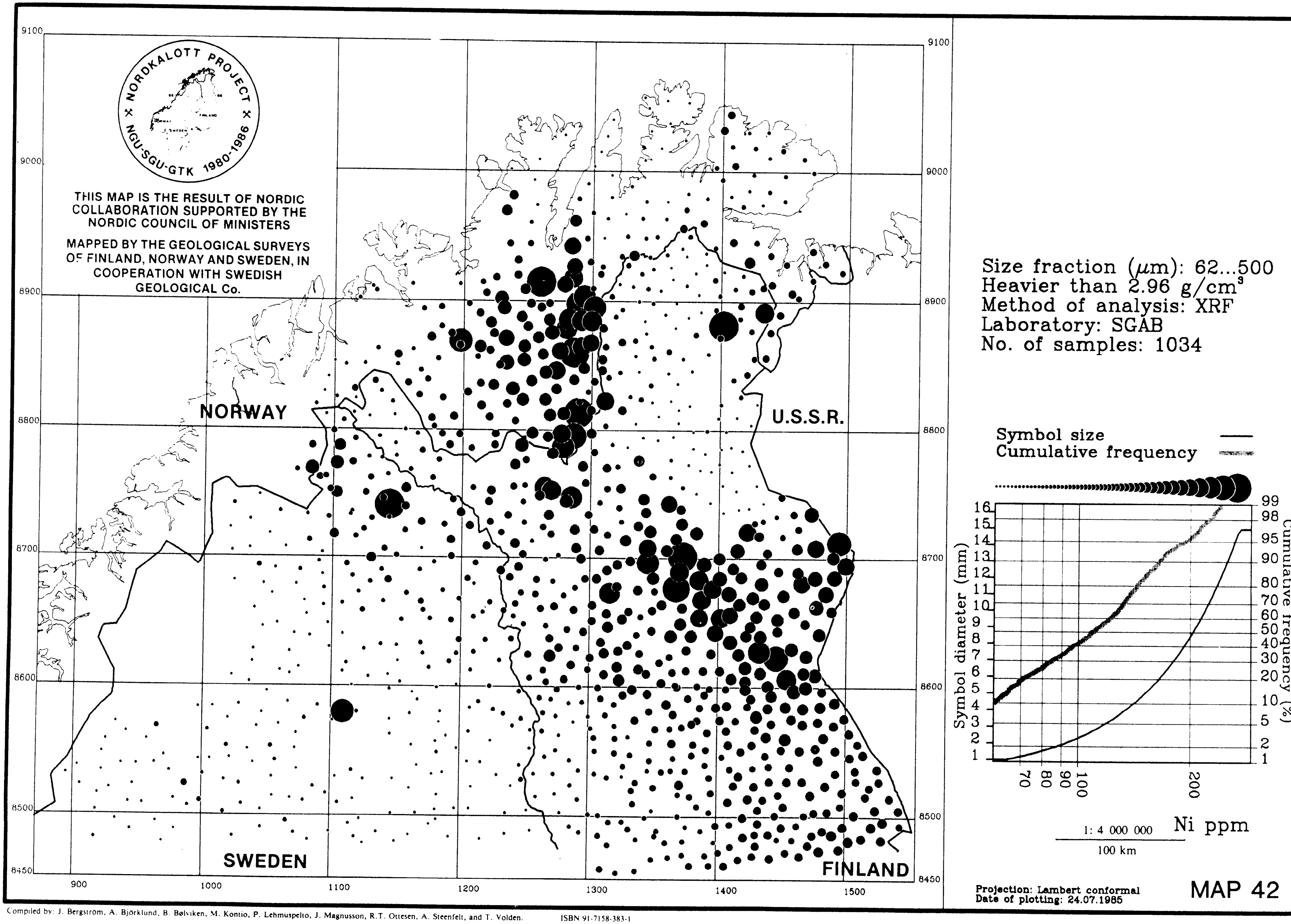

Fig. 4.5.17. Nordkalott project. Nickel content of heavy mineral fraction of tills *(Geochemical Atlas of Northern Fennoscandia,* Geological Surveys of Finland, Norway, and Sweden).

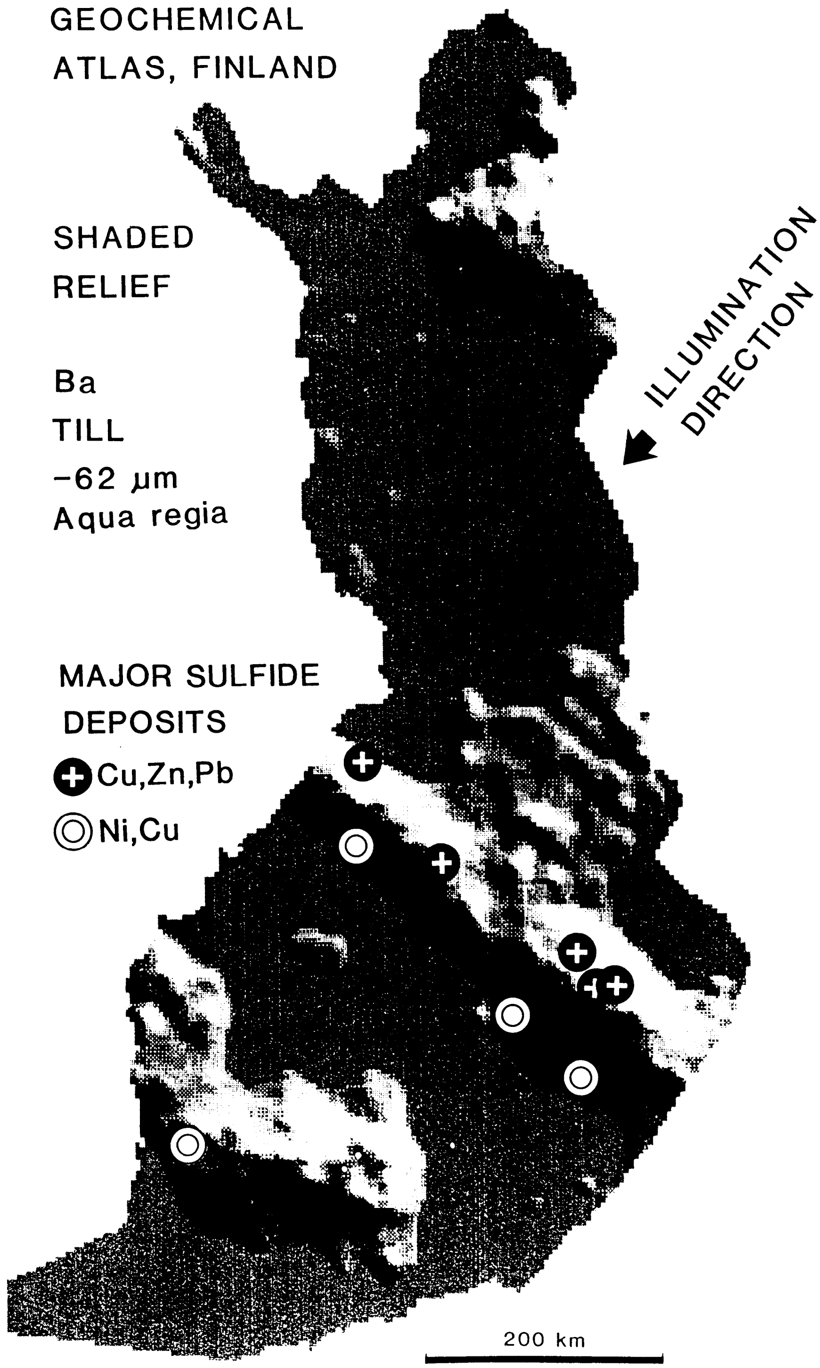

Fig. 4.5.18. Geochemical atlas of Finland. Alignment of geochemical patterns and significant mineral deposits (Björklund, 1989).

4. To help determine temperature and other conditions of deposition.

5. To establish the degrees of bacteriological involvement in certain ore-forming processes.

This expanded understanding of isotope behavior, aided considerably by improved technology, has led to the development of techniques of considerable promise for detailed prospecting.

Stable isotope investigations involve the study of the different properties of isotopes of the same element and assume that the abundance ratios of these related isotopes in geological materials have not changed over time. Since stable isotope ratios are influenced by natural processes, the understanding of this behavior can be indicative of how rocks and minerals associated in mineral deposits have been formed. The field is complex, with ratios being affected by natural processes of oxidation, reduction, precipitation, dissolution, evaporation, condensation, adsorption, desorption, diffusion, and biochemical activity. Stable isotope measurements on minute geological samples are now possible due to significant advances in laser and ion probe technology (Shanks and Criss, 1989).

When seawater is evaporated, the lighter isotopes of *oxygen* and *hydrogen* are enriched in the vapor, and consequently the $^{18}O/^{16}O$ and $^{2}H/^{1}H$ (or $\delta^{18}O$ and δD) ratios are decreased. $\delta^{18}O$ and δD values are increased relative to the vapor in the primary rainfall, and the lighter isotopes become enriched in meteoric waters progressively inland and polewards on all continents.

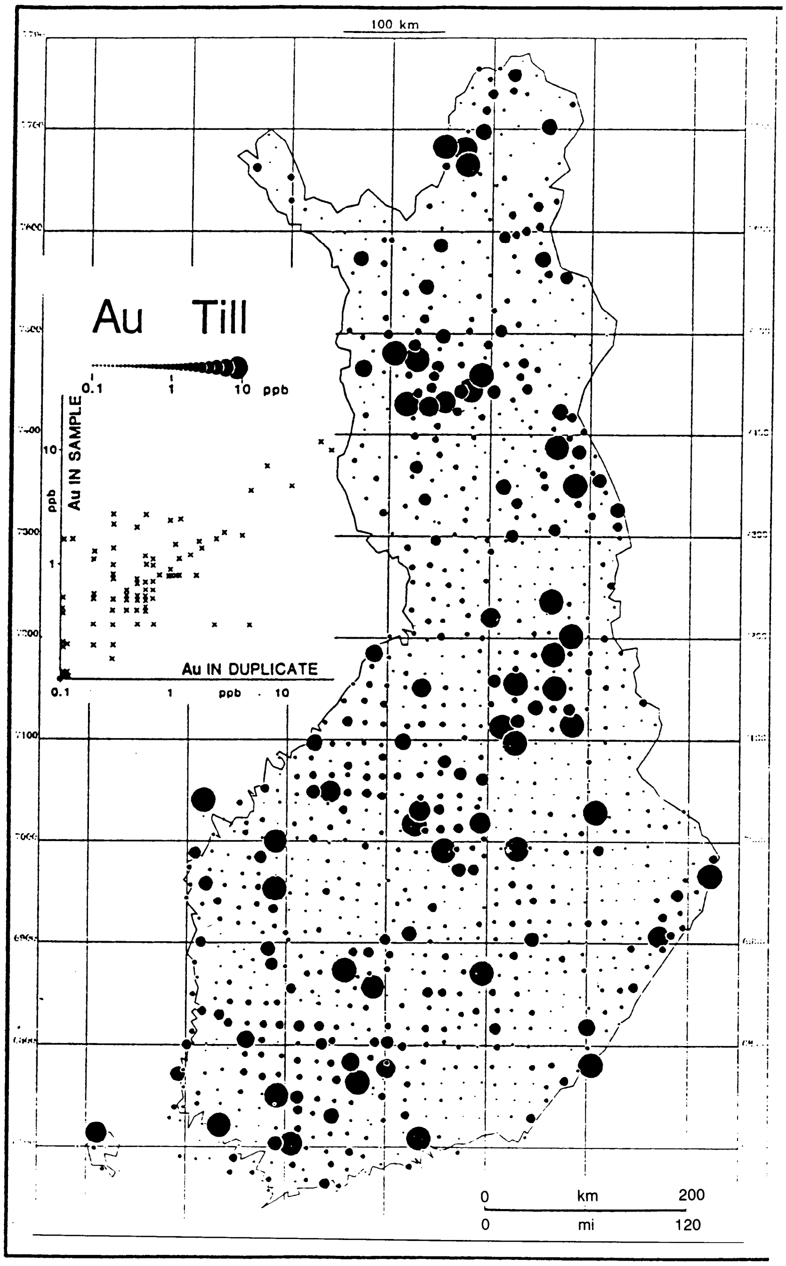

Fig. 4.5.19. Gold in till [-63 μm fraction], (— 240 mesh) *Geochemical Atlas of Finland* (Geological Survey of Finland).

Taylor (1967, 1974) incorporated these meteoric water variations into plots, which also define the fields of other waters in the earth's crust, viz., connate, metamorphic, magmatic, and seawater (or SMOW—standard mean ocean water) (Fig. 4.5.20). Fig. 4.5.21 shows variations in the $\delta^{18}O$ and δD ratios measured in various minerals associated with ore deposits. Comparison of Figs. 4.5.20 and 4.5.21 illustrates how oxygen and hydrogen isotope studies can provide information on the nature of the mineralizing fluids involved in ore formation. In practice, the interpretation of these ratios is complex, but their study is leading to an improved understanding and evaluation of mineral deposit environments.

The *sulfur isotope* ratio $^{34}S/^{32}S$ of a sample relative to the same ratio in the Canyon Diablo meteorite is referred to as $\delta^{34}S$. The $\delta^{34}S$ value for the meteorite (troilite) is zero and sulfides or igneous rocks having a $\delta^{34}S$ value at or close to zero are interpreted as being of mantle derivation. Biogenically derived sulfur is isotopically lighter and this characteristic has stimulated considerable investigations of the epigenetic or syngenetic origin of many types of mineral deposits. Other studies, necessarily integrated with careful geological observations, have illustrated many complexities related to diverse sources of sulfur and isotopic partitioning correlatable with changes in redox conditions, temperature, and sulfur concentration in the mineralizing fluid. Sulfur isotope variations in nature, from a paper by Ohmoto and Rye (1979), are shown in Fig. 4.5.22.

Radioisotopes are the product of radioactive decay of some preexisting element. Ratios of radioactive isotopes are affected only by time and the specific decay constant of the isotopes of interest. The ratios are unaffected by metamorphism or orogeny, biological activity, temperature, or intrusion and have important geochronological application. The most commonly used radioactive decay series are those of rubidium-strontium (^{87}Rb-^{87}Sr), uranium-thorium-lead (^{235}U-^{207}Pb; ^{238}U-^{206}Pb; ^{232}Th-^{208}Pb), and

Table 4.5.8. Summary of Fluid Inclusion Data for Archean Gold Deposits in the Abitibi Subprovince of Canada and the Yilgarn Block, Western Australia

Deposit or Area (Reference)	Fluid Inclusion data	Interpretation
Red Lake, Ontario Brown and LaKind (1986)	Primary low salinity H_2O-CO_2, higher CO_2 density, variable CO_2/H_2O ratios, T_h = 110° to 360°C, GOLD RELATED Secondary aqueous brine, T_h = > 350°C Secondary low salinity (< 10 equi. wt % NaCl), CO_2-H_2O, low CO_2 density	Possible fluid unmixing
McIntyre-Hollinger mine, Timmins, Ontario Smith et al. (1984)	Pseudo-secondary or primary H_2O-CO_2, T_h = 220° to 385°C, 3 to 24 mole wt % CO_2, CO_2 liquid phase homogenization, CO_2 density range 0.46 to 0.78 g/cm^2, GOLD RELATED Secondary CO_2-bearing aqueous inclusions, T_h = 160° to 215°C In graphite-bearing vein, pseudo-secondary to primary CH_4-$C0_2$-H_2O inclusions (> 26 mole % CH_4) Secondary CH_4-rich inclusions	Possible fluid unmixing; fluids buffered by QFM; depositional pressure 0.35 to 0.5 bars
Hollinger mine, Timmins, Ontario Wood et al. (1986)	Primary H_2O-CO_2, variable phase proportions; 1 to 4 equi. wt. % NaCl; 6 mole % CO_2; T_h = 225° to 355°C, mean 277°C; minor CH_4; CO_2 density 0.65 g/cm^3	Phase separation
O'Brien mine, Cadillac, Quebec Krupka et al. (1977)	Primary H_2O-CO_2 with constant phase proportions.	Early fluid homogeneous
	Coexisting secondary H_2O-CO_2; T_h of both types= 210° to 380°C	Late heterogeneous fluid
Sigma mine, Val d'Or, Quebec Robert and Kelly (1987)	Secondary H_2O-CO_2 inclusions with < 10 equi. wt. % NaCl, 15 to 30 mole % CO_2; T_h = 285° to 395°C	
	Coexisting H_2O-rich inclusions, 25 to 34 equi. wt. % $NaCl_1$, T_h = 60° to 295°C and CO_2-rich inclusions in healed fractures	Fluid unmixing intermittently
Yilgarn Block, Western Australia Ho et al. (1987)	Primary H_2O-CO_2 with both constant and variable phase proportions (20 to 30 mole % CO_2), < 2 equi. wt. % NaCl, T_h = 200° to 390°C	Early fluid, some unmixing
	Coexisting primary H_2O-rich and CO_2-rich in late vugs	Late fluid, heterogeneous

Source: Colvine et al., 1988.

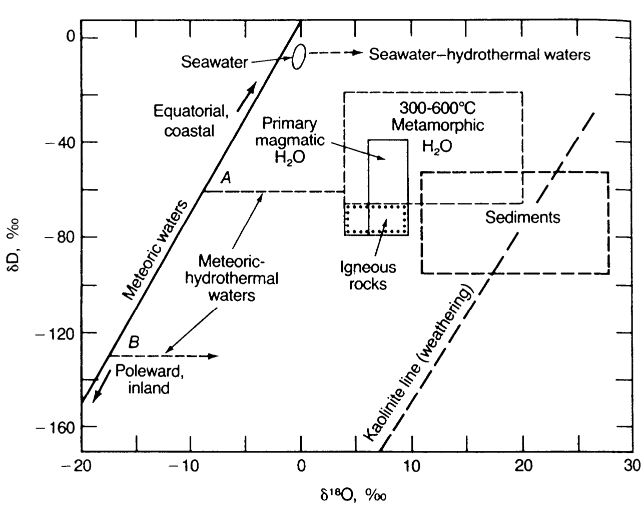

Fig. 4.5.20. Summary diagram of isotope composition of waters of different origins. Trends of ^{18}O shift due to water-rock interaction and exchange are shown for seawater and meteoric waters of compositions A and B (Guilbert and Park, 1986).

potassium-argon (^{40}K-^{40}Ar). Rubidium-strontium dating can be applied in the investigation of the source (mantle or crust) of the rocks related to mineral deposits and features of alteration. Potassium-argon dating is particularly applicable to determing the age of igneous rocks.

^{204}Pb is a stable isotope and the abundances of ^{206}Pb, ^{207}Pb, and ^{208}Pb have increased through the decay of uranium and thorium since the formation of the earth.

High-precision *uranium-lead* isotope analyses on zircon and titanite have been used extensively in dating rock units in the Canadian Shield and elsewhere. When combined with careful geological mapping and age dates from mineral deposits, these isotopic data provide a geochronological framework for determining the timing of mineralization episodes and the conditions of mineral deposit formation (Colvine et al., 1988).

Gulson (1986) reviews the growth and application of lead isotopes in exploration. This includes the development of the lead isotope method as a prospecting tool in the differentiation of significant gossans from false gossans in deeply weathered and oxidized terrain (Gulson and Mizon, 1979).

Lead isotope studies depend on the following assumptions (Richards, 1971):

1. At the time of the formation of the earth, there was a single set of lead isotope ratios throughout the earth mass, the so called primeval lead.

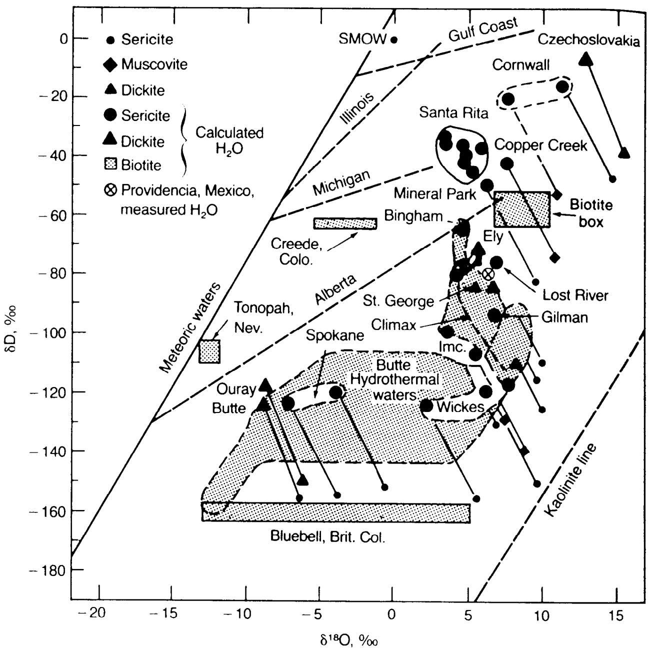

Fig. 4.5.21. Plot of δD vs. $\delta^{18}O$ for calculated hydrothermal waters from a variety of ore deposits, mainly from western North America. The stippled biotite "box" describes waters that would have coexisted in equilibrium with hydrothermal biotites from Ely, Bingham, and Santa Rita at 650°C. Also shown are the trend lines for oil-field formation waters from various sedimentary basins in the midcontinent of North America (Taylor, 1974).

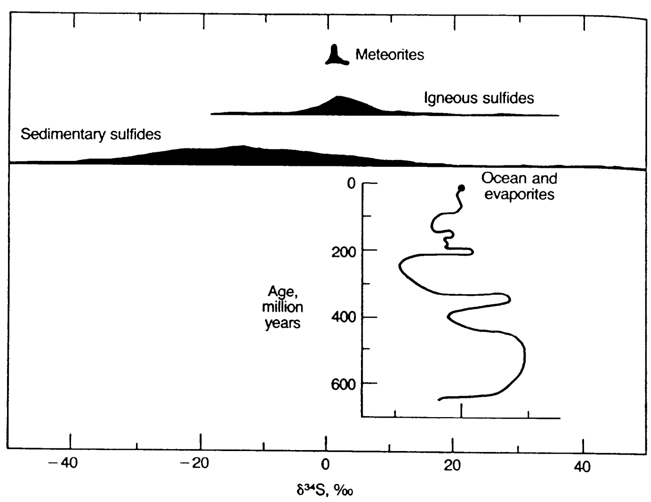

Fig. 4.5.22. Sulfur isotope variations in nature (Ohmoto and Rye, 1979).

2. Since then, all lead has been held in one or more closed systems with the proportions of uranium, thorium, and lead changed only by radioactive decay.

3. From time to time, batches of lead-bearing material have been removed from their source to form ore deposits, with negligible effect on what remains.

4. Ore leads so removed were not contaminated by other leads while traveling to the site of deposition; they reflect frozen original isotope abundances and ratios.

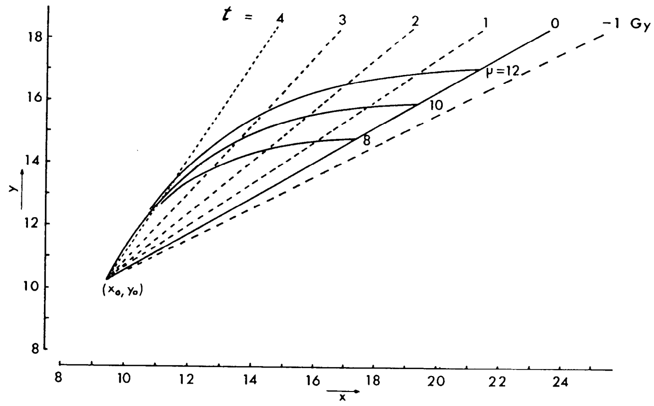

Fig. 4.5.23. Simple lead growth model. Growth curves for constant ($^{238}U/^{204}Pb$), variable time; straight-line isochrons for constant *t*, variable μ. Coordinates are $x = {}^{206}Pb / {}^{204}Pb$; $y = {}^{207}Pb / {}^{204}Pb$ (Richards, 1968, 1971).

The primeval lead value is calculated from the Canyon Diablo meteorite (which contains no uranium or thorium) at 4.55 billion years. Dating of leads is achieved by observing the positioning of measurements relative to growth curves of single stage leads and isochrons connecting points of equal geologic age (Fig. 4.5.23). An example of lead isotope ratios (and calculated age dates) plotting on a single stage growth curve is shown on Fig. 4.5.24.

4.5.2.5 Portable X-Ray Fluorescent Analyzers

Portable X-ray fluorescent analyzers (XRF) were initially developed in the 1960s (Bowie et al., 1965). The analyzers utilize a radioisotope source (Cf^{238}or Cd^{109}) to irradiate the sample material with gamma rays (Fig. 4.5.25). The resultant X-ray spectrum is selectively filtered, and the concentration of specific elements can be measured directly at the outcrop, the drill site, or the underground face.

The penetration of the sample surface by X-rays emitted by the radioactive source is very limited and generally less than 3 mm (0.1 in). The accuracy of any determination is, therefore, dependent on sample surface exposed to the detector. Measurements on unprepared rock surfaces, whether the sample is a rock face, a hand specimen, or piece of drill core is, at best, a qualitative measurement. To obtain quantitative results, it is necessary to homogenize and mix the sample through grinding and pulverization. The sensitivity of the portable XRF analyzers is generally inferior to the more conventional geochemical or assay techniques, but nevertheless, the positive identification of metals of interest in the field contributes to an increased prospecting effectiveness. The metals that can be identified and quantitatively measured by the portable analyzers include copper, tin, molybdenum, lead, zinc, nickel, iron, titanium, tungsten, barium, zirconium, and silver.

4.5.2.6 Laser Techniques

Excitation of minerals using a laser beam followed by the measurement of the resulting luminescent radiation is the basis of the *Luminex* exploration technique developed by Scintrex Ltd. for deposits of tungsten, zinc, molybdenum, gold, and other minerals. Ultraviolet sources are coupled with detectors particularly sensitive to responses from photoluminescent minerals. When mounted in a helicopter, the Luminex system can be used

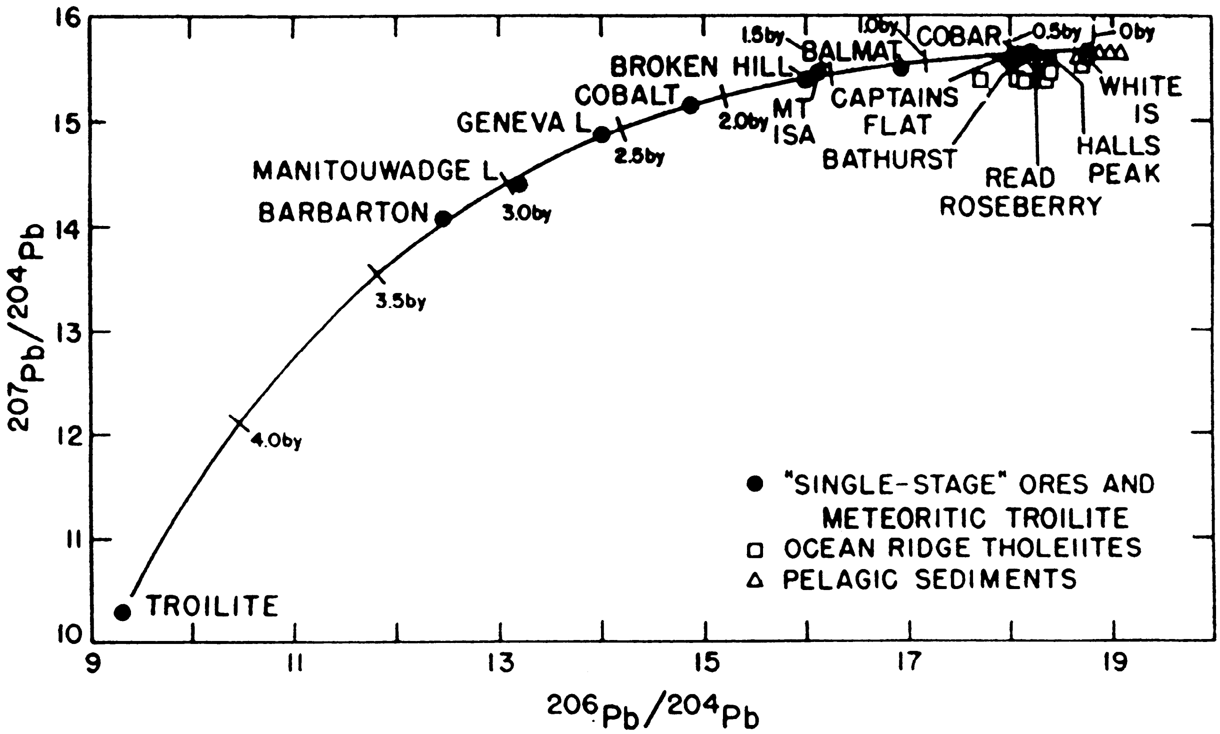

Fig. 4.5.24. Single-stage growth curve and lead isotopic ratios for uranogenic leads from selected ores, ocean-ridge tholeiites, and pelagic sediments. Parameters of the curve are: T_o = 4.43 b.y. (chosen to distribute errors in model-lead ages), ^{206}Pb / $^{204}Pb_0$ = 9.307, ^{207}Pb / $^{204}Pb_0$ = 10.294, and ^{238}U / ^{204}Pb = 9.58 (Doe and Stacey, 1974).

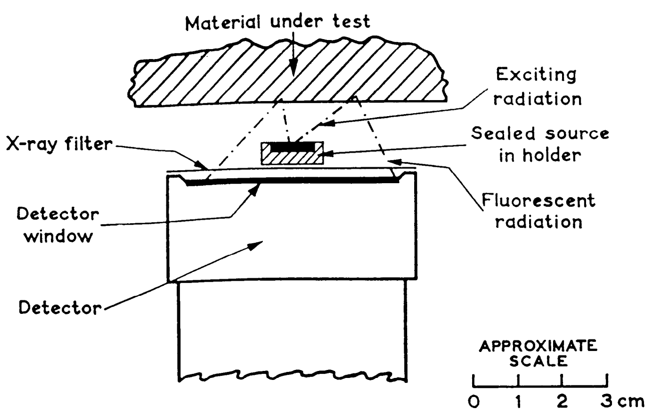

Fig. 4.5.25. Diagrammatic illustration of the relative positions of source, sample, filter, and detector of a portable radioisotope X-ray fluorescence analyser (Bowie et al., 1965). Conversion factor: 1 in. = 2.54 cm.

to detect and quantify responsive minerals occurring in outcrop, and it is probably the first of a series of prospecting methods that will be developed using laser technology.

ACKNOWLEDGMENTS

During the preparation of this summary, the author benefitted considerably from wise and pertinent comments from a large number of people. Sincere thanks and appreciation is extended to A.J. Bjorklund, D.W. Coulter, L.G. Closs, I.L. Elliott, W.K. Fletcher, G.J.S. Govett, H.E. Hawkes, W.C. Lacy, O.P. Lavin, PK. Theobald, I. Thomson, and E.F. Weiland. Mollie Uhl Eaton very efficiently organized the preparation of the manuscript, and Gary Moseley competently assisted with several of the illustrations. Their contributions are gratefully acknowledged.

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Chapter 4.6
MANAGEMENT AND NEGOTIATIONS

THOMAS N. WALTHIER (DECEASED)

4.6.1 INTRODUCTION

Exploration success depends upon a supportive corporate environment financing a well-managed exploration group. Qualified people using the most modern technology are essential. Luck is a negligible factor.

D.L. Kyle (1984) stated, ". . . *on average*, the return on all exploration in a region, by virtue of discovery value, is not adequate when considering the risks involved . . . Thoroughbred mining companies, however, show an exceptionally good rate of return on this investment . . . by applying a totally professional approach . . ." This requires "money, experience, perseverance, technical excellence, imagination, ideas, flexibility, time, confidence . . ." The author analyzes exploration risk.

Eisenbeis (1975, p. 8) writes, "Because of the great odds in making an economic discovery, an exploration department could conceivably operate inefficiently and ineffectively for many years before it becomes evident to upper management. In contrast, a well-managed creative exploration group . . . may find one or two acceptable ore bodies with an expenditure equal to the unproductive group."

There have been lucky discoveries—stumbling across an ore-grade outcrop, cutting ore when the drill site was misplaced on a survey error, etc. This is different from a consistently successful record of discovery where luck is not a significant factor (see also Bailly, 1979a). This chapter discusses organization and management that foster a high success rate and a high return on exploration dollars.

4.6.2. ROLE OF THE CORPORATION

4.6.2.1 Corporate Dedication

The board of directors and the president/CEO must truly believe that exploration is an essential part of a mining company. Exploration is necessary to replace the ore that is being mined and is equally important to the company's growth through the discovery of new profit centers.

Exploration must be recognized and accepted as a high risk, entrepreneurial activity not usually compatible with other sectors of the corporate organization. However, "Large mining companies tend to be bureaucratic (and therefore inherently conservative) enterprises engaged in a high-risk business—exploration . . . Well established management rules, which may be effective for mine management, cannot easily be applied to exploration . . . This built-in tension in working styles has always provided difficult management situations" (Anon., 1986). Wooda11 (1984a) writes, "The single most critical requirement for successful mineral exploration is the chain of confidence linking financial investors and Boards of Directors to the scientists and field operators."

Exploration is to be looked upon philosophically as an *investment* in the company's future and not an expense, no matter what the account books say. Funds for exploration, or a dividend, are equally important considerations.

If the person in charge of exploration is not a corporate officer, it indicates less than optimum company commitment. For the same reason, the incumbent should report directly to the CEO. Vice president-exploration (VP-Exp.) is a suitable title.

4.6.2.2 Objectives

Exploration guidelines should be clearly established:

1. *What do we look for?* Commodities sought should have attractive future markets as well as compatibility with the company's growth plans and expertise; i.e., long-range planning is part of exploration, and vice versa.
2. *Where?* In political jurisdictions that possess acceptable investment climates and an attractive discovery potential.
3. *What are acceptable company risks?* In the technical, financial, marketing areas?

4.6.2.3 Size of Program

Availability of funds is the long-term (and usually short-term) limitation.

Funding via equity investors is limited in amount that can be raised and is not usually repeatable without short-term success.

Debt is rarely available to fund exploration.

Income from operating mines is the most secure source of long-term exploration funds.

Two other restrictions on size of the program are

1. Competent staff, especially supervisory staff.
2. Availability of good prospects.

A competent staff can generate more good properties than there are funds available to test them.

Exploration in remote areas, and in foreign countries, adds to the cost and reduces the effective amount of exploration that can be done on the same money. Supervision becomes more difficult and complex.

4.6.2.4 Transition from Exploration to Operations

Once a discovery is made, further testing becomes less and less an exploration process, whereas engineering factors become dominant. At some point, it is appropriate to transfer responsibility to a group with an operating orientation. This is a crucial time, and the CEO must devote much attention to make sure the transition is timely and smooth. It may not be easy.

The exploration group will have developed a paternalistic attitude toward "its" discovery and may resent operations taking over. The latter will have no sentimental attachment to the new deposit. They must search for uncertainties and deficiencies in the data that have to be resolved before they will assume responsibility for a production decision. This can appear to be (and may be) unnecessarily critical. Geological ore reserves are always larger than initial engineering minable reserves. Consequently, the size and even viability of the discovery may be hotly disputed.

Intimate knowledge of the underlying contractual agreements relative to the deposit are initially not known by the operators. Commitments may be overlooked, contractual deadlines missed. Accusations of mismanagement may pass between the two groups.

To facilitate the transition, the CEO must personally see to the gradual involvement of the operating group as a discovery emerges. A separate *Mine Development Group* has been found to be successful (Regan, 1971, pp. 33-39).

4.6.2.5 Budget

Annually, the company prepares a financial plan for the coming year, probably combined with a five-year plan. Based upon these projections, the board of directors and CEO allocate a certain sum to fund the next year's exploration, usually in consultation with the VP-Exp. Emphasis on certain commodities and allocation by countries is stated. The VP-Exp., with the various exploration managers, choose those reconnaissance programs and more advanced projects that optimize the promise of discovery within the funds allowed. Work plans are prepared with estimated costs. The final budget presented by the VP-Exp. is normally the last technical review. The CEO and board concentrate on the broad objectives and whether there is a balance among commodities, political distribution, and reconnaissance vs. advanced projects.

The overall magnitude of the exploration budget is limited by a combination of funds available, corporate philosophy, and alternative investment opportunities. Moreover, there are practical limitations: above a certain budget level, an exploration group appears to lose its effectiveness—likely because of increase in bureaucracy, dogma, overstaffing, and overorganization. Conversely, below a certain level of funding, exploration does not have enough exposure, ability to attract staff, acquire quality prospects, or persevere long enough (see also Bailly, 1973).

What is a minimum-sized program that has a reasonable expectation of success? Boyd (1967) and Morgan (1969) estimated a minimum of $1 to $3 million annually for five years were required. Adjusting these figures for inflation to current 1989 dollars raises the expenditures to about $3.5 to $10 million annually, or a total commitment between $15 to $50 million. Woodall (1984a) states, "The study showed that the average cost of finding an economic metal deposit (between 1955 and 1978) in Australia was A$38 million" (in 1980 dollars). Excluding Western Mining Corp. from the statistics, the average cost increased to A$54 million (Woodall, 1984b). He found that in Canada (1946-1979) the discovery cost calculates to C$20 million, Woodall adds that two to three times this may have to be spent to increase the probability of success to 90%. Assuming an efficient and effective exploration program, a cost of US$30 to $50 million per discovery is generally accepted today. A program of $10 million annually for five years should be more effective than $17 million for only three years or $2 million for 25 years.

When the board approves the budget, part of it may be only contingent approval—depending upon favorable results of the next ensuing phase of exploration.

It is important that the VP-Exp. have some freedom to shift funds between various reconnaissance programs and drilling projects. The money is shifted away from projects that deteriorate in potential into those with improved expectations. Without the authority to move money about quickly to meet changing priorities can so frustrate attempts to better-the-competition that the exploration program suffers significantly. Overall limits in the ability to shift funds may be set by the CEO (such as 5 to 10% of the total budget).

4.6.2.5 Accounting Practice

How expenditures for exploration and mine development are shown on the account books have a marked effect on the company's P&L statement, balance sheet, taxes, and cost of production.

Consider three companies exploring the same prospect (The Prospect). Company A has no operating income but raised $1 million selling stock. Companies B and C have operating mines, which were developed by borrowing $15 million in addition to selling $1 million shares. Operating revenue is $4 million and production costs $2 million. Tax rate is 50%.

Balance Sheets, Beginning of Year

	A	B	C
Assets			
Cash	$1,000,000	$ 1,000,000	$ 1,000,000
Mine (book value)	0	15,000,000	15,000,000
Liabilities			
Debt	0	15,000,000	15,000,000
Equity	1,000,000	1,000,000	1,000,000

During the year, companies A and C consider they have made a discovery at The Prospect after spending $100,000 and proceed to spend $900,000 more on "mine development" than is capitalized. Company B questions the viability of The Prospect after spending $1 million. The changes in financial positions (simplified—omitting depreciation, et al.) during the year are

	A	B	C
Revenue	0	$ 4,000,000	$ 4,000,000
Operating costs	0	(2,000,000)	(2,000,000)
Operating profit	0	2,000,000	2,000,000
Exploration expense	$ (100,000)	(1,000,000)	(100,000)
Taxable income	0	1,000,000	1,900,000
50% income tax	0	(500,000)	(950,000)
After tax profit	0	500,000	950,000
Mine development	900,000	0	900,000

Year End Balance Sheets

	A	B	C
Assets			
Cash	0	$ 1,500,000	$ 1,050,000
Mine (book value)	0	15,000,000	15,000,000
Mine development	$ 900,000	0	900,000
Liabilities			
Debt	0	15,000,000	15,000,000
Equity	900,000	1,500,000	1,500,000

All three companies have used acceptable accounting procedures. They differ because of the subjective decision of what constitutes a commercial discovery (see also Brown, 1983).

Normally, in the long run, all three companies will end up more or less the same. If the prospect is a "real" discovery, the same book value will eventually appear on all the books. (Company B will have to recapture prior "expenses" and capitalize them.) However, if the occurrence fails to prove commercial, it should disappear on all the companies' books, incurring, perhaps sizable, write-offs in companies A and C. However, company A or C can decide not to write off The Prospect, thereby showing a higher net worth and avoiding a difficult write-off, on the belief, or ruse, that The Prospect will be minable "someday."

The three companies were keeping their accounts consistent with their own needs, discovery criteria, and legal/accounting restrictions. In joint ventures, or with minority partners, the participants may have widely different views on what constitutes a discovery and how the accounts should be managed. Serious disputes can arise.

Once a mine is in production, exploration near the mine to extend reserves is expensed; it is no longer capitalized and is considered a mine production cost.

But how close is close to the mine? It is a subjective call whether exploration in the vicinity of the mine complex is "close enough" to be a production expense or outside exploration. If exploration close-in is nevertheless called "outside exploration," it will give the appearance of lower production costs plus a more aggressive exploration program. A guide is that if any ore that might be found would (probably) be processed in the existing milling facility, it should be considered "close enough."

4.6.2.7 Criteria of Successful Exploration

Evaluating exploration success is difficult because of the absence of measurable parameters. A team may use all the appropriate exploration procedures in the correct way, most of the money going "into the ground," yet fail to find ore. They are obviously not successful, but where is the problem and how to evaluate it? Even if successful, have they been sufficiently successful?

The immediately preceding accounting example illustrates some of the evaluation difficulties. Companies A and C each found a commercial discovery after spending $100,000. Company B, by its criteria, did not after spending $1 million—yet they have all found exactly the same thing. Is company B less successful?

The exploration industry in Australia (1955-1978) was studied as to cost, number of discoveries, and cost effectiveness of exploration (Mackenzie and Bilodeau, 1984). Woodall (1984a,b,c) compares this with a similar Canadian study, 1946-1977. Cranston (1980, p. 2) points out that in any such study, the number of tabulated discoveries is less than what eventually will be counted. He states, "As times passes, yet more ore bodies may turn out to be discovered during that interval and the recognized *size* of the currently known ore bodies discovered in those year may become even larger."

In summary, the average cost per discovery is, at best, a crude yardstick of the effectiveness of an exploration group compared to industry as a whole. There are many variables.

A comprehensive worldwide review of exploration expenditures, cost of discovery, etc., is covered in Tilton et al. (1988, Chap. 2).

As noted, the number of discoveries is often used to measure success. But are five small ore bodies worse, equal to, or better than one major discovery? Suppose the sponsoring company is only interested in the "big one;" is that realistic? How often can such a discovery be expected? A low success rate is built in by such corporate guidelines and is not necessarily a measure of incompetence of the exploration team. Likewise, profitability has been suggested as a guide, but the profitability of a mine depends on many important factors such as market price of the product, operating skill, grade, tax and royalty structure, etc., *none* of which measures exploration proficiency.

The best criteria of successful exploration are

1. Is enough ore being found to (a) replenish ore as it is mined and (b) generate new profit centers in line with desired corporate growth?
2. Is ore being found cheaper than buying equivalent reserves?
3. The cost of discovery per ton of ore, or unit of salable product, should not be a material burden on profitability.
4. During the exploration stage, are recognizable fatal flaws and weak links identified and resolved early before a lot of money is spent?
5. Is cooperation good with other corporate departments?
6. Are long-range corporate goals and needs known throughout the exploration staff and do they appreciate their role in achieving them?
7. Are exploration programs, and results, explained clearly and concisely to top management?
8. Is there good budget control?
9. Is morale good?
10. What percentage of (production) (earnings) (revenue) (growth) is derived from new discoveries within the last (5) (10) (?) years? If low, this shows something is wrong, but it begs the question of "why?"

4.6.2.8 Compensation

Small exploration companies will generally approach remuneration differently from a large, mature-operating mining company. In the former, exploration success is vital to survival. Cash is in short supply. Salary and fringe benefits will be lower but offset with noncash items like large stock options and deferred expenditures like discovery bonuses. The potential reward from success must be sufficient to entice quality people away from a more secure good salary, fringe benefits, and a comfortable retirement.

Exploration is not such a vital element in a large operating company. Base salaries and fringes will be standardized company-wide, based on industry averages. Stock options may exist but will extend down equally in all corporate departments, hence are not per se a reward *for discovery*. There might be an ad hoc reward for a significant discovery. If so, it most likely will be treated confidentially and hence is not a motivation to others.

In granting discovery awards, care must be taken to ensure that *all* employees who made a material contribution are recognized. Others may think they were contributors, but really were not. They will be upset at being omitted. Some companies take the easy way out and eliminate discovery bonuses because it is difficult to determine who should get what.

There is some evidence (Regan, 1971) that a policy of extra, or special, compensation for a discovery does not *motivate* top quality professionals although such incentives can attract excellent people to the company and help retain them.

To summarize, an anonymous exploration executive has said, "Successful exploration *requires* other things beyond a good exploration approach."

4.6.3 ROLE OF THE EXPLORATION DEPARTMENT

4.6.3.1 Staff

This will comprise "support" and "technical" people. The former consist of clerical, accounting, legal, etc., and are similar in background and training to those in most business departments, the management of which has been discussed many times. The rest of this segment, therefore, will concentrate on management of the technical staff.

4.6.3.2 Technical Staff

Qualifications: A strong background in the earth sciences is essential, with particular emphasis on descriptive geology of ore deposits plus up-to-date concepts on ore genesis. A thorough understanding of modern exploration technology is crucial. Knowledge of ore bodies should encompass many types and kinds as this expands the explorer's ken to the vast number of geologic processes that can concentrate minerals and metals into minable deposits. It minimizes parochial thinking. In addition, all the staff should have an understanding of the principles of geochemistry and geophysics and the fundamentals of mining methods and extractive metallurgy. There should be a good working knowledge of the laws and regulations governing mineral rights and mining in those areas where exploration is being carried out.

Exploration makes great use of geology, but in applying geology, its role is restricted solely to finding ore and making money. This is quite different from running a geological survey or speculating on the composition of the earth's core. See also Ohle and Bates (1981).

A dedication must exist to finding ore. Most of the person's professional career should have been spent involved in the discovery and acquisition of mineral deposits.

As Parker (1988) has said, "The individual explorer must be personally committed to the general exploration program and should work in areas with which he/she has geological familiarity."

Experience in mining operations is useful, but extensive exposure to operations accentuates an engineering desire for precision, certainty, and risk avoidance. These are not the dominating personal traits of a successful explorationist.

In addition to a good technical background, the mineral explorer should have an entrepreneurial flair. It is necessary to have the courage to commit, on a continuing basis, large sums of company money where only sparse and indirect evidence exist, relying upon experience and judgment that the potential rewards are compensatingly large. It demands a personality comfortable working in this environment. This does not imply the person is reckless and flamboyant. He is *not* a gambler, but a calculating risk taker employing the most modern technology and concepts to improve the possibility of discovery.

Bailly (1979b) discusses the personality of a successful explorationist. "Good exploration people are rarely among the best of earth scientists; they are just keen on using their . . . knowledge . . . to find new deposits." Bailly further quotes Drucker (1964) that the purpose of staffing is not to minimize weaknesses but to maximize strengths. People that perform well (i.e., discover ore) are *the key* people, even though they may have weaknesses-technical or personal.

Past participation in making a discovery is almost essential: "The more ore deposits you are involved in finding, the better you get at it" (Kohls, 1988; Bailly, 1979a).

Exploration *managers* are more difficult to find than good field people. Eisenbeis (1975) discusses this at great length. He quotes Katz (1954, p. 90) that the engineer (scientist)-manager must have three managerial attributes: technical, human, conceptual. "Human" means the ability to work as a group member and leader and inspire teamwork and enthusiasm. Conceptual skill is the ability to see the total picture (company goals, resources, and political structure) and be able to operate in that environment. The VP-Exp. is the key post that integrates the technically oriented explorers into the company environment. Fellows (1968) lists inherent conflict areas between managers and technical professionals. Some are

1. Professionals will not accept direction from low-ranking nontechnical people.
2. Technical people often look upon "management" as undisciplined politicians.
3. Even a technically trained manager must use diplomacy in giving directions to professional subordinates.

The single most important duty of the VP-Exp. is the selection of staff. "His main discovery contribution, therefore, is to select and to retain only *outstanding* people in order to obtain the best technical work management at all levels" (Bailly, 1973). [Emphasis added.]

Another critical task is "to ensure that a favorable idea-generating and idea-nurturing climate is maintained." This takes effort; it seldom occurs accidentally. It is easier in some companies than in others (Regan, 1971).

The top manager of exploration, the VP-Exp. must also be acutely aware of the corporation's tolerance for risk taking, which varies widely among companies, or with a change in management.

The VP-Exp. must also have the ability (and courage) to bring to a prompt end projects that were once exciting favorites that have lost their potential.

4.6.3.3 Delegating Authority

Much field work is done by geologists acting independently in widely scattered areas without good communication. This requires a large amount of delegation of authority and responsibility by the VP-Exp. and the regional exploration managers. Clear guidelines must be given on commodities to be looked for, acceptable terms for the acquisition of mineral rights, minimum size of deposit sought, what are acceptable risks, and what are the limits to the delegated powers. A large amount of travel by the supervisors will be necessary to preserve control.

Rose and Eggert (Tilton et al., 1988) have recognized two fundamental management styles in exploration groups: (1) exploration managers who dominate the scope and direction of the program and (2) exploration managers that lead a team of specialists. They conclude that the former appears to be more successful with the proviso that the manager must be competent or the whole thing fails. "Dominate" means inspiration and leadership, *not* deciding every minute detail.

Good morale is essential, both individually and as a group. "Good" morale equates with satisfaction in attempting to achieve the company's objectives. Lack of such commitment constitutes "low" morale. First, the corporate goals should be consistent, clearly defined, and surrounded with a sense of importance and urgency (Bailly, 1979b). Furthermore, Read (1962) and Shepard (1961) have concluded that unsuccessful groups historically are very dogmatic, highly critical of the company and of members within the group.

4.6.3.4 Special Considerations

Every mineral commodity has its unique set of technical, economic, and marketing factors that impact directly on what constitutes a commercial ore body. Base and precious metal deposits are fairly straightforward, with tonnage, grade, metallurgical recovery, and infrastructure availability the major considerations. Other minerals, especially the nonmetallics, may have very critical texture, impurity, color, or location restrictions that outweigh almost everything else in evaluating commercial viability. Do not start exploration for a new commodity until a thorough study has been made as to what special factors may dominate the assessment. Special knowledge, via consultants, usually is needed.

4.6.3.5 Control of Mineral Rights

There is nothing more embarrassing and unforgivable than to make an important discovery only to find it belongs to someone else (see Chapters 3.2 and 7.1).

Absolute legal ownership of the minerals must precede discovery, which means unequivocal control occurs very early in the exploration process.

The basic procedures for obtaining control are established by the laws and regulations of the country, especially the mining law. The rights and obligations of the explorer and miner must have been thoroughly understood. This requires a competent local legal advisor.

Mineral rights may be owned privately or by the government; similarly with the associated surface rights. Private mineral and surface rights may be owned by the same or different parties.

In most political jurisdictions, mineral rights are considered senior to surface rights, which means mining (and by inference, exploration) may proceed without consent of the surface owner. Courtesy, as a minimum, requires asking permission to trespass. However, the miner must compensate for damage to, or the loss of, the surface. The compensation may be set by law or it may be left for the parties to negotiate. Agreeing to the damage amounts should be established before exploration starts and loss occurs.

Private Ownership: Property ownership can be sold, leased, or passed on by inheritance. After several generations, ownership may be widespread with many heirs unknown. If surface and mineral rights are separate, the problem is compounded. However, surface rights usually incur property taxes, so at least the heirs that are paying taxes form a good starting point in tracing ownership. Separated mineral rights, in many areas, are not taxed, and the search for unknown heirs and owners (who are often ignorant of their inheritance) often approaches the impossible. Consult local legal counsel if there are procedures for extinguishing the rights of lost owners. Some countries (Ireland, for example) have started taxing mineral rights. If taxes are not paid, the mineral rights revert to the state. This quickly sorted out mineral ownership, which had been a horrendous quagmire in Ireland. The change was subsequently followed by an exploration boom.

Once ownership is determined, negotiations commence, culminating in a written contract giving the explorer the right to test for ore and, if found, conveys the irrevocable right to exploit the discovery. Compensation to the owner of the minerals is set out. See 4.6.5 for details. Compensation to a different surface owner should also be resolved at this time.

If a third party has obtained mineral rights directly from a government (see below), the ensuing negotiations will be carried out as if the third party were a private owner. The new group, however, will have to assume all the obligations and commitments that the third party agreed to with the government. Many governments allow the free transfer of mineral rights, but sometimes the transferee must show technical and financial competence.

Government-Owned Minerals: Minerals are available by either

1. Laying claim to a tract of land (maximum size specified) by posting notice in the field and recording a copy in the appropriate government office. This is called staking or pegging a claim. Annual work commitments and/or rentals are imposed to maintain the claims in good standing. The details of staking and maintaining claims can be complex and vary state to state and province to province. Seek local legal advise on the detailed procedures. Even minor variations in carrying out the procedures may void the claim. The law and procedures pertaining to staking, filing, and maintaining claims in the US are summarized by Pruitt (1987).

2. Issuance by the government of a concession to qualified applicants permitting exploring and mining. Laws and regulations itemize the conditions and obligations. Concessions can cover large areas measured in square miles (square kilometers) as opposed to individual claims that usually encompass a few acres (hectares). Because concessions tend to be large, there are usually stipulations that a certain percentage must be dropped after specified time periods, such as 50% every two or three years. Annual work obligations and rentals are common. The terms and conditions may be standard and spelled out, or they may be individually negotiated at the time the concession is requested.

A government may consider a parcel of ground to be so attractive that it will call for an auction, requesting interested parties from around the world to submit (sealed) bids. While this is common for oil, gas, and coal concessions, it is rare for hard minerals.

4.6.3.6 Water Rights

Almost every operating mine requires a secure source of process water. Water rights may be scarce and considered more valuable than surface rights, hence very difficult and expensive to acquire.

Rights to water may be privately or government owned. All available water may be already allocated to farms (irrigation), factories, and towns. If so, somebody's rights will have to be bought. An alternative can be to set up an exploration program to discover a new underground source. Even if successful, government authorities may have a strong voice in how the discovered water will be allocated.

As soon as a prospect begins to show signs of being commercially viable, start securing water rights via options to purchase, filing for an allocation of undedicated rights, or assessing the possibilities for discovering a new aquifer.

4.6.3.7 Land Records

Any sizable exploration program will generate a large number of documents covering staked claims, joint ventures, leases, concessions, and options to purchase or lease. There will be a bewildering assortment of terms and conditions and a variety of deadlines and commitment dates. An emerging valuable discovery can be lost if a commitment is overlooked.

A systematic procedure to monitor all obligations and contracts is essential. It should be the responsibility of one person. In some large companies, a separate land department is established. However, the person in charge of a field project must also be aware of all the obligations so that work commitments are completed in a timely manner and rental payments can be justified.

Deadlines, payment dates, and commitments should be programmed into a computer retrieval system such that they are flagged in adequate time for the field people to generate the data needed to justify the next commitment. Large monetary commitments may need more advance warning if higher management review and approval is necessary.

4.6.4 EXPLORING IN FOREIGN COUNTRIES

Exploring abroad is more complex than in the home country—and usually highly complex. It is also more expensive.

Much of what follows relates to the operating stage of a mine in the foreign country. However, it is almost always the exploration division that makes the initial contacts in the country. It bears the responsibility for making certain that activities in the country are properly organized for the entire period the company will be active there.

4.6.4.1 Selecting the Foreign Country

Two fundamental considerations are

1. The political and financial climate must be acceptable and appear stable for many years.

2. The geologic setting, and evidence of mineral, is promising for the minerals being sought.

Beware of countries that have been traditionally negative about foreign interests mining their national patrimony and then, suddenly, espouse an enthusiasm for it. Perhaps the change is real and lasting, but inbred attitudes can resurface rapidly and investment incentive laws be repealed. This can coincide when sizable profits are about to be remitted or with a change of government.

Assessing the political risk of a country as a place to invest is a very difficult task. It should not be the result of a few casual trips, a visit to a local attorney, banker, and your embassy plus touristic literature. It involves an in-depth analysis of the political scene, the country's history, and how that has influenced the current socioeconomic scene. Is the regime stable and likely to last? Is the economy strong? What are the people's *real* attitude on foreign investment? It is difficult to get up-to-date, informed, unbiased data and opinions on these issues. Coplin and O'Leary (1987) discuss how to collect objective data and various do's and don't's to arrive at a reasonable evaluation.

4.6.4.2 Initial Steps

Having chosen the country, the first chore is to find a good attorney. Then, in most countries, forget about exploration for six months to a year until all the formalities related to starting a mining business are taken care of.

Performing serious exploration on tourist visas can be a serious violation of the immigration laws. Start promptly to obtain proper work visas.

4.6.4.3 Selecting an Attorney

Recommendations can come from your embassy, home country law firm with international connections, branch of a foreign bank, CPA firm, or foreign firms operating there.

Qualifications include

1. Fluency in your language as well as the local one.

2. Comprehends how the laws and tax regime of his country must mesh with your home country for combined maximum benefit.

3. Is well respected for his legal ability.

4. Has easy access to top political officials of all major parties but officially is politically neutral and noncontroversial.

4.6.4.4 Corporate Structure

Determine which legal vehicles are recognized in the country, for example, branch of the home company, limited partnership, joint venture, corporation, etc. The one that is selected by top management will be on the advice of tax, legal, and financial experts and will have three objectives

1. Maximize remittable profits.

2. Minimize the *combined* tax burden of the foreign and home countries.

3. Management flexibility and control.

This is a very complex issue and may take months to resolve.

The corporate structure is not irreversible, but subsequent changes may incur large costs in capital transfer and corporate reorganization fees as well as legal costs. Deemed income may be assessed on the shifting of corporate assets, thereby incurring sizable income and VAT (value added taxes).

4.6.4.5 Foreign Investment Agreement

Some developing countries will enter into a contract with a potential foreign investor, guaranteeing certain financial conditions to encourage the investment.

These agreements are complex documents and tedious to negotiate, but are usually considered worth the effort.

Topics that may be included are

1. Guarantees will last for a period of time, say, 25 years, life of mine, etc.

2. Guarantees will cover a stated amount of foreign investment.

3. Specify a fixed income tax rate.

4. No additional, or discriminatory, taxes, fees, or charges will be imposed.

5. Right to foreign exchange at most favorable exchange rate on day of transaction to service debt, pay foreign bills, repatriate capital, and freely pay dividends.

6. Exoneration of customs duties and relief from bureaucratic delays.

7. Number, and work conditions, of expatriate employees.

8. The operation will be treated and governed as any other business in the country. The general commercial code, general labor law, etc., will govern. There will be no discrimination in regulation of the operation.

9. If restrictions are placed on the amount of product that can be exported, the price received on prescribed domestic sales will be no less than the net amount that could have been received from the most attractive sale abroad.

10. In case of expropriation, the conditions of compensation are spelled out, and specifically how the value is to be calculated.

Which terms are included and how inclusive they are will depend upon the skill of the negotiators as well as restrictions contained in the law or decree authorizing such agreements.

4.6.4.6 Staffing the Foreign Program

Qualifications: The qualifications of the domestic exploration staff apply equally here (see 4.6.3a), but more things must be considered.

Supervision will be less frequent.

In much of the world, infrastructure will be less developed. Service industries support will be poor or absent. Considerable time will be spent improvising or attempting to import goods and services, which in the home country can be accomplished with a local phone call.

Language differences may mean that only bilingual people can be employed; hence fewer candidates can be considered.

Living conditions (for example, housing, food, schooling, medical care) may be inadequate. Many spouses and children may not readily accept the environment. Some qualified job candidates will decline for these reasons. Any foreign assignment is disruptive to a family unit. Some people will "make do" for a while, but only a small percentage really enjoy it and make it a career.

Some companies feel strongly that their senior representative in a foreign country should be an expatriate, rather than a local citizen. This stems from a feeling that a national may have a conflict of interest between what is best for the company vs. what is best for the country or local employees. It is perceived that the local government can put more pressure on a national to take (or not take) certain actions, thereby compromising the company.

It may be that local citizens are not available, either because they are poorly trained or happily employed. In many cultures, manual work is demeaning to anyone with a college education. Local geologists may look upon field work as unacceptable manual labor.

In any event, expatriates may have to fill in.

Compensation: Since candidates and families ready, willing, and able to work overseas are harder to find, qualified ones are enticed and rewarded with higher remuneration.

A number of consulting firms keep records on overseas compensation packages currently in vogue with multi-national companies. They can tailor-make a program to meet your needs.

Overseas benefits are usually based on a percentage of a base salary, which is taken as the salary that would be paid for a comparable position at home.

Common benefits and perquisites are

1. Overseas bonus, calculated as a percentage of base salary (15% is a typical number).
2. Hardship premium (percentage of base), if applicable.
3. Reimbursement of income taxes over and above what employee would have paid in home country on base salary. Only company-derived income and perks are considered in calculation of the reimbursable amount.
4. Cost of living adjustment: a payment so the employee may approximate the same life style overseas as if at home.
5. Housing allowance: may be included in the foregoing, but housing must also be an appropriate standard for the people that represent the company.
6. Company-provided car. Because of import duties, vehicles may be two or three times more expensive. When driving conditions are difficult, a chauffeur may also be provided.
7. Tuition, books, lodging, and travel, if necessary, for children through secondary school. Costs are reimbursed for the closest home-language school that meets home country college entrance requirements.
8. Payment to transport household goods to and from the assignment.
9. Annual paid trip for the family back to home country for a vacation, often of one month's duration.
10. In case of serious medical problems, paid trip(s) to a medical facility meeting home country standards.

The foregoing benefits will at least double the cost of employment overseas. Not all of the foregoing are necessary if the country of assignment has a high-quality life style and a cultural background similar to the home country.

Employees are encouraged to take only as much salary in local currency as is needed for current living. The balance will be paid into a home country bank account. Paying split salaries can cause local accounting and tax problems; check with an appropriate legal and accounting counsel. The company is not responsible for foreign exchange losses if local currency is accumulated, nor is the company responsible for losses incurred on local investments (including any home purchase).

4.6.4.7 General Conditions

In many parts of the world, infrastructure is poor to essentially absent. Travel to the field can be slow and arduous. The geologic framework may be poorly known or incorrectly interpreted. In these cases, exploration will proceed slowly. Support facilities and exploration contractors may be missing. If local ones exist (for example, an assay house), the government may insist they be used instead of foreign facilities, no matter how poor the quality and reliability.

4.6.4.8 Discovery is Made

Prior to discovery, the local exploration manager (LEXM) has been the senior representative of the company, dealing with the top government officials and enjoying high-level business contacts and an attractive social life. When parent company executives visit the country to review the exploration developments, the LEXM is host and the focal point of the visit.

Then the important discovery is made.

Before long, an operations general manager (OGM) is appointed and stationed locally. The OPM becomes the senior representative of the company with all that it implies. The LEXM is lower in the management structure because of what has been built over and around him. Further exploration may be curtailed until the cash drain from building the new mine-mill complex is completed and exploration can be financed out of locally generated cash flow.

When resumed, exploration becomes an *expense* against mine revenue, increasing the costs of the new operating mining company. Because of this, the OPM will seek a strong voice in directing further exploration. The CEO may eventually have to become involved to resolve company in-fighting.. This scenario need not happen, but without cooperation among the LEXM, OPM, and VP-Exp., and an attentive CEO, the situation can get out of hand.

If the LEXM's morale deteriorates, he might be transferred to a new country. This is expensive and his local experience is lost. There is a practical limit to how many times this solution can be used. The LEXM can be brought home to the executive office as, perhaps, Assistant VP-Exp. This is expensive; local expertise has to be rebuilt, and it has limited repeat possibilities.

Something should be done, or the LEXM can be lost to a competitor.

The best solution is for the LEXM to be made part of, and truly treated as, a key member of the local company's *management team.*

This must have the full backing of the CEO, OPM, and the head of international operations. The LEXM can contribute significantly to the preproduction mine development, even should continuing exploration be deferred.

4.6.5 NEGOTIATIONS

Many transactions are involved in the exploration business that require extensive negotiations. Those that are most crucial are the negotiations related to the acquisition of mineral, water, and surface rights, and these are the subject of this subchapter.

All contracts involving mineral and surface rights must be written and signed by both parties before they are binding and effective. A precise legal description of the property is essential, and the whole agreement depends upon proof of clear and merchantable title.

While outright purchase contracts are made for mineral rights, the usual deal involves an option to either purchase or lease. The option period allows time to test the area and justify the previously agreed to purchase price or lease terms as well as a final verification of title.

A valuable guide is found in the *Landman's Legal Handbook—A PracticaI Guide to Mineral Leasing,* published by the Rocky Mountain Law Foundation (Anon., 1982). See also Maley (1983).

The Rocky Mountain Mineral Law Foundation has prepared a compendium on US mining laws, regulations, proceedings, and pitfalls entitled *American Law of Mining* (Anon., 1984). It is updated with yearly supplements. Since July 1955, the same

Foundation has sponsored annual sessions focusing on current legal problems facing the US mining industry. The results are published under the title *Rocky Mountain Mineral Law Institute.* The Foundation has also held two workshop sessions on all types of contracts used in the industry: lease agreements, drill contracts, consulting arrangements, joint venture contracts with a section on accounting practices, etc. These two publications entitled *Mining Agreements, Institutes I and II,* are more general in content and have application beyond this country (Anon., 1979, 1981) (also see Chapter 3.2).

4.6.5.1 Nature of the Mineral Rights Market

Mineral commodities are cyclical in demand with accompanying variations in prices. When prices are high, exploration for that commodity accelerates, and it will be concentrated in those areas with high discovery potential and a good investment climate.

The demand for the right to explore will surge and wane accordingly. The terms and conditions explorers will pay will fluctuate to the same rhythm. In a loose sense, there is a market for mineral rights. One has to meet the "going price" being offered by the other companies active in the area. This equivalency of offers is largely subjective as no two tracts of land are precisely the same in size or geologic attraction, but perceived relative equivalence of offers does exist nevertheless.

Appreciably more attractive acquisition terms can be obtained by exploring areas off the beaten track. This is offset by the usually much higher discovery cost, or political risk, in working in a virgin area.

4.6.5.2 Acquiring Partly/Fully Developed Ore Bodies

The asking price will be substantial, to reimburse for the high-risk early exploration that was paid out, plus the expectation of a reward for having made the discovery. A sizable down payment is usually involved, and the other financial conditions are higher than if unexplored ground were involved. A deal will be possible only if the seller's lowest acceptable price overlaps the buyer's highest evaluation.

The property may be sold in its entirety, or only partial divestiture may be under consideration.

Before the deal is consummated, the buyer will want time to do a "due diligence" evaluation. This allows time to verify title and confirm the previous work, especially ore reserve calculations and metallurgical recoveries. A determination will be made of what government clearances and permits are in hand and those that may be needed in addition. In short, verify what has been done, outline what has to be done, and determine if there are any weak links that must be resolved. Almost certainly a major consideration will be the potential for discovering considerably more ore.

The foregoing assumes that the prior work was done properly. Problems arise if that is not the case. The prospective buyer will not want to reimburse exploration costs that were poorly planned or incompetently carried out. Critical backup data may have been lost, or the results appear suspect, and the work has to be duplicated. If the property is attractive enough, reimbursement may just have to be considered part of the purchase price, attributing little if any value to the prior results.

A corollary is that owners of minerals should not test their property unless it is done in a professional manner by experts and all the results preserved in a businesslike way.

4.6.5.3 Acquiring Undeveloped Properties

These are properties possessing only indirect evidence of ore. The potential buyer will put up the front-end exploration money and take the high risk that something will be found. Substantial front-end payments are not justified until there has been time for exploration to indicate the existence of commercial mineralization. An option agreement provides the time to do the required exploration.

The duration of the option period (commonly in the three- to five-year range, but remote, rugged terrains, or the search for deeply buried ore may double this period). The amount and schedule of rentals, the size of work obligations, and the purchase price and manner of payment, or the lease terms, are negotiated and then incorporated into a formal contract.

4.6.5.4 Manner of Payment

Mineral properties can be paid for in several ways

1. Cash: *Lump sum.*

   Installments, usually tied to start-up of the mine with small interim payments.
2. Company stock of the buyer.
3. By royalty payments: kinds of royalties are

   Advance, before production starts, and these are in reality rentals; they may or may not be credited (in whole or in part) against future production royalties.

   Production, as ore is produced, a negotiated percentage of the value is paid to the previous owner.

   Minimum, no matter how small the production royalty is, a minimum payment is made regardless (monthly or annually).
4. Royalty cap: payments made until the aggregate equals a specified amount, whereupon further royalties cease.

Production royalties can be calculated as a percentage of the total price *received from the sale* of the ore mined. This is called a "gross revenue royalty." More typically, certain charges and costs are deducted, such as freight and insurance from mine to the buyer's facility, deductions and penalties for impurities, and subsequent treatment charges, sales taxes, etc. These are commonly referred to as *net smelter return* royalties, or NSR royalties.

Some owners believe they should be paid a royalty on the full mineral content in the ground, at prices quoted in current mining periodicals. This is mistaken because

1. Ore in the ground is an estimate only and not a precise measure at any specific extraction site.
2. There will be mining losses and dilution.
3. The mill will recover only a percentage of what is mined.
4. The price received by the mine will be less than the quoted price because of freight and insurance to the buyer's location, processing and refining costs, and attendant losses by the buyer. Published prices may not reflect discounts that are the reality of the marketplace.

Royalties may be based as a percentage of the profit made by the miner. Profit has to be carefully defined. It may be gross cash operating profits, but paper charges, like depreciation and amortization, may be deducted, or it may be a percentage of "taxable profits" where additional costs, such as corporate (general and administrative, G&A), can be deducted before calculating the royalty amount. Such royalties can be refereed to as net profits royalties (NPR) or net profits interest (NPI).

The mining company prefers NPRs because the owner shares only when prices are sufficient for both parties to enjoy the profits. In times of low prices, both parties suffer together. With an NSR, the owner may be enjoying high royalty payments while the mine operator is losing money. Care must be taken that the

NSR owner is not penalized because of poor mine management and excessive costs unnecessarily reducing profits.

It is a common rule of thumb that in a well-managed operation, a stated NPI percentage is equivalent to an NSR of about half that amount (i.e., 4% NSR equates to 7 or 8% NPI). Specific conditions can grossly distort this ratio.

4.6.5.5 Holding Costs

Some owners grant long-term leases or options at only nominal holding costs. Under such relaxed conditions, the lessee may let the property go unexplored and lie idle. No other party can be enticed to explore because of the double financial burden to the owner and the lessee. To prevent such stalemates and the deep storage of good mineral properties, the owner should insist on rentals or work obligations, increasing with time, to prevent long-term control of the property without an incentive to test and develop. The rentals and obligations should be high enough to foster testing in a prompt way but not so onerous as to deter reasonable people from making a deal at all. Knowledgeable consultants can be found to advise what are acceptable and reasonable terms.

4.6.5.6 Joint Ventures (JV)

This is an arrangement whereby a party who controls a tract of mineral rights joins with a technically and financially competent party to test and develop the minerals for their mutual benefit. In essence, the first party contributes the mineral rights (plus any relevant data, facilities, and expertise) while the other party(ies) contribute money and expertise. Superficially, it appears to be a partnership, but there are important legal differences. Some countries do not have the legal structure to permit forming a true "joint venture." A "partnership" can have significantly different tax consequences.

These are complex deals to negotiate. Both parties tend to be sophisticated and financially responsible. The bargaining can become complex.

The basic items to be negotiated are

1. How much money must be spent by the incoming party, over what time period, to earn what percentage interest? After earn-in period, both parties contribute their share of costs and enjoy their share of profits and cash flow. The earn-in may be staged so that after putting up $X, a permanent Y% equity has been earned. The contributing party may stay at that equity level, whereupon both parties contribute proportionately, or the farming-in party may elect to continue to fund 100% to earn a higher interest.

2. Who manages while the farm-in party puts up all the money? Generally, the financing entity can insist on the right to manage, but for a variety of reasons may let the noncontributor be the manager.

3. Qualifying expenditures must be carefully defined, especially nebulous charges like management fees, rental charge of company-owned equipment to the joint venture, corporate overhead, depreciation, etc.

4. If the JV is to be equally owned (50-50%), how are deadlocks and stalemates resolved?

Possible solutions in case of deadlock are

a. Major decisions (defined) require unanimous vote.

b. The manager can unilaterally invoke a budget no greater than % of the last approved budget, provided it must be large enough to preserve rights to the property.

c. Invoke arbitration.

d. Either party can set a price at which the other party must sell its equity or buy out the other party.

e. The right of first refusal to buy the other participant's interest at a third party's offering price.

f. The manager (or either party) can invoke a work plan and budget after certain procedural attempts at reconciliation. The other party has the choice of contributing or diluting. Dilution of equity interest tracks the percentage shortfall of its financial contribution compared to the other party's. There may be a point below which no further dilution occurs no matter what the financial shortfall. This residual interest effectively becomes a NPI. Alternatively, dilution below a certain percentage automatically reverts to 0%.

g. The manager automatically is replaced if it becomes bankrupt.

h. If a large exploration area is involved, small portions can be spun off and set up as special projects. Percentage ownership can be different and, so too, the manager. Special projects arise when one party wishes to concentrate exploration there while the other does not and is willing to dilute in this restricted portion.

i. Convert the joint venture into a corporation, with each party owning half the shares. Outside directors provide the swing votes.

A small company may be reluctant to enter a joint venture with a major company for fear that the financial disparity will end up in dilution for them, or somehow in their being squeezed out. It can be agreed that should the small company not be able to meet a cash call, it can request the large financially-strong company to advance the money on its behalf at some agreed interest rate (prime plus). This will be subject to preferential payback out of operating cash flow to the major. This carried advance may be all the way to production, or just to the stage where the project is "bankable," that is, the junior can borrow money from the banks with the project as collateral. It should be the responsibility of the major partner to prepare sufficient documentation of the viability of the project so that it truly is bankable by the junior.

4.6.6 SUMMARY OF EXPLORATION SUCCESS

Regan (1971) canvassed the North American mining industry and from the polled data selected the seven most successful exploration companies and the seven least successful during the period 1966-1970. The first group reported 12 major commercial discoveries during the period, the second group only one. The companies were analyzed, and executives interviewed, to see whether significant differences existed in their approach and methodology toward exploration. The results, abstracted and generalized in Table 4.6.1, appear equally germane today as in the 1960s.

Corporate organization was found by Regan to vary widely, and there was no discernable pattern distinguishing the two groups' organization charts. The same was true with accounting practices.

Disparity in budget size existed between the groups (five-year total for six of the seven most successful was $177 million vs. (estimated) five-year total of the seven least successful of $50 million. This must account for some of the difference in the number of discoveries, but the other factors seem surely to have influenced the discovery rates as well.

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Table 4.6.1. Differences in Approach and Methodology of Successful and Unsuccessful Exploration Companies

Factors	Successful	Unsuccessful
Does a formal system (such as a Mine Development Group) exist between Exploration and Operations?	Yes	No
Is technical competence of staff stressed?	Definitely	Less so
Is long-range planning important?	Definitely	No, or less so
Does exploration deviate from company policies?*	Rarely	Variable, but tends to
Does top management dwell on exploration's past successes or failures?	Successes	Failures
Exploration morale	High	Low(er)
Interdepartmental communications	Well developed	Poorly developed
Annual budgets	Large	Medium to small

Source: After Regan (1971).
*Successful companies may have more flexible, or less rigid, policies so "deviation" is less necessary.

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Section 5 Project and Mining Geology

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Chapter 5.0
INTRODUCTION

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Every mineral project or mine is based on a geologic entity-an ore (mineral) deposit. A well-defined mineral deposit and its associated geologic characteristics are the only aspects of a project that cannot be altered. Mine plans can be modified to exploit the deposit using alternative approaches and yielding similar results. A variety of processing methods also are often available producing generally comparable results. Major modifications can be made to other aspects of the project without substantially changing project economics. However, all engineering and metallurgical aspects of a mine must be accommodated to the specific location and geologic characteristics of the deposit.

Geologic data and interpretations form the foundation for both mine evaluation and mine production by providing estimates of reserves and essential information for mine planning and process design or control. Proper geologic work requires a keen awareness of and an ability to anticipate the technical requirements of geotechnical engineers, hydrologists, mining engineers, metallurgists, and other specialists. Geologists are integral members of the project evaluation or production team. Presentation of pertinent data in a usable format and frequent communication of new geologic knowledge to the other members of the technical team are integral components of the geological program.

The ultimate objective of the exploration geologist is to find ore, of the project geologist it is to define the ore, and of the mine geologist it is to keep the mine in ore. Project geology follows sequentially after exploration, once a discovery of potentially economic mineralization takes place and evaluation and development proceed; mining geology begins upon the commencement of production, but also can be defined to include project geology. Project geology combines many of the elements of exploration and mining geology through the delineation of mineralization and estimation of reserves. Feasibility studies and development decisions involving large capital outlays require great accuracy in mineral deposit definition and reserve estimation. The greatest single cause of mine failure worldwide is unreliable reserve estimates. In addition, the project or mine geologist is expected to identify potential ground stability and hydrological problems.

Section 5, Project and Mining Geology, relates the procedures and goals required for sampling, assaying, geologic data collection, delineation, geologic interpretation, and reserve estimation of a mineral deposit. In Chapter 5.1 of this section, the general principles of project and mining geology are presented, including responsibilities, procedures, importance, and objectives. Responsibilities of a project or mine geologist include continuing evaluation of the mineral deposit; exploring for more reserves within the district; and contributing to the mine planning, metallurgical testing, land acquisition, land use planning, environmental, and other programs. Procedures for the collection, analysis and interpretation of a variety of critical geologic data are presented in Chapters 5.2 through 5.5. The most common methods of gathering geologic data are by mapping and sampling geologic features, both at surface and underground, when available, and by collecting and logging drillhole samples, as discussed in Chapters 5.2 and 5.3. Collection of samples manually or by drilling should provide material that is truly representative of selected portions of the original rock mass under study. In addition, geochemical surveys, geophysical surveys, mineralogical and petrographical studies, geotechnical investigations, and other studies often provide valuable insights in understanding the geologic setting of a deposit. Sample preparation and analysis, described in Chapter 5.4, are designed to produce analytical results that are accurate for the samples collected.

Interpretation of geologic and analytical data, discussed in Chapter 5.5, is essential in the understanding of a geologic setting, the determination of ore controls, the development of a geologic model, and the delineation of mineralization. Valid geological interpretation relies on reasonable assumptions and judgments as to the adequacy of sampling, assay quality, and geologic projections. Interpretative data has a major impact on mine operations through definition of the location, size, shape, attitude, tonnage, and grade of mineral reserves, as described in Chapter 5.6, and by providing other information on geologic characteristics of both ore and waste. The most important goals of project or mining geology are the accurate definition and estimation of reserves. Reserves are validated by detailed comparison with estimates produced by various alternatives such as manual, geostatistical, and computer-assisted conventional methods. Additional objectives are grouped in the broad category of providing information and support for other evaluation specialists or production engineers (Chapter 5.7). Three case studies are presented in Chapter 5.8 as examples of recent project evaluations. For a more extensive series of recent case histories of applied mining geology, the reader is referred to the SME volumes edited by Erickson (1984) Metz (1985), and Ranta (1986).

Section 5 focuses primarily on project and mining geology for metallic deposits, but, where appropriate, reference is made to placer deposits, as well as industrial minerals, coal, and other nonmetallic mineral deposits. The concepts, procedures, and goals presented are generally applicable to any type of mineral occurrence, and only the details of applying exploration and evaluation techniques may vary. Further information on the exploration and evaluation of specific types of mineral deposits can be found in Section 5 (Payne, 1973), *SME Mining Engineering Handbook,* 1st ed. Deposit types mentioned include sedimentary, disseminated, placer, coal and petroliferous solids, uranium, and miscellaneous minerals.

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Chapter 5.1
GENERAL PRINCIPLES

WILLIAM C. PETERS

Mining takes place in geologic media, and the dictates of geologic conditions prevail throughout the life of a mining operation. The resident geologist evaluates the significant conditions and their impact on the mine.

Exploration geology, described in Section 4, is directed toward discovery of a mineral deposit and the recognition of a prospect that can be developed into a mine. Mining geology, beginning as project geology in the preproduction development stage of a mining sequence, is directed to the design, planning, and operation of the mine.

Project geology deals with a succession of feasibility studies and with subsequent mine preproduction development. It is commonly done within a project task force comprising geologists, geological engineers, mining engineers, metallurgists, and other technical specialists. With the completion of preproduction work, the project becomes a mine with a resident mine geologist or a geology department.

Relatively small mines operate with a single resident geologist or with a geologist in the engineering department. Major mines have geologists and supporting personnel in a staff department or a geology section in the engineering department.

The geology department or section at a major mine may comprise groups assigned to specific sectors within the mine and groups assigned to geological engineering, ongoing district exploration, and hydrogeological and environmental work. Ore reserve estimation may be done in the geology department, but it is more commonly within the scope of the engineering department where it is handled in cooperation with the geology department. Functions that may be assigned to the mine geologists or performed in cooperation with other departments are ore sampling and grade control (planning department) and analytical laboratory work (quality control department.)

The chief geologist ordinarily reports to the mine superintendent, to a technical services superintendent, or to the general manager of the corporation's local mining unit. The closest lines of communication are, however, to mine engineering and planning. The geology department at an individual mining unit commonly has an additional line of communication and responsibility to a manager of operating properties geology at the corporation's exploration division headquarters.

5.1.1 RESPONSIBILITIES: GEOLOGISTS

Project and mine geologists are essentially responsible for evaluating known ore bodies and finding more ore. Responsibilities also relate to geologic factors in the design, planning, and operation of a mine; to characteristics of the ore that is handled in the mineral processing plant; and to environmental concerns. In each of these, there are long-term (years), intermediate-term (months), and short-term (day-to-day) elements.

5.1.1.1 Ore Body Evaluation

The ore bodies that were outlined in earlier exploration work are verified and delineated during preproduction; their limits and changes in character are defined, and new ore reserves are sought. In this, continued development work, drilling, and sampling provide for geologic resources to be included in mining reserves as their classification is changed from possible to probable to proven ore and a mine plan is devised. The geologist's and engineer's periodic inventories of ore reserves are made with a number of geologic provisions, including provisions for the differing mineralogical zones in the minable ore and for alternative cutoff grades on the periphery of each ore body.

In the short term, geologists and the planning engineers are responsible for ore-grade control and for the designation of ore boundaries in stopes and benches. In Chapter 5.6, the estimation of ore reserves is covered in detail, with discussions of sample spacing, ore reserve models, methods of calculation, and classification.

5.1.1.2 District Exploration

One of the principal responsibilities of the mine geologist is to define and interpret the patterns of mineralization throughout the immediate district, as well as in the mine. The job is essentially that of district exploration for additional ore. Nearby prospects, mining operations, and exploration activity are evaluated. The need for district-wide exploration relates to the inevitable depletion of the ore bodies currently being mined, and the information becomes additionally important when a joint venture, lease, or purchase of mineral land is considered.

Supporting materials such as coal, aggregate, limestone for cement, stope fill, and siliceous ore for metallurgical purposes may be needed. The resident geology department evaluates the sources. Mine geologists have an additional exploration responsibility that extends beyond the district: the mine and its geologic setting can provide orientation survey localities for new exploration methods, and mine geologists are often called upon to advise exploration personnel on the targeting of similar deposits in other districts and regions.

5.1.1.3 Mine Design and Planning

The geologic factors involved in design, planning, and operations are identified and assessed. Where development is planned for new ground in untested geologic and geotechnical surroundings, or when changes in mining method are anticipated, the mine geologist's projections are especially critical. In the shorter term, there is a need for monitoring water and ground conditions throughout the mine, as well as a responsibility for geotechnical investigation and for immediate advice on problems with ground control or slope stability.

Where mineral processing methods are designed and planned, bulk samples from workings and drillholes are 'tested, and the samples have an inherent geologic context. Mineralogical zoning within the ore bodies is fundamental in this respect, and the zoning patterns reflected in the appropriate computerized model also relate to production goals and to forecasts of monthly and yearly recovery of metals or minerals. In the short term, the processing plant needs continuing advice on the changing characteristics of ore from active stopes or benches. Where ore blending or selective mining is practiced in order to give an acceptable mill feed, day-to-day geologic advice is needed at the mine and at the concentrator.

5.1.1.4 Land and Environmental Geology

It is the geologist's responsibility to evaluate land for mining and for supporting nonmining use. Areas with a potential for ore mineralization are designated for exploration and for future expansion of the mine, and nonmineral sites are identified for water supply, mine dumps, tailings disposal areas, plant installations, housing areas, and access routes.

Environmental baseline data, environmental permitting, compliance with environmental statutes, and the continued monitoring of land, water, air, and biota have geologic aspects. These functions, and the requirements for land reclamation and restoration during and following mining, are often handled jointly by the geology and engineering departments.

5.1.2 PROCEDURES IN MINING GEOLOGY

The procedures of project and mining geology deal in large part with the collection and interpretation of data: with a geologic database that is begun at the very outset of work in a project. As the mine geologic database grows, it affords a mine geologic model, a conceptual picture of the specific deposit-this becomes a basis for interpretations and a platform for projections. The initial geologic model of the deposit, like the geologic models used in exploration, is revised and expanded as more is learned about the deposit's limits and variations. In the maps, sections, and charts made from information in development workings and drillholes, the geologist can begin to recognize correlations and discontinuities that were missed or that could only be vaguely projected between the earlier exploration pits and drillholes. With further development and mining, the results of geologic mapping, logging, monitoring, and sampling can then be added to the geologic database and to an improved geologic model.

As mining continues, data become available on metallurgical recovery and on pumping, explosives and bit usage, and ground support. All of these have a geologic aspect, and all are relevant in some respect to the growing geologic database and the changing geologic model. A development in "new ground" is planned on the basis of the geological and geotechnical conditions mapped and monitored in "old ground."

Sampling of the mineral deposit and of the adjacent ground is a continuing procedure in mine geology. Geologic and geotechnical interpretations in themselves need to be supported by samples taken for petrographic study, mineralogical examination, and laboratory analysis. Ore sampling for assay and in relation to design, planning, and daily operations is done in the context of the mine geologic model and its mineralogical zoning pattern.

In long-term aspect, the sampling of drillholes and of exploration and development workings relates to successive estimates of ore reserves for eventual development and mining. In the short term, the sampling of production drillholes, blastholes, and broken ore is done for grade control and immediate mine planning.

The procedures of data collection and recording are given in Chapter 5.2. Methods of data interpretation, with descriptions of the database and interpretive models are discussed in Chapter 5.5. Detailed comments on sample collection are given in Chapter 5.3, and comments on sample preparation and assaying appear in Chapter 5.4. Case studies of geologic procedures and methods at specific mines are given in SME's three volumes on *Applied Mining Geology* (Erickson, 1984; Metz, 1985; Ranta, 1986).

IMPACTING GEOLOGY AND DATA		OPERATIONS AND FACTORS: DRILL AND BLAST, EXCAVATE, LOAD, HAUL: Hardness	Abrasiveness	Fragmentation	Broken Ore Storage	Controlled Caving	Stope Filling	AUXILIARY: Ground Support	Pumping / Drainage
MAPPING AND LOGGING	Rock Type	●		●	●	●	●	●	●
	Mineralogy		●		●		●		
	Texture	●			●		●		●
	Fractures			●	●	●	●	●	●
	Structure					●		●	●
	Geohydrology				●			●	●
GEOTECHNIC TESTING	In Situ	●	●			●		●	●
	Core			●		●		●	
	Laboratory	●	●			●	●	●	
RECORDS AND MONITORING	Drill Rate / Bit Wear	●	●						
	Powder Factor			●					
	Ground Support							●	
	Drainage / Pumping							●	●
	Subsidence					●		●	
	Geotechnical Instrumentation					●		●	●

Fig. 5.1.1. Impact of geology on underground mining operations.

5.1.3 IMPACT OF GEOLOGY ON MINE OPERATIONS

The impact of geology carries throughout mine operations. The geologist's responsibility is to provide the relevant information. The most distinct impact bears upon the ore that is to be developed and mined—its location, extent, attitude, and grade. A broader and more complex set of impacts relates to the development and mining operations themselves.

The relevant information is established from the current geological model and database. Then communication with the engineering and planning staff carries the collected geologic data into an active stream of information that pertains directly to the impact of geology on the mining operation.

Figs. 5.1.1, 5.1.2, and 5.1.3 show examples of impacting geology and its relation to operational factors in underground mining, open pit mining, and mineral processing. The impacting geology is related to data accumulated through mapping and logging, geotechnical testing, and the interpretation of geology-associated records.

In each example, the impacting geology is necessarily predictive—it is based on observations in known ground, but it is applied to new sites and conditions. The most critical predictions deal with projected changes in geologic conditions. A significant change in rock type or in the mineralogy of the ore would, for example, impact the entire sequence from mine development to the disposal of tailings at the mill.

Operations and Factors / Impacting Geology and Data		Drill and Blast, Excavate, Load, Haul, Crush, Convey					Waste Disposal and Restoration		
		Hardness	Abrasiveness	Fragmentation	Rippability	Slope Stability	Overburden Character	Pumping / Drainage	Haul Roads
MAPPING AND LOGGING	Geomorphology					●		●	●
	Rock / Soil Type	●		●	●		●	●	●
	Mineralogy		●				●		
	Texture	●			●			●	
	Fractures			●	●	●		●	●
	Structure					●		●	●
	Geohydrology					●		●	●
GEOTECHNIC TESTING	In Situ	●	●		●	●	●	●	
	Core	●	●		●	●			
	Laboratory	●	●			●	●		
RECORDS AND MONITORING	Drill Rate / Bit Wear	●	●						
	Powder Factor			●					
	Secondary Breakage			●					
	Drainage / Pumping					●	●	●	
	Geotechnical Instrumentation					●		●	

Fig. 5.1.2. Impact of geology on open pit mining operations.

In the basic production cycle of drilling, blasting, loading, and haulage at an underground or open pit mine, geologic impact is related to rock type and texture, ore and gangue mineralogy, and fracture patterns, and it is related to geotechnical determinations of rock and ore characteristics and stress conditions. The relevant information is taken from mapping, logging, and testing and from the records of prior drilling and blasting performance under various geologic conditions. Where excavation is done by ripping or continuous mining, and where development involves raise boring or roadheader advance, the factors are similar, and the same geologic data are applicable.

Underground backfilling of waste, storage of broken ore in underground shrinkage stopes, and surface stockpiling and heap leaching are all affected by geologic conditions related to fragment size, shape, density, and mineralogy. Mineralogy is of special importance in the sense that the weathering and oxidation of ore and gangue minerals can cause plugging, corrosion, and cementation in the broken material.

In relating geologic impact to ground support and slope stability, the information taken from mapping and logging is combined with the information from geotechnical testing and monitoring. The requirement for geologic information is similar in caving methods of mining.

Rock type and texture are critical factors in the initial crushing and grinding stages of conventional mineral processing, but their impacts and the additional impact of changes in mineralogy continue throughout the successive stages in mineral processing. The mine geologist has an ongoing responsibility for communication with the mill engineer regarding changes in the geologic nature of the ore.

Operations and Factors / Impacting Geology and Data		Crush and Grind			Concentration			Leaching			
		Hardness	Abrasiveness	Work Index	Classification	Density	Flotability	Leach Amenability	Leaching Pad Base	Water Supply	Tailings Disposal
MAPPING AND LOGGING	Rock Type	●		●	●			●	●	●	●
	Mineralogy	●	●	●	●	●	●	●	●		●
	Texture	●		●	●	●	●	●	●	●	
	Fractures							●	●	●	
	Structure								●	●	
	Geohydrology								●	●	
GEOTECHNIC TESTING	In Situ	●	●						●	●	
	Core	●	●			●			●	●	
	Laboratory	●	●	●		●			●	●	
RECORDS AND MONITORING	Recovery Percent					●	●	●			●
	Concentrate Grade					●	●				
	Concentration Ratio					●	●				●
	Mineralogy - Tailings					●	●				●
	Crushing / Grinding	●	●	●							

Fig. 5.1.3. impact of geology on mineral processing.

5.1.4 OBJECTIVES IN MINING GEOLOGY

The ultimate objectives of the mine geologist are to keep the mine in ore and to see that geological conditions are taken into account throughout the life of the mining operation. The more immediate objectives are to fulfill these responsibilities at the current stage in the sequence of development and mining. The objectives, in both long- and short-term, are centered on an ore body or a group of ore bodies, a mine, and a related plant facility. But the objectives and the responsibilities have a broader spatial and temporal aspect as well: they pertain to the ore bodies and geologic conditions throughout a mining district, and they relate to an operation that goes from discovery and preproduction development to an eventual sequence of stages in mining.

5.1.4.1 Preproduction

During preproduction, a principal objective is to put specific dimensions on the geologic factors that were mapped, measured, and estimated during earlier feasibility studies. This amounts to the formulation of an initial database and geologic model. The focus is on mine design and on the defining of sufficient proven and minable ore reserves to support the initial capital investment. But there is more: preproduction work calls for putting dimensions on the geologic aspects of regulatory permitting, a plant site, a water supply, and supporting facilities for power, transportation, and mineral processing.

An associated aim during preproduction is to determine the pattern of geologic and geotechnical zones that will influence the subsequent stages in mining and processing. Bulk sampling and testing are important to this objective, and the initial work-

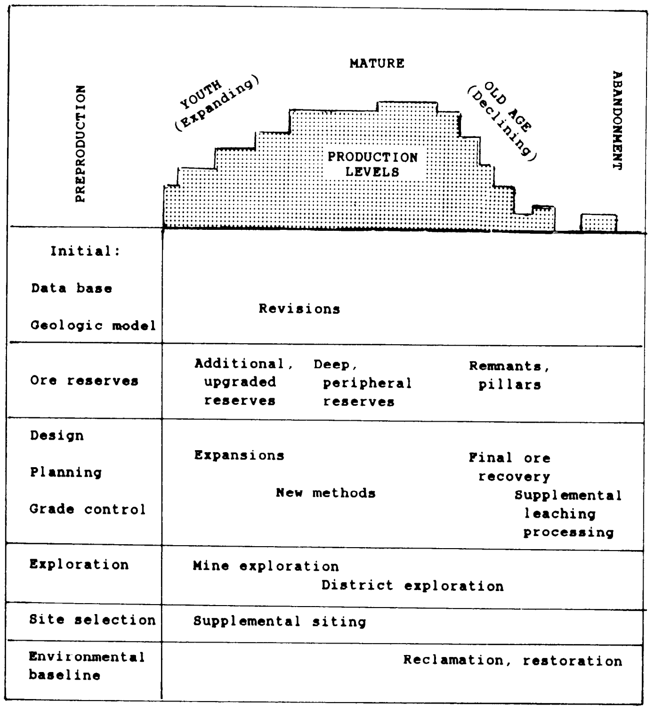

Fig. 5.1.4. Mining geology focus in the mine life cycle.

ings may have to be planned for this as well as for immediate mine development.

A broader objective is related more specifically to the long-term responsibilities that begin in preproduction and continue through the life of the mine. This objective is to establish consistent procedures in geologic investigation and data collection, with appropriate levels of detail, accuracy and precision. Then, as geologic parameters begin to develop, an adequate database is established for measuring the subsequent changes in geotechnical, hydrological, and environmental patterns.

5.1.4.2 Mining Life Cycle

A mining life cycle of youth, maturity, and old age can be characterized in spite of its variations from mine to mine. The life cycle can be as short as a few years, or the mine can be consistently productive during a century or more. It can be irregular, with idle periods and with periods of rejuvenation and expansion. Unlike the biological life cycle analogy, there can be resurrection; the graveyard of a former mining operation will often be recognized as an exploration target, and the subsequently rediscovered ore body begins a new mining life cycle.

In the characteristic mining life cycle, a mine passes from preproduction to successive stages of youthful expanding production, mature production, old-age decline, and abandonment. The focus of mining geology during the life cycle is illustrated in Fig. 5.1.4. At each stage, the mine geologist's immediate responsibilities and objectives are shared with longer-term objectives that relate to responsibilities in the later stages.

Youthful Stage Mines: In the youthful stage of a mining life cycle, the objectives of preproduction continue. Proven ore reserves are augmented from probable and possible reserves, and the limits of the deposit or group of deposits are sought and outlined. Grade control procedures, designed during preproduction, are now capable of being refined. The database, also designed earlier, now begins to fill out, to require massive data processing, and to provide stronger support for geologic projections.

A new objective for the mine geologist is to provide specific information for mine growth and expansion. This can involve site studies for new access through adjacent mines, new shafts and entries, a supplemental or extended pit, and an enlarged milling and waste disposal facility.

Mature Stage Mines: In the mature stage of a mine's life, the objectives continue to relate to grade control, ore reserve definition, and geologic information for mine planning, but there are additional considerations. In a mature mine, the apparent limits of the established ore reserves will commonly have been recognized. The workings are deeper and broader than before, and mining costs are higher. The complex outer zones of the main ore bodies are part of the picture, and the grade of the deeper and peripheral ore may be lower than that of the ore previously mined. The mining geologist's objectives are now governed by an economic squeeze.

The finding of roots to the deposit and the identification of extensions and associated deposits are now objectives. The geologic model and geologic data from earlier stages in the mine's life cycle come into full play. The economic squeeze in a mine's maturity can result in a need for raising the cutoff grade or for emphasizing higher-grade ore zones during times of low metal price. Conversely, improvements in metal price and in technology can permit the mining of lower-grade ore. In either event, changing economic conditions can bring about a change in mining method and a reassessment of economic ore reserves. The mine geologist must then provide for a delineation of different ore zones and an assessment of different geotechnical conditions in mining, access, and waste disposal.

District exploration intensifies, and it extends to prospects and even to relatively small ore bodies that could be developed and integrated with the existing plant. Exploration targets, in the district and outside, receive special attention by the mine geologist and by company exploration geologists.

Old Age Mines: A mine's old age may last through a number of years in the mining of dwindling ore bodies and pillars or in the mining of a steep-walled "ultimate" pit. The geologist's work in this stage involves a review of information collected in earlier stages, and the objectives at this point turn to a re-inventory of the remaining ore and an assessment of the geotechnical conditions for final mining in disturbed ground. The mine geologist may be concerned with an evaluation of workings for in-place leaching and with the evaluation of dumps and tailings for processing and leaching. Environmental restoration, stabilization of workings and waste, and the need for mined-land reclamation are now specific geologic concerns.

5.1.4.3 Planning for the Objectives

The geologist's continuing responsibilities and objectives require the building of an early foundation. The early foundation will be sketchy and speculative, but it amounts to a skeletal database and model that serve to identify the currently accessible geologic and geotechnical information that will be later required in a broader and deeper mining complex.

Provision for the massive data processing that will be required in the youth and maturity of a mine's life can be made at the outset of operations or during preproduction, even though the need does not seem to be immediately pressing. Determinations of the potential limits of ore mineralization at grades not now economic, the potential shape and mineralogy of the ore body borders, and the presence of additional ore bodies that may be incorporated in expanded operations—conditions of importance in the mature stage of a mine—require an established fund of data.

The needs of a mine in the stage of declining production or old age can be anticipated. This will include an assessment of values and of geological and geotechnical conditions in the final pit, in pillars, in dump leaching, and with in-place leaching operations. Geological and geotechnical data need to have been obtained during earlier stages of mining, before some of the workings and terrain became inaccessible. Mine abandonment, with the need for stabilization and environmental restoration, makes use of the geologic database that began during the preproduction period and extended through the mine's productive years.

The role of the project geologist and mine geologist requires an appreciation for the changes in design, planning, and operations that will take place in the mine. Performance of the role calls for the recognition of an ultimate geologic model, a conceptual picture of the conditions that will affect the geologist's objectives and procedures as the mine grows to maturity and is finally depleted of its ore.

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Chapter 5.2
GEOLOGIC DATA COLLECTION AND RECORDING

A. J. ERICKSON, JR.

Mining is a physical endeavor that extracts some valuable mineral resource from the earth. To do this effectively, it is essential that management has as accurate as possible a characterization of mineral-zone geometry. A good understanding and clear representation of the shape, size, quality, variability, and limits—the geologic characteristics of the mineral zone—are needed at the evaluation, development, and productive stages of a project. This characterization requires a high-quality geologic database in order for the geologist, in a clearly supportive role, to provide management with the information needed for critical project decisions. The financial success of the mining venture is directly related to the completeness and accuracy of the geologic database and the quality and understanding of the characterization that describe the mineral zone. In an expansion of an earlier statement, the geologist who fails to provide the best geologic characterization based on data available at the time "is delinquent in his or her responsibility for providing management with the best possible information for intelligent decision making" (Barnes, 1980, p. 63).

This chapter deals with basic geologic data-collection principles, the first of the three essential geologic information processing steps—data collection, analysis, and interpretation. Chapter 5.5, to follow, deals with the latter two in the development of the geologic model and input into a database for resource estimation of the mineral zone.

5.2.1 GENERAL

Geologic data vary greatly within a single deposit and from deposit to deposit. This applies if we are concerned with metallic, coal, or nonmetallic commodities. Specific geologic features will differ considerably, and likewise the importance of any specific feature will vary from deposit to deposit. The geologist is faced with the task of collecting many types of geologic data. It is seldom known in advance which features are critical; therefore, he/she must collect detailed data on all features of potential importance. Data may be collected in a number of ways, such as by drilling, surface or underground mapping, geophysical or geochemical surveys, or in specific studies examining such features as structure, rock mechanic properties, alteration effect and distribution, or mineralogy.

A great deal of time, effort, and expense is required to collect data adequately. It is essential that data-collection systems be planned in advance so that all data and descriptions are systematically gathered to insure completeness and high quality. Data-collection systems should be on at least a region-wide, but preferably a company-wide, standardized system or format to assure consistency, accuracy, neatness, legibility, objectivity, and completeness. Fact must be discernible from inference. Personnel should be trained in the requirements imposed by the recording system. It does little good to have two people, biased by personal experience, making data recordings on the same outcrop or sample, the ultimate record suggesting two entirely different geologic units. Quantification of the geologic variable should be done wherever possible. This requires estimation which, although imprecise, is sufficiently accurate for comparison or correlation purposes and far superior to vague generalizations.

Geologic data are extremely important, costly to obtain, and essential for proper evaluation and mining. The data collection may be a one chance occurrence due to constraints imposed by mining or distance. A second observation of a critical area may be impossible as the drift or bench may be mined out or the core crushed for assay. The system used to collect the data should be designed to assure timely completeness, accuracy, consistency, quantification, objectivity, and legibility. If data is collected in this manner it will provide a useful record that is timeless in character. The practice of geology is not easy and requires a great deal of patience, diligence, discipline, and simply hard work, whether it be surface or underground mapping or core or cutting logging. A high level of professionalism is required. Further comments on these topics are clearly discussed by McKinstry (1948, Chapters 19 and 17), and should be reviewed.

5.2.2 REQUIRED DATA

Every effort must be made to observe, record objectively, and describe all geologic features that may be of importance in characterizing the size, shape, and variability of the resource under study and its associated environment. Broad categories of data to be collected routinely are location information and lithologic, mineralogic, alteration, structural, and rock competency data. As experience is gained in specific areas, deposits, or suites of rocks, the capability to subdivide various units into key subunits typically will be developed. This will enable the geologist to improve description, correlation, projection, and understanding of the genesis of the deposit. More importantly, it will allow for superior resource estimation and improve recommendations needed for management decision making.

Important details within each category appear in Table 5.2.1. Keys are careful observation and clear description.

5.2.3 RECOMMENDED EQUIPMENT

Equipment varies somewhat with individuals and projects; however, in addition to normal personal gear—hats, boots, lamp—items listed in Table 5.2.2 are useful equipment that should be on hand for either mapping or core logging. Ear plugs of appropriate design *must* be worn around equipment, particularly percussion drills, to prevent long-term permanent nerve damage and resultant hearing loss. Safety glasses are essential.

5.2.4 SYMBOLS

Geologic representation, geologic mapping, core logging, data compilation, and interpretation are heavily dependent on visual presentation as "geologists think and communicate best in pictures. The basis of almost all geologic work is the making of maps, plans, sections and the like. The graphical form is the most efficient way of displaying and explaining geological observations and ideas" (Dixon, 1979, p. 5). Standardization in the way geologic data are collected, recorded, or displayed is desirable as the concepts being portrayed are frequently difficult,

Table 5.2.1. Geologic Data Collection: Key Features

Location Data

Sample, map, mine, or drill location on each sheet. May include geographic data such as state, county, section, township, range, latitude, longitude, coordinates, elevation, mining district, mine, pit, bench, level, working, claim, claim corner, or any and all information that will clearly identify the unique location of the geologic data points. Data cannot be used if the geologist does not know where they came from.

Lithologic Data

Typical data to describe rock, sample, or unit. Should include color, texture, mineralogic characteristics, lithology, and rock type. Appropriate descriptive modifiers, stratigraphic information, if known, top and bottom data, age relationships, and general gross features as hardness, competency, and bedding characteristics. Subjective generic terms should be avoided unless well established or qualified to distinguish inference from observable facts. Primary sedimentary structure and sedimentological features such as bedding, laminations, casts, soft-sediment deformation, graded bedding, burrows, bioturbation, fossil content, or banding, foliation, and lineation, with appropriate attitudes, should be noted where possible.

Structural Data

Secondary structural features that post-date rock formation. Should include clear description and attitudes of joints, fractures, faults, breccias with quantitative description of selvages, gouge zones, fragment size, and healed or recemented character of breccias. Folds, dragfolds, crenulations, lineations, and foliation should be noted. Age relationships, mineralization association, and overall effect on rock mass are important. Weathering and oxidation intensity data are usually critical and commonly structure related but may be included with lithologic data. Quantification of structural data where possible is extremely important as they may play a key role in determining minability of a deposit.

Alteration Data

Nature, mineralogy, intensity, and distribution of features. This should include color, texture, mineralogy, intensity, pervasiveness, fracture control, stages, mineralization association, and overall effect on rock mass. Weathering and oxidation intensity are important but may be included with lithologic data. Quantification where possible is extremely useful as is description of age relationships of various alteration features.

Mineralization Data

Nature, intensity, mineralogy, and distribution of the desired resource. Should include primary and secondary classification, estimates of specific and total quantity of various minerals, intensity, character of veinlet, vein or disseminations, supergene features, weathering and oxidation intensity, and associated gangue mineralogy. Vein age relationships tied to mineralogy, alteration, or lithology provide important data in understanding both zoning and grade estimates and overall deposit genesis.

Coal Data

In addition to standard lithological and structural data listed above, logging of any and all features that aid in correlation, understanding the distribution of sedimentary facies, and construction of a depositional model of the coalbed(s) and coal-bearing sequence is important. Detailed descriptions of horizons immediately above and below the roof and floor are critical as are accurate measurements of depths and thickness of all units associated with the coal. Some key features include abundance and type of marine or freshwater fossils, slickenside in roof or floor rocks, the presence of roots representing old soil horizons, pyrite bands, nodules or streaks, siderite or ironstone nodules, and plant debris. Description of individual coalbeds, either the banded or nonbanded groups, requires careful measurement or estimates of the banded lithotypes, vitrain, clarain, durain, and fusain content (Stopes, 1919; Ward, 1984). A more practical system (Schopf, 1960) describes the thickness and amount or concentration of vitrain and fusain bands in a matrix of atrital coal. The latter is described by five luster levels that range from bright to dull. Nonbanded sapropelic coals, boghead and cannel end-members descriptions, rely on identification of these massive faintly banded fine-grained accumulations of algae or spores and usually require a microscope for adequate description. The nature of cleats, partings, bone, and shale layers needs description and careful thickness measurements to separate "net" from "gross" coal bed thicknesses. Coalbed description, while straightforward, requires some supervised training to ensure adequate data recording.

Other Features

Other features that may supply extremely important information with direct bearings on mining and/or metallurgy should be recorded. This may be reasonably objective as fracture frequency, rock quality determination measurements, longest and shortest unbroken core recovered in a run, or more subjective as an overall estimate of rock strength, friability, or competency. Metallurgically significant features such as hardness which affects grindability, grain size which controls grinding for particle liberation, or oxidation intensity which affects flotation recoveries, should be noted. Added testing is almost always needed here; however, geologic data collections should indicate these and other potential problem areas requiring specialized study.

Table 5.2.2. Geologic Data Collection: Useful Equipment

The following are items of equipment routinely used in day-to-day geologic work: (1) Compass, geologist's pick, hand lens, pocket tape, and several cloth survey tapes or chains (nonmetallic for safety), (2) a knife, protractor, or 6-in. (152-mm) plastic scales with imprinted protractor, dilute HCl, magnet, nail, survey spads, cement nails, sample tags, sample bags, marking pens, aluminum sheet holder, preferably end-opening with an end extractable pencil holder attached, and (3) sample ring for holding sample bags during chip sampling, pencils, pens, mapping sheets, or logging forms. A field vest with large pockets for samples and clipboard is helpful. Access to brushes and water buckets is important for core logging and spray bottles help. Extra core blocks are useful as is a good binocular microscope.

and varied symbolism may cause confusion. A simple line or symbol on a map is a powerful decision-influencing tool, may have innumerable connotations, and therefore, should be presented in a clear, self-explanatory way. Symbol standardization, a very difficult objective, helps in obtaining the desired clarity.

There is no single standardized set of geologic and mine-related symbols in use, but those listed in Tables 5.2.3 to 5.2.6 are commonly used in the United States. Many are utilized in other parts of the world as well. They were compiled from works by Dietrich et al. (1982), Peters (1987), Berkman and Ryall (1982), Reedman (1979), Lahee (1952), Compton (1962, 1985), and McKinstry (1948), and from personal experience. Additionally, the Geological Society of London's five-volume *Handbook* series by Barnes (1981), Tucker (1982), Fry (1984), Thorp and Brown (1985), and McClay (1987) are very useful and practical. This series covers both general and specific aspects of mapping, and data recording. The volumes are small and highly portable for field use. Blackadar et al. (1968) illustrate the Canadian

Table 5.2.3. Lithologic Symbols

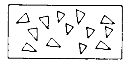
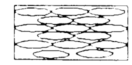
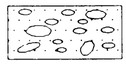
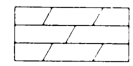
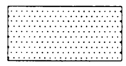
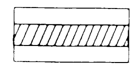
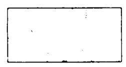
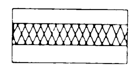
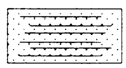
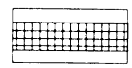
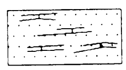
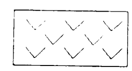
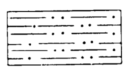
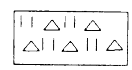
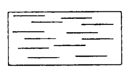
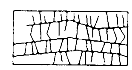
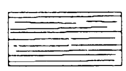
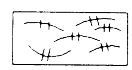
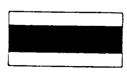
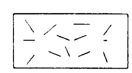
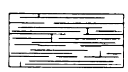
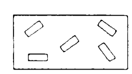
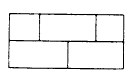
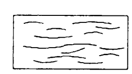
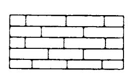
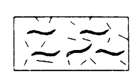
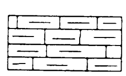
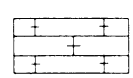
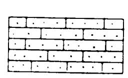
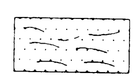
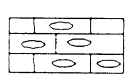
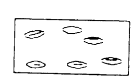
Breccias		Chert	
Conglomerate		Dolomite	
Sandstone, bedded		Gypsum	
Sandstone, crossbedded		Anhydrite	
Shaly sandstone		Salt	
Calcareous sandstone		Volcanics	
Siltstone		Tuffs, breccias	
Mudstone-claystone		Flows, basic	
Shale		Flows, other	
Coal		Granite	
Calcareous shale		Porphyritic	
Limestone, massive		Schists	
Limestone, bedded		Gneiss	
Shaly limestone		Marble	
Sandy limestone		Quartzite	
Cherty limestone		Serpentine	

Table 5.2.4. Culture Symbols, Surface or Underground

Vertical shaft		Shaft, through level	
Shaft, flooded, caved		Shaft bottom	
Inclined shaft		Inclined shaft (chevrons down)	U D
Portal, (blocked)		Raise, winze, (head)	
Portal and cut		Raise, winze, (through level)	
Prospect, open cut		Raise, winze (foot)	
Trench		Chute, manway	C M
Pit, quarry		Working blocked	
Dump		Working filled	
Mine (abandon inverted)		Lagging, cribbing	
Sand, gravel, (abandon inverted)		Stope	
Drillhole, No., (inclination)	MH31 −90°	Drillholes No., inclination	MH31 −17°
Drillhole, No., elevation, depth	2116' MH31 TD397'	Survey point, No., elevation at back, distance to sill	1571 −9.3

viewpoint, and a number of chapters in Finkl (1988) cover symbology as well as other topics on field methods.

Worldwide standardization would be useful, country-wide strived for, company-wide very desirable, regional office-wide should be mandatory, and project-wide absolutely essential. The accompanying tables are not all inclusive but illustrate many commonly used symbols. Projects with multiple units of a similar nature frequently require the creation of additional symbols or combination of symbols. The possibilities are endless, but simplicity, clarity during reproduction, and practicality should be the guides.

Some systems use black for culture, lithology, and rock data, blue for structure, and red for mineralization. This practice is convenient, useful, and readily understood. Other systems use an extensive array of colors for various geologic features leading to either highly useful or totally confusing data collection. Still others collect all data in black, which is quite useful for modern copying technology. Any system should be well planned in advance, simple to learn and use, consistent, and well documented and should aid, not hinder, data collection and understanding.

5.2.5 ABBREVIATIONS

The extensive observations and descriptions essential in geologic work are commonly facilitated by the use of abbreviations. Standardization of these abbreviations is desirable because logs

Table 5.2.5. Geologic Symbols

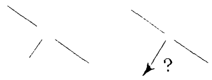
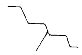
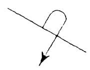
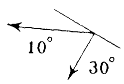
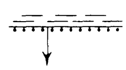
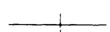
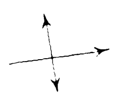
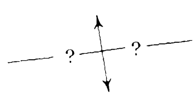
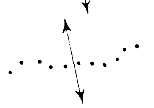
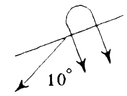
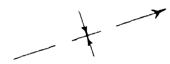
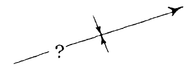
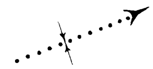
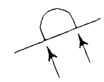
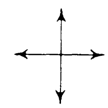
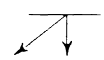
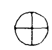
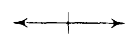
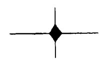
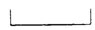
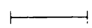
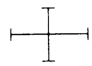
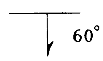
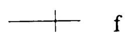
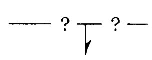
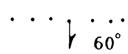
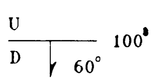
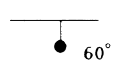
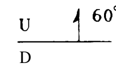
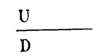
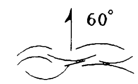
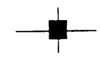
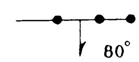
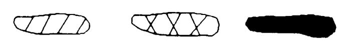
Strike and dip of bedding plane	
Uncertain	
Generalized	
Horizontal bedding	
Vertical bedding	
Overturned bedding	
Strike, dip, lineation plunge	
Contact	
Contact with dip and lithology	
Contact inferred	
Contact uncertain	
Contact concealed	
Contact vertical	
Anticline axis and plunge	
Anticline uncertain	
Anticline concealed	
Anticline overturned, limb dips, axial plunge	
Syncline axis and plunge	
Syncline inferred	
Syncline uncertain	
Syncline concealed	
Syncline overturned, limb dips	
Dome	
Lineation plunge	
Lineation horizontal	
Lineation plunge and bedding, magnitude unspecified	
Vertical beds	
Horizontal lineation	
Vertical beds, horizontal lineation	

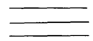
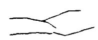
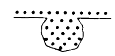
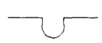
Table 5.2.5 Geologic Symbols—Continued

Foliation showing dip	
Foliation vertical	
Foliation horizontal	
Foliation, horizontal lineation	
Cleavage	
Cleavage vertical	
Cleavage horizontal	
Fault and dip	
Fault vertical	
Fault inferred	
Fault uncertain	
Fault concealed	
Fault, sense, amount of displacement, dip known	
Fault, ball or hachure on downthrown block	

Thrust fault, teeth on upthrown block	
Fault, reverse	
Fault, relative displacement	
Fault zone	
Joint with dip	
Joint vertical	
Joint horizontal	
Vein	
Vein attitude	
Mineralization, ore bodies showing increasing intensity	

Note: Bedding dips show options with and without double arrowheads. The arrowhead prevents map confusion if the blocky limestone symbol is used. It is advisable to use half arrow on veins and faults as it allows clear symbol identification and shows relationships where considerable detail is involved.

Table 5.2.6. Useful Sedimentological Symbols

Laminae			
Plane parallel		Organic material	
Curved parallel		Plant fossils	
Plane nonparallel		Marine fossils	
Curved nonparallel		Brackish fossils	
Wavy parallel		Pyrite	
Wavy nonparallel		Load cast	
Large cross stratification		Flute cast	
Ripples		Scoured surface	
Flaser bedding		Lag	
Bioturbated			
Rooted			
Claystone (seat rock) underclay			
Underclay slickenside			
Clasts, nodules			

and maps may be used by numerous individuals and therefore require some basic commonality. Additionally, long lag time may be involved between data recording and actual use. In addition to the commonly used w (with) or w/o (without), the list abridged in Table 5.2.7 from Chace (1956), with some additions, is fairly standard and used (at least in part) in the United States. Other lists appear in Compton (1962, 1985), Berkman and Ryall (1982), Blackadar et al. (1968), and Finkl (1988). Quantification of these abbreviations is achieved by the addition of superscripts or subscripts, as py^2 or py 2-10, which indicate estimates of 2% or 2 to 10% pyrite content in a rock. Cp_1 and py_2 indicate 1% and 2% chalcopyrite and pyrite, respectively, while cp:py = 1:2 or cp1:py2 indicates a ratio of chalcopyrite to pyrite of 1 to 2. Combinations are endless. The only requirement is that an explanation or legend describing the notation accompany *each* log, map, or project.

5.2.6 CORE AND CUTTING LOGGING

Drilling of one type or another provides much of the material for sampling and geologic analysis of natural resources. The core, rotary, or percussion drill provides the geologist with numerous surface or subsurface points to observe, describe, and sample, providing the facts to include in a geologic database for analysis and interpretation. Drilling is expensive and time-consuming, and in order to derive maximum benefit from the data obtained, an orderly standardized system of recording, commonly referred to as logging, is needed. When done to scale, commonly 1 in. = 20 ft (1:240) or sometimes 1 in. = 10 ft (1:120), this is actually mapping a long, narrow segment of the earth and some correctly refer to the process as core mapping rather than core logging.

Although frequently looked on with disdain and relegated to new, inexperienced geologists, good core logging is in fact a highly specialized skill requiring careful observation, clear accurate recording, and considerable personal discipline. Good drill logs are the primary record-keeping document of the geologist and are used repeatedly by individuals of varied technical disciplines. They must be clear, correct, and legible. The log records the basic data used in geologic analysis, interpretation, and resource calculations, the basis for economic evaluation and decision making.

Numerous types of logging forms exist, all allowing systematic data collection and comparison usually by means of columns for recording data. Data are recorded in the columns in varied ways—graphic, alpha, and numeric—depending upon purpose. Although varied in appearance and to some degree with project needs, all should contain the following broad classes of information, not uncommonly on several sheets or forms.

Location data, commonly referred to as header information, containing thorough location, bearing, inclination, scale, date, geologist, possibly down-hole surveys, hole size, and hole completion information such as casing record, plugging, and hole condition. Detailed geologic logs should provide space for recording depths, core recovery, perhaps rock mechanics data, lithology, alteration, mineralization, and possibly assays. All sheets should provide for hole and page number so separated sheets are not lost. Graphic columns for standardized symbols with adjacent room for notes are desirable. Note-taking should be brief, descriptive, employ standard abbreviations, and be quantified where possible. As an example, py_{10} or 10% py means approximately 10% pyrite to anyone, whereas “strong,” “weak,” or “much pyrite” usually means different things to different individuals. Logs should record as many as possible of the project’s important features listed in Table 5.2.1, using those symbols that are appropriate from Tables 5.2.3, 5.2.4, 5.2.5, and 5.2.6, with abbreviations from Table 5.2.7.

At some stage, generally early in the logging process, core is typically marked for sampling, splitting, and analysis. This should be done by the geologist familiar with the project to ensure that geologic units are appropriately sampled. Samples should be taken to reflect geologic changes and not taken across contacts. This ensures that analytical data reflect geologic variations. Occasionally, this may lead to some short samples; however, with core, the samples can be selected to allow later mathematical compositions to some equal sample length as desired.

Logging should be done to scale (multiple scales in the case of coal) and a separate summary log at a smaller scale, commonly 1 in. = 100 ft (1:1200), completed at the end of a hole. This allows the geologist to organize and simplify the detailed data and also his interpretation for more effective use. Reedman (1979) provides some practical discussion of logging, noting particularly the importance of recording attitude data on planar or linear features in relation to the core axis. He stresses the importance of summarizing observations into major units so the user is not overwhelmed by the meticulous detail normally recorded.

Two general types of formats in use are the graphic and narrative, and the data-entry style. There are combinations of both. Sheet sizes vary from 8-l/2 × 11 in. to 11 × 17 in. (216 × 279 mm to 279 × 432 mm) or sometimes larger. All can be designed adequately to do the job. Smaller 8-1/2 × 11 in. (216 × 279 mm) sheets are easier to handle while larger data-entry pad style allow easier numerical data entry but are somewhat cumbersome. Data-entry pad format requires considerable planning and standardized alpha, alpha-numeric, or numeric codes to be entered properly in appropriate columns to allow for key punch. It also requires a disciplined systematized data entry methodology that typically identifies a feature and then allows for an intensity estimate. The key is to decide what is needed, design a form to assure recording of the needed data, and train the staff in its use. An instructive method of logging coal cores, or any normal facing sedimentary sequence, is to wait until drilling of the hole is complete and then carefully log the core from the bottom up. This allows the geologist to observe the orderly change in the evolving sedimentary record, visualize the processes, and gain a better understanding of the depositional system responsible for the resource being evaluated.

If planned well, a generalized logging form should be usable from project to project within a company, requiring but little project specific modification. For specialized studies, standardized formats can be modified for the desired purpose. References by Call (1979), Perrin (1984), Ranta et al. (1984, 1984a), Atkinson and Erickson (1984), Ahrens (1984), Rowe and Hite (1984), Peters (1987), Goodwin et al. (1982), and Reedman (1979) provide additional discussions, descriptions, and philosophies on logging that are useful. Warnaars et al. (1985, p. 1563) discuss the importance of terminology, nomenclature, and abbreviation in logging and mapping in a paper published after a major deposit evaluation program in Chile. They also provide a good example of a data-entry style logging format. Ward (1984), Wallis (1978), and Merritt (1986) describe specialized requirements for coal logging and in particular of the need to record sufficient data to understand the stratigraphy essential to solving structural problems and the depositional environment influencing coal and associated noncoal distribution. Good detailed measurements and description of lithology, geologic structure, roof rolls, and slickensides in roof, coal, partings, mineral matter, and floor rocks is essential for coal work. Core photography is useful. Logging can be on a simpler form, as shown in Fig. 5.2.4c, or directly on data-entry coding forms, as illustrated by Ward (1984, p. 215). Alternatively, the coding for computer data handling can be done as a separate exercise after logging. Merritt (1983) is a well-illustrated practical reference that provides discussion of coal overburden/interburden characterization as it relates to mine planning, hydrologic, and environmental problems. Ferm and Smith (1980) and Ferm and Melton (1977) have compiled extremely detailed and comprehensive guides to cored rocks in the coal-rich Pittsburgh (Dunkard) and Pocahontas Basins in the eastern United States. These guides were published in an effort to rectify problems in inadequate description, misidentification, and the lack of uniform terminology by providing a standard basis for objective rock description and a uniform set of terms. Color photographs of core specimens of the common rock types in the basins are provided, and each is named and numbered according to a key that conveys information on lithol-

Table 5.2.7. Abbreviations

Rock Names

Igneous rocks

agglomerate	— agg
amygdaloid	— amyg
andesite	— and
anorthosite	— ano
aplite	— ap
basalt	— bas, bt
breccia	— bx, bc, br
diabase	— db
dacite	— da, dct
diorite	— di, dio
dolerite	— dole
dunite	— dun
gabbro	— gb, ga
granite	— gr
granodiorite	— grd, gd
hornblendite	— hbt
igneous	— ig
latite	— la
lava	— lava
lamprophyre	— lamp
monzonite	— monz, moz
norite	— nor
obsidian	— obs
peridotite	— pd
pegmatite	— peg
phonolite	— phon
pillow lava	— pl
porphyry	— po, por
pumice	— pu
pyroxenite	— pyxt
quartz diorite	— qd
rhyolite	— rhy, ry
syenite	— sy
trap	— trp
trachyte	— tryt, tyt
tuff	— tuff
sperulitic pillow lava	— spl

Sedimentary rocks

argillite	— argl
arkose	— ark, ak
asphalt	— asph
banded iron formation	— bif
bitumen	— bit
conglomerate	— cg, cong, cgl
chert	— ch, ct, cht
claystone	— cs
diatomite	— dtm
dolomite	— dol
edgewise conglomerate	— ewcg
graywacke	— gw
iron formation	— if
ironstone	— is, ir
limestone	— ls
marl	— ml
mudstone	— ms
phosphate	— phos
quartzose iron formation	— qif
sedimentary	— sed
shale	— sh
shaly iron formation	— shyif
sandstone	— ss
silstone	— slt, st
travertine	— trv

Metamorphic rocks

anthracite	— anth
amphibolite	— amp, amph
ellipsoidal greenstone	— el gs
gneiss	— gns
greenstone	— gs
injection gneiss	— inj gns
magnetic slate	— mag sl
magnetic graywacke	— mag gw
marble	— mb
metamorphic	— meta
paragneiss	— pagns
orthogneiss	— orgns
phyllite	— phy
quartzite	— qte,qtzt
schist	— sch
serpentine	— serp,sert
slate	— sl

General

batholith	— bath
dike	— dk
lopolith	— lopo
mineral	— min, mni
pluton	— plut
rock	— rk
specimen	— spec
stock	— stock
stone	— stn
volcano	— vol

Mineral and Metal Abbreviations

acanthite	— ac
actinolite	— act
adularia	— adu
aguilarite	— agu
aikinite	— ai
alabandite	— abn
albite	— ab
algodonite	— alg
allanite	— al;
alunite	— al
altaite	— at
amalgam	— amal
amblygonite	— amb
amphibole	— amp
analcite	— anal
andorite	— ad
anglesite	— ang
anhydrite	— anh
animikite	— anm
ankerite	— ank
annabergite	— anna
anorthite	— an
antimony	— Sb
antlerite	— ant
apatite	— ap
aragonite	— ara
argentite	— arg
argyrodite	— ay
arsenopyrite	— asp
arsenic	— as
atacamite	— ata
augite	— aug
autunite	— aut
azurite	— az
baddeleyite	— bad
barite	— ba
bauxite	— baux
becquerelite	— bec
bementite	— bem
biotite	— bio
bismuth	— Bi
bismuthinite	— bm
bornite	— bn
boulangerite	— bl
bournonite	— bo
brannerite	— bran
braunite	— br
bravoite	— bv
breithauptite	— btp
calamine	— cala
calaverite	— ca, cl
calcite	— calc
canfieldite	— can
carbon	— C, cbn
carbonate	— carb
carnotite	— carn
carrolite	— car
cassiterite	— cs, ct, cx
celesite	— cel
cerargyrite	— cer
cerussite	— ce
chalcocite	— cc
chalcopyrite	— ccp, cp
chalcanthite	— chin
chalcostibite	— cb
chalmersite	— cm
chert	— ch, cht
chloanthite	— cln, cl
chlorite	— chl
chromite	— cr
chrysocolla	— chr
cinnabar	— ci
clausthalite	— ct
clay	— clay
cobalt	— Co
cobaltite	— cob
coffinite	— cof
coloradoite	— colo
columbite	— colu
coolgardite	— cool
copper	— Cu
corundum	— cor
cosalite	— cos
covellite	— cv
crookesite	— ck
cubanite	— cn
cuprite	— cup
cummingtonite	— cum
cylindrite	— cy
datolite	— dat
descloizite	— des
diaphorite	— des
dickite	— dick
diopside	— diop
domeykite	— dom
dyscrasite	— dy
epiboulangerite	— epib
epidote	— ep
emplectite	— emp
enargite	— en
enstatite	— ens
erythrozincite	— eryz
erythrite	— ery
euxenite	— eux
famatinite	— fm
ferberite	— ferb
fergusonite	— ferg
feldspar	— fs, felds
fluorite	— fl
franckeite	— fc, fr

Table 5.2.7. Abbreviations—Continued

franklinite	— fk
freieslebenite	— freis
galena	— gn
gangue	— G
garnet	— gar
garnierite	— garn
geocronite	— gc
gersdorffite	— gf
gibbsite	— gibb
glaucodot	— gld
goethite	— goe, gt
gold	— Au
goldfieldite	— glf
goldschmidtite	— glm
gossan	— gos
graphite	— graph
greenockite	— gk
grunerite	— grun
guanajuatite	— gt
guitermanite	— gm
gummite	— gum
gypsum	— gyps
halloysite	— hl
hausmannite	— hs
hematite	— hem
hewettite	— htt
histrixite	— his
hornblende	— hb
huebnerite	— hub
ilmenite	— il
iron	— Fe
itabirite	— ita
jalpaite	— jl
jamesonite	— jm
jarosite	— jar
jasper	— jas
jasperoid	— jasp
jordanite	— jd
kalgoorlite	— klg
kaolinite	— kaol
keweenawite	— keew
kermesite	— km
krennerite	— kr
kyanite	— ky
labradorite	— lab
laterite	— lt
laumontite	— laum
lead	— Pb
lepidolite	— lep
limonite	— lim, lm
linnaeite	— ln
livingstonite	— lv
loellingite	— lo
luzonite	— lz
magnetite	— mag, mg
malachite	— mala, mc
manganite	— man, mng
marcasite	— mar, ms
matildite	— mt
melonite	— melon
meneghinite	— mene
metacinnabarite	— mc
miargyrite	— my
millerite	— ml
mineral	— min, mnl
mohawkite	— mk
molybdenite	— mo
muscovite	— mus, mv
nagyagite	— ng
naumannite	— nm
nephelite	— neph
niccolite	— nc
oligoclase	— ol
orthoclase	— or
orpiment	— orp
oxide	— ox
patronite	— pat
pearcite	— pc
pentlandite	— pn, pent
petzite	— pet
phosphate	— phos
pitchblende	— pthb
plagionite	— pg
platinum	— Pt
plumbojarosite	— pbj
polyargyrite	— plgy
polybasite	— plb
polydymite	— pm
polyhalite	— pyh
prehnite	— prh
proustite	— pu, pru
psilomelane	— psi
pyrargyrite	— pr
pyrite	— py
pyrochlore	— pyl
pyrolusite	— pyrl
pyromorphite	— pymp
pyroxene	— px, pyx
pyromorphite	— po, pyrr
quartz	— qz, qtz
quicksilver	— Hg
rammelsbergite	— rm
realgar	— rl
rhodochrosite	— rhod
rhodonite	— rho
rutile	— rt
safflorite	— sf
scheelite	— shee
semseyite	— sems
sericite	— ser, sr
serpentine	— serp, sert
siderite	— sid, sd
siegenite	— sg
silicate	— sil
silver	— Ag
sillimanite	— sill
skutterudite	— sk
smaltite	— sm
smithite	— smtt
smithsonite	— smith
specularite	— specul
sperrylite	— sperry
sphalerite	— sp
sphene	— sph
stannite	— stan
stephanite	— stp
stibnite	— stib
stilpnomelane	— stilp
stromeyerite	— strom
stutzite	— stut
stylotypite	— sty
sulfide	— sulf
sulfur	— S
sylvanite	— sv
talc	— tc
tantalite	— tan
teallite	— teal
tellurite	— tell
tellurium	— Te
temiskamite	— tk
tennantite	— tn
tenorite	— to
tetrahedrite	— td
tetradymite	— ty
thomsonite	— thom
thorite	— thor
thucholite	— thuc
tiemannite	— tie
tin	— Sn
titanite	— tit
topaz	— tz
torbernite	— torb
tourmaline	— tl
tremolite	— tm
troilite	— tr
tungsten	— W
turquoise	— turq
turgite	— turg
tyuyamunite	— tyuy
ullmannite	— ul
umangite	— um
uraninite	— uran
uranophane	— urp
vanadinite	— van
vein quartz	— vgz
vermiculite	— verm
violarite	— vl
voltzite	— vo
warrenite	— wa
whitneyite	— wh
willemite	— wy
wolframite	— wf
wollastonite	— woll
wulfenite	— wulf
wurtzite	— wurt

Color Abbreviations

black	— blk, bk, bl
blue	— blu, bl, bu
bright	— brt
brown	— brn, br, bn
buff	— buf, bf
chocolate	— choc, cho
dark	— drk, dk
drab	— drb
gray	— gry, gy
green	— grn
light	— lgt, lt
orange	— oran
purple	— purp, pp
red	— red, rd
steel	— stl
violet	— vio
white	— wht
yellow	— yel, yw, yl

Descriptive and Structural Terms

abundant	— abun
altered	— alt, alt'd
alteration	— altn
angular	— ang
arenaceous	— aren
argillaceous	— argill
asphaltic	— asph
band	— bnd
banded	— bndd
banding	— bndg
bed	— bd
bedded	— bdd
bedded, thick	— tkbd

Table 5.2.7. Abbreviations—Continued

bedded, thin	— tnbd	fold	— fld	pebble	— peb
bedding	— bdg	foliation	— foli	phenocrysts	— phen
below	— blw	foot wall	— F.W., FW	porphyry	— por
between	— btwn	formation	— frm	predominantly	— pred
bentonitic	— bent	fossil	— foss	proportion	— prop
bitumen	— bitum	fracture	— frac	rare	— rare
bleb	— blb	fracture cleavage	— frac cleav	refractory	— rfty
border	— bor	fragmental	— fragl	residue	— res
bottom	— bot	fragments	— frag	residual	— resd
break	— brk	friable	— fri	rock	— rk
broken	— bkn	glass	— gls	rocks	— rx
calcareous	— calc	gouge	— go	round	— rnd
carbonaceous	— crb,carbon	grade	— grad	sand	— sd
cement	— cmt	gradational	— gradl	sandy	— sdy
cherty	— chty	grain	— grn or grain	scarce	— scar
cindery	— cndy	grained	— grn, grnd	schistosity	— schis
cleavage	— clvg	granular	— gran	sediment	— sed
clear	— clr	ground	— gro	sheared	— shrd
coarse	— cse	gneissosity	— gny	sheeting	— shtg
compact	— cpt	hanging wall	— H.W., HW	siliceous	— silic
composition	— comp	hard	— hd	slickensides	— slick
concentration	— conc	heterogeneous	— hete	slightly	— sly
concretion	— concr	high grade	— h.g., hg	soft	— sft
condense	— condense	igneous	— ig	specimen	— spec
condition	— cond	impregnated	— impreg	sticky	— stky
conductivity	— conduct	inclined	— incl	stock	— stock
considerable	— consid	inclusions	— inclus	stone	— stn
cross-bedded	— x-bdd,xb	indurated	— indur	striae	— striae
crumpled	— crump	irregular	— irreg	strained	— strained
crushed	— crsd	interbedded	— intbd	streak	— strk
crystalline	— xln	intrusive	— intr	strike	— str
crystalline, coarsely	— xln-c, cxln	joint	— jo	strong	— strg
crystalline, finely	— xln-f, fxln	jointing	— jotg	structure	— struc
crystallized	— xld	lamination	— lam	tarnished	— tnshd
dark	— drk, dk	lean	— ln	thick	— tk
dense	— dns	lineation	— lin	thick bedded	— tkbd
dike	— dk	location	— loc	thin bedded	— tnbd
discard	— dscd	low grade	— l.g., lg	unaltered	— unalt'd
disseminated	— dissm	magnetic	— mag	undulating	— und
distributed	— distr	massive	— msv	unfavorable	— unfav
dragfold	— dfld	material	— mat	unoxidized	— unox
ellipsoidal	— elp	medium	— med	variable	— var
enriched	— enr	metamorphic	— meta	vein	— vn
extrusive	— extr	mineral	— min, mnl	veined	— vnd
fault	— flt, f	mineralized	— minz, mzd	veinlet	— vnlt
favorable	— fav	mineralization	— mzn	very	— vy, v
ferruginous	— ferr	mixed	— mxd	volcanic	— vol
fine	— fn	mottled	— mot	wash	— wash
fissle	— fss, fsl	oolitic	— ool	weathered	— wth
fissure	— fiss, fisr	opaque	— opq	weak	— wk
flakes	— fks	outcrop	— otp,oc,otc	with	— +, w
flow cleavage	— flclvg	oxidized	— ox	without	— w/o

ogy, color, grain size, composition, structure, and bedding. Geophysical logging, discussed in the next part of this review, is an essential part of coal work.

Figs. 5.2.1 to 5.2.6 are examples of types of logging formats and are presented to show the degree of variability in style. The first group (Figs. 5.2.1 to 5.2.4) show narrative-graphic styles, first with a variable interval format and subsequently a fixed interval format more useful for logging cuttings from specific sample intervals. Logs in Figs. 5.2.1, 5.2.2, and 5.2.4 use 8-1/2 × 11 in. (216 × 279 mm) forms, whereas Fig. 5.2.3 logs are 8-1/2 × 14 in. (216 × 356 mm) forms. Finally, Fig. 5.2.6 illustrates 8-1/2 × 11 in. (216 × 279 mm) data entry style used with the log shown in Fig. 5.2.5d. Other data entry style forms shown in Fig. 5.2.5 are 11 × 17 in. (279 × 432 mm) or larger. Also shown, Fig. 5.2.6c, is an example of the first of several pages in one style of summary log. Effective summaries may also be made by compiling data on standard logging format in use on a project, but at a reduced scale.

All of the logs illustrated are adequate to record varying amounts of geologic data and could be modified to suit a specific project. These examples are representative and have been collected over a number of years from various sources. Company identity has been removed to ensure confidentiality.

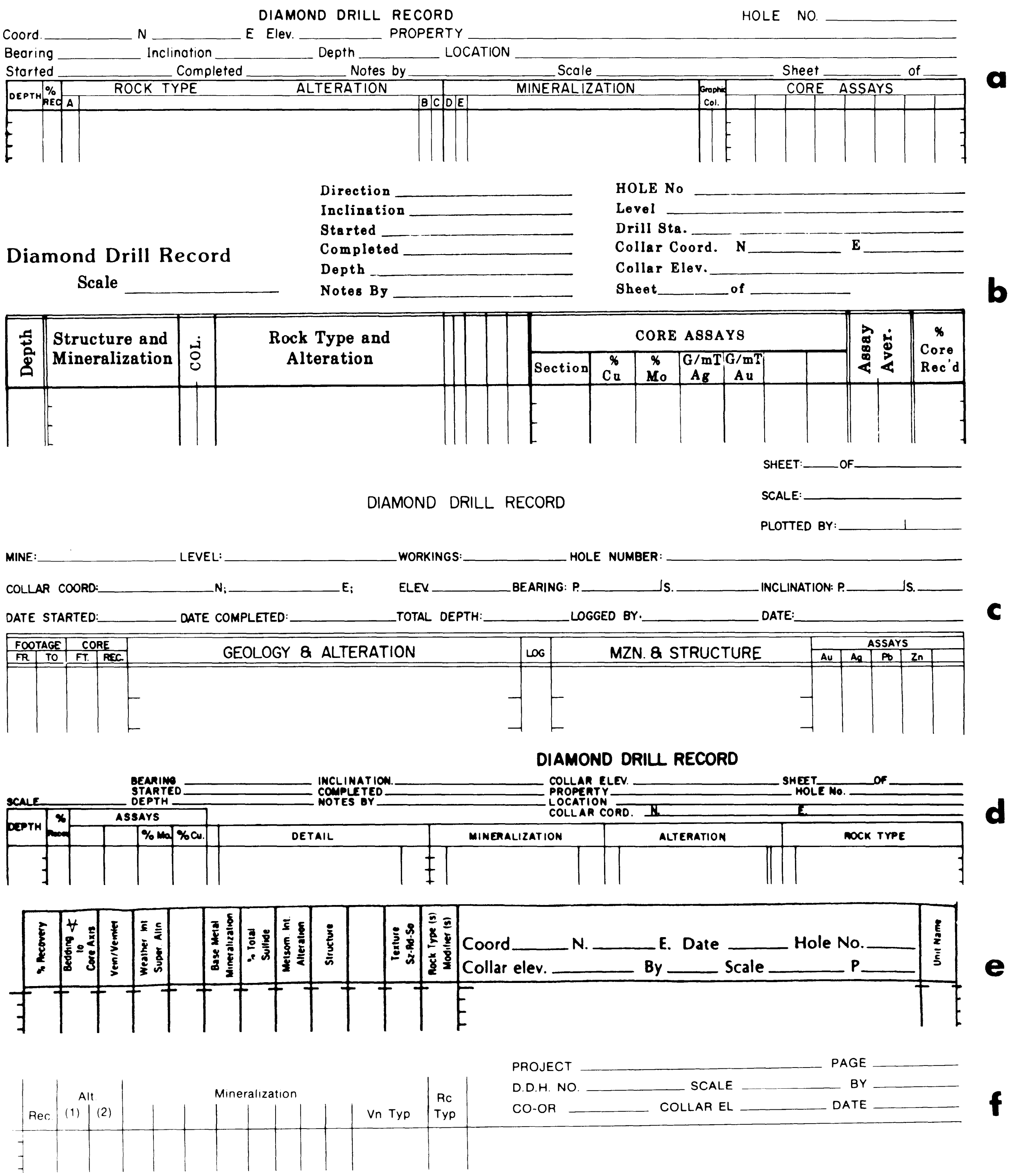


Fig. 5.2.1. Typical 8 1/2 × 11 -in. (216 × 279-mm) vertically arranged drill core or cuttings logging form of various companies.

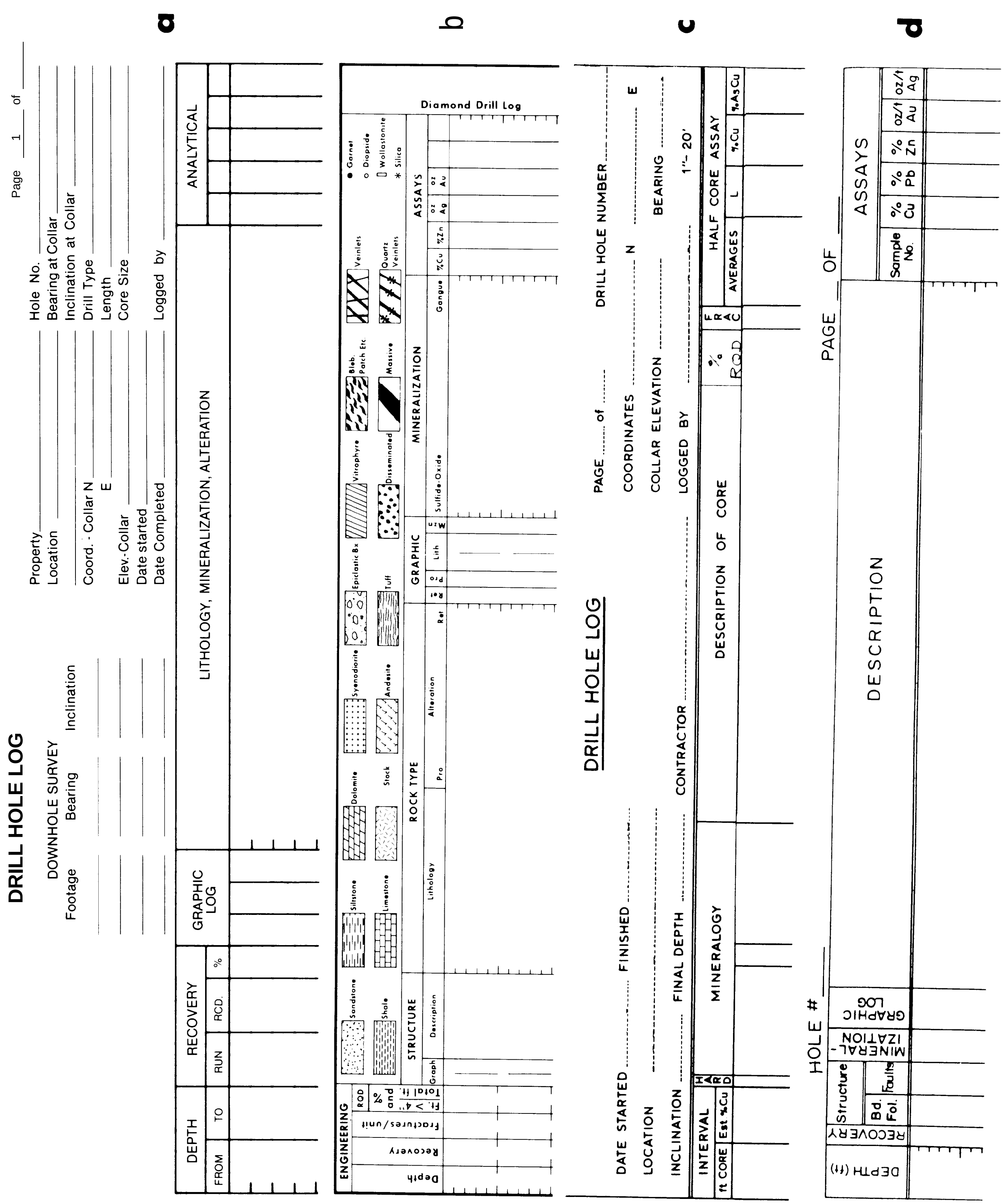


Fig. 5.2.2. Typical 8 1/2 × 11-in. (216 × 279-mm) horizontally arranged drill core or cuttings logging form of various companies.

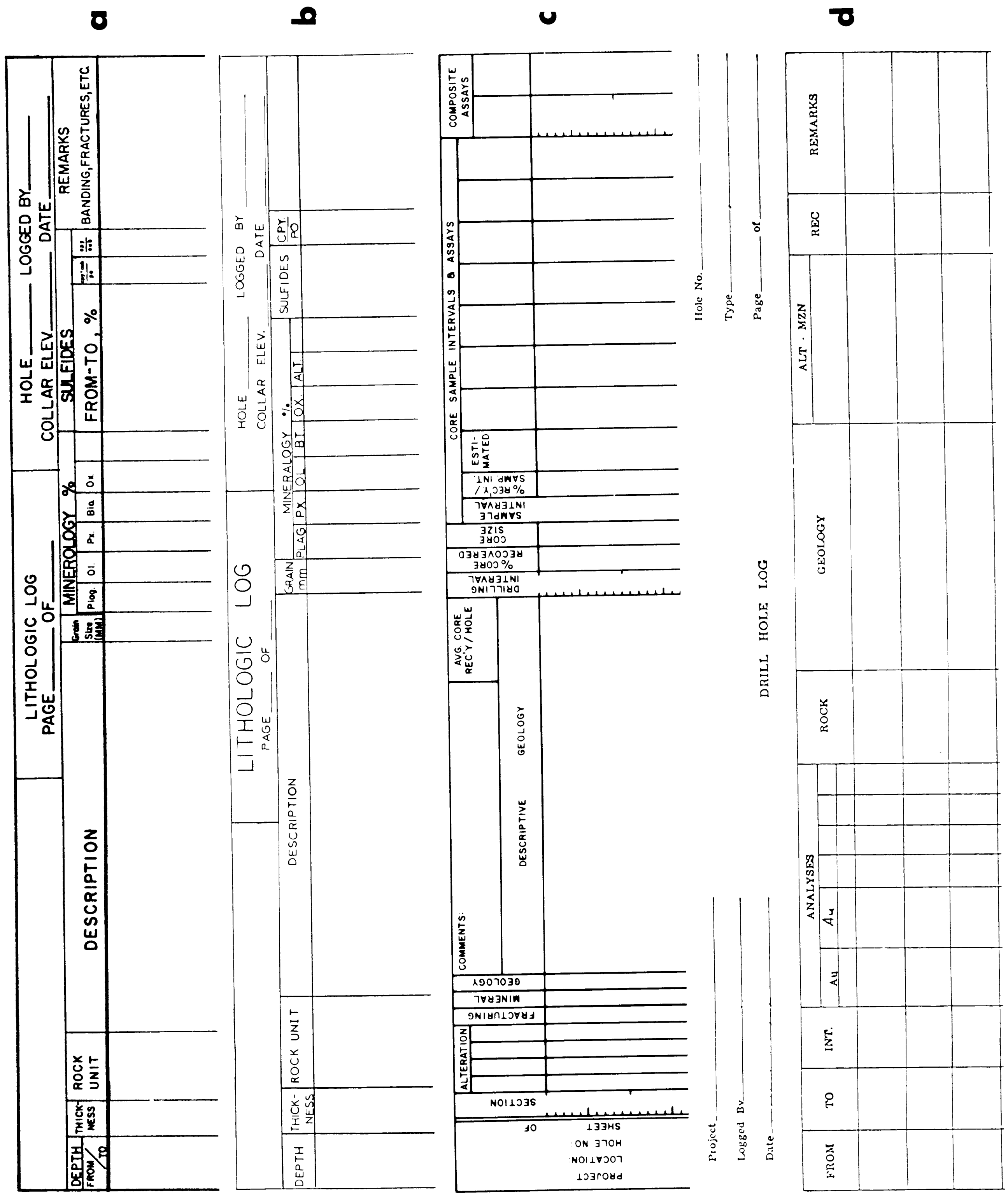
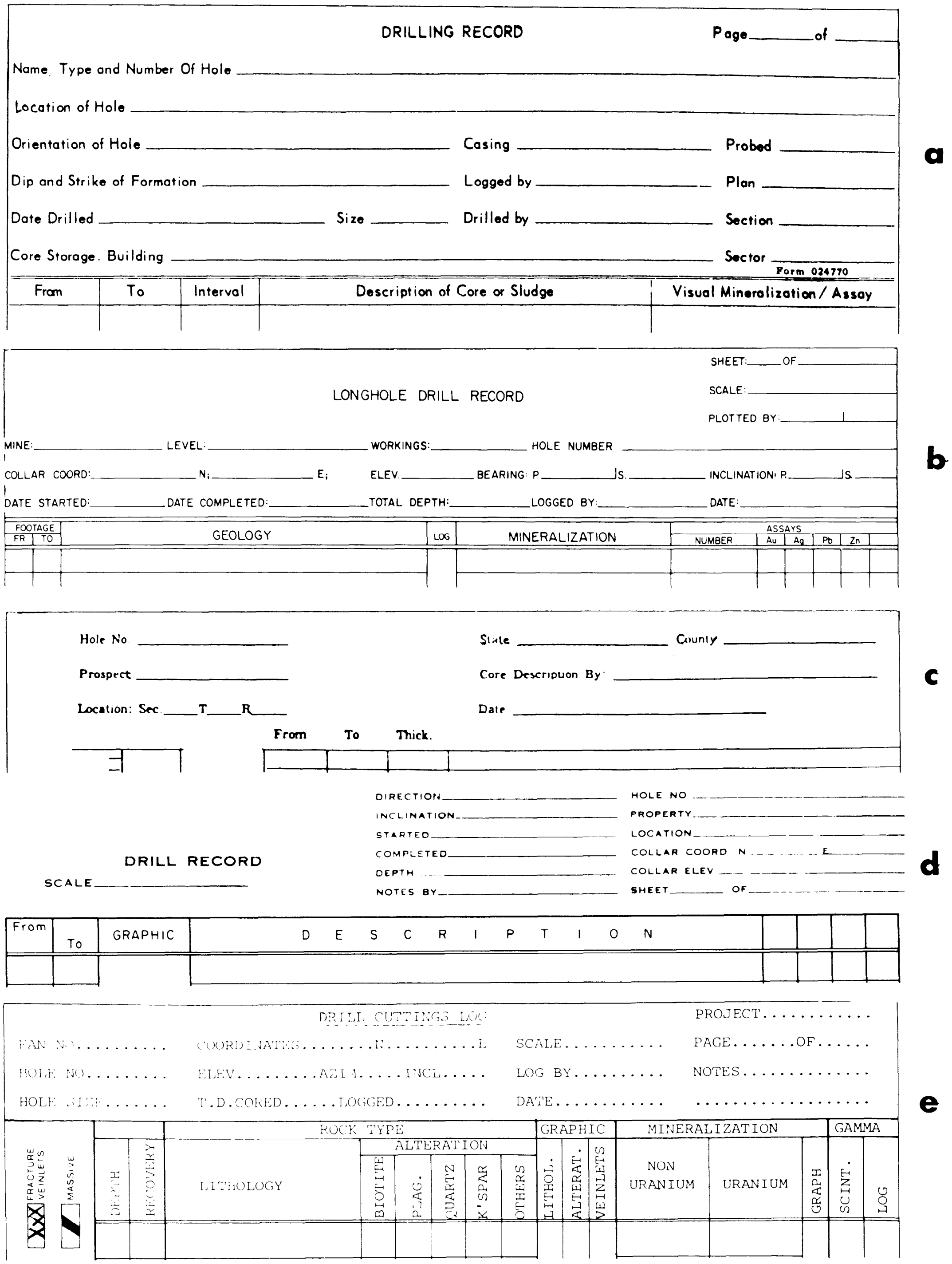


Fig. 5.2.3. Typical 8 1/2 × 14-in. (216 × 356-mm) horizontally arranged drill core or cutting logging form of various companies.



DRILLING RECORD Page ____ of ____
Name, Type and Number Of Hole ____
Location of Hole ____
Orientation of Hole ____ Casing ____ Probed ____
Dip and Strike of Formation ____ Logged by ____ Plan ____
Date Drilled ____ Size ____ Drilled by ____ Section ____
Core Storage, Building ____ Sector ____
Form 024770

From	To	Interval	Description of Core or Sludge	Visual Mineralization / Assay

a

LONGHOLE DRILL RECORD
SHEET: ____ OF ____
SCALE: ____
PLOTTED BY: ____
MINE: ____ LEVEL: ____ WORKINGS: ____ HOLE NUMBER ____
COLLAR COORD: ____ N; ____ E; ELEV. ____ BEARING: P ____ S. ____ INCLINATION: P ____ S. ____
DATE STARTED: ____ DATE COMPLETED: ____ TOTAL DEPTH: ____ LOGGED BY: ____ DATE: ____

FOOTAGE FR	FOOTAGE TO	GEOLOGY	LOG	MINERALIZATION	ASSAYS NUMBER	Au	Ag	Pb	Zn	

b

Hole No. ____ State ____ County ____
Prospect ____ Core Description By: ____
Location: Sec. ____ T ____ R ____ Date ____

From	To	Thick.

c

DRILL RECORD
SCALE ____
DIRECTION ____ HOLE NO ____
INCLINATION ____ PROPERTY ____
STARTED ____ LOCATION ____
COMPLETED ____ COLLAR COORD N ____ E ____
DEPTH ____ COLLAR ELEV ____
NOTES BY ____ SHEET ____ OF ____

From	To	GRAPHIC	D E S C R I P T I O N				

d

DRILL CUTTINGS LOG
PROJECT...........
FAN NO.......... COORDINATES........N..........E SCALE........... PAGE.......OF......
HOLE NO......... ELEV.........AZIM.....INCL..... LOG BY.......... NOTES..............
HOLE SIZE....... T.D.CORED......LOGGED.......... DATE............ ....

FRACTURE VEINLETS / MASSIVE	DEPTH	RECOVERY	ROCK TYPE: LITHOLOGY	ALTERATION: BIOTITE	PLAG.	QUARTZ	K'SPAR	OTHERS	GRAPHIC: LITHOL.	ALTERAT.	VEINLETS	MINERALIZATION: NON URANIUM	URANIUM	GAMMA: GRAPH	SCINT.	LOG

e

Fig. 5.2.4. Typical 8 1/2 × 11 -in. (216 × 279-mm) vertically arranged drill core or cutting logging form. (a) variable sample length, (b,c,d,e) fixed, constant sample length.

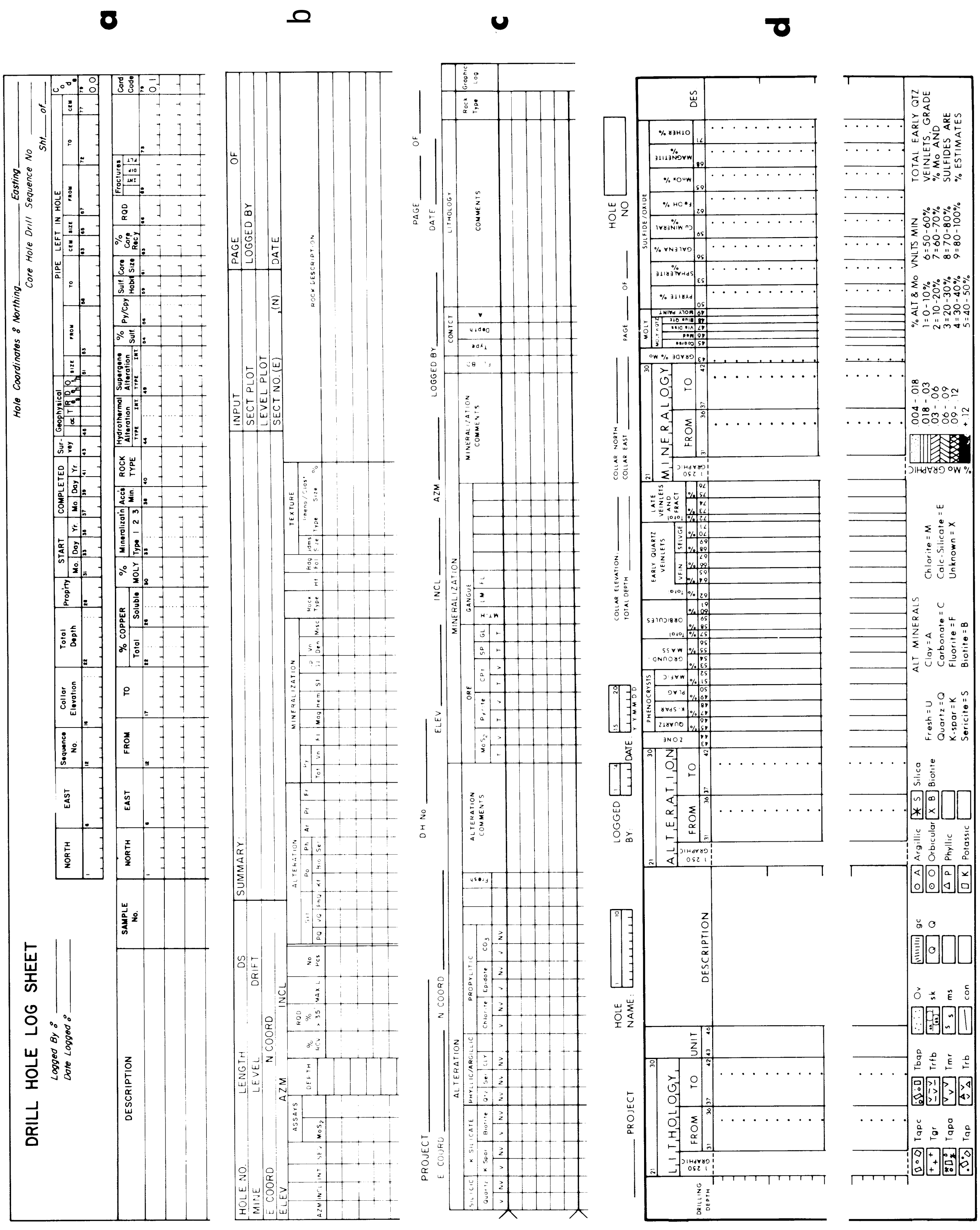


Fig. 5.2.5. Typical 11 × 17-in. (279 × 432-mm) or larger data entry pad style logging forms.

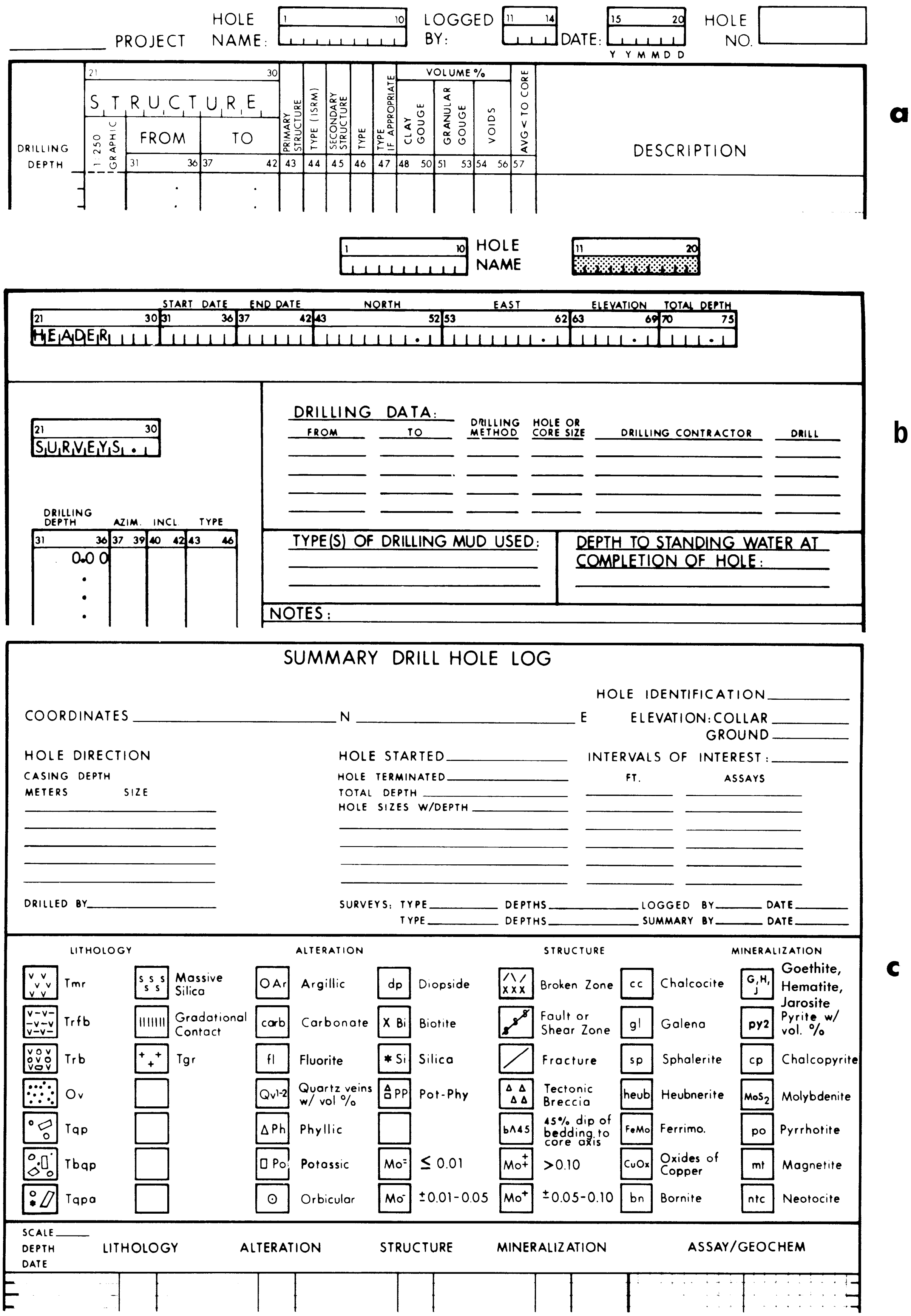


Fig. 5.2.6. Typical 8 1/2 × 11-in. (216 × 279-mm) specialized logs, header sheets, and summary logs used with data entry log form 5.2.5d.

5.2.7 GEOPHYSICAL LOGGING

Although not commonly used in typical hard-rock mineral applications, the use of in-hole electrical or geophysical logs, which record measurements of electrical, nuclear, or physical properties of rocks and adjacent fluids, is essential in coal and uranium resource evaluation. These logs, supplemented by good core or cutting logs that describe key features listed in Table 5.2.1, form the basis for geologic work and resource estimation in these industries.

Geophysical logging was developed to serve the special needs in subsurface, noncoring, work in the petroleum industry. The techniques have advanced to the status of an extremely useful, dependable, accurate, highly specialized discipline used in lithologic identification, correlation, depositional environment analysis, and formation evaluation. Only the use of logs for lithologic identification for correlation will be discussed here.

Numerous types or suites of logs are available, and descriptions of each can be found in a publication by Pickett (1971), which is particularly good yet brief, or more in-depth treatments by Asquith and Gibson (1982), the Schlumberger (Anon., 1972, 1974), or Dresser-Atlas (Anon., 1974 through 1979) sets of complementary interpretation manuals. BPB Instruments (Anon., 1981) has published a *Coal Interpretation Manual* that is useful, as are numerous papers by Reeves such as those in Argall (1979). The Plouffe (1981) summary is excellent, and the older Dodd and Eschliman (1972) review directed towards uranium work is good, as is Fitch (1990). Finally, Peters (1987) provides a useful short overview. Some supervised training in log interpretation is helpful and allows one to learn basic techniques quickly. A clear understanding of log heading, the multitrack nature of the logs, curve deflection direction, scales, and recording format is essential as there are a significant number of different log types, suites, and recording conventions.

Ward (1984), Chironis (1982), Wood et al. (1983), and Merritt (1986) provide discussions of use of various logs as well, and the summary which follows, primarily relating to coal, is largely abstracted from Chironis and Wood et al. Further, the work by Wood et al. provides a good discussion of various logging methods. The logs, when properly interpreted, provide a substantial quantity of useful geologic data, particularly on noncore holes. Some tools and measurements such as resistivity or spontaneous potential require fluid-filled uncased holes while variations in natural radioactivity, determined from the gamma log, can be measured in cased holes. In general, the systems provide the best information on porous, fresh sediments; however, variations in natural radioactivity associated with potassium-bearing clays and micas of hydrothermal alteration halos in crystalline rocks can commonly be clearly identified and mapped.

It is almost essential that suites of logs, such as gamma, spontaneous potential, resistivity, density, or other combinations, are obtained. The down-hole logging tool, sonde, typically is constructed to measure several properties, hence it provides multiple logs in a single pass.

Modern logging systems commonly supply output in the form of pictorial logs, "squiggly lines," and digital tapes or disks that feed directly into computer data processing and graphical output systems. Specific formats and products, some highly sophisticated in nature, vary with software vendor, client purpose, and need. In general, computer systems tend to provide a lithologic identification associated with the trace of varied geophysical logs, or at least space to append one manually for calibration, reconciliation, or simply as an additional set of of data to be used in geologic evaluation. Lithology determination and attendant correlation without the geologists or "mud log" can be erroneous and should be avoided.

A summary of logging techniques follows.

5.2.7.1 Spontaneous Potential (SP)

This is one of the most common logs found in all older work but not, unfortunately, with some modern coal-logging systems. The system measures natural currents developed in drillholes at lithologic contacts due to differences in salinity between borehole fluids and water in porous formations as the formation is invaded with the drill fluid. It is good for differentiating sand from shale, environmental analysis such as "fining up" sequences, but problems in distinguishing shale from coal are common.

5.2.7.2 Natural Gamma Ray

This method measures the natural gamma rays emitted by isotopes of uranium, thorium, and potassium. Coals and lignite are usually low in radioactivity, whereas the rock that usually surrounds coalbeds, shale, is high in potassium-rich clay and sometimes uranium minerals. Sometimes, however, uranium minerals from groundwater precipitate in the coalbeds, making the beds less apparent on the log and, additionally, limestone, like coal, tends to be low in radioactivity.

5.2.7.3 Bulk Density (Gamma-Gamma Density)

Interaction between induced gamma rays and electrons in the material surrounding the borehole is measured. Porous materials with light elements, such as coal and lignite, have a low apparent density on the gamma-gamma density log. This log alone may be used to identify thickness and structure of coal seams, but is even more definitive when used in combination with a caliper log that shows changes in hole size. Contract between high-density limestone or moderate-density sandstone with low-density shale or coal is quite clear.

5.2.7.4 Neutron Density

This method involves measurements similar to bulk density, except that the neutral tool emits a stream of high-energy neutrons that interact with hydrogen atoms. Because coal and lignite are hydrogen-rich and porous, and the pores usually are filled with water that is hydrogen-rich, these beds have a higher porosity index on neutron logs than associated shales or limestones.

5.2.7.5 Sonic (Acoustic) Velocity

This method measures the velocity of the compressional wave component of an acoustic signal between a transmitter and a receiver. The interval transit time is usually higher for coal than for surrounding rock. The more highly compacted coals have lower transit times. Lignite's range is 130 to 150 micro-sec/ft, while anthracite's is 120 or less. Accuracy is affected by variations in hole size and condition.

5.2.7.6 Resistivity (Electrical)

Resistivity measurements involve determining the current flow between an electrode in the logging tool and another electrode in the ground at the surface. Because coal is a poor conductor of electric current, it exhibits high resistivity values. Tight sandstone or limestone beds, however, may be confused with coal. Also the identity and thickness of a coalbed may not show up accurately, depending on the electrode spacing with respect to bed thickness. Newly developed focused resistivity tools can measure true coal seam resistivity and thus provide better resolution. Resistivity remains as one of the best logs for lithologic interpretation and correlation.

5.2.7.7 Laterolog (Electrical Conductivity, Induction Log)

This method, which measures the electrical conductivity of a bed, sends out the signal horizontally. High-rank coals exhibit low conductivity; poorer quality coals vary with ash content. Bed boundaries are likely to be more accurate than on the resistivity log but, as with the resistivity log, this log may confuse coal with tight sandstone or limestone.

Figs. 5.2.7a and 5.2.7b are summaries of typical log signatures or character to be anticipated from various lithologies during geophysical logging. Fig. 5.2.7a from Irving (1981) shows the highly irregular pattern commonly seen in actual logs, whereas Fig. 5.2.7b from Wood et al. (1983) is more stylized in nature. The suites of logs shown are slightly different.

Fig. 5.2.8a is a typical log from uranium deposit evaluation in Wyoming. Four curves are shown: two at different sensitivities recording the natural radiation, gamma rays in counts per second (cps) in the borehole environment, and one each for spontaneous potential in millivolts (mv), and for resistivity R in ohms. Deflection of the natural gamma curve to the right on the scale of 0 to 2000 cps is off the sheet, while the higher-range scale 0 to 100,000 cps clearly shows the nature of the uranium distribution, Measurements are taken at 1/2-ft (152-mm) intervals and then converted, through use of a constant (*K* factor) that is different for each instrument, to $\%eU_3O_8$. Modern logging systems perform this calculation automatically and provide printouts of the values. Periodic calibration with known standards, a test pit, and chemical analysis of core for U_3O_8 are essential to assure accuracy of the eU_3O_8 determinations. The two spikes, shaded, at 500 and 542 ft (152 and 165 m) on this log correspond to 3.5 ft (1.1 m) of 0.270% eU_3O_8 and 8.5 ft (2.6 m) of 0.171% eU_3O_8, respectively. The remaining two curves, *SP* and *R,* show the characteristic left or negative deflection of the *SP* curve and right deflection of the *R* curve corresponding to sandstone (stippled) horizons. Less deflection can be seen in the two curves adjacent to the shale or shaly (horizontal lines) horizon.

Fig. 5.2.8b is one of many types of logs displaying typical responses seen in eastern US coal work. In this case, gamma, density, and resistivity tracks or curves are illustrated. Sand and shaly sections can clearly be seen in the gamma and resistivity logs, the shale showing the higher natural radioactivity of potassium, thorium, and uranium concentrated in clays.

Coal is clearly identifiable due to the low radioactivity (gamma log), the low density shown on the density log, and the very high resistivity. Four coalbeds which have been shaded are identified on this log. Sandstone and shale horizons have not been differentiated, as in Fig. 5.2.8a.

5.2.8 GEOLOGIC MAPPING

Surface or underground mapping requires the same detail, uniformity, standardization, and systematic, unbiased, objective, data collection and recording as described in the previous section on logging. The general comments and philosophy described there apply to mapping as well. Note-taking, abbreviation, and symbology are best if they employ company-wide established methodology, reflecting common initial training of personnel involved. In addition to being consistent with logging, mapping data must be accurately located and tied to known, preferably surveyed, geographic locations.

Brunton compass and tape(s) are commonly used to construct ground outlines for data recording, to locate outcrops, and to tie in culture or survey control, and if done carefully, they will provide sufficient accuracy. This is a fairly straightforward procedure and consists of stretching a cloth measuring tape or tapes from or between known points and determining the bearing of the tape with a compass. This “tape line” is then plotted to scale in its proper orientation on the field note sheet and tick marks posted and identified each 10 ft (3 m) along the bearing of the tape line. Following this, offset measurements are taken at right angles to the cloth tape from the tape to the edge of the drift or pit bench. A small pocket tape is used to take these offset measurements at 10-ft (3-m) intervals along the cloth tape. Points corresponding to these offsets are then plotted on the field note sheet, connected, and an outline of the drift or edge of the pit bench is thus created for geologic note taking. Figs. 5.2.9a and b illustrate the methodology of developing a ground outline and shows some simple geologic notes.

Geologic field note-taking is commonly done on relatively large scale in typical mine-related work such as 1 in. = 20 ft or 1 in. = 50 ft (1:240 or 1:600). Smaller scales, 1 in. = 100 ft and 1 in. =200 ft (1:1200 and 1:2400), are also used, generally to collect data on overall resource setting or to simplify more detailed work in the mine itself. For regional work, scales of 1 in. = 1000 ft or 1 in. =2000 ft (1:12,000 or 1:24,000) may also be appropriate. Some variability is necessary, and the actual mapping scale used will depend on needs of specific projects. Advance planning here is useful. While detailed, 1 in. = 20 ft (1:240) mapping in a vein or massive sulfide deposit is desirable, attempting to get the same detail in a 15-million tpy (13.5-Mt/a) open cut coal mine or a 2-million tpy (1.8-Mt/a) underground coal operation would be inappropriate.

Mapping techniques are described in a number of good references, which vary somewhat in perspective. Proper supervised training is desirable. Peters (1987) provides considerable detail in a good description of general surface, surface open pit, and underground work. He describes an outcrop mapping system in which multiple acetate overlays (Fig. 5.2.10a), generally three, are superimposed over a base map or photo and used to take notes describing geology, mineralization, and alteration, respectively, for each outcrop. A similar system using photographically superimposed topography on orthophotos is described in more detail by Atkinson and Erickson (1984) in a volume that reviews the practice of mining geology at a number of properties such as the Santa Rita District, NM (Ahrens, 1984). The field manual by Compton (1962, 1985), provides good coverage of geologic mapping techniques as do the extremely useful Geological Society of London’s *Handbook* series previously mentioned. Earlier textbooks by McKinstry (1948) and Forrester (1946) remain the best sources on techniques in mine mapping. More recent articles by Rutherford et al. (1984), Ledvina (1986), and Krausse et al. (1979) describe the increasing use of geologic mapping in underground coal mines, stressing the importance of lithology and roof rock characterization. Data gained and properly used enhances geologic understanding, improves safety, and supports increased productivity and, ultimately, mine profitability. A report by Moebs and Stateham (1984) summarizes contract studies investigating the relationship between geologic factors and roof stability in coal mines. The work is well referenced and clearly identifies mine and core mappable geologic features that control roof stability. The reference, Finkl (1988), contains a number of sections on basic field principles from a construction site evaluation perspective and is useful. Volumes 69 and 70 in the *Benchmark Papers in Geology* series (Lacy, 1983) are compila-

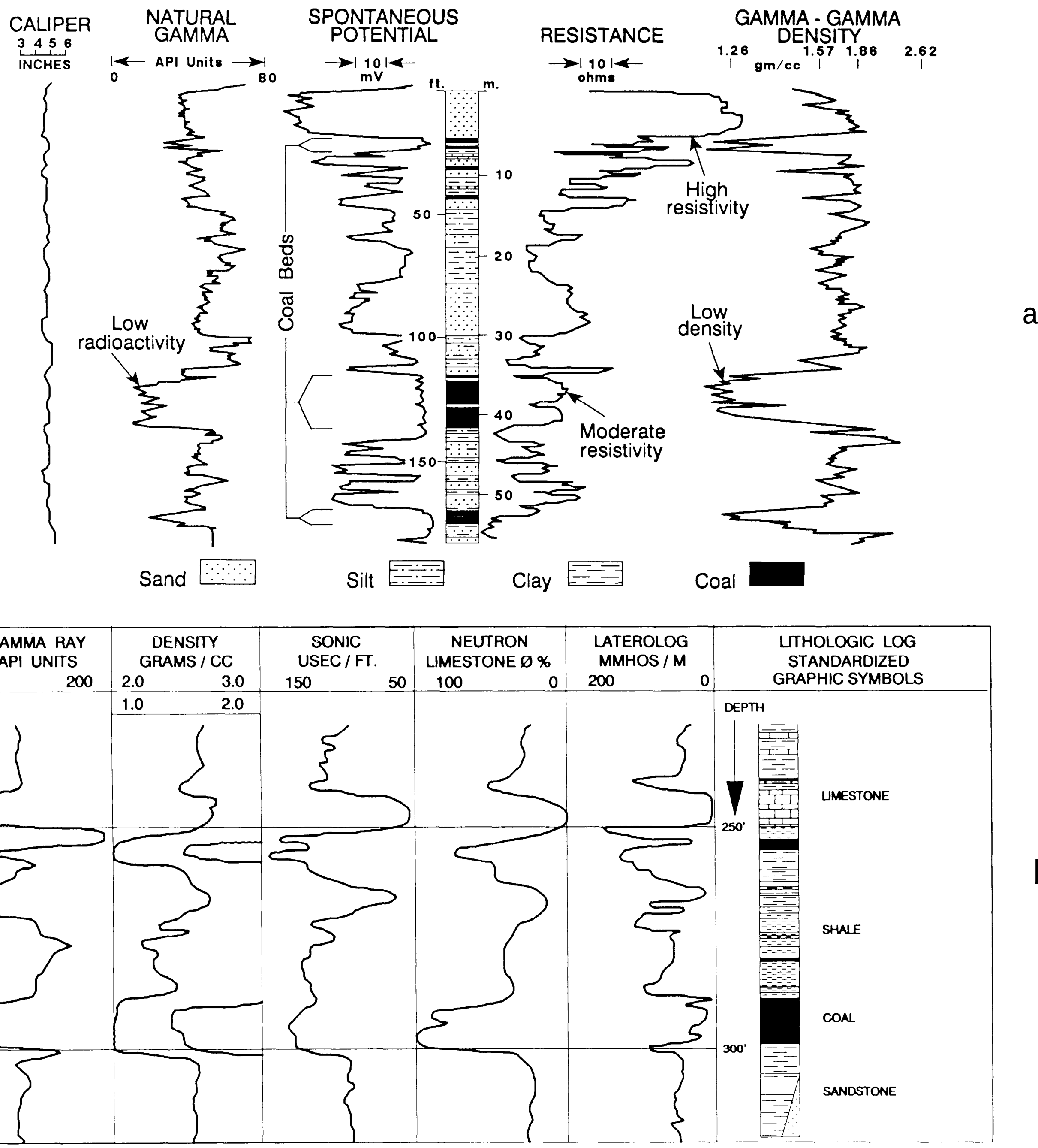


Fig. 5.2.7. Typical geophysical log characteristics for various lithologies. (a) is modified after Irving, 1981, while (b) is modified after Wood et al., 1983.

tions of key geologic papers in mining and exploration geology, and although abstracted and only partially reproduced, they are valuable as they identify key significant papers with bibliographies and provide many useful illustrations.

Early papers by Linforth (1910, 1933), McLaughlin and Sales (1933), Billingsley (1933), McLaughlin (1933), Sales (1941), or James (1946), although somewhat dated, clearly show techniques and discuss the importance of careful geologic mapping. A recent paper (Garmoe, 1984) is one of the few that contains a direct reproduction of an unedited, underground field sheet showing the technique referred to as the Anaconda method. The papers by Bakken and Einaudi (1986) and Einaudi and Bakken (1988) demonstrate the usefulness of detailed geologic mapping. In this study of the Carlin deposit, gold grade was

Table 5.2.8. Surface Mapping: General Points

Aluminum sheet holder for 8-1/2 in. × 11 in. (216 mm x 279 mm) gridded sheets, topographic bases, or airphoto base maps, are superior to bound notebooks as they allow sheet removal, filing, and reduce chance of data loss.

Adequate control is critical, and data must be tied with survey or Brunton and tape to known locations as survey points, claim corners, mine workings, key topographic features as ridges, valleys, stream junctions, and roads.

All maps need north arrows and coordinate data to determine orientation.

Data sheets must be clearly identified with location, scale, geologist, and date and, if an overlay system is used, all should be permanently attached to base map or photo and further identified with photo number.

Notetaking should be clear, legible, and accurate as "clear cut sketches indicate understanding . . . muddled mussy sketches denote confusion . . ." (Sales, 1941, p. 19.03). Keep sheets clean by keeping a folded piece of paper on which to rest hand during note taking in clipboard. Use sharp pencils.

Set up field sheet map index system in advance.

Use standardized notetaking, abbreviations, and symbols and post data collected regularly on base maps. Plot strikes and dips in their proper attitude. Plan ahead so outcrop outlines and notes complement and do not conflict with each other.

Table 5.2.9. Pit Mapping: Added Key Points

Safety: Be careful; pit benches and walls are frequently unstable and falls are common. Additionally, heavy equipment operators frequently are unable to see individuals close to the equipment. Be sure they know of your presence.

Use Brunton compass and tape to tie mapping to available survey points by drawing ground outline at toe of bench as in Fig. 5.2.9.

Clearly identify pit and bench location on 8-1/2 in. × 11 -in. (216-mm × 279-mm) grid sheets and use note-taking system parallel to one set of grid lines on paper or perpendicular to trend of workings.

Plan ahead and use systematic data posting. (Overlays may or may not be appropriate.) Plot lithologic data on one side of ground line, alteration on the other, mineralization close to actual location.

found to be correlatable with mappable alteration features, extremely useful information both from a genetic and a practical, day-to-day operating perspective.

Tables 5.2.1 through 5.2.10 contain summaries of some key points to be considered in geologic mapping on surface, in pits, or underground.

Fig. 5.2.10a from Peters (1987) illustrates the overlay system while Fig. 5.2.10b from Ahrens (1984) shows good technique in underground vein mapping. Figs. 5.2.11a and 5.2.11b from Peters (1987) illustrate multiple bench mapping with a minimum of detail and then simplified geologic note-posting, compilation on an office base map.

Table 5.2.10. Underground Mapping: Added Key Points

Be careful and constantly aware of the back, ribs, stopes, raise locations, and moving equipment. Test by light tapping with pick. Check for bad air. Keep a candle; if it will not burn in old workings there is insufficient air to breath adequately over sustained periods. Supervised training is mandatory.

Notesheet should be clearly identified as to mine, level, scale, dates, and geologist.

Tape and compass measuring should be tied to survey control usually located in the back of the workings. Control must be carried from drifts through raises and stopes to ensure proper location. This is time consuming but important. Ground outlines should be accurately drawn (Fig. 5.2.9).

A blotter paper in the aluminum sheet holder in addition to folded paper allows the geologist to keep notesheets clean and dry. A clean rag to dry hands and hat brim aids in keeping field sheets clean.

Mapping should be at waist height and ground outlines plotted by taking offset measurements from tape.

Colored pencils should be kept to a minimum of two to three colors as they frequently bleed when damp. Plan ahead in note taking.

Advance mapping should be on a day-to-day basis to allow ready access to clean drift walls. Old ribs should be washed or picked clean. A geologist cannot map what he cannot see

Notetaking should be clear and systematic with appropriate detail as needed. Too much detail, if uncluttered, is far superior to too little detail.

Advance should be posted on office base maps and dated shortly after mapping. Note sheets should be systematically filed for ready retrieval and use.

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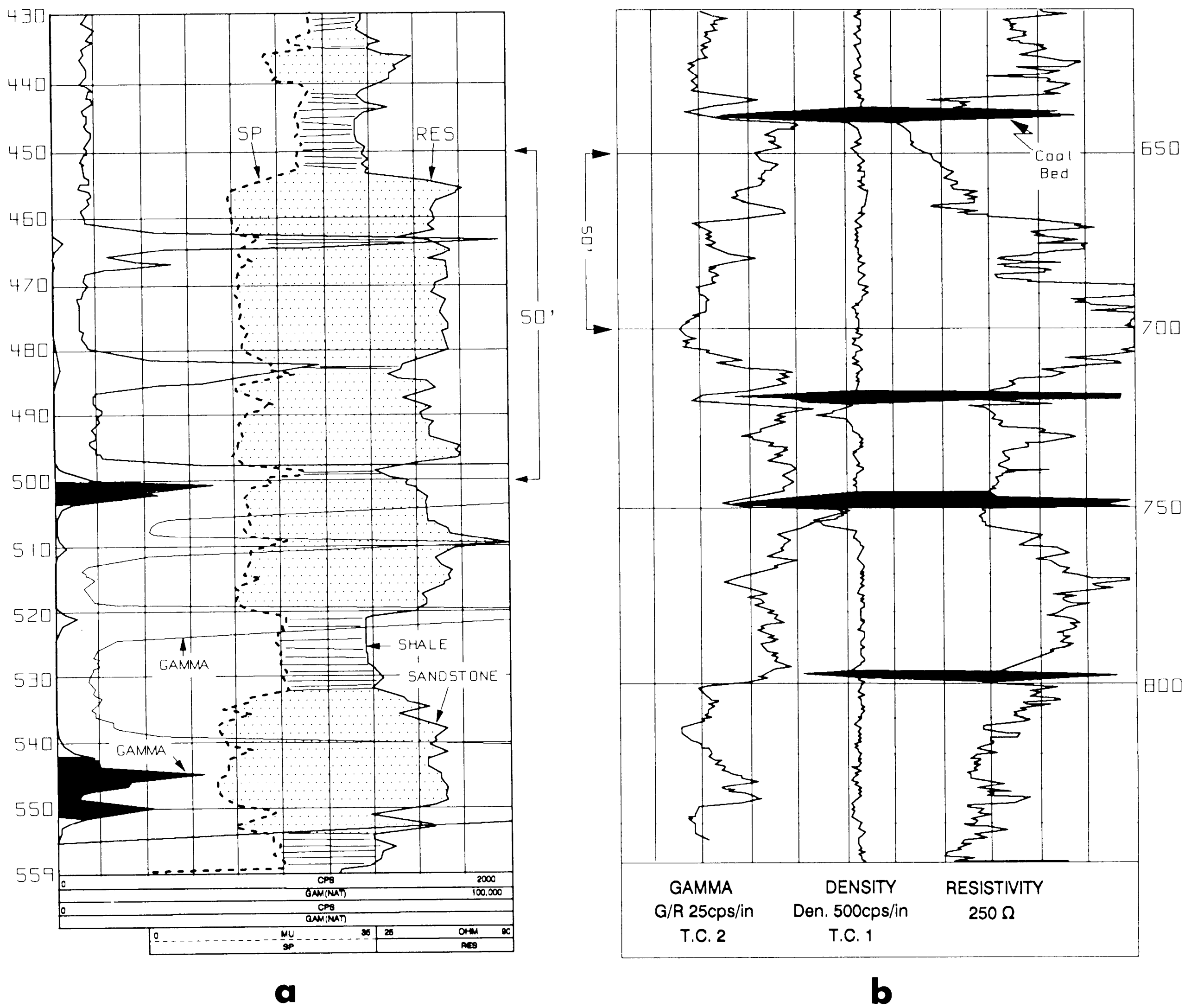

Fig. 5.2.8. Annotated geophysical logs. (a) Wyoming U_3O_8 deposit log showing typical patterns and characteristic gamma spike indicating ore grade uranium. (b) Eastern US coal deposit log with four coalbeds shaded based on low density, low radioactivity, and moderate to high resistivity.

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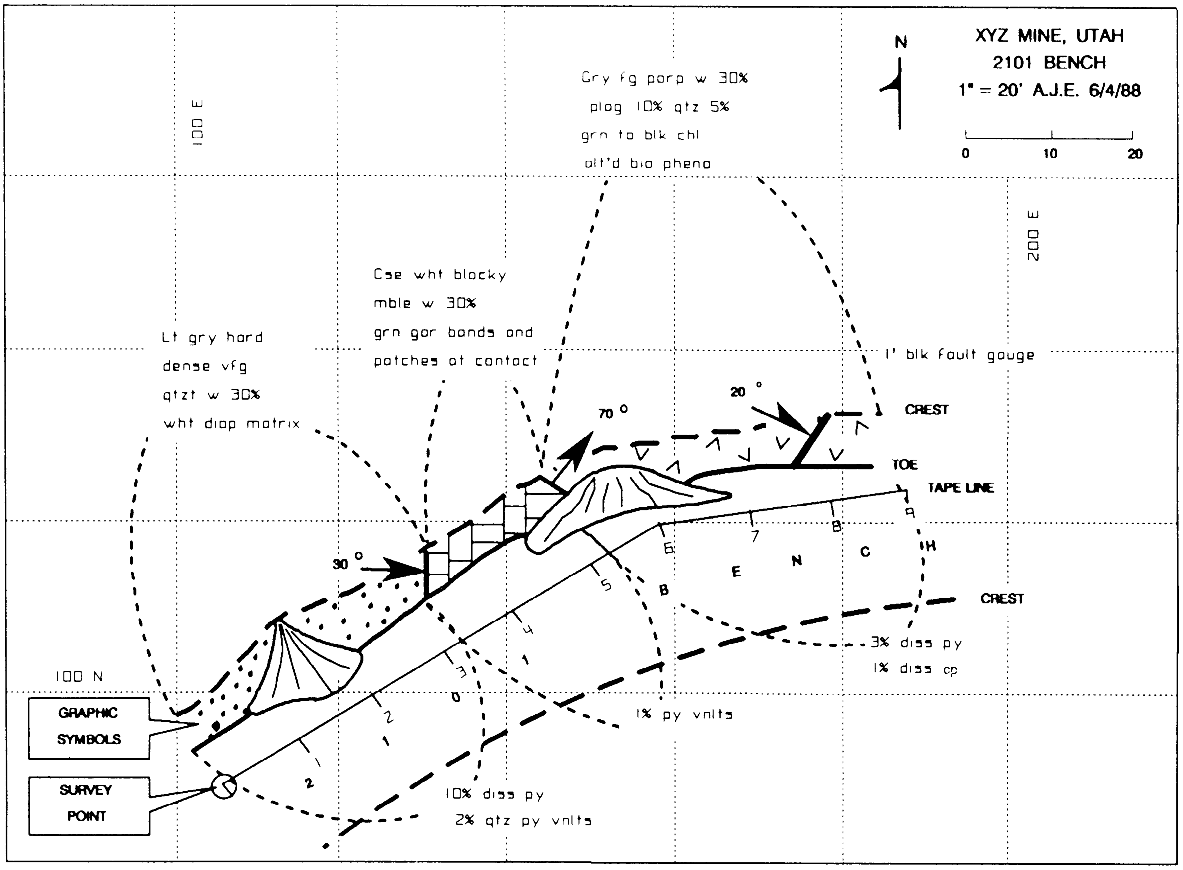

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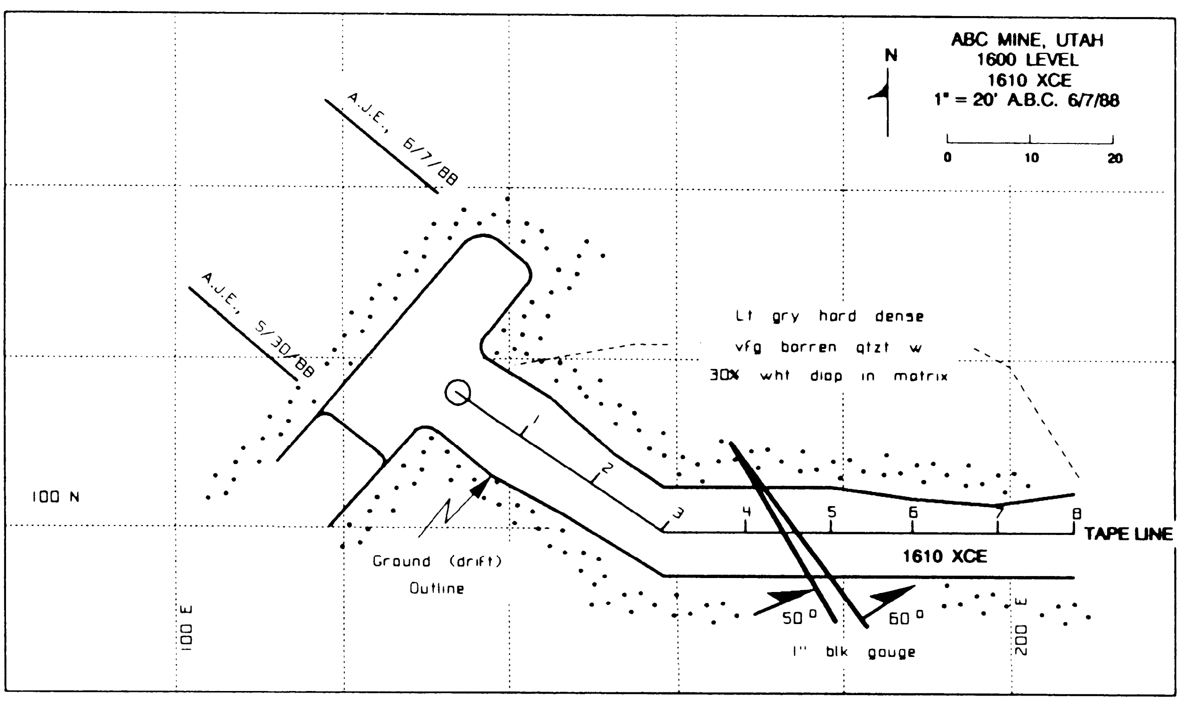
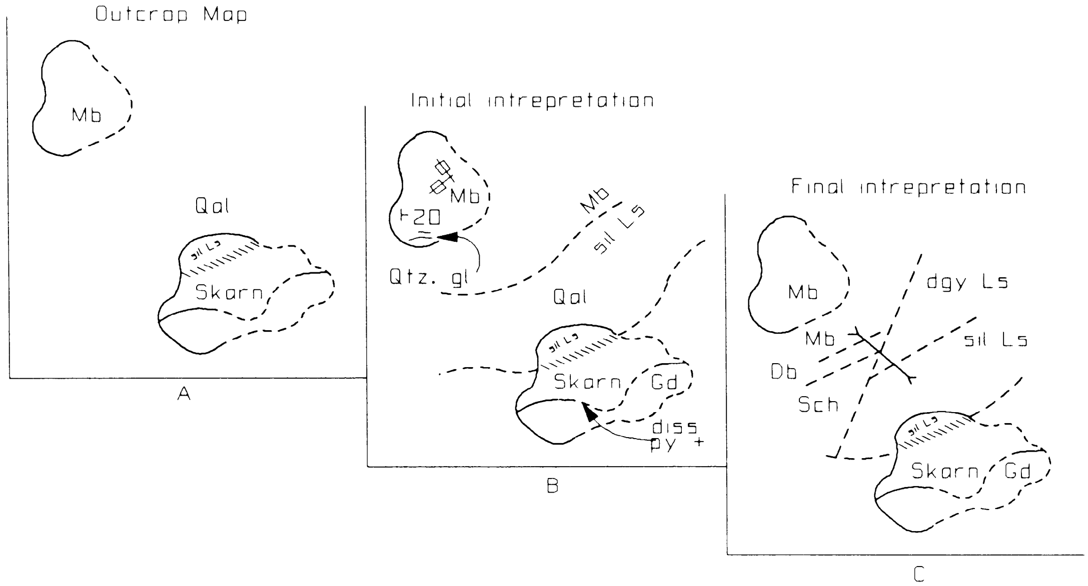


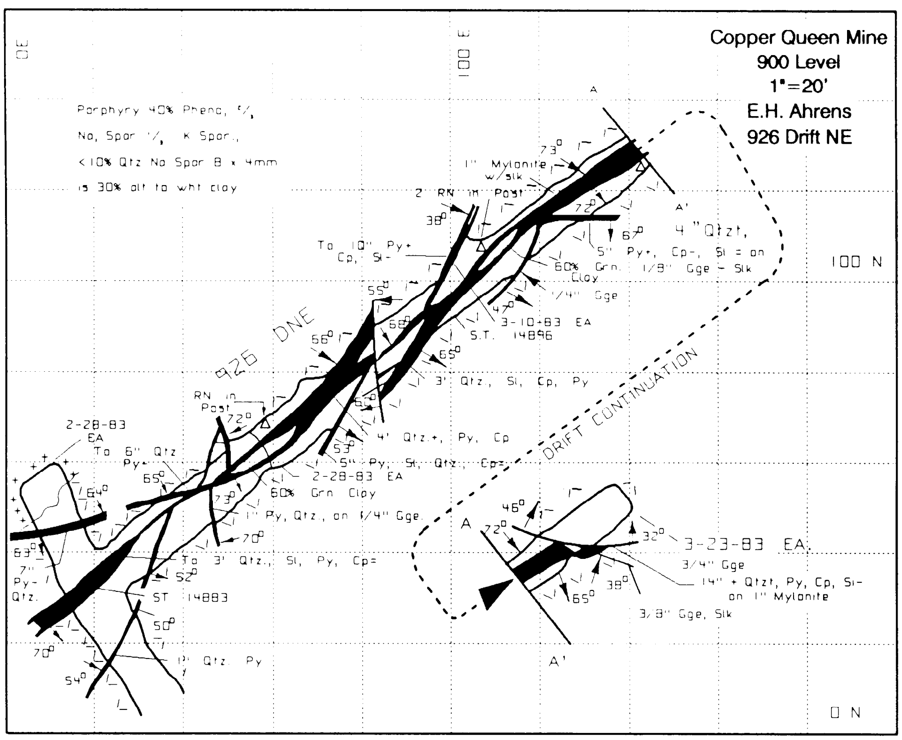
b

Fig. 5.2.9. Simple geologic field note sheet. (a) An example of mapping a pit bench showing method of control, ground outline, and geologic notes. (b) An underground drift map showing control, ground outline, and geologic notes. Location, scale, date, geologist, and orientation are clearly indicated on each example.



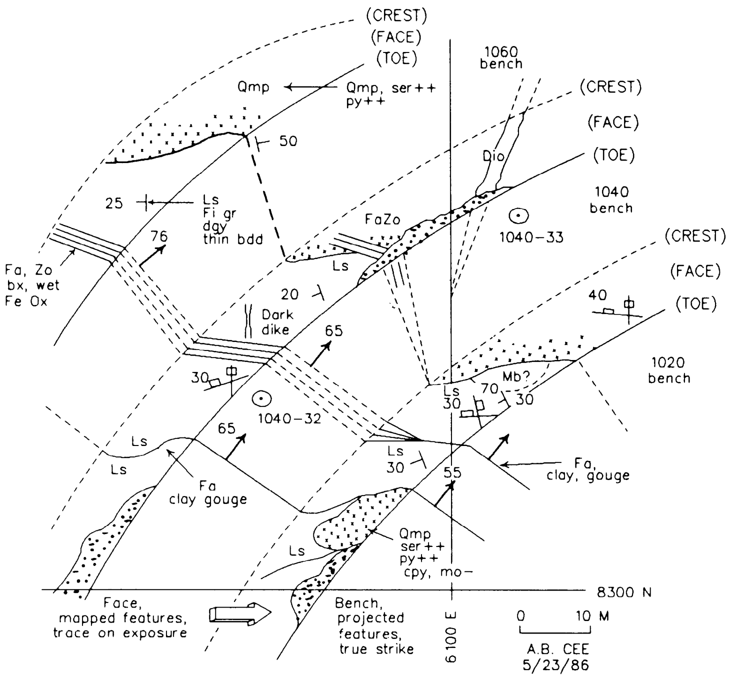
a

Part of an outcrop map. A. As plotted on a overlay sheet showing lithology and geologic contacts. B. Combined data from several overlays, with an interpretation. C. Later interpretation modified by information from a trench.



b

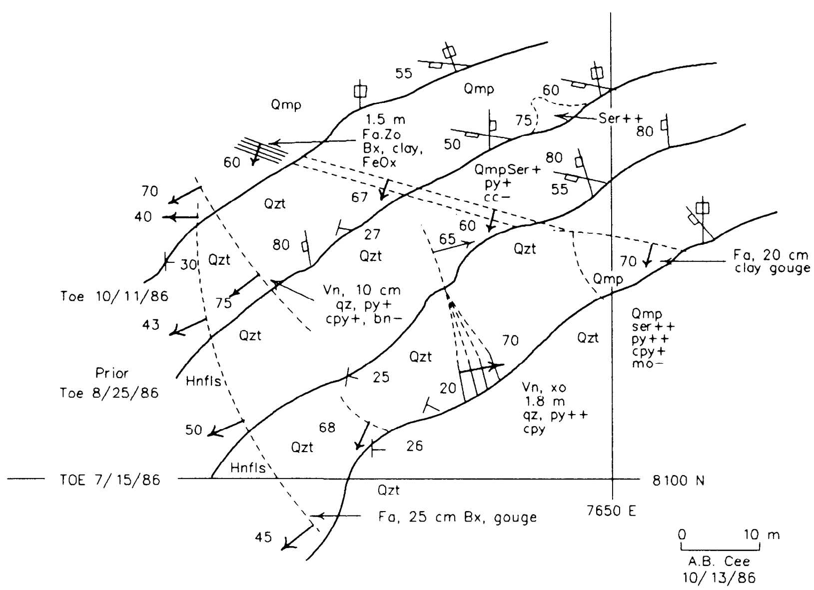
Fig. 5.2.10. Field note sheets. (a) Modified after Peters, 1987, showing the construction of an interpretative geologic map from an outcrop map using successive overlays. (b) Modified after Ahrens 1984, showing detailed mineralization and structural notes from a vein in intrusive rocks.



a

Field note sheet with open pit mine geology recorded by the face method.

Fig. 5.2.11. Field note sheet and posting sheet from an open pit mine. Modified after Peters, 1987.



b

Field note posting sheet, with open pit geology recorded on each level by the toe method.

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Note: A number of excellent papers describing current Australian methodology in mapping, planning, ore reserves, and grade control were recently presented and published by the Australian Institute of Mining and Metallurgy (Anon., 1990) in conjunction with a Mine Geologists' Conference in Mt. Isa, Qld.

Chapter 5.3
SAMPLE COLLECTION

ROBERT A. METZ

Whether for the evaluation of an exploration prospect or control of a mining operation, the process of sample collection can have significant effects on the outcome of those activities. The mathematical chances alone of collection on the order of a millionth (1:l,000,000) of a rock mass to evaluate its valuable mineral content (Bailly, 1968) would seem to dictate that the utmost care be used in that collection. It is paradoxical, then, that this function is often treated so casually in exploration and mining geology. The methods used may be carefully selected, but emphasis is often placed on obtaining data at minimum cost in the shortest possible time so as to expedite production or planning decisions. Procedures that should be rigorously followed often become mundane in practice due to their repetitive and often labor-intensive nature. Sampling personnel are not always highly paid, well-supervised, or thoroughly indoctrinated in the reasons for being meticulous in their work; also the equipment they use may not be in the best condition or have the best design to perform the tasks for which it was intended. Often the demands imposed by time, cost, and difficult working situations lead to the use of procedures and equipment that are more expedient than ideal. It is hoped, and apparently often true, that the number of samples and volume of material collected will be sufficient to compensate for variations and inconsistencies in the material being sampled. The history of mining contains abundant examples of failure due to erroneous or inappropriate sample-gathering methods or the misinterpretation thereof.

In following the trail from prospecting through discovery to a successful mining operation, the geologist will employ a number of diverse sampling methods, each applicable to the particular level of progress and scope of the project. In this day and age, seldom if ever are mines developed from an outcrop of ore alone. Some subsurface sampling must be done, usually by one or more methods of drilling. In event of a discovery, additional drilling, perhaps of a different type, must be done to determine the potential size, shape, and geologic character of the deposit.

At some point prior to mining, bulk sampling of the deposit is done to further determine mining and metallurgical properties of the deposit. These properties include not only the grade but the degree of homogeneity of mineralization and controls localizing the mineralization, rock hardness and strength, ore and gangue mineralogy and their variations, grain size of the ore minerals, and response to ore dressing processes as well.

Once the project progresses to becoming an operating mine, a different type of sampling program must be designed to monitor and control the operation for maximum profitability within the previously determined parameters, generally simplistically referred to as grade control. In actuality, many more factors than grade are usually involved, but most depend on a valid sampling of the material to be mined.

5.3.1 DRILLING METHODS AND EQUIPMENT

Sampling the subsurface usually involves one or more types of drilling, determined by the nature of the material to be sampled and the particular objective of the sampling. A number of different types will be discussed, although present-day mineral exploration and development work is virtually limited to the use of diamond core, rotary, or rotary-percussion methods (Peters, 1978; Waterman and Hazen, 1968; Sainsbury, 1979). For a discussion of drilling theory, see Chapter 9.1.

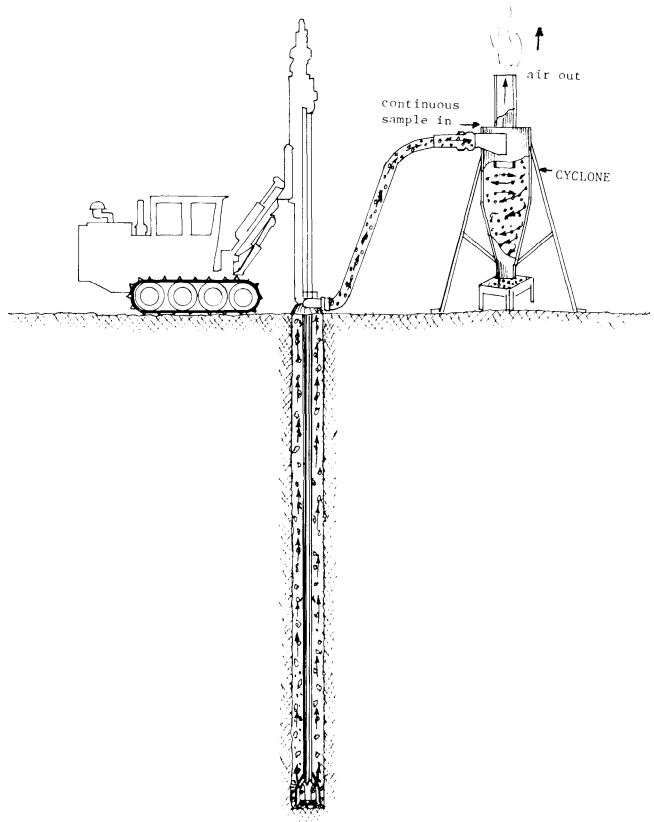

Fig. 5.3.1. Conventional rotary-percussion drill, bit, and cyclone.

5.3.1.1 Rotary-Percussion Drilling

One of the simplest, oldest, and least expensive methods, rotary-percussion drilling is essentially an evolutionary outgrowth of the single-jack driller whose arm has been replaced by a pneumatically or hydraulically driven hammer that transmits its force to a rotating drillbit through a string of hollow-drill steel or directly to the bit in the case of the down-hole hammer. Air, or sometimes water, is circulated through the drill-steel column to the bit to cool its surface and clean the hole of cuttings by forcing them along the outside of the column, hopefully to the collar of the hole (Fig. 5.3.1). Of no little importance is the power behind the circulating fluid. With air, the compressor should have the capacity in both volume and pressure appropriate for the situation. Insufficient volume will allow sorting in the fluidized column of cuttings and may result in heavy particles being left in the hole. Excessive airflow can cause excessive erosion of the hole walls and contamination of the sample. Drilling at high altitudes requires some special considerations and more powerful or multiple-stage compressors. If water is used, special arrangements must be made to allow sample cuttings to settle, minimize the loss of fines, and perhaps clarify and reclaim the water. For sample collection, the collar of the hole is usually fitted with casing and a tee connection to divert the fluidized stream of cuttings into a cyclone collector. At the end of each sampling increment, the bit is slightly retracted from the bottom of the hole and the fluid circulated until all the cuttings are removed.

This method can work well in certain situations where wall rock is competent, dry, and impermeable and drilling does not produce a great amount of fines that could be lost in the exhaust from the cyclone. Conventional rotary-percussion drilling is relatively fast and inexpensive, and equipment, particularly at an operating mine, is readily available and quite mobile, for example, blasthole drills.

However, the method has several disadvantages. Raveling of hole walls caused by vibration of the drill string or abrasion from cuttings can seriously contaminate or dilute samples. A small amount of moisture in the wall rock can be helpful by stabilizing the walls and reducing dust loss; often drillers will add a small amount of water or "mist" to the airflow to the bit to do just that. Large amounts of water flowing into a drillhole can cause serious difficulties, especially where clays are present. Swelling clay minerals can plug drillbits and adhere to hole walls, restricting and stopping airflow and/or rotation of the drill stem. Such problems can sometimes be alleviated by adding substantial amounts of water to air for circulation, but again the washing action of water either injected or from the wall rock is likely to erode the hole walls and produce an invalid sample. Fractured or permeable rock encountered in drilling will cause reduction in or loss of circulation to the surface as the drilling fluid and cuttings are spent into fractures and pores. Attempts at continuing drilling may eventually restore circulation, but any sample so obtained will be suspect; there is the additional danger of cuttings suspended in fractures settling around the drill stem and binding it in place when fluid flow is interrupted such as when adding or removing drill steel. It is generally not practical to install casing and continue with small-diameter drill tools, and the best course when circulation is lost is usually to abandon the hole. The exception would be if the desired object is the hole itself rather than a sample of the material it penetrated.

As drilling depth increases downward, the size of recovered cuttings decreases, resulting in a greater chance of losing values in fines from the sample. Nevertheless, conventional rotary-percussion drilling has a practical depth limit of 200 to 300 ft (60 to 90 m).

As mentioned, sampling by rotary-percussion drilling is relatively inexpensive due, in part, to the ready availability of equipment, the main function of which is usually blasthole drilling. This is particularly true in operating underground mines where compressed air and water supplies are installed. Sample collection is likely to be easier and the sample quality better where relatively short holes are drilled, especially when directed upward from the working and extraction of cuttings is aided by gravity. For "long holes," however, raveling of the hole wall will be exacerbated by the (usually) downward deflection of the drill stem.

By far the more common rotary-percussion drilling method for sampling uses a down-hole hammer (Fig. 5.3.2). Improvements in engineering and design have resulted in pneumatic drill hammers as small as $3^7/_8$ in. (85 mm) in diameter, which are attached to the bottom of a string of drill pipe, are activated by compressed air in the pipe, and transmit impact directly to the drillbit as the drill string is rotated. This method overcomes the disadvantage of the loss of force caused by compression of drill steel in conventional rotary-percussion drilling. Contamination and/or dilution of samples from the hole walls has been alleviated by development of dual-wall drill pipe and reverse circulation in which cuttings are forced up the center or the annulus of the pipe to the cyclone collector (Fig. 5.3.3). Although reduced, there is still the possibility of sample contamination from loose wall-rock caving, falling to the bottom of the hole, and entering the stream of cuttings. Innovations in down-hole hammer design now being tested are aimed at extracting cuttings directly from the bit face and forcing them through ports in the bit to the center of the drill pipe. If problems of bit plugging can be overcome, this concept should greatly improve sample integrity.

Initial reluctance of drilling companies to make the sizable investment in dual-wall pipe has been overcome by demand from clients for better samples, and the equipment is now universally available. At this time, reverse circulation is the most widely used method of sampling mineral deposits, and a great variety of drilling machines have been designed or adapted to work in almost any situation. The method's popularity is due to its speed, up to 825 fpd (250 m/day), and low direct cost, which generally ranges from about $5 to $15/ft ($15 to $45/m). Depth capability until recently had been effectively limited to about 650 ft (200 m), but demand has spurred modification and improvements in equipment to more than triple that capability.

Sample collection with reverse circulation drilling is relatively simple in theory (Fig. 5.3.3). Cuttings transported either through the center or the annulus of the drill pipe are retained in a cyclone collector. When the end of the sample interval is reached, the drill stem is retracted a short distance from the bottom of the hole and the airflow (or water flow) continued until it is judged that cuttings from that interval have been exhausted from the pipe. The flow is then shut off and cuttings dumped from the cyclone through a splitting device to reduce the sample size to a predetermined amount. A separate portion of the rejected sample is collected for geologic logging.

This procedure is not without problems as far as accurate sampling is concerned. The volume and pressure of the air (or other fluid) supply must first be adequate to force all of the cuttings out of the hole. This is especially important if the mineral sought has a high specific gravity. Most rigs designed for down-hole hammer reverse circulation drilling are equipped with suitable compressors, but conversion of rigs built for other types of drilling may not be. Excessive airflow causes excessive loss of fines from the exhaust stack on the cyclone; this also can be reduced by extending the length of the stack, which may produce dust leaks elsewhere in the system. By experimentation, an optimum arrangement can usually be found. Another source of sampling error can occur when the sample is dumped from the cyclone onto the splitting device. If this is not properly done, the material will billow up over the splitter and be lost from the splitting process. In some collection systems, the cyclone does not have a gate on the outlet, and the material collected flows continuously through the cyclone and splitter during drilling. Although this continuous flow may avoid the problem of spillage, it usually results in extremely dusty and often hazardous conditions around the machinery. Samples from the cyclone are commonly reduced in size by being passed through Jones or Gilson splitters (Figs. 5.3.4 and 5.3.5). The latter is an adjustable modification of the former, and both types have such a long history of use in the sampling process that their appropriateness is seldom questioned. However, some thought should be given

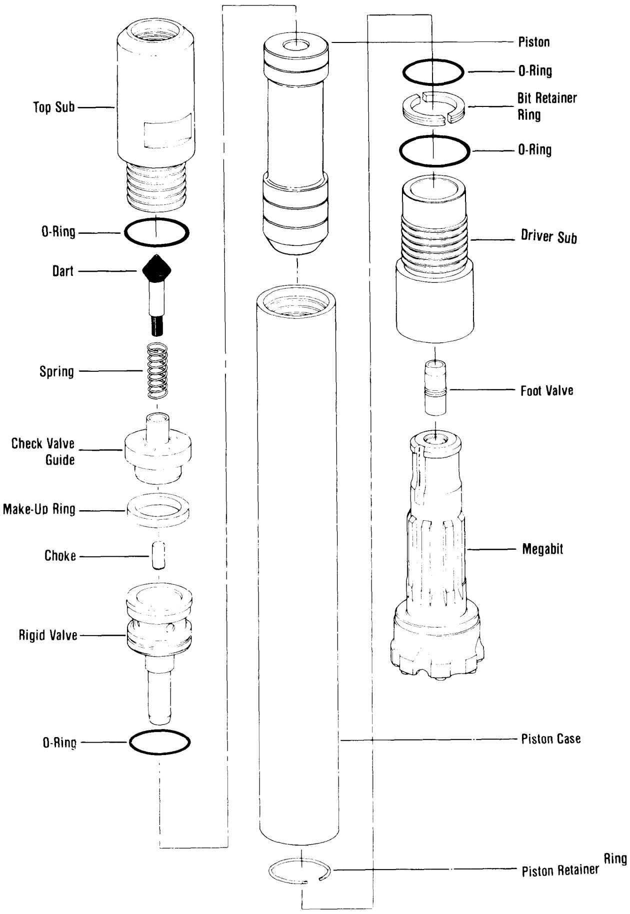

Fig. 5.3.2. Down-hole hammer (by permission: Ingersoll-Rand, Inc., left and Sandvik Mission Drilling Products Co., center and right).

to their correct use (e.g., level positioning, even distribution of the material being split, size of the slot openings relative to particle size of the material). With this simple and seemingly effective equipment, there are numerous ways to introduce bias into the final sample with only a little carelessness. Recently developed splitters such as the Accu sampler and SDI models (Figs. 5.3.6 and 5.3.7) are designed to avoid these biases and also to handle wet samples. Another type of splitter designed specifically for wet samples is shown in Fig. 5.3.8. Collection of wet samples dictates special care in washing equipment thoroughly between sample intervals.

In collecting wet samples, one is often overwhelmed by the volume and weight of the material and the problems of filtering and drying it for transport and handling. Spillage and loss of fines are valid concerns. Often the mineral sought, when in a finely divided, suspended state, will float on the water and may be lost from the sample. This is particularly true of gold and other precious metals that, due to their malleability, are easily suspended by surface tension. The problem is being addressed by a number of drilling companies and equipment manufacturers who have designed and produced a variety of stationary and rotating sampling devices intended to extract a representative proportion of the drill cuttings. However, the problems of spillage and properly cleaning the device, added to those of filtering out the water without losing values in the fines, in the author's opinion, make accurate and reliable sampling virtually impossible. These are problems that must be solved. Environmental concerns about air pollution and conspicuous dust plumes from pneumatic drilling have prompted regulations in some areas that essentially prohibit drilling with air conventionally. It is anticipated that this regulatory trend will continue and that drilling with water and collecting wet samples may eventually become mandatory procedures.

5.3.1.2 Rotary Drilling

Because of inherent elasticity or other unique physical properties, certain rock types cannot be drilled and sampled effectively with percussion equipment but are amenable to some type of rotary drilling, whether conventional or reverse circulation (Fig. 5.3.9). Most commonly, a tricone bit is used, but special situations may find the use of plug, fishtail, or drag bits more effective. Extracting and collecting the sample are essentially the same as with rotary-percussion drilling.

5.3.1.3 Diamond Core Drilling

Diamond core drilling generally provides the most useful and accurate samples of a mineral deposit and is the most versatile of all drilling methods (Peters, 1978). Whereas at one time, limitations of equipment produced small, fragmented samples, the refinements in wireline techniques and drilling fluids and additives since the 1950s have made it possible to core almost any kind of rock.

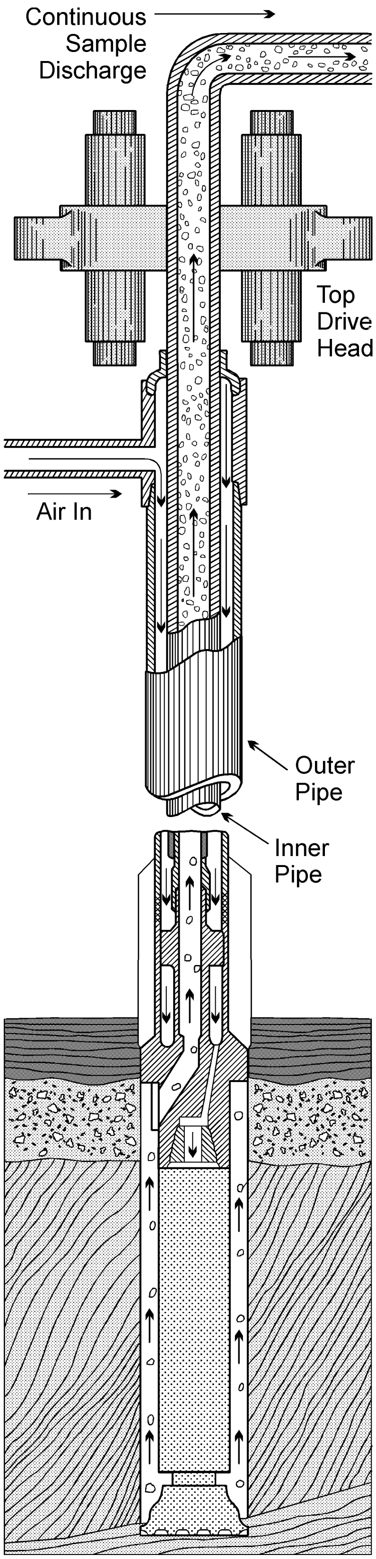

Fig. 5.3.3. Reverse circulation downhole hammer, pipe, sample recovery system. (By permission of Drilling Services Co.)

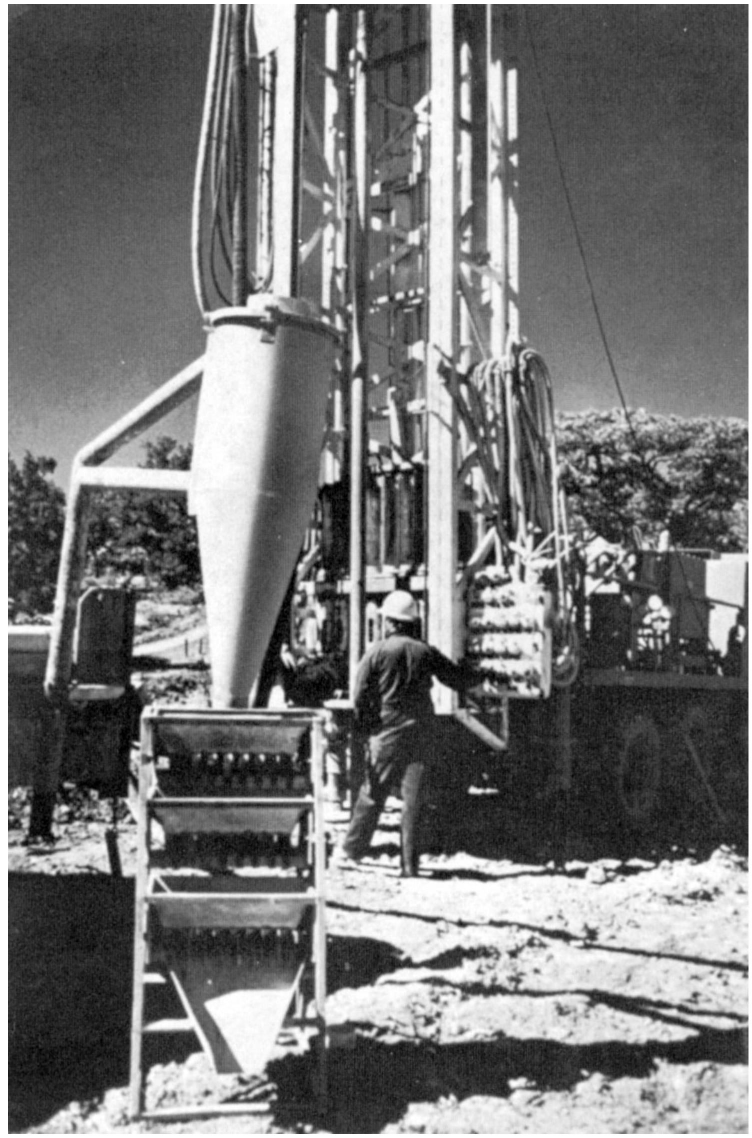

Fig. 5.3.4. Jones riffle splitter. (By permission of Drilling Services Co.)

The two methods of diamond core drilling are referred to as conventional and wireline. In drilling with conventional equipment, the core barrel is a fixed part of the drill rod string (Fig. 5.3.10) and can only be retrieved by retracting and removing all drill rods from the hole one at a time. This method has undoubtedly resulted in a great deal of core being destroyed over the years because of the reluctance of drill crews to pull the rods before finishing a run, especially in deep holes.

The wireline method uses a core barrel removable through the inside of the drill stem with a latching device on the end of a cable (Fig. 5.3.10). With this method, core can be retrieved at any desired point, usually in less time than with the conventional method, and drill rods need only be removed for bit changes, casing installation, etc. In general, wireline is the preferred method, but there are exceptions. Core diameters with wireline equipment are smaller than their conventional counterparts. In relatively shallow holes or in hard rock where bit changes are

Fig. 5.3.5. Gilson adjustable sample splitter. (By permission of Gilson Screen Co.)

frequent, there may be no time-saving advantage to using wire-line equipment.

Early conventional equipment using water as a drilling fluid commonly yielded core recoveries of less than 50% and led to numerous intricate methods of collecting sludge (material ground up by the drillbit and flushed from the hole outside the rods) samples and formulas for combining core and sludge assays to get the "correct" value. Even then, the results were often suspect and assays discounted as an extra safety factor. With the present variety of diamond bit designs and mud and polymer technology, 100% core recovery has become the rule rather than the exception, and sludge samples (fortunately) are rarely collected. The collection and use of sludge samples have the same inherent problems as those with cuttings from conventional rotary-percussion drilling. Sludge is usually believed to consist of the softer (and probably more mineralized) part of the material being drilled, but with the possibilities of losses into fractures or additions from caving walls, one can never be sure of what a sludge sample represents. Wireline tool design is believed to add stability to the coring process and also to increase core recovery.

Core sampling can also be problematic. Traditionally, core is manually split lengthwise with the aid of a mechanical splitter (Fig. 5.3.1), and one half is used for assays and the other half for "backup." However, a true split is rarely if ever achieved. This is particularly troublesome when the mineral sought occurs in friable fractured material at varying angles to the core axis. Another method, intended to obtain a more equitable split, is to cut the core lengthwise with a diamond saw; this works particularly well with hard, competent rock and more or less evenly disseminated mineralization. Other workers believe that the best course is to assay the whole core, thus avoiding bias from the splitting process.

The advantages of diamond core drilling over other methods include better geologic and geotechnical information (structure, controls, nature of mineralization, strength) and better material for metallurgical testing. Its main disadvantages are that it is slower and more costly than other methods and produces a relatively smaller sample. However, in recent years due to the amortization of drilling equipment and competition from

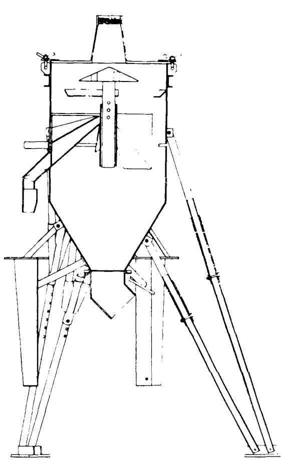

Fig. 5.3.6. Accu-sampler. (By permission of Exploratech Marketing International Inc.)

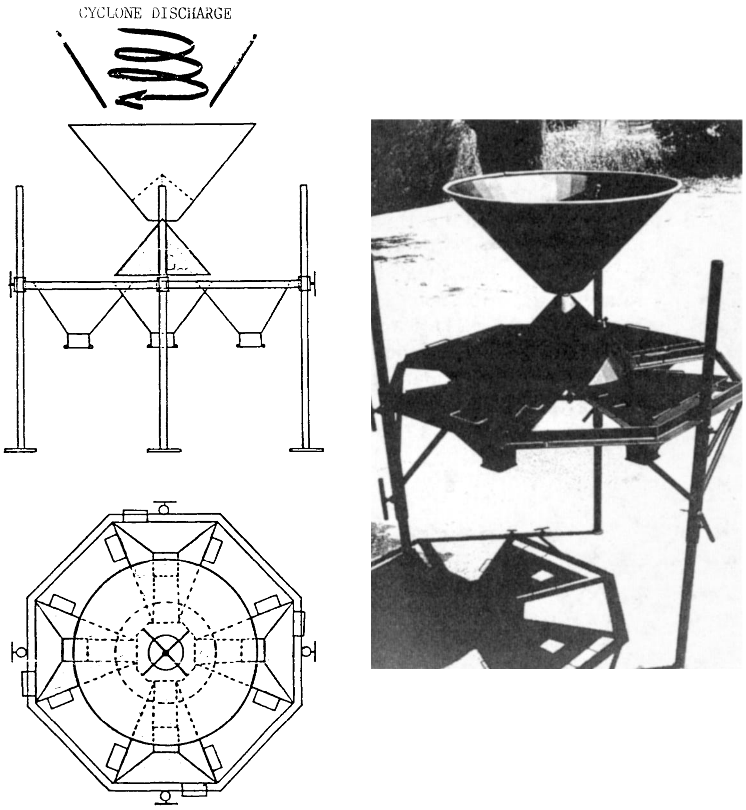

Fig. 5.3.7. SDI sampler. (By permission of Sampling Devices, Inc.)

reverse-circulation drilling, diamond drilling costs have dropped to about a third of what they were in the mid-1980s. Prices vary with circumstances, but rates of \$10 to \$15/ft (\$35 to \$50/m) are not uncommon.

5.3.1.4 Other Drill Sampling Methods and Equipment

The foregoing methods and equipment are virtually the only ones used in sampling hard-rock mineral deposits. A number of specialized types of drilling methods have been developed for sampling soft, unconsolidated materials such as alluvial or placer deposits. Many of these are based on the churn (or cable tool) drill method such as the Airplane drill, Banka or Empire drill, Keystone drill, and Becker hammer drill (Wells, 1973) (Figs. 5.3.12 and 5.3.13). The principles in all are the same: hollow casing is driven into the material to be sampled, leaving an inside core of relatively undisturbed material; the core is then attacked with a cable tool drillbit, broken up, and flushed, pumped, or otherwise lifted to the collar of the hole for sampling. The Keystone drill is a churn drill specifically designed for placer sampling; the Airplane drill is essentially a smaller version for use in areas of difficult access (Wells, 1973). The Banka or Empire drill is a prospecting drill, which uses the weight of the crew on a platform to drive the casing, rotation being accomplished by man or animal power; the drillbit and sand pumps are usually operated from the drilling platform. The Becker hammer drill, developed in the 1950s, combines the foregoing techniques with those of reverse circulation; dual-wall drill pipe is driven by a pile driver while air or water is forced down through the annulus to lift the cuttings through the inner pipe. Measurements of the samples recovered and calculations of volume and value are quite specific and critical; examples are given by Daily (1983) and Wells (1973).

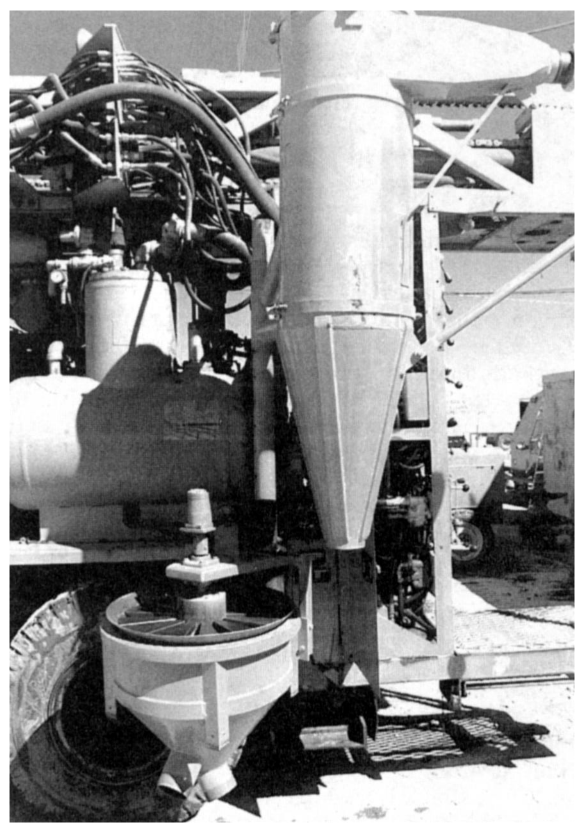

Fig. 5.3.8. Wet drill sample splitter. (By permission of Boyles Brothers Drilling Co.)

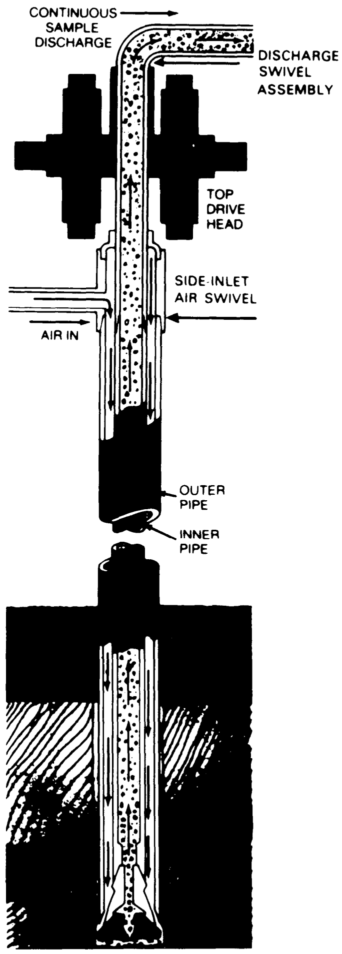

Fig. 5.3.9. Reverse circulation drilling system with tricone bit. (By permission of Systems for Drilling Inc.)

In special circumstances, a bucket drill can be used successfully, that is, in firm dry ground without large rocks. The "bucket" is a straight-walled cylinder many times larger in diameter than the drill stem, with radial excavating blades extending from the bottom where rubber flaps over wedge-shaped openings admit cuttings during drilling but seal the openings when the bucket is hoisted (Wells, 1973, pp. 39-41). Variations on this technique are rotary "clamshell" type excavators such as the "orange peel," which has several convergent tapered segments that can be closed to remove the sample. Sampling with these excavators has the inherent inability to remove all the loose material from the hole.

Large boulders can cause problems for all these methods by either blocking the drillbit, deflecting it, or being pushed aside. Ideally, the boulders would be cut by the bit and included in the sample, but this is probably seldom what happens. If numerous large boulders are known to occur in the deposit, this effect can be reduced by taking larger samples such as with a caisson drill (Fig. 5.3.14) (Hildebrand, 1976). A large-diameter caisson is driven by a combination of the rotational impact of a horizontally swinging arm onto a striker block and the weight of the equipment. The sample is removed at intervals from the inside of the caisson with a mechanical excavator and perhaps, ultimately, manually.

A recent innovation in drilling equipment, developed for water wells, is the Barber Dual Rotary (DR) system manufactured by Barber Industries. This system features two rotary drill heads: a lower one to advance carbide-tipped steel casing through unconsolidated material, and an upper one to advance a drill string connected to a down-hole hammer or tricone bit used to break up the core within the casing. The fragmented core material is flushed from the hole to a collecting cyclone by the air or other drilling fluid (Fig. 5.3.15). Reportedly, this equipment is successful in drilling through boulders as well as sands and gravels in unconsolidated alluvium and in materials with strong artesian flow. These features appear to be well suited to reliably sampling deposits such as placers with minimal loss or contamination of samples. However, any values contained in large fragments and reported in the samples from such deposits would not be recovered in normal placer operations.

Earth augers have been used in soft, relatively fine material (for example, placers) but present problems in sample removal and collection.

5.3.2 DRILLING PARAMETERS

In establishing a drill sampling program there are many factors to be considered, most of them having to do with the type of deposit involved. Different methods should apply to a high-grade precious-metal vein, a low-grade disseminated metal deposit, a specialty clay, coal seam, phosphate, or placer. A fair working hypothesis must be developed about the nature of the deposit sought, its geometry, competency, mineralogy, etc. The stage of the project is important as well, because a method used in the early exploration phases will probably not be suitable for development or production phases. If, for example, the stratigraphy is well known and the various rock types are readily discernible, reverse circulation drilling can provide adequate samples and geologic information. If the foregoing premises do not apply, diamond core drilling should be employed to provide the necessary information, at least until the geologic framework of the deposit is stabilized with some degree of confidence.

Other factors such as equipment availability and cost may dictate the use of a drill sampling method that is less than ideal for the project. If, for example, funds are critical and a standard percussion blasthole drill is readily available, one might drill a few holes "just to see if something's there." If that test meets with apparent success, it should be followed with a more accurate method. Checking such results is often done by "twinning" drillholes, but it must be realized that the sampled material from the "twin" holes will not be identical, and close agreement cannot always be expected. Caution should be used in mixing or combining results from different types of drill samples into a resource estimate. Too often an assay printed on a log or cross section becomes a "fact" on a par with all others, with no further consideration of its provenance. In the author's experience and in examples such as noted by Parratt and Bloomstein (1989), unintentional salting of samples occurred in rotary-percussion holes, and the apparent high-grade zones were disproved by subsequent diamond drilling.

The optimum drilling method for a deposit may change as the project makes the transitions from exploration and discovery through development to production. Early exploration drilling is usually done at locations chosen for specific geologic reasons. Following a discovery, a specific drillhole pattern or grid is followed to delineate and define the deposit with sufficient accuracy to determine its economic viability. Spacing of drillholes in the grid can be critical, and the optimum spacing can often be determined by the use of geology (Shaddrick, 1987) and geosta-

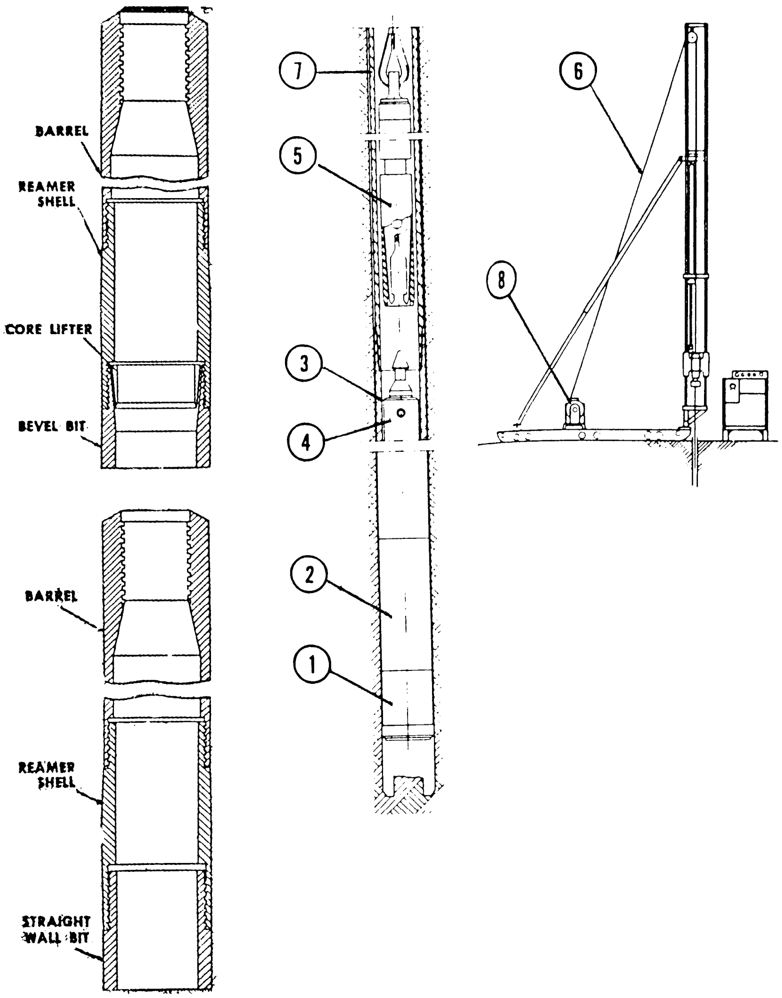

Fig. 5.3.10. Conventional and wireline core barrel systems. Left: Single-tube core barrel with bevel wall bit and core spring or "core lifter" (top) and straight wall bit (bottom). Right: Wireline drilling equipment that consists of (1) diamond drillbit, (2) diamond set reaming shell, (3) outer core barrel complete, (4) inner tube assembly, (5) overshot assembly, (6) wireline cable, (7) drill rod string, and (8) wireline hoist. (Reproduced by permission from *Diamond Drill Handbook,* 1980, pp. 55, 63.)

tistics (Call, 1979; Peters, 1978) as well as empirically. There have, of course, been deposits that in retrospect were found to have been drilled on a closer than necessary spacing, causing an undue front-end capital cost to the project. More noteworthy are the opposite situations where too large a grid spacing resulted in holes penetrating isolated pods of mineralization with insuffcient continuity between holes. Porphyry copper deposits may be adequately sampled by drillholes on a 250-ft (75-m) grid spacing, but a limestone replacement or skarn deposit may require a 50-ft (15-m) or less drillhole spacing for adequate evaluation.

Differences in mineralization grade in various rock types in a deposit should be taken into account in drillhole sampling if possible. Although generally practicable only with diamond core drilling, sample (assay) intervals should conform to geologic contacts to properly define the deposit. Even though the ore body may ultimately be mined on prescribed bench levels, assays can be averaged to represent those levels and the geologic controls can and should be used to determine those levels.

A deposit characterized by steeply dipping mineralized fractures can seldom be evaluated properly with vertical drillholes. In such deposits, preferred orientations of mineralized fractures should be determined as closely as possible and the drillholes oriented at a reasonable angle to crosscut the fractures.

Drillhole size is also an important consideration in sampling. If mineralization is expected to be relatively homogeneous and

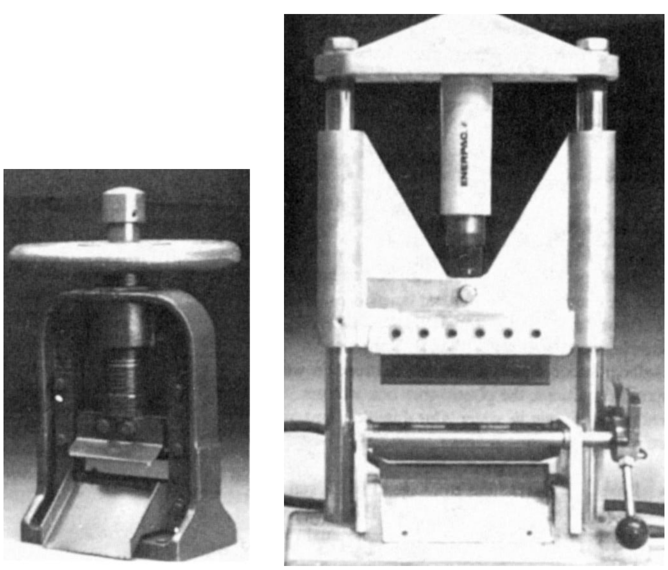

Fig. 5.3.11. Core splitters (manual, left; hydraulic, right). (By permission of Myhre's Exploration Products, Inc. *Catalog.)*

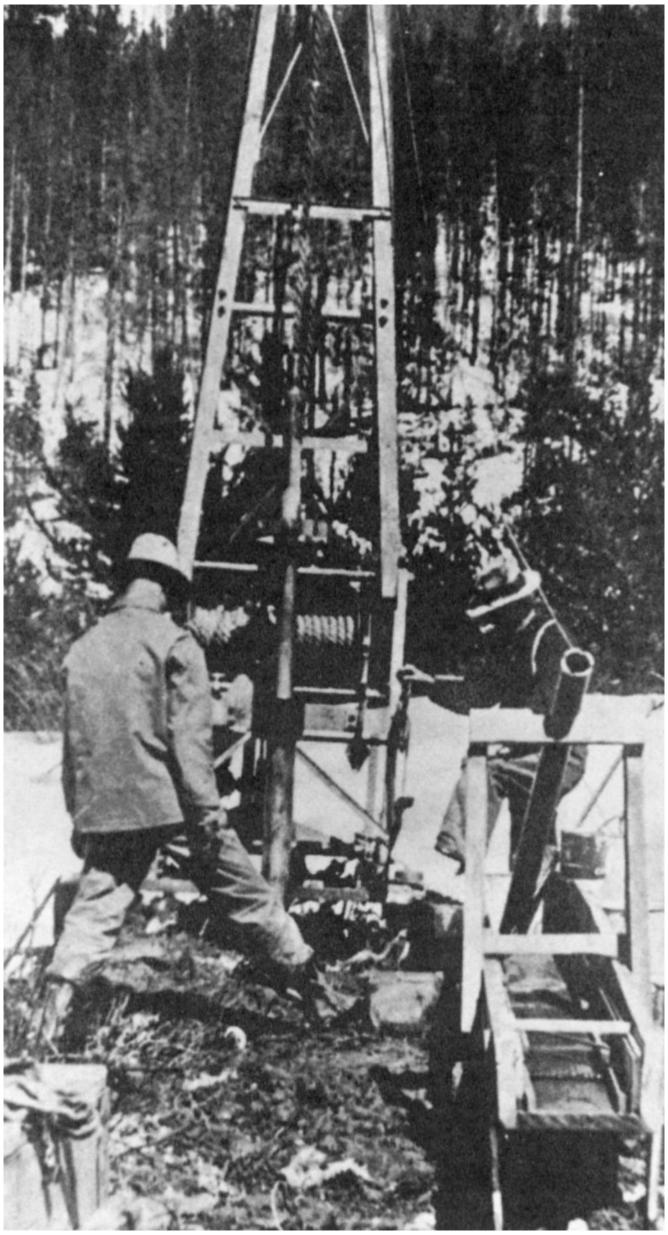

Fig. 5.3.12. Airplane placer drill (Wells, 1969, p. 51).

finely disseminated on closely spaced fractures within the host rock, adequate sampling may be done with BX size core (1.432-in. or 136.4-mm diameter). However, NX core can yield 1.7 times as much sample, a 5.5-in. (140-mm) diameter rotary-percussion hole 11 times as much, and a 12-in. (305-mm) diameter blasthole 50 times as much. The importance of adequate sample size for gold analysis was discussed in detail by Clifton et al. (1969).

With angle drilling for vein deposits as well as with relatively deep vertical drilling, it is important to know the location of the drillhole at points from which samples were cut. Such drillholes often deviate from their collar orientation as the result of wall-rock structure and composition (e.g., bedding, cleavage, foliation, alternating hard and soft layers, etc.) or excessive down pressure on the drill stem during drilling (McKinstry, 1948, p.

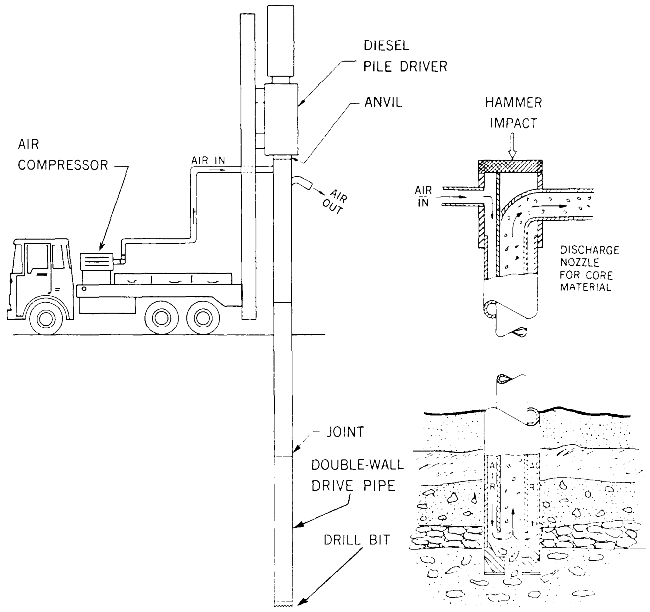

Fig. 5.3.13. Becker hammer drill. (By permission of Becker Manufacturing and Drilling Inc.)

96; Peters, 1978, p. 447). Over the decades, numerous methods of down-hole surveying have been used in diamond drilling to measure such deviations and determine the true location of the drillhole in space. Some methods such as that using simply a glass tube and hydrofluoric acid provide only the inclination of the hole at the point surveyed; others such as the Maas compass and "Tro-Pari" (Cumming and Wicklund, 1975) provide both azimuth and inclination data on the attitude of the hole but are affected by the magnetism of the wall rock. The Sperry-Sun Well Surveying Co. offers services and equipment that include multi-shot gyroscopic as well as magnetic compass devices that can provide an almost continuous survey of a drillhole, if desired. Although originally developed for oil-field drilling, presently available equipment is small enough for use in BX-size core holes (Table 5.3.1). Outgrowths of accurate drill surveying are the taking of oriented core samples (permitting measurement of true attitudes of geologic contacts, faults, veins, bedding, etc.) and directional and multiple-hole drilling. With the latter two, a targeted vein or formation can be sampled at depth at one or more specific locations, by the combined use of hole surveys and directional wedges or mud-powered bits such as the Dyna-drill or Navi drill (Fig. 5.3.16). Both Smith and Navi drill systems include a deviation device acting from the nonrotating part of the drill stem. The bit, turned by a motor powered by the force of the circulating fluid, continues to drill until the desired deflection has been achieved, while the drill string slides down the hole behind it. With the Smith Steerable System, azimuth and inclination are monitored on the surface by means of a "mud-pulse" telemetry system in the down-hole drilling unit. Although this system was developed mainly for the larger-diameter drilling in the petroleum industry, continuing refinements and miniatureization of the equipment suggest it may eventually be used more widely in the mining industry.

5.3.3 BULK SAMPLING

Once a reserve of economic-grade material has been proved, its continuity and availability to mining and metallurgical pro-

Table 5.3.1. Commonly Used Wireline Core Drilling Sizes

Boyle Bros.[a]			CBC[b]			Longyear[c]		
Designation	Core diam., in.	Hole diam., in.	Designation	Core diam., in.	Hole diam., in.	Designation	Core diam., in.	Hole diam., in.
AX	1.067	1.890	AXE	1.015	1.852	AQ	1.062	1.890
BX	1.432	2.360 2.400 2.440	BXE	1.437	2.385	BQ	1.432	2.360
NX	1.875	2.980 3.032 3.125 3.250	NXE	2.000	3.005	NQ	1.875	2.980
HX	2.400	3.650 3.750 3.937	NCE	2.406	3.685	NQ	2.500	3.782
CP	3.345	4.827	3-in.	3.000	3.915	PQ	3.345	4.827

Source: [a]Personal communication, C. Hirschi, Boyles Bros., Reno, NV.
[b]Personal communication, N. Trujillo, CBC Drilling, Tucson, AZ.
[c]Personal communication, A. O. Krause, Longyear Co., Peoria, AZ.
Conversion factor: 1 in. = 25.4 mm

cesses should be verified by bulk sampling. By the time the project reaches this stage, a great deal should be known about its physical as well as its chemical characteristics from physical examination of samples and bench-scale metallurgical tests run on them. These presumed properties need to be confirmed by examination and testing on a larger scale. The earlier stage gives information on what and how many metallurgical ore types occur in the deposit (oxide, sulfide, mixed, polymetallic), their spatial distribution, and some clues as to their structural strength. A carefully planned program of collecting several tons to several thousand tons from different portions of the deposit judged to be representative of the various significant ore types is essential to designing a successful operation. These samples may be obtained from surface excavations, underground excavations, or drillholes as appropriate to the size, location, and geometry of the deposit.

Near-surface deposits such as a placer or certain types of nonmetallic deposits may be effectively tested from surface trenching or pitting. Such excavations should be accurately measured for assay checks and computation of tonnage factors. They should also be mapped in detail to document such factors as fracture spacing, intensity, and trends; facies changes; mineralization variation and trends; and size and distribution of fragments.

Underground bulk samples are usually collected by driving raises around a drillhole from the surface or mining a test stope in a "proved" ore area. That area must be carefully selected to be as representative as possible of the whole deposit, a very difficult condition to meet. Consequently, the sampling of more than one such area is desirable if not necessary. Assay comparisons between the bulk sample and the drillhole(s) it is intended to check are likely to show some discrepancy. A method of statistical comparison is given by Waterman and Hazen (1968).

Without the convenience of existing underground workings, the cost of underground bulk sampling may seem prohibitive and must be weighed against the cost of alternative methods or that of inadequate mine and process design. That design will depend greatly on the selection of the bulk sampling site(s).

A more comprehensive system of bulk sampling uses drill cores from various parts of the deposit. For this purpose, large-diameter core (6 in. or 152 mm, or larger) is most useful for

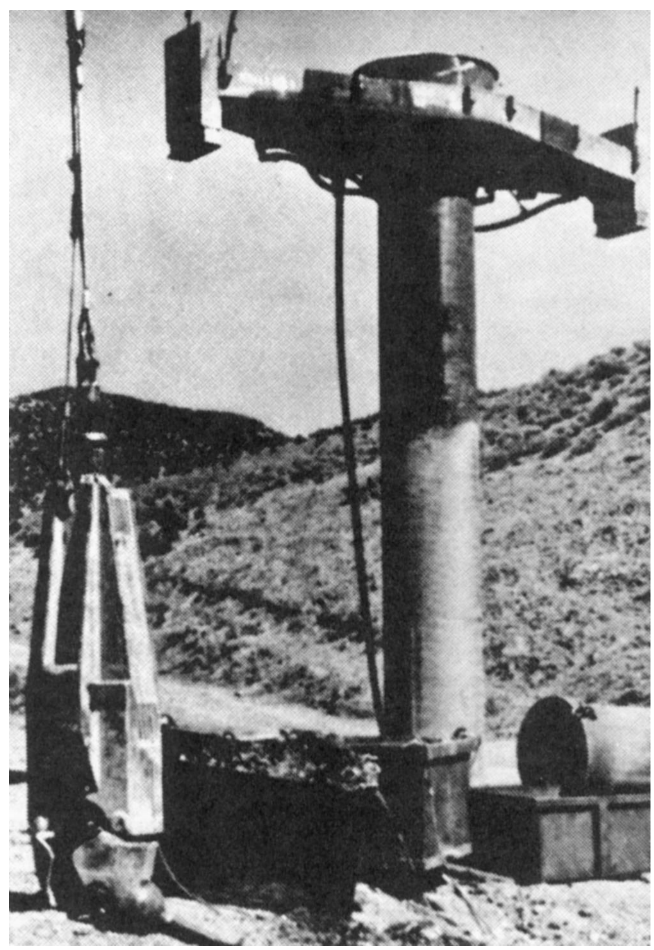

Fig. 5.3.14. Caisson drill. (Reprinted from *Mining Year Book 1976,* Colorado Mining Association, p. 131.)

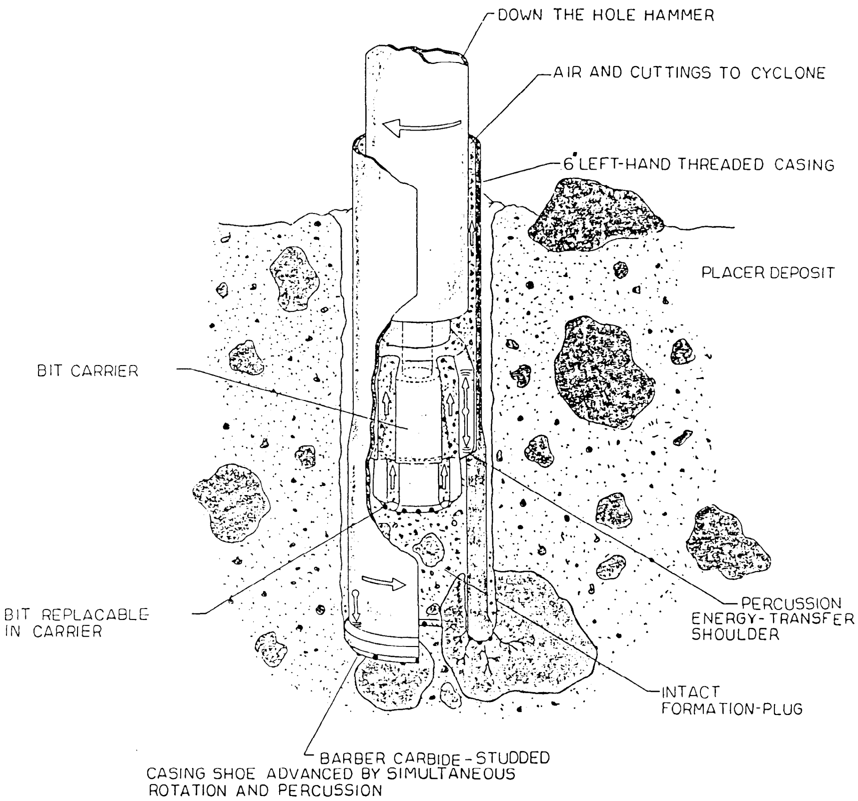

Fig. 5.3.15. Schematic diagram showing the Barber dual rotary drilling methods. (By permission of Barber Industries.)

obtaining metallurgical and geotechnical information. This system permits sampling all facies or ore types of a deposit rather than only what is hoped to be "typical." With a vein deposit, drilling wall rock can be started with noncoring equipment and converted to coring equipment as the vein intersection is approached. Ample amounts of wall rock should be cored to provide geotechnical data and address the problem of dilution as well. Bulk sampling with drill core does not provide first-hand inspection of material in place but does provide a greater amount and variety of information than other methods.

In the subject of bulk sampling, emphasis is often placed on obtaining samples of "typical" or average material. Sampling should also include some of the "worst case" or refractory portions of the ore body, especially if they are likely to be treated at an early stage of mining when achievement of production and profitability goals are especially important.

Placer deposits present a special challenge in the area of bulk sampling due to the often erratic distribution of values, great variations in particle size of material to be handled, and usual lack of consolidation. Dry land placers can be sampled with carefully excavated surface pits and trenches, hand-dug shafts, or by special techniques such as the caisson drilling method previously mentioned (Hildebrand, 1976). The subject is worthy of an entire chapter by itself; Hildebrand also discussed some of the problems to be dealt with, and Wells (1973) covered various methods and ways to address special problems and deposit types. It is especially important in bulk sampling of placers to retain any sample in its entirety without splitting for concentration by the same methods contemplated for production to provide a reliable estimate of value and recovery. Chapter 15.1 also discusses placer exploration.

Sampling industrial mineral deposits is a very specialized field, and techniques may vary from the simple use of a physical property such as specific gravity for a barite deposit to processing the sample through a series of microscopic and X-ray analytical tests. Often the purity of the sample (a bias in itself) must be estimated prior to the collection. Or, as in sampling a phosphate deposit, most of the analysis is done at the sample collection site. A comprehensive discussion of techniques, problems, and their considerations is given by Eyde and Eyde (1985).

Other specialized applications are discussed by O'Donnell and Jodrey (1985) and Greenberg et al. (1985).

5.3.4 GRADE CONTROL

During the transition from development to production, a method of grade control must be established, including correct and necessary sampling procedures. The procedures will vary with the type of deposit and, of course, be different for open pit and underground.

Presently, virtually all grade control in open pit mines is based on samples collected from blastholes on pit benches. Although a variety of techniques may be used, the best samples are obtained by using a wedge-shaped ("piece of pie") collecting pan whose vertical sides are radial to the hole and long enough to recover all particles ejected from the hole in that sector. The collector is placed against the drill collar before drilling starts and removed when the drillbit reaches the next lower bench level (blastholes are usually drilled a prescribed distance past that level to allow for caving and to assure optimum breakage). Results of such sampling give a good indication of the grade distribution of the material in situ, but that can be changed significantly by blasting, particularly if there are notable physical differences, for example, competency or fracture density, between ore and waste. What once may have been clearly separate ore and waste can

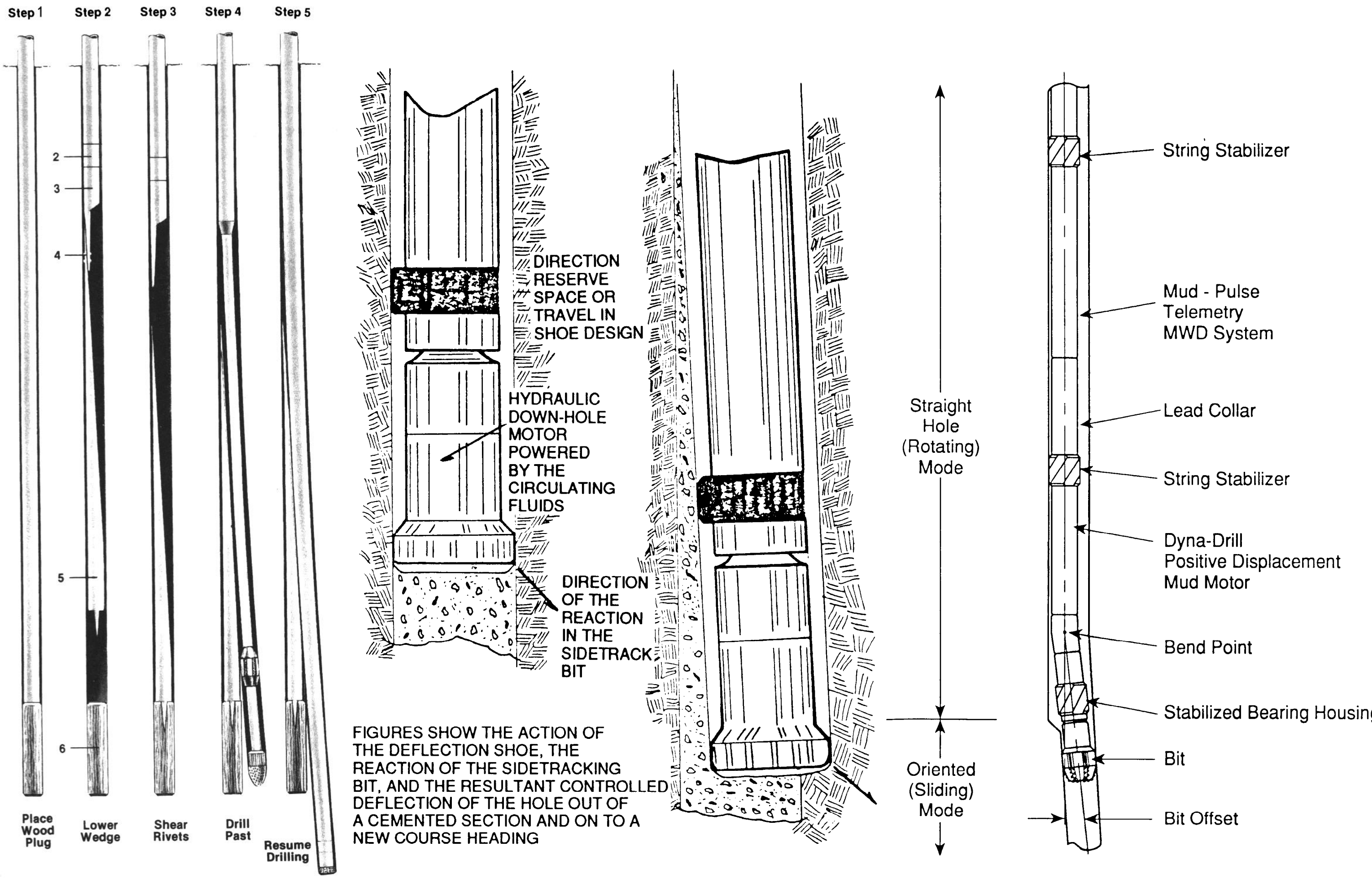

Fig. 5.3.16. Directional deflecting wedges (left), Navi-drill (center), and Smith Steerable System (right). (By permission of left, Boyles Bros. Drilling Co.; center, Christensen Boyles Co.; and right, Smith International.)

become so mixed during blasting that all is waste. If ore controls are well known, visual inspection of shot muck may suffice to redefine an ore-waste boundary, or the ore control engineer may decide to use grab samples to make that definition, a risky method. More refined selective mining techniques may be called for, and these should rely heavily on geologic input. Information gained from detailed mine level mapping can determine structural and lithologic controls and mineralization and thus ore grade and metallurgical ore types. Kornze et al (1985) and Clark et al. (1985) detail methods used in low-grade precious-metal and disseminated copper deposits, respectively.

Sampling for grade control in underground mines presents some more difficult problems, as often the material cannot be sampled until it is broken and essentially loaded and perhaps on its way to the mill or smelter. Grade control may be based on composite results from channel samples of a stope, drift face or back, muck piles, ore chutes, and car samples. Particular care must be taken with the last three because of the mineral particle segregation in fines vs. coarse material. Rogers (1985) and Ruge and Vera (1985) detail some of the intricate problems and variations encountered and methods used to resolve them at two quite different underground mines.

Grade control is usually thought of in terms of measuring the contained amounts of the metal or compound sought. However, in industrial mineral deposits, it often depends on detection and quantitative measurement of contaminants. Likewise, quality control in coal deposits depends on sulfur and nitrogen oxide contents of the products of combustion. In solution mining and tangentially Frasch sulfur production, grade control is accomplished by a complex system of computer monitoring, analysis, and regulation of solutions.

5.3.5 OTHER SAMPLING METHODS

The preceding discussion has covered the more common sampling methods currently in use, most of which are aimed at direct measurement of the sought-after commodity. For certain minerals, indirect methods are used. One of the better known methods developed from the uranium rush of the 1950s as techniques were devised to "assay" radioactive minerals in rock with Geiger counters, scintillometers, and subsequently gamma-ray spectrometers. Rotary-percussion drilling without concern for sample recovery followed by hole probing to detect zones of radioactivity was the method used and perfected because of the unique properties of uranium. Down-hole probing was refined to down-hole logging, providing precise electrical as well as radiometric measurements, particularly useful in prospecting evaporite deposits. Another down-hole sampling technique that has seen some limited use employs neutron activation analysis to assay material in situ. Evaluation of diamond deposits involves mechanically breaking down bulk samples and heavy media separation (Atkinson et al., 1985) in lieu of the earlier labor-intensive, less-quantitative method of panning alluvial or colluvial material. Mention has already been made of the drill-site sample processing and analysis in evaluating phosphate deposits; no doubt other similar techniques exist or will be developed.

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Chapter 5.4
SAMPLE PREPARATION AND ASSAYING

GORDON E. GUMBLE, EDWIN V. POST, AND WALTER E. HILL, JR.

The sample preparation and assaying procedures discussed in this chapter apply equally to in-house facilities as well as commercial laboratories. They are applicable to samples derived from reconnaissance exploration through development drilling to mine operations.

The focus is on operations in metal mines; indeed, much worldwide mining activity at this writing is concerned with exploration for and development of gold deposits. The requirements for the preparation of samples of many gold-bearing materials are more stringent than those for many other metallic deposits. Thus the practices described here generally fulfill the requirements for most metallic mineral deposits.

The preparation and testing of coal samples is a specialized field. Most of the pertinent procedures have been published by the American Society for Testing and Materials (ASTM), and they are summarized in a later segment (5.4.3) of this chapter.

As is true for coal, the preparation and testing of samples of industrial mineral raw materials is highly specialized. For many products, the preparation and testing procedures are end-product sensitive; that is, the preparation and testing procedures to be used are controlled by the properties desired in the final product. One must attempt to select sample preparation and testing procedures that will produce a product comparable to one produced by the actual process procedures to be used. The book, *Industrial Minerals and Rocks* (Lefond, 1983), presents perhaps the best overall background on a wide variety of industrial mineral products, although little specifically on sample preparation and testing. Few additional general publications exist, and no specific guidelines can be given here, other than to suggest that sample preparation and testing procedures be established for individual deposits through frequent and close communication among the exploration and mining staff, laboratory personnel, mill managers, marketing staff, and end users.

5.4.1 SAMPLE PREPARATION

Sample preparation is the process of converting samples of geologic materials from the large sample collected in the field or mine into finely divided homogeneous powders suitable for chemical analysis or other testing. This is accomplished by the mechanical reduction of pieces of rock to a smaller particle size in a stepwise sequence, alternating with the reduction of sample volume or mass by an unbiased splitting process.

Error can be introduced in many ways during sample preparation. As a consequence, attention to detail and thorough cleaning of equipment between samples is necessary. The desired end result of sample preparation is a rock powder that contains the elements to be analyzed in the same concentrations and proportions as in the original sample received. The reduction in particle size will be affected by many factors, including particle shape, hardness, specific gravity, lubricity, malleability, residual moisture, and the quantity of clay minerals or organic matter present.

5.4.1.1 Sample Preparation Equipment

The equipment required to prepare mineral samples adequately for analysis depends to some extent on the nature and quantity of the samples, and even on the climatic environment. In hot desert environments, samples can be air dried adequately under the sun, and a laboratory with only a handfull of small samples to crush and pulverize per day could do an adequate job with a manual bucking board and muller.

Dryers: Electric- or gas-fired ovens are used to remove moisture or free-standing water from samples before crushing and pulverizing. An airflow is maintained through the oven to remove water vapor released from the samples. For routine assay purposes, oven temperatures are usually maintained from 220° to 285°F (104° to 140° C), the higher temperatures being used on clays, although the temperature should not exceed 100°F (37°C) if mercury is to be determined. The submission of larger and excessively wet drill cutting samples to high-volume minerals laboratories has started a trend towards drying rooms or buildings equipped to dry large quantities of samples.

Crushers:

Jaw Crushers—The first stage of crushing normally is accomplished in a laboratory-sized jaw crusher. Many designs still in production date back a half century or more, but the trend in redesigns or new designs is towards roller bearings, short stroke, and a high reduction ratio. Some of these can be choke-fed through a hopper with feed up to 4 in. (100 mm) across, and they will produce a —8-mesh (2.4-mm) product in one pass while the operator is attending to other duties. Jaw crushers have a relatively high productivity and reduction ratio and are generally easy to clean between samples.

Cone Crushers—Normally used in a second stage of crushing, cone crushers produce a uniform-sized product with a smaller percentage of fines, considered better for metallurgical testing, and most can crush to 10 mesh (2 mm). Cone crushers are not effective on clays, but work best on hard siliceous materials. They have a relatively low productivity and are difficult to clean, especially after clayey samples.

Roll Crushers—As an alternative to cone crushers for second-stage crushing, roll crushers have a higher productivity and can produce a —10-mesh (2-mm) product. However, they produce a poorly sorted product with a higher percentage of fines and are noisy, dusty, and difficult to clean. The feed to roll crushers must be sized to —3/8 or 1/2 in. (10 or 12 mm), and the feed rate must be controlled to prevent choking.

Hammer Mills—Hammer mills have a high productivity and the potential to produce a product suitable for splitting and pulverizing in one pass from feed as large as 4 in. (100 mm). However, they are extremely noisy, dusty, hard to clean, and subject to excessive wear when processing tough siliceous materials. Their product is not well sorted, typically consisting of a large percentage of fines with a small percentage of very coarse fragments. Hammer mills are more often used to crush clays, limestone, coal, and similar softer materials.

Splitters: At some point in the sample size-reduction process, it becomes impractical and unnecessary to further reduce the particle size of the entire sample. The sample volume is then reduced by half or more, depending on the procedure selected as appropriate for the material at hand, by using a sample splitter.

Riffle Splitters—The riffle splitter, or Jones splitter, is most commonly used for sample size reduction. The technique of splitting must be carefully monitored to insure that statistically valid splits are taken. Splitter chutes should be at least three times as wide as the diameter of the largest particles in the sample, and the delivery pan should be no longer than the distance across all of the chutes. The sample should be evenly distributed along the length of the delivery pan and should be poured along the center line of the splitter—not against one side or the other. The rate of pouring must be slow enough to avoid choking the chutes. The splitter must be cleaned between samples.

Rotating Sectorial Splitters—The most effective splitter, in terms of sampling error, is the rotating sectorial splitter (Allen and Khan, 1970). Various designs exist, most being adapted to splitting the product from rotary drill rigs and thus larger than appropriate for use in the laboratory. Shop-made laboratory-sized versions of the rotating sectorial splitter are in use in some sample preparation facilities. One design consists of a rotating circular table driven by a variable-speed motor at about 10 rpm. Four plastic cartons of 1-pt or 1-qt (1/2 or 1-L) capacity rest on the rotating table. The gaps between the cartons are covered by pieces of angle iron. The sample is fed from a feed hopper by a vibratory feeder.

Pulverizers: Once the sample is reduced to an appropriate weight (typically 1/4 to 1 lb, or 100 to 500 g), it is then pulverized to a nominal 100 to 200 mesh (150 to 75 µm). Two basic types of pulverizers are in common use: the plate or disk pulverizer and the vibratory ring mill.

Plate Pulverizers—Plate pulverizers reduce the particle size of samples through a shearing action as the sample passes between a fixed and a rotating plate. Two basic designs exist: horizontal shafts and vertical shafts. Some are designed only for intermittent use, whereas the more recent versions are extremely rugged and built for high volume work. Controlling the spacing between the plates (and thus the product size) has been a problem, and has required almost constant attention by the operator, but recent models of plate pulverizers have better mechanical or even pneumatic controls that maintain a constant plate separation. Plate pulverizers have a high productivity, they can pulverize a large sample, and are easily cleaned. However, they are dusty and their product is inhomogeneous, requiring thorough blending before assaying. Mikli (1986) does not recommend plate pulverizers for the final pulverizing of nuggety gold ores, as the plate pulverizer does not significantly reduce the particle size of gold nuggets. Plate wear is high, requiring frequent changing and refinishing or replacing of plates.

Vibratory Ring Mills—Two basic versions of vibratory ring mills are made. The first consists of a steel bowl with lid, the bowl containing a cylindrical steel puck plus one or two steel rings that surround the puck. The crushed sample is placed in the open space between the walls of the bowl, the puck, and the rings. The bowl is clamped in a housing which is made to oscillate around a vertical axis by an electric motor carrying an eccentric weight on its shaft. The oscillatory motion causes the puck and rings to revolve in a planetary pattern inside the bowl, thus grinding the sample.

The second version of this mill also consists of a bowl with lid but only a single grinding element, a discus- or "flying saucer"-shaped oblate spheroid of steel with a flat rim. The center of gravity of this steel element is off center so that when the bowl oscillates, the grinding element revolves in a planetary pattern within the bowl. Bowls of this style have a capacity of 1.8 to 11 lb (800 g to 5 kg) of sample, whereas the puck and ring bowls range from 0.1 to 0.9 lb (50 to 400 g) in capacity.

Vibratory ring mills exhibit the following advantages: they require no operator adjustment, produce a relatively homogeneous product that requires no further blending, create little dust, have a low noise level (because of a supplied noise-suppressant cabinet), and have a moderate productivity of some 12 to 20 samples per hour. The productivity depends on the character of the samples, with the lower productivity being achieved on high-clay samples, which require less than 1 min to pulverize, but several minutes' cleanup time between samples. Vibratory ring mills have been described by Mikli (1986) as the only type of pulverizer that actually can reduce the particle size of gold nuggets. However, to reduce the particle size of coarse nuggety gold or to pulverize a 4.5-lb (2-kg) or larger sample thoroughly to — 80 mesh (177 µm) or finer can require from 10 to 30 min and result in excessive bowl wear.

Table 5.4.1. Representative Sample Preparation Procedure

Dry Sample
(Typical wt. 2 to 10 lb, or 1 to 4.5 kg)
Crush
(Typical product 8 to 10 mesh, or 2.4 to 2 mm)
Riffle Split
(Retain 1/2 lb, or 250 g)
Pulverize
(Typical product 100 to 150 mesh, or 150 to 100 µm)

Blenders and Pulp Splitters: Pulps prepared on plate pulverizers require blending. The most common approach (although not the most effective) is to roll the pulp on a rubberized cloth. Taggart (1945, Sec. 19, p. 71) gives instructions for proper rolling: "Rolling is accomplished by drawing the corners of the cloth horizontally toward diagonally opposite corners, causing the sample to roll over and over on itself. If the corner is lifted instead of drawn horizontally, the sample merely slides along the surface of the cloth and no mixing occurs." Plastic sheeting should not be used for blending because of the static charges that build up, causing retention of some particles and difficulty in cleaning. Some question exists as to whether or not rolling actually is very effective in homogenizing a pulp.

A far superior alternative to rolling is the use of a mechanical blender. Large V-shaped blenders have been used to blend bulk samples for use as reference standards, but small sizes suitable for blending assay pulps are not common. Individual sample blenders are slow. A multisample mechanical wheel blender that meets the productivity requirements of a high-volume minerals laboratory has been described by Gilbert (1987).

Mechanical splitters for use on pulps exist, but they are expensive, slow, and require extensive cleaning between samples. A more effective approach is to roll the pulp on a rolling cloth into a "sausage," flatten the sausage to the height of the scoop being used, and then cut several increments from the sausage using a flat-bottomed vertical-sided scoop until the required weight has been withdrawn.

5.4.1.2 Typical Preparation Methods

The procedure given in Table 5.4.1 is typical of many minerals laboratories. It is inexpensive and assumes the sample is relatively homogeneous and the ore particles small.

People experienced with sampling for nuggety gold (or any ore with a potentially large nugget effect) generally collect larger

Table 5.4.2. Typical Sample Preparation Procedure Used on Nuggety Materials

Dry Sample
(Typical wt. 8 to 20 lb, or 3.6 to 9 kg)
Crush
(Typical product 8 to 10 mesh, or 2.4 to 2 mm)
Riffle Split
(Retain 4 to 10 lb, or 1.8 to 4.5 kg)
Plate Pulverize
(Product 40 mesh, 420 µm, or finer)
Riffle Split
(Retain 1 to 2 lb, or 450 to 900 g)
Pulverize in Ring Mill
(Typical product 100 to 150 mesh, or 150 to 100 µm)

samples and prefer a sample preparation procedure similar to that outlined in Table 5.4.2.

5.4.1.3 Selecting a Sample Preparation Procedure

Virtually every mineral deposit has its own eccentricities, and an individual sample preparation procedure must be developed for each, unless the deposit is known to be fine-grained and relatively homogeneous. A "safe" sample preparation procedure is given by Royle (1989, p. 40) based on a method originally developed by Gy (1977). Similarly, Pitard (1987) presents sampling nomographs that enable one graphically to analyze an existing sample preparation process and to develop an optimum protocol. Pitard markets programs for personal computers that describe tests to perform on gold ores of unknown characteristics and enable one to plot sampling nomographs from which an appropriate sampling protocol can be developed. In their Chapter 1, Sampling, Ingamells and Pitard (1986, pp. 1-84) present a good review of the necessity for intelligent sampling, both before and during the sample preparation and analytical stages.

5.4.2 ASSAY METHODS

Two basic classes of assay methods are available: geochemical and quantitative. Geochemical procedures typically are used in prospecting and the early stages of exploration when results of high accuracy and precision are not as necessary, but low levels of detection are required. Quantitative procedures are used during exploration drilling, sampling and analysis for ore reserve estimation, and subsequent stages of mine development and operation. Geochemical methods of analysis in the past have been considered semi-quantitative, but many of these now approach quantitative methods in accuracy and precision, and they offer the advantage of considerably lower levels of detection.

Whether establishing an in-house laboratory or selecting a commercial laboratory, there is no substitute for a chemist with experience in the analysis of earth materials. The wide variety of materials to be analyzed, constituents to be determined, and matrix compositions to be encountered pose a myriad of challenges to the minerals analyst. Many constructive comments on the art of geochemical analysis are given by Ingamells and Pitard (1986).

5.4.2.1 Wet Chemistry

Wet chemical methods of analysis have been and probably will continue to be the mainstay of the minerals laboratory, in contrast to nondestructive instrumental methods. Wet chemical methods have undergone significant changes with time, perhaps the most dramatic following the evolution of microelectronics and computers that allowed the development of instrumental methods of measurement.

Classical Volumetric and Gravimetric Methods: Volumetric and gravimetric methods of analysis now are used principally for the determination of the higher concentrations of various elements in ores and concentrates. Typical examples are the determination of calcium in limestone; iron in iron ores; ferrous iron in most mineral materials; copper, lead, zinc, etc., in ores and concentrates; and uranium in ores and yellow cake. Most classical methods of analysis are well-established. Many are given in references such as Furman (1962), Young (1971), and Dolezal et al. (1968).

Colorimetric Analysis: Colorimetric methods of analysis were used extensively in the mineral industry in the mid-1900s, especially in trace analysis for geochemical exploration purposes. More recently, colorimetric methods of analysis have been supplanted by instrumental methods such as atomic absorption and inductively coupled plasma spectrometry.

Colorimetric methods of analysis are based on the development of a colored compound in solution by the reaction of the element in question with a specific chemical reagent. One of the principal colorimetric methods still in common use is the thiocyanate assay procedure for molybdenum in ores and concentrates (Ingamells and Pitard, 1986, p. 523). Simple colorimetric methods of analysis useful in geochemical exploration in remote areas are ammonium citrate-soluble heavy metals (Bloom, 1955) and cold acid-extractable copper (Canney and Hawkins, 1958). Colorimetric analytical methods for tungsten and molybdenum produce results comparable with atomic absorption procedures, but they are not as rapid.

Instrumental Analysis: Instrumental methods of analysis have dominated minerals laboratories since the development of the atomic absorption spectrophotometer in Australia in the mid-1950s and the inductively coupled plasma emission spectrometer in the 1970s.

The atomic absorption spectrophotometer (AA) enables the analyst to measure the concentration of some cations down to a fraction of a part per million and with specialized attachments down to a part per billion. The AA instrument generally is specific for the element selected, although there are various interferences and operational nuances that the analyst must be aware of and either avoid or compensate for.

Principles of AA operation are given by Slavin (1968). Many methods of analysis are given in publications of the US Geological Survey and the Geological Survey of Canada as well as the *Atomic Absorption Newsletter* (now titled *Atomic Spectroscopy)* published by the Perkin Elmer Corp.

Inductively coupled plasma spectrometers (ICPs) were developed as modifications of existing direct-reading emission spectrometers or atomic absorption spectrophotometers. The direct readers became ICPs capable of measuring 30 or more elements simultaneously, whereas the AAs became ICPs that measure a number of elements sequentially, commonly utilizing more than one wavelength per element. Thus the simultaneous instrument is faster for routine work on similar materials, whereas the sequential instrument is more versatile. Most ICP instruments utilize the high energy of an argon plasma to excite atoms of various elements in a sample solution that is aspirated into the plasma. The excited atoms emit light that is passed through a spectrometer, wherein the energy of light emitted at various wavelengths is measured electronically and converted into the concentration of each element in the sample solution.

ICPs are supplanting AAs in many high-volume mineral laboratories because they allow the determination of a large number of elements in a short period of time, whereas the AA determines one or at best a few elements simultaneously. The accuracy and precision of ICP instruments and AAs are comparable, and the capital costs, considering the productivity, probably favor the ICP. The ICP can achieve detection levels as low or perhaps slightly lower than the AA, and in addition the ICP instruments have a linear response over some five to seven orders of magnitude of concentration of the element in question, a much broader operating range than the AA.

ICPs are used to determine both trace elements and major elements in most earth materials. A wide variety of digestion and separation procedures are used to prepare the sample for ICP measurement, many of them similar to AA digestion procedures. A large literature exists.

Several other instruments are used for specific determinations in minerals laboratories. Specific ion electrodes are used to measure the concentration of cyanide in dilute solutions as well as the content of fluorine, chlorine, and several other constituents in earth materials, and the fluorometer is the standard instrument for determining trace amounts of uranium.

5.4.2.2 Fire Assay

A fire assay is a chemical fusion method for separating, concentrating, and measuring the content of gold and silver in exploration samples, ores, and concentrates. The pulverized sample is weighed, mixed with a lead oxide-alkali carbonate flux and a small amount of reducing agent such as flour, and fused in a fire-clay crucible. The reduced lead collects the precious metals as it settles down through the melt. The molten charge is then poured into a mold to cool. The lead sinks to the bottom of the mold and is broken from the glassy slag when cool. The precious metals are separated from the lead by cupellation. The lead button containing the precious metals is placed in a cupel of bone ash or magnesia which is heated in the furnace under oxidizing conditions. The cupel acts as a semi-permeable membrane allowing the lead oxide formed from the button to be absorbed into the cupel, leaving the precious metals in a tiny bead on the surface of the cupel. The gold and silver are separated chemically and quantified gravimetrically or by instrumental analytical techniques.

Fire assay is the standard method of the industry. Details, theory, and variations of the method are described in references such as Bugbee (1940), Haffty et al. (1977), and Heady and Broadhead (1976).

5.4.2.3 Nondestructive Analysis

X-Ray: The X-ray diffractometer is used in some well-equipped minerals laboratories to determine the mineral species present in a sample by recording their characteristic crystallographic patterns. The instruments are not extensively used in mine and project assay work.

X-ray fluorescence spectrometry (XRF) is used for the rapid analysis of silicate rocks and the routine determination of the concentration of selected elements in exploration samples, ores, concentrates, and mill products. XRF analysis is most satisfactory when applied to a continuing series of samples of similar matrix. The method is subject to matrix effects and interelement interferences, most of which can be corrected for in the comprehensive computer software that accompanies all modern instruments. XRF instruments require for calibration previously analyzed standards similar in bulk composition to the unknowns. The instruments are capable of excellent precision, but without proper calibration and intelligent operation, they can be rather inaccurate.

Radiometric Analysis: Radiometric analysis is used to measure the concentration of uranium in its ores by simultaneously counting the beta and gamma radiation emitted by the sample.

Multispectral radiometric analysis provides a determination of the concentrations of the radioactive elements uranium, potassium, and thorium, all of which will provide a positive response to a simple gamma ray detector such as a Geiger-Mueller counter or a scintillometer.

Neutron Activation Analysis: Neutron activation analysis (NAA) is available principally through commercial laboratories with access to nuclear reactors. Most of the laboratories offering commercial NAA analyses of earth materials at reasonable cost are Canadian-based. NAA is indicated when (1) a conventional technique does not have acceptable limits of detection for the element of interest, (2) the sample is unique and cannot be consumed in analysis, (3) only a small quantity of sample material is available, or (4) conventional methods of analysis are unacceptable because of interferences or inherent instrumental errors. The sample typically is pulverized, loaded into a plastic capsule (rabbit), and introduced to a reactor in which it is bombarded by neutrons. After recovery of the capsule from the reactor, the radioactivity induced in the sample is measured and analyzed, thus giving a measure of the concentration of each element present in the sample. NAA currently has much application in the analysis of vegetation or mull for trace amounts of gold, and in the analysis of the platinum-group elements and the rare earth elements.

5.4.3 COAL PREPARATION AND ANALYSIS

The procedures used in the preparation of coal and coke samples for analysis are similar to those for rock samples. The ASTM standard method of preparing coal samples is No. D 2013-86, published in the *Annual Book of ASTM Standards*, Pt. 05.5 (Anon., 1989). The principal difference in coal sample preparation is the use of lower temperatures and, indeed, even air drying to preclude oxidation as much as possible. Crushing and pulverizing is accomplished with much the same equipment as used on rocks. However, samples are pulverized only to —60 mesh (250 µm), and only 0.1 lb (50 g) is retained for analysis.

A variety of tests are available for coal samples. Some of these include sieve analyses, washability, Hardgrove grindability, moisture, sulfur, ash content, carbon and hydrogen content, and calorific value. Two of the most common determinations are proximate and ultimate analyses. A proximate analysis, as described in ASTM Standard Method D 3172-73, covers the determination of moisture, volatile matter, and ash, as well as the calculation of fixed carbon. According to ASTM (Anon., 1989, Method D 3172-73, p. 1), the results of a proximate analysis are used to establish the rank of coals, to show the ratio of combustible to incombustible constituents, to evaluate the coal for beneficiation and other purposes, and to provide a basis for buying and selling.

An ultimate analysis, according to ASTM Standard Method D 3176-84, p. 1 (Anon., 1989), when tabulated along with a proximate analysis, provides the data for a cursory valuation of coal for use as a fuel and coke for metallurgical purposes. An ultimate analysis includes the determination of carbon and hydrogen as well as sulfur, nitrogen, ash, and the calculation of oxygen by difference. Typically, moisture is reported as well. The procedures for an ultimate analysis are also specified by ASTM (Anon., 1989).

In addition to the analytical procedures described previously, data on the major, minor, and trace elements in coal and coke ash often are of use in the evaluation of coal quality. Accordingly, methods for these determinations are presented in ASTM Standard Test Method D 3682-87, "Major and Minor Elements in Coal and Coke Ash by Atomic Absorption," ASTM Standard Test Method D 4326-84, "Major and Minor Elements in Coal and Coke Ash by X-Ray Fluorescence," and ASTM Standard Test Method D 3683-78, "Trace Elements in Coal and Coke Ash by Atomic Absorption." The atomic absorption measurement of the major and minor elements is accomplished after a lithium tetraborate fusion of the sample followed by hydrochloric acid dissolution of the fusion product, whereas the trace elements are determined following a mineral acid dissolution process.

5.4.4 QUALITY CONTROL

Most laboratories have an adequate quality control program covering their analytical work. Some commercial laboratories actually provide results of check assays with their reports of analysis. It would appear, however, that few laboratories include sample preparation in their quality control system; yet, as Royle (1989) has commented, "Wait until you see the numbers that show all the horrible things that happen in sample preparation of gold samples!"

During sample preparation, maximum particle size readily can be monitored by screening at various stages as the material is crushed and pulverized. Other parameters are more easily checked by actually analyzing the material.

Analysis of several pulps prepared from different splits of the crushed bulk sample will provide data on reproducibility of splitting. Analogously, the homogeneity of a pulp can be checked by taking a number of replicate portions for analysis from the same pulp and calculating the precision of replication. A simple procedure to evaluate the effectiveness of sampling drill cuttings is presented by Schwarz (1989).

Analytical precision and accuracy are best established and maintained through the use of reference standard samples and analytical control samples (Hill, 1975). Certified standard samples of mineral materials are available from some governmental agencies and professional or trade groups in several countries (Abbey, 1983; Flanagan, 1986). Such standard materials are expensive and limited in quantity. The elements present and concentration ranges in certified standards may not adequately cover the character of materials that the laboratory is engaged with; nevertheless, such standard materials may help to provide initial calibration. Matrix-matched control samples to provide batch-to-batch and day-to-day calibration and quality control in the laboratory can be prepared and blended from the reject samples from the project. The analytical development of these in-house control samples can be calibrated to the certified standards. Ideally, every set of analyses produced by the laboratory should have one or more control samples and duplicates included for quality control and assurance. Some governmental agencies that contract out a large quantity of sample preparation and analytical work arrange for 3 out of each 20 samples to be controls or replicates. Those more mathematically inclined can find much of value regarding analytical quality control in the laboratory in the *Quality Assurance Handbook* compiled by the Center for Analytical Chemistry of the National Bureau of Standards (Anon., 1987). ASTM gives guidelines for evaluating laboratories performing analysis of coal and coke (Anon., 1989, p. 357).

When the results of analysis of the controls do not agree with the established values within acceptable limits, the entire group of assays should be rejected, the problem identified and resolved, and the group of samples rerun. The timely realization and resolving of sample preparation and analytical problems is the essence of quality control.

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Chapter 5.5
GEOLOGIC INTERPRETATION, MODELING, AND REPRESENTATION

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Chapter 5.2 provides a discussion of the importance of clear, complete, accurate, detailed, systematic collection of varied geologic data and suggests methodologies to accomplish this goal. Narrative guidelines and narrative-graphical formats are provided to assure the desired standardization in the acquisition of geologic information. The area of activity in the chapter is referred to as the data collection step in the three-step geologic information processing effort required in resource evaluation.

This chapter builds on Chapter 5.2 in discussing the purpose and recommending methodology in carrying out the remaining two steps, data analysis and interpretation. All three steps are required to discharge the critical responsibility of resource characterization for quantification, evaluation, mine planning, and extraction. *Resource characterization* is defined as the determination of the shape, size, quality, quantity, and variability of the geologic entity and, as importantly, determining the limits of variable geologic features. Proper characterization has both operational and exploration implications as it provides the information for synthesis of commonly subtle features into an accurate, predictive description of the resource environment. This description can then be used either in support of mining operations or in the ongoing search for additional reserves.

5.5.1 GEOLOGIC MODELING: GENERAL

Current terminology refers to resource characterization as the geologic model, or perhaps the three-dimensional geologic model, whereas earlier studies used the nearly synonymous term, zoning patterns. This model consists of a compilation of all geologic data, observations, and studies available at the time, assembled in such a way as to display and explain the observations from both an empirical and genetic point of view. The model may be extremely simple or highly complex depending on the nature of the resource, the data available, or the degree of sophistication in studies of the resource. The empirical model represents the compilation and integration of numerous types of chemical, mineralogical, structural, and not uncommonly numerically quantifiable zoning studies, whereas the conceptual or genetic model attempts to explain the distribution and origin of these features in a useful and meaningful way. The geologic model is commonly incomplete (Barton, 1986, p. iii), and as discussed by Peters (1987, pp. 214-217), the model will probably change or be revised as additional geologic data are acquired.

Erickson (1982), Eckstrand (1984), Cox and Singer (1986), and particularly Roberts and Sheahan (1988), together with reviews of these volumes by Barton (1985, p. 1758) and Skinner (1989, p. 725), present up-to-date synthesis of geologic data from a broad spectrum of mineral deposits into a number of both empirical and genetic models. Work by Dapples and Hopkins (1969), Horne and Fern (1978), Donaldson et al. (1979), Kaiser et al. (1980), Ayers and Kaiser (1984), Rahmani and Flores (1984), Walker (1984), Ward (1984), Reineck and Singh (1986), and Ayers (1986) provide excellent descriptions of coal deposits and associated geologic environments and processes associated with this resource. Models described in all these references provide a state-of-the-art picture of the products and of geologic processes and events and are extremely useful in guiding broad-scale exploration activity or more tightly controlled resource evaluation and operational support work. They are excellent and useful volumes as they are compilations from and syntheses of numerous, varied, studies of long-established mining districts or regions. Typically, these are areas where extensive exploration, development, and operational geologic programs have provided large factual geologic databases and extensive material for laboratory studies. Although each district or region may exhibit specific variations, the volumes summarize observations and studies from many deposits of a similar nature and hence provide a guide to the commonly observed typical features to be expected in a particular deposit type—the geologic model. They are extremely important and useful guides for interpreting geologic observations while work is being conducted on projects, when new district specific models are being constructed, and as aids during ongoing exploration or development programs.

Models should attempt to explain all observable facts. Geologic data should not be ignored or discarded if they do not fit the perceived model. The lack of agreement between facts and theory may indicate incorrect data, incorrect models, or insufficient data or all three, and thus point the way for further investigation.

The fundamental problem associated with resource characterization for evaluation and estimation is taking a very limited amount of geologic data, correctly analyzing and interpreting the data, extending these interpretations into unknown areas, and then making summary quality, quantity, and limit statements about the area. In simplest terms, where is the resource (ore), what are its limits (shape), what is the quality (grade), and what is the nature of the associated environment. Unfortunately, this implies sharp boundaries, which are seen in only a few specific types of deposits. More often, we are dealing with an irregular, poorly constrained distribution of values of a commodity as discussed and illustrated by Gentry and O'Neil (1984, pp. 59-61). Their illustration, here reproduced with minor modification as Fig. 5.5.1, is a lucid, simple illustration that clearly explains the resource estimation, external and internal dilution, and limit or edge problems that plague many operations.

Sound, careful geologic data analysis during model construction, frequently with much checking and rechecking as pointed out by Worthington and Brown (1984, p. 209), goes a long way towards solving the afore-described fundamental geologic problems in resource characterization. As discussed by Ranta et al. (1984, pp. 1-2), the geologic data and interpretation (the model), are the foundation of the evaluation process and are essential for the resource estimation and extraction. As Call (1979, p. 31) indicates in his discussion of statistical data handling in development drilling programs, "the validity of any analytical model must ultimately be determined by geologic interpretation." Sound geologic judgment, discipline, and hard work are needed to deduce the detailed geologic setting from drillholes, mapping, and sampling programs, which lead to the construction and understanding of the geologic model and its variations. The model is based on numerous, varied geologic studies, and assuming that all other factors (sampling, hole or data locations, analysis, etc.) are correct, ensures a reliable, best possible resource estimate, together with an understanding of

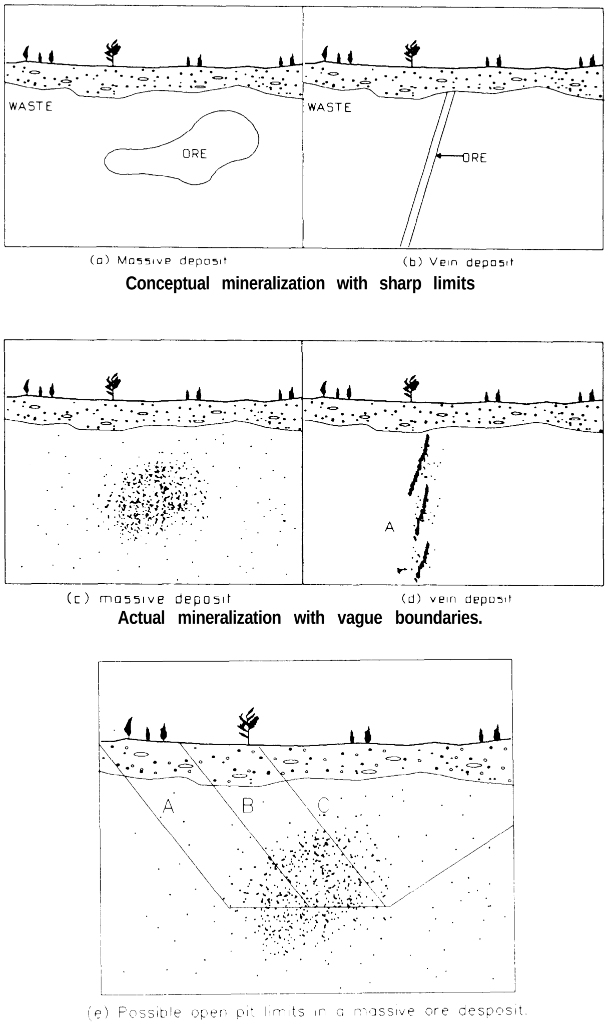


Fig. 5.5.1. Ore body representation. (a,b) Common conceptualization of ore bodies with clear, well-defined boundaries. (c,d) Actual typical distribution of a valuable commodity within an ore body with vague irregular boundaries. (e) Illustrates how an open pit mine may extract waste and the valuable commodity. Modified after Gentry and O'Neil, 1984.

ground conditions. "Without correct interpretation, drillhole data" (or any other data) can be misleading (Ranta et al., 1984, p. 2) and manual, computer-assisted, or the more sophisticated geostatistical estimates may be incorrect. The simple line on a map or cross section depicting the limit of a geologic feature to be estimated is an extremely powerful decision-influencing item, and only proper geologic interpretation assures its correctness. Sound interpretation is the only way a geologist can fulfill the challenge issued by Barnes (1980, p. 62) to "accept the responsibility of recognizing and recording boundaries" that characterize the deposit or distributions within the deposit.

In summary, "Good geology based on well-understood data is still the only recipe for good reservoir/site/deposit characterization" (Journel, 1988, p. ii). The geologist must thoroughly understand the total geologic environment and distributions to reliably estimate resources, delineate areas for mine planning, and support day-to-day operations.

5.5.2 GEOLOGIC MODELING: METHODOLOGY

With the foregoing serving as a somewhat detailed discussion of geologic characterization—the interpretative model—and the importance of developing a good model—for accurate resource evaluation—the remaining provides guidelines on how best to construct a geologic model of a resource. Emphasis is on an empirical model, one that accurately records the factual geologic observations.

The geologist normally starts with surface outcrop data and drillhole logs that, as reviewed in Chapter 5.2, are of high quality and have been collected using a rigorous standardized methodology providing unbiased factual data for compilation and analysis. The geologist builds a geologic outcrop map from surface work that contains all geologic observations. This map must clearly differentiate factual outcrop data from geologic interpretation and inference between outcrops. A common graphic methodology uses bold solid lines and patterns or dark colors for outcrop and dashed or dotted lines with similar colors, but applied in a pale or lighter fashion, for interpretations. As indicated by Herness (1977, p. 529), "there must be no screening of data to eliminate 'unimportant' facts" . . . as "often the importance of insignificant data is realized 20 or 50 years later" when mine workings or core unfortunately may not be available for remapping.

As an aside, with respect to core availability, the deliberate disposal of core from known ore deposits or established mining districts is unconscionable. Stored core is simply one of a number of the geologist's files that will be constantly reviewed as new ideas or data become available. "Core is our record; we must always go back to it" (Klohn, 1989). Numerous ore discoveries have been made because core was available for reexamination, resampling, and reinterpretation. A recent talk and abstract (Braun, 1991, p. 146, and personal communication) clearly demonstrate the importance of the availability of old drill core for relogging and resampling in the discovery of a new gold ore body. Future discoveries will be made because old core was available for examination in the light of new data, new ideas, other commodities, or, unfortunately, simply because a better or more conscientious geologist was involved. Geology is a continually evolving discipline, and new knowledge and understanding are accumulating at an astonishing rate. Discarding core because it has been logged is the equivalent of destroying survey or production records or a computer database after a set of maps or financial algorithms have been completed. It should not be done!

Gustafson (1989, pp. 987-993) in an important applied geology lecture and paper, clearly reviews the importance of good mapping to both exploration and operational geology and laments that in the modern era of specialized theoretical studies "perhaps . . . we no longer know how" to map.

After, or in conjunction with, the development of a surface geologic map, it is essential that a detailed set of geologic cross sections (and subsequently plan maps) be constructed, preferably at right angles to each other. This is easier if the critical aspect of a resource can be satisfactorily approximated by a set of N-S and E-W sections tied to a coordinate system. Other orientations are possible, although this can introduce measuring and posting problems in relationship to grid coordinates. Coordinates may also be rotated to provide local grids, but this practice may cause serious survey correction problems and difficulties in tying locations to the US Land System or various state plane coordinates. A reference line to aid in registration and data posting should be drawn on sets of sections, particularly those that do not parallel the coordinate grid system. All data from surface mapping and careful systematic core logging should be posted on appropriate sections and, as indicated by Call (1979, p. 31), "plotted, without interpretation on reproducible sheets. From these, copies can be made for use in interpretation. This process will maintain the distinction between observed facts and interpreted geology." These sections can be constructed manually as the project or drilling proceeds. If an appropriate computer-assisted package is available, data can be entered into an expandable database for rapid factual data posting to provide base maps for interpretation.

As new data are acquired, they are posted on the factual sheets, prints are produced, and new revised interpretations are developed. Additionally, completed interpretations can be transferred to a second set of reproducible bases (copies of the factual data) so multiple interpretative sets of maps can be reproduced for varied users. Drillhole traces and associated data should be plotted on the sections at a scale sufficient to allow for posting of multiple variables along the trace of the hole. Parameters normally of importance are lithology, structure, alteration (if present), grade (as a pattern, a color bar, histograms, or numerical values), and perhaps total sulfide or sulfur content. The particular parameters posted depends upon the resource being evaluated. This posting of several parameters allows for the determination of critical relationships that normally are of use in guiding interpretation and projections. An extremely risky interpretation is the simple correlation of assays (grade) from hole to hole with no consideration of associated geologic features. The process, which should be avoided, commonly leads to incorrect interpretation, overestimation of resources, and incorrect, usually overly optimistic, evaluation.

Clear cross-sectional construction allows one to determine hole-to-hole relationships and continuity of numerous key geologic features such as lithology, alteration, mineralogy, grade, structure, or perhaps features that may influence either mining, such as RQD (rock quality designation), or metallurgical treatment, such as hardness, of the resource. The construction of two sets, commonly at right angles, allows for both determination of hole-to-hole relationships in two directions, and hence section-to-section relationships in three dimensions. The development of a third set of illustrations, plan or level maps (sometimes referred to as slice maps), from the two sets of cross sections, and drillhole pierce points in the planes of the level maps is commonly the final step. This allows one to conform the sections and plans so all common points have similar x, y, and z coordinates in the now completed three-dimensional resource characterization. As indicated, multiple sets showing the important relationships of variables—the geologic model—is the product. The construction of this set of illustrations allows one to develop the fundamental geologic understanding of the resource needed for mine planning and financial evaluation. In the case of coal or uranium, the sets of plans and sections are commonly supplemented or replaced by isopach, isograde, and structure contour maps of varied features of importance. Of particular importance in this understanding is the determination of ore/resource controls such as lithology, primary structural features, secondary structures, structural intersections, or combinations of features that are directly responsible for the specific localization of the resource. Sufficient understanding, particularly of limiting features, should be developed in the process to allow the input of proper geologic controls in either conventional or computer-assisted resources estimation methodologies, such as block modeling or the gridded seam technique.

Fig. 5.5.2, modified from Herness (1951, p. 1008), illustrates the relationships of an orthogonal set of plan maps and cross sections, including the smaller field note-sheet subdivisions, used to collect and compile data and support interpretations in the development of a geologic model. This article provides an ex-

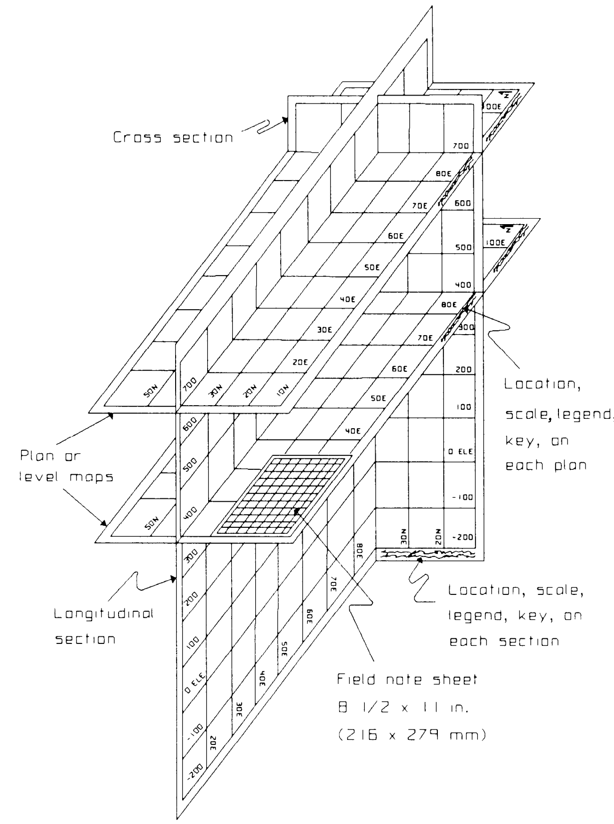


Fig. 5.5.2. Three-dimensional plan, section, and field note sheet relationship. Coordinates and elevations annotated. Scale and explanation omitted. Modified after Herness, 1951.

tremely detailed discussion, illustrations, and recommendations (some dated and others still current) on the essentials of setting up a unified geologic data recording and representation system. Figs. 5.5.3 through 5.5.6 depict the development of factual data and interpretative plans and sections needed to build the geologic model characterizing a particular resource. Combination of features such as mineralization and lithology, not shown here, is common. These types of illustrations ultimately form the basis for models illustrated in Figs. 5.5.7a and 5.5.7b. These latter two illustrations, modified after Roberts and Sheahan (1988, p. 147), illustrate an idealized volcanogenic massive sulfide model characteristics developed from studies of numerous deposits. Fig. 5.5.7b from the same reference illustrates various genetic models that account for the features and suggests processes responsible for formation of the varied deposit types of this class of deposits.

5.5.3 GEOLOGIC MODELING: PURPOSE

The geologic model is constructed to provide a clear picture of the three-dimensional geologic relationship of numerous features that limit varied distribution in the geologic resource. The plans and sections used to develop the model form the total basis for resource estimation in conventional reserve calculation schemes. They provide the input for limiting controls, generally as digitized polygons, in computer-assisted methodologies and are the primary standard of comparison between conventional

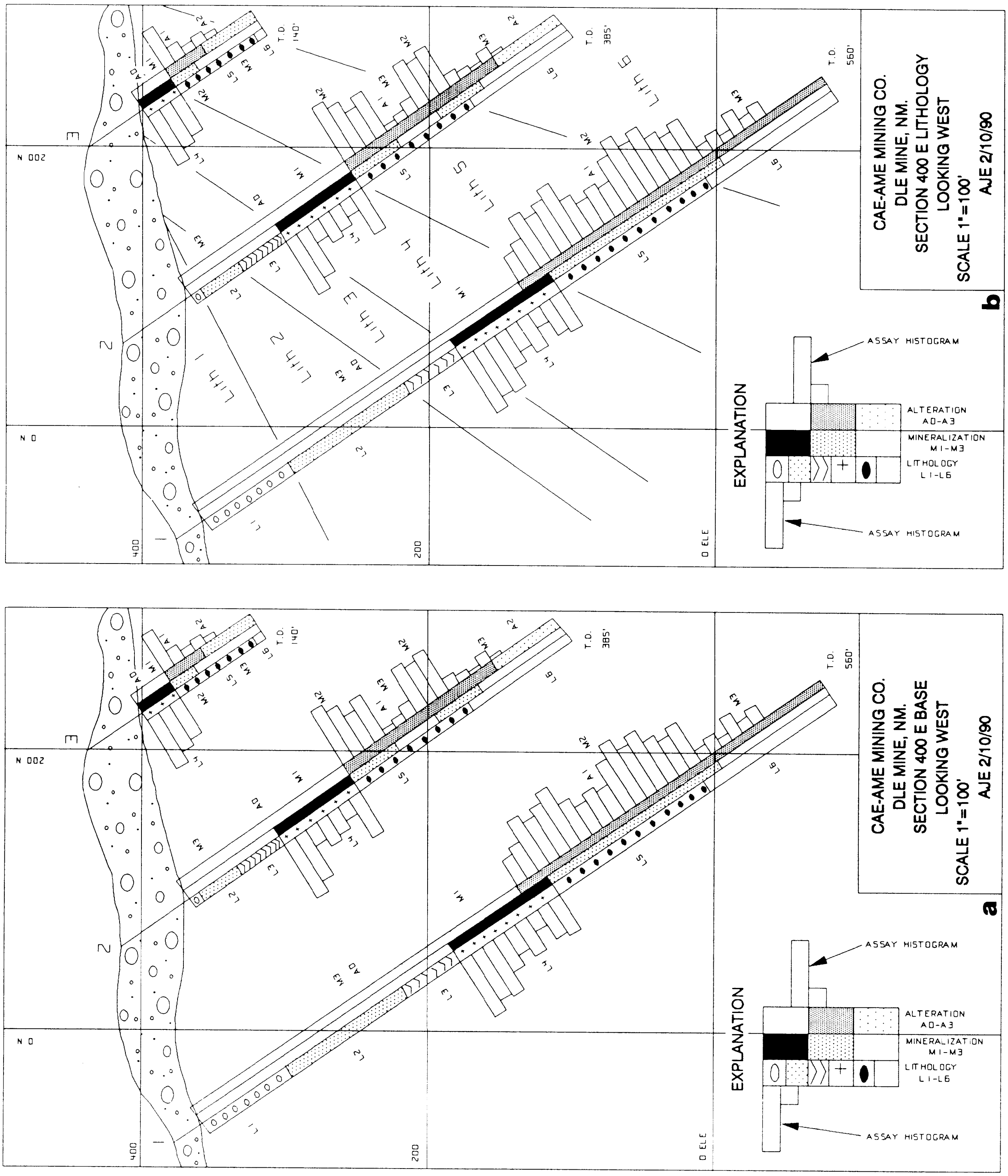
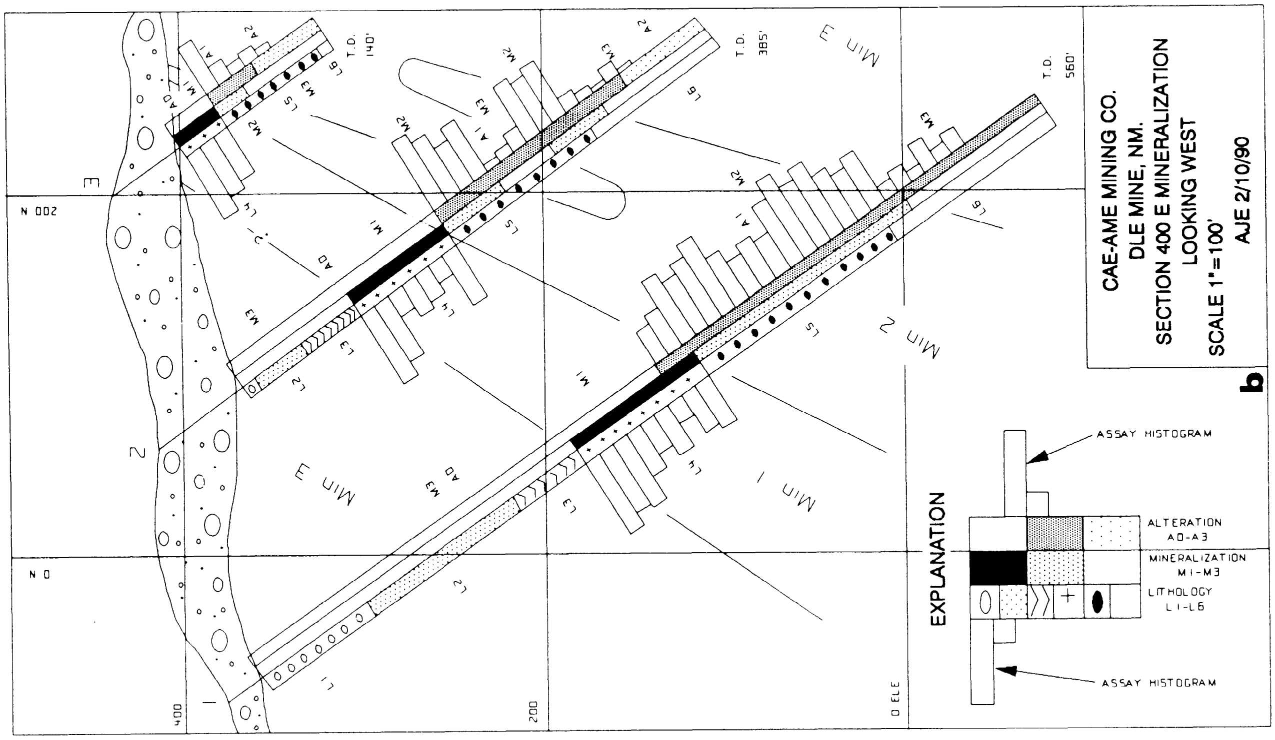


Fig. 5.5.3. Idealized cross section. (a) All factual data manually posted or computer-generated from a database on cross-sectional base showing coordinates, elevations, drillhole trace annotated with multiple geologic variables, and histograms representing metal grades. (b) Lithologic interpretation developed from data posted on the cross-sectional base in Fig. 5.5.3a.



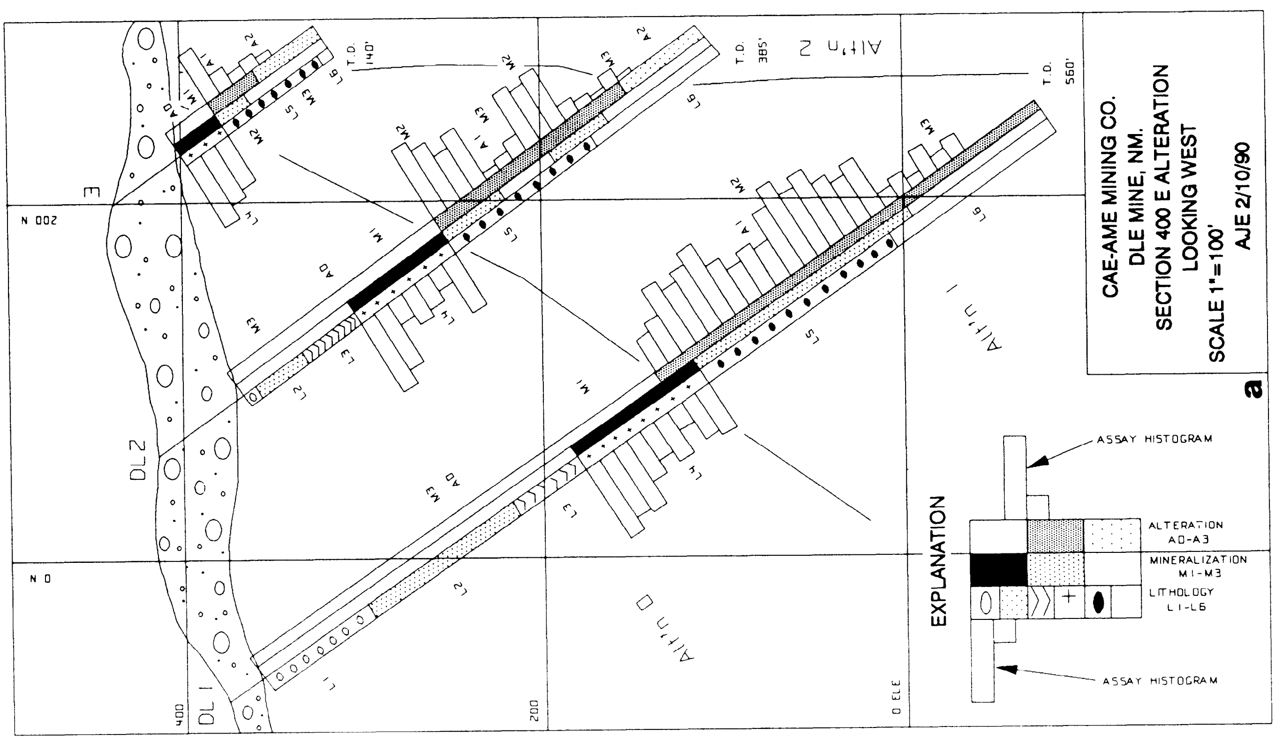


Fig. 5.5.4. Idealized cross section. (a) Alteration interpretation developed from data posted on cross-sectional base in Fig. 5.5.3a. (b) Mineralization and ore body interpretation developed from data posted on the cross-sectional base in Fig. 5.5.3a.

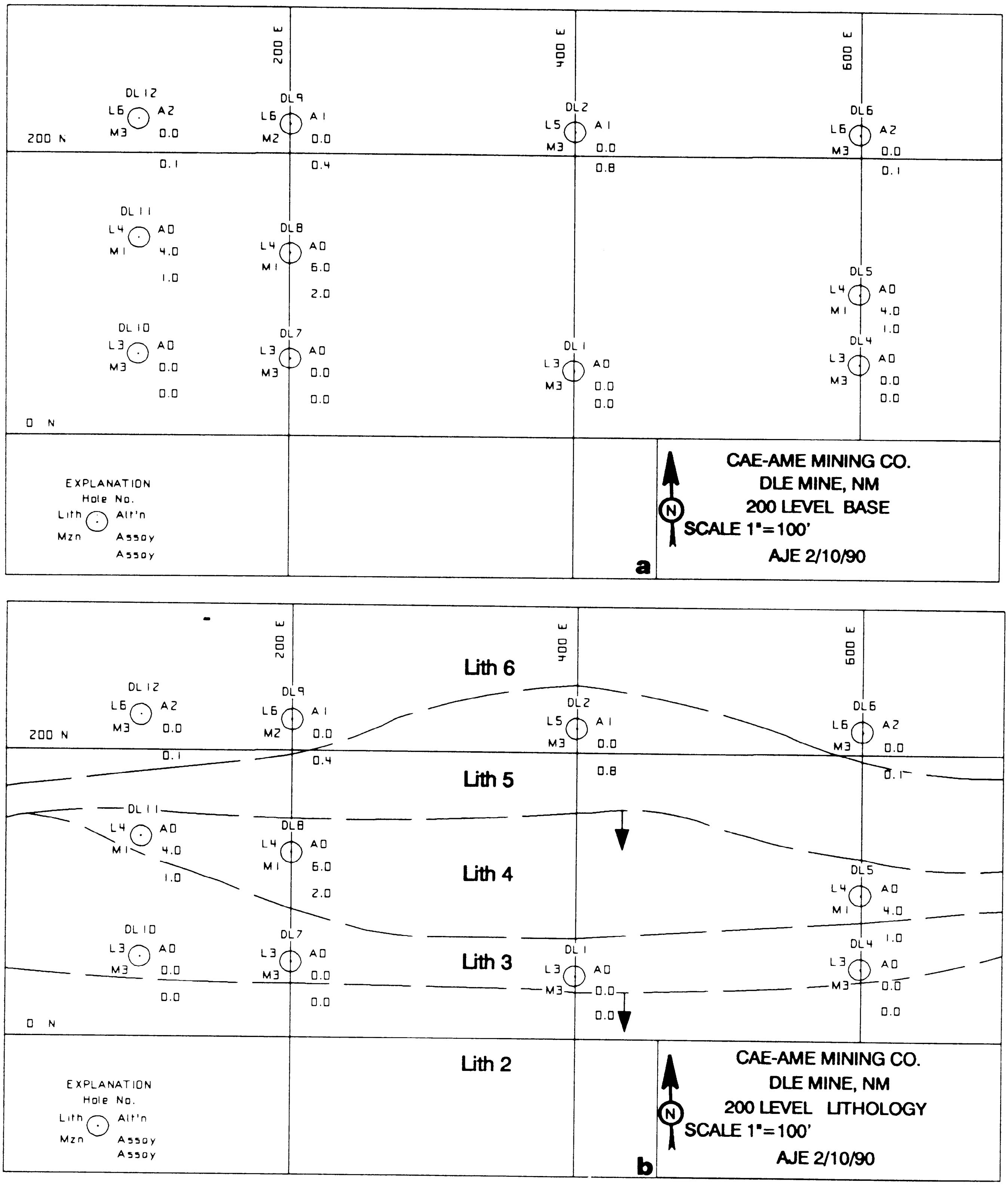


Fig. 5.5.5. Level maps. (a) Plan map at the 200 elevation showing factual data in drillholes. (b) Lithology interpretation at the elevation of the plan map developed from factual data in the drillholes and interpreted contacts transferred from cross section 400E in Fig. 5.5.5b. Only contacts transferred from one cross section are shown, whereas in actual practice data from many sections would be used.

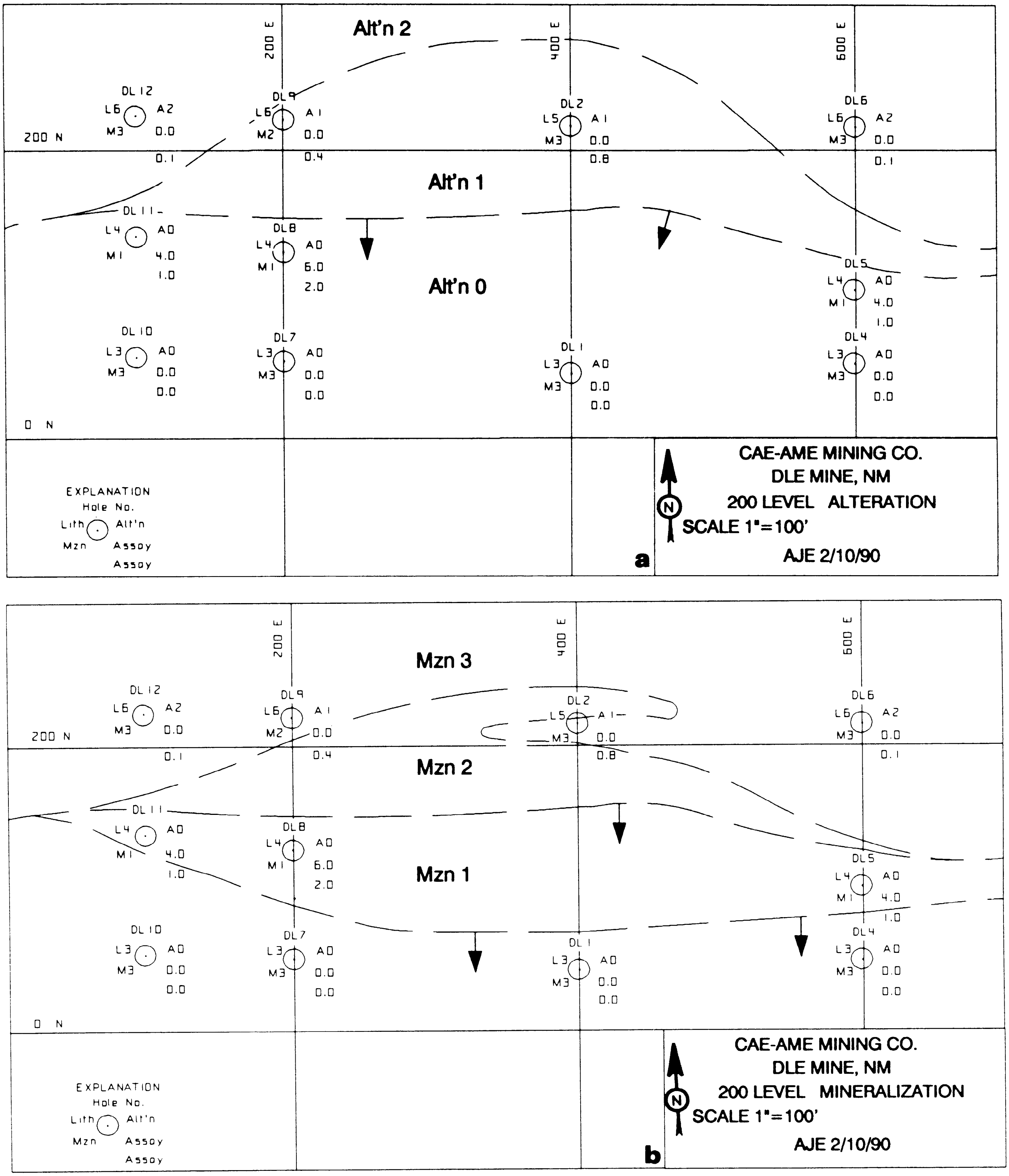


Fig. 5.5.6. Level maps. (a) Alteration interpretation at the elevation of the plan map developed from factual data in the drillholes and interpreted contacts transferred from cross section 400E in Fig. 5.5.5b. (b) Mineralization and ore body interpretation developed from data in Fig. 5.5.5a in a similar manner to lithologic and alteration interpretations shown in Figs. 5.5.5b and 5.5.6a.

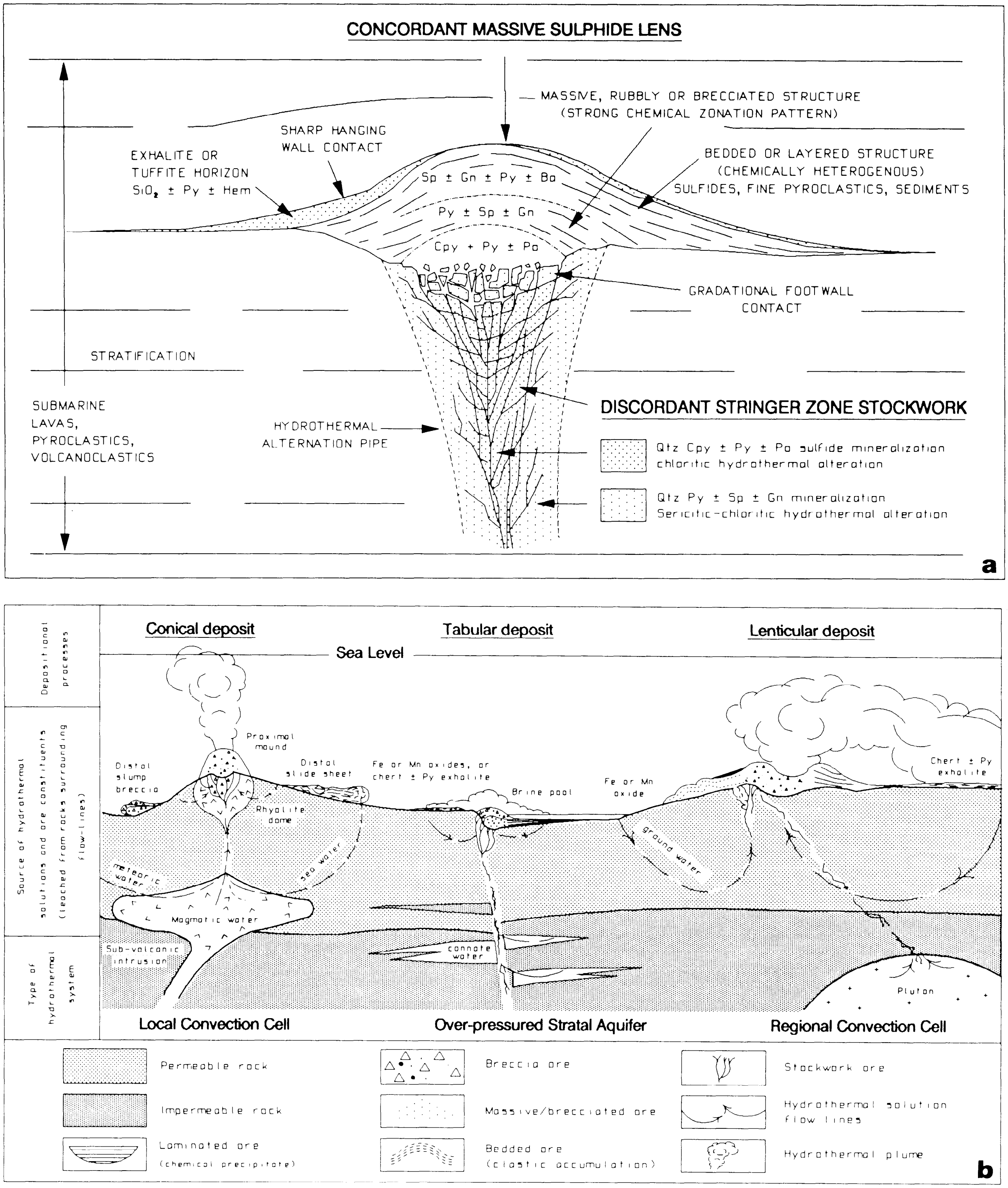


Fig. 5.5.7. Geologic models. (a) Empirical model of a volcanogenic massive sulfide (VMS) deposit showing mineral and metal distributions and features to be expected in the deposits. (b) Genetic model showing the varied geologic environments where VMS deposits are formed. Models of these types are extremely useful in interpreting observation, predicting varied geologic changes to be anticipated, and pointing the way to additional resources. Modified after Roberts and Sheahan, 1988. Not to scale.

and computer-generated resource estimates. The model provides the limits needed to prevent algorithms from estimating resources in unmineralized areas. A well-constructed model assures that the algorithms are being used on correlatable units or features. Not uncommonly, the varied relationships shown on the numerous plans and sections are combined into a few illustrative displays, cartoons, which depict the model and explain the observed geologic relationships as illustrated in Figs. 5.5.7a and Figs. 5.5.7b.

It is becoming increasingly apparent as more sophisticated computer-modeling techniques are developed that, given that correct analytical data and posting are available, the fundamental control of sound geologic interpretation is the most important factor in resource evaluation. Superior models require superior data collection and interpretative work to provide superior resource calculations.

Considering the risk associated with the heavy financial commitments in the resource industries, unquestionably, in the evaluation of all but the smallest resource, the best way to handle the massive amounts of data and to test sensitivities as parameters are varied is though computer modeling of the resource. These models, whether block or gridded seam or others, take a considerable amount of manpower to construct and control and require sound geologic judgment and experience. Once constructed, the parameters in the computer model can be rapidly varied and analyzed to determine sensitivities. It is imperative to have a correct resource determination in support of evaluation and risk assessment to assure the profitable outcome of the heavy financial commitment required for a new mine. The only way to effectively discharge the important geologic responsibility is through construction of an accurate geologic model.

5.5.4 GEOLOGIC DATA: REPRESENTATION

Geologic features of mineral resources are highly variable. A nearly constant change in three-dimensional shapes and relationships is the rule rather than the exception. Many parameters of varying significance must be considered in order to understand the resource and its normally complex geometrical shapes. It is essential that presentations or illustrations used in describing these complex relationships present sufficient, but not excessive, detail to provide accurate, clear, qualitative, and where possible quantitative, or at least semi-quantitative, representations of the important features. The illustrations must be systematic, consistent, preferably simple, and standardized, as professionals of varied disciplines, frequently nontechnical, will commonly be using them. Relationships depicted or documented by the illustration should be instantly obvious with a minimal chance for confusion or misinterpretation. The user must be able to quickly grasp the significance of relationships. Additionally, illustrations should be easy to update to assure that new data can be rapidly assimilated and available to support the decision-making process.

The basic illustrations commonly used to depict important geologic relationships should consist of an integrated set of plan maps and sections showing distributions of lithologies, alteration, and the resource under study. Structural features are commonly shown on all sets. Combinations of various parameters such as mineralization with lithology or perhaps isogrades reflecting sulfur distribution within minable areas of a specific coal seam are common. Almost any feature (parameter) showing variability can be mapped and, if of importance to resource characterization, can be illustrated in such a way as to depict the importance of the relationship. Sets of illustrations should have common coordinates and points of origins so they include identical areas. Sheet size should be standardized and neither too small to prevent adequate coverage of important areas nor so large as to be unwieldy if frequently used. Sheets 30 × 42 in. (760 × 1070 mm), a standard size readily available, are a good choice for most geologic work. Sheets should have coordinate grid lines with full numerical values of the coordinates identified, not coordinates with some digits removed for ease in posting, as this leads to questions concerning the actual location of data under discussion. (For example, 2,550,000 N should be stated as such and not abridged to 50,000 N). Similarly, elevations on cross sections should not be abridged or modified to reflect anything other than true elevation as related to sea level. Schemes which add or subtract constants to elevations tend to confuse and should be avoided.

Legends or explanations should be complete and on each illustration. The legend should include all symbology, have the scale shown as both a bar and ratio or representative fraction, and include location name, date, topic and/or subtopic, and a north arrow. Magnetic declination on plan maps may or may not be needed.

As indicated previously and shown in Figs. 5.5.3 through 5.5.6, illustrations should be clear and reproducible with sufficient detail included to provide appropriate understanding and with fact clearly discernible from inference. A base reproducible set showing all factual data should be maintained in order to provide easy copies for interpretative work. The interpretative sets, constructed from the base set, are the result of the data analysis process and form the basis for geologic predictions and projections into unsampled areas. Key simplified sets such as mineralization outlines, perhaps isograded, or coal thickness with sulfur content contours will typically be made available to operating personnel for mine planning and operational support. Fig. 5.5.8 is a highly detailed legend, or explanation, commonly found on cross sections illustrating the type of data required in the typical geologic evaluation of a porphyry copper deposit.

Scales of illustration vary as follows: extremely detailed and seldom used 1 in. = 10 ft (1:120) or the more common 1 in. = 20 ft (1:240) or 1 in. = 50 ft (1:600) for initial data compilation in a base metal deposit; the 1 in. = 100 ft (1:1200) or 1 in. = 200 ft (1:2400) for interpretative compilations and mine planning; the 1 in. = 400 ft (1:4800), 1 in. = 500 ft (1:6000), 1 in. = 1000 ft (1:12,000), 1 in. =2000 ft (1:24,000), or an inch equals a mile scale used for summary presentations or district or regional overviews.

Numerous other types of maps and illustrations can be constructed on an as-needed basis to depict or clarify important geologic relationships. Most useful among these are the following: structure contours or isopachs of geologic units or parameters, isogrades of metal values or perhaps dilutents as sulfur or ash content of coals, and ratio maps or combinations of parameters such as grade and thickness, common in uranium work, or total sulfide content. Any geologic parameter that can be quantified can be mapped or illustrated and may provide extremely useful data on trends, distributions, or variability; hence it contributes to interpretation and a better understanding of the geologic conditions associated with a resource. The key points on the use of any of these are accuracy, clarity, simplicity, and reproducibility.

Herness (1977, pp. 529-531) discusses subsurface geologic representation and provides a 19-point list of essentials for effective field and office representation of geologic data that is here abridged and summarized into 10 points as follows:

1. Techniques should be easy to master and rapid to use.
2. All megascopically recognizable features should be capable of being represented.
3. Representation legends should be logical, systematic, chromatic, or geometric sequences, clearly depicting trends.

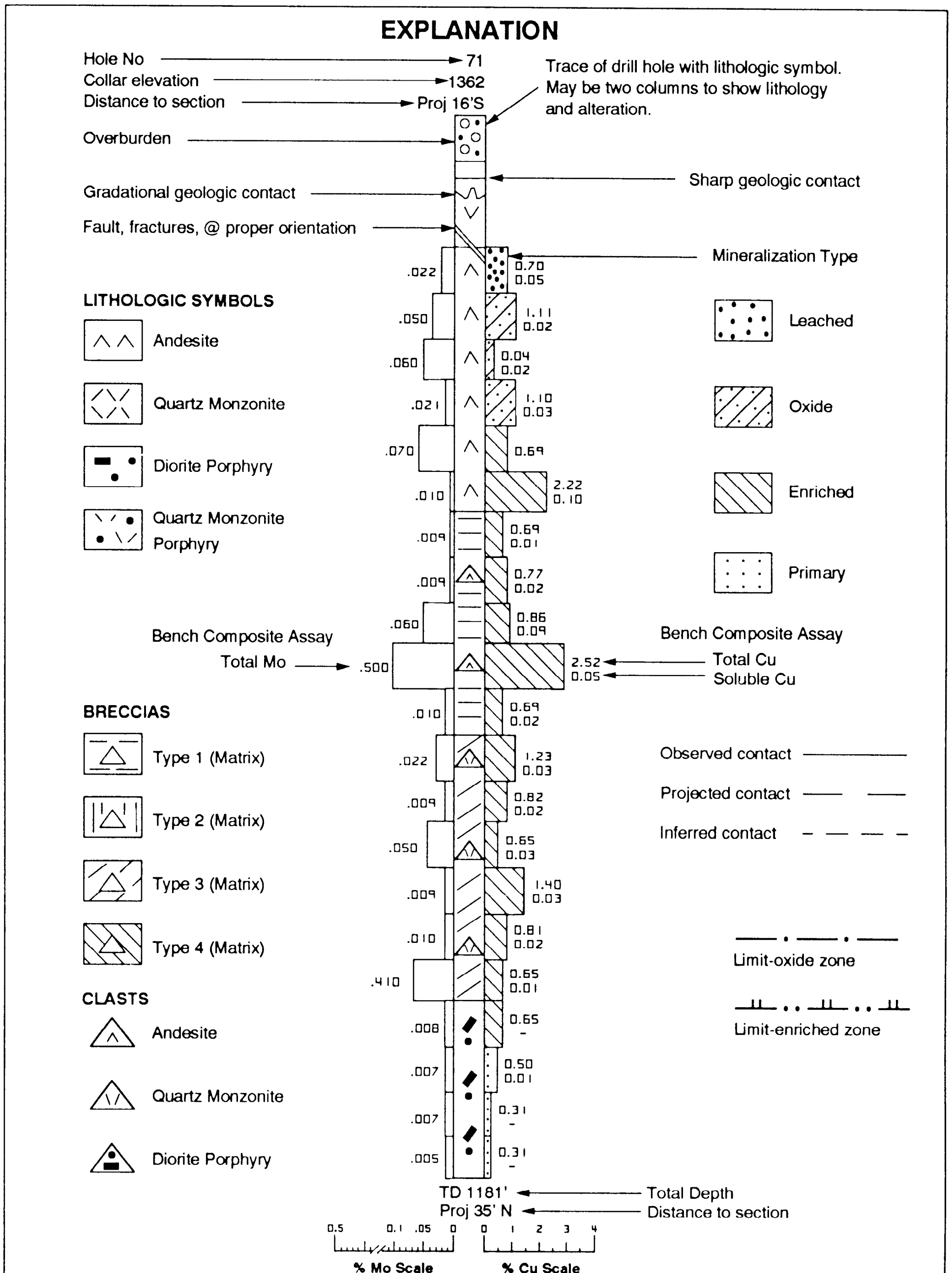


Fig. 5.5.8. Typical legend, or explanation, showing the varied data posted adjacent to each drillhole on cross sections constructed during evaluation of a porphyry copper deposit with associated breccia pipes. Modified and company name removed to ensure confidentiality.

4. Systems should be planned to prevent duplication, unnecessary maps, and map overlap.

5. Ongoing work maps should be easy to revise.

6. Sections and plan maps should be integrated to allow for efficient utilization and prevent data loss in files.

7. Note sheets should be uniform in size, kept in a logical available file, and never discarded.

8. Office maps should not be excessively large to assure ease of handling and prevent damage. (However, a good set of large, highly generalized, attractive display maps should be available for presentations.)

9. Coordinate grids should be parallel to the edges of note and map sheets, and maps should conform to a district-wide grid and not overlap.

10. Maps and note sheets should be clear, neat, and pleasing in appearance. They must have sales appeal because difficult concepts are being portrayed and a project's funding is commonly dependent on clarity of presentation. Impressive, well-executed note sheets and maps generally attest to the quality of the geologic work and of the geologist doing the work.

The variety, types, styles, and purposes of geologic illustration are too varied to allow an example of each. Any of the professional technical geologic journals can be reviewed for style and content. In addition, the following references are particularly good in their content of illustrations: LeRoy et al. (1977), Titley (1982, particularly the included papers by M. T. Einaudi); Dixon (1979), Hutchinson (1983, some are quite complex yet good), Rahmani and Flores (1984), Roberts and Sheahan (1988, very clear and simple, sometimes small), and Barnes (1981).

In summary, there is no substitute for high-quality resource characterization and evaluation based on sound geologic understanding and judgment and presented in an accurate, clear, and lucid manner.

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Chapter 5.6
ORE RESERVE/RESOURCE ESTIMATION

ALAN C. NOBLE

Ore reserve estimates are assessments of the quantity and tenor of a mineral that may be profitably and legally extracted from a mineral deposit through mining and/or mineral beneficiation. Estimation of ore reserves involves not only evaluation of the tonnage and grade of a deposit but also consideration of the technical and legal aspects of mining the deposit, of beneficiating the ores, and of selling the product. Thus a number of professional disciplines may be involved in ore reserve estimation including geology, geostatistics, mining engineering, mineral process engineering, mineral economics, land and legal issues, and environmental engineering.

This chapter, however, addresses only the aspects of ore reserve estimation that include determination of the tonnage, grade, size, shape, and location of mineral deposits. Although these are often referred to as ore reserve estimates, the term resource estimation is used rather than ore reserve estimation to emphasize that all aspects of the ore reserve estimate are not being considered.

5.6.1 RESOURCE ESTIMATION METHODOLOGY

A *resource estimate* is based on prediction of the physical characteristics of a mineral deposit through collection of data, analysis of the data, and modeling the size, shape, and grade of the deposit. Important physical characteristics of the ore body that must be predicted include (1) the size, shape, and continuity of ore zones, (2) the frequency distribution of mineral grade, and (3) the spatial variability of mineral grade. These physical characteristics of the mineral deposit are never completely known, but are inferred from sample data. The sample data consist of one or more of the following:

1. Physical samples taken by drilling, trenching, test pitting, and channel sampling.
2. Measurement of the quantity of mineral in the samples through assaying or other procedures.
3. Direct observations such as geologic mapping and drill core logging.

Estimation of the resource requires analysis and synthesis of these data to develop a resource model. Methods used to develop the resource model may include

1. Compilation of the geologic and assay data into maps, reports, and computer databases.
2. Delineation of the physical limits of the deposit based on geologic interpretation of the mineralization controls at a reasonable range of mining cutoff grades.
3. Compositing of samples into larger units such as mining bench height, seam thickness, or minable vein width.
4. Modeling of the grade distribution based on histograms and cumulative frequency plots of grades.
5. Evaluation of the spatial variability of grade using experimental variograms.
6. Selection of a resource estimation method and estimation of quantity and grade of the mineral resource.

The estimation procedure must be made with at least minimal knowledge of the proposed mining method since different mining methods may affect the size, shape, and/or grade of the potentially minable ore reserve. The most important mining factors for consideration in evaluation of the ore reserve from the resource are

1. The range of likely cutoff grades.
2. The degree of selectivity and the size of the selective mining unit for likely mining methods.
3. Variations in the deposit that affect the ability to mine and/or process the ore.

These mining factors often determine the degree of detail that is required for the resource model and thus the degree of difficulty to develop a resource model for estimating ore reserves. For example, a disseminated gold deposit may be continuous and regular in shape, if mined by bulk, open pit methods. The same deposit may be discontinuous and difficult to estimate, however, if mined by more selective underground methods at a higher cutoff grade. Such large differences in deposit shape due to variations in cutoff grade and mining method may require different ore reserve estimation methods for different mining methods.

5.6.2 DATA COLLECTION AND GEOLOGIC INTERPRETATION

Data that must be collected and compiled for the resource estimate are as follows:

1. Reliable assays from an adequate number of representative samples.
2. Coordinate locations for the sample data.
3. Consistently recorded geologic data that describe the mineralization controls.
4. Cross sections or plan maps with the geologic interpretation of the mineralization controls.
5. Tonnage factors or specific gravities for the various ore and waste rock categories.
6. Surface topographic map, especially for deposits to be surface mined.

Although small deposits may be evaluated manually using data on maps and in reports, the amount of data required for a resource estimate is often large, and data may be more efficiently evaluated if they are entered into a computer database. Computer programs can then be used to retrieve the data for printing reports, plotting on digital plotters, statistical analysis, and resource estimation. Minimum information that should be included in a drillhole database are

1. Drillhole number or other identification.
2. Hole length, collar coordinates, and down-hole surveys.
3. Sample intervals and assay data.
4. Geologic data such as lithology, alteration, oxidation, etc.
5. Geotechnical data such as RQD (rock quality designation).

Entry of data into a computer database is a process that is subject to a high error rate if not carefully controlled and checked. Some procedures that may be used to ensure that the data have been entered correctly are

1. Verification of the data using independent entry by two

persons. This is a standard procedure at many commercial data-entry shops that may dramatically reduce data-entry errors.

2. Manual comparison of a random sample of the original data sheets to a print-out of the database.

3. Scanning the data for outlier values. For example: drill locations outside the project limits, high and low assays, and sample intervals that overlap or are not continuous.

4. Comparison of computer-plotted data with manually plotted maps of the same data. Collar location maps and cross sections are especially useful to rapidly locate inconsistent collar locations and down-hole surveys.

Additional care and attention to detail and accuracy during data entry are essential. A database with a large number of errors may result in a resource estimate that is inaccurate and requires a complete revision to provide defendable results.

5.6.3 GEOLOGIC INTERPRETATION

The sample database represents a large three-dimensional array of point locations in a deposit. The sample data are quantitative and have been subjected to minimal reinterpretation after the original measurements. There is another body of geologic knowledge, however, that does not fit this description. This is the interpretation resulting from the geologist's assimilation of the large quantity of geologic data. These interpretative data are often represented on plan maps or cross sections that show outlines of the extent of geologic features or iso-grade contours that define ore zones. These interpretations combine to provide an interpretative geologic model that is one of the most critical factors in the resource estimation. Failure to develop an appropriate geologic ore body model is the most common reason for large errors in the resource estimates. As shown in Fig. 5.6.1, an inappropriate geologic model may lead to errors greater than an order of magnitude.

The geologist's interpretation of the ore body should be used as much as possible in developing the resource estimate. There are, however, practical limits to the amount of complexity that can be included in the resource model, and the geologic interpretation will be limited to critical inputs that define the shape and trends of the mineral zones at different cutoff grades and the character of the mineral zone contacts.

Examples of geologic features that are often modeled include

1. Receptive vs. nonreceptive host rocks.

2. Alteration types that accompany mineralization or create problems in beneficiation.

3. Faulting, folding, and other structural modifications.

4. Multiple phases of mineralization.

5. Post-mineral features such as oxidation and leaching.

Changes in lithology are often important variables in resource estimation because mineralization can vary due to physical or chemical attributes of the rocks. The differences may be distinct, such as the sharp contact between a skarn ore body and an unmineralized hornfels country rock. They also may be gradational, such as the gradual decrease in grade that is often observed between a favorable and slightly less favorable host in a porphyry copper deposit. Other important lithologic controls include barren post-mineral intrusive rocks, nonreceptive shale beds, and other unmineralized materials that are contained within the mineralized zone.

The effects of faulting will vary according to whether the faulting occurred before or after the mineralization, and to what processes accompanied the faulting. A simple post-ore displacement may create a discontinuity in the ore trends, preventing simple interpolation across the fault. The same type of fault occurring prior to mineralization may have little or no effect on the mineralization or may localize high-grade, vein-type mineralization that must be modeled independently of a more uniform disseminated ore body. It is also important to determine whether the fault is a thin, well-defined structure or many smaller structures in a complex, wide shear zone. In the first case, the fault is modeled as a simple surface with no thickness; in the second, the fault zone must be defined and modeled apart from the adjoining rock units.

Folding is particularly significant in sedimentary and stratabound deposits. Modeling of folding depends on whether folding happened before or after ore deposition, on the tendency of the ore zoning to follow the stratigraphy, on any remobilization that occurred with the folding, and on the creation of traps or other favorable structures. In addition to defining the shape of the folds, it is important to determine whether the mineralization follows the contours of the folds or is independent of the fold geometry.

Multiple phases of mineralization must be defined, particularly where they complicate the ore zoning pattern through overlapping, discordant trends, and through post-mineral oxidation or leaching. Secondary enrichment and oxidation will almost always require delineation of the modified ore zones.

The character of the ore zone contact must be determined and input into the resource model. A sharp contact will be handled as a discontinuity and the data used strictly independently on either side of the contact. A transitional contact, however, is a broad, gradational boundary that may require data selection from zones of tens of feet (meters) to over 100 ft (30 m) to achieve true differentiation between the different grade zones. As a transitional zone becomes thinner, it will eventually approach a sharp contact. For practical purposes, any transitional boundary thinner than the smallest selective mining unit will be modeled as a discontinuity.

In addition to definition of these physical ore controls and post-mineral modifications, a clear understanding of ore genesis will always be beneficial in creating a resource model. In the simplest case, the ore genesis will give clues to the behavior of the grade distributions and variograms; in other cases, the genetic structure is so dominant that it can be used as a direct control in the estimation of mineral resources.

5.6.4 COMPOSITING

Compositing is a procedure in which sample assay data are combined by computing a weighted average over longer intervals to provide a smaller number of data with greater length for use in developing the resource estimate. Compositing is usually a length-weighted average. If density is extremely variable (e.g., massive sulfides), however, compositing must be weighted by length times density (or specific gravity).

Some of the reasons for and benefits of compositing include

1. Irregular length assay samples must be composited to provide equal-sized data for geostatistical analysis.

2. Compositing reduces the number of data and may significantly reduce computational time, which is often proportional to the square of the number of data.

3. Compositing incorporates dilution such as that from mining constant height benches in an open-pit mine or from mining a minimum height/width in an underground mine.

4. Compositing reduces erratic variation due to a high nugget effect caused by erratic high-grade values.

There are several different methods for compositing that may be used depending on the nature of the mineralization and

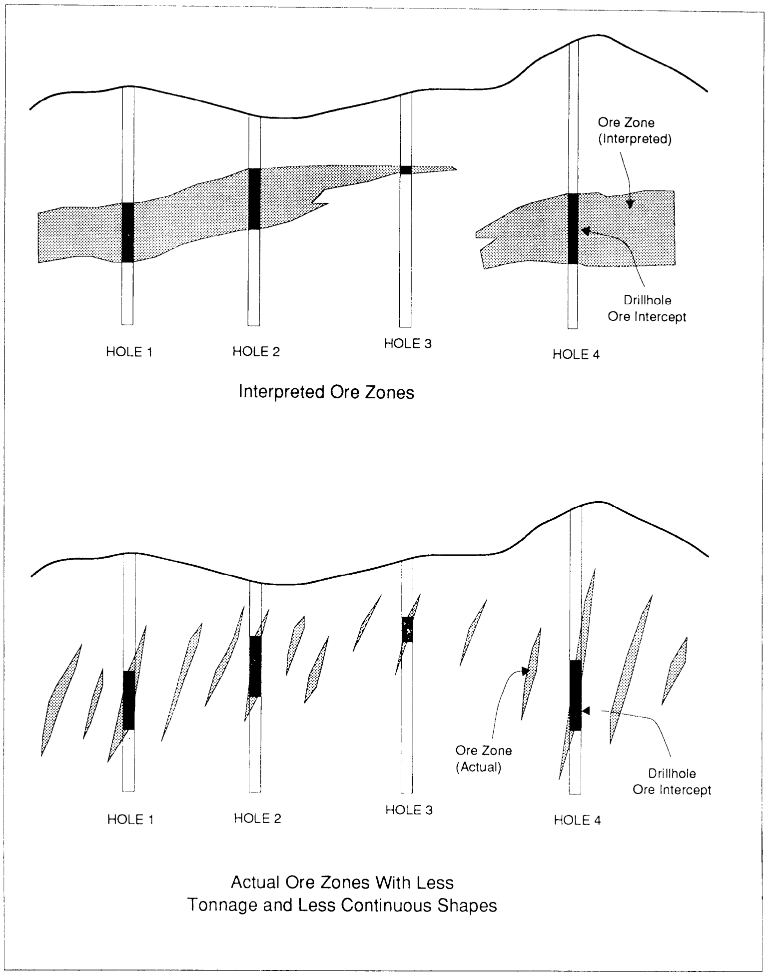


Fig. 5.6.1. Overestimation of ore reserves based on a geologic model that is less continuous than the actual ore zones.

the type of mining. Common compositing methods are (1) bench compositing, (2) constant length compositing, and (3) ore zone compositing.

Bench compositing is a method often used for resource modeling for open pit mining and is most useful for large, uniform deposits. Composite intervals for bench compositing are chosen at the crest and toe of the mining benches. Bench compositing has the advantage of providing constant elevation data that are simple to plot and interpret on plan maps. In addition, the dilution from mining a constant-height, constant-elevation bench is approximated by the bench composite.

Down-hole composites are computed using constant length intervals starting from the collar of the drillhole or the top of the first assayed interval. Down-hole composites are used when the holes are drilled at oblique angles (45° or less) to the mining benches, and bench composites would be excessively long. Down-hole composites should also be used when the length of the sample interval is greater than one-third the length of the composite interval to prevent overdilution when the sum of the lengths of the samples is much greater than the length of the composite.

Ore-zone compositing is a method of compositing that is used to prevent dilution of the composite when the width of the contact between waste and ore (or low grade and high grade) is less than the length of a composite. Use of bench compositing or down-hole compositing in this case may distort the grade distributions by adding low grade to the ore population and high grade to the waste population, resulting in underestimation of ore grade and overestimation of waste grades.

Ore-zone composites are computed by first identifying the interval containing each ore zone in the drillhole. Each ore zone is then composited individually as follows: (1) the length of the ore zone is divided by the desired length of the composite; (2) this ratio is rounded up and down to determine the number of composites that provide a length nearest the desired length when divided into the length of the ore zone; and (3) the ore zone is composited using length composites starting at the beginning of the ore zone and length as determined in the previous step.

A special case of ore-zone compositing is encountered in a vein or bedded deposit in that the width of the ore zone is determined by a combination of minimum mining thickness (height) and assay limits. In these situations, composites must be recomputed for each combination of assay cutoff grade and minimum mining thickness.

Geologic codes are usually assigned to composites according to the rock type, ore zone, or other geologic feature. This is often a simple procedure, since most composites will be computed from samples taken from a single geologic unit. Assignment of geologic codes to composites that cross geologic contacts is more complex, since the composite will be computed using data from multiple geologic units.

If the geologic contact is transitional and does not separate contrasting grade distributions, it is appropriate to assign the geologic codes according to the majority rule. If the composite crosses a sharp boundary between contrasting grade distributions, it is best to use geologic unit compositing or to assign the composite to the geologic unit with the most similar grade.

If some sample intervals in the data are missing assays, it is important to determine the reason for the missing data and account for it appropriately. Typical examples are

1. The missing zone was not assayed because it was low grade or barren by visual inspection, or the sample was missing because of poor core recovery in a barren zone.

Action: Composite using the average of the barren unit or zero grade for the grade of the missing assay.

2. The sample was missing because of poor core recovery in a narrow post-mineral fault.

Action: Ignore the missing interval when computing composites. The volume of the fault zone is small and the grade will be similar to the grades in the country rock.

3. The sample was missing because of poor core recovery in a vein that is higher grade and less competent than the surrounding country rock.

Action: Ignore the missing interval when computing composites, but retain the length of the interval for use in estimating the grade of the vein.

5.6.5 BASIC STATISTICS AND GRADE DISTRIBUTION

Computation of basic statistics and evaluation of grade distributions are the first quantitative analyses of the grade data and are basic tools to provide both feedback to the geologic analysis and input to the resource modeling. Important factors in these basic studies include

1. Detection of high-grade or low-grade outlier values.
2. Evaluation of the favorability of different lithologies as host rocks.
3. Differentiation of complex grade distributions into simple populations for resource modeling.
4. Identification of highly skewed and/or highly variable grade distributions that will be difficult to estimate.

Basic statistics should be computed for sample and/or composite grades in each geologic domain that is suspected to have different characteristics. This may include different lithologies, alteration types, structural domains, grade zones, or other grouping of data that has been recognized (or suspected) to have different grade distributions. Statistics that should be compiled include

1. Number of data (samples or composites).
2. Average grade, thickness, etc. (mean)
3. Standard deviation (std. dev.) and/or variance.
4. Coefficient of variation (COV), the standard deviation divided by average grade.
5. Histogram of grades.
6. Cumulative frequency distribution (probability plot).

The first item reviewed is the number of data; generally, at least 25 data are required to make comparisons between different geologic domains. If sufficient data are available, average grades and coefficients of variation will be compared among the various geologic domains. General rules for evaluating differences in average grade are as follows

Grade Difference	Interpretation
0% to 25%	Grade populations that do not usually require differentiation for resource modeling.
25% to 100%	Grade populations that require differentiation for resource modeling if divided by a discontinuity such as a fault, or if variograms or grade trends are dissimilar.
Above 100%	Grade distributions must be separated for modeling. Differences of 1000% or more may be observed when barren, mineralized, and/or high-grade populations are present.

Rules for analyzing coefficient of variation are as follows

COV	Interpretation
0% to 25%	Simple, symmetrical grade distribution. Resource estimation is easy, many methods will work.
25% to 100%	Skewed distributions with moderate difficulty in resource estimation. Distributions are typically lognormal.
100% to 200%	Highly skewed distributions with a large grade range. Difficulty in estimating local resources is indicated.
Above 200%	Highly erratic, skewed data or multiple populations. Local grades are difficult or impossible to estimate.

Distributions with COV greater than 25% often have a lognormal grade distribution, and the basic statistics will also be compiled for the natural logarithms of grades. For a perfectly lognormal distribution, the lognormal statistics are related to the normal statistics as follows:

$$\text{mean} = e^{(\alpha + \beta^2/2)} \tag{5.6.1}$$

$$\text{COV} = (e^{\beta^2} - 1)^{0.5} \tag{5.6.2}$$

$$\text{std. dev.} = \text{mean} \times \text{COV} = e^{(\alpha + \beta^2/2)} \times (e^{\beta^2} - 1)^{0.5} \tag{5.6.3}$$

where a is the average of the natural logarithms of grades; and b is the standard deviation of the natural logarithms of grades.

Close agreement between the mean, standard deviation, and coefficient of variation when estimated using both normal and lognormal statistics is indicative of a lognormal population and is required to use lognormal statistics.

5.6.6 GRADE DISTRIBUTION

The grade histogram and cumulative frequency distribution are used to study the relationship between the statistical grade distribution and geologic parameters. The analysis is usually begun with a histogram of sample or composite grades. If the histogram is bell-shaped and symmetrical, a normal distribution is indicated, and the cumulative frequency will be plotted on normal probability paper. Normal distributions are not usually found in mineral deposits except those with sedimentary origins.

If the histogram is skewed to the right, a lognormal distribution is indicated and the cumulative frequency distribution will be plotted on lognormal probability paper. Lognormal distributions are frequently observed in most hydrothermal precious and base metal deposits.

Normal probability paper is a special graph paper in that the y-axis is a cutoff grade and the x-axis is the percentage of samples above (or below) the cutoff grade. The x-axis is scaled such that a normal distribution will plot as a straight line, the

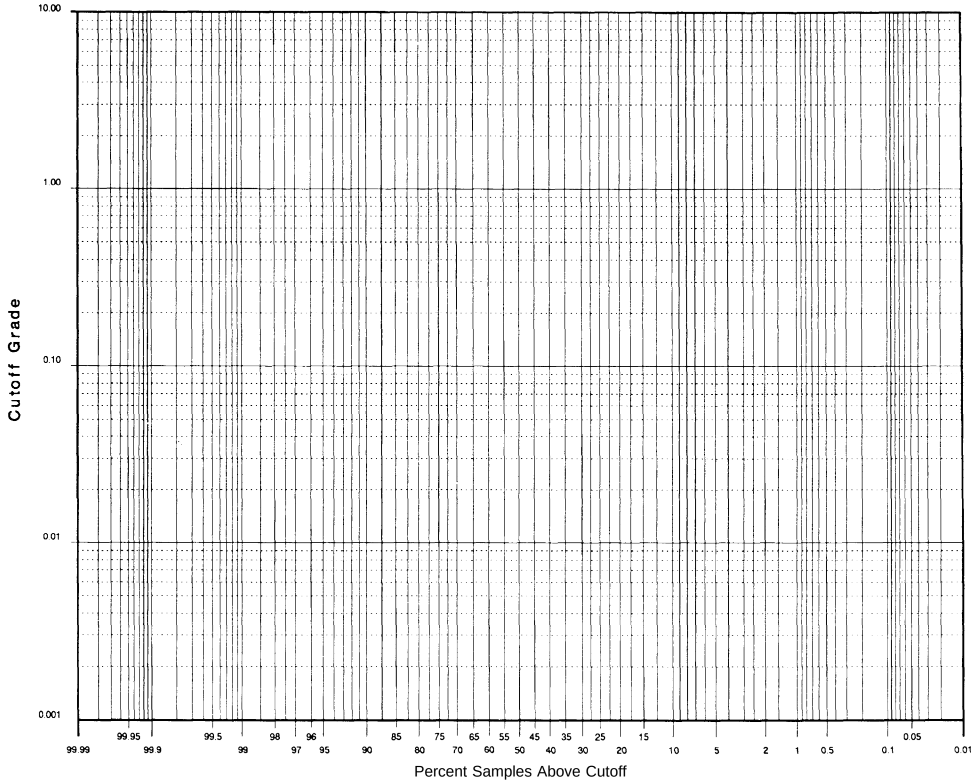


Fig. 5.6.2. Lognormal probability plotting paper.

slope of the line is proportional to the standard deviation of the distribution, and the 50th percentile is the average grade.

Lognormal probability paper is similar to normal probability paper except that the y-axis is scaled according to the logarithm of cutoff grade. The slope of the line is proportional to the standard deviation of logarithms of grade *b*, and the 50th percentile is the average of the logs of grades *a*. An example of lognorma1 probability paper is shown in Fig. 5.6.2.

The probability graph may be used to estimate the standard deviation based on probabilities from the normal probability distribution as follows:

$$\beta = [\ln(g_{97.7\%}) - \ln(g_{2.3\%})] \div 4 \qquad (5.6.4)$$

which is based on 2 standard deviations, or

$$\beta = [\ln(g_{98.0\%}) - \ln(g_{2.0\%})] \div 4.1 \qquad (5.6.5)$$

which is based on ± 2.05 standard deviations.

Often the probability graph is not a straight line, but will be composed of multiple straight lines or curves. A typical deviation from a straight line is a downward curve at the low end of the graph as shown in Fig. 5.6.3. This curve represents excess low-grade samples, and in porphyry systems is often attributed to weakly mineralized late intrusions or to post-mineral, barren dikes. On low coefficient of variation deposits, this type of graph may also represent a normal distribution that has been plotted on lognormal probability paper. The data should be examined to determine the source of the low-grade material and to determine whether that population has been or can be mapped geologically and estimated separately.

Another common deviation from a straight line on the probability plot is a steeper slope at the upper end of the curve as shown in Fig. 5.6.4. This represents excess material in the high-grade population and may be caused by two superimposed populations, such as high-grade veins within lower-grade disseminated or stockwork mineralization. Other causes of excess high-grade assays include small zones that are highly favorable to mineralization because of higher permeability, favorable chemical properties, secondary enrichment, or metamorphic remobilization. Since the high-grade mineralization usually has less continuity than the lower-grade mineralization, the source of the high grade must usually be identified and estimated separately from the remaining mineralization.

5.6.7 VARIOGRAM MODELING

The *variogram* is the fundamental tool used by the geostatistician and geologist to measure spatial continuity of grade data. The variogram is a graph of the average variability between samples vs. the distance between samples. A variogram is computed by averaging the squared differences between pairs of samples that are a given distance apart as follows

$$\gamma(h) = \text{variogram for distance } h$$

$$= \frac{1}{2N} \sum (g_i - g_{i+h})^2 i = 1,2,3 \ldots N \qquad (5.6.6)$$

where N is the number of pairs at distance *h*, and *h* is the distance between the samples.

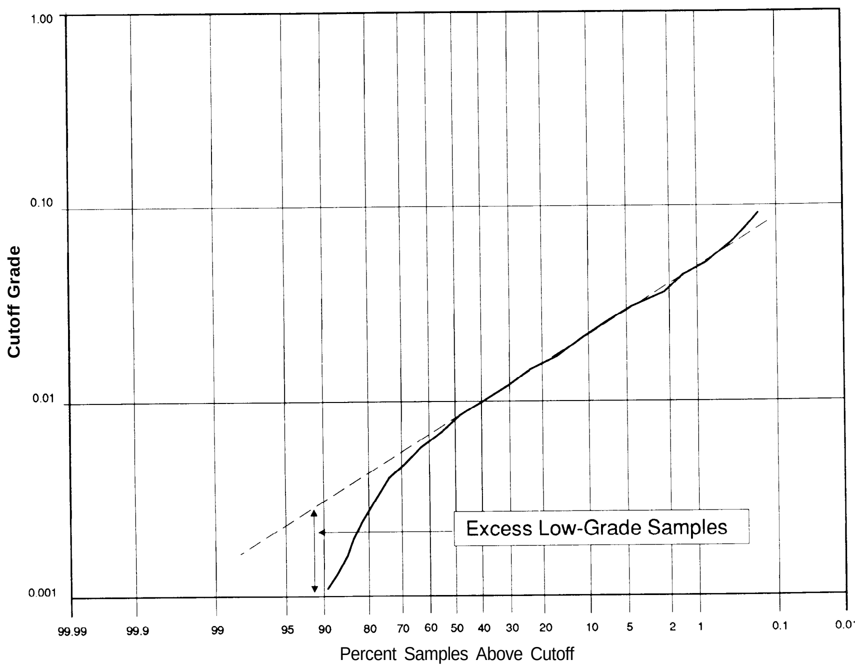


Fig. 5.6.3. Deviation from a lognormal distribution that is caused by excess low-grade samples.

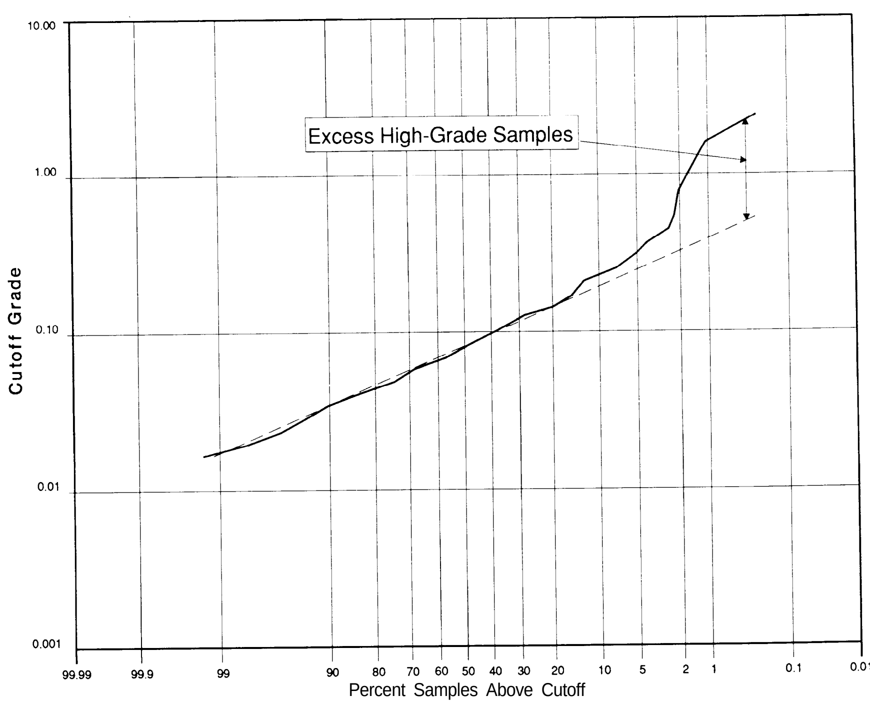


Fig. 5.6.4. Deviation from a lognormal distribution that is caused by excess high-grade samples.

The variogram function $\gamma(h)$ is computed for a number of different sample distances, to provide an experimental variogram that typically looks like the graph in Fig. 5.6.5. The most important features of the variogram are the nugget, range, and sill. The nugget value is identified as the y-intercept of the variogram curve and represents random and short-distance variability factors such as sampling error, assaying error, and erratic mineralization. High nugget values are commonly found in ore bodies where short distance variability is extremely high, where accurate sampling and assaying of ore is difficult, or where poor sampling and assaying techniques are employed. High nugget effects are found in many gold deposits because of random gold nuggets that cause large grade changes over small distances. Similar high nugget values are often found in molybdenum deposits; these are caused by small pockets of pure molybdenite in a disseminated or stockwork mineralization.

Small nugget values indicate an ideal situation reflecting good sampling techniques and locally continuous mineralization.

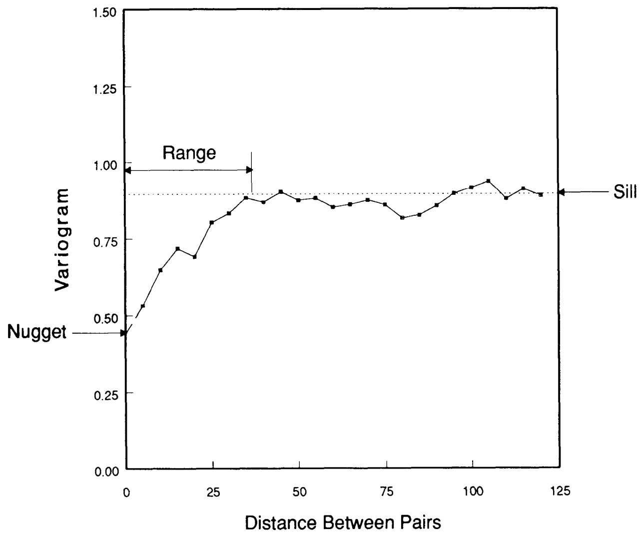


Fig. 5.6.5. Typical experimental variogram plot.

A small nugget value on a variogram confirms that the assays can be reliably used for geologic interpretation and resource estimation. Low nugget values are typically found in many types of deposits, including hypogene porphyry copper, iron ore, and coal. High nugget values have also been found for each of these types of deposits so each deposit must be analyzed individually.

Most variograms increase in value from the nugget for some distance and then level off to a constant value. This distance is called the range of the variogram, and the variogram value is called the sill. The range is equivalent to the geologist's concept of range of influence, that is, the distance beyond which samples are not correlated with other samples and beyond which grade trends should not be projected. The sill value is usually equal to the sample variance. If the sill is higher or lower than the variance, zonal effects or multiple grade distributions are usually indicated.

The slope and shape of the variogram often vary in different directions, with the range increasing in the direction of greatest continuity of the mineralization. This behavior is referred to as a geometric anisotropy.

5.6.7.1 Computing an Experimental Variogram

Computing an experimental variogram from a set of randomly spaced data involves finding pairs of data that are oriented in the required direction, determining the distance between the samples, then summing the squared differences of the grades. Since the data are usually sparse, it is necessary to use a tolerance when locating samples in the desired direction and to use a distance increment to classify samples by distance. The directional tolerance is usually achieved with a window angle, or a fixed distance, as shown in Fig. 5.6.6. These methods may be combined and/or generalized into three dimensions as shown in Fig. 5.6.7. The distance tolerance is a fixed distance increment (cell size), selected so a reasonable number of samples fall in each cell. Some guidelines to aid in computing experimental variograms are

1. Variograms must be computed within continuous zones of mineralization. Do not cross contacts between different geologic domains.

2. The maximum distance used should be less than one-half the length of the mineralized zone in the direction of the variogram.

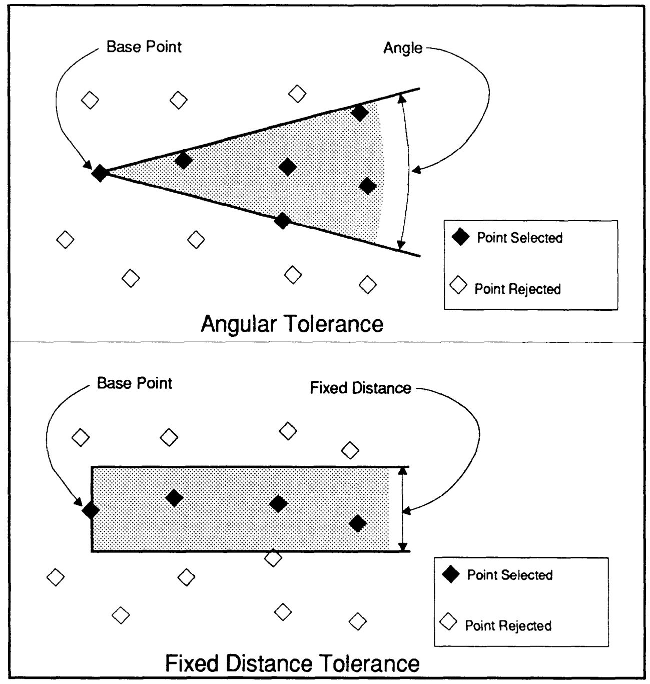


Fig. 5.6.6. Angular and fixed-distance tolerance methods for selecting variogram pairs.

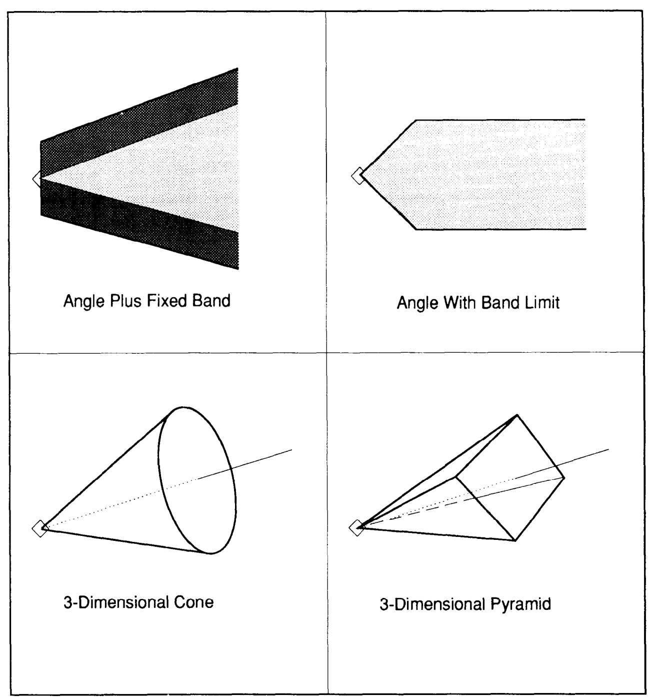


Fig. 5.6.7. Composite and three-dimensional methods for selecting variogram pairs.

3. The maximum search distance perpendicular to the direction of the variogram must be less than one-half the range of the variogram in the perpendicular direction.

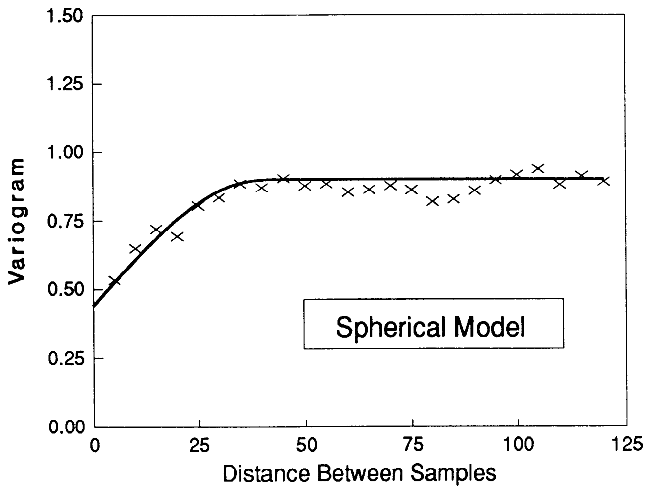


Fig. 5.6.8. Experimental variogram modeled with a spherical variogram model.

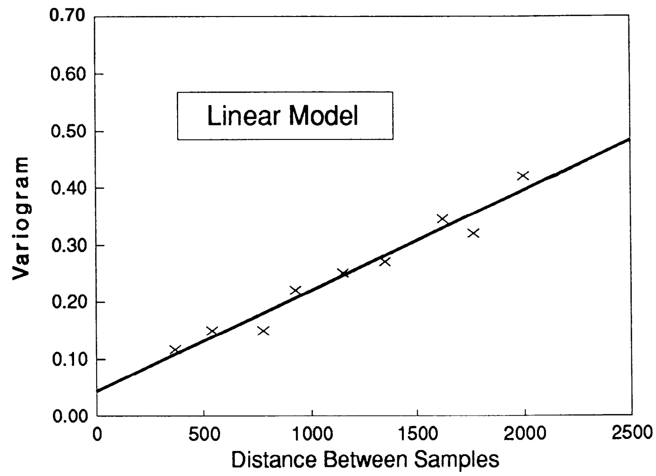


Fig. 5.6.9. Experimental variogram modeled with a linear variogram model.

4. The distance increment should be approximately equal to the average spacing between samples in the direction of the variogram.

5. At least 30 pairs of samples are required to compute a valid variogram. More pairs produce a more stable variogram.

6. All samples must be the same size and should be obtained by the same or similar sampling methods.

7. Data should be declustered before computing the variogram. In particular, a few twin holes may give a misleading impression of the nugget effect.

A model, or equation, is fitted to the experimental variogram for further geostatistical evaluations such as kriging. The most common variogram model is the spherical model shown in Fig. 5.6.8. This model has the equation

$$\gamma(h) = C_0 + C\left[\frac{3}{2}\cdot\frac{h}{a} - \frac{1}{2}\left(\frac{h^3}{a}\right)\right] \quad \text{for } h \le a \qquad (5.6.7)$$

$$\gamma(h) = C_0 + C \quad \text{for } h \ge a \qquad (5.6.8)$$

where C_0 is the nugget, C is the sill, and a is the range.

A spherical variogram model may be constructed graphically by drawing a horizontal line at the variogram value equal to the variance of the samples. This value is equal to nugget plus the sill C_0 + C. A line is drawn through the points at the short-distance end of the curve. The nugget C_0 is estimated where the line intersects the Y-axis, and the range a is estimated as 1.5 times the distance where the line intersects the variance.

Other variogram models used in resource estimation include the exponential, linear, hole effect, and various combinations of "nested" structures. Examples of some of these variograms are shown in Figs. 5.6.9 to 5.6.11.

5.6.7.2 Relative Variograms

Lognormally distributed data often exhibit a proportional effect where the standard deviation of grades increases with grade. This results in variograms with higher values in high-grade areas than in low-grade areas. This may be corrected by dividing each cell in the experimental variogram by the square of the mean of the samples that were used in the variogram

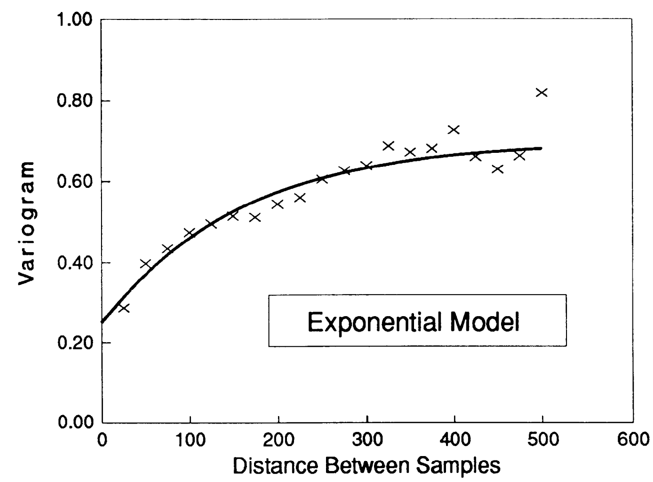


Fig. 5.6.10. Experimental variogram modeled with an exponential variogram model.

for that cell. The resulting variogram is known as a *relative variogram.*

5.6.7.3 Lognormal Variograms

If data are clearly lognormal, a variogram may be computed using the logarithms of sample grades. The resulting *lognormal variogram* is often less erratic and more easily interpreted than the variogram of untransformed values. This variogram may be used directly for lognormal geostatistics or may be transformed to a relative variogram as follows:

$$\sigma_R^2 = (e^{\beta^2} - 1) \qquad (5.6.9)$$

$$\gamma_R\,(\mathrm{h}) = \sigma_R^2 - e^{(\beta^2 - \gamma_{\ln}(h))} + 1 \qquad (5.6.10)$$

Caution must be exercised when using the lognormal variogram since small deviations from lognormality may have large effects on the transformation to a relative variogram.

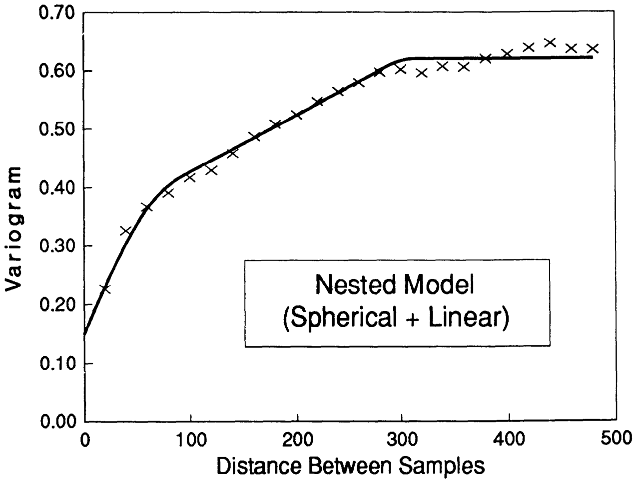
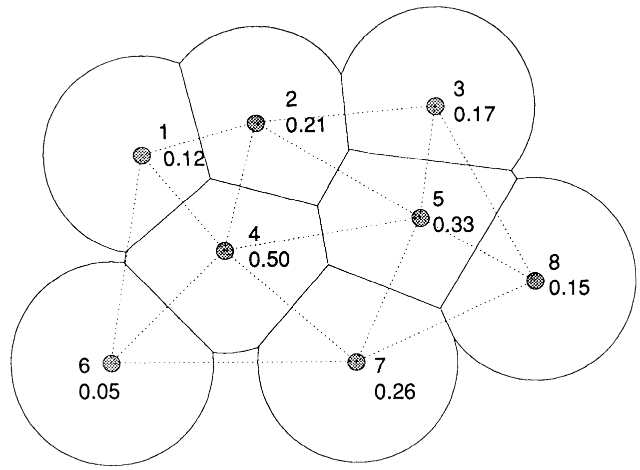


Fig. 5.6.11. Experimental variogram modeled with nested spherical and linear variogram models.



Hole	Grade	Area
1	0.12	39.4
2	0.21	37.6
3	0.17	42.0
4	0.50	37.7
5	0.33	33.8
6	0.05	50.1
7	0.26	46.8
8	0.15	46.3

Total Area = 333.7

Total Area x Grade = 71.39

Average Grade = 0.2139

Fig. 5.6.12. Computation of an estimate using the polygonal method.

5.6.8 RESOURCE ESTIMATION (MODELING)

Methods for resource estimation or modeling are generally divided into (1) traditional, geometric methods that are done manually on plans or sections and (2) interpolation methods such as inverse-distance-weighting and kriging that require the use of a computer.

5.6.8.1 Geometric Methods

Manual resource estimations are usually done on plan maps or cross-section maps that cut the deposit into sets of parallel slices. Data plotted on the maps include drillhole locations, assay values, and the geologic interpretation of the mineralization controls. The *geometric methods* used are based on geometric weighting of assays and include area averaging, polygonal, cross sectional, and triangular.

Area Averaging: The *area-averaging method* is among the simplest of all reserve estimation methods, involving only a geologic interpretation of the shape of the ore and averaging of the grades within that shape as follows:

1. Draw the outline of the ore body on each map; these are the ore blocks and may be regular or irregular shapes. If several ore zones or ore types are present, each is drawn individually.
2. Measure the area of each ore block (usually by planimetering). Multiply the area times the thickness of the ore and divide the resulting volume (cubic feet) by the tonnage factor (cubic feet per ton) to compute tons of ore; in SI units, multiply the volume (cubic meters) by the density (tonnes per cubic meter) to compute the tonnes of ore.
3. Compute the average grade of samples within each block.
4. Calculate the sum of the tonnage in the individual blocks. Average grade is the tonnage-weighted average grade of the individual blocks.

Despite its simplicity, the area-averaging method provides excellent estimates where the drilling pattern is uniform, grades are continuous, and ore boundaries are distinct and sharp. Problems may arise, however, when the drill pattern is nonuniform. With a nonuniform drill pattern, a cluster of holes in a high-grade zone will cause overestimation of grade. Area-averaging methods also may be difficult to implement on deposits with discontinuous or spotty ore zones, especially if the ore contacts are gradational, and multiple cutoff grades are desired.

Polygonal and Cross-sectional Methods: *Polygonal* and *cross-sectional methods* are related methods in that each ore interval is assigned its own polygon of influence. Tonnage and grade is then computed using the same procedure as was used for the area-average method, except that the areas used to compute tonnage are the area of each individual polygon. Polygons are drawn on plan maps based on the perpendicular bisectors of the line between each drillhole as shown in Fig. 5.6.12. The size and shape of the polygons may be limited, if desired, by a maximum distance from each hole. On cross sections, the polygons are usually drawn one-half the distance from each drillhole as shown in Fig. 5.6.13. The distance from a drillhole may also be limited to a maximum distance in the cross-sectional method.

A computer approximation of the polygonal method is the nearest neighbor estimation. This method requires superposition of a rectangular grid of blocks over the drilled area as shown in Fig. 5.6.14. The grade of the nearest sample is then assigned to each block. This method will closely approximate the polygonal method if the block size is no more than 25% of the average drillhole spacing.

Triangular Method: The *triangular method* is similar to the polygonal method except that areas of triangles are estimated, and the grade of each triangle is based on the average of the grades at each of the corners of the triangles as shown in Fig. 5.6.15.

The geometric methods all have the advantage of simplicity and ease of implementation. In addition, they will provide an unbiased estimate of the average grade of a deposit at a zero cutoff grade. A resource estimate using a geometric method provides a quick, inexpensive check to verify nonbias of a more complicated, computer-generated resource model.

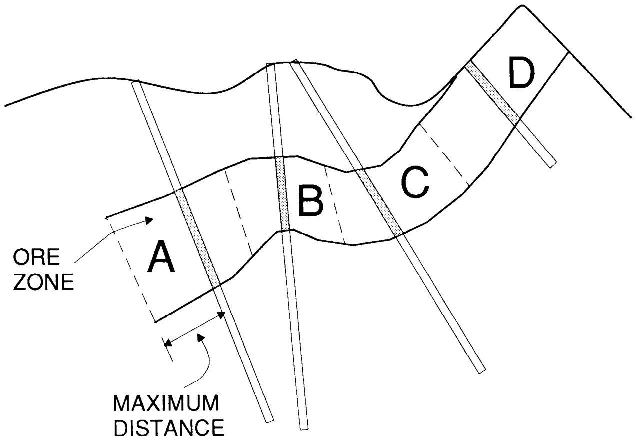
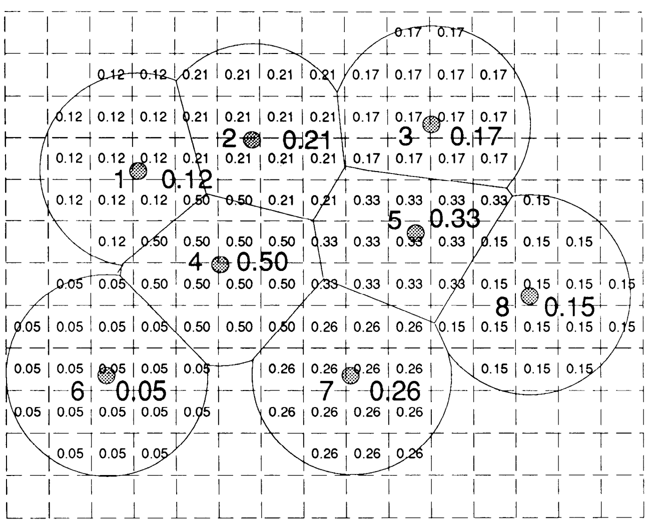


Fig. 5.6.13. Computation of an estimate using the cross-sectional method.



Hole	Grade	# Blocks
1	0.12	12
2	0.21	14
3	0.17	14
4	0.50	13
5	0.33	12
6	0.05	19
7	0.26	14
8	0.15	16

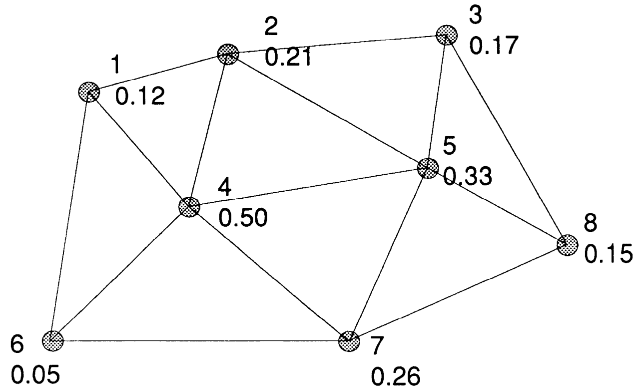
Total Blocks = 114

Sum Blocks × Grade = 24.21

Average Grade = 0.2124

Fig. 5.6.14. Computation of an estimate using the nearest-neighbor method.

The most common problem with geometric methods is that they may imply more selective mining than may be achieved by the mining method. This results from estimating the resource from samples the size of a drillhole but mining larger, less selective volumes. High-grade blocks usually include lower-grade material when they are mined, and low-grade blocks usually include



Triangle	Grade	Area
124	.277	14.5
146	.223	21.6
235	.237	21.1
245	.346	26.2
358	.217	14.9
457	.363	28.2
467	.270	29.7
578	.247	22.3

Total Area = 178.5

Total Area x Grade = 49.9

Average Grade = 0.280

Fig. 5.6.15. Computation of an estimate using the triangular method.

some higher-grade material. The resulting mined grades are different from the predicted distribution; for cutoff grades below the average grade of the deposit, the mined grade will be lower and the tons will be higher. If the cutoff grade is significantly higher than the average grade of the deposit, however, both the mined grade and tons can be lower, resulting in a severe overestimation of contained metal.

5.6.8.2 Moving Average Methods

The *moving average methods,* inverse-distance weighting and kriging, are the most widely used procedures for computer-assisted resource estimation. The basic procedure for both of these methods is as follows

1. Divide the ore body into a matrix of rectangular blocks as shown in Fig. 5.6.16.
2. If geologic controls are present and will be used to control or modify grade assignment, a geologic code must be assigned to each block.
3. Estimate the grade of each block by searching the database for the samples surrounding each block and computing the weighted average of those samples. The weighted average is computed using the following equation:

$$g^* = \Sigma w_i g_i \qquad i = 1,2,3,...n \tag{5.6.11}$$

where g* is the estimated grade, g_i is the grade of sample i, w_i is the weight given to sample i, and n is the number of samples selected.

5.6.8.3 Practical Considerations for Moving Average Methods

The determination of the block size, anisotropies, and the sample selection criteria are common considerations for either

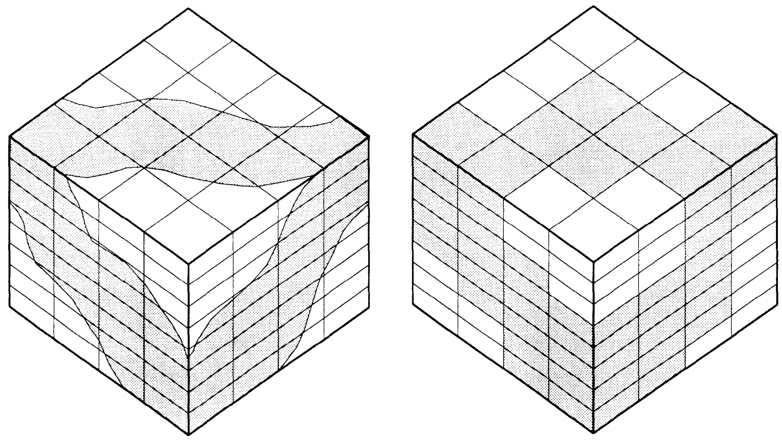


Fig. 5.6.16. Example of a geologic unit coded into a block model.

kriging or inverse distance estimation. These factors are often the most critical factors in developing a satisfactory resource model since the geologic controls are introduced via these mechanisms. Poor selection of these parameters commonly results in an unsatisfactory resource model with significant overestimation or underestimation of ore tonnage or grade.

Block Size: Factors that must be considered in determining block size include the size of the resulting model, drillhole spacing, mining method, and geologic controls. General rules for block size determination are

1. The largest block size possible should be used to minimize the size of the block model and reduce computational time and disk storage requirements.

2. The block is normally one-half to one-fourth the average drillhole spacing. A bigger block may be used if the drilling is extremely dense. Smaller block sizes provide minimal improvement in the estimation unless strong geologic controls are present.

3. The block size must be at least one-half the size of the smallest geologic feature that will be modeled. Larger blocks will destroy the location and/or size of small features.

4. Block size may be related to a proposed mining method. The block height is usually the same as bench height in a deposit that will be mined by open pit methods.

5. Most commercial software packages for resource estimation allow each dimension of the block to be a different size and may allow rotation of the entire model to an orientation other than north-south.

6. These rules are often contradictory, and the best solution will be a compromise that will vary on a case-by-case basis.

Anisotropies: Strong anisotropies, or trends, are often observed in mineral deposits that have ore zones with greater continuity in favorable orientations, commonly shown by variograms with longer ranges in certain directions.

Sample Selection: The purpose of the sample selection step is to provide a subset of the data that is representative of the region around the block. Weighted moving average methods may be very sensitive to sample selection and time spent in analyzing the sample search pattern and how it relates to the data is essential. Some rules to assist in defining a sample selection search are

1. Samples must be selected from geologic domains similar to that of the block.

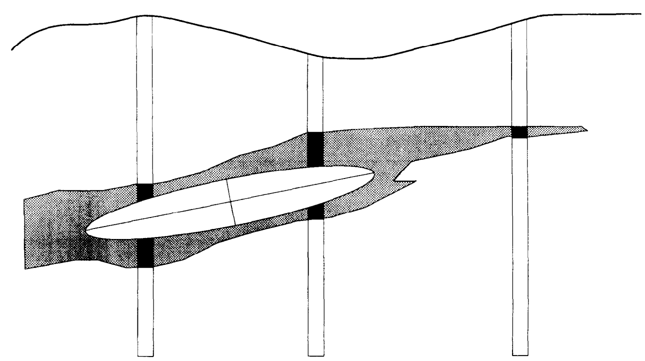


Fig. 5.6.17. Use of an oriented search ellipse to approximate the structure of an ore zone.

2. The maximum radius should be at least equal to the distance between samples to prevent discontinuities in the weighted average as samples drop in and out. (For a square grid, the maximum radius is the diagonal.)

3. The maximum number of samples is usually on the order of 10 to 20. More than 20 samples rarely improves the estimate; fewer than 10 samples may cause discontinuities in the estimated grades.

4. A minimum distance to the nearest sample may be used to prevent excessive extrapolation.

5. A search ellipse or other anisotropic pattern may be used to align the search with trends in the ore as shown in Fig. 5.6.17. The axes of the search ellipse should be oriented parallel to grade trends. The length of the ellipse axes should be proportional to the range of continuity in the respective directions. The variogram ranges and visual appraisal of the grade zones on plans and sections are both used as guides to determining the orientation and length of the search axes.

6. Three composites are usually the maximum required from a single drillhole. More than three provides redundant data and may cause strange kriging weights.

7. Search patterns may be modified to select data with quadrants or other geometric limit as shown in Fig. 5.6.18. Use of a quadrant search will improve estimations if data are clustered.

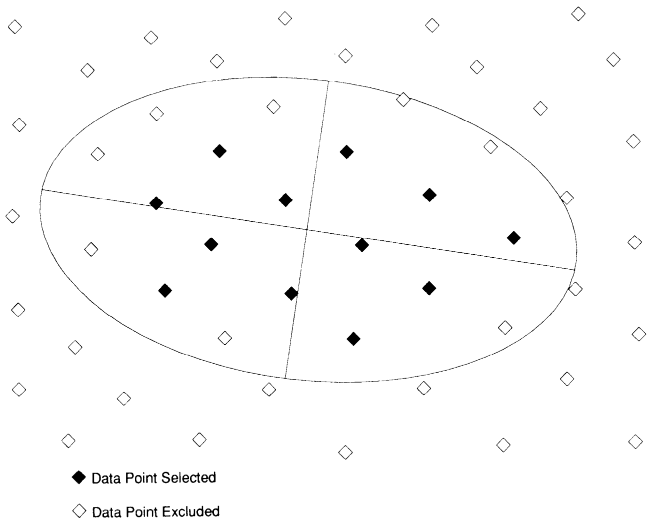


Fig. 5.6.18. Selection of data using a quadrant search and an elliptical search pattern.

8. Sketch the search pattern on plan maps and/or cross-sectional maps in both well-drilled and sparsely drilled areas. This will aid in visualization and assure that the search pattern is appropriate.

5.6.8.4 Inverse Distance Estimation

Inverse distance weighting, one of the earliest used interpolation methods, is based on an empirical observation that the weight of each sample in Eq. 5.6.11 is proportional to an inverse power of the distance from the location of the estimate to the sample. The inverse distance estimate is thus a weighted average with the individual weights computed as an inverse power of distance as follows:

$$w_i = \frac{d_i^{-\text{power}}}{\Sigma d_i^{-\text{power}}} \quad i = 1 \ldots \text{number of samples} \qquad (5.6.12)$$

where w_i is the weight computed for each sample i, each d_i is the distance between the location being estimated and sample i, and power is the inverse distance weighting power. Care must be taken with an inverse distance estimate to ensure that none of the distances d_i are very small or equal to zero, resulting in division by zero, or floating point overflow. This problem may be alleviated either by adding a small constant to each distance or by assigning the value of the closest point to the estimate if the distance is less than some threshold.

The degree of smoothing, or variance reduction, in the inverse distance estimates may be controlled by changing the weighting power and search parameters. A lower power, large search radius, and/or greater number of points used in the estimation result in more continuous estimations and a greater reduction in the variance of the estimated values. Higher weighting powers, smaller search radius, and/or fewer points result in less reduction in the variance of the estimated values and more erratic appearance of the estimated values. The appropriate parameters may only be found through trial and error and experience to achieve the desired distribution of the estimated values and trends, or contours, to match with production results, or to produce trends in the estimated values that visually match with the empirical expectations of the geologist.

5.6.8.5 Kriging

Kriging is the geostatistical estimation method developed to provide the "best linear, unbiased estimate" for grade based on a least squares minimization of the error of estimation, or kriging error. Kriging and its variants have had much theoretical development and are well described in the geostatistical literature. For a more detailed discussion of kriging, the reader may refer to David (1977, 1988) or Journel and Huijbregts (1978). Important factors in the kriging estimate are

1. The average of the estimates should not be systematically higher or lower than the true value; this is established mathematically by setting the sum of weights equal to zero, that is,

$$\Sigma w_i = 1 \qquad (5.6.13)$$

2. The error of estimation s_K^2, which is expressed as variance $(G - G^*)$, or

$$\sigma_K^2 = \sigma_{B,D}^2 - 2\,\Sigma w_i \sigma_{B,Xi} + \Sigma\Sigma w_i w_j \sigma_{Xi,Xj} \qquad (5.6.14)$$

is minimized, where G is the actual grade, G^* is the estimated grade, $s_{B,D}^2$ is the variance of blocks the size being estimated in the deposit, the $s_{B,Xi}$ are the covariances between each sample and the block being estimated, and the $s_{Xi,Xj}$ are the covariances between the individual samples.

The kriging variance in Eq. 5.6.14 is minimized using the Lagrange principle to create a modified equation that satisfies the nonbias constraint in Eq. 5.6.13 as follows:

$$\sigma_K^2 = \sigma_{B,D}^2 - 2\,\Sigma w_i \sigma_{B,Xi} + \Sigma\Sigma w_i w_j \sigma_{Xi,Xj} + \mu\,(\Sigma w_i - 1) \qquad (5.6.15)$$

where μ is the Lagrange multiplier. This equation is differentiated with respect to each of the w_i and μ, resulting in a set of simultaneous equations that may be solved for w_i and μ as follows:

$$\begin{aligned}
w_1\sigma_{X1,X1} + w_2\sigma_{X1,X2} + \ldots + w_n\sigma_{X1,Xn} + \mu &= \sigma_{B,X1} \\
w_1\sigma_{X2,X1} + w_2\sigma_{X2,X2} + \ldots + w_n\sigma_{X2,Xn} + \mu &= \sigma_{B,X2} \\
w_3\sigma_{X3,X1} + w_2\sigma_{X3,X2} + \ldots + w_n\sigma_{X3,Xn} + \mu &= \sigma_{B,X3} \\
&\vdots \\
w_1\sigma_{Xn,X1} + w_2\sigma_{Xn,X2} + \ldots + w_n\sigma_{Xn,Xn} + \mu &= \sigma_{B,Xn} \\
w_1 + w_2 + \ldots + w_n + 0 &= 1 \qquad (5.6.16)
\end{aligned}$$

The foregoing system of equations (5.6.16) is usually solved using simple gaussian elimination to determine the weights and the Lagrange multiplier μ. The kriging error of estimation is then computed based on the following relation:

$$\sigma_K^2 = \sigma_{B,D}^2 - \Sigma w_i \sigma_{B,Xi} + \mu \qquad (5.6.17)$$

which is based on the equation for error of estimation (5.6.14) and the relationship between μ and the kriging equations (5.6.16),

$$\mu = -\,\Sigma w_i \sigma_{B,Xi} + \Sigma\Sigma w_i w_j \sigma_{Xi,Xj} \qquad (5.6.18)$$

The individual covariances and block variances are computed from the experimental variogram as follows:

$$\sigma_{Xi,Xj} = \sigma_{S,D}{}^2 - \gamma(Xi,Xj) \qquad (5.6.19)$$

$$\sigma_{B,Xi} = \sigma_{S,D}{}^2 - AVE(\gamma(B,Xi)) \qquad (5.6.20)$$

$$\sigma_{B,D} = \sigma_{S,D}{}^2 - AVE(\gamma(B,B)) \qquad (5.6.21)$$

where $s_{S,D}{}^2$ is the variance of samples in the deposit, $g(Xi,Xj)$ is the value of the variogram function between samples Xi and Xj, $AVE(g(B,Xi))$ is the average value of the variogram between the block and sample Xi, and $AVE(g(B,B))$ is the average value of the variogram between all points within the block.

Lognormal Kriging: *Lognormal kriging* is a method of nonlinear kriging that was developed to improve estimation when the underlying data are distributed according to a lognormal probability distribution. The basics of lognormal kriging include: (1) the variogram is computed using the natural logs of the data, (2) the kriging system is solved to provide a weighted average of the natural logs of the data, and (3) the kriged log average is then transformed back to normal values using lognormal transformation similar to that shown earlier in Eq. 5.6.1. The mathematics of lognormal kriging are complex and are discussed in Rendu (1978) and Journel (1978, 1980).

Complications in the practical application of lognormal kriging are many, including a strict requirement for a lognormal distribution and a variogram which is stationary over the field of estimation. Serious local and global biases may occur if either of these conditions are not met. In addition, there is a tendency for lognormal kriging to overestimate the high-grade end of the population when the coefficient of variation is greater than 2.0. Lognormal kriging is recommended only for special purposes where the results can be monitored closely and adjusted to prevent biases.

Indicator/Probability Kriging: *Indicator kriging and probability kriging* are related methods that are used to improve estimation when ore zones are erratic and grade distributions are highly variable and complex. Advantages of indicator kriging include less smoothing of estimated grades than ordinary kriging and robustness in handling nonstandard grade distributions.

The first step in indicator kriging is to set one or more cutoffs with which to define indicator variables. Given a cutoff g_c, the indicator variable is set to 1 if the grade is above g_c or 0 if the grade is below g_c (the order of the { 1,0 } coding may be reversed); indicator variables are coded similarly for each desired cutoff. Variograms are modeled for each indicator variable and an expected value for each indicator is estimated using ordinary kriging and the appropriate indicator variogram.

The resulting indicator estimates, which may be interpreted as either the probability that the block will be above the cutoff or the percentage of the block that is above cutoff, are used to estimate the grade of the block as follows

$$g^* = \Sigma(I^*_j - I^*_{j+1}) \times g_j \, j = 0,1,2...n \qquad (5.6.22)$$

where each I^*_j is the estimate for the indicator for cutoff j, g_j is the estimated grade for the interval j to j+1, and n is the number of indicator cutoffs. The interval grades g_j are usually estimated as the average of the cutoff grades for the interval, or, if the interval is large, may be estimated from the kriged grade of those data in the interval j to j+1. The prior method is more precise when a large number of indicator cutoffs are defined; the latter is most often used for a single cutoff.

Other Types of Kriging: Other types of kriging that are not widely used include universal kriging, cokriging, disjunctive kriging, and soft kriging. *Universal kriging* is a method to incorporate trends into the kriging equations. If the trends are defined according to a secondary variable, it is known as universal kriging with exogenic drift. A method of universal kriging that uses the geologists' interpretation of grade-zone trends as the exogenic drift is zoned kriging. *Cokriging* is the method of kriging that accounts for the correlation of a primary variable with a secondary variable, for example, gold with silver or molybdenum with copper, etc. When cokriging is used with qualitative secondary variables such as alteration, rock type, or other geologic features, it is known as *soft kriging*. *Disjunctive kriging* is a method used which attempts to estimate not only the local block grade but also the shape of the tonnage-grade distribution within the block.

5.6.8.6 Volume-variance Effects and Recovery Functions

The volume-variance effect refers to the inverse relationship between the distribution variance and the volume of blocks. The volume-variance effect is characterized by Krige's relationship as follows

$$\sigma^2{}_{B,D} = \sigma^2{}_{S,D} - \sigma^2{}_{S,B} \qquad (5.6.23)$$

where $s^2_{B,D}$ is the variance of blocks in the deposit, $S^2_{S,D}$ is the variance of samples in the deposit, and $S^2_{S,B}$ is the variance of samples in the block. The variance of samples in the block may be estimated from the variogram as follows

$$\sigma^2{}_{S,B} = \overline{\gamma}(_{S,B}) \qquad (5.6.24)$$

where $\overline{\gamma}_{(S,B)}$ is the average of the variogram for samples within a block with the size and orientation of the mining block.

The volume-variance relationship is unimportant where the entire deposit is above the cutoff grade or where the ore is mined nonselectively. Generally, however, the cutoff grade is higher, and only a portion of the mineralized grade distribution is selectively mined as ore. The shape of the grade-tonnage distribution, as defined by the distribution variance, is then a critical factor in determining the grade and tonnage above cutoff.

For practical resource estimation purposes, the variance of mining blocks is generally larger than the variance of kriged resource estimation blocks. The variance of mining blocks is generally smaller than the variance of resource estimation blocks for polygonal estimation.

Polygonal estimation underestimates tons and overestimates grade for low cutoffs. At higher cutoffs, tonnage and grade are both overestimated. Kriging tends to overestimate tons and underestimate grade for low cutoffs. At higher cutoffs, tonnage and grade are both underestimated.

For polygonal estimation, the difference between estimated and mined reserves is usually handled with dilution factors where a fixed tonnage is added with a grade that is less than the cutoff. These dilution factors are adequate for correction of overall reserves but are not accurate for smaller areas if local grades vary significantly from the average grade. Caution must also be observed since dilution factors will vary according to the cutoff grade, the population variance, and the amount of variance reduction between the polygonal and mine block distributions. It should be noted that polygonal reserve estimates may require dilution factors for both volume-variance effects and contact-mining geometric effects.

Kriging reserves are corrected for volume-variance effects according to the distribution of mining blocks within the reserve block as (1) the variance and distribution of mining blocks within the reserve block is estimated, and (2) the tonnage and grade above cutoff is estimated for the block. The mining block distribution parameters are most effectively determined by compiling

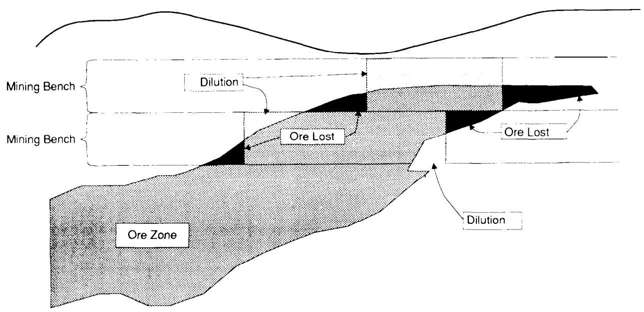


Fig. 5.6.19. Ore dilution and losses caused by mismatch between the mining geometry and the ore geometry.

production statistics of the grade-tonnage curves, or recovery curves, for several grade ranges of estimated blocks. Alternatively, a lognormal distribution may be assumed for mining blocks within reserve blocks. The variance of the distribution of mining blocks within reserve blocks may be estimated from production data. If production data are not available, the variance of mining block may be approximated by

$$\sigma^2_{b,B} = \sigma^2_{B,D} - \sigma^2_{S,B} + \sigma^2_{k,B} - \sigma^2_{k,b} \qquad (5.6.25)$$

where $\sigma^2_{b,B}$ is the variance of mining blocks in the estimated reserve block, $\sigma^2_{B,D}$ is the variance of reserve blocks in the deposit, $\sigma^2_{S,B}$ is the variance Of samples in the reserve block, $\sigma^2_{k,B}$ is the estimation (kriging) variance of reserve blocks, and $\sigma^2_{k,b}$ is the estimation variance of mining blocks (based on grade-control samples).

5.6.8.7 Dilution and Mining Losses

The estimated tonnage and grade must be adjusted for dilution of grade and losses of tonnage that occur in the course of mining. Dilution is waste that is not segregated from ore during mining, thus decreasing the grade of the ore and increasing the tons. Ore losses are due to the inability of the mining method to follow accurately and to segregate small isolated pods and small irregular offshoots from the main ore body. Dilution is most significant in deposits with sharp contacts between high-grade ore and barren waste and least significant in deposits with gradational contacts between ore and waste.

Dilution tonnage is estimated according to the quantity of waste mined with the ore based on the mismatch between ore body and mining geometry, overbreak in blasting, or lack of accurate location of the ore/waste contact as shown in Fig. 5.6.19. Care must be taken in estimating dilution that the actual ore/waste contact is not more irregular than the model since dilution will be underestimated as shown in Fig. 5.6.20. Dilution grade is estimated as the grade of the waste at the ore/waste contact. Mining losses and grades are estimated according to similar procedures.

5.6.8.8 Selection of Resource Estimation Methods

Selection of an appropriate resource estimation method depends on the geometry of the deposit, the variability of the grade distribution, the character of the ore boundaries, and the amount of time and money available to make the estimate. Deposit geometry determines the amount of detail that must be interpreted and input to the reserve estimation; the variability of the grade distribution determines the amount of smoothing that is required to estimate minable blocks; the character of the ore boundaries determines how grade will be estimated at the borders between different grade zones; and the available time and money determine the detail and effort that will be expended on the estimate. Considerations for selection of a resource estimation method are summarized in Table 5.6.1.

Cost: Simple, manual methods such as polygonal and cross-sectional estimations are the cheapest and quickest methods for estimation of resources when the quantity of data is small. This is usually the case for preliminary evaluations in exploration stages. As the number of data increase and a more detailed estimate is desired, computer-assisted methods should be used in order to save time and money. The least expensive computer-assisted methods are automated polygonal or nearest-neighbor methods and the most expensive methods involve extensive definition of geologic controls in conjunction with the more complex geostatistical methods. For further discussion of computer applications to ore estimation, see Chapter 8.4.

Ore Boundaries: The appropriate reserve and dilution estimation method is determined by the character of the ore/waste contacts. Sharp, simple boundaries are modeled with linear outlines defining discrete mineral zones. Individual estimations are made for each mineral zone; dilution is estimated based on the intersection between the shape of the mineral zones and the shape defined by the geometry of a mining method. A sharp, irregular boundary is also described with linear boundaries defining mineral zones; the actual ore-waste contact is much more irregular than the interpreted boundary, and dilution must be increased accordingly. Geometric methods are usually appropriate for ore bodies with sharp contacts, although kriging or inverse-distance methods may be used within the zones if supported by sufficient data.

Gradational boundaries are handled as transitional between different mineral zones; kriging or inverse-distance methods are most appropriate to model ore bodies with gradational contacts. Sufficient dilution for a gradational contact is usually incorporated in the modeling method.

Extremely erratic, irregular boundaries are difficult to define accurately and are most appropriately estimated using methods such as indicator kriging.

Deposit Geometry: *Simple geometry* is often found in tabular, stratabound deposits, veins, and structural zones. The geometry of these deposits is easily described using two-dimensional methods such as contouring of thickness and elevation. Few additional controls are required other than boundaries to limit the lateral extent of the mineral zones.

Deposits with *moderately complex geometry* include both deposits with simple geometry that have been moderately folded or faulted and deposits with large, simple, massive shapes such as porphyry copper and molybdenum. Definition of deposit geometry includes definition of fold axes, fault boundaries, and zoning of trends within the deposit. While these controls are not usually difficult to define, their definition is necessary to provide accurate resource estimates.

Deposits with *very complex geometry* are usually associated with structural deformation and are folded, faulted, stretched, and twisted to form extremely discontinuous shapes that are difficult to describe and model. Multiple ore controls such as a combination of stratigraphic and structural controls or multiple, overlapping pulses of mineralization also commonly form very complex shapes. Definition of deposit geometry requires detailed examination of structural geology and ore controls to provide cross sections or plan maps which define the shape and location of mineral zones. These sections or maps may then be used directly for manual resource estimation or may be digitized to provide control for a computer block model or three-dimensional

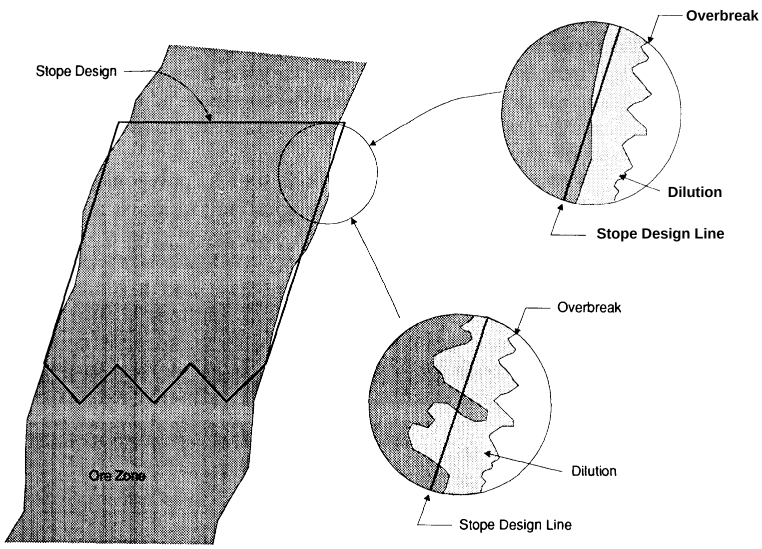


Fig. 5.6.20. Underestimation of dilution because of an irregular ore-waste contact.

wire frame model. *Deposits with complex geometry are prone to large estimation errors due to misinterpretation of deposit geometry and ore controls; order of magnitude errors are common.*

Grade Variability: Deposits with low variability may be estimated with many methods. Common methods include automatic contouring and polygonal methods with cross-sectional estimation or area-averaging techniques for more complex geometry.

Weighted averaging methods (kriging and inverse distance) are most commonly used for estimation of deposits with moderate variability although polygonal or cross-sectional methods are also used. Weighted averaging methods may require recovery functions and polygonal methods may require dilution to compensate for volume-variance effects, although in most cases the adjustments are small, on the order of 5 to 15%.

Weighted averaging methods are most commonly used for estimation of deposits with high variability. Other appropriate methods may include indicator kriging, polygonal, and cross-sectional methods. Volume-variance effects are often large with these deposits and must be compensated for with recovery functions for weighted averaging methods and large dilution of polygonal and cross-sectional reserves. For coefficients of variation above 2.0 or 3.0, local grade estimates are extremely difficult and must be tempered with judgment and caution.

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Table 5.6.1. Selection of Estimation Method Based on Deposit Geometry and Variability

	Low Variability, COV < 0.25	Moderate Variability, COV > 0.25 < 0.75	High Variability, COV > 0.75
Simple Geometry			
Deposit description	Tabular, continuous grade and thickness Flat or constant dip	Tabular, large ore pods Moderately variable grade	Tabular, small ore pods Highly variable grade
Example deposits	Evaporite Sedimentary Iron Limestone Coal	Stratiform copper Mississippi Valley lead Simple porphyry copper, molybdenum	Gold veins Gold placers New Mexico uranium Alluvial diamond
Estimation methods	Grade and thickness using any 2-dimensional method: polygonal, contouring, inverse distance, kriging. Geometric controls for boundaries of ore zone, faults, and fold axes	2-dimensional methods. Inverse distance or kriging. Polygonal or cross section with 5 to 15% dilution	2-dimensional methods. Inverse distance or kriging with recovery functions. Polygonal with 15 to 35% dilution
Moderately Complex Geometry			
Deposit description	Simple, bedded. Uniform grade but erratic thickness, gentle folding, or simple faulting	Simple three-dimensional geometry. Moderately variable grade	Simple three-dimensional geometry. Two-dimensional with smaller, more erratic ore pods. Simple folding, faulting
Example deposits	Bauxite (variable thickness) Lateritic Nickel (variable thickness) Salt Dome	Porphyry copper Porphyry molybdenum	Stockwork and Carlin-type gold Volcanogenic base metals
Estimation methods	Estimate grade, thickness and elevation using any 2-dimensional method Must define structural geology (faults, fold axes) Variability of thickness may be difficult to predict	Inverse distance or kriging with external controls to define the shape and grade trends. Polygonal and cross-sectional methods may be used but will require dilution/volume variance correction	Inverse distance of kriging with recovery functions. Polygonal or cross section with 15 to 35% dilution
Complex Geometry			
Deposit description	Otherwise simple deposits that have been severely folded and faulted	Complex geometry due to faulting, folding, or multiple mineralization controls. Moderately variable grade	Deposits with extremely variable grade and highly contorted, complex ore shapes. Typically little continuity between individual ore zones. General mineral envelope definable but with 50% or less ore
Example deposits	Talc Gypsum (deformed)	Tungsten skarns (folding/faulting). Base metal skarns (erratic shape). Copper porphyry combined with local skarns or replacements (multiple controls)	Archean gold deposits Roll front uranium
Estimation methods	Cross-section methods with detailed definition of structural geology. Difficult to define geometry for 3-dimensional block models and geostatistical methods	Cross-sectional methods with detailed input to describe structural geology and or zones. Geostatistical methods may be appropriate but difficult to implement because of geometric complexity	Estimation very difficult. Size, shape, and grade not locally predictable. Cross-section, area-outline methods, indicator kriging applicable. Errors of 50 to 100% typical. Tonnage often overestimated because of incorrect geologic model

Chapter 5.7
EVALUATION AND PRODUCTION SUPPORT

DONALD E. RANTA

The value of geologic data gathered at mineral projects or mines lies in their effective use by other evaluation or production specialists. Little benefit is gained from accurate geologic information and sound geologic ideas if data are not properly organized and presented and if reports documenting the results are poorly written. Thus written and oral communications of geologic information are essential to guide the evaluation process and to achieve production goals.

Geologic data and interpretations form the foundation for both mine evaluation and mine production by providing essential information for estimating ore reserves and for mine planning and process design. Proper geologic work requires a keen awareness of and an ability to anticipate the technical requirements of geotechnical engineers, hydrologists, mining engineers, metallurgists, and other technical specialists who all rely on the geologic data and samples. Presentation of pertinent data in a usable format and frequent communication of new geologic knowledge to the other technical specialists are integral parts of the geologic program.

Geologic support for other specialists involved in mine evaluation and production includes the following (Ranta et al., 1984):

1. Gathering and assessing geologic data and samples for geotechnical analysis.
2. Collecting groundwater data for hydrological investigations.
3. Defining the ore body and high-grade areas for mine planning; exploring for additional ore bodies and other materials in the district.
4. Identifying ore types and collecting samples for metallurgical testing.
5. Evaluating geologic characteristics and ore potential of sites designated for waste dumps, a mill, leach pads, shops, offices, and associated facilities.
6. Assisting with land, legal, environmental, and permitting studies.

5.7.1 GEOTECHNICAL SUPPORT

Geotechnical engineers assist in mine planning and operations by collecting, interpreting, and applying geologic data to the solution of mine engineering problems. Geologists support this effort by collecting additional geologic data specifically useful to the engineer as well as standard geologic information. Geotechnical engineers use the data in helping to select a mining method, to design the mine, and to define and mitigate potential ground stability problems.

Geologic data most useful for initial geotechnical analysis consist of descriptions and map and section representations of rock types, structures, and alteration, which could be considered the "geologic model" of an ore deposit. First, ore types, wall rock, and overburden are separated on maps into structural domains and then physical characteristics are used to define rock strengths and rock mass fabric within each domain. Separate structural domains occur between major faults, which are the zones of greatest weakness, and are partly dependent on lithology and alteration. Structural domains are initially determined by the geotechnical engineer through review and analysis of the basic geologic data, including geologic maps, drillhole logs, and interpretive geologic sections and plans.

Rock strengths are determined by rock substance between structural discontinuities and by fracture strength. Rock mass fabric elements include faults, joints, bedding, veins, and foliation (Stewart and Sacrison, 1984). These types of data are used to establish classification systems in predicting and mitigating potential ground stability problems.

Additional geologic data collected specifically to support the geotechnical programs include estimates of rock competency and hardness, photographs of unsplit core that has natural fractures marked, representative rock or whole core samples for laboratory testing of physical strength, measurements of fracture characteristics in core and bedrock, and records of various water levels and flow rates. Engineering logs of fracture characteristics per unit interval of drill core often include the following: core recovery percentage, maximum length of unbroken core, fracture frequency, rock quality designation (RQD), and rock competency estimate. Noting the fracture orientation, fracture-filling material, and vein-filling material is also important. Monitoring of geotechnical equipment for evidence of ground movement in a mine is often the responsibility of the geology department. These data are collected by the geologic staff for use by the geotechnical engineer. According to Pillar and Drummond (1975), "the best information on the strength characteristics is obtained when the geologist logging the core can see the interrelationships of the various physical features and can make the appropriate notations."

5.7.2 HYDROLOGIC SUPPORT

Hydrologists have the responsibility of locating sources of both process and potable water for the mine operation. Several superior mineral deposits in desert regions are not "ore" bodies because of a lack in the quantity or quality of local water. Other deposits are not viable because local rainfall or groundwater inflow exceeds economic pumping capacity. Either too little or too much water is a hydrological problem that often is a function of the local geologic setting.

Geology is helpful, and often essential, in locating available sources of water. Water encountered during exploration drilling or in underground workings should be noted for location and measured for flow rate, pH, various dissolved solids, and temperature. Water courses should be correlated with known geologic features, especially structures or aquifers, to predict potential water source or problem areas. Surface water flow, springs, and seeps should be noted and plotted on maps, and measured, if possible, as a part of routine geologic mapping.

The mine planning department must have a reasonably accurate estimate of groundwater flow that will be encountered in open pit or underground mine development. Dewatering by a series of wells may be required in advance of the development. An accurate estimate of the time and costs of dewatering or of water development is necessary for proper planning.

5.7.3 MINE PLANNING SUPPORT

Mine planning engineers require definition of the location, size, shape, attitude, boundaries, and internal grade distribution of a mineral deposit for mining method selection, mine layout and design, and production scheduling. Tonnage and average grade above various cutoff grades of the geologic resource must be accurately estimated with a high degree of confidence. Through a detailed mine plan, a large portion of the geologic resource is designated as the minable "ore" reserve. Delineation and reserve estimation of high-grade zones as the most valuable portion of the ore deposit is especially important in selection of an initial mining area. Any changes in ore body configuration resulting from new drilling and sampling results must be quickly reported to mine planning engineers. Geologic resource data can be transmitted by sets of grade zone sections and plan maps that should be at the same scale as mine planning maps. A complete computer database of sample, assay, and pertinent geologic data in a format compatible for mine planning also must be transmitted. A reliable inventory of all mineralization should be estimated, also noting inferred resources and additional exploration potential. Ore types, such as oxide, sulfide, mixed oxide and sulfide, strong argillization, strong silicification, etc., should be identified to ensure proper production scheduling and blending, if needed. Chemical analyses for a variety of elements should be carried out on representative samples of ore and waste to identify potential byproducts and contaminants.

Potential byproducts should be considered in an equivalent grade model if their value could have significant economic impact on the mining operation. Unusual groundwater flow, faults, alteration, or rock types that may affect ground stability and mine design also should be located and described. Types and thickness of overburden, sharp assay boundaries, and post-mineral dikes and faults are identified to provide information on grade discontinuities that may limit mining recovery of ore or cause dilution. This detailed information is used by the mine geologist for grade control on benches or in stopes. Selective mining is largely dependent on a thorough geological understanding of a deposit. Specific gravity measurements of the various ore and waste types are made to determine their tonnage factors for converting volumes to short tons (tonnes). Specific gravity measurements are also needed for deposits having highly variable rock densities, for the selection of haulage equipment, and in planning production schedules.

Long-term planning can benefit from knowledge of additional ore bodies and potential ore bodies in the district. Discovery of new deposits could change the following: sequence of development and mining, optimum production rate, economies of scale, and mine life. If a significantly better deposit were found, the overall operation could be substantially improved. Also transmittance of the ore deposit data to an associated exploration group could benefit the entire company. Continuing exploration in and near the existing deposit is a vital function of the geological department.

5.7.4 METALLURGICAL SUPPORT

Ore types, which may behave differently in ore processing, must be identified and defined for the deposit according to their geologic characteristics. The various ore types should be plotted on sections and plan maps to show their spatial distribution throughout a deposit and transmitted to the metallurgist as the data become available for his/her short-term planning. An initial indication of the proportions of various ore types in a deposit can be determined by summing the cumulative drillhole footage of each ore type and calculating their percentages in the deposit. Composite samples of the various ore types are first identified and then assembled by a geologist, in close coordination with a metallurgist, to ensure that the samples are representative of the entire deposit (or selected portions of the deposit) and are adequate for bench-scale metallurgical testing. Samples of ore are usually collected from coarse rejects or cuttings resulting from drillhole samples that have been prepared for assaying. Other samples for testing can be collected from surface trenches, outcrops, mine benches, or underground workings. All coarse rejects of drillhole samples from a mineral deposit should be retained for possible metallurgical testing. If larger bulk samples are required for pilot plant-scale metallurgical testing, then large-diameter drillholes, test pits, or underground exploration/development workings into geologically specified areas may be necessary. The bulk samples should be representative of all the deposit or, at least, of the most important ore types.

Geologic characteristics that define various ore types include lithologic type, alteration type and intensity (e.g., argillization vs. silicification), mineralization type and intensity, sulfide vs. oxide, etc. Detailed mineralogic investigation of a deposit is necessary to define the ore types. Of the various ore types, unusually hard or soft rocks and deleterious trace elements, minerals, or rock types should be identified to help alleviate potential processing problems. Detailed information on the ore and gangue minerals, such as mineral assemblages, grain sizes and shapes, and textural relationships (e.g., interlocking or coating) should be described for the metallurgist. Potential byproducts should be considered and evaluated. Routine spectrographic analyses of representative samples will help identify potential byproducts and deleterious trace elements. Mineralogic examination of the intermediate materials resulting at different stages of the processing is often beneficial in determining how to improve metallurgical recovery.

5.7.5 LAND AND LEGAL SUPPORT

Initial support for land and legal aspects of a project in the United States can be provided by establishing mineral "discoveries" (valuable minerals in place) on all claims within the mineralized area, thereby validating the claims in accordance with the mining law. In addition, an evaluation of the mineral potential and "mineral-in-character" of areas designated for plant sites, waste dumps, tailing ponds, leach dumps, and ancillary facilities is necessary from both the engineering and permitting aspects of a project or expansion of a mine. The project or mine geologist should make recommendations for acquisition or disposition of property and evaluate nearby property submittals. Patenting of claims requires significant geologic input.

5.7.6 ENVIRONMENTAL SUPPORT

Identification of environmental geology hazards, such as potential landslides, flood plains, avalanche chutes, and active faults, can help avoid risk-prone areas in siting surface facilities. Environmental baseline studies and permitting often require geologic input. Environmental engineers require data on deleterious trace elements that may be naturally leached from waste dumps, tailings, or mine workings. Final reclamation of the entire mine site may be partly controlled by geologic conditions, such as acid leachate from the mine or dumps, or the future preservation of a low-grade resource. Because the exploration or project geologist is often the first company employee to reside at a property, he may be aware of valuable historical, political, sociological, and geographical information to pass on to the environmental and permitting specialist.

More generalized discussion of several of these topics is found in Sections 3 and 7.

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Chapter 5.8
CASE STUDIES

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5.8.1 DEVELOPMENT GEOLOGY OF THE COPPER FLAT PORPHYRY COPPER DEPOSIT

PETER G. DUNN

5.8.1.1 Introduction

The scope of geologic work expands significantly as a result of a successful exploration program. Geologists involved in subsequent development work obviously continue to be concerned with the control and distribution of the mineralization, but must also address such factors as:

1. Careful checks on the sampling and assaying.
2. Examination of underground workings driven to provide samples for both metallurgical testing and assay comparisons.
3. Selection of representative samples used to establish recoverable amounts of precious metals in base metal ores.
4. Location of an adequate water supply.
5. Determination of plant site and waste dump areas as nonmineral.

These investigations are necessary for engineering design, for acquiring environmental permits, and for financial analysis.

Development of the Copper Flat porphyry copper deposit in south-central New Mexico provides a case history of such studies. The mining district had produced gold from placer workings and both precious and base metals from vein deposits surrounding a porphyry stock. The porphyry deposit was recognized early (Harley, 1934), and had been evaluated in the period 1956 to 1972; the major development effort took place in 1975 and 1976. Plant construction began in 1980, and production began in 1982. The Copper Flat mine, however, had a short operating life due to depressed copper prices in 1982.

One aspect of this work that needs to be stressed is the careful recording and checking of the data. It is important not only for the mining company to know what it really has, but also for any possible joint venture partners and engineering firms to have access to the necessary data. It is particularly important, if the project is to be financed, that the lending institution is satisfied that the various studies have been well carried out and meticulously recorded.

5.8.1.2 Geology

The Copper Flat porphyry deposit is approximately 1500 by 2000 ft (450 by 600 m) in plan and occurs in the center of a small quartz monzonite stock that had intruded a circular block of andesite about 4 mi (7 km) in diameter. Latite dikes, 5 to 20 ft (2 to 6 m) thick, cut the quartz monzonite and extend across the andesite terrane. Most of the vein deposits in the district occur along the dike contacts.

The porphyry deposit includes a large hydrothermal breccia pipe about 1400 ft (425 m) long and 500 ft (150 m) wide, immediately below the alluvium. It extends to a depth of at least 1000 ft (300 m) and increases in size with depth. The breccia consists principally of quartz monzonite fragments within a matrix of quartz, biotite, and potash feldspar, with pyrite, chalcopyrite, and minor molybdenite (Dunn, 1982).

Copper occurs almost exclusively as chalcopyrite with minor amounts of chalcocite and copper oxide minerals. Molybdenum occurs as molybdenite. The sulfide mineralization first formed in narrow veinlets and as disseminations in the quartz monzonite with weakly developed sericite alteration. This stage of mineralization was followed by the formation of the breccia pipe with the introduction of abundant pyrite, chalcopyrite, and molybdenite with strong potassic alteration.

5.8.1.3 Diamond Drilling

Drillholes completed prior to 1975 had been spaced about 400 ft (120 m) apart, and breccia mineralization had been intersected in about eight holes. After the recognition of the mineralized breccia in the center of the deposit, the initial phase of the 1975 development drilling was undertaken to confirm that the eight breccia intersections represented a single breccia pipe.

Grade changes within the quartz monzonite are generally gradual, and by the completion of the development drilling program in early 1976, drillholes outside the pipe were spaced about 200 ft (60 m) apart. Grade changes within the breccia pipe, however, are commonly abrupt, so the drillholes in and immediately around the breccia pipe were spaced about 100 ft (30 m) apart. All told, 175 vertical holes with a total length of 125,300 ft (38.2 km) were drilled in the mine area; 135 of these holes, 96,500 ft (29.4 km), were drilled in 1975 and 1976. Almost all the core was NX, approximately 2 in. (50 mm) in diameter.

The principal purpose of the diamond drilling was to provide the best possible sample for assay. Because of the overall low grade of the Copper Flat deposit and because of the erratic nature of the mineralization within the breccia pipe, core recovery was important to the evaluation. For this reason, a core recovery incentive bonus, payable to both the contractor and the drill crews, was included in the drilling contract. Core recovery was calculated on a weight basis for the entire hole, and the bonus was paid on those holes with recovery above 88%, with a corresponding penalty (to the contractor only) for those below 88%. The overall core recovery on the project during the time the bonus provision was in effect was over 96%, and only three of the holes failed to gain the bonus. This figure can be compared with 89% recovery in holes drilled without the incentive bonus. The bonus increased the direct drilling cost about 10% ($1.60/ft or $5.25/m), but the amount was considered to have been well spent.

In drilling prior to 1975, the core had been split and one-half used for assay. The other half from selected intervals was later assayed, and the two results commonly did not correspond well—particularly for samples of coarse-grained breccia mineralization. It was therefore decided to use the entire drill core as the assay sample; each box of core was photographed, and the slides were kept as a visual record of structural features in the core. Short specimens were collected every 10 to 20 ft (3 to 6 m) and retained for a permanent core record. These specimens provided thin sections for later petrographic work and samples for rock strength tests. The coarse rejects from the sample preparation were also retained and were used for later metallurgical work.

5.8.1.4 Sample Preparation and Assaying

The importance of proper sample preparation and assaying cannot be emphasized too strongly. It is essential that a program of assay checks be undertaken throughout the drilling program, and that the results of these checks are available early so that any necessary changes in the procedures can be made before too much of the work has been completed. Two aspects of this procedure need to be checked: reduction of the core down to the assay sample and the analytical work itself.

At Copper Flat, sample preparation was checked by preparing a second pulp from the coarse reject and comparing the assay results from the two samples. The assaying was checked by having the original pulp reassayed by the original lab and by separate labs.

At the beginning of the drilling program in 1975, the core was brought to Tucson, and the pulps were prepared by a commercial assayer. Results from the first 78 reject checks indicated the sample preparation procedure was not acceptable. The differences between original and check copper assays averaged 0.04% copper, but with a standard deviation of 0.23. The *differences* between two pulps from the same sample interval ranged as high as 1.14% copper—nearly three times the average grade of the deposit. The differences between molybdenum assays from the same samples averaged 0.003% but with a standard deviation of 0.022. The highest difference was 0.12% molybdenum—10 times the average grade.

The problem in sample preparation was due to the splitting of samples, which averaged about 36 lb (16 kg), before they were crushed fine enough. The procedure was revised, and a sample preparation laboratory was built on the site. Under the new procedure, each entire sample was crushed to — 1/2 in. (12 mm) before being split, using a Jones splitter, down to one-eighth of the original weight; this one-eighth fraction was crushed down to — 10 mesh before being split again. The resulting split, approximately 2 lb (1 kg), was pulverized to — 100 mesh, and 0.5 lb (0.25 kg) of pulp was selected for assay.

Under the revised procedures, the reject checks were judged acceptable. Over 500 sample pairs were compared, with an average difference of 0.004% copper with a standard deviation of 0.07; the average difference in the molybdenum checks was nil with a standard deviation of 0.009.

Because of logistical problems, the results of the original reject checks were not available until nearly 30 holes had been completed. It was necessary to re-prepare nearly 2000 samples from these holes and reassay them at a substantial cost. It clearly would have been preferable to do it correctly from the onset.

The samples were assayed for total copper and molybdenum; the upper portion of each drillhole was also assayed for nonsulfide copper. The assay checks indicated that the precision was within acceptable limits. Although the range of differences between two assay values from the same pulp was slightly wider in the higher grade samples, there was no significant bias. A set of standards, prepared from Copper Flat ore, was included with each shipment of pulps, and the assays of these standards were also within acceptable limits.

Both gold and silver were known to occur in the ore body, but their concentrations in drill core were generally too low for accurate analysis. Because only the gold and silver recovered in the copper concentrate are paid for, the precious metal contents determined through bench flotation tests were the values used in the financial evaluation. Rougher flotation concentrates were prepared from 330 drillhole composites using the coarse assay rejects. Each composite represented about 80 ft (24 m) of core and weighed about 40 lb (18 kg). Each of these concentrates was assayed for gold and silver to provide data concerning the spatial distribution of precious metals in the deposit.

These rougher composites were subsequently combined into three samples that corresponded to the ore that would be mined during various stages of the planned production. Test work was carried on to the second cleaner concentrate on these three major composites before they were assayed. These results indicated the range of precious metals expected from the copper concentrates: a minimum of 50% of the gold and over 90% of the silver. Although little mineralogical work was done on the concentrates, these high recoveries suggested that most of the precious metals occur with the chalcopyrite.

It is important to remember that assay samples represent a minute portion of an ore body—even in the case of Copper Flat where the drillholes are close-spaced. The core drilled within the minable deposit amounted to about 200 tons—only slightly more than 0.0003% of the minable reserve. Assay pulps from that amount of core totaled about 3 tons (2.7 t). That may seem like a great deal of assay pulp, but it represented only about 0.000005% of the ore body. The time and money spent to provide confidence in precision of the sample data are clearly justified; engineers evaluating a mining property, particularly those representing lending institutions, are interested in more than just tonnage and grade. They are also interested in the precision of the assay data and in the efforts made to determine that precision.

5.8.1.5 Ore Reserve Estimates

Manual reserve estimates were made, using both cross sections and polygons, at various stages of the project. The reserve estimate made for the feasibility study, however, was computer-based using an ore body model of 50 × 50 × 40-ft (15 × 15 × 12-m) blocks. The copper and molybdenum grades assigned to each block were the grades of the nearest drillhole over the same elevation interval—the computer equivalent of a drillhole-centered polygon reserve. It was believed that this method most correctly characterized this particular ore body—that the marked grade changes in the breccia pipe would be offset by the close-spaced drilling, and that the gradual grade changes in the surrounding quartz monzonite would allow the more widely spaced drilling.

The minable reserve was estimated at 60 million tons (54 Mt) with an overall grade of 0.42% copper and 0.012% molybdenum plus credits in both gold and silver. The high-grade breccia pipe was estimated to contain nearly one-half of the recoverable metal but to comprise only one-third of the minable tonnage. Because much of the breccia ore was available for mining early in the project life, its existence had a positive effect on the predicted financial outcome. For that reason, the early development drilling had been directed at defining the grades within the breccia pipe.

To develop confidence in the computer-calculated reserve estimate, a final manual polygon estimate was made using over 100 polygons. The two reserve estimates had the same copper grade and a difference in the molybdenum grade of only 0.001%; the computer reserve indicated about 2% more ore. The differences were not considered to be significant.

This manual reserve estimate took about two employee-months, but it turned out to be well worth the time spent. The work included a calculation of the total tonnage in the pit using vertical cross sections. Although the two ore tonnages matched well, the two estimated waste tonnages did not. On checking, it was found that the computer tonnage had been calculated using a pit model with a severe overhang in part of the pit wall. With that error corrected, the two stripping ratios matched within 1%.

Assay data from the underground metallurgical sampling program, described in the following, provided additional checks on both the drillhole assays and the estimated block grades. Nearly 1600 ft (490 m) of drift were driven to obtain bulk samples. They were designed to intersect five drillholes in the breccia pipe and five in the quartz monzonite.

Each round was assayed for copper and molybdenum, and these results were compared both with the drillhole assays and with the block estimates. In the first case, assays along the drifts from the midpoint between drillholes were compared with assays for the same distance along the drillhole above and below the drift. The second comparison was made using the estimated block grades from the computer block model and the average grade of that segment of the drift that intersected the block. Thirty-eight of the blocks were intersected by drifts, with an average length in each block of about 40 ft (12 m).

The results from the two comparisons were similar and provided some confidence in the typical practice of estimating the ore reserve for porphyry copper deposits on the basis of vertical drillholes. In the first case, the drift samples averaged 0.02% lower copper and 0.007% higher molybdenum than the drillhole samples. In the second case, the drift samples had a lower copper grade in high-grade blocks but a higher copper grade in average and low-grade blocks. The molybdenum results showed the same 0.007% increase in the drift samples as in the first case.

No geostatistical treatment of the assay data was undertaken. Although it seems unlikely that a reserve estimate using geostatistics would have been appreciably different, it would have established confidence limits on the reserve figures. With a geostatistical program in place it also would have been possible to determine whether the close-spaced drilling in the breccia pipe had been necessary. A considerable saving might have resulted.

5.8.1.6 Metallurgical Samples

Preliminary flotation tests on Copper Flat ore were done using the coarse sample rejects from selected intervals of drill core. After it was determined that the high-grade intervals represented a single continuous breccia pipe, underground mining was initiated to collect bulk samples for metallurgical test work. The drifts were planned to intersect roughly equal amounts of breccia and quartz monzonite ore, and the two ore types were kept separate throughout the metallurgical testing. The workings were about 165 ft (50 m) below the surface in order to assure that the samples were free from both oxidation and enrichment; access was through a 15º decline about 625 ft (190 m) long.

A sample tower was built on the site to crush and split the muck; each round was treated separately. The entire round was first crushed to — 1/2 in. (12 mm), and split using a Vezin splitter. Through successive crushing and splitting, the plant produced one sample of 1/4-in. (6-mm) material, representing about 2.5% of the original weight of sample, and a smaller sample of 10-mesh material, about 0.25% of the original. The larger sample was used for pilot plant flotation test work, and the smaller sample was used for assay. Each reject was stacked as a separately marked pile; some of these later had to be used for additional metallurgical work. Several test samples were run through the sample tower to test that assays of each split of the sample corresponded well.

The sample plant crushed and split about 3200 tons (2900 t) of breccia ore and 2800 tons (2500 t) of quartz monzonite, providing roughly 80 and 70 tons (73 and 64 t) of metallurgical samples for the two types. The head grades of the samples were 0.50% copper and 0.017% molybdenum for the breccia and 0.47% copper and 0.021% molybdenum for the quartz monzonite. These grades corresponded well with the estimated head grades for the first few years of production.

Underground development also provided an opportunity to collect large run-of-mine samples; slab rounds from several places in the drifts were collected. These samples were representative of both ore types and were used for crushing and grinding tests.

Unfortunately, in 1975, no consideration was made of semi-autogenous grinding (SAG) in the Copper Flat mill, and no sample was collected to test the amenability of the ore to such a circuit. After the underground sampling was completed, the portal was closed and the drifts were allowed to cave. When it was later decided that a SAG mill might be preferable to conventional crushing, it was necessary to drill 6-in. (15-cm) diameter core holes to provide the test sample. Two holes, one in each ore type, were drilled in 1979 with a total length of 380 ft (115 m). The test work indicated that the ore was suitable for semi-autogenous grinding, and a SAG mill was installed.

It is worth remembering that large-diameter drill core is expensive and is not as suitable as a run-of-mine sample for grinding tests. It is also difficult to get a representative sample from a few isolated drillholes. It would have been simpler and much cheaper to provide a better sample four years earlier.

5.8.1.7 Water Supply

An adequate water resource is critical to the development of any new mine. At Copper Flat, this phase of the development work was actually initiated before the major diamond drilling program got underway in 1975. Discovery of a suitable water supply turned out to be a more difficult task than was anticipated.

Eight-inch (200-mm) rotary drillholes were drilled in the Tertiary Santa Fe Formation east of the proposed tailings area. These holes had a maximum depth of 2000 ft (600 m) and bottomed in the Santa Fe Formation. Pump tests indicated that, although there was adequate water, a sufficient amount could not be produced because of low permeability.

Rotary holes were drilled successively farther to the east until a zone of adequate permeability was discovered about 7.5 mi (12 km) east of the plant site. The water table was approximately 350 ft (105 m) below the surface and 1200 ft (365 m) below the plant site. Because of apprehension that the New Mexico State Engineer might close the water basin to development, three production wells were drilled immediately in early 1976 to assure the water rights. These production wells were 26 in. (660 mm) in diameter, gravel-packed, and cased. They were pump-tested but not equipped with pumps at that time. Pump tests indicated that the individual wells would be capable of producing from 1000 to 2000 gpm (65 to 130 L/s), that maximum pumping from the wells was not expected to lower the water table more than 100 ft (30 m), and that the water supply in the field would have been adequate for the life of the mine.

Eventual design changes to the concentrator meant that the three wells would not have had sufficient capacity for start-up, during which time no water would be returning from the tailings thickener. A fourth well was drilled in the same field during 1980 to cover that shortfall.

5.8.1.8 Nonmineral Exploration

Soon after it was determined that a viable ore body existed at Copper Flat, it was necessary to start a definite drilling and sampling program to locate barren areas suitable for the mill, the waste dumps, and the tailings disposal area. Such an effort is required not only to prevent building the mill over part of the ore body, but also to satisfy the Bureau of Land Management (BLM) that the mill site locations are actually on nonmineral ground. This nonmineral exploration program had three phases.

The first phase was diamond drilling within three potential plant sites. One of these sites was found to be underlain by ore-grade mineralization just outside the perimeter of the proposed pit. The other two sites were underlain by barren andesite, and the selection between the two was based on the haulage distance.

The second phase consisted of surface sampling along the dikes and veins that crossed the plant site and part of the waste dump. Sampling was concentrated along the andesite-dike contacts. Surface samples were supplemented by assays of drill core in the same areas. The samples were assayed for copper, molybdenum, gold, and silver, with consistently low values.

The third phase was the most extensive aspect of the nonminera1 program, and involved a placer evaluation of the tailings area. BLM was particularly interested in this phase because the area coincided with a pre-World War II gold operation that had been the most productive placer in the state during this century (Johnson, 1972). The examination was directed by a consultant recommended by BLM.

Backhoe trenches, up to 30 ft (9 m) deep, were dug throughout the area, and close-spaced large samples were cut vertically down the walls of each trench. Because placer samples should not be assayed, the in-place volume of each sample was measured, run through a mechanical gravity separator, and then run through a sluice. The heavy fraction was then panned, and the free gold was collected and weighed.

The evaluation indicated that little placer gold remained, and it was restricted to recent alluvium within the present drainages. The tailings area was underlain by well-cemented alluvium with no significant placer gold. BLM allowed the site to be used for the tailings disposal area.

5.8.1.9 Costs and Staff

The total cost of the development program described previously was approximately $2.75 million, as detailed in the following:

Development Costs

Drilling (including bonus)		$1,400,000
Six-in. (152-mm) coring		60,000
Sample preparation—assaying		115,000
Underground development		400,000
Adit	$160,000	
Drifts	240,000	
Bulk sample plant		100,000
Construction	75,000	
Operation	25,000	
Water development		525,000
8-in. (203-mm) holes	150,000	
26-in. (660-mm) holes	375,000	
Nonmineral exploration		100,000
Consulting, computer time		50,000
Total		$2,750,000

These figures do not include the costs of metallurgical test work, environmental studies, land acquisition, or the cost of the feasibility studies that were based on the results obtained from this development work. The costs also do not include salaries of the professional staff who worked on the development program. For most of the period from January 1975 to April 1976, the staff consisted of four geologists, one of whom was the project manager, and a drilling supervisor, a surveyor, and a rodman.

Management is commonly not aware of the scope of the geological work necessary in the development of a new mine; the staff at Copper Flat was stretched fairly thin to cover all the various tasks that were required.

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5.8.2. NORTH GENERATOR HILL GOLD DEPOSIT, JERRITT CANYON DISTRICT, NEVADA

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The North Generator Hill gold ore body is the largest of a number of discrete bodies of gold mineralization that are collectively mined as part of the Enfield Bell mine and mill complex, located within the Jerritt Canyon Mining District in the Independence Mountains of northern Elko County, NV (Fig. 5.8.2.1). The North Generator Hill gold deposit is typical of other gold deposits within the Jerritt Canyon mining district and is also similar to other bulk-minable open pit gold deposits located in northeastern Nevada in that it contains micron- to submicron-sized gold particles disseminated in altered and highly fractured Paleozoic carbonaceous sedimentary host rocks.

Mineral exploration, originally directed towards antimony, commenced in the Jerritt Canyon area in 1971. By 1973, FMC Gold Corp. (then operating as FMC Corp.) discovered gold mineralization and turned its activities to that commodity. FMC was joined by Freeport Exploration Co. (manager prior to Independence Mining Co.) in 1976, following the signing of a joint venture agreement conveying to Freeport 70% of the interest in, and management of, its holdings within the Jerritt Canyon area. An area of interest was also created by the joint venture such that lands and leases acquired in an area peripheral to the existing holdings would become part of the joint venture agreement. An accelerated exploration program commenced in the summer of 1976, and by the fall of that year, high-grade economic gold mineralization was discovered in the fifth drillhole in the Marlboro Canyon area. Continued drilling discovered ore-grade mineralization in the southern portion of the North Generator Hill area during 1978. By the middle of 1979, encouraging results had been received from both the metallurgical work and ore reserve delineation in the Marlboro Canyon and North Generator Hill areas, and Freeport commenced preparation of a feasibility study and environmental impact study for the Jerritt Canyon area. Mining in the Jerritt Canyon area began in the middle of 1980 in the North Generator Hill and Marlboro Canyon regions. The North Generator Hill ore deposit was not explored aggressively until 1979, and exploration and development drilling along with mining, which began in 1980, has continued to the present.

As of Jan. 1, 1989, the North Generator Hill ore body contains a proven and probable ore reserve of 7,432,329 tons (6.74 Mt) at an average grade of 0.129 oz Au/ton (4.42 g Au/t) at an average waste-to-ore ratio of 2.5:1. The North Generator Hill ore body has produced over 901,000 oz (25 t) of gold, giving an

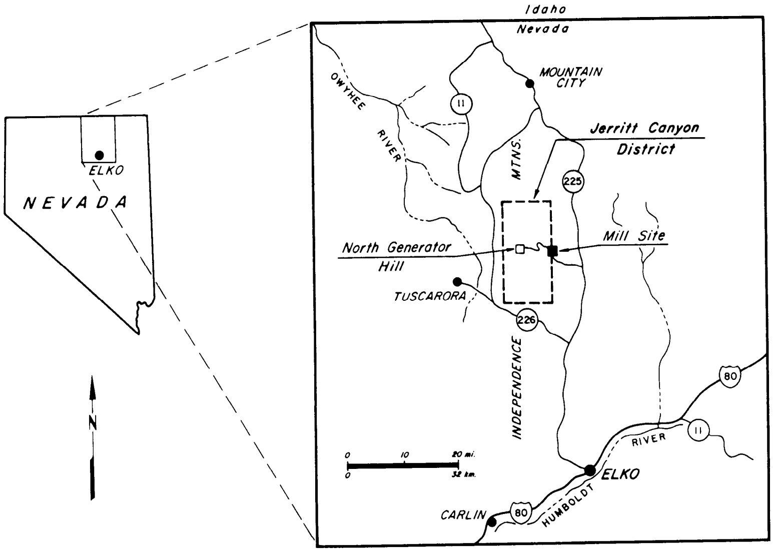

Fig. 5.8.2.1. Index map showing the Jerritt Canyon district, North Generator Hill deposit and Enfield Bell mill site.

overall recoverable reserve in excess of 1,950,650 oz (55 t) of gold. The deposit is currently being mined by open pit methods, and the gold is recovered by standard carbon-in-leach technology, which is preceded by a chlorine oxidation step. Commencing in 1989, a portion of the ore from the North Generator Hill ore body will be oxidized by passing the finely ground ore through a fluid bed roaster. This expanded capacity will give the overall Jerritt Canyon operation a capacity of 2.45 million tpy (2.22 Mt/a) of ore, which will produce between 388,000 and 423,000 oz (11 and 12 t) of gold per year for the joint venture.

This paper describes the discovery and geology of the North Generator Hill gold ore body and discusses the evolution of a geological resource to an ore reserve, with the development of evaluation techniques that led to the current ore reserve calculation practices at the Enfield Bell mine at Jerritt Canyon.

5.8.2.1 Discovery

FMC initially explored in the Independence Mountains for antimony in the summer of 1971. During 1973, FMC geologists recognized the potential for gold in the region and carried out wide-spaced soil sampling that covered, in addition to other areas, a portion of the North Generator Hill gold deposit. The soil samples were analyzed for gold by conventional fire assay techniques, and a gold anomaly was detected in the North Generator Hill area. During the 1974 field season, six NX 1.8-in. (45-mm) diameter core holes totaling 695 ft (212 m) tested the soil anomaly. Because of the difficulties in core drilling the massive jasperoidal silicification so prevalent over the deposit area, the holes did not penetrate the Hanson Creek Formation, which later proved to be the host for most of the gold mineralization in the Jerritt Canyon district. This initial six-hole drilling program encountered only sporadic low-grade intercepts of gold mineralization in the Roberts Mountains Formation. Exploration drilling did not resume on North Generator Hill until 1978.

Following geologic mapping and a comprehensive soil and rock geochemical survey carried out in 1977, several low-level geochemical anomalies and geologic targets were discovered in North Generator Hill. Drilling began in 1978, and by the summer of 1980, a minable ore reserve of 4.1 million tons (3.73 Mt) grading 0.212 oz Au/ton (7.26 g Au/t) had been defined. This initial reserve of 955,924 oz (27 t) of gold has yielded over 901,000 oz (25 t) of gold production with remaining reserves of nearly 1,057,920 oz (30 t) of gold. This increase has been achieved through a continued program of exploration drilling and engineering optimization.

Geology, Alteration, and Mineralization: The North Generator Hill gold deposit is hosted within faulted, folded, and altered sedimentary carbonate and clastic rocks of the upper and lower plates of the Roberts Mountains thrust sequence (Fig. 5.8.2.2, Hawkins, 1982). The (Ordovician/Silurian) Hanson Creek and (Silurian/Devonian) Roberts Mountains Formations form the lower plate of the thrust fault. These shelf and marginal-basinal facies carbonate rocks are truncated by the Roberts Mountains thrust fault, the hanging wall of which contains a thick sequence of deep-water, eugeosynclinal sedimentary, and volcanic facies assigned to the Ordovician Valmy Group. Displacement of these units along the Roberts Mountains thrust fault is believed to have taken place during the Antler Orogeny (Roberts et al., 1958). Subsequently, in the North Generator Hill area, these lithologies have been subjected to several pulses of folding accompanied by both high- and low-angle faulting, igneous intrusion, and, culminating with extensional tectonism, forming the Basin and Range physiography during the Miocene.

The detailed stratigraphic column of the North Generator Hill area is shown in Fig. 5.8.2.3, which has been adapted from Birak and Hawkins (1985).

The relative abundance of gold, organic carbon, and pyrite in the different rock types in North Generator Hill is shown by the bar charts in Fig. 5.8.2.3. The lengths of the bars are qualitative representations of the three components relative to the different rock types. Organic carbon and pyrite are important constituents of the North Generator Hill ore in terms of alteration and ore processing characterization.

Structure: Several tectonic events affected the rocks of the North Generator Hill area. Compressional tectonics dominated the early geologic history of the area. During the Devonian/Mississippian Antler Orogeny, eugeosynclinal, Ordovician rocks were thrust eastward over miogeosynclinal, Devonian and Silurian rocks along the Roberts Mountains thrust fault. This was followed by Permo-Triassic compression related to the Sonoma Orogeny, which resulted in the numerous east-west trending folds in the district and North Generator Hill area. Finally, Cretaceous compression, related to the Sevier Orogeny, produced major east-west faults, such as the Ridge Fault (Fig. 5.8.2.2), and broad north-south folds.

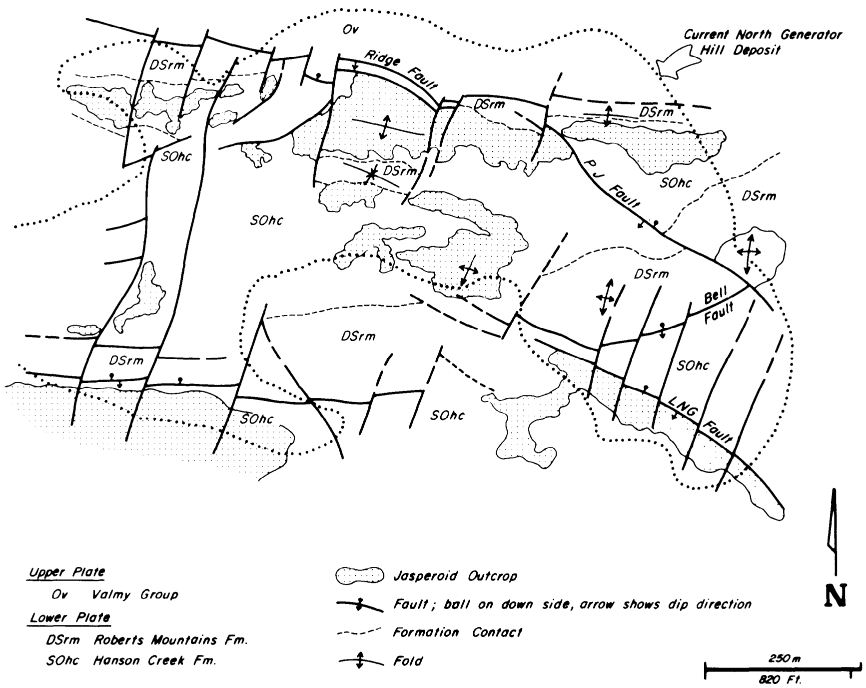

Fig. 5.8.2.2. Generalized geologic map of the North Generator Hill area.

Extensional tectonics became pronounced in the Tertiary, beginning perhaps as early as Oligocene time. The complex pattern of high-angle northeast- and northwest-trending faults, recognized in the area, are related to several episodes of Tertiary extension, culminating with Late Miocene, Basin-and-Range extension.

Alteration of the carbonate and clastic host rocks in the North Generator Hill deposit consists of extensive silicification, carbonization, decalcification, oxidation, and argillization (Birak and Hawkins, 1985). The jasperoid outcrops in the North Generator Hill area are the result of silicification and replacement of the carbonate minerals in the host rocks by quartz (Birak et al., 1987). Three stages of silicification have been documented in jasperoids from the Jerritt Canyon district and the North Generator area (Hofstra and Rowe, 1987; Northrop et al., 1987). The initial stage of silicification is believed to be the result of thermally and/or diagenetically driven connate fluids that were in chemical equilibrium with the host rocks. The second stage of silicification was structurally localized and in part coeval with gold mineralization. Isotopic evidence indicates that the second stage of silicification resulted from the mixing of deep-seated, isotopically exchanged connate fluids with meteoric waters. The final stage of silica deposition post-dates gold mineralization and may represent a continued and enhanced dilution of the second-stage, deeper-seated fluids with meteoric water. Gangue minerals, which may occur in varying abundances with all three stage of silicification, are barite, stibnite, calcite, gypsum/anhydrite, pyrite, variscite, or realgar.

In addition to silicification, carbonization and decalcification are important alteration features at North Generator Hill. Areas of carbonization and decalcification host some of the highest-grade gold mineralization observed at Jerritt Canyon. These alteration types frequently occur together and appear as black, sooty, carbon-enriched sheared zones in calcite- and dolomite-depleted limestone and siltstone host rocks. The carbon in this type of alteration consists of micron-sized discrete grains and aggregates of poorly crystalline graphite (Leventhal et al., 1987). Realgar, orpiment, and cinnabar are common gangue minerals. Carbonate veining occurs peripheral to the carbonized zones and may be remobilized from decalcified areas. Carbonization, based on paragenetic information and carbon crystallinity measurements, occurred early in the hydrothermal history of the rocks of the North Generator Hill area and may have preceded gold mineralization. Some of the carbon may have played a role in precipitating gold from subsequent hydrothermal solutions, although evidence to support this unequivocally has not been found.

Oxidation is characterized by the conversion of pyrite to iron oxides and destruction of organic carbon. The term argillization is used to describe rock that is variably enriched in clay minerals and/or sericite and nearly devoid of pyrite, organic carbon, and primary carbonate minerals. Argillization may destroy primary sedimentary textures, while oxidation alone often preserves even the most delicate of these features. Oxidation and argillization appear to be largely supergene as both are well developed at the present topographic surface, decreasing downward along favorable structures, and are the deepest along contacts, faults, and in the highly fractured cores of tight anticlinal folds. Oxidation and argillization post-date all other alteration types.

Gold occurs as discrete micron- and submicron-sized grains and grain clusters disseminated throughout the host rocks. It has been observed adhering to pyrite and organic carbon, interstitial to terrigenous grains, and included within hydrothermal barite and jasperoidal silica. The North Generator Hill area does not contain intrusive igneous rocks. As such, dating of gold mineralization by direct crosscutting relationships and igneous age-dating is not possible. However, age dating of a granodiorite stock south of the North Generator Hill deposit, coupled with structural paragenesis, suggests that gold mineralization occurred no earlier than 39.2 m.y. ago, and probably no later than mid-Miocene.

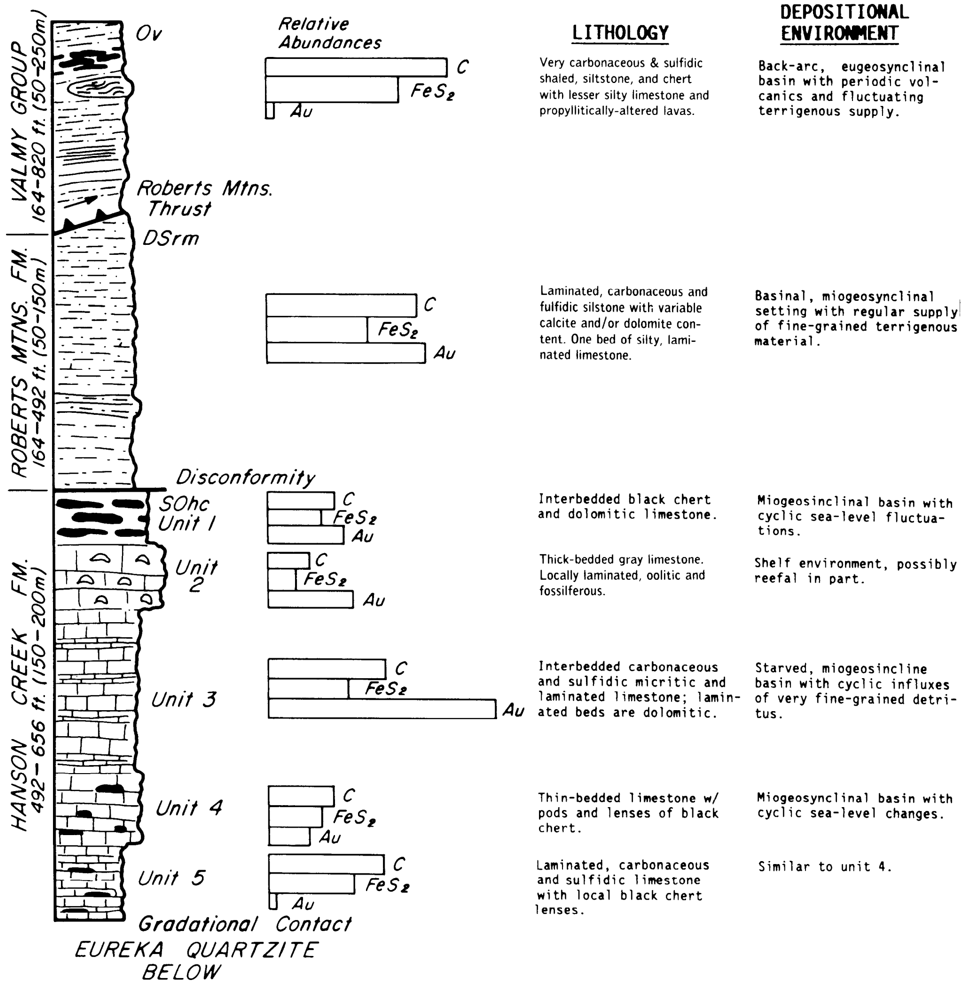

Fig. 5.8.2.3. Stratigraphic section of the North Generator Hill area.

Surface Geochemistry: Several surface geochemical sampling surveys have covered the North Generator Hill area. The earliest, in 1973, was part of a district-wide sampling program during which 135 soil samples were collected on ridges and spurs and random locations across the area that overlays the North Generator Hill gold deposit. Samples, collected with a shovel from holes dug approximately 8 to 12 in. (200 to 300 mm) deep and below the humic layer, were sieved to 80-mesh (0.18 mm) to remove organic material contamination and assayed with standard fire assay techniques. Fig. 5.8.2.4 shows the location of the 1973 soil sampling program, the resulting gold anomaly, and the outline of the North Generator Hill gold deposit. Concentrations of soil gold values greater than or equal to 0.003 oz/ton (0.09 g/t) are outlined on the map. This program was successful in defining a single multiple-sample gold anomaly in the extreme south-southeast portion that covered an area roughly 400 × 1200 ft (122 × 366 m). Several single-sample anomalies were also detected by this initial sampling program.

This initial geochemical sampling program was not an effective exploration tool for several reasons. First, the samples were assayed utilizing conventional fire assay techniques. The resulting range of detection limits was between 0.001 and 0.005 oz Au/ton (0.03 and 0.17 g Au/t). Later sampling programs determined that this detection limit and precision were inadequate. Second, at the time of the survey, sporadic Pleistocene glacial debris partially covered the subcrop of underlying mineralization. This formed a geochemically impermeable layer that covered 20% of the underlying mineralization. A third factor related to prioritization of the anomalies. In 1973, the Roberts Mountains Formation was the favored target for gold exploration in Nevada and accordingly received most of the attention in the Jerritt Canyon district. In the northern portion of the North Generator Hill area, the single-sample soil anomalies were developed over the subcrop of the Hanson Creek Formation, and these anomalies were downgraded.

Drilling: The initial exploration drilling for gold in the Generator Hill area was carried out by FMC in 1974. Six short 1.8-in. (45-mm) diameter core holes were drilled to test the gold soil anomaly over the subcrop of the Roberts Mountains Formation. Holes were not drilled deeper than the Roberts Mountains/Hanson Creek contact for the following reasons: (1) intense silicification and fracturing at the Hanson Creek/Roberts Mountains contact precluded effective core drilling, and (2) lack of knowledge of the potential of the Hanson Creek Formation to act as a host to gold mineralization. Fig. 5.8.2.4 shows the location of the initial six core holes with respect to the original gold soil anomaly and outline of the surface projection of the current ore reserves.

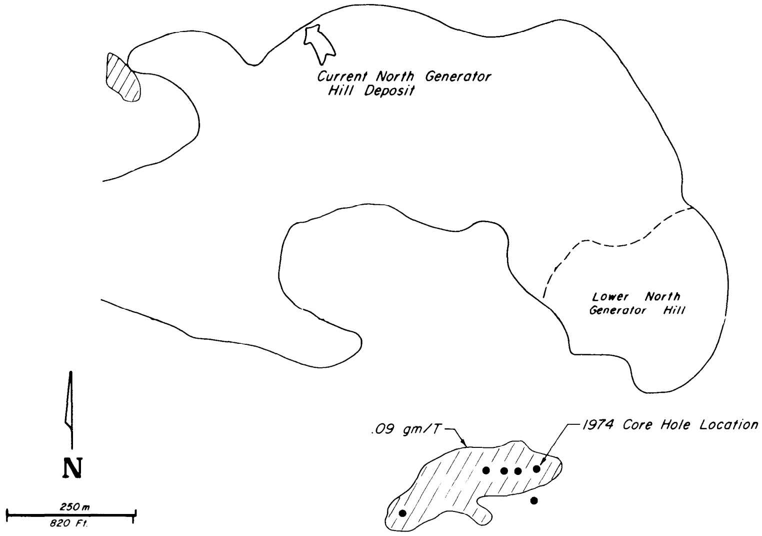

Fig. 5.8.2.4. Location of 1973 anonalous soil samples (cross-hatched areas) and first six holes drilled in North Generator Hill. Conversion factor: 1 oz/ton = 31.25 g/t.

Assay difficulties were encountered in analyzing the 1974 core drilling from Generator Hill. The initial gold assay, performed by standard sodium cyanide extraction and analysis by atomic absorption (AA), yielded accurate results only on the oxidized portions of the core. A considerable amount of the core was unoxidized, containing organic carbon and pyrite that did not allow the gold to be solubilized by sodium cyanide. It was later discovered that fire assaying or a roasting step prior to cyanidization was required to determine the gold content of the carbonaceous and sulfidic host rock. With this knowledge, all subsequent analyses were performed using either fire assaying or cyanide extraction preceded by roasting or heating the sample pulp in air at approximately 1000°F (550°C), which oxidized the carbonaceous material and sulfide minerals.

Drilling commenced in North Generator Hill in early 1978 following the identification of geochemical and geologic targets developed during the 1977 field season. In early 1978, seven widely spaced holes were drilled using rotary-percussion drilling. Six of these holes detected encouraging gold mineralization and three encountered ore-grade mineralization; > 5 ft grading 0.05 oz Au/ton (> 1.5 m grading 1.7 g Au/t). The location of these seven holes, as well as the six core holes drilled in 1974, is shown on Fig. 5.8.2.5. The most important aspect of the 1978 drilling was the discovery of gold hosted within the Hanson Creek Formation. This opened up a much larger area to exploration throughout the Jerritt Canyon district and indicated that North Generator Hill could host a significant tonnage of bulk-minable gold mineralization.

5.8.2.2 Cost of Discovery

Prior to 1977, FMC had spent approximately $1.5 million in exploration in the Jerritt Canyon district. Of that amount, it is estimated that less than $50,000 had been spent in the area of the North Generator Hill ore deposit.

Between 1978 and 1980, a total of over 95,000 ft (29 km) of drilling in approximately 200 holes had been completed in the North Generator Hill area. The total cost of this program was approximately $1.4 million.

The procedures established in the development drilling of the North Generator Hill deposit are largely followed today in our exploration and reserve definition practices throughout the Jerritt Canyon district.

5.8.2.3 Delineation Drilling and Sampling

Drilling: Three types of exploration and delineation drilling were carried out at North Generator Hill: rotary-percussion drilling with conventional circulation, rotary-percussion drilling utilizing dual wall pipe and reverse circulation, and core drilling. Core drilling was not extensively used due to the high costs associated with drilling in intensely silicified and fractured zones. Conventional circulation was used in areas yielding good sample recovery (solid ground conditions and minimal voids), and reverse circulation was used in cavernous areas or areas of broken ground. To check the quality of samples returned from rotary drilling, a series of drill twinning tests were made. These consisted of drilling proximal to several rotary drill sites with either reverse circulation rotary methods or with core. The holes were usually between 5 and 15 ft (1.5 and 4.5 m) apart. Closer spacings were not possible due to the probability of lost sample circulation from the twin hole into fractures adjoining the original hole. Overall, the comparison between the twin and original hole sample assays was acceptable. However, it was discovered that a low-grade "assay tail" of contamination that occurred in several of the conventional circulation rotary holes was not present in the reverse circulation or core twin holes. This finding resulted in a switch from conventional circulation to reverse circulation drilling in all further exploration holes at Jerritt Canyon. It was also necessary to determine if the reverse and conventional circulation drilling sampling methods were obtaining a representative sample. This was verified by collecting blow-by dust and comparing the assays with the coarse cuttings samples. The dust samples yielded gold assays that were slightly higher in grade than the coarse fraction sample assays. The overall correlation coefficient between the coarse and dust sample assays was 0.89 with the worst correlation in the < 0.02 oz/ton (< 0.69 g/t) assay range. Since the poorest comparison between dust and coarse sample assays was in the lowest grades, and since the remainder of the samples compared well, it was determined that air-circulation drill samples were acceptable.

The initial grid of exploration drilling at North Generator Hill was carried out in the spring of 1978 and consisted of 20 drillholes totaling approximately 6890 ft (2100 m), scattered across the eastern portion of the area (Fig. 5.8.2.5). These initial holes were established on a staggered 400-ft (122-m) grid to cover as much of the primary target area as possible during the

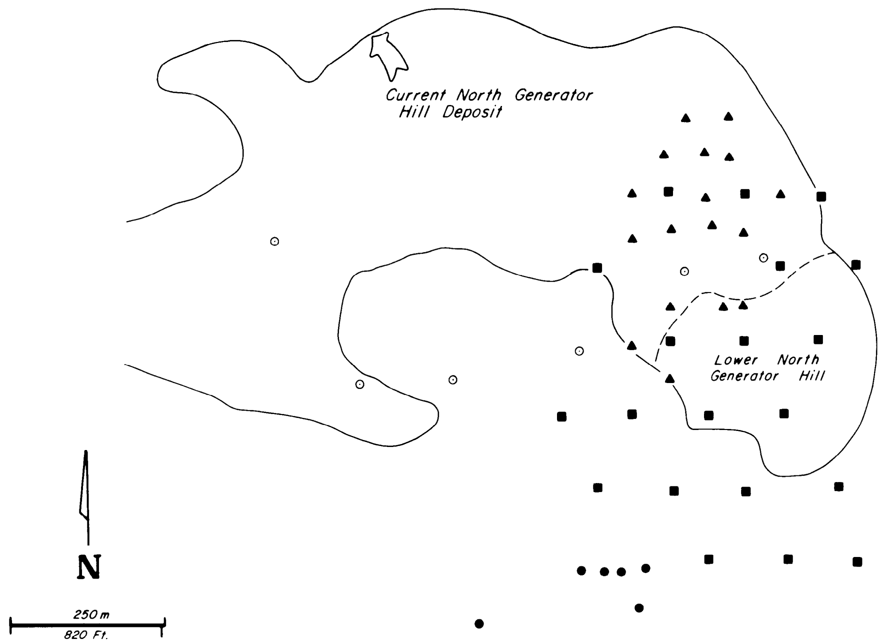

Fig. 5.8.2.5. Location of all holes drilled in North Generator Hill by the end of 1978. Denoted are 1974 exploration (solid dot), early 1978 exploration (open dot), 1978 400-ft (122-m) grid (solid square) and 1978 200-ft (61-m) grid (triangles) holes.

early portion of the field season. Seven of the 20 holes encountered encouraging thicknesses and grades of gold mineralization, with the best single intercept being 85 ft (26 m) averaging 0.264 oz Au/ton (9.05 g Au/t).

Because of the wide-spaced nature of the gold mineralization discovered in this first-pass drilling and the significance of some of the grades encountered, fill-in drilling on a 200-ft (61-m) grid was initiated. Seventeen holes totaling 4626 ft (1410 m) were completed (Fig. 5.8.2.5), seven of which intersected ore-grade gold mineralization. Although the percentage of ore-grade holes marginally increased from the first to the second drilling phase, the area of gold mineralization increased significantly, and the decision was made to accelerate the program and continue drilling through the winter of 1978-1979. The drill grid was modified in this accelerated program so that holes would form a five-spot pattern with a hole at each of the corners of a 200-ft (61-m) (east-west) and 400-ft (122-m) (north-south) rectangle and one hole in the center. Drilling continued throughout the spring and summer of 1979, and by the end of that year, a total of 226 holes had been completed totaling over 68,898 ft (21 km) of drilling. Additional delineation drilling continued on a five-spot grid during the summer seasons of 1980, 1981, and 1982.

During 1983, the drill grid was closed to 100-ft (30.5-m) hole spacings, and this tighter control resulted in the addition of approximately 820,120 tons (744 kt) of reserves containing an average grade of 0.244 oz/ton (8.36 g/t). The tighter grid was deemed necessary because of abrupt variations in the continuity and grade of mineralization and the need to determine more accurately grade boundaries for mine planning purposes. Reserve development drilling today continues on a 100-ft (30.5-m) grid, as this has been determined to be the optimum spacing for mine design requirements and ore-zone definition.

Sampling: In 1978, drill sampling was standardized to a regular 5-ft (1.5-m) interval. Previously, sampling intervals were established subjectively and based on logged geologic intervals of nonuniform lengths.

The 5-ft (1.5-m) interval for a rotary-percussion drillhole 5 in. (127 mm) in diameter would theoretically generate 99 lb (45 kg) of sample. However, fractured and open ground in addition to unrecovered dust resulted in considerably less sample being collected for each interval. At the drill site, the sample was reduced by passing it over a single tier riffle-type splitter with adjustable splitting bars. A technician at the drill site was responsible for the collection, splitting, and labeling of the samples. Three samples were collected from each interval: an 8-lb (3.6-kg) assay sample; an 8-lb (3.6-kg) duplicate assay/metallurgical sample; and a 10.6-oz (300-g) geologic sample. The geologic sample was collected in a clear plastic bottle and stored in a conventional cardboard core box. In this manner, the sample was readily retrievable for geologic logging.

All samples were initially analyzed for gold using AA preceded by an oxidative roasting. Samples returning > 0.02 oz Au/ton (> 0.68 g Au/t) were additionally analyzed with a dilute cyanide extraction not preceded by roasting to determine the amount of readily “leachable” gold as a metallurgical control. Assays returning ≥ 0.02 oz Au/ton (≥ 0.68 g Au/t) were also fire assayed as an additional check on the roasted AA assay. Using commercial laboratories, assay turn-around time varied between two and four weeks for all three assays.

5.8.2.4 Calculation of Reserves/Resources

A variety of methods, including cross sections, plan and block-model polygons, and geostatistics, have been used to estimate the reserves in the North Generator Hill deposit.

Initially, the geologic resource was determined utilizing cross-sectional polygons as follows:

Table 5.8.2.1. North Generator Hill Reserve Calculations, March 1979

Method	Tons (Tonnes)	Grade, oz/ton (g/t)	Total oz (kg) Au	Cutoff grade,[a] oz/ton (g/t)	
Cross sections	1,308,113	0.25	357,114	0.04	(O)
	(1,186,700)	(8.53)	(10,124)	(1.37)	(O)
				0.09	(C)
				(3.11)	(C)
Plan polygons	1,159,521	0.28	352,140	0.04	(O)
	(1,051,900)	9.49	(9,983)	(1.37)	(O)
				0.10	(C)
				(3.43)	(C)
Block model polygons	901,360	0.25	246,071	0.05	(O)
	(817,700)	(8.53)	(6,976)	(1.61)	(O)
				0.12	(C)
				(4.01)	(C)

[a](O) = oxide ore; (C) = carbonaceous ore.

1. North-south oriented cross sections were compiled at a scale of 1 in. = 50 ft (10 mm = 6.0 m) and spaced 100 ft (30.5 m) apart. Vertical drillholes were plotted on the sections.
2. Drillhole gold assays were subdivided into oxidized and carbonaceous metallurgical types.
3. Cutoff grades were estimated to be 0.04 and 0.10 oz Au/ton (1.37 and 3.43 g Au/t) for oxidized and carbonaceous ores, respectively.
4. A 10-ft (3-m) minimal drill intercept thickness (anticipated bench height) was used for grade averaging.
5. Gold grades were projected halfway to sections on either side of the reserve section.
6. Cross-sectional areas were determined by manual planimetering.
7. The areas were multiplied by 100 ft (30.5 m) to derive volumes.
8. A tonnage factor of 14 ft^3/ton (0.40 m^3/t) was applied.

In addition to the cross-sectional calculation method discussed previously, both plan-view polygons and block model polygons were used to prepare tonnage and grade estimates as a check to the cross-sectional method. These results are tabulated on Table 5.8.2.1.

Both cross-sectional and plan-view polygon ore reserve calculations indicated the same relative amount of contained ounces of gold; however, tonnage calculations varied by 13% and grade by 11%. These estimates were not conditioned by mining and processing costs, ore recoveries, stripping ratio, or pit design and were only used to model the overall picture of the in-situ gold resource.

The third reserve calculation method, block-model polygons, was derived from regularly shaped and sized computer-generated blocks throughout the deposit. The block model was created by compositing drillhole assays into 15-ft (4.6-m) lengths corresponding to the planned mining bench elevations. The deposit was then subdivided into a three-dimensional grid of 50 × 50 × 15 ft (15.2 × 15.2 × 4.6 m). The composited assay data was merged with the block file so that grades could be assigned using horizontal polygonal-extensional methods. Grades were assigned from the composited grade of the closest exploration drillhole to the block. In assigning grades, a search radius of 100 ft (30.5 m) was used except at the periphery of the deposit where a search radius of 50 ft (15.2 m) was applied. This minimized the overextension of reserves at the periphery of the reserve.

In summary, the tonnage and grade differences expressed on Table 5.8.2.1 by the various ore reserve calculation methods are attributable to

1. Varying cutoff grades assigned to the oxide and carbonaceous ores.
2. Vertical compositing to uniform bench thickness of 15 ft (4.6 m) in the block model reserve.
3. Mine design limitations which exclude some low-grade intercepts or isolated high-grades as uneconomically minable.
4. Search radius of 50 ft (15.2 m) on the edges of the ore zone.
5. Data from eight additional drillholes were included in the cross-sectional and plan-view polygonal estimates and not included in the computer-generated block model.

None of the reserve estimations were final at this point as the zone was open to further drilling in a number of directions and production cost estimates were preliminary. However, the minable reserve analysis generated from the block-model polygons indicated an economically exploitable body of gold mineralization and justified initiation of a feasibility study and continued exploration. The discrepancy between the reserve and resource estimates by the various calculation methods was anticipated and did not alter the decision to proceed with plans for development.

Until late 1979, the polygonal block-model reserve estimates were believed to be representative of future mine production. As only a few disseminated gold deposits were being mined at that time, potential ore estimation problems were not well understood. However, several other operations were experiencing unanticipated dilution, and this potential problem was addressed by the project staff at Jerritt Canyon.

To better assess the potential for grade dilution during mining at North Generator Hill, three additional ore-reserve calculation procedures were made: (1) insertion of an arbitrary dilution factor, (2) a kriged geostatistical approach, and (3) a modified polygonal block reserve utilizing a capped maximum assay value.

The dilution model involved inflating the tonnage factor by 15% and keeping the contained ounces constant. Subsequent production experience proved that a uniform dilution factor provided insufficient correction for the mining dilution and the method was abandoned. In 1980, Fluor Mining and Metals was contracted to make a kriged geostatistical study of the Lower North Generator Hill portion of the deposit. This is the same area that was previously evaluated with cross-sectional, plan-view, and block-model polygons in 1979. The objectives of this study were to determine (1) how much error could be expected in the polygonal block model, and (2) if the error could be minimized by grade-control procedures. A series of variograms was compiled in an attempt to model the entire ore zone, grade selective portions, and geologically selective portions. Using a

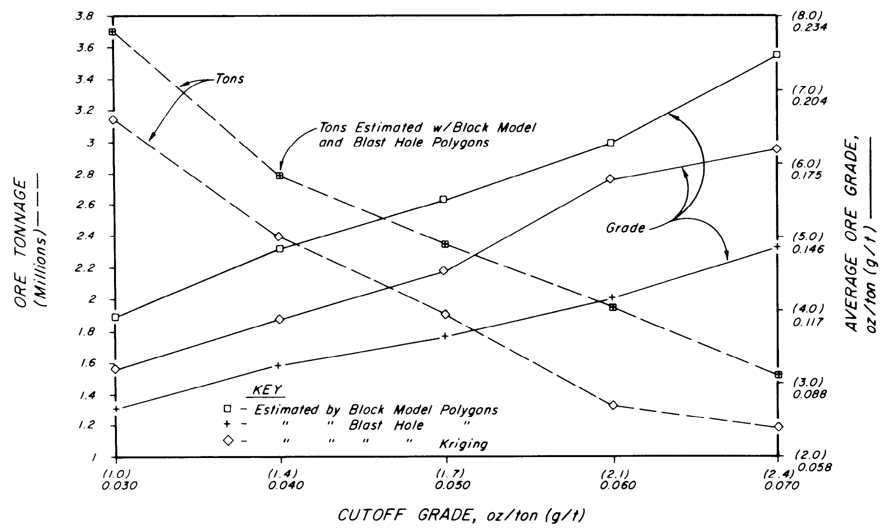

Fig. 5.8.2.6. 1980 North Generator Hill geostatistical evaluation, grade and tons, with different estimation methods.

volume-variance relationship, it was possible to estimate the amount of ore above any given cutoff grade that would be recovered with blasthole sampling and mining. Metallurgical and mining cost parameters were not considered in this exercise.

Resource estimates of tonnage and grade using block-model polygons were compared to estimated minable tonnage and grade using blasthole polygons and blasthole kriging techniques. The data in Fig. 5.8.2.6 show that the block-model polygons overestimated grade when compared to the grade selected using blasthole polygons and overestimated both tonnage and grade compared to kriged blasthole estimations. Kriging yielded grades between those of the block model polygons and the blasthole polygons. Based on these findings, additional series of geostatistical evaluations were designed to address the impact of selective mining unit (SMU) sizes and the geometry of these units.

In an effort to determine the best reserve calculation method, a second modification to the polygon block reserves was made that consisted of capping the block grades to a maximum value. As was later determined, each deposit in the district would have its own maximum capped grade. In North Generator Hill, the capped grade was determined to be 0.45 oz Au/ton (15.42 g Au/t). Capping has been effective in reducing the overestimation of the spatial extent of high compositive drillhole grades while not affecting the lower grades. The method for determining this value was as follows:

1. All composited exploration drillhole assays were sequentially ranked by decreasing grade.
2. Anomalously high-grade samples were eliminated from the sample population. In the case of North Generator Hill, values in excess of 0.75 oz Au/ton (25.7 g Au/t) were eliminated.
3. Subsets were established of the remaining rank-ordered composite data into groups of different numbers of composites (i.e., 4, 5, or 6 composites in a group).
4. The composited gold grades were averaged for each of the different group sizes.
5. The average composited gold grade was plotted against the sequential group number.
6. The major inflection point of the curve was visually established and the capped grade determined.

Cumulative frequency plots have also been used in the mining industry to determine capped grades. However, unlike cumulative frequency, the rank-order method is not as strongly affected by the range of the data set. The rank-order cap grade calculation was determined to be the most accurate grade control and is the method currently practiced at Jerritt Canyon.

All the mine plans and reserve estimates for the deposits in the Jerritt Canyon District are made using a US$450/oz gold price, current or forecasted operating and capital costs, recovery parameters, and open pit slope angles determined by geotechnical studies. The average pit slope angle is approximately 42º. Ore reserves are calculated at the end of each year following completion of the year's exploration and delineation drilling program and the deduction of the tons (t) of ore and ounces (kg) of gold produced.

Table 5.8.2.2 lists North Generator Hill ore reserves as tabulated at the end of each year since 1979. The overall increase has been accomplished by continued exploration and development drilling, ore processing revisions allowing a reduction of cutoff grade, and a bench height increase. Heap leaching became a viable process for North Generator Hill ores in 1985, thus adding to the ore reserve inventory.

Reserve estimates are only as good as production experience will allow. North Generator Hill production has been tracked against local reserve estimates since 1981. Since conversion from 15- to 20-ft (4.6- to 6.1-m) bench heights, block model polygons in North Generator Hill have been found to underestimate mined tonnage by 14.9% and overestimate grade by 9.4%. Obviously, the variance is affected by both the amount of low-grade ore mined vs. predicted and the accuracy of the grade estimate on high-grade exploration composites. Because of the impracticality of drilling a regular grid of delineation holes tighter than 100 ft (30.5 m) apart, low-grade ore will often be more abundant than predicted. In addition, low-grade ore often occurs blanketing high-grade and in separate ore zones without a high-grade core. Usually the latter case is the most difficult to define with delineation drilling. Fig. 5.8.2.7 shows the geology of a typical mining level of North Generator Hill superimposed with blasthole sample grade contours for comparison. Exploration drillhole locations are also denoted in Fig. 5.8.2.7. Note that the mineralization consists of several separate ore zones. Ore is frequently controlled by high- and low-angle structures as well as in the cores of anticlinal folds. The individual ore zones are much more

Table 5.8.2.2. End-of-Year Ore Reserves, North Generator Hill; Ore Reserves (Proven and Probable)

Year-end	Tons (tonnes)	Grade, oz Au/ton (g Au/t)	Contained gold, oz (kg)	Mined oz (kg)	Cumulative mined + reserves, oz Au (kg Au)
1979	3,464,895	0.19	705,479		705,479
	(3,143,300)	(6.37)	(20,000)		(20,000)
1980	4,119,337	0.21	955,924		955,924
	(3,737,000)	(7.23)	(27,100)		(27,100)
1981	4,508,012	0.21	1,044,109	63,493	1,107,602
	(4,089,600)	(7.23)	(29,600)	(1,800)	(31,400)
1982	3,884,876	0.21	892,431	186,952	1,142,876
	(3,524,300)	(7.16)	(25,300)	(5,300)	(32,400)
1983	5,611,536	0.20	1,231,061	8,818	1,492,089
	(5,090,700)	(6.85)	(34,900)	(250)	(42,300)
1984	5,665,329	0.19	1,209,897		1,470,924
	(5,139,500)	(6.68)	(34,300)		(41,700)
1985	6,527,557	0.17	1,227,534	63,493	1,552,054
	(5,921,700)	(5.86)	(34,800)	(1,800)	(44,000)
1986	6,247,460	0.16	1,149,931	165,788	1,640,239
	(5,667,600)	(5.58)	(32,600)	(4,700)	(46,500)
1987	6,626,324	0.15	1,125,239	190,479	1,806,027
	(6,011,300)	(5.31)	(31,900)	(5,400)	(51,200)
1988	7,434,428	0.13	1,047,637	222,226	1,950,650
	(6,744,400)	(4.42)	(29,700)	(6,300)	(55,300)

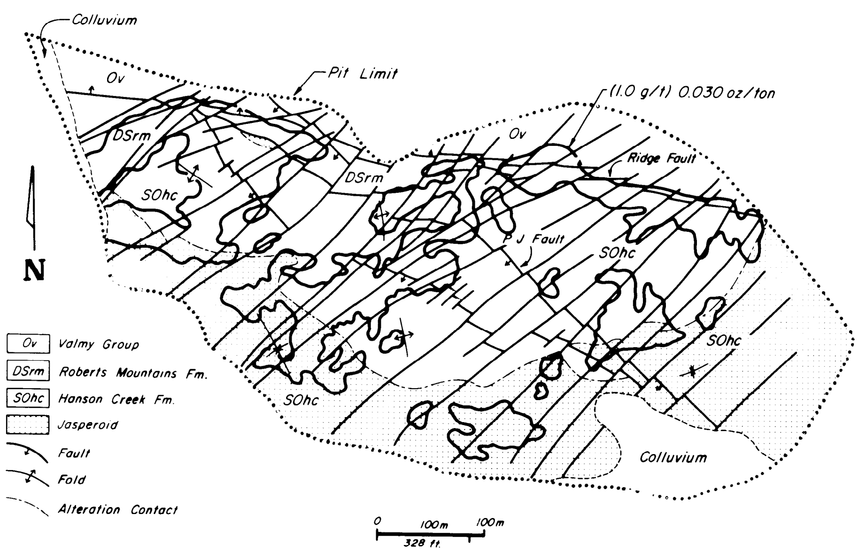

Fig. 5.8.2.7. Geologic map of the 7820-ft (2384-m) level in North Generator Hill with 0.030 oz/ton (1.0 g/t) contour of the blasthole value.

continuous horizontally than vertically. Frequently, the mine geologist will drill holes to fill in geologically critical ore zones for grade and volume definition. These holes will be located to yield the most geologic information on ore zone size and shape rather than on a predetermined grid.

Grade Control: Calculation of the grade of a selective mine unit is the fundamental role of a mine grade control program; the SMU being a function of production requirements balanced against dilution. Production requirements include milling rate, relative cost savings between different mine bench heights, rock fragmentation characteristics, and ore to waste ratios. At North Generator Hill, dilution consists of both mining and estimation types.

Mining dilution refers to the intermingling of ore and waste by raveling at ore and waste boundaries, drill sampling errors, shovel or loader operator error, or truck driver misclassification. It is a relatively fixed problem that must be addressed with proper training and supervision. As such, Independence Mining Co. does not apply a mining dilution correction factor into grade control.

Estimation dilution is a bigger problem than mining dilution and one that can be modeled and mitigated by use of geostatistics.

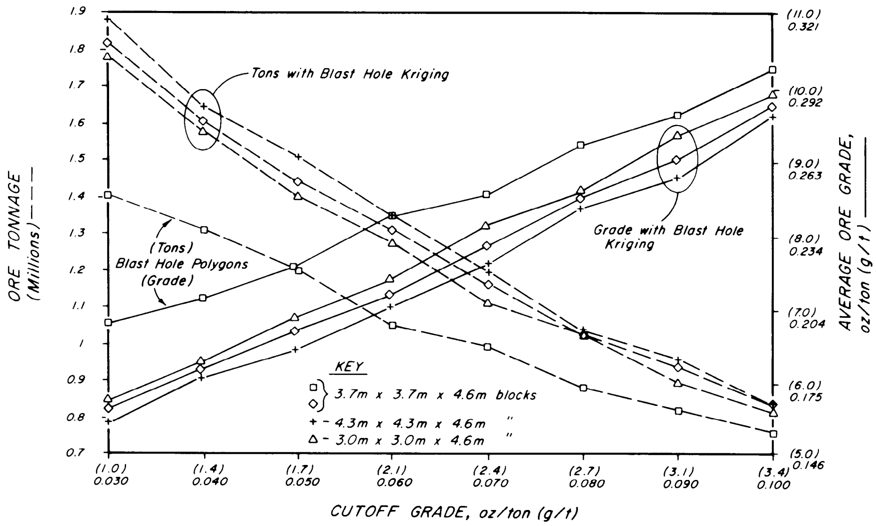

Fig. 5.8.2.8. 1980 North Generator Hill: recoverable grade and tons with different selection methods and selective mining unit sizes.

Estimation dilution refers to the variance inherent in the selection method, such as kriging or polygons, when compared to the actual grade mined for the same volume. These errors are usually normally distributed and their effects can be quantified by normal probability laws. Of the two selection methods, only kriging will minimize the estimation variance and thereby reduce the effects of estimation dilution.

The effect on recoverable reserves by both SMU size and selection method was examined in detail during preproduction and early production phases of North Generator Hill development. Fig. 5.8.2.8 graphically displays how recoverable tons and grades are affected by decreasing the horizontal dimension of the SMU. These curves were modeled using kriging and the volume-variance relationship taken from the deposit variogram. The volume-variance relationship relates the estimation variance to block volume: large blocks will be less variable than small blocks, but will result in a lower average grade. After considering the three curves and the estimated mining costs for each horizontal SMU size, the 12 × 12 × 15-ft (3.7 × 3.7 × 4.6-m) SMU size was chosen (Fig. 5.8.2.8, bold lines). As a comparison, the recovered tons and grade with central blasthole polygon of 12 × 12-ft (3.7 × 3.7-m) dimension are also displayed. Blasthole polygons would estimate less recovered ore tons but at significantly higher grade. This follows one of the basic axioms of geostatistics applied to mining: polygons tend to underestimate the spatial influence of low-grade samples and overestimate the spatial influence of the high-grade ones.

Metallurgical Evaluation: North Generator Hill ores were part of a district-wide metallurgical evaluation of all the ores in the Jerritt Canyon district. This decision ensured the timely completion of the mine and mill feasibility study for the entire Jerritt Canyon project. However, implementation of standard sampling and analytical procedures occurred much earlier and formed the foundation for later metallurgical tests.

Pre-metallurgical work began in early 1977 with a change in the assay procedure of exploration drill samples. The first change involved utilization of either a roast AA or fire assay for total gold determination. Both methods produced more accurate measures of the total in situ gold grades than the no-roast AA method used previously. Later, fire assaying replaced the roast AA assay for total gold analysis, the accuracy of which was dependent upon amount of sulfide-sulfur, organic carbon, and carbonate minerals in the ore samples (constituents that do not hinder accurate fire assaying). The no-roast AA assay, performed on samples with fire assay values 0.02 oz Au/ton (0.68 g Au/t), was used only to give an indication of the "leachability" of the total in situ gold. This measure, expressed by the ratio of the no-roast AA assay to fire assay, is referred to as the "leach ratio." Samples with leach ratios ≥ 0.60 were classified as oxidized and those < 0.60 were carbonaceous. Thus the total inventory of mineralized rock could be metallurgically categorized.

Based upon leach ratios, it was clear that the North Generator Hill carbonaceous and oxidized ores were quite intermixed. A limited program of core drilling was conducted in 1978 and 1979 to (1) provide pristine samples of different ores for metallurgy and grinding tests and (2) enable a more detailed geologic examination of the ore zones. All of the core, roughly 1.8 in. (45 mm) in diameter, was photographed, weighed, and split in half. Photographs provided a record of the core, while weight measurements facilitated bulk density calculations. After splitting and assaying, the geologist logged and selectively sampled the core for specimens of typical and unusual ore, waste, and gangue mineralogy. In addition, measurements of rock quality (RQD) provided information for future geotechnical studies pertinent to mine designing. Last, mineralized sections of the remaining core became the stock from which grinding and more accurate leachability tests were performed.

Because of the high costs, relative to rotary drilling, core drilling for metallurgical testing was site- and goal-specific. To decrease the costs of metallurgical sampling, a program of rotary drill bulk-sampling was begun in 1978. In addition to the three assay and geologic samples collected from every 5 ft (1.5 m) of rotary drilling, a large sample weighing approximately 40 lb (18.2 kg) was also collected. Bulk samples that contained ore-grade gold mineralization were retained for metallurgy. Bench-scale tests performed with the drill bulk samples of varying leach ratios yielded a relationship to leachability that was used in subsequent mill designs and mine planning. The tests involved pulverizing the drill sample to approximately 98% passing 0.008 in. (0.208 mm), followed by rolling agitation of 1.3 lb (600 g) of ore for 20 hr in the presence of 0.2 oz/a (0.6 g/a) NaCN solution and 7 oz (20 g) of activated charcoal. The gold recovery was

Table 5.8.2.3. Mined vs. Milled Ore; Total Jerritt Canyon (1981 through 1988)

	Mined	Milled and stockpiled	% Difference
Tons of ore (tonnes of ore)	10,042,497 (9,110,440)	10,231,874 (9,282,200)	+3.6
Grade, oz Au/ton (g Au/t)	0.23 (7.74)	0.21 (7.35)	-5.2
Oz Au (kg Au)	2,491,646 (70,637)	2,444,803 (69,309)	-1.9

determined by comparing the total gold in the feed sample to that remaining in the pulp after leaching. The degree of leachability, a function of several geologic characteristics of the ore, was affected most by the amount of organic carbon in the ore. Tests indicated that at leach ratios of 0.60, mill-modeled recoveries would be at least 75%. The ratio of oxidized ore to carbonaceous ore was roughly 50:50, and the mill feed rate and recovery circuit was designed accordingly.

A significant portion of the ore in the Jerritt Canyon district and North Generator Hill was not only unamenable to straight cyanide leaching but would scavange gold from gold-bearing cyanide solutions. This phenomenon, termed "preg-robbing," was known to exist in other disseminated gold deposits in Nevada. However, neither the strength nor extent of the preg-robbing Jerritt Canyon ores was known. A simple analytical test helped put the problem into perspective. The test was similar to a standard no-roast AA assay with the exception that a gold-bearing cyanide solution was used as the lixiviant instead of pure cyanide. The gold concentration of the pregnant solution was 0.29 oz/ton (10 g/t). Atomic absorption analysis of the final leach liquor, after contact with the drill sample, yielded a gold value that was a percentage of the total gold content of the sample plus the known standard. This was expressed as the percentage of gold robbed by organic carbon constituents of the ore. Leach ratios could not provide this information alone due to inherent inaccuracies in the no-roast AA assay. The work revealed that below leach ratios of 0.10, carbonaceous ore samples would rob between 97 and 100% of the total available gold; the actual limits of the preg-robbing were not measured, but are finite. Consequently, research began to find a method to beneficiate the carbonaceous ores in Jerritt Canyon. The result was a precursor to the chemical oxidation step, using chlorine, that is used on selected ores from the Enfield Bell mill today.

Metallurgical Balance: An accurate comparison of ore tons mined from North Generator Hill vs. milled is difficult because the Jerritt Canyon mill simultaneously processes ore from several deposits. All of the deposits in the Jerritt Canyon district have different ore-to-waste ratios. This, coupled with the fact that the mining operation is located approximately 7 mi (11.3 km) from the mill site, precludes direct feed of ore into the primary crusher. Mill-site ore stockpiles and several mine-site stockpiles must be used to maintain a balanced feed to the mill while keeping pace with waste stripping requirements. A comparison of mined to milled ore tons is performed on a year-to-year basis as a check of the grade-control program.

The comparison of tons mined to tons milled for the entire Jerritt Canyon project, from the beginning of production to the end of 1988, is depicted in Table 5.8.2.3.

This analysis shows that the mill at Jerritt Canyon receives a greater tonnage and lower-grade material than is produced by the mining operations. This variance reflects the total cumulative project dilution incurred since the commencement of production. These errors resulting in the dilution are a function of inaccuracies in mill throughput calculation, stockpile volume calculation errors, ore misclassification, assay variability, bulk-density variations, bank-volume calculation errors, and estimation variances inherent to the grade-control program. These overall variations are considered to be within the tolerance of anticipated project economics and in general reflect a high level of precision in the accuracy of grade determination and control in the North Generator Hill ore deposit. As each ore deposit will have its own characteristics, it is important that procedures for ore grade determination and mining control be developed in a flexible manner that is responsive to the geological and metallurgical characteristics of the individual deposit.

5.8.2.5 Conclusions and Current Practice

The North Generator Hill gold deposit is a typical "Carlin-type" disseminated gold deposit hosted in Paleozoic sedimentary rocks. The deposit was discovered by a combination of geological mapping, surface soil geochemistry, and rotary-percussion drilling. The first prospecting work in the area, based on a model that assumed the Roberts Mountains Formation to be the sole host to ore-grade gold mineralization, failed to test the potential of the deposit as it is currently known. Later review and reassessment of this initial exploration philosophy, as well as refinements in low-level gold analyses and rotary drilling methods, were successful in the discovery of the deposit. Yearly exploration and development costs reflect the changing priorities of the deposit spurred by these reviews and reassessments. Table 5.8.2.4 lists the yearly costs incurred in the North Generator Hill area and yearly footage of exploratory and development drilling.

The yearly costs include those typically incurred in an exploration program such as drilling, road and drill-pad construction, geochemical sampling and analysis, salaries, land payments, materials, and miscellaneous items. Costs after 1980 are not included in Table 5.8.2.4, even though delineation drilling is on-going, since the deposit was put into full production in 1981.

The North Generator Hill ore deposit is mined by open pit methods utilizing 20-ft (6.1-m) benches. Benches, drilled using rotary, down-hole hammer drills, are blasted with ANFO and excavated by hydraulic shovels and mechanical-drive trucks. Samples from blasthole drilling are assayed for total gold content and cyanide soluable gold content. The blastholes, drilled on 14 $\times$ 14-ft (4.3 $\times$ 4.3-m) square grids, define the current minimum selective mining unit size; the corners of the SMU are defined by blastholes. The blasthole gold values are used to calculate the grade of the SMUs by averaging the sample assay values of the four holes in a single SMU. This system was designed using geostatistics to determine the effect that different blasthole geometries and spacings would have on recoverable reserves. Compared to actual milled production, this grade-control system works quite well with less than 2% error on tonnage and not more than 4% error on grade.

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Table 5.8.2.4. Exploration and Development Costs (U.S. Dollars)

Year	Total Cost	Ft (m) Drilled	$/Ft ($/m)
1973-76	$ 50,000.00 [a]	695 (212)	$71.94 ($235)
1977	41,743.20	650 (198)	$64.22 ($211)
1978	229,458.06	16,150 (4,921)	$14.21 ($ 47)
1979	761,996.12	53,630 (16,348)	$14.21 ($ 47)
1980 [b]	1,425,489.80	26,125 (7,964)	$13.10 ($ 43)
Total	$1,425,489.80	97,250 (29,642)	$14.66 ($ 48)

[a]Estimated incurred costs.
[b]Mining began in 1981.

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5.8.3 EXPLORATION AND DEVELOPMENT GEOLOGY OF THE TWENTYMILE PARK UNDERGROUND COAL DEPOSITS, NORTHWEST COLORADO

R.G. MATEER AND J.M. MERCIER

This case study focuses on the development of a coal resource and covers the exploration effort required to identify and evaluate an underground coal reserve in northwest Colorado. It further describes the key geologic factors and considerations for successful underground mine design and production planning.

The area of interest, Twentymile Park, is located in Routt County in northwest Colorado. Cyprus Coal Co., a wholly owned subsidiary of Cyprus Minerals Co., controls 20,400 acres (82.6 Mm^2) of coal lands in the area. Cyprus acquired the property through its acquisition of Getty Coal Co. in 1985. When acquired, the property assets included an operating surface mine with attendant surface facilities and a small pilot mine (Foidel Creek mine) with federal, state, and private coal leases comprising the Twentymile Park underground reserve. In early 1981, studies indicated the Twentymile area contained a potential for over 300 million tons (272 Mt) of in-place, underground minable coal of excellent quality. An earlier 1979 test mine in the underground reserve, however, encountered adverse geologic conditions including multiple small faults, poor roof and floor rock, and high water inflow rates. Because of the apparent size and high quality of the underground reserve, a concerted effort, the Twentymile Park Project, was put forth to address the complex subsurface geology and to outline, as expeditiously as possible, an economically minable reserve. A primary focus of the Twentymile Park Project was to enhance confidence levels on various aspects of resource minability and provide accurate information for subsequent mine planning and development.

Cyprus Yampa Valley Coal Corp. (subsidiary of Cyprus Coal Co.), is in the final stages of surface reclamation due to surface coal reserve depletion. After several years of exploration, planning, and development, the Twentymile underground reserve successfully produced moderate development tonnages, which were marketed to regional utility companies. A longwall face was installed in the mine during 1989. Average raw tons per longwall shift for the first six months of 1991 averaged over 6300.

5.8.3.1 Introduction

Twentymile Park is located approximately 150 mi (240 km) northwest of Denver, CO. Access is from Steamboat Springs, 20 mi (32 km) by US Highway 40 and Colorado State Highway 131 to Oak Creek, CO, thence by paved county roads to the facilities of Cyprus Colorado Yampa Coal Co. A spur of the Denver and Rio Grande Western Railroad, present along the eastern margin of the area, ends at the Cyprus Colorado Yampa loadout facilities at the southern end of the Twentymile Park area (Fig. 5.8.3.1) shows the location of the Twentymile Park area and Cyprus' coal operations).

Twentymile Park is located within the Yampa Coalfield, which is approximately 600 mi^2 (1554 km^2) in area and is bounded on the east by the crystalline core of the Park Range and on the west by volcanic rocks of the White River Plateau. Topography in the area consists of "parks" (semi-enclosed basins), rolling and sloping plains, resistant bounding cliffs, and incised gulches developed above gentle folds. Surrounding the area are the rugged Elkhorn Mountains to the north and the Park Range to the east. Rough rolling hills to the west comprise the Williams Fork Mountains. Major drainage is supplied by meandering tributaries of the Yampa River. Elevations range from 6600 to 7600 ft (2012 to 2316 m) above sea level.

The climate of the Twentymile Park area is semi-arid with an annual precipitation of approximately 16 in. (401 mm) per year. A significant amount of this precipitation is in the form of snow.

Exposed slopes and rolling plains are vegetated with scrub grasses, brush, or are dry-farmed. Soil cover in the project area is generally thick (10 to 30 in. or 250 to 760 mm) and yields crops of winter wheat and barley.

Cyprus' Twentymile Park property has been subdivided into three generalized blocks (Fig. 5.8.3.1). The Tow Creek area encompasses the northern portion of the project area and consists of 11,000 acres (44.5 Mm^2) of private and state leases. The Middle Creek area includes the old Middle Creek test mine and adjacent past surface mine operations on the eastern border of Twentymile Park. Approximately 3200 acres (13 Mm^2) are

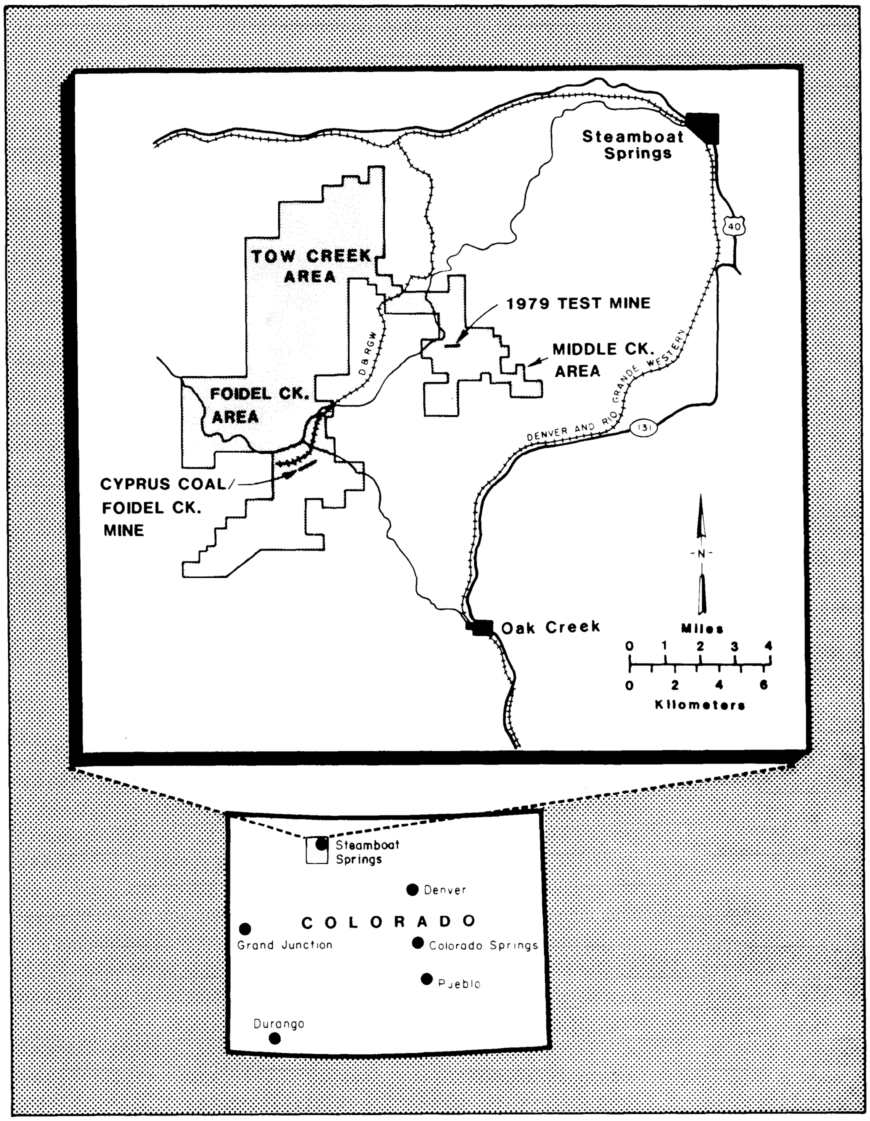

Fig. 5.8.3.1. Location map of the Twentymile Park area, Routt County, CO.

controlled through federal, state, and private leases in the Middle Creek area. The Foidel Creek area is located in the southern portion of Twentymile Park and contains approximately 8300 acres (33.6 Mm^2) of federal, state, and private leases. The entries for Cyprus' Foidel Creek mine are located in this area, with access to the north through a strip pit highwall left open for such purpose. This case study addresses the exploration effort and the resultant geologic considerations for the development of this deep mine. Numerous company reports summarize in detail the various geological and engineering aspects of this study especially Getty and Cyprus internal reports (Anon., 1983; Anon., 1985-1988) and Norwest Resource Consultants reports (Anon., 1982).

5.8.3.2 Geologic Overview

Stratigraphy: Coal-bearing strata within the project area include the Illes and Williams Fork Formations. The major coal seams are found within the lower portion of the Williams Fork Formation. These coal-bearing strata are enclosed within a stratigraphic sequence that is bounded by the massive Trout Creek and the Twentymile Sandstones (Fig. 5.8.3.2). These sandstones form massive cliffs along the margins of the Twentymile Park Basin. Intervening strata between the massive sandstones erode to form gentle slopes, which are characterized by poorly exposed outcrops. Overlying the Twentymile sandstone is the Holderness Member, a shoreline sequence that, in turn, is capped by the marine Lewis Shale.

The Wadge seam and the Upper and Lower Wolf Creek seams are the seams of economic interest (Fig. 5.8.3.2). The Wadge seam, the focus of this exploration and mine development effort, is extremely consistent in both thickness and lateral continuity. Across the main part of the Twentymile Park Basin, the Wadge seam ranges in thickness from 8 ft (2.4 m) to over 13 ft (4.0 m), averaging almost 10 ft (3.1 m). The seam crops out along the margins of the basin and reaches a maximum depth of 1800 ft (549 m) in the basinal center. The Wadge has been extensively mined in the past by surface methods.

The Wolf Creek seams are located 140 to 190 ft (43 to 58 m) below the Wadge seam and vary in thickness from less than 1 to over 13 ft (0.3 to 4 m). Drill data indicate a depositional

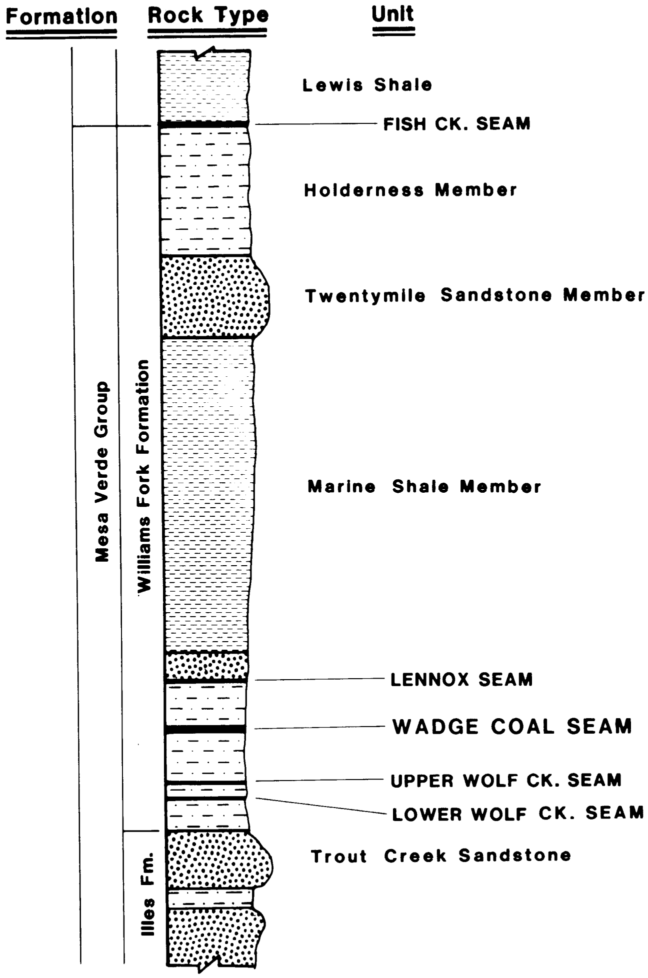

Fig. 5.8.3.2. Generalized stratigraphic column of the Upper Cretaceous strata of interest, Twentymile Park area, Northwest CO.

sequence containing several thin coal seams and two seams of economic interest. The two main seams, named the Upper and Lower Wolf Creek, contain rock bands and are locally split into subseams. Because seam quality is generally lower than the Wadge coal, the Wolf Creek seams were not rigorously studied during the project.

In general, the strata overlying the Wadge seam are found to be very predictable units (in terms of lateral continuity) of deltaic and marine origins. The strata below the Wadge seam, including the Wolf Creek seams, are highly variable and are comprised of interbedded coastal plain deposits with a high degree of fluvial influence.

Structure: Strata in the Twentymile Park area have undergone a long history of structural deformation that is predominantly compressional. Large folds and reverse faults formed early in the history while oblique normal faults apparently formed later. Three large, north-trending asymmetrical folds are the most prominent structural features that form the Twentymile Park basin. They cause the strata to dip in excess of 15% near the periphery of the basin and form the outer boundaries of the mine area. The reverse fault located in the west central part of the basin, with a displacement ranging from 20 to 80 ft (6.1 to 24.3 m), follows an arcuate line northward through the center of the basin (Fig. 5.8.3.3). The strata including the coal measures are also disturbed by northwest-striking normal fault zones, with displacements ranging from 0 to 35 ft (0 to 10.6 m), which occur in broad zones. These zones are 1 to 3 mi (1.6 to 4.8 km) wide and contain a number of faults that have similar characteristics to each other. The zones correlate to faults in the crystalline basement (Kucera, 1962) that were apparently reactivated during deformation of the sedimentary section.

Two major joint systems displaying consistent orientations occur throughout the basin. A shear system oriented northwest is very planar and terminates at lithologic boundaries. An extension system, oriented northeast, is sub-planar, open with mineral filling, and cuts indiscriminately across lithologic boundaries. Cleats in the Wadge seam share orientations with the joint sets and range in density from 6/ft (20/m) to spacings of one every 2 ft (0.6 m).

Hydrology: The Twentymile Park study area is an enclosed groundwater basin. Within the coal sequence strata, there are three significant groundwater aquifers: the Twentymile Sandstone, the overburden immediately above and including the Wadge seam, and the underlying Trout Creek Sandstone (Fig. 5.8.3.2). These aquifers are confined by shale layers above and below, and display artesian pressure levels in the center of the Twentymile Park basin. All three aquifers are recharged at the west and south flanks of Twentymile Park. Groundwater flows through the aquifers toward the east flanks, where it discharges into a perennial tributary to the Yampa River. During the study, joints and faults were recognized as conduits between lithologic units.

5.8.3.3 Twentymile Park Exploration Program

Pre-Exploration Data: Pre-exploration information was comprised almost entirely of drillhole data. These data consisted primarily of widely spaced holes completed into the Wadge seam from which geophysical logs were available. The quality of these logs ranged from unacceptable to excellent, with most in the good to acceptable range. If available, lithologic and core descriptions were usually too generalized to be of significant value. Drillholes were completed on ½- to ¾-mi (0.8- to 1.2-km) centers in the Foidel Creek area, except in the previously surface mined areas where drilling was shallow and of limited use. Drilling to the north in the Tow Creek area was particularly wide spaced, with no data available throughout the southern half of the area; holes were drilled on l- or 1 1½-mi (1.6- or 2.4-km) centers in the northern half of the area. Holes were somewhat more closely spaced in areas between old surface pits and the Middle Creek test mine. Drillhole data obtained on adjacent lands were generally spaced on ¾-mi (1.2-km) centers.

In addition to drillhole information, published reports of outcrop mapping were available. The US Geological Survey prepared a report (Bass et al., 1955) describing the general geology of the region, while two PhD dissertations concerning the stratigraphy, structure, and depositional environments of Cretaceous strata of much of the region were completed in the 1960s (Kucera, 1962; Masters, 1966).

Numerous mine pit exposures at nearby surface mines to the south were available to study seam character and continuity and structure.

In May 1979, Energy Fuels Corp., which controlled the property at that time, opened up an underground test mine in the Middle Creek area. This test mine experienced difficult mining conditions almost from startup, and ceased production in June 1980. Information on this failed project profoundly influenced the direction and goals of the Twentymile Park exploration program and subsequent mine planning.

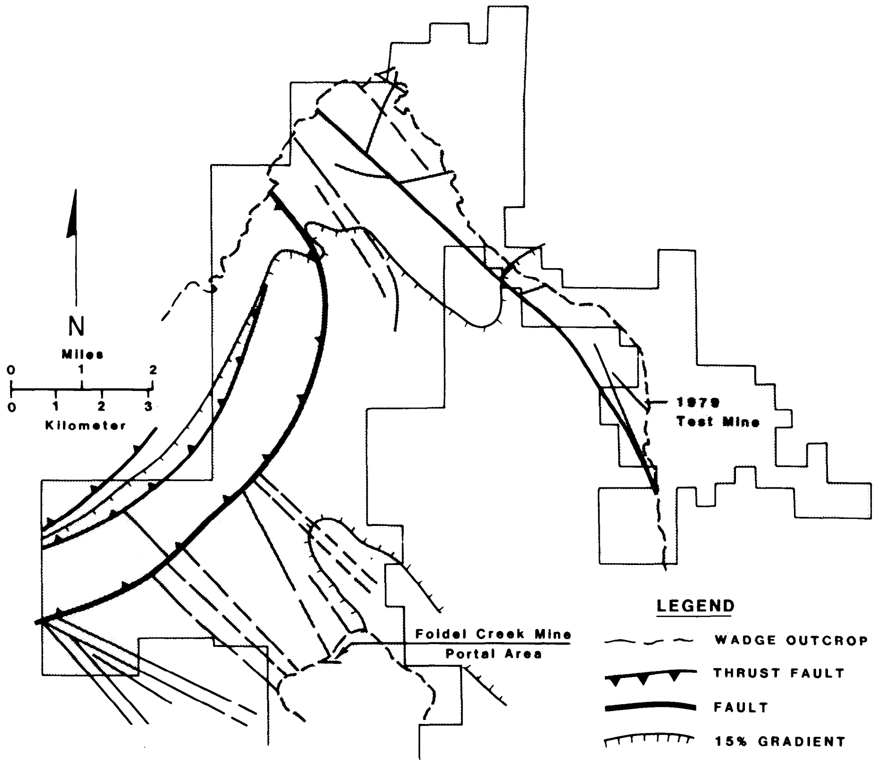

Fig. 5.8.3.3. Data of structure in the Twentymile Park area, Northwest CO.

Ongoing hydrologic and environmental studies in the Twentymile Park Basin were available at the beginning of the program along with a number of government and industry reports on the adjacent surface mine that were useful in the permitting process.

Scaling of the Exploration Effort: For three years, a multifaceted exploration program was conducted on the Twentymile Park area to address a variety of requirements and concerns. The phased program was comprised of conventional drilling and coring, field mapping, quality analyses, geotechnical testing of core, geophysical surveys, hydrologic data collection, overburden geochemical assaying, and preliminary wildlife, vegetative, and soil data collection. Steps in the program were designed to (1) achieve a broad database on the property, (2) focus efforts on a potential pilot mine(s) area, and (3) finalize mine planning details to support mine start-up efforts.

Project Specifics: Prior to the exploration effort, the underground coal resources were attached to the permit for the adjacent active surface mine. That existing permit allowed for the development of underground reserves from final pit exposures as a technical revision. It was decided that a pilot mine would be developed under the premise that the conditions encountered in the pilot mine would be found to be surmountable prior to the company application for a stand-alone mine permit.

Despite relatively harsh winter weather, field activities were conducted almost year around to achieve project goals. Actual drilling/coring was accomplished utilizing truck-mounted rotary drilling rigs (Gardner-Denver 2000, Gardner-Denver 15W, and a Midway 15M-2). Drilling was done with mud circulation systems. Holes averaged about 1500 ft (458 m) in depth. Completed holes were generally filled with an abandonment gel and the coal zone and upper portion of the hole cemented. A significant number of holes were completed as water monitor wells. Support equipment for the drills included a dozer and road grader (road and site work), backhoe (site work and reclamation), water truck, and a utility trailer. Four-wheel-drive vehicles were necessary because of the frequent muddy conditions.

Core was conventionally recovered at almost every site. Generally, $8\frac{5}{8}$-in. (219-mm) OD casing was set through surface alluvium, while rotary intervals were drilled $5\frac{5}{8}$ to 6 in. (143 to 152 mm) in diameter. As 3-in. (76-mm) core was desired for sampling, $4\frac{7}{8}$-in. (124-mm) OD chrisbit-variety core bits and/or conventional diamond bits were utilized. Cored zones were reamed if deeper seams were to be sampled. Six-in. (152-mm) diameter core was recovered from four sites to collect a reasonable bulk sample in anticipation of washability and certain geotechnical tests. Three drillholes yielded oriented core samples for geotechnical and structural studies. All holes were geophysically logged recording natural gamma, density, resistance, deviation, caliper, occasional neutron curves, temperature, sonic, dip meter, spontaneous potential (SP), and focused electric. Detailed seam thicknesses, ash analysis, and strength index interpretations were also computed. All logging information was stored on magnetic tape.

Cuttings were collected and logged on 5-ft (1.5-m) intervals. Core was logged for geotechnical properties including rock type, bedding, induration/competency, hardness, discontinuity characteristics, groundwater indicators, and effects. Core was photographed on l-ft (0.3-m) intervals and logged in detail prior to sealing in plastic and boxing. Core description included location, recovery, RQD (rock quality designation), discontinuity graphics and descriptions, three-digit lithology codes, lithologic and geotechnical descriptions, and sample designations. Samples of coal-bearing strata and coal were subjected to a wide range of ASTM standard quality tests. Quality analysis tests for seam composites included full proximate, ultimate, ash fusion, mineral analysis of ash, sulfur forms, hardgrove grindability, free swelling index, specific gravity, equilibrium moisture, and trace elements.

Short proximate analyses were also performed on selected immediate roof and floor samples and incremental seam samples. Although the property was anticipated to produce a run-of-mine (ROM) product, selected cores were subjected to washability studies. Specific gravity separations ranged from 1.30 to 1.80.

A geotechnical program was conducted to define rock mass parameters by measuring material properties, in situ horizontal stresses, and the geochemical nature of the confining strata. Stress relief overcoring and strain relief techniques were used to define in situ stresses. Geotechnical testing and analysis involved samples from across the property, but focused in the proposed Foidel Creek mine area. Rock properties testing was conducted by rock type and included Brazilian, uniaxial compressive, triaxial compressive, bulk density, moisture, slake durability, indirect tensile, and swelling index.

In general, the Wadge and Wolf Creek seams were cored in every hole and sent to the laboratory for qualitative testing. Along with each coal seam, 40 ft (12.2 m) of roof and 20 ft (6.1 m) of floor material was cored for geotechnical testing and engineering purposes. The Wolf Creek seam in nearly all the holes was paired, and a complete series of screen analyses and gravity separation tests were conducted to determine the preliminary washability characteristics of the coal.

Samples of the Wadge seam and immediate roof, obtained during drilling operations, were monitored for methane desorption from four drillholes. No significant amounts of methane were found associated with the coal seams. Petrographic analyses were also performed on the Wadge and Wolf Creek seams and included maceral composition and vitrinite reflectance.

During the drilling periods, a number of monitoring wells were installed in specific aquifer horizons throughout Twentymile Park. These wells were incorporated into a hydrologic monitoring program and designed to take advantage of drillholes that intersected saturated aquifers directly above and below the Wadge and Wolf Creek seams.

Two project-specific field mapping efforts were completed in the Twentymile Park area. The Twentymile and Trout Creek Sandstone outcrops were mapped, and all the accessible Wadge seam open pit highwalls were recorded with joint and fault data noted. Field mapping was found to be one of the most useful and cost-effective sources of structural and stratigraphic data.

Lineament studies were conducted to address regional structural questions and utilized satellite imagery and high-altitude photographs, in addition to computer-enhanced low-altitude areal photographs.

Surface geophysical surveys were also utilized to locate and help evaluate local structural features. Field surveys included resistivity, induced polarization, very low frequency-electromagnetic (VLF-EM), and high-resolution seismic traverses.

All sites and access routes were inspected by registered archeologists to confirm that no sites of prehistoric or historic significance were disturbed in the exploration process. As far as possible, all sites impacted by exploration activities were reclaimed by recontouring, replacing topsoil, fertilizing, and seeding the same year as impacted.

Phase I Highlights—The first phase of the exploration drilling program was started in September 1981, and extended until March 1982. During this time, 36 deep drillholes were completed and surveyed (43,850 ft or 13.4 km drilled, and 3929 ft or 1198 m cored), predominantly in the Tow Creek area (Fig. 5.8.3.1). This phase of the program was primarily designed to define geologic structure, seam thickness, coal quality, hydrology, and geotechnical properties of the coal and rock strata.

In addition to the normal coring program, four holes were designated for extensive continuous coring below a specific depth to evaluate interburden properties. A velocity survey profile (VSP) program was completed for two holes. Continuous coring was accomplished in only one hole because of the time and cost was substantially more than anticipated. Nine sites were completed as hydrologic monitor sites.

During this early phase, a project evaluation task group was formed to assess the economic feasibility of mining the Twentymile Park underground coal reserves. Conceptual engineering utilizing incoming geologic data shifted emphasis from the Tow Creek to the Foidel Creek area (Fig. 5.8.3.1). (Pre-1981 evaluations indicated that the initial access should be into the Tow Creek area, with entries driven from a site near abandoned strip pits.) Preliminary results of this phase of exploration indicated that this access should not be evaluated further due to complex geologic structure with intense fracturing and faulting, combined with steep gradients.

In addition to the foregoing, results of phase I indicated that a high priority should be given to the geotechnical program and that it should include in-situ stress measurements. Further, the old 1979 test mine area (Middle Creek) should be included in the studies so it could be used as a "yardstick" for comparison with the Tow Creek and Foidel Creek reserves.

This first phase of exploration established a sizable reserve for the Wadge seam based on a minimum seam thickness of 5 ft (1.5 m) and seam dips less than 15% gradient (Fig. 5.8.3.4). The Wadge reserves for the Foidel Creek and Tow Creek areas totaled over 230 million tons (209 Mt) of in-place coal and suggested a controlled minable reserve of over 140 million tons (127 Mt).

Phase II Highlights—The second phase of exploration was planned to define geologic factors impacting mine planning and feasibility. This phase included additional drilling, mapping, strength testing, stress measurement, oriented core, pump tests, washability tests, geophysical surveys, satellite imagery, high-resolution seismic, and depositional modeling. Studies were concentrated in the area of the first five years of planned mining, with detail in other areas sufficient for long-term mine planning and economic feasibility.

During this second phase, 35 deep coal exploration holes were drilled and logged (44,100 ft or 13.4 km drilled, and 3895 ft or 1187 m cored). All were cored to recover roof, coal, and floor, and 16 were completed as hydrologic monitor wells. A total of 30 mi (48.3 km) of resistivity lines was completed, 10 mi (16.1 km) of high-resolution seismic lines were shot, 7 mi (12.1 km) of VLF-EM survey lines were run, 6 aquifer tests were performed, and 6 mi (10.5 km) of deep seismic data were acquired. Fifteen shallow (250 ft or 76 m) holes were also drilled to investigate possible offset in the Twentymile Sandstone.

Results of phase II and detailed strata analyses indicated that the proposed Foidel Creek portal site exhibited reserve characteristics that were substantially superior to those present in all other potential sites (Fig. 5.8.3.5). Additional drilling and seismic and resistivity surveys were recommended to obtain data adjacent to the major faults and to validate the structural model in the proposed pilot mine area. In addition, another monitor well was proposed in the mine area to analyze potential hydrologic hazards.

Potential impacts by geologic features on production were identified (Fig. 5.8.3.6) and their cumulative effect overlain on the reserve area. Significant concern in the proposed pilot mine area focused on potential hydrologic impacts and dilution effects from exposed roof strata.

Phase III Highlights—This phase of the program entailed the development of water wells for mine consumption, one deep water-monitor well and two alluvial monitor wells. Concurrent with the Foidel Creek mine start-up, eight drillholes were placed to gain greater structural control and to test rock strength and conditions in the test mine area. A l-mi (1.6-km) portion of seismic line was reshot, and approximately 5 mi (8 km) of surface induced polarization (IP)-resistivity survey data were gathered.

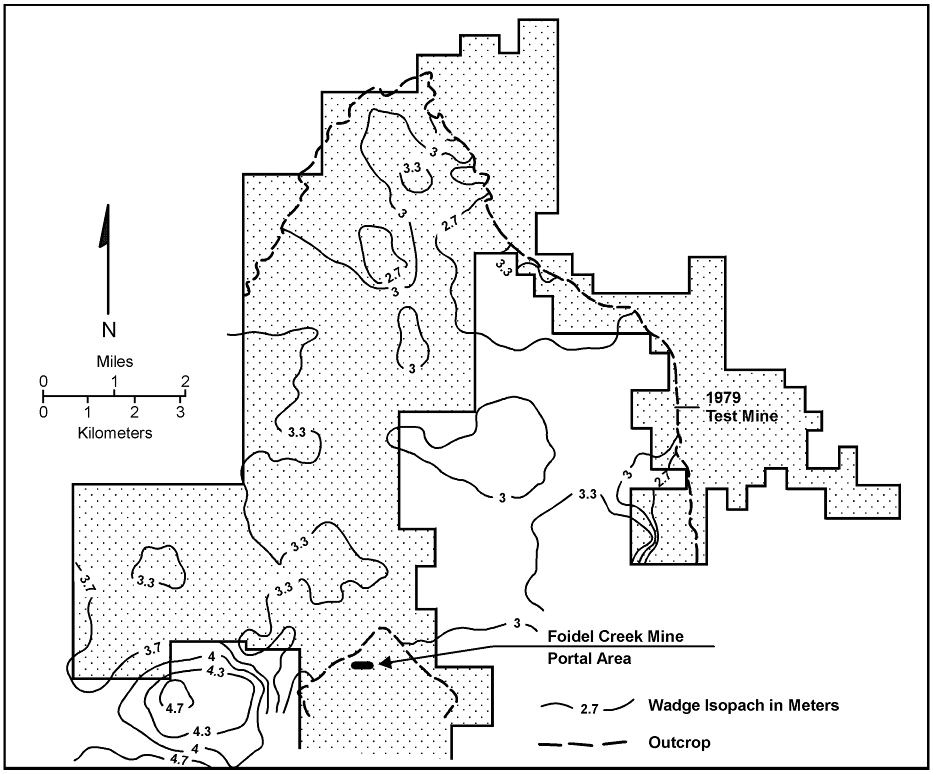

Fig. 5.8.3.4. Isopach of the Wadge coal seam, Twentymile Park area, Northwest CO.

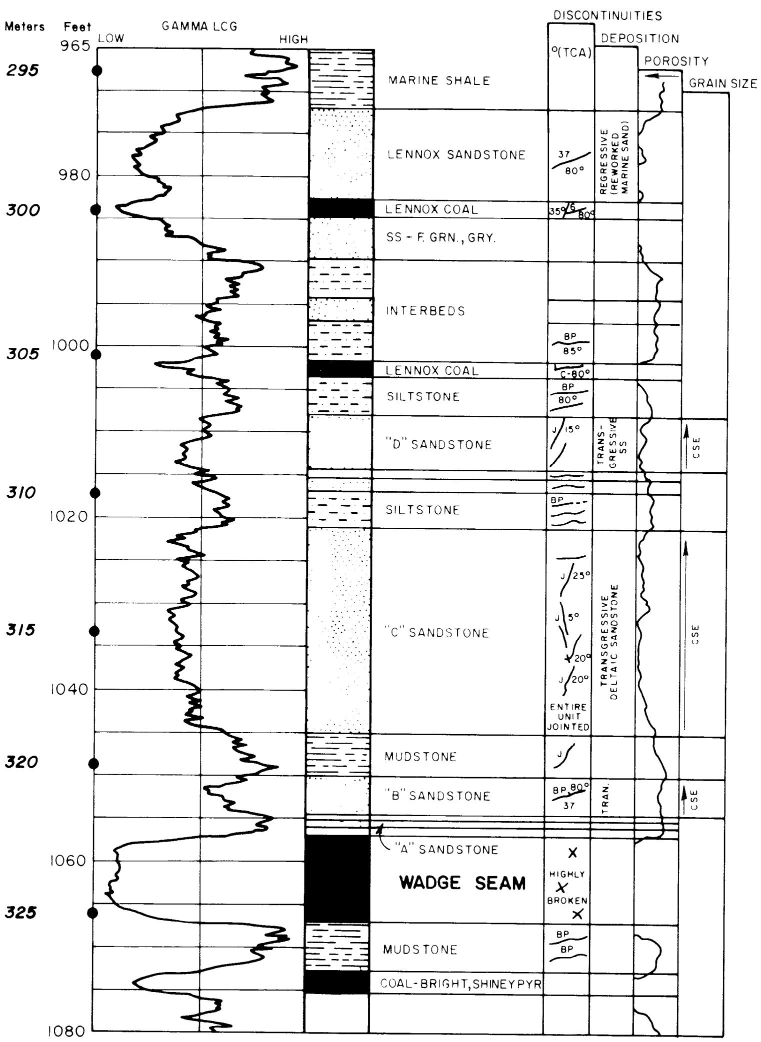

Fig. 5.8.3.5. Example of data collection detail on Wadge coal seam and roof characteristics.

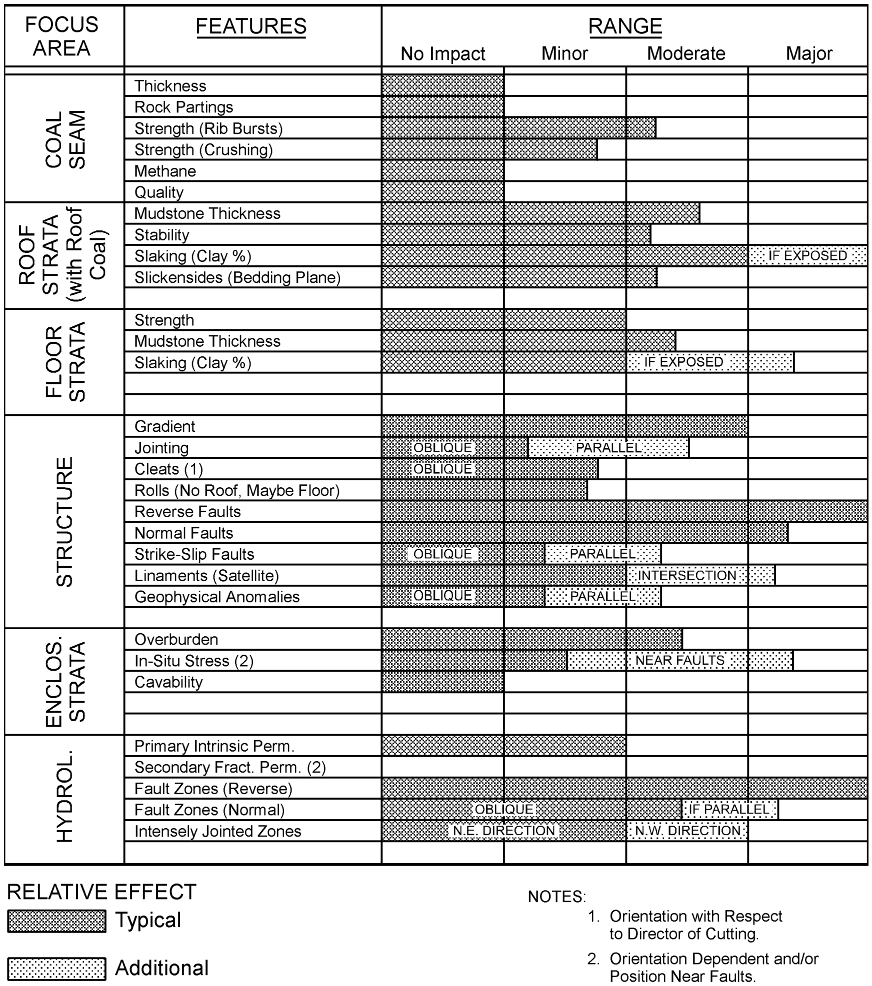

Fig. 5.8.3.6. Detail of analysis on relative potential productivity impacts of Wadge seam geologic features.

Table 5.8.3.1. Wadge Seam Quality Variation (As Received)[a]

	Top Coal	Mid Seam	Bottom Coal	Full Seam
Moisture, %	9.1	8.6	8.2	8.6
Ash, %	12.1	8.1	11.7	8.8
Sulfur, (%)	0.6	0.4	0.5	0.5
Heating Value,				
Btu/lb	10,900	11,600	11,100	11,500
(Mj/kg)	(25.35)	(26.98)	(25.81)	(26.86)

[a]These statistics are averaged for the total in-place Twentymile Park reserves base.

Several miles of VLF-EM data were also gathered. Over 9400 ft (2865 m) of rotary drilling were completed during this phase of the program, and 225 ft (69 m) of core were recovered. Portions of the earlier planned seismic surveys were dropped, due to potential scattering effects created in areas of previous surface mining. Rock mechanics and coal quality analyses were immediately incorporated into the planning and evaluation of the developing pilot mine. Ongoing hydrologic monitoring suggested hydrologic effects would lessen deeper in the basin because of the "tight" nature of most discontinuities. A selected mid-seam mining zone was chosen from detailed quality data for optimum ROM product, as shown in Table 5.8.3.1.

5.8.3.4 Project Costs

Exploration and project evaluation costs varied widely over the three phases of the program but reflect progressive levels of concern, typical pricing, and a strong focus on state-of-the-art exploration technology and detailed data gathering and interpretation. Analysis and interpretation efforts typically bridged the program phases but are reflected, as best as possible, in their relative categories (Table 5.8.3.2).

Phase I costs approximated $0.9 million. Significant costs were tied to direct drilling expenses 56% (plugging and coring, fuel, soap, etc.), staff support (11%), down-hole geophysics (8%), and geotechnical analysis (7%). Relatively mild winter weather kept access costs under control during this phase to an unusual degree.

Phase II and exploratory interpretive efforts resulted in an overall cost of over $2.2 million. Major expenditures were incurred by geotechnical consultants (18%), geophysical surveys (18%), and direct drilling (32%). Geotechnical testing and the hydrology program consumed 6% each of the budget.

Phase III, which was conducted concurrently with the startup of the pilot mine, cost slightly over $0.2 million. Principal expenditures were from drilling activities (28%) and hydrological studies (23%).

Overall, a total of nearly $3.3 million was spent on a 3-year exploratory, evaluation, and permitting effort to bring a previously undeveloped underground coal resource to an identified coal reserve in a producing state. Percentage breakdowns of major expense categories are depicted in Table 5.8.3.3.

As would be expected, drilling/coring was found to be the best way to obtain data at specific locations on the property, with rotary costs equaling about $5.70/ft ($18.70/m) overall ($48.75/ft or $160/m, for conventional coring, and recovering oriented core cost about $60.75 ft ($199.31 m). Costs associated with these rates are shown in Table 5.8.3.4.

Table 5.8.3.2. Cost Breakdown by Phase, Twentymile Park Exploration Program

Expenditure	Phase I	Phase II	Phase III	Total
Archeology	$ 8,700	$ 4,800	$ —	$ 13,500
Site preparation/ reclamation	25,600	27,750	6,700	60,050
Mobilization	3,000	2,000	1,000	6,000
Rotary drill	250,950	251,170	52,180	613,450
3-in. (76-mm) core	168,520	219,180	12,005	399,705
6-in. (152-mm) core	—	181,600	—	181,600
Cementing	23,890	24,200	4,150	52,240
Misc. supplies	32,670	73,875	16,710	123,255
Hydrologic	20,850	119,450	53,005	193,305
Geophysical logging	73,445	71,995	12,550	157,990
Geotechnical	56,550	136,350	13,000	205,900
Geophysical	20,000	340,285	19,020	379,305
Analytical	28,700	26,250	4,200	59,150
Rental equipment	10,200	8,125	5,450	23,775
Survey	6,180	11,750	3,600	21,530
Computer	—	18,500	2,080	20,580
Project evaluation	—	112,100	2,900	115,000
Geotechnical consultants	—	430,000	—	430,000
Staff support	98,335	100,795	17,220	216,350
Staff G&A	33,920	41,315	5,820	81,055
Total Project	$861,510	$2,201,490	$231,590	$3,294,590

Table 5.8.3.3. Percentage Breakdown of Major Cost Items, Total Project

Category	Percentage, %
Direct drilling	1
Hydrologic	6
Drillhole geophysics	5
Geotechnical	12
Surface geophysics	12
Field staff	9
Project evaluation	17
Quality analysis	2
Other	6
Total	100

Table 5.8.3.4. Typical Additional Costs to Direct Drilling Rates

Item	$/ft	$/m
Roadwork/site preparation	0.55	1.80
Geophysical logging	1.47	4.80
Rental equipment	0.22	0.72
Staff support	2.10	6.90

Drilling rates were approximately 17 ft/hr (5.2 m/hr) and coring rates generally about 2 ft/hr (0.61 m/hr). (Bit and mud costs are included in the $5.70/ft ($18.70/m) expense and were nearly $1.10/ft ($3.61/m) and $0.71/ft ($2.33/m), respectively.)

High-resolution seismic and resistivity—I.P. surveys were found to be effective tools to define structure across broad areas of the Twentymile Park Basin. Costs for these items equalled $25,000 to $30,000/mi ($15,530 to $18,640/km) and $2200/mi ($1370/km), respectively. The methods had to be used in conjunction with other types of data input. Actual field mapping and VLF-EM surveys were found to be very cost effective at about $300/day/individual field person.

5.8.3.5 Project Results

The Twentymile Park exploration and evaluation effort lead directly to the timely and successful development of a phased underground pilot mine during a period of economic contraction in the coal industry. The detailed evaluation enhanced confidence levels on all key aspects of resource minability.

Results of the project follow:

1. An in-place underground coal reserve of over 400 million tons (363 Mt) of controlled Wadge and Wolf Creek coal was delineated, of which 230 million tons (209 Mt) are accessible via the Wadge seam.
2. The minable Wadge seam underlying the Twentymile Park area contains a high-quality, low-sulfur coal reserve characterized by extreme lateral continuity and generally superior mining conditions.
3. The strata containing the Wadge seam has undergone deformational episodes, but discontinuities are generally predictable and can be recognized in the mine planning process.
4. Modeling of the groundwater hydrology indicated that water inflow rates for a full mine operation could be over 1500 gpm (95 L/s) at periodic peaks and reach equilibrium levels around 900 gpm (57 L/s).
5. Geotechnical data allowed optimum mine plan layout to minimize in-situ stress and entry deterioration. Pillar sizing, cavability, and roof and floor stability were also addressed and resolved.

Through the development period, development production averaged over 700 tons (635 t)/machine-shift, with highs ranging up to 1000 tons (907 t)/machine-shift. Mine product has been marketed on a ROM basis averaging about 10.0% ash, 0.46% sulfur, and 11,000 Btu/lb (25.6 MJ/Kg). A main source of dilution to the coal product is cap rock (roof coal is difficult to hold), although high production rates minimize the impact. Water inflow rates have been much lower than anticipated, due primarily to closure of discontinuities in deeper portions of the basin.

Because of the success of the pilot mine, the mine permit was revised to reflect a stand-alone underground mining operation during application for renewal in late 1984. (The mine permit was issued in late 1987 by the Colorado Mined Land Reclamation Board.) By early 1989 over 12,000 ft (3660 m) of mains had been developed in Cyprus' Foidel Creek mine; a variable cover pillar section had been developed and pulled to evaluate caving characteristics and a three-entry longwall gateroad plan developed.

In the latter part of 1988, management of the Cyprus Coal Co. approved the purchase of a new 640-ft (195-m) face longwall to take advantage of the proven uniform conditions of the Wadge seam and to improve mine productivity and overall ROM quality. A longwall face was installed in the mine during 1989. Average raw tons per longwall shift for the first six months of 1991 averaged over 6300.

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Section 6 Mine Evaluation and Investment Analysis

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Chapter 6.0
INTRODUCTION

DONALD W. GENTRY

This section deals with one of the most fascinating aspects of mining—the evaluation of mine investment opportunities. Before proceeding further, it is important to clarify the meaning of some fundamental terms used frequently in the text.

6.0.1.1 Mine Valuation vs. Evaluation

The words "valuation" and "evaluation" in a mining context are often used interchangeably. However, for the purposes of this section, a distinction is made between the two terms in accordance with commonly accepted practice. The word *valuation* in a mining project context has the rather narrow meaning of placing a dollar or other currency value on the worth of the project as a whole. In other words, the value of a mining project refers to a measure of the desirability of ownership of that property. As such, the major item of interest is, "what is the mine worth?," or "what is the value of the mine?" The value of a mining project may actually be determined in the marketplace at any specific point in time, or it may be estimated by one of several methods. The topic of mine valuation is discussed in more detail in Chapter 6.1.

The word *evaluation* in a mining project context connotes the broader meaning of determining the numerical values of all possible factors or variables that are important in establishing the worth of a mining project. In other words, mine evaluation denotes the assessment of the relative economic viability of the mining project or investment opportunity. In this regard, estimates of project ore reserves, mining rates, revenues, costs, expected returns and associated risks, etc., as well as the dollar worth, are made for each project or investment opportunity available to the organization. Various aspects of the mine evaluation procedure are discussed in detail in subsequent chapters contained in this section. For instance, Chapter 6.2 discusses mine feasibility studies, Chapter 6.3 describes procedures for estimating capital and operating costs, while Chapter 6.4 deals with project operating strategies.

6.0.1.2 Investment Analysis

In this section the term *investment analysis* connotes continuation of the project evaluation process. That is, after the numerical values of all possible project variables have been estimated and the dollar worth of the entire project determined, appropriate evaluation criteria are calculated for each investment opportunity available to the firm. All projects are then ranked

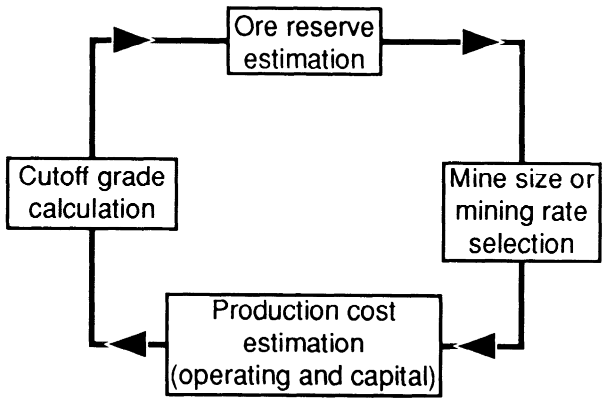

Fig. 6.0.1. Generalized iterative procedure for mine evaluation.

according to the preferred investment criteria and incorporated into the corporate capital budgeting process. Chapter 6.5 discusses mining project investment analysis as it relates to estimating project economic viability.

The mine evaluation procedure used for investment analysis is usually iterative in nature. The general procedure may be represented by Fig. 6.0.1 (Gentry and O'Neil, 1984). The tonnage and grade of the estimated ore reserve established from the exploration program are important variables in determining optimum mine size. Mine size, in turn, affects production costs, as economics of scale are often enjoyed with larger production rates. Finally, the level of production costs for the project as a whole determines what material can be mined at a profit (cutoff grade) and therefore determines the magnitude of the ore reserve.

The important point here is to recognize that each time a variable changes, the analyst must assess the impact of this change on all other project variables and on the subsequent financial and economic results. This iterative procedure must be repeated until the most economical design is achieved for the mining project being analyzed. This is, indeed, a time-consuming process, but it represents the essence of the mine evaluation process for investment analysis purposes.

6.0.1.3 Capital Budgeting

As noted in the previous section and as discussed in detail in Chapter 6.5, the mine evaluation procedure and associated investment analysis are incorporated into the corporate capital budgeting process. The capital budgeting process refers to the sequence of decisions that ultimately lead to the firm's acceptance or rejection of investment proposals along with subsequent management of the proposals accepted (Gentry and O'Neil, 1984). The capital budgeting process is normally considered to comprise the activities of planning, evaluation, selection, implementation, control, and, finally, continual reevaluation and auditing of results. In essence, *capital budgeting* is that element of capital investment dealing with the allocation of capital to projects in some optimal manner. The perceived benefits from these projects are projected to be realized at some time in the future.

After the firm has reconciled the investment decision, the financing decision must be addressed. The financing decision concerns the selection of source(s) and timing of new capital necessary to implement the investment decisions. This aspect of financing mining projects is discussed in Chapter 6.6.

6.0.1.4 Wealth Maximization

Although economic theory usually assumes that both private and public organizations strive to maximize profits from their investment decisions, various portions of this section are formulated on the premise that the fundamental objective of any private organization is to maximize its value or wealth to its owners (stockholders). In this sense, the term *wealth* refers to the total current market value of the firm's assets. In the case of corporations, the wealth or value is considered to be represented by the market price of the firm's common stock.

It is important to note that wealth maximization is a more appropriate and inclusive goal for a firm than profit maximization. Indeed, there is a difference between the two objectives in most situations. For example, there are a number of ways for a firm to increase profits, but many of these activities would rarely cause net shareholder worth to increase. Indications are that total profits are not as important to the investment community as are other indicators. Over the long term, the success or failure of an organization will be a function of how well the management of the organization handles investment and financing decisions, for the results of these decisions will directly impact the value of the firm in the marketplace.

6.0.1.5 Mining as a Unique Investment Environment

Certainly the investment environment associated with the mining industry is unique when compared with the environment encountered by most other industries. Section 2 of this book discusses various aspects of minerals commodities markets, supply-demand relationships, pricing, trading, and taxation issues that are often unique to the mining industry. Many of these unique characteristics are the result of some fundamental features of the minerals industry, which, in their combination, result in a unique business environment. These special features are described by Gentry (1988) and Gentry and O'Neil (1984) as follows.

Capital Intensity: Mining ventures are extremely capital intensive. Although the magnitude of the capital investment required for a new mining venture varies with the type of commodity, mining method, mine size, location, and other parameters, major new mines may require financial commitments that range from $500 million up to as much as $8 to $10 billion. The infrastructure alone for mines in remote locations may cost several hundred million dollars. Even small, high-grade precious-metals operations that employ a small work force may require multi-million dollar investment.

Cost Structure: The capital intensity results in a unique cost structure for the mining industry. The total average cost of production—including fixed and variable costs—per unit of salable product is often higher than the marginal or variable cost for the same unit. The average cost includes a high fixed-cost component, which primarily represents capital recovery. Consequently, in periods of low demand and price, a mining operation may be covering its marginal cost but actually losing money if the average cost per unit is considered.

Also, because high fixed costs represent a large component of a mining operation's total costs, the breakeven production level for mining facilities is closer to capacity than for other types of facilities with lower fixed costs. This is one reason why operators attempt to run mines at capacity, often employing three-shift, seven-day/week work schedules.

Long Preproduction Periods: Once the occurrence of an ore deposit has been established, several years of intensive effort are required before the property is brought on stream and ore is produced on a continuous basis. The preproduction period may range from 3 to 12 years, depending on the mining and processing methods, size and location of the deposit, and complexity of the operating and environmental licensing procedures, as well as other factors.

The significance of these long lead times is amplified when considered in conjunction with the capital intensity of the industry. Not only are companies committing extremely large capital resources to a new mining venture, but they also are exposed financially for a considerable period prior to project start-up. The longer the lead time, the higher is the probability of undesirable change in key engineering and economic parameters that were utilized in the initial investment decision. Also since expenditures of capital are required throughout the preproduction period, the longer the lead time, the greater are the returns required to offset the lost investment opportunities represented by the preproduction period.

Nonrenewable Resource: The aspect of the minerals industry that perhaps distinguishes it most from other industries is the fact that it deals with the extraction of a nonrenewable resource. The implications of a depletable resource are numerous. For example, one result is that revenues from mining are derived from a piecemeal disposal of the project's major asset, the ore body. As a result, the return of and return on the capital investment must be obtained within the finite life of the ore body. This unique feature has led many countries to provide for special tax treatment of mining ventures in the form of a depletion (or equivalent) allowance.

Because mines have finite lives that are determined by the size of the ore deposit and the mining rate, new deposits must be discovered and developed through continuing programs of exploration. Also the fact that a mining operation has a finite life may cause government agencies to require special concessions from the developer as a result of the finite nature of the benefits. One example may be special fiscal requirements that stem from the realization that government-provided infrastructure will no longer be required after the inevitable exhaustion of the deposit.

Indestructibility of Product: Another special feature of the mining industry centers on the fact that many metals are, essentially, indestructible. The consequence is a secondary market and a reduction in the amount of primary ore necessary to provide the required supply. Recycling has considerable economic advantages due to energy and other cost savings and contributes significant percentages to the US consumption of such metals as aluminum, copper, lead, zinc, silver, and iron and steel. In addition to reduced energy consumption and further increase in supply, recycling also affects metals markets in the following ways (Gocht et al., 1988):

1. The new supplier can weaken oligopolistic or monopolistic market positions.

2. World trade diminishes because recycling takes place directly in the consuming countries, with few exceptions.

3. Mineral resources are preserved.

4. Recycling of waste contributes to protecting the environment.

In the planning for any new mining venture, projections of recycling volumes and the growth in secondary markets should be included when estimating the overall future supply of the commodity considered for production.

High Risk: In addition to the obvious risks associated with capital intensity and long preproduction periods, the risks associated with mining ventures include some that may be under the control of the investor and others that are clearly uncontrollable. In general, these risks may be placed under the general headings of technical risks (geologic and engineering risks), economic or market risks, and political risks.

Although technical risks have been notably reduced in recent years, business risk—the chance of major loss due to market factors—has increased dramatically. This has been caused, fundamentally, by two primary factors: the rapidly rising capital requirements of new projects and the increasingly unpredictable future economic conditions. These difficulties, of course, are compounded by the long preproduction periods required for large, new mining projects (O'Neil, 1982).

In addition to the obvious risks associated with mineral markets and widely fluctuating metals prices, another economic risk is that associated with inflation. The impact of high rates of inflation on mineral project evaluation can be significant indeed. Also related to inflation, monetary exchange rates have recently become an important consideration in mine evaluation. When a firm sells its product in one currency and buys services in another, changes in exchange rates between the currencies can have grave consequences.

Political risk, although often overlooked, has become increasingly important in recent years when considering mining investments. There is an accelerating trend to greater participation in mining projects by host governments throughout the world. In the limit this can take the form of outright expropriation. Therefore, mining companies contemplating any new venture must assess these growing political risks to ensure that the added financial exposure is warranted.

The chapters that follow deal with the more important issues associated with mine valuation, mining project evaluation, investment analysis, and mine financing. The reader is referred to the references at the end of each chapter for further information on these topics.

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Chapter 6.1
MINE VALUATION*

DONALD W. GENTRY AND THOMAS J. O'NEIL

As noted in the introduction to this section, the term *mine valuation* implies the assigning of a dollar or other currency value to the worth of a mine or mining project and provides a measure of the desirability of ownership of that property. As such, several types of value may be encountered in performing a mine valuation study. These are

1. Market value.
2. Full cash value.
3. Salvage value.
4. Replacement value.
5. Capitalized value.
6. Book value.
7. Assessed value.
8. Insured value.

Each of these has a specific meaning that can be applied to determine a monetary amount in a specific situation.

Of interest in this chapter is the broader question of "what is the value of the mine?," or "what is the mine worth?" In this context the value of interest is the market value of the property. *Market value* is the value (price) established in a public market by exchange between a willing buyer and a willing seller when neither is under duress to complete the transaction. Thus the term *market* suggests the idea of barter.

The term market value is often used synonymously with the term fair market value. The courts have come to accept the legal definition of *fair market value* as the amount in cash, or in terms reasonably equivalent to cash, for which in all probability the property would be sold by a knowledgeable owner willing but not obligated to sell to a knowledgeable purchaser who desired but is not obligated to buy. Therefore, the *determination* of market value of a specific mining property can only be made by the market through an actual sales transaction, in accordance with the foregoing caveats pertaining to the absence of duress on the part of either the seller or the buyer.

Typically, mineral economists, appraisers, and government tax officials, among others, are concerned with the *estimation* of market value for mineral properties. This estimated market value must be based on the time and conditions existing as of a specified date. Consequently, market value is a dynamic property that constantly changes as market conditions and expectations change. Following is a discussion of the approaches most often utilized to estimate the market value of mining properties. The specific recommended methodology for estimating the value of a mining project is provided in Chapter 6.2.

6.1.1 APPROACHES TO MINE VALUATION

To estimate the market value of any asset, most appraisers initially consider three generally accepted approaches to value estimation. All three approaches are based on the very important appraisal principle of substitution. A closer look at each of these approaches and its applicability to mining or mineral properties is provided in the following discussion.

6.1.1.1 Cost Approach

The *cost approach* to mine valuation attempts to determine the depreciated replacement cost for the asset in question. That is, what would it cost to reproduce an asset of identical quality and state of repair? The fundamental concept with this approach is that a purchaser would not be justified in paying more for a property than it would cost him to acquire land and construct improvements that had comparable utility with no undue delay.

The cost approach is rarely applicable in mining because the correlation between construction costs and the value of the property is very imperfect. For example, if one were to build mines with production capacities of 100 tpd (90 t/day) each, one on a very rich ore deposit and one on an economically marginal deposit, construction costs might be very similar, but fair market values of the two mines would, clearly, be substantially different.

Another problem arises when the cost approach is applied to newly discovered mineral properties that have no surface improvements or equipment of any kind. The very nature of mineral exploration and mining dictates that the discovery value of an ore deposit is generally greater than the cost incurred in making that discovery. If this were not true in the aggregate, investment could not be justified for exploration. Furthermore, the notion of estimating the cost of acquiring a comparable asset (ore body) is not very useful. This cost could, for example, be infinite if nature failed to provide a duplicate for explorationists to find.

The cost approach is not only the least applicable method in the valuation of mining properties, but it generally is the least reliable also.

6.1.1.2 Market (Comparable Sales) Approach

This approach is considered by most appraisers and the courts to provide the best indicator of fair market value, since it reflects the balance of supply and demand in the marketplace. The *market approach* assumes that a purchaser would not be justified in paying more for a property than it would cost him to acquire an equally desirable substitute property. The concept of market value also presumes conditions of an open market, exposure for a reasonable time, knowledgeable buyers and sellers, absence of pressure on either the seller to sell or the buyer to buy, and a sufficient number of transactions to create a stable market.

The market approach encounters serious practical problems when applied to mining transactions. This is mainly due to two facts: first, there are very few sales of mining properties, and therefore few comparative data are available; and, second, since each mineral deposit is unique in quality, size, geographical location, degree of development, and many other parameters, any market data are of modest value at best. To be applicable, the market data must not only relate to similar assets but must also be for a similar point in time.

* This chapter is compiled exclusively from materials contained in Chapters 1 and 2 of *Mine Investment Analysis* by D.W. Gentry and T.J. O'Neil (1984) and from the article "Minerals Project Evaluation — An Overview" by D.W. Gentry (1988).

Experience in the area of mineral property transactions suggests that the open-market, unpressurized dealing and other assumptions previously mentioned in association with this approach are seldom reflected in reality. When such criteria and assumptions are met, it is often extremely difficult to ascertain the actual or true value of the sale because of stipulations pertaining to production commitments, deferred payments, exchanges of stock, production payments, and other subtle factors that can affect the value significantly.

6.1.1.3 Income (Earnings) Approach

With the *income approach,* the value of an asset or investment-type property is estimated by calculating future annual net earnings generated from the producing property or asset and then discounting this earnings stream to the present time by the use of an appropriate discount rate. The approach assumes that a purchaser would not be justified in paying more to acquire income-producing property than the present value of the income stream to be derived from the property. Because of the calculation procedure utilized, many analysts refer to this as the capitalized income approach.

In essence, the income approach is one step removed from the market (comparable sales) approach. If comparable sales data are unavailable or if one is estimating the value of a mineral commodity in situ, it is possible to arrive at a value estimate by combining the selling price of the commodity *produced* with the associated costs of producing it from the property in question. By proper incorporation of these data into a discounted cash flow analysis, it is possible to arrive at an estimate of property value even in the absence of actual production. It is important to remember, however, that the estimate thus obtained is not a direct estimate of the market value of a commodity in place, but rather an estimate of potential income generated from mining and selling the commodity.

Capitalized future income is a unit valuation method. That is, a single value is assigned to the ore deposit, to surface and subsurface improvements, and to all real and personal property used in the production process. To a considerable degree, real property at mines has value only because of the presence of ore, and unit valuation is therefore appropriate.

Because mines have limited operating horizons, and because there are well-established markets for most mineral commodities, the income approach is widely used in valuing mineral properties. The approach is commonly used by the mining industry in the assessment of investment rates of return and to determine appropriate purchase prices for mines or mineral prospects. From a practical standpoint, the income approach has the added capability of incorporating more obtainable, realistic data for analysis and therefore is the preferred approach to mine valuation. In addition, the income approach is consistent with the generally accepted definition of the value of a mineral property. That is, the value of a mining or mineral property at a specific point of time is simply the present value of all the future net annual proceeds that are expected to accrue from ownership. The basic element in the income approach to mine valuation is the pro forma income statement, which is discussed in more detail in Chapter 6.2.

6.1.1.4 Other Types of Value

As mentioned at the beginning of this chapter, there are several types of value that may be encountered when performing mine evaluation studies. Following are brief descriptions of some of them.

Salvage Value: Salvage is the net sum, over and above the cost of removal and sale, realized for a property or asset when it is retired from service. Salvage value and scrap value are synonymous when the property or asset retired from service is scrapped for the value of its materials.

Replacement Value: Replacement value refers to the existing value of a property or asset as determined on the basis of what it would cost to replace the property or its service with at least equally satisfactory and comparable property and service. The concept of replacement value is fundamental to the cost approach utilized by appraisers.

Book Value: Book value is the original investment in the property or asset as carried on the organization's books less any cumulative allowance for depreciation or amortization entered on the books.

Assessed Value: The assessed value of a property is the value entered on the official assessor's records as the value of the property applicable in determining the amount of ad valorem taxes to be paid by the property owner.

Insured Value: The insured value of a property refers to that value at which the property has been insured against loss or disaster. This value is generally associated with replacement value for tangible assets and earning capacity for property such as mines (ore deposits).

Capitalized Value: The capitalized value of a property is the sum of discounted future annual net earnings generated by the property. The capitalized value concept is synonymous with the income approach to value estimation for mining properties.

6.1.2 PURPOSE OF MINE VALUATION STUDIES

There are many reasons for conducting studies on estimating the value of a mining property. Regardless of the specific purpose for estimating the value of a mining property, the ultimate objective of the study is to arrive at a monetary value or worth for the property in question. A specific value, or range in values, for a specific property is often required for one or more of the following purposes.

6.1.2.1 Acquisition

The acquisition of mineral properties may transpire at any point in time between a raw prospect and an actual operating mine. Obviously, the actual amount of data available on a property will depend upon where it lies within this spectrum. Depending upon the state of development of the property, the purchaser is acquiring assets with varying levels of risk. As such, the estimated value of the property must reflect not only the potential of the mineral deposit but also the relative risks associated with these assets. Certainly the distribution of value estimates for an existing operating mine would be expected to have a rather low variance as contrasted to that for a raw prospect.

6.1.2.2 Taxation

Mineral properties must also be valued for taxation purposes. Perhaps the classic example here is with ad valorem property taxes, levied by most state and local governments. The difficulty with value estimation of a mineral property for taxation purposes is that a *single* value is required for property worth.

Most states have enacted tax provisions that attempt to approximate the value of a mineral property through a formula or other mechanism that rarely serves as an adequate measure of property value for an actual sale. These mechanisms are seldom based on strong economic foundations and only serve as a convenient proxy for mineral property values. As a result, significant discrepancies can occur between the appraised value of a property for tax purposes and the value as perceived in the marketplace.

6.1.2.3 Financing

The mode, mechanism, and magnitude of financing new mining properties or ventures are functions of the estimated property or project value. Certainly, the risk of default must also be considered in mining and must be assessed in regard to the perceived intrinsic value of the property. This aspect is becoming increasingly important in view of the popularity of international joint ventures as a means of dispersing project risks.

The fundamental concern of lending institutions is not whether a specific rate of return is achieved by the project owner, but rather that the project will generate adequate cash flows to service the acquired debt. Thus lenders approach mine valuation studies from a different perspective.

6.1.2.4 Regulatory Requirements

Even the federal government has found it necessary to wrestle with the problems associated with estimating the value of federally controlled mineral lands. This results from the fact that the leasing of federal lands for some mineral commodities is through the competitive bidding process, and the government is obligated by law to accept no bid that is less than the fair market value of the mineral occurrence. As a result, the federal government is often required to estimate the value of certain leases prior to competitive bidding in order to assure that bonus bids and royalty provisions represent fair market value and are therefore acceptable. The federal government is faced with a similar valuation problem when determining or negotiating royalty provisions on other leased minerals.

Gentry, D.W., 1988, “Minerals Project Evaluation—An Overview.” *Transactions*, Institution of Mining and Metallurgy, Vol. 97, pp. A25-A35

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Chapter 6.2
MINE FEASIBILITY STUDIES*

DONALD W. GENTRY AND THOMAS J. O'NEIL

Feasibility studies are the heart of the mine evaluation process. A *feasibility study* of a mining project represents an engineering/economic appraisal of the commercial viability of that project. As such, it is the result of a relatively formal procedure for assessing the various relationships that exist among the myriad of factors that directly or indirectly affect the project in question. In essence, the objective of a feasibility study is to clarify the basic factors that govern the chances for project success. Once all the factors relative to the project have been defined and studied, an attempt is made to quantify as many variables as possible in order to arrive at a potential value or worth of the property.

As a mining project progresses from raw exploration through to the time when a management decision is made to develop and mine the property, a number of analyses will be conducted on the property, each of which will be based on increasing amounts of data, will require increasing amounts of time (and therefore expense) to prepare, and will have increasing degrees of accuracy. For example, as exploration occurs on a mining property, the intersection of mineralization by a few drillholes typically triggers the need for some type of initial analysis to assist with necessary decision making. These types of studies are identified by various names (Gocht et al., 1988; Taylor, 1977), but in each case they are designed to answer questions pertaining to (1) what magnitude of deposit might exist rather than what is known to exist, (2) should further expenditures be incurred to look for what might exist, (3) should the project be abandoned, or (4) what additional effort and/or expense is necessary before making any of these decisions.

Assuming a favorable decision for continuation of the project, the next sequence of decisions must be predicated on studies utilizing much more detailed information. These so-called prefeasibility studies or intermediate economic studies are based on increasing amounts of data pertaining to geologic information, preliminary engineering designs and plans for mining and processing facilities, and initial estimates of project revenues and costs. They are constructed to support a continuum of decisions relating to the next major spending requirement. Intermediate economic studies of this type typically contain the following information and analysis (Gocht et al., 1988):

1. *Project Description:* geographic area, existing access routes, topography, climate, project history, concessionary terms, schedule for development of mine and any processing facilities.
2. Geology: regional geology, detailed description of the project area, preliminary reserve calculations, plans for detailed target evaluation.
3. *Mining:* geometry of the ore body, proposed mining plan (and alternatives), required plant and equipment.
4. *Processing:* technical descriptions of the ore and concentrate, processing facilities.
5. *Other Operating Needs:* availability of energy, water, spare parts, and equipment (diesel oil, explosives, replacement parts, etc.).
6. *Transportation:* description of the additional, necessary transportation facilities (roads, air strips, bridges, harbors, rail lines).
7. *Towns and Related Facilities:* housing for workers, schools for children of workers, medical facilities, company offices.
8. *Labor Requirements:* estimates of work force broken down according to qualifications (skills) and local availability.
9. *Environmental Protection:* plans to reduce or minimize environmental damage, description of relevant environmental legislation.
10. *Legal Considerations:* review of mining laws, taxation, foreign-investment regulations, political risk.
11. *Economic Analysis:* cost estimates for plant and equipment, infrastructure, materials, labor, other factors; market analysis, including production, consumption, and price formation for the relevant minerals; revenue forecasts based on expected production and mineral prices; cash flow and net present value analysis; sensitivity analysis.

Assuming the project continues to appear favorable throughout the intermediate economic studies, as these studies progressively focus more on engineering and economic aspects and less on geologic parameters, the project must be formally assessed through a comprehensive feasibility study. The feasibility study represents a detailed analysis of all the parameters contained in the intermediate economic studies, along with other pertinent factors relating to political and legal aspects affecting project viability. Specific data requirements for incorporation into feasibility studies are contained in the following segment of this chapter; however, in general, the study contains analyses of the project's geology and deposit characteristics, mineralogy, mineral processing characteristics, designs and plans for mining and processing equipment requirements, construction schedules, investment requirements and timing, estimates of revenues and costs, marketing plans, cash flow calculations, sources and methods of financing, and risk and sensitivity analyses of important project variables. As stated previously, the purpose of the feasibility study is to assess the technical and economic viability of the project and to assist the organization in making the "go/no-go" decision regarding project development.

Although there is no prescribed format for reporting the results of a project feasibility study, the final report must fulfill the following essential functions (Taylor, 1977):

1. Provide a comprehensive framework of established and detailed facts concerning the mineral project.
2. Present an appropriate scheme of exploitation complete with plans, designs, equipment lists, etc., in sufficient detail for accurate cost estimation and associated economic results.
3. Indicate the most likely profitability on investment in the project, assuming the project is equipped and operated as specified in the report.
4. Provide an assessment of pertinent legal factors, financing alternatives, fiscal regimes, environmental regulations, and risk and sensitivity analyses on important technical, economic, political, and financial variables affecting the project.
5. Present all information in a manner intelligible to the owner and suitable for presentation to prospective partners or to sources of finance. The document must be "bankable."

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* This chapter draws heavily from various chapters in *Mine Investment Analysis* by D.W. Gentry and T. J. O'Neil (1984).

6.2.1 DATA REQUIREMENTS

Nothing improves the results of a project feasibility study more than good input data. Unfortunately, those preparing feasibility studies for mining projects never possess all the information they would like. In addition to inadequacy or unavailability of some needed data, care must be taken not to overlook any variable that may influence project viability. In this regard, it is often helpful to compile an outline of factors to be considered when preparing feasibility studies on mining properties.

6.2.1.1 Factors for Consideration

Table 6.2.1. is an outline of some of the pertinent variables that must be studied, considered, and analyzed when evaluating mining properties. The significance of each variable will be a function of the specific property being investigated and the mineral commodity (metallic, nonmetallic, fuel) involved. Nonetheless, all these variables should be assessed during preparation of the final feasibility study.

A review of Table 6.2.1. suggests there are some fundamental issues that are applicable to all mining property feasibility studies—regardless of the commodity involved. For instance, one of the first tasks associated with any mining property is estimating the magnitude and quality of the ore reserves (Chapter 5.6). Ore is, of course, an economic term and is a function of commodity prices, production costs, mining method, recoveries, dilution, and a number of other variables. Because ore reserves are determined by ever-changing economic conditions, the exact amount of ore contained in a deposit cannot be precisely determined until production ceases. This uncertainty with respect to ore reserves has a significant impact on the evaluation of mineral properties where long-term contracts at stipulated selling prices are not available.

Production technology is another key area of concern when performing mining feasibility studies. Technological advancements in equipment, mineral processing, and other areas can significantly impact projected operating and capital costs. A good example of the impact of technology changes on mining costs is the comparison between direct mining costs per ton (tonne) of rock for underground and surface operations. While underground operating costs have been increasing at significant rates in recent years (mainly because of the lack of major technological advancements and the labor intensity that exists), unit operating costs at surface operations have changed far less. The technological advancements in mining equipment, yielding productivity increases, have prevented direct operating costs in surface operations from escalating at the rates experienced by underground producers. The relative stability in surface operating costs has not been entirely free, however. The technological advancements in mining equipment that contributed to this stability in operating costs carried with them significant increases in capital costs.

Operating and capital cost requirements must be determined separately when formulating the feasibility study. However, in the limiting case, it is the combination of the magnitude and timing of both these costs that ultimately influence the analysis. Any changes in future production technologies must be carefully analyzed and the impact assessed on overall operating and capital cost requirements. The estimation of capital and operating costs for mining projects that have not progressed to the detailed planning and layout stage is discussed in Chapter 6.3.

Another area of fundamental interest in feasibility studies of mining properties that requires considerable data generation and analysis is the estimation of project revenues. The timing and magnitude of mining revenues depends upon factors such as ore reserves, production rates, commodity prices, markets, and metallurgical recoveries. These variables are often extremely difficult to estimate or predict—particularly for commodities traded in international markets. This topic is discussed in more detail in the following portion of this chapter.

The overall operating environment is another area of major concern. In recent years, the national and, to a lesser but growing extent, the international operating environment of mining properties has been impacted significantly by environmental and other regulatory requirements. These constraints have invariably increased operating and capital cost requirements for the industry and have reduced or delayed the production of mineral commodities. The operating environment of mining operations is also affected by direct economic variables, such as royalties and taxes mandated by federal, state, and local taxing authorities. All these costs, whether direct or indirect, impact profit margins, ore reserves, mineral conservation, and ultimately project viability.

6.2.1.2 Variable Quantification

As stated previously, the objective of any feasibility study is to assess, clarify, and ultimately quantify the basic factors that govern the chances for project success. Once all the geologic, engineering, and other technology-related factors relative to the property have been defined and studied, an attempt is made to quantify as many of the variables as possible to arrive at a potential value or worth of the property. In this regard, two general categories of quantification are extremely important yet quite troublesome to compute: revenue estimation and cost estimation.

Revenue Estimation: Annual mine revenue is calculated by multiplying the number of units produced and sold throughout the year by the sales price received per unit. While the arithmetic associated with calculating annual mine revenue is trivial, determining the best value to use for each of these two critical variables is much more difficult.

There are a number of important considerations in estimating the number of units produced and sold annually. For example, estimates must be made of the tonnage of ore produced, the grade of ore mined (including dilution), the percentage recovery of the valuable mineral in the ore, and, finally, the number of payable units available for sale.

The second major component of the mine revenue calculation is the unit sales price. Estimating future mineral prices—particularly prices far enough into the future to be of use in mine investment analysis—is much more difficult than estimating the production-related variables and is an exercise in which a high error of estimation invariably exists. As pointed out in Section 2.0, mineral prices, like those of any other product, are ultimately determined by supply and demand. However, there are major complications on both sides of the supply-demand equation that seriously impair the value of quantitative econometric modeling for estimating mineral prices.

Clearly, the analysis of supply and demand for most mineral commodities is complicated. In fact, few analysts are willing to suggest that reliable forecasts of prices useful in mine investment analysis are possible. The current popular approach to this problem is for analysts to occupy safer ground and issue price *projections*—that is, prices that are likely if certain assumed events actually occur.

Regardless of the difficulties associated with forecasting or projecting mineral prices into the future, estimates of mining project annual revenue must be established. Generally, fungible

Table 6.2.1. Salient Factors Requiring Consideration in a Mining Project Feasibility Study

I. Information on Deposit
  A. Geology
    1. Mineralization: type, grade, uniformity
    2. Geologic structure
    3. Rock types: physical properties
    4. Extent of leached or oxidized zones
    5. Possible genesis
  B. Geometry
    1. Size, shape, and attitude
    2. Continuity
    3. Depth
  C. Geography
    1. Location: proximity to population centers, supply depots, services
    2. Topography
    3. Access
    4. Climatic conditions
    5. Surface conditions: vegetation, stream diversion
    6. Political boundaries
  D. Exploration
    1. Historical: district, property
    2. Current program
    3. Reserves
      a. Tonnage-grade curve for deposit, distribution classification; computation of complete mineral inventory (geological and mining reserves) segregated by ore body, ore type, elevation and grade categories
      b. Derivation of dilution and mining recovery estimates for mining reserves.
    4. Sampling: types, procedures, spacing
    5. Assaying: procedures, check assaying
    6. Proposed program
II. Information on General Project Economics
  A. Markets
    1. Marketable form of product: concentrates, direct shipping ore, specifications, regulations, restrictions
    2. Market location and alternatives: likely purchasers, direct purchase vs. toll treatment
    3. Expected price levels and trends: supply-demand, competitive cost levels, new source of product substitutions, tariffs
    4. Sales characteristics: further treatment, sales terms, letters of intent, contract duration, provisions for amendments and cost escalations, procedures/requirements for sampling, assaying, and umpiring.
  B. Transportation
    1. Property access
    2. Product transportation: methods, distance, costs
  C. Utilities
    1. Electric power: availability, location, ownership right-of-way, costs
    2. Natural gas: availability, location, costs
    3. Alternatives: on-site generation
  D. Land, Water, and Mineral Rights
    1. Ownership: surface, mineral, water, acquisition or securement by option or otherwise, costs
    2. Acreage requirements: concentrator site, waste dump location, tailings pond location, shops, offices, changehouses, laboratories, sundry buildings, etc.
  E. Water
    1. Potable and process: sources, quantity, quality, availability, costs
    2. Mine water: quantity, quality, depth and service, drainage method, treatment
  F. Labor
    1. Availability and type: skilled/unskilled in mining
    2. Rates and trends
    3. Degree of organization: structure and strength
    4. Local/district labor history
    5. Housing and transport of employees
  G. Government Considerations
    1. Taxation: federal, state, local
      a. Organization of the enterprise
      b. Tax authorities and regimes
      c. Special concessions, negotiating procedures, duration
      d. Division of distributable profits
    2. Reclamation and operating requirements and trends: pollution, construction, operating and related permits, reporting requirements
    3. Zoning
    4. Proposed and pending mining legislation
    5. Legal issues: employment laws, licenses and permits, currency exchange, expatriation of profits, agreements among partners, type of operating entity for tax and other purposes.
  H. Financing
    1. Alternatives: sources, magnitudes, issues of ownership
    2. Obligations: repayment of debt, interest
    3. Type of operating entity: organizational structure
    4. Division of profits: legal considerations
III. Mining Method Selection
  A. Physical Controls
    1. Strength: ore, waste, relative
    2. Uniformity: mineralization, blending requirements
    3. Continuity: mineralization
    4. Geology: structure
    5. Surface disturbance: subsidence
    6. Geometry
  B. Selectivity
    1. Dilution, ore recovery estimates
    2. Waste mining and disposal
  C. Preproduction Requirements
    1. Preproduction development or mining requirements: quantity, methods, time
    2. Layout and plans: schedule
    3. Capital requirements
  D. Production Requirements
    1. Relative production
    2. Continuing development: methods, quantity, time requirements
    3. Labor and equipment requirements
    4. Capital requirements vs. availability
IV. Processing Methods
  A. Mineralogy
    1. Properties of ore: metallurgical, chemical, physical
    2. Ore hardness
  B. Alternative Processes
    1. Type and stages of extraction process
    2. Degree of processing: nature and quality of products
    3. Establish flowsheet: calculation of quantities flowing, specification of recovery and product grade
    4. Production schedule
  C. Production Quality vs. Specifications
  D. Recoveries and Product Quality
    1. Estimate effects of variations in ore type or head grade
  E. Plant Layout
    1. Capital requirements
    2. Space requirements
    3. Proximity to deposit
V. Capital and Operating Cost Estimates
  A. Capital Costs
    1. Exploration
    2. Preproduction development (may also be considered operating costs)
      a. Site preparation
      b. Development of deposit for extraction
    3. Working capital
      a. Spares and supplies (inventory)
      b. Initial operations
      c. Financing costs (when appropriate)

4. Mining
   a. Site preparation
   b. Mine buildings
   c. Mine equipment: freight, taxes and erection costs, replacement schedule
   d. Engineering and contingency fees
5. Mill
   a. Site preparation
   b. Mill buildings
   c. Mill equipment: freight, taxes and erection costs, replacement schedules
   d. Tailings pond
   e. Engineering and contingency fees

B. Operating Costs
1. Mining
   a. Labor: pay rates plus fringes
   b. Maintenance and supplies: quantities, unit
   c. Development
2. Milling
   a. Labor: pay rates plus fringes
   b. Maintenance and supplies: quantities, unit costs
3. Administrative and supervisory
   a. Overhead charges
   b. Irrecoverable social costs

Source: Gentry and Hrebar, 1978; Taylor, 1977.

commodities—such as most metals traded on exchanges—suffer from the greatest future price uncertainty. Most metal markets are notoriously cyclical, and the amplitude and the period of the cycles defy accurate prediction. The recommended approach to mineral price forecasting is not limited to the application of any one or two specific analytical techniques, and it definitely is not a mechanical process. Rather, it is a painstaking blend of economic theory, industry analysis, market analysis, and competitor analysis combined with sound, experienced judgment.

Cost Estimation: The economic evaluation portion of a feasibility study ultimately must be based on information that provides an answer to the question, "what is it going to cost?" Unfortunately, the answer to this question is not simple, primarily because of the significant misunderstandings associated with cost data. Therefore, the components of so-called "total production cost" or "total operating cost," for example, must be carefully identified and defined.

When preparing a mine feasibility study, it is essential also to distinguish between operating costs, expenses, and capital costs. *Operating costs* are considered to be all expenses incurred at the plant site, whereas *general expenses* are off-site management or corporate-level expenditures. This latter classification of expenses may be directly related to mine or plant size, or it may contain indirect items incurred by headquarters and allocated across all production divisions in accordance with some corporate allocation scenario.

Direct costs, or variable costs, relate to items such as labor, materials, energy, and supplies that are consumed directly in the production process and are used roughly in direct proportion to the level of production. On the other hand, *indirect costs,* or fixed costs, are expenditures that are independent of the level of production—at least over certain ranges. It is obvious that, in the limit, few costs are absolutely fixed.

Capital costs (or first cost, or capital investment) are those expenditures made to acquire or develop capital assets, the benefits of which will be derived over several years. The largest portion of capital costs is incurred in the initial stages of project start-up, but some capital expenditures are incurred annually throughout the life of the mine.

In general, capital costs fall into one of three classes, depending on the treatment of the cost for tax purposes. These are depreciable investment, expensed or amortizable investment, and nondeductible investment. Because of differing tax treatments, the *type* of capital expenditure involved can be a very important factor in the evaluation of a new project, in addition to its magnitude.

The estimation of operating and capital costs for mining projects is extremely difficult and must be performed with great care. Chapter 6.3 illustrates procedures for estimating initial capital and operating costs for mining projects being analyzed via intermediate economic studies. However, the final feasibility must contain operating and capital cost estimates based on actual design and layout drawings, manning tables, flow charts, and equipment lists, specifications, and manufacturer quotations. These estimates should be predicated on data relative to unit operations, job functions, job requirements, timetables, and so on. This requires considerable time and effort; there are few, if any, acceptable shortcuts.

6.2.2 CASH FLOW ANALYSIS

The importance of the *pro forma* income statement in establishing the value of a mining project via the income (earnings) approach was alluded to in Chapter 6.1. Inasmuch as there are generally major differences between accounting profits and actual net cash benefits derived from an investment, investors are using almost exclusively the concept of project cash flows as the primary measure of real benefits produced by a capital project. This is predicated on the knowledge that cash flow analyses and accounting concepts depict investments differently, as a result of the timing of costs, and on the belief that the proper method for evaluating a capital investment is to compare the present investment outlay with the expected positive net cash flows that will accrue from the project in the future. In making this comparison, it is essential that the timing of the various cash flows be recognized by the use of an appropriate interest (discount) rate. This aspect of cash flow analysis is discussed briefly in the following portion of this chapter.

As indicated, cash flow analyses relate the expenditures associated with investment to the subsequent revenues or benefits generated from such investment. Cash flows are routinely calculated on an annual basis for evaluation purposes and are determined by subtracting the annual cash outflows from the annual cash inflows that result from the investment. Consequently, a cash flow analysis may be performed for any investment with which income and expenditure are associated. Also the annual cash flows resulting from an investment may be either positive or negative. Typically, the net annual cash flows for a new mining property will be negative during the preproduction years due to large capital expenditures. After production commences, the cash flows will usually be positive, and an inflow of cash results from investment in the project.

In the US income tax law, net annual cash flow is treated basically as a combination of two components: the return *on* the investment and the recoupment *of* the investment. In the minerals industry, net cash flow is generally defined as net profit after taxes plus depreciation and depletion minus capital expenditures and working capital. Within this definition, net profit after taxes represents the return on the investment, whereas depreciation and depletion represent the recoupment of the investment.

In a cash flow analysis, each investment receives credit for income taxes saved. Since the accounting allowances for depreci-

Table 6.2.2. Components and Basic Calculation Procedure for Developing Cash Flows

Calculation	Component
	Revenue
Less	Royalties
Equal	Gross income from mining
Less	Operating costs
Equal	Net operating income
Less	Depreciation and amortization allowance
Equal	Net income after depreciation and amortization
Less	Depletion allowance
Equal	Net taxable income
Less	State income tax
Equal	Net federal taxable income
Less	Federal income tax
Equal	Net profit after taxes
Add	Depreciation and amortization allowances
Add	Depletion allowance
Equal	Operating cash flow
Less	Working capital
Equal	Net annual cash flow

Source: Gentry and O'Neil, 1984.

Table 6.2.3. Parameters for Consideration in Cash Flow Analysis of a Mining Property

Preproduction Period	
Exploration expenses	Land and mineral rights
Water rights	Environmental costs
Mine and plant capital requirements	Development costs
Sunk costs	Financial structure
Working capital	Administration
Production Period	
Price	Capital investment—replacement of expansions
Processing costs	Royalty
Recovery	Mining cost
Postconcentrate cost	Development cost
Revenues and percent removable	Exploration cost
Grade	General and administration
Investment tax credit	Insurance
State taxes	Production rate in tons per year
Federal taxes	Financial year production begins
Depletion rate	Percent production not sent to processing plant
Depreciation	Operating days per year
Postproduction	
Salvage value	Contractual and reclamation expenditures

Source: Laing, 1977.

ation and depletion reduce the amount of taxable income (and therefore reduce the amount of taxes paid), they have the effect of saving the organization money. Consequently, they constitute a credit in the cash flow calculation and are added to net income after taxes. It must be recognized that depreciation and depletion are noncash items and do not actually "flow" anywhere.

Table 6.2.2. illustrates the components and basic calculation procedure for determining annual cash flows for a mining project. Table 6.2.3. lists some of the more important parameters relating to preproduction, production, and postproduction mining activities that require consideration in the preparation of cash flow analyses. The appropriate use and manipulation of these input variables are an extremely important facet of the cash flow analysis. The concept of a cash flow analysis is a particularly useful technique for the evaluation of mineral-related projects because of the important impact of the depletion allowance in the United States.

In view of the foregoing, it is worthwhile to reiterate the fact that in a cash flow analysis, each investment receives credit for income taxes saved. Therefore, for profitable organizations, it is advantageous to maximize pretax deductions and thereby reduce the amount of taxable income and, consequently, income taxes paid. In order to take advantage of these tax savings as soon as possible, the firm would opt to expense all possible expenditures in the year incurred as opposed to capitalizing them followed by subsequent write-offs over the amortization period. Although the total amount of the pretax deduction would be the same in either case, by expensing as soon as possible, the firm realizes an earlier return of the resulting tax savings. This early return of tax savings enables the firm to utilize these dollars sooner than would otherwise be possible.

6.2.3 TIME VALUE OF MONEY

If it were not for the existence of interest, the analysis of investment opportunities would be greatly simplified. In the absence of interest, investors would be indifferent as to when cash outlays are made or cash benefits received. It would, in fact, be irrelevant whether the outlays preceded or followed inflows, as long as both amounts are known with certainty.

Of course, it does make a considerable difference whether, for example, a firm receives $1 million now or five years from now. The reason is that money does have a value, which is a function of time. Interest is how this time value is measured.

Interest is generally defined as money paid for the use of borrowed money. Interest may be likened to a rental charge for using an asset over some specific time period. The rate of interest is the ratio of the interest chargeable at the end of a specific period of time to the money owed, or borrowed, at the beginning of that period. Interest exists to compensate for a number of concerns experienced by lenders; these are related primarily to risk, inflation, transaction costs, opportunity costs, and postponement of pleasures. The level of interest is, like the price of other assets, determined by supply and demand.

The history, philosophy, and theoretical underpinnings of interest are covered exhaustively in a large number of texts and reference books (e.g., Newman, 1980; Smith, 1973). This material need not be repeated here.

It is sufficient simply to recognize that money has earning power. That is, the timing of when payments are made and earnings are received in a capital project is very important.

6.2.3.1 Interest Formulas

Cash flows at different points in time are related by a series of six basic interest formulas. These formulas in turn are based on five variables, as follows:

> F is a future sum of money, P is a present sum of money, A is a payment in a series of n equal payments, made at the end of each period of interest, i is effective interest rate per period, and n is number of interest periods.

Every interest problem is composed of four of these variables: three are given, and the fourth must be determined. The standard notation for describing the particular type of problem involved lists all four variables of concern, in the following manner:

For example: *(F/P,i,n)*, means, "Find *F*, given *P,i,* and *n*"

Similarly, *(A/F,i,n)*, means, "Find *A*, given *F,i,* and *n*"

The notation is often shortened to simply *F/P* or *A/F*, etc., where it is understood that *i* and *n* are given. This notation is widely accepted in the field of engineering economy.

The six basic interest equations are developed and described.

1. Single payment compound amount, *(F/P,i,n)*.

$$F = P(1+i)^n \qquad (6.2.1)$$

Example 6.2.1. Find the amount that will accrue at the end of 7 years if \$1250 is invested now at 8%, compounded annually. Given: P = \$1250, n = 7 years, i = 8%.

Solution.

$$F = P(1+i) = \$1250\ (1+0.08)^7 = \$2142.28$$

The quantity $(1+i)^n$ = *(F/P,i,n)* has, like other interest factors, been tabulated for various values of *i* and *n* in so-called interest tables. Such tables are provided in the appendix of this volume for select interest rates and discrete compounding.

However, many hand-held calculators can now solve such problems directly so that interest tables are becoming less necessary.

2. Single payment, present worth *(P/F,i,n)*.

This is Eq. 6.2.1 solved for *P*.

$$P = F/(1+i)^n \qquad (6.2.2)$$

Example 6.2.2. If \$6500 will be needed in 5 years, how much should be invested now at an interest rate of 7.5%, compounded annually?

Solution. Given: F = \$6500, n = 5 years, i = 7.5%

$$P = F(1+i)^n$$
$$P = \$6500/(1+0.075)^5 = \$4527.63$$

Using interest tables to solve this problem becomes somewhat more difficult because one must interpolate between 7 and 8%. For most interest problems, a linear interpolation is satisfactory. The widespread use of programmed calculators and computers for solving interest problems facilitates exact solutions so that interpolation is often unnecessary. However, the intellectual problem of calculating exact solutions from highly inexact data is of major concern in all investment studies.

3. Uniform series, compound amount *(F/A,i,n)*.

Here the concern is to determine the terminal amount when equal annual payments are made to an interest-bearing account for a specified number of years. It is important at this point to recall that *A* is defined as occurring at the *end* of the interest period. Therefore, for

$$n = 1,\ F = A$$
$$n = 2,\ F = A(1+i) + A \qquad (6.2.3)$$
$$n = 3,\ F = A(1+i)^2 + A(1+i) + A$$

or in general,

$$F = A(1+i)^{n-1} + A(1+i)^{n-2} + \ldots + A(1+i) + A$$
$$= A[(1+i)^{n-1} + (1+i)^{n-2} + \ldots + (1+i) + 1]$$

Multiplying Eq. 6.2.3 by $(1+i)$ yields

$$F(1+i) = A[(1+i)^n + (1+i)^{n-1} + \ldots \qquad (6.2.4)$$
$$+ (1+i)^2 + (1+i)]$$

Now subtracting Eq. 6.2.3 from 6.2.4,

$$iF = A[(1+i)^n - 1]$$

or (6.2.5)

$$F = \frac{A[(1+i)^n - 1]}{i}$$

Example 6.2.3. If payments of \$725 are made at the end of each year for 12 years to an account which pays interest at the rate of 9% per year, what will be the terminal amount?

Solution .

$$F = \frac{A[(1+i)^n - 1]}{i}$$

$$= \frac{725[(1+0.09)^{12} - 1]}{0.09} = \$14{,}602.02$$

It obviously makes a considerable difference if the annual payments are made at the beginning of the year (an *annuity due)* rather than at the end of the year (an *immediate annuity).* However, the end-of-year convention is more common, and nearly all interest tables and computer programs are constructed on this basis. To verify easily that any particular table or computer program adheres to this convention, note that for n = 1, F = A regardless of the interest rate.

4. Uniform series, sinking fund, *(A/F,i,n)*.

Solving Eq. 6.2.5 for *A* will enable the analyst to determine what annual payments must be made to accumulate a specified amount by some future date at an interest rate *i:*

$$A = iF/[(1+i)^n - 1] \qquad (6.2.6)$$

Example 6.2.4. With interest at 6%, how much must be deposited at the end of each year to yield a final amount of \$2825 in 7 years?

Solution .

$$A = iF/[(1+i)^n - 1]$$
$$A = 0.06\ (\$2825)/[(1+0.06)^7 - 1]$$
$$= \$336.58$$

The concept of a sinking fund is well known, so that the *A/F* interest factor is often called the *sinking fund factor*.

5. Uniform series, present worth *(P/A,i,n)*.

This type of problem arises when the current value of a future series of cash flows is desired. This is often the case with investments in securities where an expenditure now will provide equal interest or dividend payments for several future periods.

For Eq. 6.2.5,

$$F = \frac{A[(1+i)^n - 1]}{i}$$

Substitute Eq. 6.2.1,

$$F = P(1+i)^n$$

$$P(1+i)^n = \frac{A[(1+i)^n - 1]}{i}$$

and

$$P = \frac{A[(1+i)^n - 1]}{i(1+i)^n} \qquad (6.2.7)$$

Example 6.2.5. An investment will yield $610 at the end of each year for 15 years. If interest is 10%, what is the maximum purchase price (i.e., present value) for this investment?

Solution.

$$P = \frac{A[(1+i)^n - 1]}{i(1+i)^n}$$

$$= \frac{610[(1+0.1)^{15} - 1]}{0.1(1+0.1)^{15}}$$

$$= \$4639.71$$

6. Uniform series, capital recovery, *(A/P,i,n)*

This is the reverse of the previous problem, and the equation is derived simply by solving Eq. 6.2.7 for *A:*

$$A = \frac{iP(1+i)^n}{[(1+i)^n - 1]}$$

Example 6.2.6. If an investment opportunity is offered now for $3500, how much must it yield at the end of every year for 6 years to justify the investment if interest is 12%?

Solution.

$$A = \frac{iP(1+i)^n}{[(1+i)^n - 1]}$$

$$= \frac{0.12\ (\$3500)\ (1+0.12)^6}{[(1+0.12)^6 - 1]}$$

$$= \$851.67$$

The preceding formulas pertain to annual compounding and, where *A* enters the problem, to annual cash flows. In practice, these constraints are routinely violated as neither payments/receipts nor compounding need be on an annual basis. In fact, there is a continuous spectrum of possibilities with discrete annual compounding and discrete cash flows at one end of the continuum, and continuous compounding and continuous flow of funds at the other end. The reader interested in learning more about the procedures for handling specific cases within this spectrum is referred to Chapter 3 in Gentry and O'Neil (1984).

6.2.3.2 Interest Factor Relationships

The relationships between the interest factors identified in the preceding formulas are often not as clear as they might be—particularly when values are extracted from interest tables. Following are some relationships based on given values of *i* and *n*.

1. The single payment compound amount factor *(F/P,i,n)* and the single payment present value factor *(P/F,i,n)* are reciprocals:

$$\frac{1}{(P/F,i,n)} = (F/P,i,n)$$

2. The sinking fund factor *(A/F,i,n)* and the compound amount factor for an annuity *(F/A,i,n)* are reciprocals:

$$\frac{1}{(A/F,i,n)} = (F/A,i,n)$$

3. The capital recovery factor *(A/P,i,n)* and the present value factor for an annuity *(P/A,i,n)* are reciprocals:

$$\frac{1}{(A/P,i,n)} = (P/A,i,n)$$

4. The capital recovery factor *(A/P,i,n)* is equal to the sinking fund factor *(A/F,i,n)* plus interest rate:

$$(A/P,i,n) = (A/F,i,n) + i$$

5. The present value factor for an annuity *(P/A,i,n)* is equal to the sum of the first n terms of single payment value factors *(P/F,i,n):*

$$(P/A,i,n) = (P/F,i,1) + (P/F,i,2) + \ldots + (P/F,i,n)$$

6. The compound amount factor for an annuity *(F/A,i,n)* is equal to 1.00 plus the sum of the first (n-l) terms of the single-payment compound amount factor *(F/P,i,n):*

$$(F/A,i,n) = 1.00 + (F/P,i,1) + (F/P,i,2) + \ldots + (F/P,i,n\text{-}1)$$

6.2.4. SELECTING A DISCOUNT RATE

As indicated in the previous section of this chapter, future project receipts and expenditures must be discounted to permit valid comparisons with current cash flows. Although the concept of discounting is widely accepted, selection of an appropriate discount rate has been the source of considerable debate and much disagreement.

The fact that interest exists suggests that all money has a cost associated with its use. The cost of this money may be the result of either explicit or implicit charges or some combination of the two. Indeed, any time an individual or a firm consumes less than the total earnings generated in a specified time period, that individual or corporation has either consciously or unconsciously decided to "invest" these excess funds in some type of activity—even if the funds are maintained in a cash account.

Intuitively, it seems reasonable to accept the fact that no money is free. Certainly, there is a cost associated with raising investment capital. For instance, there is a cost associated with going to the debt markets and borrowing money for investment purposes. This explicit cost of borrowing results from the fact that interest exists. Similarly, it is important to recognize that an investor who purchases the firm's common stock (equity) has the same expectations of making a return on his investment as a banker who loans money. Additionally, it is important to

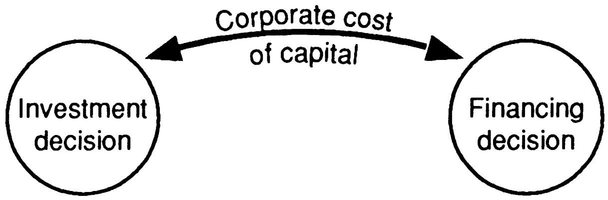

Fig. 6.2.1. Relationship between investment decision and financing decision.

recognize that a firm may allocate investment capital to alternative investment opportunities that may be available and that promise specific rates of return on the investment capital. This concept of alternative investment opportunities suggests that there also are *opportunity costs* (implicit costs) associated with investment capital. Clearly, then, there are explicit and implicit costs associated with procuring and utilizing investment capital, and, at least to some extent, these costs are influenced by inflationary trends and expectations.

Remembering that the firm's primary objective should be the maximization of shareholder wealth, it then intuitively seems reasonable to suggest that the firm should not invest in any project where the anticipated return does not exceed the cost of funds (capital) committed to the project. Indeed, if the firm always invests in projects having returns in excess of the cost of capital committed to them, then the wealth of the firm (as measured by the price of common stock) should be increased to the stockholders. This cost of funds, known as the *cost of capital,* is the direct linkage between investment policy and the firm's objective.

Further, the cost of capital also relates the financing decision to the investment decision. Indeed, the cost of capital is the *only* link between these two decisions. The relationship between the financing decision and the investment decision in corporate finance via the cost of capital is illustrated in Fig. 6.2.1.

An alternative way to view the cost of capital is to recognize that the *minimum acceptable rate of return* (MARR) on an investment may be defined as the minimum rate of return a firm must earn on its investments in order to leave unchanged the value of existing shares of its common stock. Investing at the cost of capital will achieve this objective, and therefore the MARR is quantitatively equal to the cost of capital. The logical conclusion from this observation is that this cost of capital represents the "hurdle rate" or the appropriate discount rate to be used in conjunction with discounted cash flow analyses of investment opportunities.

6.2.4.1 Components of Discount Rate

Following are four major components to the discount rate. Some of these components stem from sound theoretical and practical foundations, while others are judgmental. Rarely is the combined discount rate calculated by summing these separate components, as the capital markets do a good job of determining the overall cost of funds to various uses. Nonetheless, the combined rate must be sufficient to cover these four elements.

Base Opportunity Cost: There is always some base opportunity cost associated with procuring and utilizing investment capital. This cost is the return foregone by diverting funds from the next most attractive project and exists whether the funds are obtained externally or internally. In either case, funds appropriated for a particular use will carry a cost related to the return those funds could have achieved elsewhere. The other components of the discount rate are typically added to this base opportunity cost for investment capital.

Transaction Cost: Whether corporate investment capital is procured from the debt market or the equity market, the firm will experience transaction costs regardless of whether or not a new security issue is involved. These costs include broker and investment banker fees, costs of prospectuses and various filings, sales discounts, and other flotation costs. Although the aggregate transaction costs are generally much less than the base opportunity cost, they are significant and should be incorporated into the determination of the appropriate discount rate.

Increment for Risk: The cost of procuring funds, either debt or equity, includes a component for the investor's perception of risk. Consequently, some organizations add some *additional* percentage increment for risk to the cost of raising investment capital before applying this rate to discounted cash flow analyses. The rationale here is that high-risk projects should be discounted at some higher rate to compensate for their relative riskiness.

There are two fundamental problems associated with this approach to accounting for project risk. First, the increments for risk assigned to the discount rate are subjective by nature and cannot be quantitatively equated to project risk in any systematic manner. The second problem with this procedure is the manner in which project risk (business risk) is addressed. Clearly, the discount rate only adjusts for the present value of money; it does not by itself adjust for project business risk, as measured by the variability in annual project cash flows. Thus the discount rate provides no information on how annual project cash flows might vary, nor does it provide any insight into why these cash flows might fluctuate from year to year. Consequently, the discount rate does not provide a measure of project risk and, in general, should not be used to do so. Superior approaches to accounting for project risk are discussed in Chapter 6.5.

Increment for Inflation: It seems reasonable to assume that investment capital raised in the marketplace includes some cost premium for investor expectations of future inflation. Therefore, presumably a market-determined cost of capital includes a component that is based on investors' perceptions of future inflation. As a result, this market-determined cost of capital should be used as the appropriate discount rate when analyzing inflation-adjusted project cash flows. The amount of downward adjustment in the market-derived cost of capital for use with constant-dollar analyses can be shown to be roughly equal to the general rate of inflation.

6.2.4.2 Cost of Specific Types of Capital

The manner in which a firm chooses to finance itself will largely determine the cost of capital to the organization, since each source of financing has a specific cost. Thus the *capital structure* of the firm, defined as the mix of long-term sources of funds used to finance assets, becomes important in that the resulting cost of capital to the firm directly influences the investment decision. Since the firm's investment decisions influence common stock prices and therefore the value of the firm, the capital structure of the firm and the resulting cost of capital become key parameters in corporate finance.

Many corporations choose to develop a capital structure by utilizing a mixture of debt, preferred stock, and equity financing. Each of the various financial instruments available within these general categories has its own specific cost to the organization. The cost of each specific type of financing follows; however, the cost is essentially that rate which equates the funds received with all discounted future outlays relevant to that source. This may be expressed as

$$I_o = \frac{C_1}{(1+k)} + \frac{C_2}{(1+k)^2} + \ldots\ldots + \frac{C_i}{(1+k)^i} \quad (6.2.9)$$

where I_o is net proceeds received at time zero, C_i is cash outflow in the ith period, and k is cost of this specific source of financing. The C_i's are fairly easily determined in the case of debt and preferred stock financing. Equity financing costs, however, have generated a great deal of debate, and, while Eq. 6.2.9 applies in theory, one of several proposed simplifying approximations must be adopted to determine the costs of equity financing in practice.

Cost of Debt: In keeping with the preceding discussion, the cost of debt is that discount rate that equates net proceeds from the debt issue to the future cash outlays. Further, these future payments are net of taxes. This is very significant with debt financing since interest payments, unlike dividends, are tax deductible. If k_d is the cost of debt and k is the yield to maturity of the debt issue determined by the application of Eq. 6.2.9, then

$$k_d = (1 - t)^k \quad (6.2.10)$$

where t is the marginal corporate tax rate.

Cost of Preferred Stock: In theory, because preferred stock dividends do not represent a legal obligation on the part of the firm, preferred stocks offer a distinct advantage over debt financing. In practice, however, the omission of a preferred stock dividend is viewed by the financial community as an indication of serious financial difficulty in the firm. Thus firms usually view preferred stock dividend payments as de facto obligations.

The cost of preferred stock is usually computed in a manner similar to debt perpetuities since both instruments have fixed annual costs and no maturity. There is one important difference, however. Preferred stock dividend payments are not tax deductible to the firm.

If the annual dividend paid is D dollars, and I_o represents the net proceeds per share for a new stock issue, the cost of preferred stock, k_P, is

$$k_p = \frac{D}{I_o} \quad (6.2.11)$$

From an investment viewpoint, preferred stocks offer one advantage over bond investments to a corporation. Eighty-five percent of the dividend income to the corporation is nontaxable, while interest income from bonds is taxable at the full applicable rate.

Cost of Equity Capital: Equity financing comes from two sources—new common stock issues and retained earnings. Clearly, there is a cost of funds obtained through the issuance of new common stock. Proceeds from such a transaction must be invested to return sufficient earnings to maintain the firm's earnings per share. While all analysts agree that there are real costs involved in equity financing through the issuance of common stock, no such agreement exists on calculating these costs in practice.

The so-called *dividend valuation model* is one of the popular methods for calculating the cost of equity financing. The premise for this model is that common stocks only have a value because of an expected future stream of dividends. In theory, this model is consistent with the generalized cost of capital model previously discussed and may be represented as

$$P_o = \frac{D_1}{(1+k_e)} + \frac{D_2}{(1+k_e)^2} + \ldots\ldots + \frac{D_i}{(1+k_e)^i} = \sum_{i=1}^{\infty} \frac{D_i}{(1+k_e)} \quad (6.2.12)$$

where P_O is value of stock at time O, D_i is dividend payment in the ith year, and k_e is cost of equity applicable to firms in this risk class.

Obviously, dividend payments in future periods are unknown at the present time. Investors, however, have subjective estimates of what these payments will be, and these estimates are generally based on the current dividend and the historical long-run growth of dividends. An intuitive expression of these relationships might be:

$$k_e = \frac{D_o}{P_o} + g \quad (6.2.13)$$

where D_o is current dividend and g is annual growth rate of dividends.

Another popular method for computing the cost of common stock financing is the *earnings:price ratio* (E/P ratio). This approach is based on the assumption that investors really purchase earnings when buying common stock and that anticipated future earnings per share determine the value of a share of common stock.

The E/P model obviously is more suitable than the dividend valuation model for firms that pay low dividends or none at all. For firms that have strong earnings growth records, a growth term is sometimes appropriate in the E/P model to reflect shareholder expectations that such growth will continue. The mathematical expression for this model is

$$k_e = E/P_o + g \quad (6.2.14)$$

where E represents current earnings per share, P_O is value of stock at present, and g is growth term for earnings per share.

It should be noted that both of these models incorporate market prices for common stock. However, if a new common issue is planned, net proceeds per share, P_f should be used rather than the current market price, P_O. P_f will be lower than P_O by the amount of discount required to sell the new issue, plus the underwriting costs.

Although other models exist for estimating the cost of equity, they are more complex to use in practice and are not covered here (see Gentry and O'Neil, 1984, for more complete coverage).

That there is also a cost of *retained earnings* (net profits minus dividends) is less obvious than for common stock. However, if the firm retains, rather than distributes, earnings, and if it is dedicated to the maximization of shareholder wealth, it clearly must invest these funds above some minimum rate of return. Furthermore, the minimum rate in general is the cost of common stock financing, for this is the rate the investor implicitly agreed to by purchasing stock in the first place. If no investment opportunities are available to the firm above this minimum rate, all earnings should be returned to the owners (shareholders) to enable them to reinvest the funds themselves. Thus the cost of retained earnings is equal to the cost of common stock financing adjusted, where applicable, for flotation costs.

Readers interested in the application of the foregoing equations to calculating a firm's cost of capital are referred to Chapter 11 in Gentry and O'Neil (1984).

6.2.4.3 Weighted Average Cost of Capital

Given that each of the various types of financing available to a corporation has a specific cost, it seems reasonable to conclude that it should be possible to combine these various forms of financing into a corporate capital structure that would result in a minimum weighted average cost of capital to the firm. In other words, the implication is that there should be an optimum capital structure that minimizes the firm's weighted average cost of capital. Thus if a firm finances itself in this optimum manner, it should generate the lowest possible weighted average cost of capital. If this cost of capital is then used as the appropriate discount rate when evaluating new investment opportunities, the firm should find that more investments become acceptable. This should subsequently result in an optimum investment program and generate more wealth to shareholders.

Gentry and O'Neil (1984) provide more extensive discussions on the issues of optimum capital structures, marginal weighted average cost of capital calculations, and the practical problems associated with these calculations.

6.2.4.4 Common Errors in Evaluation

One of the most common errors made in discounted cash flow financial analysis involves using the cost of specific financing in evaluating projects rather than the average cost of all capital used by the firm. When a capital project involves an expenditure of, say, $20 million through a bank loan carrying an interest rate of 12%, it often is difficult to understand that (1) the interest charges from the loan should *not* be levied against the project for evaluation purposes, and (2) the appropriate discount rate to use is *not* 12% (or its approximate after-tax equivalent), but rather the marginal weighted average cost of *all* capital used by the firm.

Another common error in financial analyses is the inappropriate use of a market-derived cost of capital as the discount rate with constant dollar project cash flow estimates. These errors are sufficiently common to deserve further discussion.

Error No. 1: Charging Specific Capital Costs to the Project. The cost of capital, being the link between the investment decision and the financing decision, occupies a unique place in the evaluation procedure. All other project costs (i.e., operating costs, plant overhead, depreciation, etc.) are handled explicitly in the pro forma income statement. The cost of capital, however, *is* the discount rate and does not receive a separate line in these statements. While such a procedure may be applicable in other types of economic studies or analyses of financing alternatives, it is incorrect to do so in discounted cash flow analyses.

Unfortunately, this procedure has led many practitioners to believe that the cost of funds has been omitted in the analysis and should be listed separately in a manner similar to the other cost items. This misconception occurs most frequently when debt capital is raised and the accompanying interest payments are not charged directly to the project in the discounted cash flow return on investment (DCFROI) calculations. However, it is important to recognize that to do so would clearly double-count the cost of capital—once through the discount rate and a second time through the explicit interest charges. This point is illustrated in the following.

Example 6.2.7. (Gentry and O'Neil, 1984)

Suppose Hikki Mining Co. contemplates an investment opportunity having the following anticipated revenues and costs.

Investment = $100
Net annual cash flows (year 1-5) = $27.74
Cost of capital = 12%
NPV = $27.74 $(P/A, 12, 5) - 100 = 0$

In other words, the project should just break even after returning the investment plus the 12% cost of capital. This is verified in the following.

Solution .

	Year 1	2	3	4	5	Total	
Cost of capital @ 12%	12.00	10.11	8.00	5.63	2.97	38.71	
Total money owed before year-end payment	112.00	94.37	74.63	52.49	27.72		
General payment	27.74	27.74	27.74	27.74	27.74	138.70	Total payments
Money owed after year-end payment	84.26	66.63	46.86	24.75	—		
						100.00	Principal

Thus the discounting procedure assures that all project costs are covered—including return *of* and return *on* capital. To have levied financing charges explicitly against the project prior to discounting would clearly have forced the project to pay these costs twice.

Nonetheless, it continues to bother many that interest payments resulting from additional debt obligations are not charged directly to the project, even though these obligations must be met from the earning power of the firm's investments. This has caused some firms to incorporate a procedure whereby projects are evaluated on (1) a full equity basis and (2) various degrees of debt leveraging. The following example serves to illustrate the types of calculations resulting from this *improper* procedure.

Example 6.2.8. (Gentry and O'Neil, 1984)

An underground conveyor system is proposed to reduce haulage costs at a zinc mine. Benefits to be derived from this project are estimated to be $200,000 annual savings in operating costs. Depreciation is straight-line, and the income tax rate is 50%. Depletion is constrained by 22% of gross income from mining and can, therefore, be ignored as being constant in all alternatives. If the total investment is estimated to be $1,146,000, and the service life is 12 years with no salvage value, calculate the rate of return on equity capital for (1) all equity case, (2) 50/50, debt/equity financing, and (3) 95/5, debt/equity financing.

Assume the before-tax cost of debt is 8% compounded annually, and for simplicity assume that the entire principal amount will be repaid at the termination of the project.

Solution .

1. Equity investment
   A. $1,146,000
   B. $ 573,000
   C. $ 57,000
2. Debt investment to be repaid at the end of the year 12
   A. 0
   B. $ 573,000
   C. $1,088,700
3. Annual cash flows

	Case A	Case B	Case C
Net operating savings	$200,000	$200,000	$200,000
Less: depreciation	— 95,500	— 95,500	— 95,500
: interest	0	— 45,840	— 87,096
Pretax net income	104,500	58,660	17,404
Less: income tax	52,250	29,330	8,702
Incremental net profit	52,250	29,330	8,702
Add: depreciation	95,500	95,500	95,500
Incremental net cash flow	$147,750	$124,830	$104,202

4. DCF return on equity

Case A: $\$1,146,000 = 147,750\ (P/A,\ i,\ 12)$
$i = 7.5\%$

Case B: $573,000 = 124,830 ($P/A$, i, 12)–573,000)(P/F, i, 12)
i = 14.8%

Case C: $57,300 = 104,202 ($P/A$, i, 12)–1,088,700 (P/F, i, 12)
i = 182.5%

The foregoing analysis seems to suggest that the use of debt financing can turn a very marginal project into one yielding a phenomenal rate of return. However, it must be recognized that the DCFROI on the equity portion of the investment is always increased with leveraged financing if the after-tax cost of debt is less than the DCFROI of the project figured for the all-equity case. Thus the most marginal of investments can often be made to appear positively superb when a high percentage of debt capital is used!

A basic principle of investment analysis is that a project must stand on its own merits and must compete for funds on an equal basis with other investment opportunities. Each such opportunity must share fully in the benefits of debt leveraging if the same weighted marginal average cost of capital is used for all projects.

In summary, the following should always be observed in project investment analysis:

1. To avoid counting capital costs twice, do not charge costs of specific capital sources against the project. These charges are embedded in the discount rate.

2. Calculate returns on investments on total capital invested (i.e., all-equity case) to insure that all projects are compared on an equal basis.

Error No. 2: Using Specific Capital Costs as the Discount Rate. Debt financing is the source of another common error in capital project evaluations. There is a tendency on the part of some analysts to use the cost of a specific source of financing—rather than a weighted average of all capital sources—as the discount rate in measuring the attractiveness of a project. The following example illustrates this flaw.

Example 6.2.9. (after Quirin, 1967)

Global Mineral Ventures, Inc. was presented with a similar set of investment opportunities in three successive years, as shown:

	Year 1		Year 2		Year 3	
Project	**Amount of investment $**	**Rate of return $**	**Amount of investment $**	**Rate of return $**	**Amount of investment $**	**Rate of return $**
A	100,000	20	100,000	20	100,000	20
B	200,000	15	200,000	15	200,000	15
C	200,000	11	200,000	11	200,000	11
D	200,000	8	200,000	8	200,000	8
E	200,000	6	200,000	6	200,000	6

Solution.

In year one, Global had no long-term debt, and the corporate treasurer found that the full $900,000 could be raised by selling debentures bearing an annual interest rate of 5½%. He convinced Global's board that, since every project returned more than 5½%, they should all be accepted.

In year two, the treasurer was able to borrow a further $700,000 at 7%, and using the same logic as the previous year, accepted all projects except E, which offered a return below the 7% marginal cost of debt.

In the third year, however, Global's treasurer found only a limited amount of additional debt available to him—$100,000 at 18% from a finance company. Since this was still below the estimated 19% for a new equity issue, the treasurer used the debt to accept only project A.

Thus over the three-year period by using the cost of a specific source of debt as his investment criterion, the treasurer had invested $1.7 million at a weighted average return of 12%. It is clear, however, that the treasurer's eagerness to accept projects in year 1 precluded the acceptance of better projects in year 3. In fact, the same $1.7 million could have been invested to yield an average rate of 13.3% by accepting projects in the following sequence:

Year 1 A, B, C, D
Year 2 A, B, C
Year 3 A, B, C

This example illustrates that debt financing is only possible if an adequate equity base exists. Clearly, the marginal cost of capital is not simply the cost of debt but also the cost of equity that will be required in the future to support the added debt. Thus every investment must carry its proportionate share of the necessary—but higher cost—equity funds whenever debt is added to the capital structure of the firm. In the foregoing example, using the marginal weighted average cost of *all* capital sources would have resulted in rejecting the lower-value projects in year one, thereby permitting the acceptance of higher-value projects in year three.

In summary, it is important to recognize that it is incorrect to use the cost of a specific source of capital as the cash flow discount rate, or equivalently, as the minimum acceptable rate of return for capital projects.

Error No. 3: Using the Appropriate Discount Rate with Inflated and Constant Dollar Cash Flows. Some organizations insist on performing cash flow analyses in terms of constant dollars, while others prefer to perform analyses in terms of inflated cash flows. Although the preparation of constant dollar pro forma cash flows is highly recommended to promote better technical understanding of the project, discounting should always be performed on inflated dollars to avoid miscalculation of income taxes. Although there are specific virtues associated with both approaches, the important point here is to recognize that a market-determined cost of capital includes a component that is based on investors' perceptions of future inflation. As a result, this market-derived cost of capital should be used as the appropriate discount rate when analyzing inflation-adjusted project cash flows. If this rate is applied to project cash flows that are not adjusted for inflation, the project will be seriously *undervalued*. The amount of downward adjustment in the market-derived cost of capital for use with constant-dollar analysis is roughly equal to the general rate of inflation.

A more complete discussion of the impact and treatment of inflation in mining investment decision analysis is provided in Chapter 10 of Gentry and O'Neil (1984).

6.2.5 AN ITERATIVE PROCESS

This chapter has focused on the engineering and economic parameters associated with project feasibility studies. Estimates necessary for the quantification of many of these parameters in the pro forma income statement are provided in Chapter 6.3. The calculation of project net annual cash flows can be performed after completion of the pro forma income statement using the appropriate cost of capital or discount rate. Given these estimates of relative benefits, costs, and annual cash flows for a project, it then becomes necessary to convert these estimates into measures of relative desirability or attractiveness. The criteria and techniques typically utilized to determine project acceptability or desirability are the topics of discussion in Chapter 6.5.

It must be remembered, however, that completion of a feasibility study or an intermediate economic study incorporating one set of project parameters represents only one alternative in an iterative process. As noted in the introduction to this section, the process of evaluating mine investment opportunities is iterative nature. Each time a project variable or parameter changes, it is necessary to assess the impact of this change on all other project variables and on the subsequent financial results.

This iterative process must be repeated until the most economic design is achieved for the project being analyzed. This may require changes in cutoff grade, mining reserves, mine size or mining rate, or alternative extraction scenarios having differing capital and operating cost characteristics. Whatever the case, the most efficient combination of project parameters must be combined to meet the firm's primary objective: wealth maximization to its shareholders.

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Chapter 6.3
COSTS AND COST ESTIMATION

T. Alan O'Hara and Stanley C. Suboleski

6.3.1 ESTIMATION OF COSTS

A mineralized deposit should not be developed into a mine unless the estimated annual operating profit after taxes is judged to be sufficient to recover, with interest, the estimated capital cost of developing the mine. The accuracy of estimation of capital costs and operating costs depends on the quality of the technical assessment and knowledge of expected mining and mineral processing conditions.

6.3.1.1 Cost Estimation for Preliminary Feasibility Studies

Estimation of capital and operating costs of a proposed mining venture is usually required after ore reserves have been determined, but before major capital costs are committed for detailed ore exploration, mine design layout, detailed metallurgical studies, and general plant design. At this stage of preliminary feasibility studies, there is usually insufficient technical knowledge to accurately estimate costs, and costs are estimated approximately to provide guidance on probable mine feasibility, optimum plant size, and possible need for additional geological studies and further exploration to prove and extend possible ore reserves. Estimates of costs are based primarily on computed average costs of existing mining projects and operations, with appropriate allowances for general site conditions, mining methods, and milling processes.

The costs estimated in a preliminary feasibility study are unlikely to be more accurate than ± 20%, and this degree of accuracy is insufficient to provide a sound basis for major mine financing or confident assurance of a profitable mining operation.

6.3.1.2 Estimation of Costs for Detailed Feasibility Studies

Estimation of costs with an accuracy of ± 10%, which is needed for a detailed feasibility study, requires completion of extensive technical work and studies on mine planning, general plant layout and design, environmental studies, and assessment of supplies, labor, and equipment required for mining, milling, and service operations. The cost and time required for the completion of the technical activities to permit accurate cost estimation required for a detailed feasibility study may not be warranted if the preliminary study indicates the proposed mining venture will not be adequately profitable.

Accurate capital costs are estimated from the lengths, sizes, and unit costs of planned mine development; manufacturers' quotations for specific equipment; quantities and contractors' unit costs for excavation, concrete foundations, and installations of piping, electric services, and equipment.

Accurate operating costs are estimated from the quantities and unit costs of all components of supplies and labor, as determined by accurate knowledge of the ore body, the mine planning, and the plant design.

Any aspect of the mine, plant, or service facilities that is not adequately determined in terms of technical requirements and quantities of supplies, labor, and construction requirements is incorporated in a contingency allowance, that is added to capital costs and operating costs. The contingency allowance expresses the probability of capital costs and operating costs being higher than anticipated when it is difficult to determine the precise characteristics of an aspect of the mine or plant.

Because accurate cost estimation requires tailoring of mine and plant design and operating characteristics to the localized characteristics of the ore body and plant site, the actual capital costs and operating costs of apparently similar mines should not unduly influence the estimated capital costs and operating costs of a prospective mining venture. The experienced cost estimator should attempt to visit one or more mines with known similarities to the prospective mine, so that a judgment may be made as to whether the local conditions at the visited mine are more or less favorable than those at the prospective mine. This judgment should be reflected in the estimate of higher or lower operating costs for the proposed operation.

Unlike capital cost estimation, which is based primarily on the size and unique nature of the mine and plant site, estimation of operating costs depends on the assessment of the probability of imperfect coordination of human effort in equipment operation and consumption of supplies.

To standardize calculations, the majority of the equations employed here are stated in English units. Also the emphasis in this chapter is on noncoal mining.

6.3.2 ASSESSMENT OF MINING CONDITIONS AFFECTING COSTS

The capital costs and operating costs of a mining project will be influenced by many factors that must be assessed before costs can be estimated for a preliminary feasibility study. The most important factor affecting costs is the size of the mine and processing plant as expressed in terms of tons of ore mined and milled per day of operation.

6.3.2.1 Mine Size or Production Rate

After discovery of an ore body, the mining and milling rate for the proposed mine project should be chosen such that the operating profit maximizes the return on capital invested in developing the mine and constructing the plant and services. If the mine size is too large in relation to the reasonably assured ore reserves, the operating life of the mine will be too short to yield an adequate return on capital invested in the mine project, and there will be insufficient opportunity to adjust or correct plant defects or operating inefficiencies before the ore reserve tonnage is significantly depleted.

If the mine size is too small in relation to ore reserve tonnage, the operating profit will be too small to recover invested capital and necessary return over the first few years of operation. After the mine is placed in production, the capital cost of enlarging the mine plant will be much higher than the additional cost of an initially enlarged mine plant.

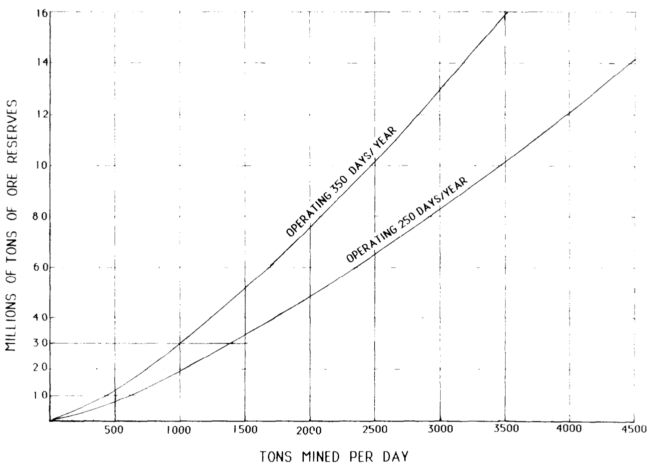

Fig. 6.3.1. Daily tonnage rate vs. ore reserves.

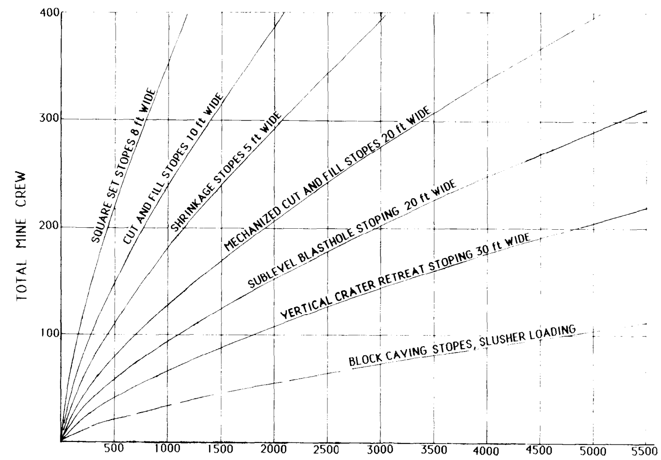

Fig. 6.3.2. Underground mine crew for various stoping methods and ore widths.

A useful guide to the optimum mine tonnage rate is the Taylor (1986) formula (Fig. 6.3.1). This formula, modified for English short ton units, is

$$\text{Optimum mine tonnage rate } T = \frac{4.88\, T_r^{0.75}}{D_{yr}} \qquad (6.3.1)$$

where T is short tons (2000 lb) of ore mined or milled per operating day, T_r is estimated short tons of diluted ore reserves that are judged to be reasonably assured (i.e., proven ore plus probable ore, but excluding possible ore with no assurance of its existence), and D_{yr} is the number of days per year of operation at full capacity. D_{yr} is approximately 250 for a mine operating on a 5-day week and 350 for a mine or mill operating continuously 7 days per week with only minor shutdowns or holidays per year.

6.3.2.2 Personnel Requirements

Operating costs and capital costs are influenced by the number of personnel required to operate the mine, mill, and services at any specific daily tonnage rate, because the number of personnel required varies with the methods to be used for mining and milling and whether or not the mine plant is extensively mechanized or computerized.

Guides for the number of personnel required for mining, milling, and services follow.

For underground mines the following relationships may be used to estimate the number of mine personnel required for mines using various mining methods (Fig. 6.3.2). T is the short tons of ore mined per day, W is the average stope width in feet, and Nmn is the number of persons required by the mine.

$Nmn = 8.0\ T^{0.7}/W^{0.5}$ for unmechanized square set mines (6.3.2)

$= 6.5\ T^{0.7}/W^{0.5}$ for small mines resuing narrow veins (6.3.3)

$= 6.0\ T^{0.7}/W^{0.5}$ for unmechanized cut and fill mines (6.3.4)

$= 2.5\ T^{0.7}/W^{0.3}$ for mechanized cut and fill mines (6.3.5)

$= 3.2\ T^{0.7}/W^{0.5}$ for unmechanized shrinkage mines (6.3.6)

For mining methods unsuitable for narrow stopes, where the slope width W is typically greater than 20 ft (6 m):

$Nmn = 0.75\ T^{0.7}$ for sublevel stoping mines with blastholes of small diameter (1 to 2 in. or 25 to 50 mm) (6.3.7)

$= 0.53\ T^{0.7}$ for vertical crater retreat mines using large diameter (3 to 6 in. or 75 to 150 mm) blastholes (6.3.8)

$= 0.72\ T^{0.7}$ for room and pillar mines in flat dipping hard rock (6.3.9)

$= 0.38\ T^{0.7}$ for room and pillar mines in flat-bedded, soft rock (increase up to 70% if mine is water-bearing) (6.3.10)

$= 0.35\ T^{0.7}$ for block caving mines using load-haul-dump equipment for loading ore (6.3.11)

$= 0.27\ T^{0.7}$ for block caving mines using slusher loading (6.3.12)

$= 0.42\ T^{0.7}$ for continuous mining in flat-seam mines (6.3.13)

The number of mine personnel required in open pit mines may be estimated from the following formulas in which Nop is number of open pit personnel and T_p is tons of ore and waste mined daily.

$Nop = 0.034\ T_p^{0.8}$ for open pit mines in hard rock using shovels and trucks for loading and haulage of ore and waste (6.3.14)

$= 0.024\ T_p^{0.8}$ for open pit mines in competent soft rock (6.3.15)

The number of mill personnel Nml required to operate mills treating T tons of high-grade underground ore daily may be estimated from the following formulas:

$Nml = 0.78\ T^{0.6}$ for cyanidation of precious metal ores (6.3.16)

$= 0.57\ T^{0.6}$ for differential flotation of base metal ores (6.3.17)

$= 0.95\ T^{0.6}$ for leaching, solvent extraction, and precipitation of uranium ores (6.3.18)

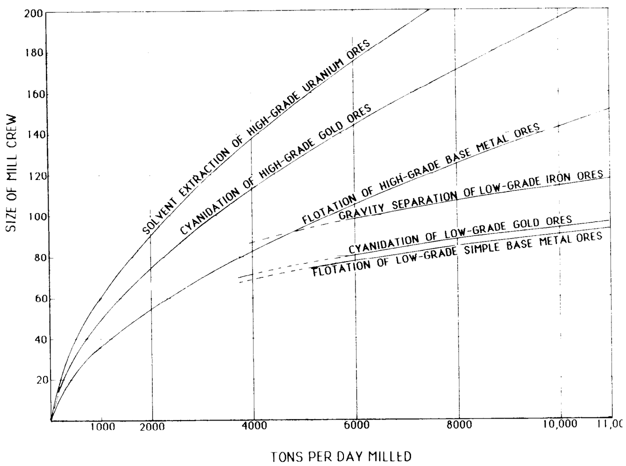

Fig. 6.3.3. Mill crew vs. mill process and size.

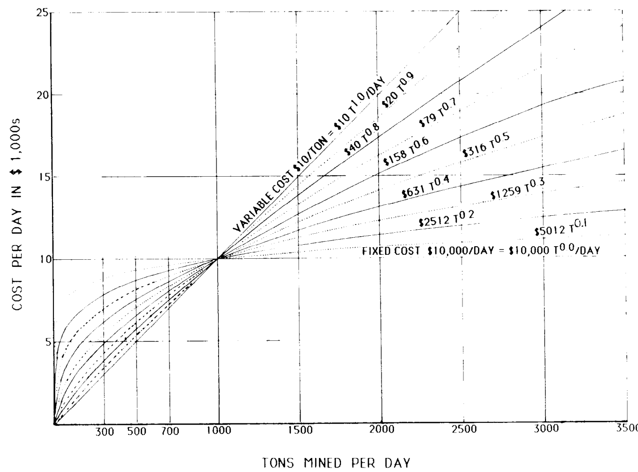

Fig. 6.3.4. Patterns of cost variability.

The number of mill personnel *Nml* required to operate mills treating *T* tons of low-grade ore mined by open pit methods may be estimated from the following formulas:

$Nml = 5.90\ T^{0.3}$ for cyanidation of precious metal ores (6.3.19)
$= 5.70\ T^{0.3}$ for flotation of low-grade base metal ores (6.3.20)
$= 7.20\ T^{0.3}$ for gravity concentration of iron ores (6.3.21)

See Fig. 6.3.3 for mill crew vs. mill process and size.

The number of service personnel *Nsv* may be estimated as a percentage of the total mine and mill personnel as shown below:

Nsv = 37.5% of ($Nmn + Nml$) for medium and large underground mines that are mechanized in drilling and ore transport (6.3.22)
= 20.6% of ($Nmn + Nml$) for small and medium underground mines with manual drilling and little mechanization (6.3.23)
= 25.4% of ($Nop + Nml$) for open pits mining low-grade ore (6.3.24)

The number of administrative and technical personnel *Nat* required for a mining and milling plant may be estimated as a percentage of the total required for mining, milling, and services:

Nat = 12% of ($Nmn + Nml + Nsv$) for underground mines and mills (6.3.25)
= 11% of ($Nop + Nml + Nsv$) for open pit mines and mills (6.3.26)

It should be noted that the formulas for personnel required for mining, milling, services, and administrative and technical activities do not allow for personnel required for smelters, refineries, mine townsite services, concentrate transport, or offsite head offices, because these services may not be required for many mine projects. Whenever these services can be financially justified for the mine project circumstances, the additional personnel should be estimated separately.

Patterns of cost variability are shown in Fig. 6.3.4.

6.3.2.3 Electrical Power Demand

The peak load in kilowatts per month and the average daily power consumption in kilowatt hours can be estimated from the following formulas:

Peak load in kW $= 165\ T^{0.5}$ for underground mines milling T tons of ore daily (6.3.27)
$= 78\ T^{0.6}$ for open pit mines milling T tons of ore daily (6.3.28)

Power consumed in kWh/day $= 1800\ T^{0.57}$ for underground mines hoisting ore to a surface concentrator (6.3.29)
$= 1400\ T^{0.6}$ for open pit mines with shovel and truck haulage to concentrator (6.3.30)

Typically, the concentrator and related facilities account for about 85% of the total power consumption for open pit mines and concentrators, whereas the concentrator and related facilities account for only about 45% of the total power consumption for underground mines hoisting ore to a surface concentrator.

6.3.2.4 Mine Site Clearing

Area A to be cleared in acres $= 0.0173\ T_p^{0.9}$ for open pits mining T_p tons of ore and waste daily (6.3.31)

The area to be cleared extends to the ultimate pit limits.

Area A to be cleared in acres $= 0.011\ T^{0.7}$ for underground mines hoisting ore up a central shaft around which service buildings are sited. This area to be cleared does not include the mill and accessory buildings (6.3.32)

Area A to be cleared in acres $= 0.05\ T^{0.5}$ for concentrator building, crusher building, substation, warehouse, and ancillary buildings (6.3.33)

6.3.2.5 Assessment of Underground Mines

Costs of a proposed mining project are difficult to estimate unless the specific underground conditions are numerically assessible. These should be assessed by a person familiar with the site topography, ground conditions, and structural geology of the ore body.

Underground Mine Drainage System: This system comprises the underground sumps, multistage pumps, controls, standby pumps, and piping for pumping drainage water from the mine. The cost of this system is a function of the total installed horsepower of the operating pumps (but excluding the standby pumps), which in turn is a function of the total of the gallons per minute (liters per second) multiplied by the pumping head in feet (meters) for each of the installed pumping stations.

The rate of pumping in gallons per minute (US) for each pump is several times the inflow at each station sump, and the pumping head will typically be between 400 and 1500 ft (120 and 450 m). It is difficult to estimate the probable inflow of water in an ore body that has been drilled but not developed by underground crosscuts, drifts, and raises. However, some indication of whether the water inflow will be slight or heavy can be attained by examination of the drill core data to determine the presence of faulted water-bearing zones and drilling records showing loss of drilling water.

Total pump system horsepower Hp = total of (gpm × Hd)/2350 (for all pump stations) (6.3.34)

This formula assumes metric units are converted to US gallons per minute and to feet of pumping head in numerically determining the horsepower required at each pumping station. In general, when the pumping system has not been planned in detail, the installed pump horsepower can be approximately estimated from the following formulas:

Installed horsepower Hp $= 8.0\ T^{0.5}$ for dry mines with little inflow and mine depth less than 1000 ft (300 m) (6.3.35)

$= 26\ T^{0.5}$ to $32\ T^{0.5}$ for mines with medium inflow and 1500 to 3000 ft (460 to 900 m) depth. (6.3.36)

$= 62\ T^{0.5}$ for mines with heavy inflow (6.3.37)

Underground Ventilation System: The cost of installing and operating a ventilation system varies with the total installed horsepower of all mine fans in the system. The total installed horsepower varies with the total quantity of air in cubic feet per minute, multiplied by the average fan pressure in inches water required to move this quantity of air. In general, larger mines require larger quantities of air than smaller mines, but the fan pressure required to move larger quantities of air increases in relation to the square of the velocity. The larger mines usually have larger development openings and larger bulk mining stopes, thus the average fan pressure in large mines is generally not much larger than in small mines ventilated by smaller quantities of air.

Quantity of air required Q in cfm:

$Q = 1400\ T^{0.8}$ for underground gold and metal mines (6.3.38)

$= 1900\ T^{0.8}$ for underground uranium mines (6.3.39)

$= 500\ T^{0.8}$ for underground nonmetallic mines without dust of a siliceous nature (6.3.40)

Typical fan pressure $= 2.4\ T^{0.1}$ in. water (6.3.41)

Total installed fan Hp = Q cfm × in. water/3800 (6.3.42)

$= 0.88\ T^{0.9}$ approximately (6.3.43)

The typical fan pressure and installed horsepower may vary widely if mine openings are small in relation to tonnage of ore mined daily.

Compressed Air Plant:

Capacity of plant C in cfm $= 170\ T^{0.5}$ for underground mines using small-hole drilling and stoping by shrinkage or cut and fill methods in medium-width stopes (6.3.44)

$= 230\ T^{0.5}$ for small underground mines using small-hole drilling, air-powered slushers, and loaders in narrow stopes (6.3.45)

$= 130\ T^{0.5}$ for large underground mines using large blastholes in wide stopes with diesel-powered mechanized equipment for loading (6.3.46)

Hoisting Equipment: Two types of hoists are used in hoisting ore in underground mines: double-drum hoists and friction hoists. Double-drum hoists are suitable for hoisting ore or transporting men and supplies from several different levels for all sizes of mines. Friction hoists are suitable for deep mines hoisting ore from the lowest level and generally consume less power than double-drum mines hoisting the same tonnage from deep levels. Despite the operating economies of friction hoists, double-drum hoists are more often used because they are applicable over a wider range of operating conditions, and also because of the availability of used double-drum hoists that can be reserviced for operations at less cost and sooner than purchasing a complete new hoist.

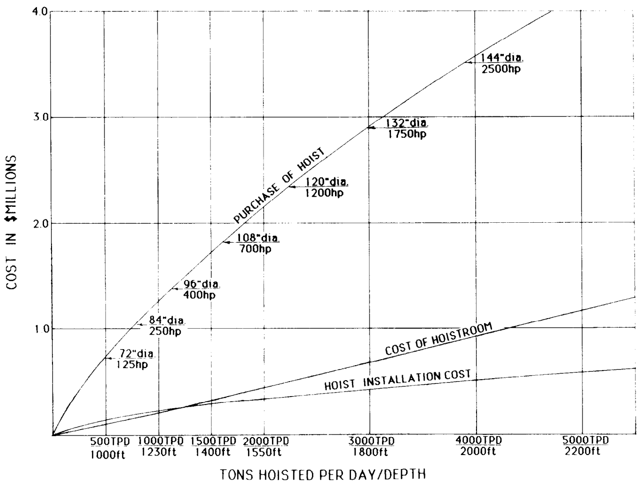

Fig. 6.3.5. Costs of hoisting plant.

The cost of a double-drum hoisting plant (Fig. 6.3.5) depends on the hoist drum diameter D in inches and on the horsepower of the hoist motor Hp, which in turn will be dependent on the loaded skip weight and rope speed. Usually small mines hoisting less than 1500 tons of ore in two 8-hr shifts/day install one hoist with combined skips and cage to hoist ore and transport miners and supplies. Medium-sized mines hoisting 2000 to 4000 tpd generally install one large hoist for hoisting ore only and a second smaller hoist for transporting miners and supplies in a multi-compartment shaft. Larger mines hoisting more than 5000 tpd will generally have two separate hoisting plants.

The optimum rope speed S in feet per minute (fpm) for hoisting ore is about:

$$S \text{ in fpm} = 1.6\, h^{0.5}\, T^{0.4} \text{ for hoisting } T \text{ tpd from a depth of } h \text{ ft} \tag{6.3.47}$$

The rope speed for transporting miners and supplies is generally about 30% slower than the optimum rope speed for hoisting ore.

The hoist drum diameter D in inches should be about

$$D = 4.13\, T^{0.3}\, h^{0.14} \tag{6.3.48}$$

The total installed horsepower of the hoist motor, or motors, should be:

$$\text{Horsepower Hp} = 0.5\,(D/100)^{2.4}\, S \text{ where } D \text{ is drum diameter in in. and } S \text{ is the rope speed in fpm} \tag{6.3.49}$$

Transport of miners and supplies is normally at slower rope speeds by a hoist with a drum diameter at least 80 times the rope diameter, and the hoisting rope diameter must be such that there is an adequate safety factor when hoisting the cage with the maximum load of miners and equipment.

The area A of the hoist room required for double-drum hoists with drum diameters of D inches will need to be about:

$$A(\text{in ft}^2) = 0.10\, D^{2.2} \text{for one double-drum hoist} \tag{6.3.50}$$

$$= 0.085\,(D_1^{2.2} + D_2^{2.2}) \text{ for two double-drum hoists with drum diameters } D_1 \text{ and } D_2 \tag{6.3.51}$$

Headframe Size: The height of the headframe above the shaft collar must be sufficient to allow the skips to dump into ore bins that must have storage capacity of ore adequate for the daily tonnage rate, plus a safe vertical distance for skip overtravel and hoist braking distance below the sheave center elevation.

The headframe height H in feet of the sheave center above the shaft collar is shown by:

$$\text{Headframe height } H = 8.0\, T^{0.3} + 1.2\, S^{0.5} \tag{6.3.52}$$

in which T is the tons of ore mined daily, which is hoisted to surface ore bins 16 hr/day, 5 days/week, and S is the rope speed in fpm. The 8.0 $T^{0.3}$ factor represents the allowance for skip dump height and the 1.2 $S^{0.5}$ factor represents the allowance for skip overtravel.

The weight of structural steel W in pounds in a steel headframe with a sheave center height of H ft is approximately:

$$W = 0.12\, H^3\, (D/100)^2 \text{ for a headframe safely designed for the breaking strength of the hoist ropes of a diameter not less than 1/80th of the drum diameter of the hoist} \tag{6.3.53}$$

If the headframe also serves another smaller hoist employed as a cage hoist, the weight of structural steel should be increased by about 20%.

Shaft Area: Because of the trend to more extensive mechanization of mines over the last two decades, the cross-sectional area of shafts sunk during the 1980s is somewhat larger in order to deliver and service larger loading and drilling equipment to the underground workings.

The shaft area A in square feet of rectangular shafts hoisting T tons of ore daily is now:

$$A = 24\, T^{0.3} \text{ for underground mines hoisting up to 5000 tpd in two skip compartments and cage hoisting miners and supplies in cage compartments} \tag{6.3.54}$$

The shaft diameter (D in feet) of circular shafts for skip hoisting ore and cage hoisting miners and supplies is:

$$D = 5.5\, T^{0.15} \text{ for circular shafts of underground mines hoisting less than 5000 tpd and transporting miners, supplies and equipment in separate cage compartments} \tag{6.3.55}$$

See Fig. 6.3.6 for shaft sinking costs vs. shaft depth.

6.3.2.6 Mine Development Required for Underground Mines

Mine development consists of two items: (1) development of drifts, crosscuts, ramps, raises, orepasses, ventilation raises, substations, and sumps to provide access to and services for the mining of sufficient ore for the first few years of mine production; and (2) stope preparation of sufficient stopes to permit subsequent mining of ore for six months, during which time current stope preparation will have prepared sufficient ore for a further six months of mining.

The costs of development (Fig. 6.3.7) for ore access and mine services (Fig. 6.3.8) together with the cost of initial stope preparation are considered to be preproduction capital costs, and mine development costs are typically the largest component of

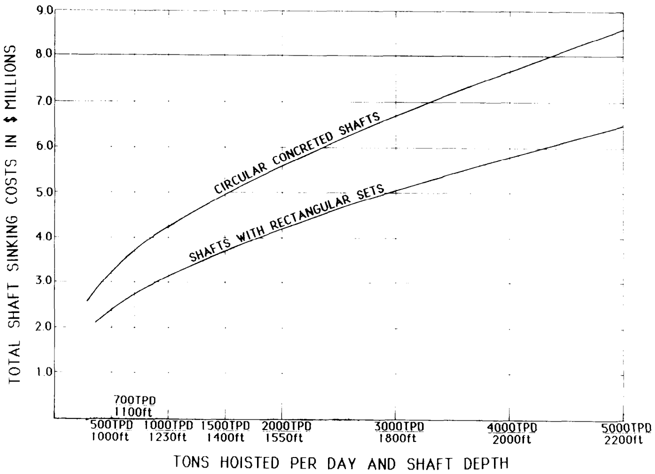

Fig. 6.3.6. Shaft sinking costs vs. shaft depth.

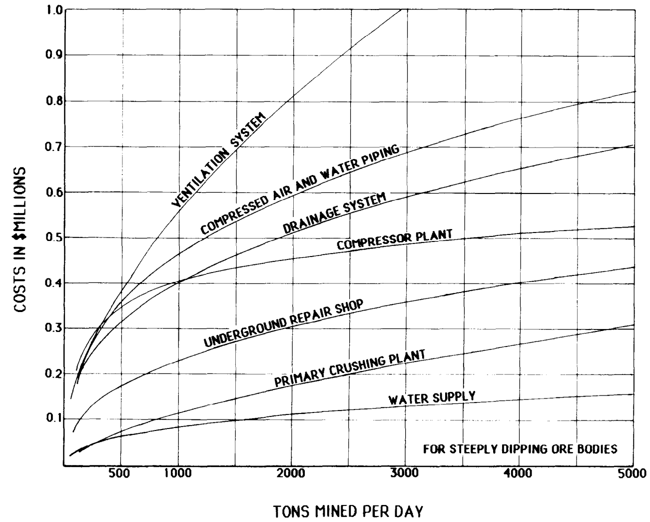

Fig. 6.3.8. Costs of mine services.

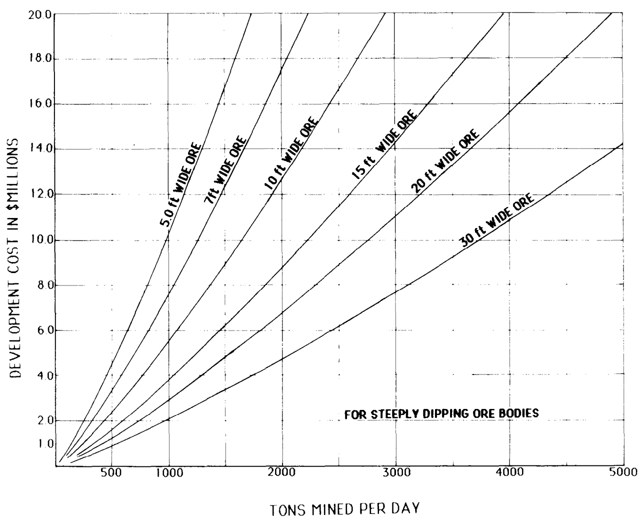

Fig. 6.3.7. Cost of mine development.

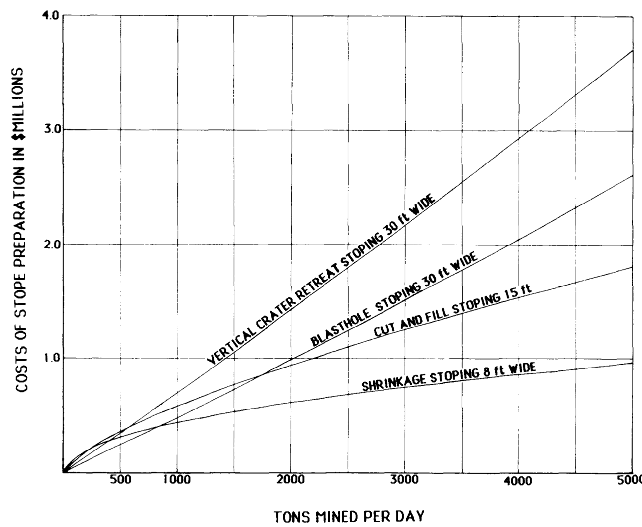

Fig. 6.3.9. Costs of preparing stopes for mining 125 days of ore.

mining capital cost and requires the longest time period of any mine activity in preparing the mine for production.

Mine development is often the most difficult cost to estimate of all preproduction capital costs prior to detailed mine planning, because of the uncertainty of many optional development plans that may or may not be suitable for subsequent efficient mine production.

In general, the mine development for a new mine should access ore equal to 1800 days of production and prepare stope ore equal to 125 days of production (Fig. 6.3.9). Less development leads to problems in maintaining steady ore production, and more development may be excessively costly.

The amount of mine development varies in relation to the volume of ore reserves, rather than the tonnage of ore reserves, so that an ore with a high specific gravity requires less development in waste to access this ore than an ore with a lower specific gravity.

One ton of ore with a typical specific gravity of 2.7 has an in situ volume of 11.866 ft^3; therefore, the ore reserve tonnage of T_r tons has a volume of:

$$\text{Ore reserve volume in ft}^3 = 32.04\ T_r\ /\ \text{SG} \qquad (6.3.56)$$

where SG is the specific gravity of the ore.

The estimator should estimate the length and height (in feet) of the rectangular shape, which fully encloses 1800 days of ore tonnage in the longitudinal section (or the length and width of 1800 days of ore tonnage in a flat-lying ore body). This length and height (or length and width) determines the amount of mine development necessary. The normal location of the ore hoisting shaft is generally near the center of gravity of the rectangular shape but offset from the ore zone by 200 ft or more. Mines in mountainous areas may be accessed by adits that extend well beyond the ore zone, and for such mines that do not require a hoisting plant, the mine development required is determined by the number and lengths of these adits.

When mine planning has not been done, an approximate measure of the amount of mine development required may be estimated from the length and height (or length and width) of the shape enclosing 1800 days of ore reserve tonnage, but the cost of the mine development will also depend on the sizes of the drifts, crosscuts, ramps, orepasses, raises, and various excavations. In general, the cost per foot of any excavation of sectional area A in square feet is proportional to the 0.6 exponent of the area A. Thus one can relate the cost per foot of length of any sized excavation of cross-sectional area A to the cost per foot of a standard 8 × 8-ft drift by the following ratios:

Drifts or crosscuts of cross-sectional area A	$0.0825\ A^{0.6}$
Inclined ramps of cross-sectional area A	$0.0970\ A^{0.6}$
Timbered raise @ 60° of cross-sectional area A	$0.1122\ A^{0.6}$
Bare raise @ 50° of cross-sectional area A	$0.0923\ A^{0.6}$
Service excavations of cross-sectional area A	$0.0948\ A^{0.6}$

By using these cost ratios, it is possible to convert all mine development openings into equivalent feet of 8 × 8-ft drift that would cost the same.

In general, the larger-tonnage mines require larger drifts, ramps, crosscuts, ventilation raises, orepasses, and service excavations than smaller mines, because the larger mines use larger equipment and move larger quantities of air and larger tonnages of ore through the mine openings.

The amount of mine development expressed in equivalent length of 8 × 8-ft drift is difficult to determine by any universal formula because of the variety of shapes and sizes of mine ore bodies. The amount of development required to access and service ore equivalent to 1800 times the daily tonnage has been estimated for hypothetical steeply dipping ore bodies. These hypothetical ore bodies are assumed to dip at 80°, to have a specific gravity of 2.7, to have a horizontal extent that is twice the vertical extent, and to have waste distributed over 30% of the overall extent of ore. The average width of ore in each ore body is assumed to increase by 63% for each doubling of daily mining rate. The results of this hypothetical study are tabulated below:

Mining rate in tpd	500	1000	2000	4000	8000
Ore width in ft	5.0	8.1	13.2	21.4	34.8
Height of ore in ft	1230	1491	1654	1835	2036
Length of ore in ft	2460	2982	3308	3670	4072

Amount of development in equivalent length of 8 × 8-ft drift:

Shaft stations, etc.	2290	3639	5299	7720	11,255
Levels and ramps	21,068	35,552	50,278	71,103	100,550
Orepasses and pockets	1149	1941	3013	4679	7269
Vent raises, manways	3069	5297	8902	15,347	26,954
Service excavations	422	724	1263	2248	4085
Total ft of 8 × 8-ft	27,998	47,153	68,755	101,097	150,111

The results of this study indicate that the following formulas should apply to development in similar shaped ore bodies, which have an average ore width of W ft and a tonnage rate of T tpd:

$$\text{Equivalent ft of } 8 \times 8\text{-ft drift} = 74.29\ T^{1.2}/W^{0.9} \tag{6.3.57}$$

If the specific gravity of the ore differs from 2.7, this formula converts to:

$$\text{Equivalent ft of } 8 \times 8\text{-ft drift} = 244.66\ (T/\mathrm{SG})^{1.2}/W^{0.9} \tag{6.3.58}$$

These formulas may be used to compute the approximate amount of mine development for steeply dipping ore bodies of similar shape to that shown, but if the ore body differs drastically in shape, attitude, and width, the formulas become unreliable.

6.3.2.7 Stope Preparation for 125 Days of Ore at Daily Tonnage Rate

The amount of stope preparation for any given vertical area of ore depends on the stoping method, the level interval, and the degree of mechanization intended for stoping practice. Since the tonnage contained in any vertical area of ore depends directly on the width of the ore, the amount of preparation required for an ore tonnage equivalent to 125 days of mine production normally varies inversely with the ore width.

Although larger mines generally mine ore from wide ore bodies, whereas small mines typically mine ore from narrow ore bodies, some small mines may have portions of their ore bodies that are locally quite wide. In such instances, the small mine may develop the locally wide portion of the ore body with larger development openings to utilize larger mobile mucking and drilling equipment.

The formulas shown in the following offer guides to the amount of stope preparation required for mines with differing stoping methods, different daily tonnage rates, and differing ore widths.

Amount of stope development for 125 days of stoping, expressed as equivalent length of 8 × 8-ft drift:

Feet of 8 × 8-ft drift = $72\ T^{0.48}\ W^{0.2}$ for shrinkage stoping mines in steeply dipping ore bodies with ore less than 15 ft wide in which T is the number of tons mined daily and W is ore width in feet (6.3.59)

Feet of 8 × 8-ft drift = $8.15 T^{0.7}\ W^{0.5}$ for cut and fill stoping mines with ore less than 15 ft in width (6.3.60)

Feet of 8 × 8-ft drift = $16.25\ T^{1.06}/W^{0.6}$ for blasthole stoping mines that have ore widths over 15 ft (6.3.61)

Feet of 8 × 8-ft drift = $24.5\ T^{1.04}/W^{0.6}$ for vertical crater retreat mines that have ore widths over 20 ft (6.3.62)

Feet of 8 × 8-ft drift = $185\ T^{0.6}\ H^{0.2}$ for block caving mines stoping flat-lying ore at least 40 ft thick. H is the height of ore stoped (6.3.63)

It may be noted that stoping methods, such as shrinkage and cut and fill that require the least amount of stope preparation per stope, are usually higher in operating costs per ton than the bulk mining methods such as blasthole mining, VCR (vertical crater retreat) stoping, or block caving.

6.3.2.8 Assessment of Open Pit Mines

Open pit mines may have a greater diversity of ore body shape and daily tonnage of ore than underground mines. However, since waste overlying the ore is removed prior to open pit ore mining, the method of mining the ore is not influenced by the need to support waste from diluting the ore. The capital and operating costs of open pit mines is influenced by the number and sizes of equipment for drilling, blasting, loading, and haulage of open pit ore and waste.

The typical open pit mine in North America produces about 43,000 tpd (39 kt/day) of ore and waste from a pit depth of about 400 to 500 ft (120 to 150 m), with an oval shaped periphery 2200 ft (670 m) wide and 4700 ft (1430 m) long. Pit benches are typically 40 ft (12 m) high, and the overall pit slope (excluding roads) is about 57° in pits with competent rock, and 44° in pits with oxidized or altered rock, with in-pit haulage road gradients averaging 9%.

Pits may vary greatly in shape, size, and pit slope, especially in mountainous areas or where the ore and/or waste rock varies greatly in competence. The formulas shown in the following for equipment sizing, preproduction stripping, and maintenance facilities presume that the shape and type of open pit is similar, except in daily tonnage, to the typical open pit described previously.

Size and Number of Open Pit Drills: The size, hole diameter, and number of drills required depends on the tons of ore and waste to be drilled off daily. In general, there should be not less than two drills, and not more than four, for open pit mines mining less than 60,000 tpd (54.4 kt) of ore and waste. In medium-drillable rock, with a penetration rate of about 500 ft (152 m) per shift, the tons of ore or waste that is drilled off by a drill with a hole diameter of d inches is:

$$\text{Tons of ore or waste } T_p \text{ per day} = 170\ d^2 \tag{6.3.64}$$

$$\text{For easily drillable rock, } T_p \text{ per day} = 230\ d^2 \tag{6.3.65}$$

$$\text{For hard drilling rock, } T_p \text{ per day} = 100\ d^2 \tag{6.3.66}$$

Typically, drill hole sizes have standard diameters (in inches) of 4, 6.5, 7.875, 9.875, 10.625, 12.25, 13.25, and 15 (or 102, 165, 200, 250, 270, 310, 336, and 380 mm); thus drill selection will be limited to one of these sizes. For tonnages up to 25,000 tpd (22.7 kt/day), two drills of appropriate hole diameter should be chosen, three drills should be adequate for up to 60,000 tpd (54.4 kt/day), and four or more drills will be required for daily tonnages over 60,000.

Size and Number of Shovels Required: The optimum shovel size S in cubic yards of dipper size in relation to daily tonnage of ore and waste (T_p) to be loaded daily is

$$S = 0.145\ T_p^{0.4} \tag{6.3.67}$$

The number of shovels N_s with dipper size S that will be required to load a total of T_p tons of ore and waste daily will be

$$N_s = 0.011\ (T_p)^{0.8}/S \tag{6.3.68}$$

In practice, the size of shovel chosen will be one with a standard dipper size close to the size calculated by Eq. 6.3.67, but the number of shovels N_s required is usually a fractional number that should be rounded off to the nearest smaller unit number. The omitted fractional number expresses the need for either a smaller-sized shovel or a front-end loader for supplemental loading service, as long as this smaller shovel or front end loader is adequate to load into trucks of a size appropriate to the shovels with dipper size S

Size and Number of Trucks Required: The optimum truck size t in tons that is well matched with shovels of S bucket size in cubic yards is

$$\text{Truck size } t \text{ in tons} = 9.0\ S^{1.1} \tag{6.3.69}$$

The total number of trucks N_t oft tons capacity required for the open pit truck fleet, plus an allowance for trucks under repair, should be approximated by the following formula:

$$\text{Number of trucks required } N_t = 0.25\ T_p^{0.8}/t \tag{6.3.70}$$

The formula for N_t determines the size of the truck fleet under the typical conditions where the average haulage distance and gradient outside the pit periphery is less than the haulage distance and gradient inside the pit periphery. If the waste dump and the ore dump over the primary crusher are well removed from the pit boundaries, or if the haulage road beyond the pit has a steep gradient, it may be necessary to increase the truck fleet size to allow for the longer trip time per load.

Amount of Preproduction Waste Stripping: Before open pit mining can begin, the soil and rock overburden above the ore must be stripped to expose sufficient ore to supply the planned daily tonnage of ore for four to six months. The soil overburden should be stripped to the peripheral limits of the ultimate pit and an estimate made of its amount. An acre of moist soil that averages 10 ft (3 m) in thickness contains about 23,000 tons (20.9 kt) of soil; thus if the area of the ultimate pit periphery is known and the average soil thickness can be found from drilling logs or ultrasonic techniques, the amount of soil overburden can be calculated.

It is assumed that the location and area of the uppermost ore, sufficient for four to six months of mining, can be determined from ore body mapping, and the average thickness and area of the waste rock overlying this ore can be computed. Each acre of waste rock that averages 10 ft (3 m) in thickness contains about 40,000 tons (36 kt) of waste.

Because of the inverted conical shape of the ultimate open pit, the waste/ore tonnage ratio at each horizontal bench decreases with each lower bench, but the uppermost ore bench to be exposed typically has a waste/ore ratio of at least twice the waste/ore ratio of the ultimate pit. If insufficient ore is exposed by the preproduction stripping of waste, it may become difficult to continue mining ore because of the proximity of waste benches where blasting, loading, and haulage of waste may interfere with ore mining.

Open Pit Maintenance Facilities: The size of maintenance facilities for repair and maintenance of open pit equipment depends primarily on the number and size of the main haulage trucks, which in turn depends on the daily tonnage of ore and waste to be hauled. Repair and maintenance of the shovels and drills, which are slow in moving, is normally performed on the site by mobile repair vehicles.

The area in square feet required by the open pit maintenance shop, which should be located close to the open pit, is as follows:

$$\text{Area of open pit repair shop} = 360\ T_p^{0.4} \tag{6.3.71}$$

Thus the areas of repair shops required for open pit mines are:

Mine size, tpd	10,000	20,000	40,000	80,000
Repair shop area, ft^2	14,300	18,900	25,000	33,000

6.3.2.9 Assessment of Miscellaneous Characteristics of Mine Projects

In addition to the numerical assessments of various mining features there are many other characteristics that cannot be easily assessed numerically, but which affect capital costs and operating costs. These characteristics can be described or tested so that a judgment can be made as to their effect on costs.

Climate at Mine Location: Weather station records at localized towns, airports, and specific sites should be collected and analyzed in terms of topographic similarity, exposure to prevailing winds, differences in elevation, which tend to decrease temperatures by 5 °F for each 1000-ft (9.1 °C for each 1000 m) increase in elevation, and differences in growth of plants and trees.

Climatic factors may affect building design, seasonal restrictions on pouring of concrete, leaching of metals from ore, size and cost of heating plants, etc.

Access to Mine Plant: The estimator should determine the distance and regional topography between the mine site and the nearest terminal of roads, power lines, and railroads, plus the nearest town where employees could find housing within acceptable commuting distance to the mine plant.

Construction Materials: The estimator should investigate the nearest site where adequate quantity and quality of gravel, water, and tailings storage can be obtained or stored. The sensitivity to pollution of the environment must be assessed and a study of the environmental impact of the mine plant by a team of experts may be warranted at an early stage.

Metallurgical Testing: Testing of drill core samples suggests the preferable ore processing method and the probable recovery of valuable minerals by ore processing. In general, these tests will indicate, but not positively assure, metallurgical performance. However, bulk samples of ore for comprehensive testing and/or pilot plant tests are preferable. The cost of extensive metallurgical testing or pilot plant runs of bulk samples may not be justified before the preliminary feasibility study.

Geologic Assessment: The geologist should examine the core samples, core recovery, and the records of core drilling to assess faulting or areas of incompetent wall rock. This could provide the necessary information on the probable selection of mining method, need for rock bolting of drifts and crosscuts, preferred areas for shaft location or drainage water sumps.

Overburden Removal: Assessment of overburden depth from drilling records and the general competence of subsurface rock strata offers guides to the amount of excavation and foundations for the surface plant.

Location: If the location of the mine is so remote that it is not feasible for employees to commute to work from an existing townsite with adequate available housing and service facilities, it may be necessary to establish a mine townsite or trailer camp adjacent to the mine. The site of such a mine townsite or trailer camp must be chosen with great care and concern for the quality of life of the subsequent residents.

6.3.2.10 General Characteristics of Capital Costs and Operating Costs

Cost formulas are offered as estimators of capital costs and operating costs where the numerical value of the main factor or factors affecting these costs is incorporated into an algebraic equation of the form

$$\text{Cost} = KQ^x, \text{ or Cost} = KQ^x\,T^Y$$

where K is a constant, Q and T represent the numerical value of the factor or factors having the greatest influence on the costs, and x and y are exponents (normally between zero and 1.0) that measure the rate at which changes in the value of Q or T result in changes in costs.

Conventional accounting practices tend to regard costs as being either *fixed* (i.e., costs that remain relatively constant regardless of the size or complexity of the mining plant) or *variable* (i.e., costs that vary in direct relationship to some quantity Q that reflects the size of the mining plant). Alternatively, some costs may be considered to be a mixture of fixed costs and variable costs.

In mining practice, however, no costs are truly fixed or truly variable, neither are they a mixture of fixed and variable costs. It is more accurate to regard all mining costs as being somewhere between slightly variable (cost $= KQ^{0.1}$) and strongly variable (cost $= KQ^{0.9}$).

Virtually all items of purchased capital cost vary with plant capacity C at an exponent of about 0.6 or 0.7; consequently one large item of equipment will invariably be less costly to purchase and install than two smaller items of equipment with the same total capacity. Most items of labor cost per day vary in relation to daily tonnage at an exponent of between 0.4 and 0.8, whereas most items of supplies vary with tonnage at an exponent of between 0.6 and 0.9. The net result of this cost behavior is that if two or more mines are operated in the same manner under the same conditions, the lowest operating cost per ton will be attained by the mine with the largest tonnage output per day.

6.3.3 COST GUIDES FOR CAPITAL COSTS OF MINING PROJECTS

The cost formulas for mining projects described in this segment are based on the actual costs of mine projects completed since 1980, which have been escalated by statistical indices to the equivalent costs for the third quarter of 1988.

6.3.3.1 Underground Mine Projects

Site Clearing: The capital cost of clearing the site for the mine headframe, hoistroom, changehouse, and miscellaneous service buildings depends on the area A in acres to be cleared and, to some extent, the density of tree growth and the slope of the area to be cleared. The choice of site for the hoisting plant allows a limited amount of flexibility to optimize the costs of clearing the site, while avoiding adverse rock conditions for sinking the shaft or unstable ground for the headframe and hoist room foundations.

The area to be cleared can be determined by Eq. 6.3.32 or by judgment of the local site conditions:

$$\text{Clearing cost for mine site} = \$2000\,A^{0.9} \qquad (6.3.72)$$

Capital Cost of Shaft Sinking: The cost of shaft sinking depends on the area of the shaft, which can be estimated from Eqs. 6.3.54 for rectangular shafts or 6.3.55 for circular concreted shafts. The costs of sinking a shaft include a fixed cost of erecting a temporary sinking plant and concreting the shaft collar.

The major cost for shafts sunk 1000 ft (300 m) or more is the unit cost of shaft sinking per foot (meter) of shaft. These unit costs tend to increase as the shaft deepens because of the longer hoisting trip time for hoisting shaft muck. The unit costs also include the cost of excavating shaft stations as the shaft deepens:

$$\text{Fixed costs for rectangular shafts} = \$140{,}000\,A^{0.25} \qquad (6.3.73)$$

$$\text{Unit costs per ft} \times \text{shaft depth in ft} = \$139\,A^{0.45}\,D_s^{1.05} \qquad (6.3.74)$$

where A is area in ft^2 of the rectangular shaft sets and D_s is shaft depth in ft.

$$\text{Fixed costs for circular shafts} = \$135{,}000\,d^{0.5} \qquad (6.3.75)$$

$$\text{Unit costs per ft} \times \text{shaft depth} = \$307\,d^{0.7}D_s^{1.05} \qquad (6.3.76)$$

where d is concreted shaft diameter in ft and D_s is shaft depth in ft.

Capital Costs of Hoisting Plant: The cost of the hoisting plant depends on the size and type of hoist, or hoists, hoisting rope speeds, shaft depth, and the tons to be hoisted per day. For mines hoisting less than 1500 tpd in two 8-hr shifts, it is probable that the optimum hoisting system will be a double-drum hoist with combination skip/cages for hoisting ore and transporting miners and supplies. This hoist should have a drum diameter D in inches as determined by Eq. 6.3.48, and a motor horsepower Hp as determined by Eq. 6.3.49, if the rope speed is close to the optimum rope speed S in feet per minute as determined by Eq. 6.3.47.

Cost of hoist: $= \$700\ D^{1.4}\ \mathrm{Hp}^{0.2}$ for new hoist skipping ore plus cage service (6.3.77)

$= \$540\ D^{1.4}\ \mathrm{Hp}^{0.2}$ for reconditioned used hoist shipping ore plus cage service (6.3.78)

$= \$700\ (0.9\ D)^{1.4}\ \mathrm{Hp}^{0.2}$ for new hoist for ore skipping only. (Drum diameter needs to be only 90% of D as determined by Eq. 6.3.48.) (6.3.79)

$= \$700\ (0.8\ D)^{1.4}\ \mathrm{Hp}^{0.2}$ for new hoist for cage service only. Hoist drum diameter needs to be only 80% of D as determined by Eq. 6.3.48 (6.3.80)

Hoist installation $= \$64\ D^{1.8}$ for installing a hoist with an actual drum diameter of D inches (6.3.81)

Hoistroom construction $= \$4.90\ A^{1.4}$ for a hoistroom with an area of A square feet as determined by Eqs. 6.3.50 and 6.3.51 (6.3.82)

Capital Cost of Headframe: The cost of the headframe depends on the weight of steel required, which in turn depends on the height H in feet and the breaking strength of the hoist rope. Eqs. 6.3.52 and 6.3.53 estimate the headframe height and the weight of steel in the headframe.

Cost of headframe structure $= \$19\ (W)^{0.9}$ for single-hoist headframe structure including shaft collar and foundations (6.3.83)

$= \$19(1.2W)^{0.9}$ for two-hoist headframe structure with shaft collar and foundations (6.3.84)

Note: Add 15% to costs if headframe is sheathed and insulated, with heating plant to ensure moist ore never freezes in skip dumps and ore bins. Cost of enclosed ore bins, skips and skip dumps, cages, and counterweights will tend to vary with daily tonnage, but costs are also affected when the mill operates for more days per week than the mine, thus requiring more ore storage for weekend milling.

Cost of ore bins, skips, etc. $= \$700\ T^{0.7}$ for mines that operate the same work schedule as the mill (6.3.85)

$= \$1150\ T^{0.7}$ for mines that operate 5 days/week while the mill and crusher operate continuously (6.3.86)

Mine Development and Stope Preparation: The amount of mine development and stope preparation that must be completed in the preproduction phase before the mine can start production of ore is discussed in 6.3.2. This amount of development, expressed in terms of equivalent footage of 8 × 8-ft drift is shown in Eqs. 6.3.57 to 6.3.63. The estimated cost of an 8 × 8-ft drift in 1988 is $148/ft of drift.

The cost of $148/ft for an 8 × 8-ft drift is appropriate for hard-rock drifting requiring little or no rockbolting. In incompetent rock requiring extensive rockbolting or grouting of water inflow, this unit cost should be increased.

The cost of mine development for a steeply dipping ore body with ore having a specific gravity of 2.7, an average ore width of W ft, and an expected production rate of T tons of ore daily would be:

Mine development cost $= \$11{,}000\ T^{1.2}/W^{0.9}$ (6.3.87)

$= \$36{,}200\ (T/\mathrm{SG})^{1.2}/W^{0.9}$ when SG is the specific gravity of ore greater than 2.7 (6.3.88)

The cost of initial stope preparation of ore stopes containing ore equivalent to 125 times the expected daily tonnage of ore to be mined depends on the type of stoping method to be employed as shown in the following:

Cost $= \$10{,}620\ T^{0.48}\ W^{0.2}$ for shrinkage stopes (6.3.89)

$= \$1{,}200\ T^{0.7}\ W^{0.5}$ for cut and fill stopes (6.3.90)

$= \$2{,}400\ T^{1.06}/W^{0.6}$ for blasthole stopes (6.3.91)

$= \$3{,}630\ T^{1.04}/W^{0.6}$ for VCR stopes (6.3.92)

$= \$27{,}400\ T^{0.6}\ H^{0.2}$ for block caving stopes (H is stope thickness in feet) (6.3.93)

Cost of Drilling, Loading, and Haulage Equipment: This cost includes all equipment for drilling, loading, and hauling ore, where such equipment is not fixed in place nor installed on foundations.

The mobile equipment cost of similarly equipped mines varies with tons mined daily, but mines with the same daily tonnage may vary in the cost and degree of mechanization if the mines have differing ore widths.

The width of ore in the stopes determines the feasible use of mobile drills, large capacity loaders, and haulage equipment. To accommodate such equipment, larger stope development openings are required. Mines with narrow ore bodies are restricted in the choice of stoping method; drilling is performed with manually controlled drills, and ore loading or scraping is accomplished with small air-powered equipment.

$$\text{Cost of equipment} = \$24{,}600\ T^{0.8}/W^{0.3} \qquad (6.3.94)$$

Although a highly mechanized mine will attain a higher productivity in tons per manshift than an unmechanized mine, it also will require more extensive ventilation and a larger maintenance crew and facilities.

Cost of Mine Ventilation System: The cost of the ventilation system is influenced by the extent of mechanization in the activities of drilling, loading, and haulage, but this cost will be somewhat lessened by the large development openings necessary to accommodate this equipment.

Ventilation costs are adversely affected when the mine is very deep and increased air temperatures at depth decrease human energy and comfort. The quantity of ventilation air and the costs are also increased if siliceous dust or radiation emanates from the mining of ore.

In general, the most reliable measurement of the cost of the installed ventilation system is the total installed horsepower (Hp) of all ventilation fans in the system. The total installed horsepower can be estimated from Eqs. 6.3.38 to 6.3.43.

Cost of ventilation system = $14,000 $\mathrm{Hp}^{0.6}$ for gold and base metal mines (6.3.95)
= $16,800 $\mathrm{Hp}^{0.6}$ for uranium mines (6.3.96)
= $7,500 $\mathrm{Hp}^{0.6}$ for nonmetallic mines without siliceous dust (6.3.97)

Cost of Mine Pumping System: The cost of the mine drainage system depends on the total installed pump horsepower (Hp) as estimated in Eqs. 6.3.34 to 6.3.37. This cost includes the concreting of dams and pump stations, the installed cost of pumps, the cost of standby pumps, the installation of piping from pump stations to shafts (but shaft piping is included in shaft sinking costs), pump control equipment, and sludge removal equipment.

Pumping system cost = $3,400 $\mathrm{Hp}^{0.7}$ for mines with medium water inflow and 1500 to 3000 ft (460 to 910 m) depth (6.3.98)
= $1,400 $\mathrm{Hp}^{0.7}$ for shallow mines with little water inflow and depth of less than 1000 ft (300 m) (6.3.99)
= $5,800 $\mathrm{Hp}^{0.7}$ for mines with heavy water inflow at depths below 1000 ft (300 m) (6.3.100)

Cost of Water Supply System: The cost of the water supply system depends primarily on the amount of drilling and the type of drills used for mine development. Small tonnage mines typically use jacklegs and stopers for drilling, but the larger tonnage mines typically use larger jumbo drills and large-bore drilling in stopes and development. A typical 500-tpd mine uses about 43,000 gal (162 kL) of water daily, while a typical 8000-tpd mine uses about 230,000 gal (870 kL) per day.

$$\text{Cost of water supply} = \$5{,}300\ T^{0.4} \qquad (6.3.101)$$

where T is tons of ore mined daily.

Cost of Primary Crusher Installed Underground: The primary crusher is usually installed underground, except in small mines where the stoped ore is finely broken with little oversize. Primary crushing of ore prior to hoisting reduces problems with hangups in skip loading pockets, skip dumps, and conveyor transport of mined ore.

Jaw crushers are suitable for primary crushing of stoped ore for underground mines, except very large underground mines over 10,000 tpd (9 kt/day) where the size of rock broken in stopes requires a gyratory crusher. Jaw crushers are available in sizes from 24 × 36 in. (610 × 910 mm) to 48 × 60 in. (1220 × 1520 mm) and require much less head room for installation than gyratory crushers.

The area (A in square inches) of the feed opening for jaw crushers determines the ore capacity in tons per day T that can be crushed in most hard rock mines:

$$= 29\ T^{0.5} \qquad (6.3.102)$$

$$\text{Cost of jaw crusher} = \$24.50\ A^{1.2} \qquad (6.3.103)$$

$$= \$1{,}370\ T^{0.6} \text{ approximately} \qquad (6.3.104)$$

The cost of installing a jaw crusher in an excavated crusher station, including foundations, ore feeder system, and dust collection, is:

$$\text{Installation cost} = \$\ 210\ T^{0.7} \qquad (6.3.105)$$

Cost of Underground Repair Shop: Although small mines producing less than 600 tpd (544 t/day) of ore typically have a surface repair shop to service mining equipment plus small mobile equipment from the mill and services, large mines, which are extensively mechanized with large equipment, usually prefer to locate the mine repair shop underground to avoid delays in hoisting the mine equipment to a surface repair shop.

The cost of equipping and stabilizing an underground repair shop excavation for mines with an ore production of T tons of ore daily is estimated to be:

$$\text{Cost of maintenance shop} = \$14{,}600\ T^{0.4} \qquad (6.3.106)$$

This maintenance shop is usually located adjacent to the hoisting shaft and may also include a facility where rock drills are serviced.

Cost of Mine Compressor Plant: The compressed air capacity C in cubic feet per minute required for underground mines can be estimated from Eqs. 6.3.44, 6.3.45, and 6.3.46. The cost of the compressors and all accessory equipment installed in a compressor house on concrete foundations can then be estimated as follows:

$$\text{Cost of compressor plant} = \$920\ C^{0.7} \qquad (6.3.107)$$

Cost of Compressed Air and Water Distribution: The cost of piping installed to distribute compressed air and water to all working places in the mine depends mainly on the length of lateral development expressed in equivalent length of 8 × 8-ft drift, and partly on the total compressor capacity in cubic feet per minute. The length of lateral development L is usually a function of daily mined tonnage T and stope width W:

$$\text{Length of lateral development } L \text{ in ft} = 1276\ T^{0.6}/W^{0.4} \qquad (6.3.108)$$

$$\text{Cost of pipe installation underground} = \$2.80 L^{0.9}\ C^{0.3} \qquad (6.3.109)$$

Cost of Fill Distribution System (for Cut and Fill Mines): The cost of the hydraulic fill system depends on the length of lateral development L in feet and on the tons per day of ore T mined by the cut and fill method.

$$\text{Cost of fill distribution} = \$1.30\ L^{0.9}\ T^{0.6} \qquad (6.3.110)$$

Cost of Underground Electrical Distribution: The cost of

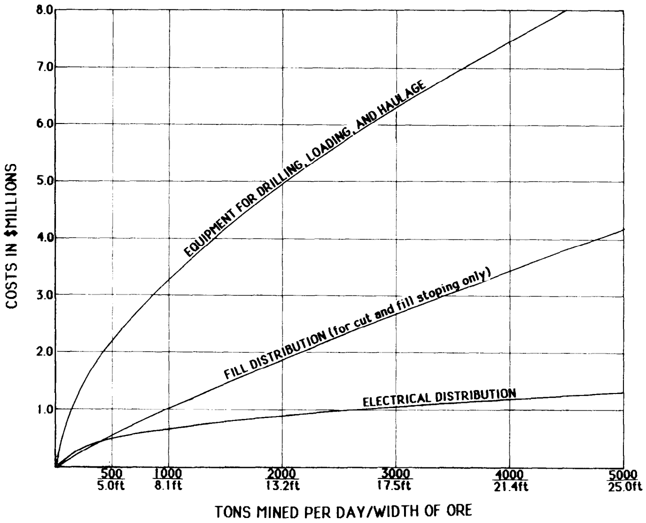

Fig. 6.3.10. Mobile equipment, fill and electrical.

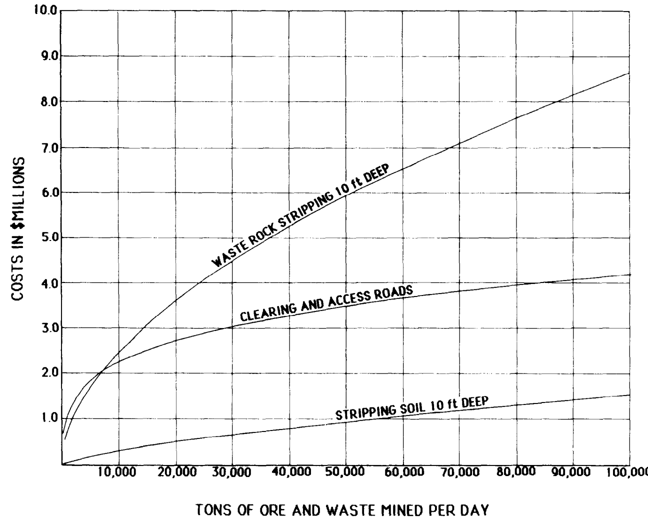

Fig. 6.3.11. Clearing, stripping, and road costs for open pit mines.

substations and power cables underground depends on the average peak load of the mine in kilowatts, as estimated from Eq. 6.3.27 that determines the overall peak load for an underground mine and a surface mill. Since the surface hoist is fed from the surface electrical distribution, the power consumption and peak load of the mine facilities located underground generally will not absorb more than 15% of the total electrical load. Thus if the plant peak load in kilowatts is estimated to be 165 $T^{0.5}$ for a mine and mill producing T tons of ore daily, the portion of this load that is attributable to the underground mining facilities is about 24.75 $T^{0.5}$ in kW

The cost of installing the underground substations and power cables is estimated to be:

$$\text{Cost} = \$1600\,(\text{kW})^{0.9} \tag{6.3.111}$$

where kW is the portion of the peak load attributable to the underground facilities.

See Fig. 6.3.10 for costs of mobile equipment, fill, and electrical distribution.

6.3.3.2 Open Pit Mine Projects

The cost formulas for open pit mines were derived from the actual costs of North American mine projects completed since 1980. The weighted average cost for each item of capital cost has been escalated by statistical indices to be appropriate for the third quarter of 1988.

Clearing Costs for Open Pit Mines: The capital cost for clearing the area where the open pit is located depends on the area A in acres (see Eq. 6.3.31) and the clearing cost per acre.

$$\text{Total clearing cost} = \$1600\,A^{0.9} \text{ for 20\% slopes with light tree growth} \tag{6.3.112}$$

$$= \$300\,A^{0.9} \text{ for flat land with shrubs and no trees} \tag{6.3.113}$$

$$= \$2000\,A^{0.9} \text{ for 30\% slopes with heavy trees} \tag{6.3.114}$$

Clearing, stripping, and road costs for open pit mines are shown in Fig. 6.3.11.

Preproduction Waste Stripping: If T_s is the tons of soil, and T_w is the tons of waste rock that must be stripped to expose an amount of ore to sustain four to six months ore production, then the estimated costs of waste stripping will be

$$\text{Soil stripping costs} = \$3.20\,T_s^{0.8} \text{ for soil not more than 20 ft deep} \tag{6.3.115}$$

$$\text{Waste stripping costs} = \$340\,T_w^{0.6} \text{ for rock requiring blasting, loading, and haulage} \tag{6.3.116}$$

Cost of Open Pit Drilling Equipment: The cost of drilling equipment depends on the number of drills Nd and the hole diameter d in inches drilled in order to prepare the daily tonnage of ore and waste rock for production (see Eqs. 6.3.64, 6.3.65, and 6.3.66).

$$\text{Drilling equipment cost} = Nd \times \$20{,}000\; d^{1.8} \tag{6.3.117}$$

This formula includes a 25% allowance for drilling and blasting supplies and accessory equipment.

Cost of Shovels and Loading Equipment: The cost of loading equipment depends primarily on the number Ns and size S in cubic yards of the shovels, as shown by Eqs. 6.3.67 and 6.3.68. The total cost of the fleet of shovels supplemented by auxiliary bulldozers and front end loaders will be

$$\text{Loading equipment cost} = Ns \times \$510{,}000\; S^{0.8} \tag{6.3.118}$$

Cost of Trucks and Accessory Road Maintenance Equipment: The cost of haulage equipment depends primarily on the number Nt of trucks, and the truck size (t in tons), as shown by Eqs. 6.3.69 and 6.3.70.

$$\text{Haulage equipment cost} = Nt \times \$20{,}400\; t^{0.9} \tag{6.3.119}$$

Open pit equipment costs are shown in Fig. 6.3.12.

Cost of Open Pit Maintenance Facilities: The cost of constructing and equipping the pit maintenance shop varies with

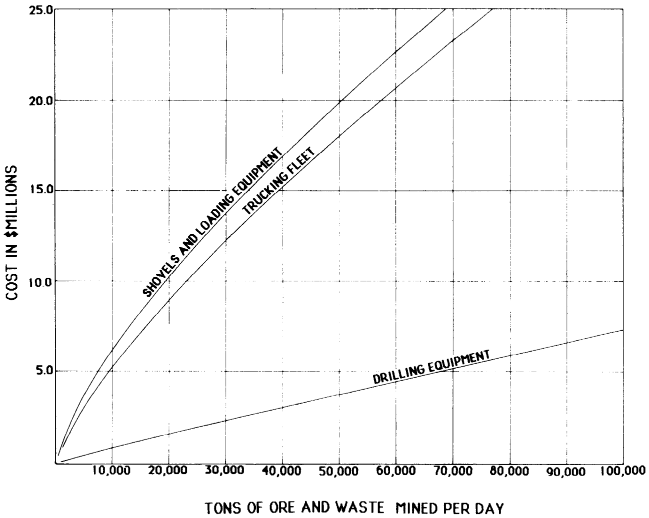

Fig. 6.3.12. Open pit equipment costs.

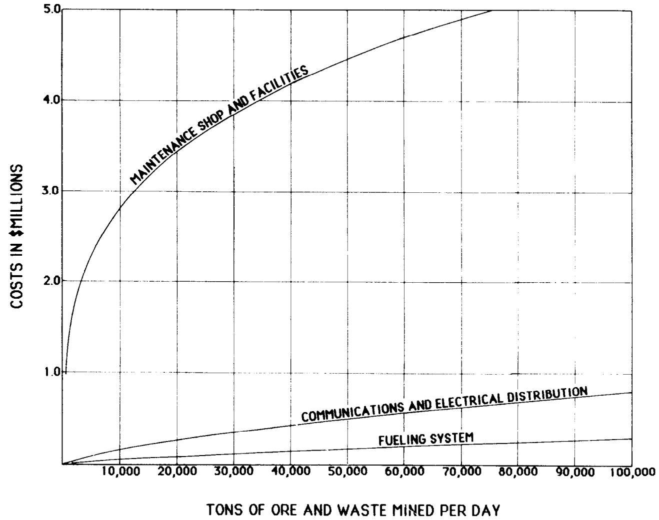

Fig. 6.3.13. Open pit services costs.

the shop area A in square feet as shown by Eq. 6.3.71 and the size t of the trucks in the truck fleet.

$$\text{Cost of pit maintenance facilities} = \$6000\, A^{0.6}\, t^{0.1} \tag{6.3.120}$$

Cost of Open Pit Communications and Electrical Distribution: This cost includes the installed costs for a surface telephone system with mobile and base radio units with one or more repeaters depending on the size of the mine. The electrical distribution includes the installed costs of primary substations, transmission lines, portable skid-mount transformers, and trailing cables, all of which depend on the size of the open pit mine as measured by the daily tons T_p of ore and waste mined.

$$\text{Cost of communications/electrical} = \$250\, T_p^{0.7} \tag{6.3.121}$$

Cost of Open Pit Fueling System: This cost includes the storage and services for diesel fuel, gasoline, lubricants, and coolants for the truck haulage fleet and mobile service vehicles

$$\text{Cost of fueling system} = \$28\, T_p^{0.8} \tag{6.3.122}$$

Open pit services costs are shown in Fig. 6.3.13.

6.3.3.3 Concentrator and Surface Facilities for Mine Projects

It is assumed that the mill operates at three 8-hr shifts for 7 days/week, regardless of the shifts worked by the underground mine or the open pit mine. Some underground mines operate 2 shifts/day, 5 days/week; but other underground mines and most open pit mines operate for 7 days/week. Thus the daily tonnage of ore milled may be the same as the daily tonnage of ore mined, or it may be only 71% of this tonnage.

The crushing plant may operate for 5, 6, or 7 days/week, depending on the mine schedule and whether or not there is adequate fine ore storage capacity to keep the mill supplied with ore when the crusher is shut down for repairs or regular maintenance.

The cost guides in this section are based on the assumption that the mill capacity is 7 1% of the daily tonnage mined 5 days/week, and that the crushing plant has the same daily capacity as the mine, but will work 6 days/week to ensure that the mill will be supplied with crushed ore if the fine ore bins have insufficient capacity to keep the mill supplied with ore during the two-day mine shutdown. In all cases where costs are presented as a function of T tons of ore daily, the number T is the daily capacity in tons of the facility being costed, regardless of whether the facility works 5, 6, or 7 days/week.

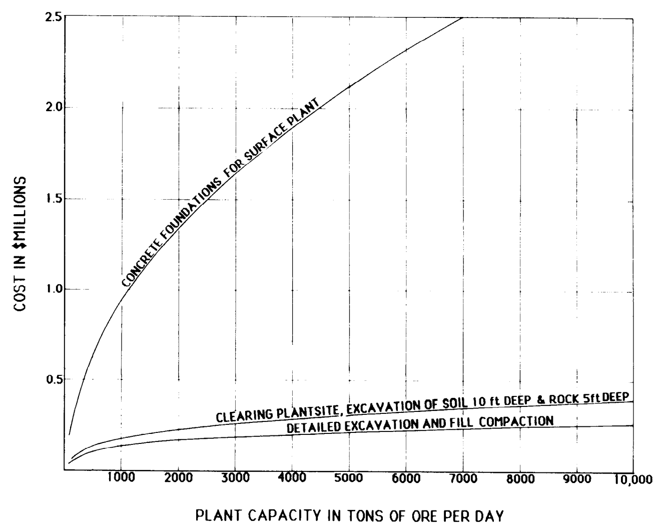

Fig. 6.3.14. Surface plant clearing, excavation, and foundations.

Clearing Costs for Concentrator, Crushing Plant, and Service Facilities: The area A in acres that must be cleared (Fig. 6.3.14) of trees, roots, and shrubs before construction of the concentrator and related facilities can begin is shown in Eq. 6.3.33. In addition to this clearing, roads must be constructed from the nearest existing suitable road to provide access to the site of the concentrator, the hoisting plant, the proposed tailings basin, and the source of water supply.

Costs for clearing and access roads for the surface plant are estimated to be:

Clearing costs = $2000 $A^{0.9}$ for lightly treed area with slopes of not more than 20% gradient (6.3.123)

Access roads = $280,000 per mile for 30-ft (9-m) wide graveled road in mildly hilly region (6.3.124)

These formulas should be modified ± 30% for more adverse or more favorable slope and tree growth conditions.

Excavation of Overburden: Soil overburden must be stripped wherever buildings and facilities are to be sited. The cost of stripping soil overburden *Do* feet deep over an area of *A* acres will be:

$$\text{Cost of soil stripping} = \$1000\ A^{0.8}\ Do \qquad (6.3.125)$$

After the soil overburden is removed and the underlying rock or basal strata is exposed, this rock or strata will require localized removal, probably by drilling and blasting, to establish sound foundation conditions over levelled areas for the plant buildings and plant equipment. If there are *Cu* cubic yards of rock requiring drilling, blasting, and haulage to a dump site, this mass excavation will cost:

$$\text{Cost of mass excavation:} = \$200\ Cu^{0.7} \qquad (6.3.126)$$

for excavations of up to 100,000 yd^3 (76 km^3)

If the mass excavation is in rock that can be broken by ripping, the mass excavation will cost only 20% of the foregoing costs.

When the mass excavation has been completed, detailed excavation to tailor the rock surface to the exact levels for pouring concrete foundations can be done. At the same time, suitable fill will be placed and compacted over level areas where deep trenches of soft soil have been removed. If there are *Cd* cubic yards of rock to be excavated by detailed excavation and *Fc* cubic yards of compacted fill to be placed, the cost will be:

$$\text{Excavation and fill compaction} = \$850\ Cd^{0.6} + \$75\ Fc^{0.7} \qquad (6.3.127)$$

Concrete Foundations for Concentrator Building: Concrete costs for the foundations of the concentrator building, fine ore bins, and concentrator equipment probably will cost between $350 and $900/$yd^3$, depending on whether the concrete pour is for a simple form with little reinforcing steel or for a complex form that is heavily reinforced. The concrete cost may be significantly higher per cubic yard if concrete is scheduled to be poured in winter months when the temperature is below 40 °F (4.4 °C) and heating of aggregate and water and heating of concrete forms is required for sound concrete.

It is difficult to estimate the shape and volume of concrete forms before these forms have been designed, and hence concrete costs related to concrete volume are unreliable for preliminary estimation.

$$\text{Approximate concrete foundation costs} = \$30{,}000\ T^{0.5} \qquad (6.3.128)$$

for concentrators milling *T* tons daily (assuming no difficulties).

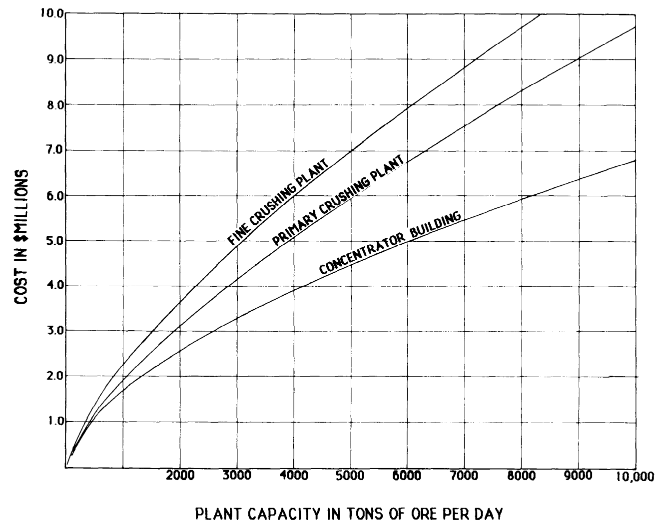

Fig. 6.3.15. Concentrator building and crushing plant costs.

Concentrator Building Costs: The cost of the concentrator building (Fig. 6.3.15) includes all costs of constructing the building above the concrete foundations and enclosing the building, plus the cost of internal offices, laboratories, and changerooms; however, it does not include the cost of process equipment, piping, or electrical wiring, because these items are included in the costs of each functional area. The equipment in operating concentrators generates a substantial amount of heat and comfortable working conditions can be attained with little or no insulation, as long as the concentrator is located in a region with a mild climate.

$$\text{Cost of building} = \$27{,}000\ T^{0.6} \qquad (6.3.129)$$

for flotation mills milling *T* tons of ore daily and located in a mild climate.

A "mild climate" is defined as a region where the degree days are about 7000 (in °F) or 4000 (in °C) per year. Weather stations usually record the "degree days" (°F × *D*, or °C × *D*), which represents the average number of days times the degrees that the temperature is below 65°F or 18°C. In hot climates, where freezing temperatures are not experienced, the building costs may be reduced by only partially enclosing the building and by locating thickeners and other hydrometallurgical equipment outside the building. In cold climates, the additional cost of insulation, heating, and snow loading is likely to increase the building cost by about 10% for each increase of 1800°F × *D* above 7000 (or 1000°C × *D* above 4000).

Primary Crushing Plant with Gyratory Crusher: Although nearly all underground mines place the primary crusher underground to eliminate problems in loading skips and conveyors prior to hoisting the ore, open pit mines generally place the primary crusher on the surface outside the pit, within convenient conveying distance to the coarse ore stockpile and the fine ore crushing plant. Open pit trucks normally discharge ore into a truck dump grizzly mounted over the gyratory crusher, which discharges crushed ore to a conveyor. Because of the headroom required to operate and discharge the crushed ore from a gyratory crusher, a substantial excavation and volume of concrete is required for the primary crusher plant. The cost of the primary crusher depends on the size and capacity of the gyratory crusher selected for an open pit mining *T* tons of ore daily:

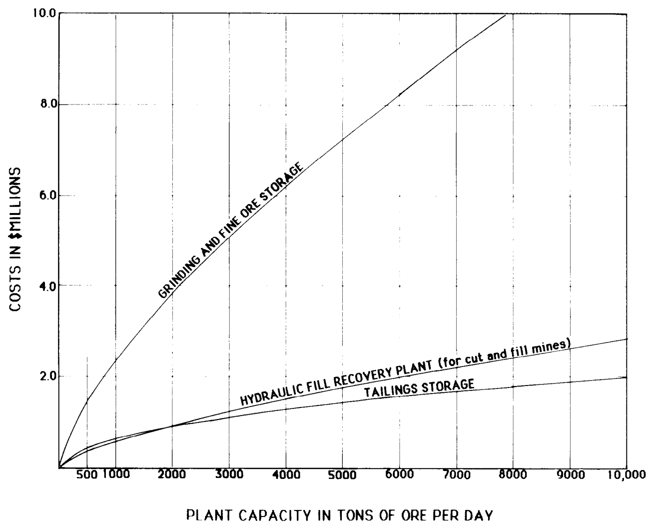

Fig. 6.3.16. Grinding, storage bins, and tailings storage.

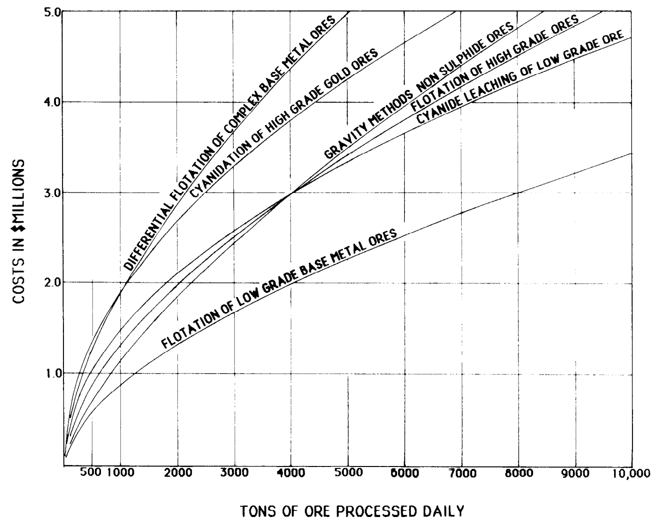

Fig. 6.3.17. Costs of processing section.

$$\text{Cost of gyratory crusher} = \$63\ T^{0.9} \tag{6.3.130}$$

The cost of excavating and concreting the foundations for the primary crusher, installing the crusher, construction of the truck dump and grizzly, plus the coarse ore conveyor and feeder under the crusher is:

$$\text{Cost of primary crushing plant} = \$15{,}000\ T^{0.7} \quad \text{(excluding crusher cost)} \tag{6.3.131}$$

Cost of Fine Ore Crushing and Conveyors: This cost includes the crushing plant building, installed equipment and conveyors.

$$\text{Cost of fine ore crushing plant} = \$18{,}000\ T^{0.7} \tag{6.3.132}$$

Note: Cost may be 12% higher if conveyors must be enclosed and heated.

Grinding Section and Fine Ore Storage Costs: The fine storage bins must have sufficient live capacity to provide mill feed for at least the number of days that the crushing plant is idle per week. The cost of the fine ore bins (Fig. 6.3.16) will be proportional to the weight of steel used in constructing these bins, and the weight of steel will be proportional to $T^{0.7}$, where T is the tons of ore milled daily.

The size and cost of the grinding mills depend on the tons of ore to be ground daily by each mill, but they also depend on the hardness of the ore as measured by the work index and the fineness of grind that is required to attain the desired concentration and recovery of valuable minerals.

Cost of grinding and bins = $18,700 $T^{0.7}$ for medium hard ore with a work index of 15, ground to 70% passing 200 mesh (6.3.133)

= $12,500 $T^{0.7}$ for soft ores ground to 55% passing 200 mesh (6.3.134)

= $22,500 $T^{0.7}$ for hard ores with a work index of higher than 17, ground to 85% passing 200 mesh (6.3.135)

Cost of Processing Section and Related Sections: The capital costs in this section cover the purchase and installation of all equipment required to concentrate or extract valuable minerals from the slurried ground ore, and process the concentrates or extracted minerals into dried solids or impure metals that are directly salable as dry concentrates, ingots of precious metals, uranium yellowcake, or impure metallic gravity concentrates of alloy metals. These capital costs include equipment and tanks for thickening, filtering, precipitation, leaching, solvent extraction, etc., plus all process piping, electrical wiring, and process control.

Process costs (Fig. 6.3.17) for different types of ore by different methods are listed in the following in relation to the tons of ore milled daily T.

1. High-grade gold ores leached by cyanidation, followed by zinc dust precipitation of gold by Merrill Crowe process, filtering, drying, and gold refining:

$$\text{Process capital costs} = \$60{,}200\ T^{0.5} \tag{6.3.136}$$

2. Low-grade ores, cyanide leaching, CIP (carbon-in-pulp) or CIL (carbon-in-leach) adsorption, refining:

$$\text{Process capital costs} = \$47{,}300\ T^{0.5} \tag{6.3.137}$$

3. High-grade gold ores with base metal sulfides; cyanide leaching, secondary flotation, carbon adsorption by CIP or CIL process, filtering, thickening, drying, and refining:

$$\text{Process capital costs} = \$103{,}200\ T^{0.5} \tag{6.3.138}$$

4. Simple low-grade base metal ores of copper with minor content of gold, which can be recovered as smelter credits. Flotation, thickening, filtering, and drying of auriferous copper concentrates:

$$\text{Process capital costs} = \$13{,}700\ T^{0.6} \tag{6.3.139}$$

5. Pyritic gold/silver ores where the precious metals are locked in the pyritic minerals. Differential flotation, selective roasting, recovery of deleterious materials, cyanidation, thickening, precipitation, filtering, and refining.

$$\text{Process capital costs} = \$180{,}000\ T^{0.5} \tag{6.3.140}$$

6. High-grade Cu/Pb ores, Cu/Zn ores, Pb/Zn ores, Cu/Ni ores. Recovery by differential flotation, thickening, filtering, and drying of separate concentrates:

$$\text{Process capital costs} = \$20{,}600\ T^{0.6} \tag{6.3.141}$$

7. Complex base metal ores containing at least three valuable metals, with recoverable minor amounts of precious metals; Cu/Zn/Pb ores, Pb/Zn/Ag ores, Cu/Pb/Ag ores, Cu/Zn/Au ores. Recovery by differential flotation, separate thickening, filtering, and drying of several concentrates and/or bulk concentrates.

$$\text{Process capital costs} = \$30{,}100\ T^{0.6} \tag{6.3.142}$$

8. Nonsulfide ores containing specialty metals such as columbium (niobium), tantalum, tungsten, and tin in minerals that do not respond to flotation, and which are separated by specialized gravity concentration methods:

$$\text{Process capital costs} = \$5000\ T^{0.7} \text{ to } \$13{,}000\ T^{0.7} \tag{6.3.143}$$

9. Uranium ores: acid leaching, countercurrent decantation, clarification, solvent extraction and yellowcake precipitation:

$$\text{Process capital costs} = \$150{,}000\ T^{0.5} \text{ to } \$200{,}000\ T^{0.5} \tag{6.3.144}$$

Capital Cost of Initial Tailings Storage: There are many aspects of tailings storage such as topography, distance from mill to tailings site, localized environmental concerns, etc., that could drastically alter the costs of tailings storage. If, however, all adverse aspects are absent, and a suitable tailings site is available within two miles of the mill, and the nature of the tailings does not have adverse environmental effects, the minimum cost of tailings storage may be:

$$\text{Minimum tailings storage cost} = \$20{,}000\ T^{0.5} \tag{6.3.145}$$

Very few mines have such favorable conditions, and if the area topography is steep or the environmental constraints are stringent, the tailings storage costs could be several times as high as the foregoing cost guide.

Capital Costs of Hydraulic Fill Recovery/Storage: If the underground mine uses the cut and fill stoping method, the hydraulic fill requirements usually can be attained from the tailings by cyclone recovery of the coarser sizes in the ground tailings. The recovery of suitable fill material is usually imposed on the mill plant personnel, as is the storage and slurrying of the recovered fill. The cost of the fill recovery and storage plant is:

$$\text{Cost of fill recovery and storage} = \$4500\ T^{0.7} \tag{6.3.146}$$

Capital Cost of Water Supply System: The cost of fresh water pumping plants, reclaim water plants, and provision for fire protection water supply, plus potable water supply, varies according to the local topography and the proximity and nature of nearby sources of year-round supplies of water. If there is a suitable source of water within two miles of the mill, and the intervening topography is moderately level, the water supply system would cost:

$$\text{Cost of water supply system} = \$14{,}000\ T^{0.6} \tag{6.3.147}$$

The cost of the water supply system for the mine, mill, and plant (but excluding the mine water distribution system) will be much higher if the local topography is steep and rugged or if there are severe constraints on sources of fresh water.

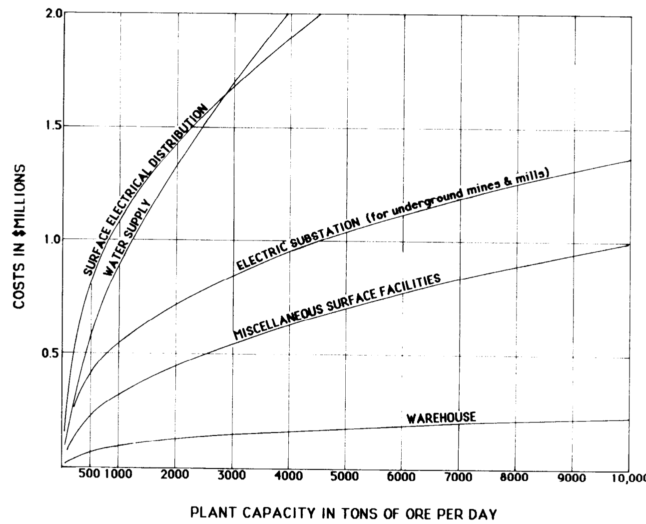

Fig. 6.3.18. Costs of plant service facilities.

Cost of Electrical Substation and Surface Electrical Distribution: The capital cost of electrical facilities for a mining/milling plant depends primarily on the size of the electrical peak load in kilowatts, which can be estimated by Eqs. 6.3.27 and 6.3.28. The cost of power supply depends on whether the power is generated by an existing electric-utility or by a mine diesel-electric plant. Small mines in remote areas may be forced to generate their own electric power, because the cost of a lengthy transmission line from an existing utility may be too high due to the low peak load and low electric power consumption of a small mine.

If the mine is supplied with utility power, the cost of a utility substation with step-down transformers will be

$$\text{Cost of substation} = \$580\ (\text{kW})^{0.8} \tag{6.3.148}$$

where kW is peak load.

The cost of installing low-voltage power distribution to the surface concentrator, crushing plant, and surface facilities, including the mine hoist and compressor plant but excluding the distribution to the surface open pit or the underground mine, is likely to be

$$\text{Cost of surface power distribution} = \$1150\ (\text{kW})^{0.8} \tag{6.3.149}$$

A diesel-electric generating plant may be required for a small mine in a remote area or by a larger mine supplied with utility power that may require a standby electric power plant for protection of vital equipment.

Costs of plant service facilities are shown in Fig. 6.3.18.

$$\text{Cost of diesel-electric plant} = \$6000\ (\text{kW})^{0.8} \tag{6.3.150}$$

Cost of General Plant Services: These costs include the costs of constructing, furnishing, and equipping the general administrative office, general warehouse, electrical and mechanical repair shop (for smaller mill equipment and services equipment), vehicle garages, changehouses, first aid and mine rescue stations, security stations plus general purpose vehicles, parking lots, and yard fencing.

The size of the buildings tends to depend on the number of employees served by each building, which can be estimated by Eqs. 6.3.2 to 6.3.26. It is necessary to estimate the building size in square feet before estimating building cost, which will vary with the area of each type of building.

1. Administrative office. The floor space per person tends to increase as the number of administrative and technical staff *Nat* becomes larger. This reflects the more complex records of accounting and technical staff and the consequent requirement of more space for computer facilities, mining plans, and reference file facilities.

$$\text{Office area } A \text{ required in ft}^2 = 35\ (Nat)^{1.3} \qquad (6.3.151)$$

$$\text{Cost of office} = \$155\, A^{0.9} \qquad (6.3.152)$$

2. Surface plant maintenance shop. Maintenance personnel *Nsv* will require about 85 ft^2/person for maintenance and repair of movable equipment from the mill and service departments.

$$\text{Cost of shop} = \$102 \times (85 \times Nsv)^{0.9} \qquad (6.3.153)$$

3. Mine changehouse. The mine changehouse requires about 24 ft^2/person on the mine payroll (*Nmn* for underground mines or *Nop* for open pit mines) and includes the first aid station and mine rescue facilities.

$$\text{Changehouse cost} = \$125 \times (24 \times Nmn)^{0.9} \qquad (6.3.154)$$

4. Surface warehouse. This should accommodate all supplies and spare parts for the mine, mill, and service facilities that must be kept indoors. Bulky supplies such as rough lumber, structural steel, etc., can be stored outdoors in most climates.

$$\text{Surface warehouse cost} = \$5750\, T^{0.4} \qquad (6.3.155)$$

where T is tons milled per day.

5. Miscellaneous surface facilities. This includes general purpose vehicles and garages, security stations and fencing, parking lots, and miscellaneous services.

$$\text{Miscellaneous surface facilities} = \$10{,}000\, T^{0.5} \qquad (6.3.156)$$

6.3.3.4 Mine Project Overhead Costs

In addition to the direct costs for specific facilities for a mine project, which may total many millions of dollars, there are substantial costs and expenses involved in project design, general site costs, supervision and administration, and provision of working capital. These overhead costs (Fig. 6.9.19) may be estimated as a function of the total direct costs D in dollars.

Engineering: This includes the costs of feasibility studies, environmental impact studies, design engineering, equipment specifications and procurement, and specialized consulting services:

$$\text{Engineering costs} = \$2.30\, D^{0.8} \qquad (6.3.157)$$

General Site Costs: This includes construction camp costs, specialized construction equipment, and general construction site costs:

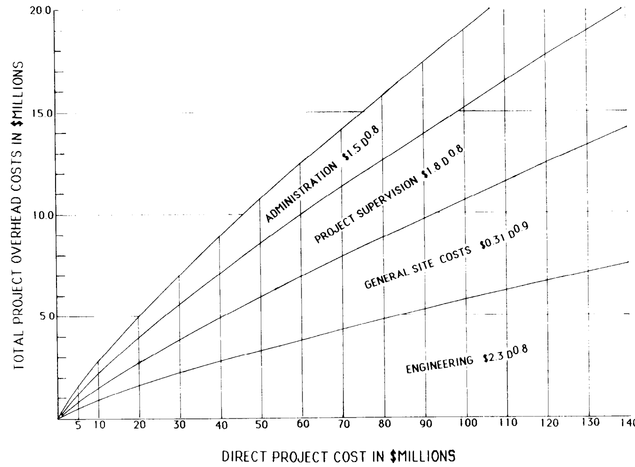

Fig. 6.3.19. Project overhead vs. direct costs.

$$\text{General site costs} = \$0.310\, D^{0.9} \qquad (6.3.158)$$

Project Supervision: This includes project supervision, scheduling and budgeting, and construction management:

$$\text{Project supervision costs} = \$1.80\, D^{0.8} \qquad (6.3.159)$$

Administration: This includes local office administration by corporate owner's representatives, accounting and payment of general contractor, legal costs, plus preproduction employment of key operating staff:

$$\text{Administration costs} = \$1.50\, D^{0.8} \qquad (6.3.160)$$

Project overhead costs as a percentage of direct project costs tend to vary depending on the size and complexity of the project; the lower percentages of 4 to 6% would be typical for $100 million projects and conventional technology, whereas the higher percentages of 8 to 11% would apply to smaller $10 million projects that are technically novel or complex.

Working Capital: The allowance for working capital for a mining project should be sufficient to cover all operating costs plus purchase of the initial inventory of capital spares and parts until revenue is received from smelters or purchasers of metallic products. The time period elapsing before receipt of revenue sufficient to pay imminent operating costs will vary depending on the smelter terms or marketing terms, but the typical allowance is about 10 weeks after the concentrator is operating at full capacity.

Typical working capital allowance
= operating costs for 10 weeks

after commissioning of concentrator plus cost of purchasing initial inventory of capital spares and parts.

Whenever the mine or mill design is based on extensive usage of reconditioned used equipment, there is a higher frequency of equipment downtime that requires additional time allowance of working capital; this will decrease the apparent savings of used equipment.

6.3.4 COST GUIDES FOR OPERATING COSTS OF MINES AND MILLS

The cost guides provided in the form cost = KT^x compute the cost of each mining activity as a function of the tons T of ore mined and express this cost as a cost per day of mining activity. The cost per ton can be derived from the formula by dividing the cost per day by the tons mined per day so that cost per ton $= KT^x/T$ which equals $K/T^{(1-x)}$. Thus if 100 $T^{0.7}$ represents the cost per day, the cost per ton $= 100/T^{0.3}$.

When a mine is producing T tons per day from all mining methods, but the mining method being costed produces only t tons of ore per day, then the cost of the mining method per day equals $Kt/T^{(1-x)}$, although the cost per ton remains at K/T^{1-x}.

6.3.4.1 Underground Mine Operating Costs per Day

The cost per day for different mining methods includes the cost of labor and supplies for drilling, blasting, ground support, loading, and haulage of stoped ore from each stope. This cost covers work done by the stoping and mucking crew only and does not include labor and supplies involved in crushing, transport, and hoisting of ore; neither does it include general mine services, supervision, or mine activities that are not specific to the mining methods being used to recover ore from the stopes.

Stoping Costs:

1.	Shrinkage stoping costs	$= \$146\ T^{0.6}$ per day	(6.3.161)
2.	Cut and fill stoping costs	$= \$185\ T^{0.6}$ per day	(6.3.162)
3.	Fill distribution	$= \$22\ T^{0.7}$ per day	(6.3.163)
4.	Longhole stoping costs	$= \$160\ T^{0.6}$ per day	(6.3.164)
5.	Fill distribution for bulk filling longhole stopes	$= \$12\ T^{0.7}$ per day	(6.3.165)
6.	VCR stoping costs	$= \$125\ T^{0.6}$ per day	(6.3.166)
7.	Room and pillar stoping costs (hard rock)	$= \$130\ T^{0.6}$ per day	(6.3.167)
8.	Room and pillar stoping costs (soft rock)	$= \$85\ T^{0.6}$ per day	(6.3.168)
9.	Sublevel caving costs	$= \$115\ T^{0.6}$ per day	(6.3.169)
10.	Block caving costs	$= \$105\ T^{0.6}$ per day	(6.3.170)

These cost guides to stoping costs are based on the presumption that the stope widths (or heights) are as follows: shrinkage 8 ft (2.4 m), cut and fill 15 ft (4.6 m), longhole stoping and VCR stoping 30 ft (9.1 m), room and pillar stoping 12 ft (3.7 m) high, but if the actual width W(or height H) differs from the assumed width Wa (or height Ha), the stoping costs should be corrected by the ratio of the assumed width divided by the actual width with the quotient taken to the 0.4 power.

$$\text{Stoping cost correction} = (Wa/W)^{0.4}, \text{ or } (Ha/H)^{0.4} \quad (6.3.171)$$

Stope Preparation Costs per Day: Stope preparation costs may be estimated as follows:

$$\text{Preparation costs} = \$85\ T^{0.48}\ W^{0.2} \text{ for shrinkage stopes} \quad (6.3.172)$$
$$= \$9.60\ T^{0.7}\ W^{0.5} \text{ for cut and fill stopes} \quad (6.3.173)$$
$$= \$19.20\ T^{1.06}/W^{0.6} \text{ for blasthole stopes} \quad (6.3.174)$$
$$= \$29.04\ T^{1.04}/W^{0.6} \text{ for VCR stopes} \quad (6.3.175)$$
$$= \$2{,}200\ T^{0.6}\ H^{0.2} \text{ for block caving stopes} \quad (6.3.176)$$

Crushing and Hoisting Costs per Day: When underground ore is crushed and hoisted, the cost per day depends primarily on the tons of ore crushed or hoisted per day. The size of ore fragments to be crushed or hoisted makes little difference to the daily crushing cost or daily hoisting cost; consequently the type of stoping method employed has little effect on crushing costs or hoisting costs.

$$\text{Crushing costs per day} = \$2.00\ T^{0.8} \quad (6.3.177)$$

$$\text{Hoisting costs per day} = \$4.70\ T^{0.8} \quad (6.3.178)$$

Cost of Mine General Services: This cost includes all labor and supplies required to maintain all direct mine services, including ventilation, pumping of drainage water, repair and maintenance of mine equipment, maintenance of development levels and ground support, plus mine supervisory staff. This cost includes all cost items that are not already costed previously and that are directly supervised by mine supervision; it should not include any "distributive costs," which are allocated and supervised by staff other than the mine supervision.

General service costs should be retained in the department that has supervisory responsibility over the activity that incurred these costs. Thus the cost of electric power utilized by the mine, concentrator, and general services should not be allocated to each department on some expedient basis, unless each departmental usage of electric power is metered directly and for which the departmental supervision is to be held accountable.

Thus the cost formulas shown in the following for mine general services apply only for those costs that are committed by personnel under the supervision of mine staff.

$$\text{Cost of mine general services} = \$75\ T^{0.8} \quad (6.3.179)$$

$$\text{Cost of mine supervision} = \$12\ T^{0.7} \quad (6.3.180)$$

6.3.4.2 Open Pit Operating Costs per Day

The operating costs of open pit mines depends on the size and numbers of drills, shovels, and trucks, which in turn is dependent on the tons per day of ore and waste. In most open pit mines mining low grade ore, there is little if any difference in the specific gravities, blasting characteristics, and drillabilities of ore or waste, and the haulage distance to the ore dump usually does not differ very much from the waste haulage distance. Consequently, the cost of mining a ton of ore will be virtually the same as the cost of mining a ton of waste.

The cost of open pit mining can be assessed against the total ore and waste tonnage (T_p) mined daily.

Drilling cost per day	$= \$\ 1.90\ T_p^{0.7}$	(6.3.181)
Blasting cost per day	$= \$\ 3.17\ T_p^{0.7}$	(6.3.182)
Loading cost per day	$= \$\ 2.67\ T_p^{0.7}$	(6.3.183)
Haulage cost per day	$= \$18.07\ T_p^{0.6}$	(6.3.184)
General services cost per day	$= \$\ 6.65\ T_p^{0.7}$	(6.3.185)

The open pit general services cost includes the cost of pit maintenance, road grading, waste dump grading, pumping, and open pit supervision, but it does not include the cost of primary crushing or electric power.

6.3.4.3 Concentrator and Services Operating Costs per Day

The operating costs per day for the crusher, concentrator, and general surface facilities are grouped together, because these costs are generally interrelated in terms of sequential activities applied to the run-of-mine ore as received from the mine to convert this ore into a salable product shipped from the plant to the purchaser. Thus, although the gyratory crusher may be located at the edge of the open pit, the costs of operating it are grouped under milling costs, rather than as open pit operating costs, as the first stage of ore treatment. Conversely, the cost of operating an underground jaw crusher is usually considered a mining cost because it is operated and repaired by the mine crew.

The cost of treatment of tailings in storage, or the cost of recovery of coarse tailings for mine backfill, and the cost of gold refining, or the contract trucking of concentrates from the mill to the purchaser, can be regarded as the final stages of ore treatment.

Operating costs that apply to the complete mining and milling plant, such as electric power consumption, which is metered only at the main electric substation, is grouped with concentrator costs. This results because the concentrator is usually the largest consumer of power in the mining and milling complex, and because most attempts to allocate power costs to different sections of the plant are uncertain at best, unless the power distribution to each section of the plant is separately metered.

The design of the milling flowsheet is usually optimized after extensive testwork on the types of processes tailored to the characteristics of the ore; at the preliminary feasibility stage however, the optimum processing requirements are not known with accuracy, and the costs of processing can only be approximately estimated.

The following cost guides are offered as rough estimates of crushing and concentrating costs per day.

Primary Crushing Costs per Day: This cost includes the cost of primary crushing, the cost of conveying the primary crushed ore to the coarse ore stockpile, plus operating costs of the coarse ore stockpile.

Crushing costs per day = $7.90 $T^{0.6}$ for open pits and mills (6.3.186)

= $2.00 $T^{0.8}$ for underground mines and mills (usually included in mining costs) (6.3.187)

Fine Crushing and Conveying Costs per Day: This includes fine crushing, conveying from coarse ore storage, and conveying to the fine ore bins.

Fine crushing costs per day = $12.60 $T^{0.6}$ (6.3.188)

Grinding and Fine Ore Bins Cost per Day: This cost includes the fine ore bin storage and the rod mills, ball mills, and/or SAG (semiautogenous grinding) mills:

Grinding section costs per day = $4.90 $T^{0.8}$ (6.3.189)

Process Section Costs per Day: This includes the operating costs of all sections that involve concentration of ore by flotation or by gravity, leaching of metals from ore, thickening of slurries, ion exchange, precipitation, filtering, drying, and recovery of metallic concentrates, or deleterious materials that would otherwise penalize smelter revenue.

Processing costs per day = $65.00 $T^{0.6}$ for cyanidation of gold/silver ores (6.3.190)

= $54 $T^{0.6}$ for flotation of simple base metal ores (6.3.191)

= $34 to $41 $T^{0.7}$ for complex base metal ores varying in complexity (6.3.192)

= $65 $T^{0.7}$ for uranium ores by leaching, CCD, solvent extraction, and precipitation (6.3.193)

= $45 $T^{0.7}$ for nonflotable nonsulfide ores responding to gravity separation (6.3.194)

Tailings costs per day = $0.92 $T^{0.8}$ for all concentrators (6.3.195)

Assaying costs per day = $1.27 $T^{0.8}$ for all concentrators (6.3.196)

Supervision, maintenance, and general costs per day = $40.80 $T^{0.6}$ for all concentrators (6.3.197)

Processing costs should be decreased to 55% of those shown by the foregoing formulas when low-grade ore, typically mined by open pit mining, is being treated by a concentrator that rejects tailings at an early stage.

Electrical Power Costs per Day: The peak load and daily electric power consumption of the mine plant, consisting of an underground mine or open pit, crushing plant and concentrator, plus surface services and general administration office can be estimated from Eqs. 6.3.27, 6.3.28, 6.3.29, and 6.3.30. These equations indicate that a small underground mine and plant processing 500 tpd (454 t/day) of ore will have a peak load of about 3700 kW, and a daily consumption of about 62,000 kWh.

Under these conditions, the unit cost of power is typically about 8¢/kWh, and the unit cost is normally reduced for larger mines that use larger quantities of electric power. Usually, the unit cost of electric power is reduced in stages by about 4.3% each time the power consumption is doubled, and by using these typical assumptions, it is possible to estimate the daily cost of electric power for mines and mills larger than 500 tpd.

Cost of electric power = $ 164 $T^{0.56}$/day for underground mines and plants processing T tons of ore/day (6.3.198)

= $145 $T^{0.56}$/day for open pit mines and plants processing T tons of ore/day (6.3.199)

Surface Services Cost per Day: The daily cost of each person in the surface maintenance and general services departments is estimated to be $141 in wages and fringe benefits, plus an average cost of $16 in supplies consumed. If the number of maintenance and general services personnel is *Nsv* as estimated from Eqs. 6.3.22, 6.3.23, and 6.3.24, then the daily costs of maintenance and general services departments is

Services cost per day = $157 *Nsv* (6.3.200)

Costs of Administration and Technical Staff: The daily costs of the administrative and technical staff, including supplies and services required by them, plus fixed costs for local property taxes and legal fees paid by administrative services, are proportional to the number of staff *Nat,* as estimated in Eqs. 6.3.25 and 6.3.26.

Each staff person is estimated to cost on the average $185 in salary per day, and to consume $37.60 in supplies and services per day.

Total cost per day for administrative and technical staff salaries and supplies $= \$222.60\ Nat$ (6.3.201)

6.3.5. CONCLUSION

The consequences of inaccurate estimation of capital and operating costs in feasibility studies may include the commitment of major amounts of capital funds before it is realized that the mining project will not be profitable, or the rejection of a proposed mining project that could be profitable. Accurate cost estimation is possible only after a substantial amount of technical activity has been competently completed. This technical activity should include the following:

General mine planning of all mine development, including scaled maps and drawings from which the lengths, quantities, and unit costs of all mine excavations and openings can be estimated.

Assessment of the sizes, types, numbers, and prices of mine and mill equipment that will be required for the planned rate of production.

Environmental impact studies for the regional area surrounding the proposed mine site and the nature and content of all fluids and gases that will be discharged from the mine and plant.

Metallurgical tests of bulk samples of mine ore treated by the proposed concentrator process.

General plant layout must be completed to show the dimensions of all plant buildings and the depth and amount of excavation required for sound building and equipment foundations. Access roads, transmission lines, water lines, and tailings lines must be mapped and the topography contoured.

Cost formulas can provide some guidance as to the order to magnitude of capital and operating costs, but the accurate estimation of costs depends on measured quantities taken from the design planning, plus unit costs quoted by contractors or derived from unit costs of recently completed projects similar in nature. In general, the accurate estimation of project costs must be entrusted to consulting engineering firms and project management firms with experience in many similar projects.

Most mining engineers employed in operating mines do not have the necessary expertise or training to determine the design features or costs of surface mining plants, and these should be determined by experienced firms employing engineers skilled in civil, structural, mechanical, chemical, metallurgical, and electrical engineering.

Additional assistance in cost estimation may be obtained from the following sources or publications:

1. *General Construction Estimation Standards,* 6 volumes, revised annually, published by Richardson Engineering Services, Inc., P.O. Box 1055, San Marcos, CA 92069.
2. *Means Construction Costs,* revised annually, and published by Robert Snow Means Co. Inc., 100 Construction Pl., Kingston, MA 02364.
3. *US Bureau of Mines Cost Estimating System Handbook,* 2 volumes, Information Circular 9142 (surface and underground mining), and Information Circular 9143 (mineral processing). Mining and milling costs are as of January 1984. The two volumes, IC 9142 and IC 9143, are available from the Superintendent of Documents, US Government Printing Office, Washington, DC 20402
4. *Canadian Construction Costs: Yardsticks for Costing,* revised annually; available from Southam Business Publications, 1450 Don Mills Rd., Don Mills, ON, Canada, M3B 2X7.
5. *Mining and Mineral Processing Equipment Costs and Preliminary Capital Cost Estimations,* Special Vol. 25, 1982; published by The Canadian Institute of Mining and Metallurgy, 1 Place Alexis Nihon, 1210-3400 de Maisonneuve Blvd. W., Montreal, PQ, Canada H3Z 3B8.

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Chapter 6.4
PROJECT OPERATING STRATEGY

DONALD D. HAAS

6.4.1 INTRODUCTION

The purpose of this chapter is to assist the owner of a mining property in the development of the proper strategy for bringing the property into production. The owner basically has five alternatives, or a combination of these five alternatives, for bringing the property into production. The five alternatives are (1) develop the property solely as the owner and operator, (2) employ a contract miner to perform some or all of the development and/or mining, (3) employ a contract miner to initiate production and to establish an operating mine and subsequently then, as an owner/operator, to operate the entire property, (4) lease the property to an established mining company or developer for exploitation, and (5) joint venture the project with another mining company.

Each of these alternatives is described in detail in 6.4.2, and each has its unique advantages and disadvantages. The use of any particular alternative depends on the advantages and disadvantages of a unique set of circumstances, the economic consequences of the alternatives, and the corporate philosophy of the property owner.

6.4.1.1 Definitions

The following definitions are provided to minimize misunderstandings that might otherwise occur when analyzing the operating strategies offered in this chapter.

Owner: The owner is the individual or company that has ownership and/or control of a property. In this chapter, the owner is attempting to develop the property into an operating mine.

Owner/Operator: This term refers to a condition where the owner of the mineral resources, either through actual ownership of the mining claims or a lease of mining claims, fee land, or other type of ownership from others, is the entity that constructs, initiates and continues mining production, and operates the milling and/or processing facilities. Some of the work on the project may be under contract to others, but total control and operation of the mine and plant are the responsibility of the owner.

Contract Mining: For surface mining, contract mining is, at a minimum, the removal of waste or overburden, the extraction of ore, and the delivery of that ore to a processing facility. For underground mining, contract mining is, at a minimum, the development of the workings, the extraction of ore from stopes or other production openings, and the delivery of that ore to the client, either at the shaft loading pocket or at the surface. Contract mining may include many other activities for both surface and underground mining. For example, in surface mining, other activities may include development of an infrastructure for the operation; construction of access to the site; clearing and grubbing the site; the removal, stockpiling, and replacement of topsoil; water control around the mine area; crushing and other preprocessing activities; construction of the processing facilities; and management and/or operation of the ore processing plant. For underground mining, these activities may include development of access to the site; development of the underground mine (including construction of the headframe or other surface facilities); sinking the shaft; development of the stations and levels; and construction, operation, or management of the processing facilities.

Startup: The startup portion of a mining operation, sometimes called mine development, is that period of time during which access to the mine is being developed (other than the access needed for exploration activities); the main haul roads are being constructed; the initial shaft, incline, or adit is being driven; the first pit or cut is being established; the haulage system is being developed; and the mine is being brought to full production levels. These activities also include stabilizing the labor force for the mine, establishing supervisory personnel and staff activities, and stabilizing the production and costs of the operation.

Lease: A contract that allows an individual or company other than the owner of the property to develop and operate a mine on a property, produce a product, and sell the product. The owner derives benefit from the leasing of the property either through a royalty arrangement such as a sharing of the revenues or profits from the operation, a set fee per weight of ore or refined product, or a flat fee per period of time.

Joint Venture: A joint venture is the joining of two or more companies for the purpose of developing and operating a mining property. A joint venture, as used in this chapter, is a joint operation where each partner brings a needed knowledge or expertise to an operation and is not to be confused with joint ventures formed for the purpose of providing financing for the project or formed for the purpose of acquiring a property for only one of the parties to operate. Marketing of the mineral may be a needed knowledge. A joint venture merely to share the risk of a project is not addressed. This type would be the result of corporate philosophy, not necessarily economics. The interested reader is referred to Gentry and O'Neil (l984) for more extensive discussions on the various forms and the financial implications of joint ventures utilized in the minerals industries.

6.4.2 PROJECT ALTERNATIVES

A more detailed discussion of the five major project alternatives follows, along with some of the major ramifications, advantages, and disadvantages of each. The owner of a property must consider the different alternatives for the development of a property before selecting the appropriate alternative.

6.4.2.1 Owner/Operator

The owner/operator alternative is the most common system of mine development; consequently, the bulk of this section addresses this type of alternative. The owner of the property either stakes the ground or acquires leases to the property for the expressed purpose of developing the property into a profitable mining operation. The staking of claims or the acquisition of leases may have occurred long before a determination is made to develop the property into a mine.

The feasibility studies necessary to assist with the decision to develop the property into a mine are discussed in detail in Chapter 6.2 of this section. Chapter 6.3 outlines methods and procedures for developing cost estimates for mining properties. Analyses of the investment decision and the financing of the operation are discussed in Chapters 6.5 and 6.6 of this section, respectively.

The owner/operator may decide to contract some of the activities at the mine such as detailed design and/or construction of the milling and processing plant, the erection of mining equipment, construction of access to the property, and even some removal of overburden, and still be classified owner/operator of the project. The owner/operator mines and processes the ore in the normal operation of the project.

Advantages and Disadvantages: The primary advantage of the owner/operator alternative is the total control the owner has of the development and exploitation of the property. In each of the other alternatives, the owner must relinquish some control of at least some aspect of the operation. The owner directly gains any advantages from reduced costs or improved recoveries at the operation. For example, when a contractor is employed, the owner is dependent upon the good mining practices of the contract miner for recovery of all the ore and for minimizing dilution in mining of the ore. Also for the same basic reason in a lease, the owner is dependent on the lessee recovering all the ore in the pit or in the stopes and then extracting the mineral commodity from the ore.

The owner's profit is dependent generally upon the amount of product sold. Thus it is advantageous to have total control over all aspects of the operation that directly influence this parameter.

A disadvantage is that the owner may experience higher cost per volume of commodity recovered using his own operation than using a contract miner or leasing the property to someone who possesses more expertise or efficiency. Typically, the owner reduces the risks associated with operating costs by employing a contract miner or by leasing the property. Or the owner shares the risk by obtaining a joint venture partner or partners.

The fact that the owner retains all risks associated with development of the property may be either an advantage or disadvantage. The owner loses if the cost of mining or processing increases; similarly, he gains if the cost decreases. The owner loses if the amount or grade of ore developed does not exist; he gains if there is more or higher-grade ore than anticipated.

6.4.2.2 Contract Miner

For a surface mine, the contract miner operates the mine, including the removal of overburden waste, mines the ore, and delivers the ore to a milling or processing facility. For underground mining, the contract miner develops the workings, extracts the ore from the stopes or production openings, and delivers that ore to the client, either at the shaft loading pocket or at the surface. The contract miner may perform many other tasks for the owner, including construction of access to the site and other preproduction activities; crushing, milling and other processing activities; reclamation of the site after mining is complete; development of the underground mine; construction of the headframe and other surface facilities; sinking the shaft; and station and level development. The contract miner does not generally operate a processing plant for the owner, due to the difficulty in monitoring the contract miner's efficiency in recovering the marketable commodity from the ore. The contract miner may operate the processing plant during start-up, but the owner generally assumes responsibility for operations shortly after the operation reaches full production and the processing facility has been debugged.

Advantages and Disadvantages: As an owner considers whether or not to contract all or a portion of his operation, he is faced with a choice that has both positive and negative consequences no matter what the decision. Each property situation is different; therefore, there is no universally superior answer for the industry. Such factors as the financial strength of the owner, degree of development of the mine plan, corporate philosophy concerning control and growth, mine location, possible parallel operations at other sites controlled by the owner, etc., all come into consideration when determining which course to follow.

The following list of advantages and disadvantages should be viewed and evaluated on a site-by-site basis, and each item should be weighed as to its relative importance under the specific circumstances associated with the property and the controlling interest.

Possible Advantages—

1. Contract mining reduces the magnitude of initial capital investment required to bring the property into production. The owner already has invested considerable capital resources in exploration and/or acquisition, engineering, and permitting for the mine. The capital outlay for equipment represents a substantial load on company resources and could restrict activities or even be an insurmountable barrier to bringing the property into production. The capital funding from the owner is preserved for the construction of the processing plant, other facilities, and possibly other mining properties. The financial resources of the owner are conserved by using a contract miner.

2. Contract mining reduces direct payroll and administrative costs. Items such as recruiting, training, and supervision become the responsibility of the contract miner, along with the responsibility of health care benefits, insurance, and all the other costs and planning required to maintain a qualified work force.

3. Contract mining supplies a well-defined fixed cost for the mining operation. An owner should develop an estimate of mining costs, but it is difficult to anticipate and accurately estimate all cost items such as maintenance, accidents, equipment availability, and so forth. Since the contractor has control over these activities, he assumes all the risks for any cost increases that might be incurred.

4. Fixed operating expenses may facilitate financing for the project. If a respectable contractor is chosen and operating costs are contractually defined, financial institutions often recognize that the project has a better chance of meeting projected costs and production levels because the contractor has assumed some of the risks inherent in project start-up. This, of course, largely is dependent on a properly engineered mine plan.

5. The owner has greater flexibility during the life of the mine. Should the owner buy all necessary equipment at the start of production and then, for example, wish to increase or decrease production because of any number of changes in key project variables, the owner may find that the wrong equipment was purchased to perform the required function efficiently. The owner is then faced with a possible financial loss while trying to sell some equipment and procure other replacements. Most active contract miners are also construction companies; it is probable they can change out the equipment with an adjustment to unit costs that saves money for the owner.

6. A good contract mining agreement improves efficiency for the project. The contract miner typically has had to win the award on a competitive basis. To do so, the contractor uses production figures based on experience in similar circumstances; the only way the contract miner can show a profit is to meet or exceed these projections. When circumstances arise that defeat these objectives, the contract miner makes every effort to overcome the problem. Where necessary, he works with the owner to correct conditions he cannot control. (An example would be if the contractor becomes ore-bound or waste-bound, he may not be able to get full utilization from his crews and equipment.) Although this may seem to be a problem, it is actually an advan-

tage, since it forces the owner and operator to work jointly to keep overall costs down. Without this relationship, especially at a profitable mine site, additional inefficiencies may be tolerated because the mine is providing an adequate profit margin. For a more complete discussion of this aspect, see Haas, 1989.

7. Contract mining reduces costs for mines with short lives. The owner may have other operations where equipment can be placed in use as a mine curtails production. However, if this is not the case, the owner faces extraordinary losses as the owner is forced to depreciate the cost of equipment over an inordinately short time period. In contrast, contractors usually depreciate equipment over its expected life rather than on a project basis, since the contractor can transfer the equipment from job to job.

Disadvantages—

1. The owner relinquishes some control of the project. It is necessary for the owner to work closely with the contract miner to ensure that the overall operation enables the contractor to meet required production projections.

2. Total operating costs per unit of mined material may be higher than those resulting from an owner/operator property. However, this observation should not be determined subjectively; it must be determined based on careful financial evaluations utilizing *all* costs (operating and capital), annual cash flows, and time value of money concepts. Such an evaluation from a cost minimization standpoint is provided in 6.4.7.

3. Selection of an inefficient and unreliable contract miner is a risk. Choosing the wrong contract miner brings on more problems than normally encountered when operations are conducted by the owner. The owner must evaluate the contract mining company being considered to ensure it has the financial resources, personnel, experience, and general operating philosophy necessary to meet project needs. This is best accomplished by checking qualifications carefully and asking pertinent questions at pre-award meetings.

6.4.2.3 Contract Miner During Start-up

The owner of a mining property may wish to use a contract miner during project start-up so as to alleviate some of the problems normally associated with start-up and to reduce capital requirements during this period. The contract miner also may be used to perform many of the preproduction activities at the mine and thereby reduce the number of contractors working on a project site during preproduction. A contractor may be employed especially during the start-up of a processing facility to utilize his experience and expertise in debugging the process and in developing the procedures necessary for efficient operation.

The contract mining bidding process and the procedures for monitoring the contractor are the same as when using a contractor for continuing operation. These are discussed in subsequent portions of this chapter.

Advantages and Disadvantages: The same basic advantages and disadvantages apply to the use of a contract miner during project start-up as they do for an entire mine project, except that they exist for only a short period of time—the preproduction development period. The advantages, in addition to those previously stated, are

1. Work crews are established by the contract miner. The contract miner recruits the hourly work force during the development period. The owner may choose to employ all or part of this work force when the contract miner's obligations are completed. Also, all or a portion of the contractor's staff labor may be employed by the owner. A distinct advantage of this arrangement is that the owner has not had to develop the training programs necessary or to perform the selection and "weeding out" process in establishing a work force.

2. Contractor's equipment may be acquired by the owner. If desired, the owner can construct the agreement with the contract miner so that the owner can acquire the needed equipment for continuing operation. This equipment can be purchased by the owner at reduced prices (fair market value for used equipment or other agreed upon method) and reduce the initial capital investment. Also the owner can select the equipment desired for maximum operating efficiency and minimum operating costs under the operating procedures or techniques planned.

6.4.2.4 Leasing of the Property

Leasing of a mining property usually occurs when the owner of the property does not have the financial capability, expertise, or desire to explore, develop, and/or operate a mining project. Leasing normally involves the transfer of all rights to explore, develop, and operate the property to another individual, organization, or company.

Leasing a mining property to another person or company has the principal advantages of reduced risk and reduced investment in financial and human resources. Mine development is extremely risky in terms of financial and human factors. An old adage in mining is that for every 100 discoveries made, one is worthy of being called a prospect and deserving of significant investment in an effort to explore further. Furthermore, for every 10 prospects, one makes a mine that develops sufficient returns to reimburse the owner for the investment of finances and personnel. The investment to explore, develop, and operate a mine is significant and is generally in the order of tens of millions or hundreds of millions of dollars. Some owners of potential mining properties do not have sufficient financial resources to explore, develop, and operate a mine. If the necessary financial strength is not available, either from the owner directly or through external financing, the owner may wish to either lease the property to a company with the required financial strength or joint venture the property.

The primary disadvantages of leasing are the loss of control over property development and the loss of significant upside potential from mine operations if a major ore body is discovered or high-grade ore is located. The loss of control does not allow the owner to direct subsequent efforts in exploration, development, and operation. If a large or high-grade ore body is discovered and/or the mining venture is very profitable, the lessor receives only the negotiated royalty. Some of this disadvantage can be mitigated by negotiating the royalty provision based on net profits from the mining venture; however, this approach also increases the owner's risk by reducing overall return if the mining venture is only marginally profitable.

6.4.2.5 Joint Venture

Joint ventures usually involve the affiliation of two or more companies in order to benefit from the expertise of all companies and the property ownership of one of these companies. Joint ventures, as the term is used in this chapter, represent the partnership of two or more companies to develop a mining property for the purpose of benefitting from the combined technical and/or operating expertise of the companies and not as a means to finance the venture.

The major advantages in using a joint venture are the benefits derived from the expertise of another company and the sharing of risks. Sometimes, after a property has been discovered and explored to the extent that a viable project appears to be a distinct possibility, the owner often determines he is deficient in the expertise needed to exploit the property to its fullest potential. Joint venturing the project with one or more other companies is one of the means of obtaining the desired expertise for the successful exploitation of the deposit. Also some projects may become too large or complex for one company to undertake. In this case, the company may desire to share the risk of the development with another company through a joint venture.

The major disadvantages of using a joint venture are the sharing of proceeds of the project and the loss of control. When a project is developed jointly, the owner must share some of the proceeds with the partner(s). If the expertise gained with the partner(s) does not exceed the cost of sharing the proceeds, a joint venture probably is not the alternative to be used. Also, when a project is developed jointly, decisions regarding property development must be shared. Each partner needs to have sufficient voice in the decisions to protect their investment and to insure an adequate return on his investment.

6.4.3 CONTRACT MINING BIDDING PROCESS

The contract mining bidding process involves several steps to develop a workable agreement with the successful contractor who will perform contract mining on the owner's project. These steps are as follows: (1) develop the bidder's list, (2) prepare the request for proposal (RFP), (3) develop the proposal by the bidder, (4) evaluate the proposals, (5) select the contract miner, and (6) write the agreement. Each of the ensuing discusses briefly the suggested procedures to be followed and the importance of the relevant factors in each area.

6.4.3.1 Developing the Bidder's List

The role of contract mining companies to provide services for the mining of ore and waste and the crushing and transportation of ore has increased significantly in the past decade. Although the concept of contract mining is not new, the scope of services provided is growing.

The basic concept of limiting contract miners to early-stage development and other preproduction activities has changed dramatically. The primary reasons for this change are (1) owners may not want to commit the necessary human or financial resources to operate a mine, (2) the owner does not desire to purchase and maintain an equipment fleet, (3) the owner does not wish to invest in developing a professional operating staff and skilled labor force for short-term projects, (4) the financial risk capital for the owner is minimized, and/or (5) the contractor shares in the risks of mining property start-up and operation.

The goal of the entire contract-mining bidding process is to (1) select the most qualified bidder, (2) obtain the lowest competitive cost, and (3) permit both parties to make a reasonable profit.

After a mine owner decides to utilize contractor services to operate the proposed mine, a bidder's list of qualified contract miners must be developed. Ideally, the mine owner wants to award the contract to the most qualified contract miner, while obtaining the lowest competitive unit costs. The resulting owner/contractor relationship should enable both to realize a profit on the ensuing activities.

Compiling a bidder's list can be either formal and structured or informal. The formal procedure involves preparing a request for qualifications (RFQ) and sending it to potential bidders who, in turn, respond with a qualification proposal detailing their capabilities to perform contract mining. Information requested in the RFQ is generally divided into two categories: (1) company profile and general information and (2) project-specific qualifications.

The company profile generally includes five components: (1) general company data, (2) type of organization, (3) financial resources and references, (4) approach to work, and (5) personnel resources. *General company data* should include the company's name, address, and contact person. *Type of organization* should indicate if the firm is incorporated, the date and state of incorporation, the parent company (if applicable) or major subsidiary to be used, and an organizational chart or list of company officers.

Financial resources and references should include average annual income (revenue) and backlog, banking and bonding references, bonding capacity, any significant pending litigation, and the company's latest annual report or financial statement. *Approach to work* should indicate the type of work typically performed by the contractor's own forces, what work is generally subcontracted to others, the existence of an affirmative action program and other desired or mandated programs, details of safety programs and the record of those programs, affiliation with any labor organization, and the dollar value limits (maximum and minimum) for work undertaken by the contractor. *Personnel resources* should detail the number and types of personnel generally available for contract mining work by category and location.

Project-specific information is divided into three areas: (1) the contractor's approach to the project, (2) the proposed project personnel and organizational structure, and (3) the company or personnel experience relative to the project. *Approach to the project* should include the type of venture (joint venture or prime contractor with subcontractors) and the role each will have in the project. It should also include a preliminary project schedule, equipment availability and status report, and the type of agreement (including reimbursement) anticipated, preferred, and/or acceptable to the contractor. *The personnel and project organization* should include a preliminary project organizational chart, identification of key project personnel (including alternates) and their resumes, and the availability of these individuals for the project. Although most contractors are unable to totally commit personnel in a RFQ response, they can supply a list of people who may be available for assignment. *Relative project experience* should address directly related operating experience, experience on projects similar to the one being considered, and any other project experience that the contractor wishes to include in the evaluation of his credentials. Client references should include company name, address, phone number, and contact person.

Upon receipt of the completed qualifications proposal, the owner's technical staff evaluates each contractor's qualifications carefully. Particular items requiring careful scrutiny are the contractor's previous experience in performing similar work, the ability to finance the contract mining portion of the project (purchase the required equipment, working capital, etc.), the work force (number and type of personnel) available, and the working relations with mine owners on existing and/or completed projects. Using this and similar data, the owner's technical staff can determine which bidders are qualified and reputable contract miners. Certain risks and problems are encountered, particularly in meeting production levels and maintaining grade control, when owners engage the services of a contractor with little or no mining experience.

As opposed to this formal procedure, an informal approach is more often used in the mining industry because of the close association of individuals and companies—particularly in distinctive segments of the mining industry. Each company has its own method of compiling a list of qualified contractors to be considered and included in the distribution of a request for proposal. Generally, these companies contact other mining companies that are or have been using contractors (industry recommendations) or rely on the internal resources of in-house personnel ("networking," the use of prior contact with individuals to determine appropriate contract miners for inclusion in the process). Business development personnel of contract miners can also provide a list of current and past experiences, along with client references.

The goal of this procedure is the same as for the formal procedure—to develop a list of qualified bidders. Although this method involves less time and money, it runs the risk of being influenced by the personal biases of the technical reviewers and/or decision makers.

In summary, once a mining company decides to employ a contract miner, it is desirable to select the most qualified contract miner. The compilation of a qualified bidder's list is one of the more important aspects of overall screening and selection of the contractor. Contract miners can provide valuable services; therefore, selection of the most qualified and reputable miner is an important decision to overall company performance.

6.4.3.2 Preparation of the Request for Proposal

In preparing the request for proposal (RFP), it is important to include as much pertinent information as possible. The topics that should be covered in a request for proposal are (1) scope of work, (2) site visit, (3) proposal due date, (4) proposal format, (5) contractual terms, (6) statement of qualifications, (7) acceptability of alternatives, (8) seasonal operating constraints, and (9) list of owner-furnished items.

The most important of this information is the scope of work. Without a complete scope of work, the job will not be bid accurately, and wide differences will exist between competing contractors. It then becomes very difficult to determine if this difference is due to a competitive edge or a misunderstanding of the work to be performed. If the winning contractor misunderstands the scope of work, then major problems often occur after the job commences that may force the contractor to renegotiate the agreement or possibly even default. The final result is delays in start-up, expensive cost overruns, and an adverse effect on the credibility of the contractor.

Scope of Work: A complete *scope of work* covers every aspect of the work expected of the contractor and any exclusions to be incorporated in the contractor's bid. Exclusions may comprise such items as operating crushing facilities and processing plants, mine planning, and construction of facilities.

The scope of work should include the following components:

1. *Expected Life of the Mine.* This is important to the contractor because it has great impact on the economics of almost all aspects pertaining to the contractor's bid, from equipment selection to unit prices.

2. *The Mining Schedule.* The mining schedule should include a timetable for major events during the life of the mine. These events may include initial construction completion dates, expected date of full-scale operation, significant shifts between mining areas, and periods of reduced or increased production levels. This timetable allows the contractor to determine when specific equipment is needed at the site and to anticipate manpower levels.

3. *Planned Mining Rates.* The owner's engineering staff should have a well-developed plan for the movement of ore and waste throughout the life of the mine to maximize cash flow and ore recovery rates. The life of the mine and the mining rate must be balanced to minimize operating costs and to avoid improper equipment selection. If the crushing or processing facilities have previously been selected, it is very important to include its capacities, along with any equipment size constraints that these may impose, such as maximum allowable size of the material fed to the crusher, feed rate of the crusher, etc.

4. *Detailed Mine Plans.* These plans should include all mining areas and associated phased production, as described in item 2. In addition, all construction required for site development, such as roads, shop and office locations, topsoil stockpiles, and impoundments that the contractor may be required to build, or must plan his work around, should be included. The physical layout of the mine, including bench heights, ramps and grades, stockpiles, waste dumps, crusher locations, leach pads or tailings ponds, and loadout facilities are of extreme importance to the contractor, if a competitive bid is to result. A well-engineered set of plans helps to alleviate any problems or questions that may arise about the scope of work. The thoroughness of these plans indicates to the contractor the level of mining competence that can be expected when the job commences. For example in underground mines, this section should make the contractor aware of final dimensions of all excavations, whether or not overbreak is acceptable, if ground support is required, if the excavation is to be a permanent installation with specifications or if it will be left to the contractor's discretion, depending on ground conditions. Details of timber or steel support structures must be clearly articulated. If significant amounts of water are anticipated, what facilities, both underground and on surface, will be required to handle the inflow?

5. *Miscellaneous Restrictions.* The scope of work must indicate any restrictions that may be imposed by the owner or as the result of specific geologic conditions. Typically, especially in precious-metal mines, the size of the blast patterns is restricted by the owner so that the drill cuttings can be used to determine ore grade and delineate ore zones. Environmental constraints, such as areas and methods for waste disposal or special material handling, should be carefully delineated. The abrasiveness and hardness of the material can greatly affect the type of equipment required, as well as the associated maintenance program.

For an underground mine, an accurate description is needed of the physical and administrative environment that will control the contractor's operation, underground construction, and mining operations' work in a restricted environment. These constraints, in part, are due to the physical limits placed on the excavations, but in a large sense they stem from the logistics of moving men, materials, and equipment in and out of the mine. The contractor must know if his operations are to be coordinated with other schedules and, if so, what are the rules that govern these schedules. The contractor also must be made aware of any unusual operating constraints that would affect performance. For example, do agreements with state or local authorities exist that restrict ground vibrations from blasting or other applicable restrictions?

Other Basic Information: In addition to the scope of work, there are a number of other basic items that should be included in the RFP. These items are just as important as the scope of work, although not nearly as comprehensive. Following is a list of some of the critical items that must be included, depending on site-specific circumstances.

1. *Invitation to Visit the Site.* Arrangements should be made for each potential bidder to make an on-site inspection. Proper coordination of this visit includes representatives of key personnel from the mine's management group in order to observe the contractor's team and assess the overall knowledge level of the team.

2. *Proposal Due Date.* A deadline for receiving all proposals from contractors must be specified and adhered to if subsequent problems are to be avoided.

3. *Proposal Format.* In order for the owner to properly evaluate and compare bids, there must be a stipulated format or form on which all contractors submit bids. This format should include specific tasks or functions for which the owner is requesting unit costs. In short, the bid form should include a method of pricing for every cost item described in the scope of work.

4. *Contractual Terms Required by Owner.* The owner may impose certain requirements or constraints on the contractor resulting from regulatory agencies, financial backers, or other agencies. Other terms may cover the recourse both parties have through errors and omissions in the bid, as well as arbitration procedures. Other items that may be included are proof of insurance, liability, clauses for termination or suspension, force majeure, clauses for agreement modifications, contractor's warranty, records and accounting, subcontracting, title to materials found, and agreement interpretation. Because of the important role these items play in maintaining the cooperation and working relationship between the owner and the contractor, fair allowances must be made for both parties in the agreement.

5. *Statement of Qualifications.* A statement of qualifications is required to assist the owner in selecting a competent contractor who can perform the job effectively. By allowing each bidder to state their qualifications and verifying information presented on past jobs, the owner can gain insight into how well each contractor performed on similar projects.

6. *Alternatives Developed by Bidding Contractors.* Some of the bidding contractors may develop alternative ideas concerning a portion of the scope of work that results in the same end product but may provide a competitive advantage to that contractor. The proposal should include a clause stipulating the amount of flexibility, if any, acceptable to the owner.

7. *Seasonal Operating Constraints.* Many jobs have seasonal constraints or are hampered by extreme weather conditions. Knowledge of these conditions is important to the contractor, since it directly affects equipment utilization, manpower fluctuations, and other operating procedures and costs.

8. *Owner-Furnished Utilities and Facilities.* If utilities are available at the site, then the local rates for these should be included, along with any utilities that may be furnished by the owner. If no utilities exist at the site, it is even more important to note this so the contractor can evaluate the needs and include appropriate provisions in the bid. This is also important for facilities, such as shops, offices, fuel and oil storage, water sources, and so on. Any plans for site locations of the contractor's facilities must be included in the proposal.

In summary, without a complete and comprehensive request for proposal, the owner will not receive accurate bids from contractors. When the RFP includes the type of information discussed, a congenial and cooperative atmosphere between the owner and successful bidder should prevail throughout the relationship.

6.4.3.3 Proposal Preparation by the Bidder

One of the most important considerations in preparing a proposal is often the most overlooked by bidders. The proposal must present information to the owner that outlines how the operator will meet the *needs of the owner.* The best starting point in preparing a proposal is to ascertain from the owner exactly what is expected from the successful bidder. This is accomplished through personal meetings, site visits, and a thorough review of the request for proposal. A well-prepared proposal outlines the human and financial resources, including experience, expertise, and equipment available to the contractor and how the contractor intends to mobilize these resources to the benefit of the owner.

The proposal should contain a clear description of the scope of services being offered. This may be accomplished in narrative and/or in reference to the request for proposal. A project plan should be offered describing specific components of the operator's proposed operations. This may include equipment lists, maintenance programs, safety programs, environmental safeguards, coordination with the owner, on-site support facilities, home-office support, and so on.

The proposal should also present an organization chart and the names of key individuals who will be assigned to the project, along with their resumes. Because each contractor may be bidding on several projects at the same time, it may be necessary to offer more than one person for each position. Listing more than one person for each position allows the owner to select his or her preference. It then becomes necessary during the pre-award meeting to commit specific people to the project.

Where appropriate, the contractor may include a project schedule indicating production rates and concurrence with the overall mine plan.

A good proposal also will contain a company resume giving background information and specific data on similar projects. Financial information, indicating that the company has the resources necessary to be responsive to the scope of the project, should be provided as well.

The commercial portion of the proposal should provide the unit price basis for compensation in the format provided in the request for proposal. Other information that should be included, if not required in the request for proposal, is the method of measuring work progress, schedule, procedure for invoicing, and the turnaround time for payment. The operator should delineate any special concerns or clarifications to requests made in the request for proposal.

The proposed contractor expects to perform a specific scope of work in a professional and profitable manner. Certainly the proposal is prepared with this in mind. The contractor must remember that the owner also has expectations for the planned venture. If the contractor prepares the proposal with consideration for these needs, the offering will receive a more favorable reception.

6.4.3.4 Evaluation of the Proposal

The most important aspect of the contract mining bidding process is the selection of a qualified and reliable contractor to perform the work. Evaluation of the proposal provides the owner an opportunity to select the best contractor available for the project. The mine owner, through the technical review team, must evaluate carefully the credentials of each bidder. Along with providing recommendations to management, the technical review team should narrow the field to two or three final candidates. The primary factors in the evaluation of contractors and the associated bids are (1) cost for the project (the price submitted by the bidders), (2) qualifications of the contractor and associated personnel, (3) availability of contractor equipment and personnel, (4) ability of the contractor to perform the required tasks, (5) contractual terms, and (6) owner/contractor relationship.

Cost alone should not be the deciding factor. A combination of all the foregoing factors, weighted by the importance of each factor to the owner, will yield the best selection. An evaluation and selection based on many factors is illustrated in 6.4.3.5.

The process is facilitated if the owner provides the contractor with an outline or format for the proposal to be submitted. The evaluation of all submitted proposals will be greatly simplified and the evaluation results more valid and consistent with the outline or format.

Cost: The contractor's bid "price" for performing the scope of work is the "cost" to the owner. The contractor's proposal is the entire document submitted, while the bid is the "price" or "cost" section of the proposal.

If the proposal is organized and properly detailed, the cost portion of the bid should be the easiest portion to analyze and evaluate. In order to compare the bids, unit costs must be converted to total dollars for the scope of work to be performed by the contractor on an annual or period basis. The owner also should evaluate the sensitivity of costs to changes in conditions. For example, the following questions need to be addressed. How will gold price fluctuations affect the owner's costs incurred? What would be the effect on profitability and continued operation of the mine if the price of gold rose to $600/oz ($19.29/g) or fell to $200/oz ($6.43/g)? Also, what would be the effect on the mine plan and the ratio of waste to ore tons if the price of gold rose to $600/oz ($19.29/g) or fell to $200/oz ($6.43/g)? What would be the effect if the ore control, based on blasthole drill samples, found more or less ore than planned? What would be the effect if processing the ore was more or less efficient, in terms of both processing costs and recoveries, than planned?

In evaluating the prices submitted by the contractor, two main methods of analysis generally are used: (1) total cost for the work provided by the contractor and (2) project discounted cash flow. The *total cost method* involves determining the total cost of the bid by multiplying the unit costs by the expected volumes and adding any lump sum bid items. The *discounted cash flow method* involves converting the costs into total dollars for each time period and discounting this stream of cash flows at an appropriate discount rate to determine the present value of the cost of the bid. The total cost method has the advantages of being simpler to analyze, appropriate for short contracts, and suitable for constant volumes/tonnages of material to be moved. The discounted cash flow method is more complex to perform; however, it has distinct advantages for longer-term contracts and is suitable for changing volumes/tonnages of material to be moved. When in doubt, the discounted cash flow method is recommended for all comparative analyses.

Three examples are provided to illustrate these two methods for bid comparison and evaluation. The first example (6.4.1) shows the effect of different lump sum mobilization costs with the appropriate changes in unit prices (cost) on the evaluation of bids. The second example (6.4.2) shows the effect of changing volumes over time on the evaluation of bids. The third example (6.4.3) illustrates the difference in single unit price bidding vs. unit prices as a function of production levels and years.

The analyses are simplified to enable easy analysis and explanation. The numbers utilized are selected to provide a contrast between the two methods and are not necessarily representative of actual prices for any specific set of mining conditions. The present value (present cost) determination is simplified in these analyses by not considering tax implications, assuming constant production rates throughout each year, assuming end-of-year cash flows for the annual costs, and by not considering the effect on working capital. A 15% discount rate is used to determine the present value (present cost).

Example 6.4.1. Mobilization Costs vs. Unit Price.

The bid prices received from contractor X and contractor Y are shown in Chart 1.

Chart 1

	Bids	
Bid Volumes	Contractor X	Contractor Y
Mobilization	$600,000	$200,000
Year 1 — 1,000,000 tons	$1.00/ton	$l.10/ton
Year 2 — 1,000,000 tons	$1.00/ton	$l.10/ton
Year 3 — 1,000,000 tons	$1.00/ton	$1.10/ton
Year 4 — 1,000,000 tons	$1.00/ton	$1.10/ton
Year 5 — 1,000,000 tons	$1.00/ton	$1.10/ton

Solution. Total Cost Analysis Method.

The bid unit prices are converted to total dollars per year and added to the mobilization cost to determine the total cost. (See Chart 2.)

Chart 2

Bid Volumes	Contractor X	Contractor Y
Mobilization	$ 600,000	$ 200,000
Year 1 — 1,000,000 tons	$1,000,000	$1,100,000
Year 2 — 1,000,000 tons	$1,000,000	$1,100,000
Year 3 — 1,000,000 tons	$1,000,000	$1,100,000
Year 4 — 1,000,000 tons	$1,000,000	$1,100,000
Year 5 — 1,000,000 tons	$1,000,000	$1,100,000
Total	$5,600,000	$5,700,000

The lowest cost bid utilizing this analysis is contractor X, with a total undiscounted cost difference of $100,000.

Solution. Discounted Cash Flow Analysis Method.

Using the annual cash flows developed previously, the present value of these costs is calculated. The discount factors illustrated in Chart 3 represent a 15% discount rate.

Chart 3

Year	Discount Factor	Contractor X	Contractor Y
−1	1.00000	$ 600,000	$ 200,000
1	0.869565	$ 869,565	$ 956,522
2	0.756144	$ 756,144	$ 831,758
3	0.657516	$ 657,516	$ 723,268
4	0.571753	$ 571,753	$ 628,928
5	0.497177	$ 497,177	$ 546,895
Total		$3,952,155	$3,887,371

The lowest cost bid by this analysis is contractor Y, with a cost difference of $64,784.

Note that the analysis method affected the results of the analysis and the preferred contractor based only on the bid cost. This example shows that the discounted cash flow method is clearly superior for comparisons of this type.

Example 6.4.2. Changing Volumes Over Time.

Bids have been received from contractor X and contractor Y as shown in Chart 4.

Chart 4

	Bids	
Bid Volumes	Contractor X	Contractor Y
Mobilization	$600,000	$200,000
Year 1 — 1,500,000 tons	$1.00/ton	$l.10/ton
Year 2 — 1,500,000 tons	$1.00/ton	$1.10/ton
Year 3 — 1,000,000 tons	$1.00/ton	$1.10/ton
Year 4 — 500,000 tons	$1.00/ton	$1.10/ton
Year 5 — 500,000 tons	$1.00/ton	$1.10/ton

Solution. Total Cost Analysis Method (Chart 5).

Chart 5

Bid Volumes	Contractor X	Contractor Y
Mobilization	$ 600,000	$ 200,000
Year 1 — 1,500,000 tons	$1,500,000	$1,650,000
Year 2 — 1,500,000 tons	$1,500,000	$1,650,000
Year 3 — 1,000,000 tons	$1,000,000	$1,100,000
Year 4 — 500,000 tons	$ 500,000	$ 550,000
Year 5 — 500,000 tons	$ 500,000	$ 550,000
Total	$5,600,000	$5,700,000

The lowest cost bid by this analysis is contractor X, and the total cost difference remains at $100,000.

Solution. Discounted Cash Flow Analysis Method.

Using the annual cash flows developed previously, the present value of these costs is calculated. The discount factors illustrated in Chart 6 represent a 15% discount rate.

Chart 6

Year	Discount Factor	Contractor X	Contractor Y
−1	1.000000	$ 600,000	$ 200,000
1	0.869565	$1,304,348	$1,434,782
2	0.756144	$1,134,216	$1,247,638
3	0.657516	$ 657,516	$ 723,268
4	0.571753	$ 285,876	$ 314,464
5	0.497177	$ 248,589	$ 273,447
Total		$4,230,545	$4,193,599

Lowest cost bid by this analysis is contractor Y, but the cost difference is only $36,946.

Note that the analysis method affected the results of the analysis and the preferred contractor based only on the bid cost.

Example 6.4.3. Single Unit Pricing vs. Variable Unit Pricing.

Bids have been received from contractor X and contractor Y as shown in Chart 7.

Chart 7

	Bids	
Bid Volumes	Contractor X	Contractor Y
Mobilization	$600,000	$600,000
Year 1 — 1,500,000 tons	$1.00/ton	$0.90/ton
Year 2 — 1,500,000 tons	$1.00/ton	$0.90/ton
Year 3 — 1,000,000 tons	$1.00/ton	$1.00/ton
Year 4 — 500,000 tons	$1.00/ton	$1.35/ton
Year 5 — 500,000 tons	$1.00/ton	$1.35/ton

Solution. Total Cost Analysis Method (Chart 8).

Chart 8

Bid Volumes	Contractor X	Contractor Y
Mobilization	$ 600,000	$ 600,000
Year 1 — 1,500,000 tons	$1,500,000	$1,350,000
Year 2 — 1,500,000 tons	$1,500,000	$1,350,000
Year 3 — 1,000,000 tons	$1,000,000	$1,000,000
Year 4 — 500,000 tons	$ 500,000	$ 675,000
Year 5 — 500,000 tons	$ 500,000	$ 675,000
Total	$5,600,000	$5,650,000

Lowest cost bid by this analysis is contractor X, and the total cost difference is $50,000.

Solution. Discounted Cash Flow Analysis Method.

Using the annual cash flows developed previously, the present value of these costs is calculated. The discount factors illustrated in Chart 9 represent a 15% discount rate.

Chart 9

Year	Discount Factor	Contractor X	Contractor Y
–1	1.000000	$ 600,000	$ 600,000
1	0.869565	$1,304,348	$1,173,913
2	0.756144	$1,134,216	$1,020,794
3	0.657516	$ 657,516	$ 657,516
4	0.571753	$ 285,876	$ 385,933
5	0.497177	$ 248,589	$ 335,594
Total		$4,230,545	$4,173,750

The lowest cost bid by this analysis is contractor Y, and the cost difference is $56,795.

Note that the analysis method affected the results of the analysis and the preferred contractor based only on the bid cost.

As illustrated by the foregoing examples, selection of the type of analysis influences the results and the selection of the preferred contractor based only on the bid cost evaluation. The type of analysis employed by the owner should be communicated to the contractor, because the contractor may structure a bid to enhance his competitive position and still meet their financial goals using the concept of equivalency resulting from time value of money concepts. It is necessary to reiterate the fact that cost is only one of several factors requiring consideration when evaluating proposals from contractors. A total evaluation and selection is illustrated in 6.4.3.5.

Qualifications: Qualifications of the contractor and associated personnel are important to project success. The owner should evaluate the experience of the contractor and staff in performing the type of project envisioned and similar projects completed. Important issues relate to questions, such as, does the contractor recognize the importance of ore control? How does ore control impact the viability of the project? Does the contractor have appropriate personnel on the project (e.g., a mine engineer and safety engineer) to perform the required tasks? Can the contractor mobilize equipment and personnel in an appropriate time frame to meet the owner's schedule?

During the technical evaluation of the proposals, a suitable rating system should be utilized to insure appropriate ranking of the various proposals and contractors. The rating system (see 6.4.3.5) must be developed by the owner to perform the tasks expected of the contractor. For example, if water control is a major concern, the availability or on-staff placement of a hydrologist should be reflected in the ratings.

If the owner does not desire to develop a complex rating system, the technical evaluation should at least rank the proposals in general terms for the qualifications of the contractor and associated personnel.

Equipment and Personnel Availability: Although the contractor must have the senior-level staff personnel and equipment necessary to perform the required tasks, it is usually preferable, for both the owner and the contractor, to hire a maximum number of local qualified personnel to ensure a stable work force. For the proposal, the contractor should provide a list of individuals to be considered for the project, with some positions listing more than one person if the contractor so desires. The listing of several candidates allows the owner to select from several individuals and, therefore, obtain the best-suited personnel for the project. Once a short list of two or three contractors is developed, a firm commitment can be solicited for the personnel desired on the project.

The equipment required to perform the scope of work can either be part of the contractor's existing fleet or purchased specifically for the project from vendors. The owner should inquire into the availability of major pieces of equipment from the contractor's fleet or availability from vendors. Along with equipment availability, the owner should check into the condition of the equipment to insure that it is capable of handling the work required. The performance of the contractor in providing equipment for previous projects also provides additional information for the evaluation of the proposal.

As with the contractor's qualifications, a rating system should be developed to rank the various candidates. The rating system and the weighting of each variable should reflect the importance given to the availability of personnel and equipment.

Ability to Perform: The ability of the contractor to perform the required tasks should be reviewed, including both financial and execution abilities. Financial ability can be determined by a review of the financial status of the contractor. The owner must establish if the contractor has the financial strength to purchase the required equipment and to provide necessary working capital to perform the work specified. An analysis of the annual report, plus other financial reports requested in the RFP along with research on the level of performance on previous contracts, should provide the information necessary for the evaluation.

The contractor's ability to execute the required tasks must be assessed by the owner. Can the contractor hire, train, and maintain a stable and qualified work force? The factors necessary to develop a rating system in this critical area are obtained primarily from previous work on other similar projects.

As stated previously, a rating system should be developed to rank the various candidates. The rating system and the weighting of this area should reflect the importance given to the contractor's ability to perform.

Contractual Terms: The contractual terms of the agreement must be acceptable to both the owner and the contractor. Each party to the agreement should review the concerns of the other to determine the relative importance of the conditions requested. Although the owner usually rates this area either "acceptable" or "not acceptable," the owner may wish to rank the concerns expressed by the contractor in relation to their importance. If the variances requested by the contractor are not acceptable, the owner should contact the contractor to determine the reason and the importance of the variance and the acceptability of alternative wording or conditions.

Owner/Contractor Relationship: The most important aspect of the evaluation is the owner/contractor relationship. An adversarial relationship creates additional work for both parties and probably impacts the overall cost and profitability of the project. The owner should check the relationship the contractor has or had with other clients on several current and/or previous projects. It must be remembered that the owner/contractor relationship is a two-way street. As stated for the other factors, the owner should develop a rating system on owner/contractor relationship for ranking the proposals.

6.4.3.5 Selecting the Contract Miner

Selecting the contract miner for a project should be relatively simple if the steps for developing the bid list and preparing the request for proposals previously discussed are followed completely. It is very important for the selection process to incorporate analysis of all parameters and not cost only.

Selection Criteria: Before the bids are received, the owner should develop a selection system based on criteria determined to be reasonable for the organization and the job to be performed. The selection system should be based on the criteria established and the weighting of those criteria. A sample system is described in the following paragraphs.

A sample selection weighting table is shown in Example 6.4.4 that follows. The factors were taken from the evaluation criteria discussed earlier in the chapter. After the factors have been established, ranking of the factors should be determined. Cost, although not the only factor, is usually ranked as the most important factor. The ability to perform the job was determined to be the second most important factor because the owner wants to be assured that the desired work is performed satisfactorily. The third factor is the qualifications of the contractor and his personnel, followed by the owner/contractor relations. The contractual terms factor was determined to be fifth in importance, followed by the availability of equipment and personnel. The owner may have other factors to add to the list or he may put more or less importance on each. However, this system should be developed prior to receipt of proposals so as to remove any bias that may develop after the bids are received. A sample set of factors and weighting is shown in Chart 10.

Example 6.4.4. Contractor Selection.

Chart 10. Weighting Table

Factors	Ranking		Weighting
Cost	1		40%
Qualifications — Contractor		9%	
— Personnel		9%	
Total	3		18%
Availability — Equipment		2%	
— Personnel		2%	
Total	6		4%
Ability to perform	2		20%
Contractual terms	5		5%
Owner/Contractor relationship	4		13%
Total			100%

Solution. A simplified sample evaluation is provided based on a fictitious request for proposal. The RFP was for a precious-metal mine in Nevada with production requirements as shown in Chart 11.

Chart 11. Production

Year	Tons
1	1,000,000
2	2,000,000
3	2,000,000
4	1,500,000
5	1,000,000
Total	7,500,000

Five bids were received with the pricing as shown in Chart 12.

Chart 12. Prices ($/ton)

Year	Contractors # 1	# 2	# 3	# 4	# 5
Mob	$800,000	$400,000	$200,000	$400,000	$300,000
1	$1.50/ton	$1.25/ton	$1.60/ton	$1.70/ton	$1.40/ton
2	$1.00	$1.25	$1.15	$1.20	$1.35
3	$1.00	$1.25	$1.15	$1.20	$1.35
4	$1.20	$1.25	$1.30	$1.30	$1.30
5	$1.40	$1.25	$1.50	$1.40	$1.30
Demob	$600,000	$200,000	$200,000	$400,000	$300,000

The total annual costs are shown in Chart 13.

Chart 13. Annual Costs ($)

Year	Contractors # 1	# 2	# 3	# 4	# 5
—1	$ 800,000	$ 400,000	$ 200,000	$ 400,000	$ 300,000
1	$ 1,500,000	$1,250,000	$ 1,600,000	$ 1,700,000	$ 1,400,000
2	$ 2,000,000	$2,500,000	$ 2,300,000	$ 2,400,000	$ 2,700,000
3	$ 2,000,000	$2,500,000	$ 2,300,000	$ 2,400,000	$ 2,700,000
4	$ 1,800,000	$1,875,000	$ 1,950,000	$ 1,950,000	$ 1,950,000
5	$ 1,400,000	$1,250,000	$ 1,500,000	$ 1,400,000	$ 1,300,000
6	$ 600,000	$ 200,000	$ 200,000	$ 400,000	$ 300,000
Total	$10,100,000	$9,975,000	$10,050,000	$10,650,000	$10,650,000

The proper evaluation technique for a project of this duration is a discounted cash flow analysis. The present value determination was simplified by not considering tax implications, assuming constant production rates throughout each year, assuming end-of-year cash flows, and not considering the effect on working capital. Using a 15% discount rate, the present value of the cash flows was calculated and is shown in Chart 14.

Chart 14. Present Value of Cash Flows

Year	Contractors # 1	# 2	# 3	# 4	# 5
—1	$ 800,000	$ 400,000	$ 200,000	$ 400,000	$ 300,000
1	$ 1,304,348	$1,086,956	$ 1,391,304	$ 1,478,261	$ 1,217,391
2	$ 1,512,288	$1,890,360	$ 1,739,131	$ 1,814,746	$ 2,041,589
3	$ 1,315,032	$1,643,790	$ 1,512,287	$ 1,578,038	$ 1,775,293
4	$ 1,029,155	$1,072,037	$ 1,114,918	$ 1,114,918	$ 1,114,918
5	$ 696,048	$ 621,471	$ 745,766	$ 696,048	$ 646,330
6	$ 259,397	$ 86,466	$ 86,466	$ 172,931	$ 129,698
Total	$ 6,916,268	$6,801,080	$ 6,789,872	$ 7,254,942	$ 7,255,219

These figures illustrate that the lowest bid in total dollars is not necessarily the lowest cost to the owner. In this case, the lowest present value is actually the second lowest bid.

The information submitted by the contractors (except cost), based on the criteria listed above in the weighting table, is summarized as follows.

Qualifications

Contractor

#1 A large and experienced contractor miner with 10 operating mines.

#2 A highway contractor with no mining experience but has many years in highway construction.

#3 A highway contractor with one small operating mine and considerable experience in highway construction.

#4 A small contract mining company with 2 small operating mines.

#5 A combination highway contractor and contract miner with operating contract mines.

Personnel

#1 The project manager is a mining engineer with 20 years experience, all in metal mines. The foremen have 5 to 10 years experience, mostly in mines. The engineer is a mining engineer with five years experience, all in metal mines.

#2 The project manager is nondegreed with 20 years experience in highway construction, but none in mining. The foremen have 5 to 10 years experience in highway construction, but none in mining. The contractor is not planning to have an engineer on the job.

#3 The project manager is a civil engineer with 15 years experience of which 2 years were in a metal mine. The foremen have 5 to 10 years experience with half of the foremen with two years in a metal mine. The contractor proposes to use a civil engineer from the home office on a part-time basis.

#4 The project manager is a mining engineer with 10 years experience, half in metal mines and half in coal mines. The foremen have an average of five years of experience with about half of them experienced in metal mines. The engineer is a mining engineer with five years experience in metal mines.

#5 The project manager is a civil engineer with 20 years experience, half in metal mines and half in highway construction. The foremen have 5 to 15 years experience with half of the foremen having half of their experience in metal mining. The contractor proposes to use a mining engineer from the home office on a part-time basis.

Availability

Equipment

#1 The major pieces of equipment will come from existing mines with 10% being purchased new from equipment companies. The used equipment is two to three years old and in reasonably good shape.

#2 Approximately 75% of the equipment will be purchased new with 25% of the equipment purchased used, and only a minor amount will be obtained from the existing highway construction fleet. The new equipment is available from the vendors, currently under conditionary order, and in the vendor's inventory.

#3 Approximately 30% of the equipment will be purchased new, 50% of the equipment will be purchased used, and the remaining 20% will be obtained from the existing mine operation or from the existing highway construction fleet. The exact equipment for purchase has not been identified.

#4 Approximately 50% of the equipment will come from existing mines, 25% will be purchased new, and 25% will be purchased used. Existing equipment is in good condition. The used equipment has not been located. The new equipment is under conditionary order and in stock.

#5 Most of the equipment will come from existing mines, 10% will be purchased used, and 10% will be purchased new. Existing equipment is in fair to poor condition.

Personnel

#1 All personnel are available from existing mines, and they have provided two to three alternative persons for the significant positions.

#2 Most of the personnel are available from existing construction jobs, with only minor positions hired for this project.

#3 Most of the personnel are available from existing mines, but half of the foremen are hired from outside the company.

#4 Over half the personnel are available from existing mines, but half of the foremen and the mine engineer are hired from outside the company.

#5 Over half the personnel will be hired from outside the company, including the mine manager and the mine engineer.

Ability to Perform

#1 The contractor has proven his ability to perform from existing mining operations and has developed well-run operations in difficult situations.

#2 Although the contractor does not have experience in mining, he has proven his ability to perform well on highway construction work in difficult situations.

3 The contractor has limited experience in mining and has had difficulty in satisfactorily meeting performance standards of the project for both the mine and some of his construction projects.

#4 The contractor has limited experience in mining but has performed acceptably in the two mines he is currently operating.

#5 The contractor has had some difficulty in one of the mines he is currently operating but has performed well in the other mines and has an excellent reputation as a highway contractor.

Contractual Terms

#1 The contractor took exception to many terms and conditions of the draft contract transmitted with the RFP that the owner believes to be important. The contractor has expressed a desire to negotiate with the owner.

#2 The contractor accepted the draft contract as transmitted with no exception.

#3 The contractor took only one exception to the draft contract; it was a minor clause that the owner has no objection to changing.

#4 The contractor has expressed a desire to discuss several of the major terms and conditions.

#5 The contractor took exception to a few of the major terms and conditions and many of the minor terms.

Owner/Client Relationship

The owner has contacted all of the clients of the contractors with mining operations and most of the clients of the contractors with highway construction work. A list of questions was compiled to insure that all clients were asked the same questions. Also the owner provided space for general comments on the relationship with the clients.

#1 All clients made favorable comments about this contractor. They commended the contractor on his working relationship with them to make a better project for both the contractor and the client.

#2 All clients made favorable comments about this contractor.

#3 Most of the highway construction clients made acceptable comments about the contractor but indicated that some problems had occurred. The client for the mining project expressed some concern about the contractor's desire to work with the client.

#4 The clients had generally favorable comments to make about the contractor, but they also stated that no serious problems had occurred to test the working relationship.

#5 Most of the clients, both from the highway work and the mines, had good comments, but one of the mining clients and one of the highway clients had problems in dealing with the contractor.

With the information describing the proposals that were received for this project, the owner now ranks each of the factors for each of the contractors, as shown in Chart 15. The ranking for each factor has a maximum of 10 points. Receipt of 10 points is a perfect score for that particular factor, or is the lowest bid in the case of costs. The procedure used for quantifying the cost factor in the system devised is 10 points minus the percentage difference in price. For example,

Lowest bid (present worth) = \$6,789,872

Bid being evaluated = \$6,916,268

$$\text{Percentage difference} = \frac{\$6,916,268 - \$6,789,872}{\$6,789,872} \times 100$$

$$= 1.862\%$$

Ranking = 10 – 1.862 = 8.138

Chart 15. Table of Ranking

Factors	Contractors 1	2	3	4	5
Cost	8.138	9.835	10.000	3.151	3.588
Qualifications — Contractor	10	5	7	7	9
— Personnel	9	5	7	9	8
Availability — Equipment	8	9	5	7	3
— Personnel	10	9	7	5	2
Ability to perform	10	8	3	9	6
Contractual terms	2	10	9	7	5
Owner/Contractor relationship	10	9	6	8	3

Now that the factors have been ranked for each of the contractors, the table for determining the selection can be made. The selection table (Chart 16) uses the ranking for each contractor for each factor and the weighting of each of the factors to quantify each factor for each contractor in relation to the other contractors. The selection table that follows quantifies the example given:

Chart 16. Selection Table (Ranking times Weighting)

Factors	Contractors				
Cost	3.255	3.934	4.000	1.260	1.435
Qualifications — Contractor	0.900	0.450	0.630	0.630	0.810
— Personnel	0.810	0.450	0.630	0.810	0.720
Availability — Equipment	0.160	0.180	0.100	0.140	0.060
— Personnel	0.200	0.180	0.140	0.100	0.040
Ability to perform	2.000	1.600	0.600	1.800	1.200
Contractual terms	0.100	0.500	0.450	0.350	0.250
Owner/Contractor relationship	1.300	1.170	0.780	1.040	0.390
Total	8.725	8.464	7.330	6.130	4.905

Using the system described, the successful contractor is #1, even though he did not have the lowest cost bid. The lowest bid submitted actually ranked third in the final analysis. This system states that factors such as "ability to perform," "qualifications," and "owner/client relationship" are important in the selection of a mining contractor. This system helps to ensure that the mine will be successful in the mining of the ore, the removal of the waste, and achieving the lowest total cost for the product being extracted.

6.4.3.6 Writing the Agreement

Two points should be considered as the owner and contract miner negotiate an agreement. First, the primary objective of each company is wealth maximization, as discussed in the Introduction, 6.0. The best agreement is one where this goal can be achieved by both parties, and the best project is one where all parties benefit. As soon as either party to the agreement begins to lose money or feels that a *fair* rate of return is not being achieved, the entire relationship typically deteriorates to the detriment of the project. Presumably, a negotiated agreement represents an agreement perceived to be equitable and fair to both parties. Events occurring after the agreement is negotiated may alter this situation, although this alone is rarely a reason to renegotiate unless both parties agree to do so. The contract miner should not look at the profits of the owner and feel he deserves a greater share. The proposal was made on the basis of a reasonable return, and the contract miner should be satisfied with that. At the same time, the owner should not begrudge profits of the contract miner but realize these are the cost of doing business. This relationship can be complex; it also must be based on both parties being reasonable and acknowledging the need for adjustments as a result of changes encountered during performance of the agreement.

Second, the contract miner is usually a construction company, and historically, performance is measured by increased production. The client, or owner in this case, generally has a stipulated production schedule and may not be interested in increasing the materials-handling portion of the operation beyond the capacity of the downstream recovery or processing facilities. The primary considerations usually are for constant production with much care given to the segregation of ore from waste. This requires a philosophical adjustment for the contract mining company—one that is best learned through experience. The owner should ensure that this requirement is understood and defined clearly in the agreement.

Preparing a good agreement sets the stage for a good project. The documents should be as detailed as possible in outlining the specific responsibilities of both parties. The parties to the agreement should be identified, along with any contractual obligations of parent companies or other third parties. A detailed scope of work is critical to obtaining desired performance in all areas covered by or affected by the contract miner. Specific responsibilities should be delineated based on the mine plan, the subsequent request for proposals, and the final bid documents. Quantities and production rates must be specified, and limits to variations on these figures must be established. (This is required because the contract miner based his proposal on crews, equipment, and support required to operate efficiently at a given production rate. Decreasing production by one-half does not decrease costs by an equal amount.) The specifications used during the bidding phase should be made part of the agreement. These specifications normally include such items as mine drawings, mine design, and operating criteria. The contract miner's proposal also should be made part of the agreement. In addition, any qualifications, assumptions, or clarifications made by the contract miner in developing the bid should be included.

Another important item that should be included in the agreement is the procedures between the owner and contract miner for coordinating identification and segregation of ore and waste. Responsibility for support activities, such as surveying, sampling, laboratory analysis, etc., also should be specified. Typically, samples are taken during drilling, and analysis is completed with sufficient lead time such that the contract miner is able to keep equipment and crews fully occupied. The plan and agreement should allow enough flexibility of operation so the project will not become ore-bound or waste-bound. The owner should realize that if this limitation is not avoided, the contract miner will incur expenses for equipment downtime.

A good agreement also has provisions for the contract miner to be compensated for work that becomes necessary but is not outlined in the specific responsibilities section of the agreement. This usually takes the form of hourly or daily rates for personnel and equipment, sometimes called "Force Account" rates. With such a provision, the owner is provided flexibility in calling on the contract miner to perform small, nonproductive activities such as access roads, snow removal, pad construction, etc.

The agreement should also list the responsible party for utility costs and installation, along with any limitations that may exist.

The financial or payment section of the agreement can be quoted as unit price, fixed price, or cost-plus-a-fee. The unit price contract is the type of agreement used most often for contract mining. The agreement has a unit price per weight or volume for the production plus, typically, lump sum payments for items such as mobilization, demobilization, etc. The fixed price contract is not generally used in contract mining. Its pricing is a lump sum for the work to be performed. The cost-plus-a-fee type agreement is not typically used in contract mining. It reimburses the contractor for his costs plus a fee established by the agreement.

Where the agreement duration is over one year, provisions should be made for escalation and de-escalation based on a weighted formula using various price indexes (labor, fuel, explosives, etc.). The premise is that as prices escalate, the market price of the salable product will move in the same direction, and the parties will maintain equality in the agreement. Without such a provision, the owner is asking the contract miner to assume all risks resulting from inflation—a risk he generally is hesitant to take. If provisions for escalation are not included in the agreement, the contract miner would include a contingency in the price and that contingency would probably be more expensive than having the owner assume the risk. In most contract mining agreements, the owner accepts and bears the risk of inflation. For a more detailed discussion, read the appropriate section in Gentry and O'Neil (1984).

The financial or payment section also should stipulate the method for measuring work performed, price schedule, invoicing and payment timing requirements, and other pertinent information. Delineation of the pay items should stipulate the responsible party for measurement, method of measurement, and any verification documents required.

Other primary sections must be included in the agreement to address mutually acceptable terms concerning changed conditions, premium and penalty, liability and indemnification, disputes, delays (force majeure provisions), suspensions, terminations, and default. These sections often require the greatest amount of discussion to reach a mutually agreeable agreement. Premiums and penalties should be used together so that the contractor is not only motivated to avoid penalties, but also to obtain premiums. The areas for premiums and penalties, if used, require accurate and readily definable terms and conditions.

Additional contract sections should include methods for modifying the agreement, the assignability of the agreement, and methods of notification and communication between the parties.

The agreement should include as much written detail and definitions as practical. It should anticipate and provide for solutions to unanticipated conditions that may occur at a later date. Provisions should be flexible enough to accommodate actual field conditions. If this approach is used in writing the agreement, it will be to the mutual benefit of all parties, and disagreements during the performance of the agreement should be minimized.

6.4.3.7 Typical Agreement Headings

Table 6.4.1 lists the headings of topics typically addressed in an agreement. Every agreement is different, because one agreement can not cover all the specifics of each different type of mine operations, property characteristics, commodities, contract miner attributes, and owner traits.

6.4.4 MONITORING THE CONTRACT MINER

Monitoring the contract miner's performance is important to ensure that the contract miner is performing the work desired by the owner to make the project a profitable venture. Monitoring the various risk factors discussed subsequently greatly assist in assuring profitability. Mining always has been and continues to be a complex and risky business. It is rare to have an initial mine design and plan remain constant through the start-up and production of an operation without experiencing at least some major changes.

These changes may be aligned with metallurgical or materials handling characteristics of the ore body and could dictate many changes to the operation. The results of these changes could include revised mine plans; changed mining practices, including ore control; and modified contractual terms, including mining volumes, prices, and equipment. The changes could even dictate closure or sale of the project before, during, or after development.

The owner's risk in developing a new project can be affected by many factors, such as overall mining and recovery methodology; recovery characteristics of the ore; front-end capital investments for exploration, acquisition, and development; ore tonnage and grade; metal sales; marketing strategy; and variations due to external factors. Monitoring the contract miner lessens the owner's risk by competitive bidding and selection, grade control, mining practice, mine design, contract terms, pricing, overall performance, visibility, working relationships, equipment, and productivity.

The owner can lessen overall project risk through the selection of a competent contract miner. Selection of the wrong contractor will lead to increased management requirements on the part of the owner, as well as increased costs due to poor overall performance.

The secret to making a contract mining situation work becomes one of mutual understanding and trust of each party's problems. Therefore, the concept of working together toward a common goal should be the first objective of both parties.

6.4.4.1 Owner/Contract Miner Interface

The contract miner and owner interface at many places in the mine operation. The contract miner must attempt to use the equipment to ensure optimum productivity at minimum cost. Grade control procedures are normally the owner's responsibility to design and implement. However, if ore is hauled to the waste dump and waste is hauled to the crusher, both parties loose. The owner must design the pit, haul roads, and dumps to effect low-cost production and should continually work with the contractor to improve designs. The contractor also should be formulating better and improved ways to perform his work. An effective design must be produced and implemented to control the project. Contract terms must be monitored on a continual basis to ensure factual record keeping and to ensure that engineering information is being produced and monitored. Payment and material production schedules must be agreed to on a regular basis.

The overall performance of the contractor is very important in terms of perception by outsiders. If the mining contractor has continual difficulty meeting the agreed upon goals, the end result will not be achieved and the owner will incur economic loss, and credibility problems will result. The owner must monitor the visual appearance of the premises and mode of operation conducted by the contractor. Sometimes the wrong picture will negatively impact future opportunities for both parties. The working relationship and openness of the owner/contractor can impact each party. As an example, a wage increase given by the contractor could impact the owner in other areas of the mine. These items affect overall labor relations and need to be discussed and planned in advance to lessen negative impacts. As the contractor works on the owner's property, the protection of property should be considered with respect to contamination and damage through misuse. The selection of equipment by the contractor should be discussed with the owner to avoid a mismatch and offset any potential productivity problems. Environmental issues must be discussed by both parties and impacts to performance levels and expectations addressed immediately. Impacts on towns and communities should be a concern of both owner and contractor with regard to quality of work force, services, etc.

Table 6.4.1. Typical Agreement Headings

- Recitals
- Definitions
  - Aerial volume
  - Agglomerate
  - Agreement
  - Contract documents
  - Contract miner
  - Contract miner's representative
  - Contract payment(s)
  - Crushing plant
  - Cubic yard(s)
  - Disposal sites
  - Drawings and specifications
  - Fixed facilities
  - Force account
  - Heap construction
  - Labor rate
  - Leach pad loading
  - Lump sum
  - Major slope failure
  - Mine plan
  - Mining
  - Mobile equipment
  - Ore
  - Ore rehandling
  - Ore stockpile(s)
  - Ore stockpile area(s)
  - Overburden
  - Owner
  - Owner's representative
  - Penalty
  - Pit
  - Plant
  - Premium
  - Processing facility
  - Project
  - Properties
  - Stripping
  - Subcontractor
  - Ton
  - Unit price(s)
  - Waste
  - Work
- Project Terms and Conditions
  - Effective date
  - Commencement of work
  - Work execution schedule
  - Stripping and mining schedule
  - Coordination of activities
  - Use of drawings and specifications
  - Modifications
  - Corrections at contract miner's expense
  - Disposal sites
  - Pit drainage
  - Heap construction
  - Fixed facilities sites
  - Survey control
  - Roadways
  - Pit limits
  - Construction of pit walls and benches
  - Deviations from owner's design and specifications
  - Option to purchase equipment
- Representations and Warranties of Owner
  - Authority to execute and deliver contract
  - Operating rights
  - Owners data
- Representations and Warranties of Contract Miner
  - Authority to execute and deliver contract
  - Examination of site
  - Qualifications
  - Owner's data
- Responsibilities of Contract Miner
  - Unit price work
  - Force account work
  - Mobilization and demobilization
  - Haul road construction
  - Topsoil removal, storage, and replacement
  - Lump sum work
  - Maintenance service for owner's equipment
  - Other work
  - Subcontracts
  - Compliance with laws of general applicability
  - Nondiscrimination and equal employment
  - Compliance with laws regarding labor
  - Solid wastes and hazardous or toxic wastes or substances
  - Cleaning up of premises
  - Inspection of work
  - Insurance
  - Payment of contract miner's obligations
  - Indemnity
  - Information on subcontractors
  - Owner's occupancy of pit area
  - Local labor and materials
  - Contract miner's representative
  - Standard of care
- Contract Payment by Work Items
  - Compensation
  - Stripping and mining
  - Description
  - Measurement
  - Disagreement regarding measurement
  - Payment
  - Payment schedule
  - Disagreement regarding invoices
  - Contract payment adjustment
  - Sample calculations and payment adjustments
  - Calculations, stripping and mining
  - Calculations, pad loading
  - Calculations, weightometer recalibration
  - Mining and stripping, aerial survey
  - Contract miner's price adjustment (escalation/de-escalation)
  - Labor adjustment (escalation/de-escalation)
  - Material and supply adjustment (escalation/de-escalation)
  - Premium/penalty
  - Calculation, premium and penalty
- Responsibilities of Owner
  - Work
  - Owner provided supplies
  - Permits
  - Ore and mining
  - Water control
  - Obligation for pit design
  - Liability for faulty pit design
  - Surveys
  - Volume determination
  - Owner's representative
  - Final reclamation
  - Additional owner obligation
  - Fixed facilities
- Miscellaneous
  - Default by owner
  - Default by contract miner
  - Force majeure
  - Extension
  - Suspension
  - Termination for cause
  - Termination for convenience
  - Performance and payment bond
- General Contract Provisions
  - Identification and integration of contract documents
  - Deviation from design and specifications
  - Relationship of the parties
  - Assignability
  - Notices
  - Accounting records and audits
  - Governing laws
  - Article headings
  - Time is of the essence
  - Waiver
  - Confidentiality
  - News release
  - Mutual cooperation
  - Complete agreement, modification
  - Multiple counterparts
  - Severability
  - Disputes
  - Survival of representations, warranties, and covenants
- Exhibits
  - Detailed scope of work
  - Drawings and specifications
  - Stripping and mining schedule
  - Insurance
  - Price schedule
  - Price escalation/de-escalation
  - Force account provisions

Both parties must be good citizens. Human welfare and safety are a common concern.

6.4.4.2 Formal Methods of Implementation

Formal methods of implementation should include regularly scheduled production meetings; material reports on daily, weekly, and monthly basis; monthly billing and audit sequence; quarterly operations review; and ongoing communication in the working environment.

6.4.5 MINE LEASING PROCESS

The purpose of leasing a mining property to a mining company or other entity is to transfer some or all of the rights to explore, develop, and operate the property to a company in exchange for certain considerations such as cash, royalty provisions, or other type of reward. This section discusses in general the mine leasing process and the major items for consideration in leasing a property. The process is discussed in the following order: (1) property evaluation, (2) property rights, (3) objectives and goals, (4) marketing the property, (5) negotiating the lease, and (6) an outline of a mining lease. In this section, the *lessor* is referred to as the person or entity who owns or controls the property and is desiring to lease the property to another person or entity. The *lessee* is referred to as the person or entity who is obtaining operating and ownership rights to a property through the lease.

6.4.5.1 Property Evaluation

An accurate and objective evaluation of the property is essential in the leasing process. Without appropriate knowledge of the value of the property, it is difficult to determine the objectives and goals for leasing the property, marketing the property, or negotiating a lease. Other chapters in this section (e.g., 6.1 and 6.2) go into details of property evaluation.

6.4.5.2 Property Rights

When leasing a mining property, it is important to specify the property rights that are being sold or transferred. The main types of property rights are (1) fee land, (2) unpatented mining claims, (3) patented mining claims, and (4) leased land from a private landowner or governmental entity. A title examination is important to determine who has title to the property and what rights they have to the property. Other chapters in this *Handbook* (e.g., 3.2 Mining Law) go into details of title examination. Property rights may be restricted to one or some or all of the minerals, to surface or underground mining rights, or restricted surface rights with the mineral rights, or total surface and mineral rights. The end product should be an accurate title report of the property to be leased and the rights pertaining thereto.

6.4.5.3 Goals and Objectives

The goals and objectives of the lessor are critical to the leasing process. The goals of the leasing process should be developed in detail and thoroughly defined so that the lessor can direct the process to achieve those goals. The lease must contain the necessary incentives and provisions to ensure that lessor goals and objectives are accomplished.

6.4.5.4 Marketing the Property

Before the lessor starts to market the property, he must determine to whom he wants to market the property and how much is he willing to spend on the marketing effort. The major types of entities who may lease a mineral property are established mining companies, new or recently established mining companies, organizations desiring to become mining companies, and promotional companies.

The major ways to market a property are to (1) advertise it, (2) list the property with a real estate broker, (3) list the property with a listing agency, and (4) direct contact with mining companies or venture capital firms who promote mining properties.

The places to advertise a property are (1) in local and/or regional newspapers, (2) in national newspapers, (3) in trade publications, and (4) at trade shows and conferences. The latter two places listed provide better direction to potential lessees.

6.4.5.5 Negotiating a Lease

In negotiating a lease, it is important to remember that the lease must be fair to all parties involved in the lease or problems will occur throughout the life of the lease. The "real" objectives of the lessor and the lessee must be understood at the start of negotiations. The lessor must determine if production or simply income is the primary objective. The lease must be site-specific and product-specific because general leases do not and cannot contain all the terms and conditions appropriate for all or even most situations. During negotiations, it is necessary to recognize that nothing is binding until a letter of understanding (or letter of intent) or a contract is signed.

Listed are the subject areas normally covered in lease negotiations. (Zierold, 1989) A brief discussion of each item follows this listing:

Lease term	Attorney's fees
Payments to lessor	Arbitration
Work commitments	Force majeure
Operator rights	Notices
Indemnifications	Assignment of rights
Responsibility for taxes	Lessee rights
Default and forfeiture	Assessment work
Recordation of lease	

Lease Terms: The term of the lease must be determined, specified as primary term and secondary term. They should be defined as specific dates, not as "per anniversary date" or other vaguely defined nomenclature. Also the terms and conditions for renewal of the lease should be negotiated. It is important to identify the term for lease renewals and, if on production only, are periods of nonproduction allowed and for what cause and length of nonproduction.

Payments to Lessor: Payments to the lessor are generally divided into three categories: (1) signature bonus or payment, (2) periodic payments, and (3) production royalties. The signature bonus is a payment or commitment to pay at the time the contract is signed. This payment can be in terms of cash or stock or other remuneration. The signature bonus as well as the periodic payments represent compensation for loss of possession. The signature bonus should cover all or most of the costs of the lessor in marketing the property and negotiating a lease.

The periodic payments can be rent, minimum royalties, minimum advance royalties, or other types of payments. These payments also represent compensation for loss of possession of the property. Rent payments are only for possession of the property. Royalty payments are payments for production of the property. Minimum royalty sets a minimum payment per period of time that must be paid, but the payment could be higher depending upon the quantity of production. Minimum advance royalties set minimum payments per period of time, but any payments over the payment required by the amount of production are a credit to the producer and recoupable when production payments exceed the minimum royalties. The minimum or advance minimum royalties can be established in terms of dollars per period of time, or set on a scale that changes as time passes, or they can be escalated bases on certain index(es). The escalation can be based on the price of the mineral or a governmental index (e.g., consumer price index, producers price index, etc.).

Production royalties vary considerably with commodity and location. On commodity, for example, a high-grade gold deposit has a much higher value than a gravel deposit. Also a gravel deposit near downtown New York City has a much higher value than a gravel deposit in central Nevada. The basis of the royalty can be established on dollars per weight or volume (generally for bulk materials like gravel), on percentage of gross value of the product produced (generally for more valuable commodities like metals), on the percentage of net operating profit of the operating venture, or a combination of these bases. For the bases

Table 6.4.2. Typical Table of Contents, Lease Agreement

- Recitals
- Definitions
  - Assessment year
  - Calendar quarter
  - Continuous production
  - Day
  - Lease year
  - Leased substances
  - Leased property
  - Lessee
  - Lessor
  - Mill
  - Mining operation
  - Net smelter return
  - Ore
  - Section
  - Tailings material
  - Waste material
- Term
- Nonexclusive Possession
- Title
  - Representations and warranties
  - Title documents; data
  - Title defects, defense, and protection
  - Lessee's interest in perfecting title
  - Lessor interest provisions
  - Payments in the event of title litigation
  - Change in law
  - General
- Duty to Explore, Develop, and Mine
- Payments to Lessor
  - Initial payment upon signing
  - Minimum advance royalty
  - Production royalty
  - Fractional interest
  - Adjustment in minimum "continuous production" royalty payments
- Option to Purchase; Overriding Royalty
- Operations; Authorized Activity of Lessee
  - Authorized activity
  - Conduct of operations
- Assessment Work
  - Performance of assessment work
  - Recordation of assessment work with county recorder
  - Recordation of assessment work with Bureau of Land Management
  - Expense of recordation of assessment work
- Patent to Mining Claims
- Commingling
- Timber
- Liability, Insurance and Indemnification
  - Protection from liens
  - Insurance
  - Indemnification
  - Damage to lessor's property
- Taxes
- Inspection
- Default and Forfeiture
- Termination
  - Termination by lessor
  - Termination by lessee
  - Obligations of lessor upon termination
  - Relinquishment of record
  - Enforcement action after termination
- Additional and After-Acquired Rights
- Removal of Property
- Ownership of Tailings Materials
- Data and Regular Reports
- Force Majeure
- Arbitration
- Notices; Addresses of Parties
- Memorandum of Mining Property Lease; Recording
- Regulatory Approval
- Amendment
- Counterparts
- Effect
- Assignment
- Designation of Sole Representative
- Applicable Law; Forum for Litigation
- Entire Agreement
- Miscellaneous Provisions
  - Titles
  - Promotion
  - Unit of currency
- Further Actions
- Signatures and Acknowledgments
- Exhibit "A," List of Unpatented Mining Claims

Source: Zierold, 1989.

stated previously, it is very important to define all terms on which the payment is to be made.

The dollars per weight royalty and the percentage of gross revenue have lower risks for the lessor, who in turn has less potential to achieve higher returns for his property. The percentage of profit royalty has higher risks because the lessor is captive to the operating efficiencies of the operator, but the potential for higher returns is greater.

Work Commitments: The work commitments for a project depend on the objectives of the lessor and the lessee. The lessor normally seeks some commitments to perform work, in a timely manner, if his objective is to get the property into production. However, if the lessor's objective is to receive income, especially in the short term, he may reduce or eliminate work commitments in favor of higher payments for signature bonus and/or higher periodic payments. If the project is highly speculative, work commitments are normally kept to a minimum.

Lessee's Rights and Obligations: The lessee's rights and obligations in the development of the property must be clearly defined. The lessee's right should be stated to develop and operate a mine, to disturb the surface as required, to use any of the surface for mining and nonmining-related activities, and so forth. The lessee should have the right and the obligation to obtain all permits required for the exploration, development, and operation of the mine and also be responsible for any and all required reclamation. The requirement for regular and thorough reports throughout the lease should be included.

Indemnification: An agreement on indemnification must be reached in the negotiations, including insurance claims and indemnification of the leases. Indemnification is the means of compensating for loss or damage.

Responsibility for Taxes: The responsibility for all taxes must be stated in the contract. The party responsible for all taxes associated with the property and any tax savings or obligations resulting from exploration, development, and operation should be stated. These taxes include property taxes, production taxes, severance taxes, and personal property taxes.

Default and Forfeiture: The basis on which both parties can terminate the contract should be defined. For example, if the lessee determines that he no longer wants to lease the property, what notification and obligations does the lessee have? If the lessee fails to make periodic or production payments, does the lessor have the right to terminate the agreement and what are both parties' obligations on such a termination?

The liability for taxes, payments, reclamation, vendor liens, production royalty, etc., should be stated. Long-term liabilities, such as waste dumps, tailings, toxic material, etc., need to be addressed. This may require that the lessee actually obtain ownership for that portion of the land that has any long-term liability potential. The property rights must be addressed as to equipment and improvements on the land and time frames to remove or pay for the cost of moving. The information that the lessee must provide to the lessor should be stated, such as drill results, assay values, geologic mapping, mine planning, etc. The disposition of interpretive reports must be specifically stated, because most mining companies do not want to provide such information to the lessor. The contract should also include relinquishment of the lease at the time of termination, including the recordation of the relinquishment at the appropriate county court house. Also if the lessee has staked any additional mining claims in a defined area of interest around the lessor's property, the lessor has the right to have those claims "quitclaimed" to him, at his option.

A quitclaim is a legal document for the transfer of title from one person to another.

Recordation of Lease: The contract should address the filing of the lease or a memorandum of the lease in the appropriate county court house. The responsibility for creating and filing the memorandum of the lease should be stated.

Attorney's Fees: The handling of attorney's fees, if a disagreement occurs between the parties, should be addressed. This statement might be that the party that loses the claim pays both parties attorney's fees. Another method could be that each party pays his own attorney's fees, unless the claim is determined to be frivolous by the appropriate judicial authority.

Arbitration: If the parties decide to use arbitration to settle disagreements between them, the procedures for such arbitration should be addressed in the contract.

Force Majeure: What makes up "force majeure" should be defined in the contract. Force majeure is an event or effect that cannot be reasonably anticipated or controlled, such as an "act of God." What types of force majeure will the lessor allow and the lessee willing to accept? The terms must be stated as to the lease obligation that can be put on hold and the obligations that must be met during force majeure. Work commitments are usually put on hold in case of force majeure but payments to the lessor generally continue.

Notices: Names and addresses for notices must be included in the contract. They are used for the making of payments, notices of default, notices of force majeure, and any other notices or communications stated in the contract.

Assignment of Rights: The rights of the lessee to assign the lease to other parties should be stated. The lessor may decide to restrict the assignment to the lessee or a subsidiary of the lessee or may accept any assignment subject to his approval that will not be unjustly withheld. If the lease is assigned to another party, what if any are the rights and obligations of the original lessee?

Lessor Rights: The rights of the lessor to entry onto the property should be stated, including his/her rights to other activities on the area (if he legally has those rights, such as timber sales, grazing rights, etc.). The lessor must have reasonable right of inspection to determine the work being performed by the lessee and to assess compliance with the lease requirements. The lessor should receive, at least annually, a report from the lessee on all appropriate activities on the property.

Assessment Work: The lease must state which party is responsible for performing and recording assessment work on unpatented mining claims, if any are part of the lease. Assessment work is usually performed and recorded by the lessee. Notification and copies of the recordation must be stated so that the lessor has time to perform and record the assessment work if it is not performed and recorded by the lessee.

A typical table of contents of a lease agreement is given in Table 6.4.2.

6.4.5.6 Royalty Rates

Royalty rates vary considerably with conditions like the commodity, grade (quality), location, and demand for the commodity. Typical royalty rates have been published in *Mining Engineering* (Bourne, 1988, 1989).

6.4.6 JOINT VENTURE PROCESS

The purpose of entering into a joint venture agreement generally is to obtain expertise and experience not currently available within the sponsoring company. Another purpose, or possibly an additional purpose. of a joint venture is to share the risk of property development. A definite need should be established before a search is undertaken to find a joint venture partner. The following paragraphs start with the definition of the need for a joint venture, the formal search procedure, the informal search procedure, the request for proposal, the evaluation of the proposals, the negotiation of the contract, and a list of typical headings addressed in the contract.

6.4.6.1 Definition of Need

Before initiating the search for a joint venture partner, the owner should carefully determine the need and desire for a joint venture and define the conditions and requirements for the development of the project. The project should be defined in detail as to the current status, the work required to develop the project, a tentative schedule for development, the technical specifications of the project, an estimate of capital and operating costs, areas of concern that may affect the development, and specific areas of expertise that may be required. An *honest* appraisal of the expertise available in the owner's company is very important in the analysis and determination of need. This project definition will greatly assist in the determination and definition of need. Also the project definition will be needed in the development of the request for proposal.

An example of the potential of a joint venture is provided. The sample project is a precious-metal mine with a production schedule as shown in Chart 17.

Example 6.4.5. Joint Venture Benefits.

Chart 17. Production Schedule (1000 tons)

Year	Ore	Waste
—1	—	2,000
1	500	4,000
2	1,000	4,000
3	1,000	4,000
4	1,000	2,000
5	1,000	1,000
Total	4,500	17,000

The ore contains gold at an average grade of 0.08 oz/ton (2.74 g/t) with a sales price of $400/oz ($12.86/g). Recovery in the heap leach and process plant is expected to be 60%. The projected operating and capital costs for the project are shown in Chart 18.

Chart 18. Owner/Operator Capital and Operating Costs ($1000 and $/ton)

	Mine		Plant	
Year	Capital	Operating	Capital	Operating
—1	$5,000	$0.60	$12,000	$3.50
1	—	$0.60	—	$3.50
2	—	$0.60	—	$3.50
3	—	$0.60	—	$3.50
4	—	$0.60	—	$3.50
5	—	$0.60	—	$3.50

Solution. In order to perform a simplified analysis of the project, it is assumed that all capital is depreciated on a straight-line basis over the life of the project and there is no salvage value. Federal income taxes are 34% (no state or local income taxes), and the depletion allowance is 15%. The annual cash flows are calculated as shown in Chart 19.

Chart 19. Cash Flow Analysis ($1000)

Years	—1	1	2	3	4	5
Revenue	-0-	$9,600	$19,200	$19,200	$19,200	$19,200
Operating costs (–)	$ 1,200	$4,450	$ 6,500	$ 6,500	$ 5,300	$ 4,700
Operating income	($ 1,200)	$5,150	$12,700	$12,700	$13,900	$14,500
Depreciation (–)	$ 833	$3,233	$ 3,234	$ 3,233	$ 3,233	$ 3,234
Depletion (–)	-0-	$ 958	$ 2,880	$ 2,880	$ 2,880	$ 2,880
Taxable income	($ 2,033)	$ 958	$ 6,586	$ 6,587	$ 7,787	$ 8,386
Income tax (–)	($ 691)	$ 326	$ 2,239	$ 2,239	$ 2,647	$ 2,851
After tax income	($ 1,342)	$ 632	$ 4,347	$ 4,438	$ 5,140	$ 5,535
Depreciation (+)	$ 833	$3,233	$ 3,234	$ 3,233	$ 3,233	$ 3,234
Depletion (+)	-0-	$ 958	$ 2,880	$ 2,880	$ 2,880	$ 2,880
Capital (–)	$ 17,000	-0-	-0-	-0-	-0-	-0-
Salvages (+)	-0-	-0-	-0-	-0-	-0-	-0-
Cash flow	($ 17,509)	$4,824	$10,461	$10,461	$11,253	$11,649

The annual cash flows calculated in the foregoing, along with the discount factors and present values for the project, are shown in Chart 20.

Chart 20. Present Value ($1000)

Year	Cash Flow	Discount Factors	Present Value
—1	(17,509)	1.000000	(17,509)
1	4,824	0.869565	4,195
2	10,461	0.756144	7,910
3	10,461	0.657516	6,878
4	11,253	0.571753	6,434
5	11,649	0.497177	5,791
Total	31,139		13,699

If a joint venture partner is used and the partner has expertise in processing, the recovery of gold in the processing plant is assumed to increase to 70%, and the plant capital and operating costs are reduced as shown in Chart 21.

Chart 21. Joint Venture Capital and Operating Costs
($1000 and $/ton)

	Mine		Plant	
Year	Capital	Operating	Capital	Operating
—1	$5,000	$0.60	$10,000	$3.00
1	—	$0.60	—	$3.00
2	—	$0.60	—	$3.00
3	—	$0.60	—	$3.00
4	—	$0.60	—	$3.00
5	—	$0.60	—	$3.00

Per the joint venture agreement, the joint venture partner spends the first $10,000,000 in cost, and then all costs are shared in a 50/50 basis including the profits.

The same assumptions used in the owner case applies to the joint venture case. The year-by-year cash flows are calculated as shown in Chart 22.

Chart 22. Cash Flow Analysis ($1000)

Years	—1	1	2	3	4	5
Revenue	-0-	$11,200	$22,400	$22,400	$22,400	$22,400
Operating costs (–)	$ 1,200	$ 4,200	$ 6,000	$ 6,000	$ 4,800	$ 4,200
Operating income	($ 1,200)	$ 7,000	$16,400	$16,400	$17,600	$18,200
Depreciation (–)	$ 833	$ 2,833	$ 2,834	$ 2,833	$ 2,833	$ 2,834
Depletion (–)	-0-	$ 1,680	$ 3,360	$ 3,360	$ 3,360	$ 3,360
Taxable income	($ 2,033)	$ 2,487	$10,206	$10,207	$11,407	$12,006
Income tax (–)	($ 691)	$ 845	$ 3,470	$ 3,470	$ 3,878	$ 4,082
After tax income	($ 1,342)	$ 1,642	$ 6,736	$ 6,737	$ 7,529	$ 7,924
Depreciation (+)	$ 833	$ 2,833	$ 2,834	$ 2,833	$ 2,833	$ 2,834
Depletion (+)	-0-	$ 1,680	$ 3,360	$ 3,360	$ 3,360	$ 3,360
Capital (–)	$15,000	-0-	-0-	-0-	-0-	-0-
Salvage (+)	-0-	-0-	-0-	-0-	-0-	-0-
Cash flow	($15,509)	$ 6,155	$12,930	$12,930	$13,722	$14,118

The annual cash flows as calculated previously, along with the discount factors and present values, are shown in Chart 23, with the present value determined based on a 15% discount rate. The cash flows represent the cash flows for the project first, and then for the owner.

Chart 23. Project Present Value ($1000)

Year	Cash Flow	Discount Factors	Present Value
—1	(15,509)	1.000000	(15,509)
1	6,155	0.869565	5,352
2	12,930	0.756144	9,777
3	12,930	0.657516	8,502
4	13,722	0.571753	7,845
5	14,118	0.497177	7,019
Total	44,346		22,986

Owner's Present Value ($1000)

Year	Cash Flow	Discount Factors	Present Value
—1	(2,754)	1.000000	(2,754)
1	3,077	0.869565	2,676
2	6,465	0.756144	4,888
3	6,465	0.657516	4,251
4	6,861	0.571753	3,923
5	7,059	0.497177	3,510
Total	27,173		16,494

6.4.6.2 Search for Joint Venture Partner

The search for potential joint venture partners can be either formal or informal. An informal search is generally used because the owner and/or some of the people working for the owner have a working knowledge of the mining companies, or other companies engaged in or interested in engaging in the industry, that have the expertise and financial capability to assist in the development of a project. The formal search involves (1) using a consultant with expertise in the area of need to compile a list of potential joint venture partners, (2) researching professional publications for companies with the appropriate expertise, and/or (3) employing other means to compile a list of appropriate companies. Compilation of the list should include, at a minimum, a preliminary review of the expertise level and financial capabilities of the companies being considered. The review should include prior history with the required type of expertise, corporate and public records of the company, and their reputation in the industry. Compilation of the list of appropriate companies should be the only significant difference between the formal and informal methods unless the owner is predisposed to select a particular joint venture partner.

6.4.6.3 Request for Proposal

Depending on the review of the expertise and financial capabilities of the potential joint venture partners, the list should be reduced to those companies with the appropriate qualification. The request for proposal can be either formal or informal. The formal approach includes the development of a formal request for proposal that is submitted to the list of companies finalized in the selection process in the foregoing. This system tends to make the proposals easier to evaluate because terms, conditions, and format of the proposal have been established.

The informal method is more difficult to evaluate because each company will provide its proposal in a different format, and comparison between companies is more difficult. However, the informal method allows the proposing companies to be more innovative and original in their thought and structure of the presentation. The project definition discussed previously is an excellent means to convey the specifications of the project.

6.4.6.4 Evaluation of the Proposals

Evaluation of the proposal is extremely important in selecting the appropriate joint venture partner. The suggested procedure is very similar to that used in selecting the contract miner, except for the areas to be considered. The major areas to be considered are corporate history, number of operations, successes and failures, number and successes in other joint ventures, individual personnel involved and their ability to relate to your personnel, and common corporate philosophy. The final list developed by the owner may differ from these, but it should cover at least these basic areas. For example, corporate philosophy may be divided into organizational structure, accounting methods, ethics, etc.

The following example provides a format for developing a weighting table (Chart 24) for the evaluation. The numbers inserted on the list are provided simply to illustrate the procedure and not to represent a particular set of circumstances.

Example 6.4.6. Joint Venture Partner Evaluation.

Chart 24. Weighting Table

Area	Ranking	Weighting
Corporate history	3	10%
Number of total operations	11	3%
Number of similar operations	5	8%
Technical expertise	4	10%
Successes/failures	9	5%
Number of joint ventures	8	5%
Successes/Failure in joint ventures	10	4%
Individual people competence	7	7%
Relations with individuals	6	8%
Corporate philosophy	2	15%
Financial offer	1	25%
Total		100%

Following the development of the weighting table, a value must be developed for each of the categories. The value is often judgmental but should be quantitative wherever possible. The value may be, at a minimum, merely a ranking of the individual companies. A value between 1 and 10 for each category is suggested. This value is then multiplied by the weighting developed in the weighting table to determine the weighted value. The weighted values are added together to narrow the list of potential partners. Depending on the results of the evaluation, the joint venture may be narrowed to two or three or may actually indicate the preferred company. An example of a value table and an evaluation table is shown in Chart 25. The value table provides the ranking of the individual companies while the evaluation table provides the results of applying the weighting of each area to the ranking provided in the value table. Again the numbers inserted on the table are provided simply to illustrate the procedure and not to represent a particular set of circumstances.

Solution.

Chart 25. Value Table

Area	Weighting	Companies 1	2	3
Corporate history	10%	10	6	8
Number of total operations	3%	9	8	10
Number of similar operations	8%	2	6	9
Technical expertise	10%	4	7	8
Successes/Failures	5%	2	6	7
Number of joint ventures	5%	8	5	1
Successes/Failures in joint venture	4%	4	8	9
Individual people competence	7%	6	9	8
Relations with individuals	8%	4	8	9
Corporate philosophy	15%	3	9	7
Financial offer	25%	10	9	8

Evaluation Table

Area	Weighting	Companies 1	2	3
Corporate history	10%	1.00	0.60	0.80
Number of total operations	3%	0.27	0.24	0.30
Number of similar operations	8%	0.16	0.48	0.72
Technical expertise	10%	0.40	0.70	0.80
Successes/Failures	5%	0.10	0.30	0.35
Number of joint ventures	5%	0.40	0.25	0.05
Successes/Failures in joint ventures	4%	0.16	0.32	0.36
Individual people competence	7%	0.42	0.63	0.56
Relations with individuals	8%	0.32	0.64	0.72
Corporate philosophy	15%	0.45	1.35	1.05
Financial offer	25%	2.50	2.25	2.00
Total	100%	6.18	7.76	7.71

If, as shown in this case, two of the companies' evaluations are too closely ranked to make the final selection, the two companies should be invited to discuss their proposal in more detail with the owner. These discussions may include a formal presentation and/or informal dialogue on the project and the proposal. Some of the values chosen for the companies may change and thus change the evaluation. This process will lead to the designation of the selected company.

6.4.6.5 Negotiating the Contract

Negotiation of the contract for the joint venture is similar to the negotiation of a lease, as illustrated in 6.4.5.5. The contract must be perceived as fair to all parties involved in order for the joint venture to be successful. The objectives of both parties must be stated at the start of negotiations and should be consistent with the request for proposal provided by the owner and with the proposal provided by the potential joint venture partner.

Listed in the following is the general content of normal negotiations. A discussion of each item follows the listing. The joint venture agreement that results from these negotiations can be in many configurations and structures. Each agreement is individualized for each particular joint venture and is extremely dependent on the companies involved. A general outline is provided in Table 6.4.3, but the content of the negotiations dictates the important aspects of the agreement.

Type of Joint Venture
Term of Joint Venture
Area of Commitment to the Joint Venture
Personnel Assignments and Obligations
Financial Responsibilities
Operating Responsibilities
Organizational Structure-Joint Venture
Accounting Methods
Default
Arbitration
Notices

Type of Joint Venture: The joint venture can take many forms, from a loosely associated partnership to the formation of a corporation to handle the joint venture. The type selected depends on the tax implications for all parties, the operating responsibilities of the parties, the liabilities that the parties are willing to accept, and the individual desires of the parties with respect to ownership.

Term of Joint Venture: The term of the joint venture is generally determined by either the time required to fully exploit the property committed to the joint venture or the desires of the parties to have a longer-term relationship. The term may be flexible to account for the unknowns in the property being developed.

Area of Commitment to the Joint Venture: The boundaries of land, mineral, depth, etc., committed to the joint venture must be defined in detail to reduce the potential for disagreements in the future as to the land (claims) that are involved or the mineral that is being developed. Some parties may commit only the surface minable or underground minable reserves to the joint venture. Definition of the terms of surface or underground minable must be delineated in detail to reduce potential conflicts in the future.

Personnel Assignments and/or Obligations: Each party may commit certain personnel or types of personnel to the joint venture, either on a full-time basis (possibly as an employee of the joint venture) or on a part-time basis. For example, one of the parties may commit to provide the metallurgical expertise to ensure the optimum performance of the processing facility.

Financial Responsibilities: These responsibilities are often defined to some degree in the proposal and during discussions of the proposal. These responsibilities must be well defined, and the accountability of the expenditure of these funds also must be defined. For example, if one party was to expend the first $10,000,000 on the development, was the commitment to provide the joint venture with the funds or for that party to expend the funds as it performed the work? The commitment of funds for property development is generally well defined, but the ability of the joint venture to call for additional funds also must be carefully articulated. This commitment may vary, depending on whether the need is for ongoing operation or for expansion of the mine or for additional development. If one of the parties is unwilling or unable to provide committed or additional funding, the change of ownership division must be stated or the recourse available to the other party.

Operating Responsibilities: Any responsibilities or commitments by the parties to perform part of the operation or development of the project must be defined. For example, is one party going to operate the mine and the other party operate the processing plant? If that is the case, the interface between the parties must be defined, that is, party A puts the ore in the stockpiles

Table 6.4.3. Typical Table of Contents, Joint Venture Agreement

- Recitals
- Definitions
  - Accounting procedures
  - Affiliate
  - Agreement
  - Area of interest
  - Assets
  - Assumed obligations
  - Budget
  - Deadlock
  - Development
  - Effective date
  - Exploration
  - Feasibility report
  - Initial contribution
  - Insurance
  - Joint account
  - Joint venture
  - Manager
  - Mining
  - Operating procedures
  - Operations
  - Participant
  - Participating interest
  - Prime rate
  - Products
  - Program
  - Project
  - Properties or property
  - Royalties
  - Transfer
  - Venture
  - Year, calendar or fiscal
- Representations and Warranties
  - Capacity of participants
  - Representations and warranties
  - Disclosures
  - Acceptance of title
  - Conveyance of assets
  - Joint loss of title
- Name, Purpose, and Term
  - General
  - Name
  - Purpose
  - Limitations
  - Terms
- Relationship of Participants
  - Joint venture agreement
  - Indemnification
  - Federal tax elections and allocations
  - State income tax
  - Tax returns
  - Other business opportunities
  - Transfer or mortgage of rights of properties
- Contributions by Participants
  - Participants initial contributions
  - Assumed obligations
  - Additional cash contributions
  - Failure to make contributions
- Interests of Participants
  - Participating interests
  - Profits and losses
  - Capital accounts
  - Additional contributions
  - Changes in participation
  - Participant's loans
  - Venture loans
- Management Committee
  - Organization
  - Decisions
  - Meetings
  - Action without meetings
  - Matters requiring approval
- Manager
  - Appointment
  - Powers and duties of manager
  - Standard of care
  - Resignation; deemed offer to resign
  - Payments to manager
  - Transactions with affiliates
  - Activities during deadlock
- Programs and Budgets
  - Initial programs and budgets
  - Operations pursuant to programs and budgets
  - Presentation of programs and budgets
  - Review and approval of proposed programs and budgets
  - Election to participants
  - Deadlock on proposed programs and budgets
  - Budget overruns; program changes
  - Emergency or unexpected expenditures
- Accounts and Settlements
  - Monthly statements
  - Cash calls
  - Failure to meet cash calls
  - Audits
- Disposition of Production (Product)
  - Taking in kind or sales
  - Enforcement
- Withdrawal and Termination
  - Termination by expiration of agreement
  - Termination by deadlock
  - Withdrawal
  - Continuing obligations
  - Disposition of assets on termination
  - Noncomplete covenants
  - Right to data after termination
  - Continuing authority
- Acquisitions within Area of Interest
  - General
  - Notice to nonacquiring participant
  - Option exercised
  - Option not exercised
  - Competition
- Transfer of Interest
  - General
  - Limitations on free transferability
  - Preemptive right
  - Exceptions to preemptive rights
- Disputes
  - Reference to chairman
  - Arbitration
  - Confidentiality
  - General
  - Exceptions
  - Duration of confidentiality
- Abandonment and Surrender of Properties
  - Surrender or abandonment of property
  - Reacquisition
- General Provisions
  - Notices
  - Waivers
  - Modification
  - Force majeure
  - Governing law
  - Rule against perpetuities
  - Further assurance
  - Survival of terms and conditions
  - Entire agreement; successors and assigns
  - Memorandum
  - Currency
- Exhibits
  - Properties
  - Assets
  - Liabilities or assumed obligations
  - Accounting procedures
  - Operating procedures
  - Tax matters
  - Feasibility report
  - Insurance

at location B. Also the accounting for the expenditures charged to the joint venture must be defined.

Organizational Structure: The structure of the organization is determined by the type of joint venture and the operating responsibilities of the parties. Following those designations, the structure will dictate the responsibilities and authorities of the personnel working for the joint venture or committed to or working on the joint venture to correspond with these designations. The definition of responsibilities that dictate the organizational structure assists in developing accountability in the operation. Also, the structure of the management committee must be defined for the overall direction of the joint venture, including the decision-making powers.

Accounting Methods: The accounting methods to be used for the joint venture must be designated so that the parties know how to handle the accounting for their company and how it will impact their financial status. This should include not only those methods needed for "accounting" purposes but also how individual cost items are going to be traced for purposes of analyzing the efficiency of the operation.

Default: The basis by which each party can terminate this agreement must be defined. This area is frequently overlooked or handled on a cursory basis. In developing the default terms and conditions, a pessimistic point of view often must be used to ensure that potential defaults are addressed in a realistic manner. Also, long-term liabilities of the parties and the property must be addressed, including environmental and personnel factors.

Arbitration: The method for settling disagreements should be stated in the agreement. If arbitration is to be used, the method of arbitrating the difference should be delineated. If another method is to be used, that method should be stated and the procedures defined.

Notices: The names and addresses for notices must be included in the agreement. The names and addresses are used for the transfer of funds, notices of default, and any other notices or communications stated in the agreement.

6.4.7 EVALUATION OF ALTERNATIVES

As a basis for discussing the evaluation of alternative methods of developing a property, a hypothetical mining project is selected as an example. The evaluation discussed in this section of the chapter is strictly an economic evaluation based on the expected costs of each of the alternative methods of developing and operating the property. Factors other than economics may enter into the decision making, such as corporate philosophy, risks, ability to finance the project, etc. The costs given in the following are costs typical of an open pit, heap-leach, gold-mining operation in Nevada, but do not represent any one operation. However, the procedure shown can be used in any mine operation, whether it is open pit or underground, precious metal or base metal, or metallic or industrial or energy fuel.

Example 6.4.7. Alternative Development Methods.

Base Criteria—Basic criteria for the property and the mine operation are shown as follows:

Minable reserves	10,000,000 tons (9.1 Mt)
Grade	0.06 oz/ton gold (2.06 g/t)
Strip ratio	2.5 tons waste/ton ore (2.5 t waste/t ore)
Production rate	2,000,000 tpy (1.8 Mt/a)

Capital and Operating Costs—Capital and operating costs shown in the following are illustrated from the perspective of the owner of the property. The operating and capital costs are its costs as determined by an estimate prepared according to the procedure of Chapter 6.3. The contract mining costs assumes the owner decided to use a contract miner to remove the overburden, mine and crush the ore, and place the crushed ore on the heap leach pad. The leasing alternative assumes the property was leased to a mining company that mined and produced the ore according to the same production schedule as the other alternatives. The lease assumes a straight royalty basis and includes a signature bonus and periodic payments. The joint venture alternative assumes a joint venture partner was selected for its mining expertise. The owner performed the exploration prior to the development of the joint venture. The joint venture partner expended the first $10,000,000 of the development as his buy-in.

6.4.7.1 Owner/Operator

The analysis has been simplified to illustrate the evaluation process. All the complexities of all possible ventures could not be addressed in one example.

The owner has determined that the mine will be operated on the production schedule shown in Chart 26.

Chart 26. Production Schedule (1000 tons)

Year	Ore	Waste	Total
—1	-0-	2,000	2,000
1	2,000	5,000	7,000
2	2,000	5,000	7,000
3	2,000	5,000	7,000
4	2,000	5,000	7,000
5	2,000	3,000	5,000
Total	10,000	25,000	35,000

The owner has performed a cost estimate and has determined that expected capital and operating costs are as follows:

Capital Costs	
Mobile equipment	$ 7,750,000
Crushing and agglomeration	$ 6,250,000
Leach pad and plant	$ 7,000,000
Ancillary Facilities	$ 4,000,000
Infrastructure	$ 2,000,000
Total	$27,000,000
Operating Costs	
Ore mining	$0.70/ton ($0.77/t)
Waste removal	$0.65/ton ($0.72/t)
Crushing and agglomerating	$0.80/ton ($0.88/t)
Heap construction	$0.30/ton ($0.33/t)
Leach pad and plant operation	$2.85/ton ($3.14/t)
Overhead	$2,200,000/yr

The capital costs include the mobilization (freight and erection) of equipment and working capital for the project. The operating costs for the tasks are labor and supplies only (including operating costs). Supervision, property taxes, insurance, etc., are contained in overhead cost.

In the exploration and development of the property, the owner has spent $6,000,000 in exploration and development drilling and in permitting and other developmental costs. The preproduction and development years are designated as year —3, —2, and —1. These preproduction and development costs were incurred as follows: year — 3, $2,000,000; year —2, $3,000,000; year — 1, $l,000,000. These costs have been capitalized and amortized over the ore production years. The annual amortization cost is $1,200,000.

The annual operating costs for the project are shown in Chart 27.

Chart 27. Annual Operating Costs ($1000)

Year	Ore	Waste	Crush/ agglom.	Heap constr.	Leach & plant	Overhead	Total
—1	-0-	$ 1,300	-0-	-0-	-0-	$ 2,200	$ 3,500
1	$1,400	$ 3,250	$1,600	$ 600	$ 5,700	$ 2,200	$14,750
2	$1,400	$ 3,250	$1,600	$ 600	$ 5,700	$ 2,200	$14,750
3	$1,400	$ 3,250	$1,600	$ 600	$ 5,700	$ 2,200	$14,750
4	$1,400	$ 3,250	$1,600	$ 600	$ 5,700	$ 2,200	$14,750
5	$1,400	$ 1,950	$1,600	$ 600	$ 5,700	$ 2,200	$13,450
Total	$7,000	$16,250	$8,000	$3,000	$28,500	$13,200	$75,950

For simplification, the depreciation base for all applicable capital is established at 10 years, straight-line, with zero salvage value. The depreciation for the project is $2,700,000/yr. Salvage value on projects less than 10 years is assumed to be book value and is taken in the last year of project life. The salvage value reflects the recoupment of working capital and the demobilization of the equipment.

Revenue for the project is calculated on an average ore assay of 0.06 oz/ton (2.06 g/t) of gold with an average recovery of 70% and an average gold price of $400/oz ($12.86/g). The annual revenue for this project is $33,600,000 with a total project revenue of $168,000,000.

A simplified cash flow analysis is shown in Chart 28. No royalty exists on the mining claims. Taxes, other than income tax, are included in the overhead. (Nevada does not have a corporate income tax.) The federal tax rate is assumed to be 34%. The tax calculation assumes consolidated returns with other operations so that tax losses can be taken in the year in which they occur.

Chart 28. Cash Flow Analysis ($1000)

Years	—3	—2	—1	1	2	3	4	5	Total
Revenue	-0-	-0-	-0-	$33,600	$33,600	$33,600	$33,600	$33,600	$168,000
Operating costs (–)	-0-	-0-	$ 3,500	$14,750	$14,750	$14,750	$14,750	$13,450	$ 75,950
Operating income	-0-	-0-	($ 3,500)	$18,850	$18,850	$18,850	$18,850	$20,150	$ 92,050
Depreciation (–)	-0-	-0-	$ 2,700	$ 2,700	$ 2,700	$ 2,700	$ 2,700	$ 2,700	$ 16,200
Amortization (–)	-0-	-0-	-0-	$ 1,200	$ 1,200	$ 1,200	$ 1,200	$ 1,200	$ 6,000
Depletion (–)	-0-	-0-	-0-	$ 5,040	$ 5,040	$ 5,040	$ 5,040	$ 5,040	$ 25,200
Taxable income	-0-	-0-	($ 6,200)	$ 9,910	$ 9,910	$ 9,910	$ 9,910	$11,210	$ 44,650
Income tax (–)	-0-	-0-	($ 2,108)	$ 3,369	$ 3,369	$ 3,369	$ 3,369	$ 3,811	$ 15,179
After tax income	-0-	-0-	($ 4,092)	$ 6,541	$ 6,541	$ 6,541	$ 6,541	$ 7,399	$ 29,471
Depreciation (+)	-0-	-0-	$ 2,700	$ 2,700	$ 2,700	$ 2,700	$ 2,700	$ 2,700	$ 16,200
Amortization (+)	-0-	-0-	-0-	$ 1,200	$ 1,200	$ 1,200	$ 1,200	$ 1,200	$ 6,000
Depletion (+)	-0-	-0-	-0-	$ 5,040	$ 5,040	$ 5,040	$ 5,040	$ 5,040	$ 25,200
Exploration/Development (–)	$ 2,000	$ 3,000	$ 1,000	-0-	-0-	-0-	-0-	-0-	$ 6,000
Capital (–)	-0-	-0-	$27,000	-0-	-0-	-0-	-0-	-0-	$ 27,000
Salvage (+)	-0-	-0-	-0-	-0-	-0-	-0-	-0-	$10,800	$ 10,800
Cash flow	($2,000)	($3,000)	($29,392)	$15,481	$15,481	$15,481	$15,481	$27,139	$ 54,671

The cash flows developed in the foregoing were discounted at 15% to determine the present value of the cash flows, as shown in Chart 29.

Chart 29. Present Value ($1000)

Year	Cash Flow	Discount Factors	Present Value
—3	(2,000)	1.000000	(2,000)
—2	(3,000)	0.869565	(2,609)
—1	(29,392)	0.756144	(22,225)
1	15,481	0.657516	10,179
2	15,481	0.571753	8,851
3	15,481	0.497177	7,697
4	15,481	0.432328	6,693
5	27,139	0.375937	10,203
Total	54,671		16,789

6.4.7.2 Contract Miner

The same operation example is used in this case as was used in the owner/operator case. The only difference is consideration of a contract miner to perform the waste removal, ore mining, crushing and agglomerating, and the placement of the crushed and agglomerated ore on the leach pad. The owner has developed a bidder's list of contractors and transmitted an RFP to the contractors on the bidder's list. The owner has received proposals from the contractors and has selected the best contractor based on the price, the qualifications of the contractor and his personnel, the availability of equipment and personnel, the ability to perform the job, the contractual terms, and the type of owner/contract miner relationship he expects. The contract miner's bid is as follows:

Ore mining	$1.15/ton of ore ($1.27/t)
Waste removal	$1.10/ton of waste ($1.21/t)
Crushing and agglomeration	$1.50/ton of ore ($1.65/t)
Heap construction	$0.50/ton of ore ($0.55/t)

The owner's expected cost for the remaining operations are

Capital Costs	
Leach pad and plant	$7,000,000
Infrastructure	$2,000,000
Total	$9,000,000
Operating Costs	
Leach pad & plant operation	$2.85/ton of ore ($3.14/t)
Overhead	$1,000,000/yr

In the exploration and development of the property, the owner has spent $6,000,000 in exploration and development drilling and in permitting and other developmental costs. These costs were incurred as follows: year — 3, $2,000,000; year —2, $3,000,000; year — 1, $1,000,000. These costs have been capitalized and amortized over the ore production years. The annual amortization cost is $1,200,000.

The annual operating costs for this operation are shown in Chart 30.

Chart 30. Annual Operating Costs ($1000)

Year	Ore	Waste	Crush/ agglom.	Heap constr.	Leach & plant	Overhead	Total
—1	-0-	$ 2,200	-0-	-0-	-0-	$1,000	$ 3,200
1	$ 2,300	$ 5,500	$ 3,000	$1,000	$ 5,700	$1,000	$18,500
2	$ 2,300	$ 5,500	$ 3,000	$1,000	$ 5,700	$1,000	$18,500
3	$ 2,300	$ 5,500	$ 3,000	$1,000	$ 5,700	$1,000	$18,500
4	$ 2,300	$ 5,500	$ 3,000	$1,000	$ 5,700	$1,000	$18,500
5	$ 2,300	$ 3,300	$ 3,000	$1,000	$ 5,700	$1,000	$16,300
Total	$11,500	$27,500	$15,000	$5,000	$28,500	$6,000	$93,500

For simplification, the depreciation period for all applicable capital is taken as 10 years and assumed to be straight-line with no salvage. The depreciation for the project is $900,000/yr. Salvage value on projects less than 10 years is assumed to be book value and is taken in the last year of project life. The salvage value reflects the recoupment of working capital and the demobilization of the equipment.

Revenue for the project is calculated on an average ore assay of 0.06 oz/ton (2.06 g/t) of gold with an average recovery of 70% and an average gold price of $400/oz ($12.86/g). The annual revenue for this project is $33,600,000 with a total project revenue of $168,000,000.

A simplified cash flow analysis is shown in Chart 31. No royalty exists on the mining claims. Taxes, other than income tax, are included in the overhead. The federal tax rate is assumed at 34%. The tax calculation assumes consolidated returns with other operations so that tax losses can be taken in the year in which they occur.

Chart 31. Cash Flow Analysis ($1000)

Years	—3	—2	—1	1	2	3	4	5	Total
Revenue	-0-	-0-	-0-	$33,600	$33,600	$33,600	$33,600	$33,600	$168,000
Operating costs (–)	-0-	-0-	$ 3,200	$18,500	$18,500	$18,500	$18,500	$16,300	$ 93,500
Operating income	-0-	-0-	($ 3,200)	$15,100	$15,100	$15,100	$15,100	$17,300	$ 74,500
Depreciation (–)	-0-	-0-	$ 900	$ 900	$ 900	$ 900	$ 900	$ 900	$ 5,400
Amortization (–)	-0-	-0-	-0-	$ 1,200	$ 1,200	$ 1,200	$ 1,200	$ 1,200	$ 6,000
Depletion (–)	-0-	-0-	-0-	$ 5,040	$ 5,040	$ 5,040	$ 5,040	$ 5,040	$ 25,200
Taxable income	-0-	-0-	($ 4,100)	$ 7,960	$ 7,960	$ 7,960	$ 7,960	$10,160	$ 37,900
Income tax (–)	-0-	-0-	($ 1,394)	$ 2,706	$ 2,706	$ 2,706	$ 2,706	$ 3,454	$ 12,884
After tax income	-0-	-0-	($ 2,706)	$ 5,254	$ 5,254	$ 5,254	$ 5,254	$ 6,706	$ 25,016
Depreciation (+)	-0-	-0-	$ 900	$ 900	$ 900	$ 900	$ 900	$ 900	$ 5,400
Amortization (+)	-0-	-0-	-0-	$ 1,200	$ 1,200	$ 1,200	$ 1,200	$ 1,200	$ 6,000
Depletion (+)	-0-	-0-	-0-	$ 5,040	$ 5,040	$ 5,040	$ 5,040	$ 5,040	$ 25,200
Exploration/Development (–)	$ 2,000	$ 3,000	$ 1,000	-0-	-0-	-0-	-0-	-0-	$ 6,000
Capital (–)	-0-	-0-	$ 9,000	-0-	-0-	-0-	-0-	-0-	$ 9,000
Salvage (+)	-0-	-0-	-0-	-0-	-0-	-0-	-0-	$ 3,600	$ 3,600
Cash flow	($2,000)	($3,000)	($11,806)	$12,394	$12,394	$12,394	$12,394	$17,446	$ 50,216

The cash flows developed previously were discounted at 15% to determine the present value of the cash flows, as illustrated in the owner/operator case with the present value determined to be $19,778,000.

6.4.7.3 Lease

In the lease case example, the owner has decided to lease the property to a mining company. The payment terms call for a signature bonus of $50,000, a minimum annual advance royalty of $100,000/yr, and a royalty rate of 5% of gross value. The signature bonus occurs in year —2. The owner spent $40,000 on marketing the property and negotiating the lease. He also spends $5,000/yr monitoring the lessee when the lessee is not mining ore and $25,000/yr when the lessee is mining ore. He has not experienced or will not experience any other costs or any capital investment.

A simplified cash flow analysis is shown in Chart 32. The federal tax rate is assumed to be 34%.

Chart 32. Cash Flow Analysis ($1000)

Years	—3	— 2	— 1	1	2	3	4	5	Total
Payments	$ 50	$ 100	$ 100	$ 1680	$ 1680	$ 1680	$ 1680	$ 1680	$ 8650
Deduct advances (–)	—	—	—	$ 200	—	—	—	—	$ 200
Net revenue	$ 50	$ 100	$ 100	$ 1480	$ 1680	$ 1680	$ 1680	$ 1680	$ 8450
Costs (–)	$ 40	$ 5	$ 5	$ 25	$ 25	$ 25	$ 25	$ 25	$ 175
Net income	$ 10	$ 95	$ 95	$ 1455	$ 1655	$ 1655	$ 1655	$ 1655	$ 8275
Depletion (–)	-0-	-0-	-0-	$ 252	$ 252	$ 252	$ 252	$ 252	$ 1260
Taxable income	$ 10	$ 95	$ 95	$ 1203	$ 1403	$ 1403	$ 1403	$ 1403	$ 7015
Income tax (–)	$ 3	$ 32	$ 32	$ 409	$ 477	$ 477	$ 477	$ 477	$ 2385
After tax income	$ 7	$ 63	$ 63	$ 794	$ 926	$ 926	$ 926	$ 926	$ 4630
Depletion (+)	-0-	-0-	-0-	$ 252	$ 252	$ 252	$ 252	$ 252	$ 1260
Cash flow	$ 7	$ 63	$ 63	$ 1046	$ 1178	$ 1178	$ 1178	$ 1178	$ 5890

These cash flows were discounted at 15% to determine the present value of the cash flows, as illustrated in the owner/operator case, with the present value determined to be $3,008,000.

6.4.7.4 Joint Venture

The same operating example is used in this case as was used in the owner/operator case. The only difference is the use of a joint venture to exploit the property. The joint venture partner that was selected by the owner has mining expertise that will help the project. Also the joint venture partner has equipment available to use. The joint venture has determined that the mine will be operated on the production schedule shown in Chart 33.

Chart 33. Production Schedule (1000 tons)

Year	Ore	Waste	Total
—1	-0-	2,000	2,000
1	2,000	5,000	7,000
2	2,000	5,000	7,000
3	2,000	5,000	7,000
4	2,000	5,000	7,000
5	2,000	3,000	5,000
Total	10,000	25,000	35,000

The joint venture has developed a cost estimate of the project for the joint venture: this estimate follows. The owner should note that the operating costs are lower, which reflect the expertise gained from the joint venture partner. Also the capital costs are lower, reflecting the low book value equipment available from the joint venture partner.

Capital Costs

Mobile equipment	$ 4,750,000
Crushing and agglomeration	$ 6,250,000
Leach pad and plant	$ 7,000,000
Ancillary facilities	$ 3,000,000
Infrastructure	$ 2,000,000
Total	$23,000,000

Operating Costs

Ore mining	$0.65/ton	($0.72/t)
Waste removal	$0.60/ton	($0.66/t)
Crushing and agglomerating	$0.80/ton	($0.88/t)
Heap construction	$0.30/ton	($0.33/t)
Leach pad and plant operation	$2.85/ton	($3.14/t)
Overhead	$2,200,000/yr	

The capital costs include mobilization (freight and erection) of the equipment and working capital for the project. The operating costs for the tasks are labor and supplies only. Supervision, property taxes, insurance, etc., are included in the overhead cost.

In the exploration and development of the property, the owner has spent $6,000,000 in exploration and development drilling and in permitting and other developmental costs. These costs were incurred as follows: year — 3, $2,000,000; year —2, $3,000,000; year — 1, $1,000,000. These costs have been capitalized and amortized over the ore production years. The annual amortization cost is $1,200,000. The joint venture partner will invest the first $10,000,000 in the project as his buy-in.

The annual operating costs for the project are shown in Chart 34.

Chart 34. Annual Operating Costs ($1000)

Year	Ore	Waste	Crush/ agglom.	Heap constr.	Leach & plant	Overhead	Total
—1	-0-	$ 1,200	-0-	-0-	-0-	$ 2,200	$ 3,400
1	$1,300	$ 3,000	$1,600	$ 600	$ 5,700	$ 2,200	$14,400
2	$1,300	$ 3,000	$1,600	$ 600	$ 5,700	$ 2,200	$14,400
3	$1,300	$ 3,000	$1,600	$ 600	$ 5,700	$ 2,200	$14,400
4	$1,300	$ 3,000	$1,600	$ 600	$ 5,700	$ 2,200	$14,400
5	$1,300	$ 1,800	$1,600	$ 600	$ 5,700	$ 2,200	$13,200
Total	$6,500	$15,000	$8,000	$3,000	$28,500	$13,200	$74,200

For simplification, the depreciation for all applicable capital is established at 10 years, straight-line, with no salvage. The depreciation for the project is $2,300,000/year. Salvage value on projects less than 10 years is assumed to be book value and is taken in the last year of the project. The salvage value reflects the recoupment of working capital and the demobilization of the equipment.

Revenue for the project is calculated on an average ore assay of 0.063 oz/ton (2.06 g/t) of gold with an average recovery of 70% and an average gold price of $400/oz ($12.86/g). The slightly higher grade reflects the superior ore control techniques employed in mining the ore. The annual revenue for this project is $35,280,000 with a total project revenue of $176,400,000.

A simplified cash flow analysis is shown in Chart 35. No royalty exists on the mining claims. Taxes, other than income tax, are included in the overhead. Again, the federal tax rate is assumed to be 34%. The tax calculation assumes consolidated returns with other operations so that tax losses can be taken in the year in which they occur.

Chart 35. Cash Flow Analysis ($1000)

Years	—3	—2	—1	1	2	3	4	5	Total
Revenue	-0-	-0-	-0-	$35,280	$35,280	$35,280	$35,280	$35,280	$176,400
Operating costs (–)	-0-	-0-	$ 3,400	$14,400	$14,400	$14,400	$14,400	$13,200	$ 74,200
Operating income	-0-	-0-	($ 3,400)	$20,880	$20,880	$20,880	$20,880	$22,080	$102,200
Depreciation (–)	-0-	-0-	$ 2,300	$ 2,300	$ 2,300	$ 2,300	$ 2,300	$ 2,300	$ 13,800
Amortization (–)	-0-	-0-	-0-	$ 1,200	$ 1,200	$ 1,200	$ 1,200	$ 1,200	$ 6,000
Depletion (–)	-0-	-0-	-0-	$ 5,292	$ 5,292	$ 5,292	$ 5,292	$ 5,292	$ 26,460
Taxable income	-0-	-0-	($ 5,700)	$12,088	$12,088	$12,088	$12,088	$13,288	$ 55,940
Income tax (–)	-0-	-0-	($ 1,938)	$ 4,110	$ 4,110	$ 4,110	$ 4,110	$ 4,518	$ 19,020
After tax income	-0-	-0-	($ 3,762)	$ 7,978	$ 7,978	$ 7,978	$ 7,978	$ 8,770	$ 36,920
Depreciation (+)	-0-	-0-	$ 2,300	$ 2,300	$ 2,300	$ 2,300	$ 2,300	$ 2,300	$ 13,800
Amortization (+)	-0-	-0-	-0-	$ 1,200	$ 1,200	$ 1,200	$ 1,200	$ 1,200	$ 6,000
Depletion (+)	-0-	-0-	-0-	$ 5,292	$ 5,292	$ 5,292	$ 5,292	$ 5,292	$ 26,460
Exploration/Development (–)	$ 2,000	$ 3,000	$ 1,000	-0-	-0-	-0-	-0-	-0-	$ 6,000
Capital (–)	-0-	-0-	$23,000	-0-	-0-	-0-	-0-	-0-	$ 23,000
Salvage (+)	-0-	-0-	-0-	-0-	-0-	-0-	-0-	$ 9,200	$ 9,200
Cash flow	($2,000)	($3,000)	($25,462)	$16,770	$16,770	$16,770	$16,770	$26,762	$ 63,380

The cash flows developed were discounted at 15% to determine the present value of the cash flows, as illustrated in the owner/operator case, with the present value determined to be $22,402,000 for the joint venture.

The net present value for the owner can be determined by subtracting the $10,000,000 invested in year — 1 by the joint venture partner and using half of all the cash flows except the exploration and development expenses. The exploration and development expenses were spent prior to the joint venture agreement in years — 3 and —2, and we assume the agreement between the companies was based on that. The cash flows just developed were discounted at 15% to determine the present value of the cash flows, as illustrated in the owner/operator case, with the present value for the owner determined to be $14,982,000.

6.4.7.5 Evaluation Summary

Results of the evaluation of the four alternatives using the examples stated previously are shown in the following:

Alternative	Net Present Value
Owner/Operator	$16,789,000
Contract miner	$19,778,000
Lease	$ 3,008,000
Joint venture	$14,982,000

The numbers used here are simulated and should not be construed as an endorsement of contract mining as the proper economic choice in all circumstances. The choice, using net present value only, is the use of the contract miner. The second choice would be the owner/operator; the third choice would be the joint venture; the fourth choice would be the lease. However, other factors may enter into the decision making. Corporate philosophy may dictate that the company desires to become an owner/operator and is unwilling to use a contract miner. The accuracy of the cost number used in the evaluation must be considered when making the decision. Possibly, an upside potential exists for lowering the costs such that the owner/operator is the preferred alternative. Another overriding consideration may be that the inability to raise capital to invest in the operation or the desire to reduce the risks of the project either dictates the use of a contract miner or possibly even the lease alternative. The decision is seldom black or white, and many factors must be considered when making the decision.

6.4.8 CORPORATE PHILOSOPHY

The philosophy of the corporation can and should play an important role in the decision on project strategy. The corporation should develop both short-term and long-range goals. These goals should reflect the philosophy of the corporation with respect to the development of mining properties. Short-term goals may be different than long-range goals. The establishment of these goals is discussed in the following. The development of goals and objectives and the strategy to achieve the goals is given a cursory discussion in the following paragraphs. For a more detailed discussion of goals, objectives, and strategy, see Quinn, et al. (1988).

6.4.8.1 Short-term Goals

The short-term goals of a company establish where the company expects to be five years hence. These goals should be well thought out to insure they are supportive of the long-range goals.

The goals should be well defined in quantitative terms. A strategy should be developed on how to achieve the goals. This strategy should be detailed sufficiently to know just how the company plans to achieve the goals and that the resources, both in terms of human and fiscal resources, are committed to the achievement of the stated goals.

6.4.8.2 Long-range Goals

The long-range goals of the company should be developed with considerable thought into the long-range profitability and business direction (strategy) of the company. These goals must be set so that the short-term goals and the strategies to achieve those goals can complement the long-range goals. These long-range goals may dictate the actions taken on the development of an individual mining property. If the company is an established mining company, the actions may be dictated purely by economic analyses. However, if a company is currently a small mining company and desires to become a major supplier in the market of a particular commodity, it may desire to mine the property itself as an owner/operator to assist in the growth of the company. A company just getting into the mining business may desire to have a contract miner during start-up and then take over the property later to establish itself in the expertise of mining. A company that does not have expertise in the mining of a commodity and does not have the goal to become a mining company may use a contract miner for all aspects of the mining venture or lease the property to another company for development and operation.

6.4.8.3 Areas of Expertise

A company must review the expertise it has or lacks when making the decision on the development and operation of a mine. If a company has the expertise, it can utilize that expertise and experience to develop the mine to the optimum. However, if the company is missing the needed expertise, it must decide whether or not it wants to acquire the expertise. If it desires to acquire the expertise, the company can either directly hire outside personnel or hire a contract miner for the start-up and then hire selected personnel from the contract miner to acquire the expertise. If the company does not desire to acquire the expertise, they can use a contract miner for all aspects or lease the property to another company.

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Chapter 6.5
INVESTMENT ANALYSIS

THOMAS J. O'NEIL AND DONALD W. GENTRY

6.5.1 OBJECTIVES

Previous chapters of this section provide insight and guidance into selecting and quantifying input variables pursuant to the financial evaluation of a mining venture. Chapter 6.5 takes the analysis to the next step: putting the data into an analytical framework to help determine whether the contemplated investment should be carried out. The basic approach in most such analytical models is the same. The goal is to determine whether or not the project will provide cash benefits sufficiently in excess of the cash costs of establishing and operating the project to justify (1) the cost of funds employed and (2) the risk involved.

Although the project evaluation tools described herein are very useful for decision-making purposes, the analyst must remember always that there is no substitute for accurate input data. Virtually every troubled mining project can trace its difficulties to poor estimates of key variables, not to the use of some improper project evaluation method.

6.5.2 TYPES OF EVALUATIONS

As a raw exploration prospect travels the long, tortuous path to become an operating mine, and as detailed in Chapter 6.2 of this section, it is almost continuously subjected to evaluation. The level of effort and choice of analytical method vary considerably, however, depending on the specific objectives of the evaluation.

6.5.2.1 Stage of Development

Early in the exploration phase, the goal is generally to determine whether the next stage of exploration spending is justified. This often is a fairly subjective and limited evaluation, because typically little hard data are available.

As the prospect continues into progressively more costly exploration and development stages, the evaluation becomes more detailed. One of the discounted cash flow techniques is generally employed in such studies (intermediate economic studies) when exploration has reached a relatively advanced stage.

For the final investment decision, a thorough financial analysis is mandatory for all significant capital investments. Furthermore, for most mining investments, complete sensitivity analysis and risk assessment also are essential.

6.5.2.2 Purpose of the Evaluation

Although the evaluation process is almost continuous, there are specific milepost evaluations that are common in the mining industry. Three standard mileposts are the prefeasibility study, feasibility study, and bankable feasibility study. The level of effort and detail in the financial analysis of the project increases as a project progresses through this sequence.

As described in Chapter 6.2, the *prefeasibility study* is generally the first broad engineering-economic review of the project as a commercial venture. The study typically involves a broad range of disciplines producing input data, some for the first time. The objective of the prefeasibility study is to (1) determine the overall merit of the project at a relatively early stage and (2) identify the key variables upon which the project's viability relies and that, therefore, usually require further investigation.

The formal prefeasibility study can take from one to six months to complete, depending on the size and complexity of the project; and the financial analysis generally is fairly modest, with usually only a few sensitivities on key variables.

When the project has been sufficiently explored and tested to reduce the technical risks to reasonable levels, a final *comprehensive feasibility study* is undertaken. This is the principal document justifying capital investment in the project and, as such, should contain a complete financial analysis of the project, analysis of key variables, and assessment of downside risks and upside potential. The feasibility study usually requires three to nine months to complete—again depending on the size and complexity of the project, but also on the amount of additional data the report authors must independently develop to complete the study.

With the growth of external financing of mining ventures, another version of the feasibility study, the *bankable feasibility study*, is sometimes required. This is, in general, an even more exhaustive version of the feasibility study, usually performed by impartial consultant(s) selected by the lender. The objective is to satisfy the financier about the ability of the project to repay the loans. For this reason, some of the topics that may be of less concern to the owner (e.g., debt servicing) must be treated in great detail here.

Financial analysis in bankable feasibility studies is complex and tailored to meet the needs of the funding agency. Leveraged economic analysis may be employed, particularly when gold loans or other project financings are contemplated.

6.5.3 INVESTMENT CRITERIA

When a company is confronted with several investment opportunities, it becomes necessary to evaluate the attractiveness of each proposal. Any evaluation criterion utilized should provide company management with a means of distinguishing between acceptable projects in a consistent manner. In other words, the criterion should help answer the question, "Is project A and/or project B good enough to justify capital investment by the company?" To provide this necessary information for investment decision making, any satisfactory evaluation criterion must respect two basic principles:

1. Bigger benefits are preferable to smaller benefits.
2. Early benefits are preferable to later benefits.

The following project evaluation criteria section are not intended to represent an exhaustive list available to the analyst. Rather, those discussed represent the major evaluation criteria utilized for evaluation investment proposals within the minerals industry.

6.5.3.1 Simplified Criteria

A number of simplified methods for assessing the investment value of capital projects have evolved. The most common of these are presented in this chapter.

Table 6.5.1. Accounting (Average) Rate of Return

	Year 1	Year 2	Year 3	Year 4	Average
Net operating income	$3,000	$4,000	$5,000	$6,000	$4,500
Depreciation	2,000	2,000	2,000	2,000	2,000
Taxable income	1,000	2,000	3,000	4,000	2,500
Taxes @ 50%	500	1,000	1,500	2,000	1,250
Net profit	500	1,000	1,500	2,000	1,250
Book value:					
January 1	$10,000	$8,000	$6,000	$4,000	
December 31	8,000	6,000	4,000	2,000	
Average	9,000	7,000	5,000	3,000	6,000

$$\text{Average rate of return} = \frac{1250}{6000} \times 100 = 20.8\%$$

Degree of Necessity: There are times when company management must make investment decisions based only on limited quantitative data of dubious accuracy. These types of investments may be referred to as *degree of necessity* investments and are characterized by the fact that the decisions are either obvious or can be quantified only to a limited degree. To illustrate the concept, suppose the main production hoist at a large, profitable underground mine suddenly failed. Under these conditions, it is conceivable that some comparative analyses could be performed in order to help decide what kind, model, etc., of hoist should be purchased. However, the investment decision to actually purchase a new hoist need not be predicated on rigorous economic analyses resulting in calculated rates of return or some other yardstick of profitability, unless the operation is already marginal. Indeed, performing a formal benefit/cost analysis for an investment proposal indicated in this example is not a simple task, and, in this case, may simply delay the actual decision required.

Other examples closely related to the minerals industry are expenditures in the areas of research and development (R&D) and exploration. How much, if any, capital should be allocated to R&D activities? What quantity of capital should be allocated to exploration activities ? Should this be a fixed percentage of the annual corporate budget, some amount commensurate with industry average, or what?

Clearly these kinds of investment decisions fall within the realm of management judgment and are predicated more on corporate strategies than on any specific economic criterion. Thus some major investment decisions can not be easily analyzed with quantitative criteria.

Accounting Rate of Return: One of the more common versions of the accounting rate of return calculation is often referred to as the average rate of return. The average rate of return is calculated by dividing average annual profits after taxes by the average investment in the project (average book value after deducting depreciation).

Example 6.5.1. Table 6.5.1 illustrates the calculation procedure for determining the average rate of return on a project requiring an investment of $10,000 with an estimated salvage value of $2,000 at the end of year 4. Estimated annual profits are given. Depreciation is on a straight-line basis.

Another version of the accounting rate of return uses original investment for the denominator rather than average book value. In the example presented, the calculation would be represented as $1{,}250/10{,}000 \times 100 = 12.5\%$. This version is somewhat less informative since income is averaged but investment is not. However, as an approximation to the internal rate of return (see discussion on Internal Rate of Return), it usually provides better results than using the average investment.

The primary advantages of the accounting rate of return criterion are (1) it is simple to calculate, (2) it makes use of readily available accounting information, and (3) it provides a “rate of return” number to which most managers seem to relate. Once the calculation has been performed for a project, the rate can be compared with the company’s required, or cutoff, rate to determine whether the project should be accepted or rejected.

The principal disadvantages of the method are that (1) it is based on accounting profits rather than actual cash flows, and (2) it does not take into account the timing of these profits. These are very serious disadvantages, as they violate some basic concepts and requirements set forth in this chapter. In reality, it takes little additional effort to work with actual annual cash flows and incorporate the concept of time value of money into the analysis.

Payback (Payout) Period: One of the most common evaluation criteria used by mining companies is the payback or payout period. The payback period is simply the number of years required for the cash income from a project to return the initial cash investment in the project. Although benefits resulting from the investment can be measured in terms of net profit for calculation purposes, modern practice has resulted in the use of annual net cash flows for the denominator.

Example 6.5.2. Table 6.5.2 illustrates five investment proposals having identical capital investment requirements but differing expected annual cash flows and lives. The payback period is calculated for each.

The investment decision criterion for this technique suggests that if the calculated payback period for an investment proposal is less than some maximum value acceptable to the company, the proposal is accepted; if not, it is rejected. In other words an investment proposal having a payback period of three years is acceptable to a company having a hurdle value of five years and is preferable to a second project having a payback period of four years.

An interesting situation arises when calculating the payback period for a typical new mining venture where several years of negative cash flows (investment) are anticipated prior to project start up. Should the payback period start from the beginning of investment or the beginning of positive cash flow? That is, is the payback period for the project in Fig. 6.5.1 four years or nine years?

Table 6.5.2. Payback Period

	Annual Net Cash Flows				
	Proposal A	Proposal B	Proposal C	Proposal D	Proposal E
Initial investment	$10,000	$10,000	$10,000	$10,000	$10,000
Project year 1	2,000	7,000	1,000	6,000	6,000
2	2,000	2,000	2,000	2,000	2,000
3	2,000	1,000	7,000	2,000	2,000
4	2,000	2,000	2,000	-0-	3,000
5	2,000			-0-	4,000
6	2,000			-0-	1,000
7	2,000			-0-	1,000
8				-0-	500
Payback period, years	5.0	3.0	3.0	3.0	3.0

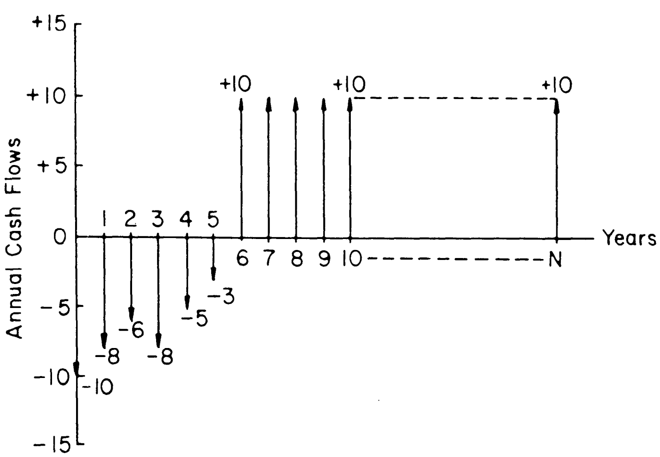

Fig. 6.5.1. Annual cash flows for a new mining project, illustrating effect of preproduction period on payback period calculation.

Table 6.5.3. Discounted Payback Period

Proposal A		Present worth factor, (P/F) at 7%	Present value of cash flows
Initial investment:	$10,000	1.0	$10,000
Cash flows:			
Project year 1	$2,000	0.9346	1,869
2	$2,000	0.8734	1,747
3	$2,000	0.8163	1,633
4	$2,000	0.7629	1,526
5	$2,000	0.7130	1,426
6	$2,000	0.6663	1,333
7	$2,000	0.6228	1,246
Undiscounted payback period	5.0 yr	Discounted payback period	6.4

From an economic and financial standpoint, the investment payback period is theoretically more correct, because it represents the commitment of investment throughout the preproduction period, particularly the opportunity cost associated with the investment during this period. However, both payback period calculations provide useful information to the decision maker, and *consistency* in application is the most important feature. It is often important to know how long it will take to recoup the investment in a project after actual start-up. When such information can aid in helping management make the correct investment decision, it should be used even though it may not be theoretically as correct as another calculation procedure.

The payback period criterion has some significant disadvantages. First, the payback period fails to consider cash flows after the payback period; therefore, it cannot be regarded as a suitable measure of profitability. For example, proposals D and E in Table 6.5.2 have identical payback periods and would be rated equally by the criterion. However, proposal D never earns a profit, while proposal E continues to generate income after its payout period. Obviously, proposal E is better than proposal D, but the payback period does not recognize this distinction.

An additional disadvantage of the method is that it does not consider the magnitude or timing of cash flows during the payback period. Rather, it only considers the recovery or payback interval as a whole. For example, proposals B and C in Table 6.5.2 have identical payback periods. However, proposal B is clearly preferable to proposal C because of the large return of funds early in proposal life that could then be reinvested by the firm. When time value of money concepts are taken into consideration, one proposal may be clearly preferable to the other, even though the payback period does not indicate this difference. This troublesome point may be alleviated by calculating a *discounted payback period* for investment proposals. The discounted payback period for proposal A in Table 6.5.2 is calculated in Table 6.5.3. As expected, the discounted payback period will always result in a longer payback interval than the undiscounted calculation procedure.

The last major disadvantage with the payback period is the problem of establishing the appropriate hurdle rate or maximum acceptable value. Someone must decide whether or not a proposal having, say, a four-year payback period is acceptable. Establishing this rate, or hurdle, is often the result of subjective and arbitrary decisions. Should one hurdle rate be applied to all investment proposals even though some may be considerably more risky than others? What should the appropriate hurdle rate be, and how is it determined for investments having varying degrees of risk? Purely subjective determinations are often unsatisfactory when utilizing analytical techniques to assist with investment proposal decision making.

A payback period is, however, widely used for the following reasons:

1. Payback period is simple and easy to calculate and also provides a single number that can be used as an index of proposal profitability.

2. Payback period provides some protection to management from exposure to excessive risk. Although risky investments should have shorter payback periods due to the risk involved, someone still has to decide what an acceptable payback period should be. These are often subjective decisions having no theoretical basis. Alternatives for analyzing risk in mining ventures that are inherently superior to simply adjusting the payback period are discussed in 6.5.5.

3. Some argue that payback period can minimize "lost opportunity risk" to the firm because cash inflows will be returned to the firm within a short span of time, thus allowing the firm to seize unexpected investment opportunities that may become available.

An objective appraisal of the payback period indicates that it can provide some useful information to the decision maker when considering investment proposals. However, the technique has too many drawbacks to be used alone. It should *not* be used as the sole quantitative tool for making investment decisions, but rather in a supplementary role to the other more sophisticated methods. Many firms use the payback period criterion as a hurdle that investment proposals must clear before progressing to more rigorous and sophisticated forms of analyses.

6.5.3.2 Discounted Cash Flow Methods

Any one of the several discounted cash flow methods (DCF) discussed in this part is superior to the simplified methods described in 6.5.3.1. The DCF methods require (1) determination of periodic project cash flows over some uniform planning period and (2) consideration of the time value of money through the use of an appropriate interest rate.

Essentially these methods compare the cash benefits from a project with the cash costs of the project, both adjusted for timing by the use of an interest rate.

Present, Future, and Annual Value: The *present value* (PV), or present worth, method of measuring investment proposal desirability is a widely used technique. The term present value (PV) simply represents an amount of money at the present time (t = 0) that is equivalent to some sequence of future cash flows discounted at a specified interest rate. In other words, this technique recognizes the time value of money and provides for the calculation of an amount at the present time that is equivalent in value to a series of future cash flows.

Present value calculations are most frequently performed to determine the present worth of income-producing property, such as an existing mining operation. If the future annual cash flows can be estimated, then by selecting an appropriate interest rate, the present value of the property can be calculated. This value should provide a reasonable estimate of the price at which the property could be bought or sold.

In the more general case of investment proposal evaluation, one is interested in determining the difference between cash outflows and cash inflows associated with the proposal on a present-value basis. This calculation procedure is referred to as the *net present value* (NPV) method and is simply the difference between the sum of the present value of all cash inflows and the sum of the present value of all cash outflows. NPV can be expressed as follows:

$$\text{Net present value} = \Sigma \text{ present value of cash benefits} - \Sigma \text{ present value of cash costs} \quad (6.5.1)$$

If the NPV of the proposal is a positive value (NPV> 0), then the project should be accepted. A positive NPV indicates that the investment proposal will provide for the recovery of invested capital, a return on the unrecovered capital throughout the project life at the stipulated interest rate utilized in the calculation, as well as some surplus amount. In other words, the project promises to yield a return in excess of that rate used in the calculation procedure. If the rate used in the calculation is the rate of return investors expect the firm to earn on investments, then proposals having a positive NPV should increase the wealth of the firm. Similarly, proposals yielding a negative NPV at the required discount rate should be rejected.

The following example illustrates the calculation procedure for net present value determinations.

Example 6.5.3. Given the following conditions:

Initial investment:	$100,000
Project life:	10 years
Salvage value:	$ 20,000
Annual receipts:	$ 40,000
Annual disbursements:	$ 22,000

Minimum acceptable rate of return (discount rate): 12%

Calculate the NPV.

Solution. Use Eq. 6.5.1.

	Present Value
1. Annual receipts = $40,000 (*P/A*,12%,10)	$226,000
2. Salvage value = $20,000 (*P/F*,12%,10)	6,440
Total PV of cash inflows:	$232,440
3. Annual disbursements = $22,000 (*P/A*, 12%,10)	124,300
4. Initial investment =	100,000
Total PV of cash outflows	$224,300
Net present value (NPV):(PV inflow-PV outflow):	$ 8,140

Since the NPV > 0 in the foregoing example, the project should be accepted according to this criterion. Note that annual *net* benefits (receipts – disbursements) could have been discounted as a single stream, and the NPV would be the same. This is *not* the case for the benefit/cost ratio criterion (see Benefit/Cost Ratio) where separate discounting of receipts and disbursements is essential.

Under certain conditions, the present value concept also can be used for evaluating projects on a cost basis. For example, it is often desirable to determine what it will cost, in today's equivalent, to operate alternative pieces of equipment over some future period. When comparing investment proposals the NPV decision criterion would suggest that the proposal promising the largest PV of inflows should be selected, or if working with costs, the proposal promising the lowest PV of outflows should be selected. However, the analyst must use caution in any study based upon cost minimization. Often the lowest cost alternative is to do nothing; the objective function really is to minimize cost *subject to the completion of some particular task.* Thus selection of the alternative based on minimum PV of costs assumes that alternative can perform the required assignment.

It is instructive to calculate the net present value associated with different interest rates for an investment proposal (see Fig. 6.5.2.). This shows that the present value of an investment proposal is a function of the interest rate selected. The connection between the interest rate selected for calculation purposes and the firm's required rate of return is obvious, in that only those projects promising a positive NPV at the required rate of return should be accepted.

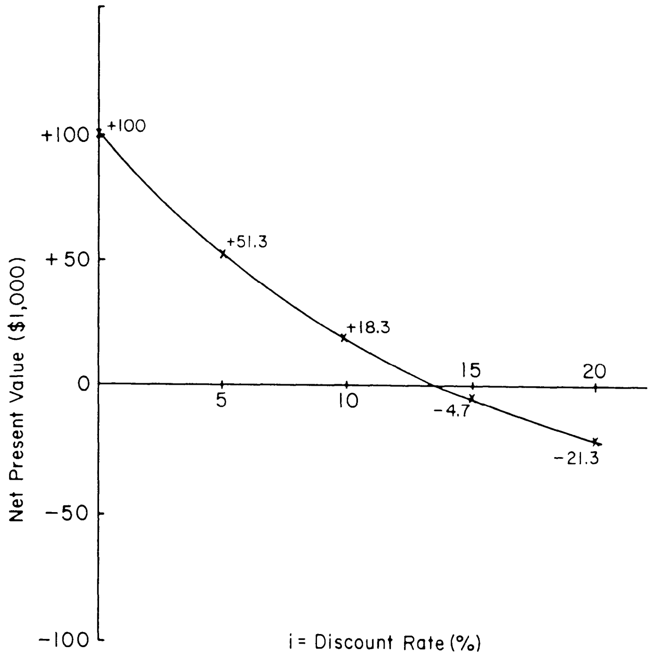

Fig. 6.5.2. Present value profile for previous example.

Investment proposals can also be evaluated on the basis of how much money they will accumulate at some future point in time, usually the end of project life. Value determinations calculated in this manner are referred to as *future values* and represent the future worth amount for a specific proposal at some point in time at a given interest rate.

Recognizing the time-value-of-money concept, it is apparent that this method is just the reverse of the present value concept. In fact, the future value amount can be calculated by first determining the present value amount of the cash flows in the following manner.

$$F = P\ (F/P,\ i,\ n) \tag{6.5.2}$$

Likewise,

$$P = F\ (P/F,\ i,\ n) \tag{6.5.3}$$

In other words, for given values of interest rate i and years n, the future value amount is simply the present value times a constant. Consequently, if an investment proposal has a present value amount two times as large as an alternative proposal's present value, it will also have a future value amount two times as large as the alternative proposal.

In view of the foregoing discussion, it makes no theoretical difference if projects are evaluated on the basis of future values or present values. However, most project evaluators prefer to work in terms of present value amounts, probably because they are considering amounts in equivalent dollars at the time the accept/reject decision is under consideration ($t = 0$). In general most investment decisions in the minerals industry are based on present value determinations rather than future value determinations.

There are times when it may be more convenient to evaluate investment proposals in terms of *annual value* or *annual cost,* as opposed to present value or future value amounts in the aggregate. For example, many analysts prefer to compare pieces of operating equipment in terms of annual costs simply because associated benefits are often difficult to quantify and cost records are readily available.

Annual value refers to a *uniform annual series of money (annuity)* for a given period of time, which is equivalent in amount to a specified sequence of annual cash flows under consideration. The concept of equivalency suggests that the equivalent annual value for a series of cash flows can be determined by first calculating the present value amount of the actual cash flow series and then multiplying this amount by the *capital recovery factor* that is, in effect, an annualizing factor. Therefore, the annual value amount [sometimes referred to as the *equivalent uniform annual value* (EUAV) may be represented as follows:

$$\text{Annual value (EUAV)} = (\text{PV of cash flows}) \times \text{capital recovery factor} \tag{6.5.4}$$

The following example is intended to demonstrate the calculation procedure for determining the annual value of a project and to illustrate the cost relationship with the present value method.

Example 6.5.4. Suppose a new piece of equipment is being considered for purchase which promises to generate annual benefits in the amount of $10,000, annual costs of $5000, and a life of 10 years. If the initial cost of the machine is $40,000 and the expected salvage value at the end of 10 years is $2000, what is the net annual worth if interest on investment capital is 10%?

Solution. Use Eq. 6.5.2.

		Annual Worth
Benefits:	$10,000/yr	$10,000
Salvage:	$ 2,000 (*A/F,* 10% 10)	125
Costs:	$ 5,000/yr	— 5,000
Investment:	$40,000 (*A/P,* 10%, 10)	— 6,508
	Net annual value:	— 1,383

Since the net annual value is less than 0, the calculation shows that the project is expected to earn less than the 10% interest rate used.

Comments—Present, future, and annual values are all measures of equivalency and differ only in the times at which they are determined. Therefore, for fixed values of i and n, the techniques provide consistent bases for comparison of investment proposals. They will all provide the same accept/reject decision for the proposal in question.

Benefit/Cost Ratio: The *benefit/cost ratio* (B/C ratio), sometimes referred to as the *profitability index* (PI), is generally defined as the ratio of the sum of present value of future benefits to the sum of the present value of present and future investment outlays and other costs. This ratio is expressed as follows:

$$\text{B/C ratio (PI)} = \frac{\Sigma\text{PV of net cash inflows}}{\Sigma\text{PV of net cash outflows}} \tag{6.5.5}$$

In order to perform this calculation an interest rate must be specified prior to present value determination. If the calculation results in a PI > 1.0, the investment proposal should be accepted; if not, it should be rejected. This is the same as saying the project should be accepted if it has a NPV > 0. Indeed, the only difference between the NPV calculation and the PI calculation is that the NPV is the *difference* between the present value of inflows and outflows, whereas the PI is the *ratio* between the two.

For any given project, the NPV method and the PI method will provide the same accept/reject decision, assuming the calculation is performed at the same interest rate. However if a *choice* must be made between two investment proposals, these methods may provide inconsistent project rankings. This aspect is illustrated in the following example.

Example 6.5.5.

	Project A	Project B
Present value cash inflows	$500,000	$100,000
Present value cash outflows	$300,000	$ 50,000
Net present value	$200,000	$ 50,000
Benefit/cost ratio	1.67	2.0

Which of these present-value-based techniques is correct, and why do they provide different rankings? There are really two answers to these questions. In absolute terms, the expected economic contribution of project A to the firm is greater than that promised by project B. This aspect is correctly represented by the NPV technique. However, the B/C ratio reflects the *relative profitability* of the two projects. In this case, the B/C ratio indicates that project B promises a greater return per dollar of outflow. Stated in other terms, the relative gain from the capital resources committed to project B is greater than from those committed to project A.

If A and B are mutually exclusive projects, an important question to ask is, "What happens to the extra $400,000 if project B is selected?" If this is invested in a third project, X, then the correct comparison is the B/C ratio for A with the composite B/C ratio for projects B and X.

The PI is the one investment criterion where it makes a difference whether one treats separate streams of benefits and costs rather than net benefits. Separate discounting of benefits and costs is preferable (Gentry and O'Neil, 1984).

In summary, the NPV method is preferred for determining the absolute expected economic contribution of a project. However, it is often relative profitability of a project that is of interest, particularly in capital-rationing situations, and it is here that the project-ranking capability of the B/C ratio is most appropriate.

Internal Rate of Return: When evaluators in the minerals industry speak of a rate of return on an investment proposed, they are almost always referring to the so-called *discounted cash flow return on investment (DCF-ROI)* or the *discounted cash flow rate of return (DCF-ROR).* These terms are special versions of the more generic term, *internal rate of return (IRR),* or marginal efficiency of capital. This criterion is employed more in the minerals industry for investment proposal evaluation than perhaps any other technique.

The internal rate of return is defined as the interest rate that equates the sum of the present value of cash inflows with the sum of the present value of cash outflows for a project. This is the same as defining the IRR as the rate that satisfies each of the following expressions:

$$\begin{aligned} \Sigma\text{PV cash inflows} - \Sigma\text{PV cash outflows} &= 0 \\ \text{NPV} &= 0 \\ \text{PI} &= 1.0 \\ \Sigma\text{PV cash inflows} &= \Sigma\text{PV cash outflows} \end{aligned} \tag{6.5.5}$$

In general, the calculation procedure involves a trial-and-error solution, unless the annual cash flows subsequent to the investment take the form of an annuity. The following examples illustrate the calculation procedures for determining the internal rate of return.

Example 6.5.6. Given an investment project having the following annual cash flows, find the IRR.

Year	Cash Flow
0	$ — 30,000
1	— 1,000
2	5,000
3	5,500
4	4,000
5	17,000
6	20,000
7	20,000
8	— 2,000
9	10,000

Solution: Step 1. Pick an interest rate and solve for the NPV. Try 15%.

$$\begin{aligned} \text{NPV} = &-30{,}000\ (1.0) - 1{,}000\ (P/F,1,15\%) + 5{,}000\ (P/F,2,15) \\ &+ 5{,}500\ (P/F,\ 3,15) + 4{,}000(P/F,4,15) + 17{,}000(P/F,5,15) + 20{,}000(P/F,6,15) + 20{,}000(P/F,7,15) \\ &- 2{,}000(P/F,8,15) + 10{,}000(P/F,9,15) \\ = &\ \$+5{,}619 \end{aligned}$$

Since the NPV > 0, 15% is not the IRR. It now becomes necessary to select a higher interest rate in order to reduce the NPV value. If r = 20% is used, the NPV = $-1,664 and, therefore, this rate is too high. By interpolation, the correct value for the IRR is determined to be 18.7%.

Example 6.5.7. Given an investment that promises the following uniform annual cash flows, find the IRR.

Year	Cash Flow
0	$ — 20,000
1	6,000
2	6,000
3	6,000
4	6,000
5	6,000

Solution. Since the annual cash flows subsequent to the investment are in the form of an annuity, the solution is simpler and takes the following form:

$$\begin{aligned} \Sigma\text{PV outflows} &= \Sigma\text{PV inflows} \\ \$20{,}000 &= 6{,}000(P/A,5,\text{X}\%) \\ \frac{20{,}000}{6{,}000} &= (P/A,5,\text{X}\%) \\ 3.33 &= (P/A,5,\text{X}\%) \end{aligned}$$

The solution is found by looking for the value 3.33 under the P/A column for n = 5 years in the appropriate interest table. The value 3.33 lies between 15 and 16%. By interpolation, the correct IRR is 15.2%. Most hand-held business calculators have pre-programmed IRR routines for the rapid solution of all such problems.

The acceptance or rejection of a project based on the IRR criterion is made by comparing the calculated rate with the required rate of return, or cutoff rate, established by the firm. If the IRR exceeds the required rate the project should be recommended; if not, it should be rejected.

Because the internal rate of return is such a popular evaluation criterion throughout the minerals industry and others, it deserves careful scrutiny to ensure that it is truly worthy of this popularity. The following points are offered for consideration:

1. Even though the IRR provides for the determination of an internal percentage rate, it still must be compared with the

hurdle, cutoff, or required rate of return established by the firm before the accept/reject decision can be made. Presumably this stipulated required rate of return established by the firm is related to the firm's cost of capital or required cutoff rate and carries with it the implicit borrowing and reinvestment assumptions of any discounting process. If this is not the case, and the required rate of return is a subjectively determined value, then similar criticism to that offered in the payback period discussion is warranted.

2. Perhaps the most significant problem associated with the IRR lies in what engineers and managers perceive it to mean. When people speak of a project's projected rate of return they typically are thinking in terms of the project's "rate of return on investment." This implication brings forth some interesting questions.

a. What is the real return generated by a project? Should this return be measured in terms of profits or cash flows, since cash flows represent a return on and a return of investment?

b. What is the investment? Since part of the investment is returned annually, the amount of investment remaining must be continually declining. Therefore, *all* of the investment is not working on an an annual basis. Consequently, what should be used for "investment" in the calculation procedure?

Clearly, these are complex questions, and anyone who has worked in the area of project evaluation appreciates the difficulty involved in arriving at answers. However, these questions do illustrate some misconceptions many individuals have with respect to the meaning of the term rate of return.

The *rate of return* calculated in the foregoing fashion is the percentage or rate of interest earned on the *unrecovered* portion of the investment such that the cash flow schedule makes the unrecovered investment equal to zero at the end of the investment's life. Thus the *whole* of the investment can only earn the calculated rate of return if the intermediate cash flows are reinvested at the calculated rate for the remaining life of the project.

Because the IRR is merely the mathematical solution to an algebraic equation, some questions regarding its suitability as an investment criterion have arisen under certain circumstances. Some of these questions concern the following:

1. Implicit reinvest assumptions.
2. Multiple solving rates.
3. Appropriate hurdle rates.

Among these problems, only the second—multiple solving rates—occasionally prevents the IRR from accurately ranking investment proposals. A necessary but not sufficient condition for there to be more than one interest rate that will solve the basic IRR equation is that there must be more than one sign reversal in the cash flows. A typical investment proposal contains one such reversal: cash outflow during construction followed by cash inflow during operation. If additional sign reversals occur, additional solving rates may exist. In such a case, no one solving rate is any more valid than any other, and, therefore, some other investment criterion should be used.

The reader is referred to Gentry and O'Neil (1984) for a complete discussion of the three potential problem areas cited.

In spite of the limitations mentioned, the IRR remains widely used in the minerals industry. Virtually every new mining project must pass an IRR test before construction approval is granted.

6.5.3.3 Comparison of Methods

The following discussion relates to some specific relationships among evaluation criteria which are generally of interest to investment proposal analysts.

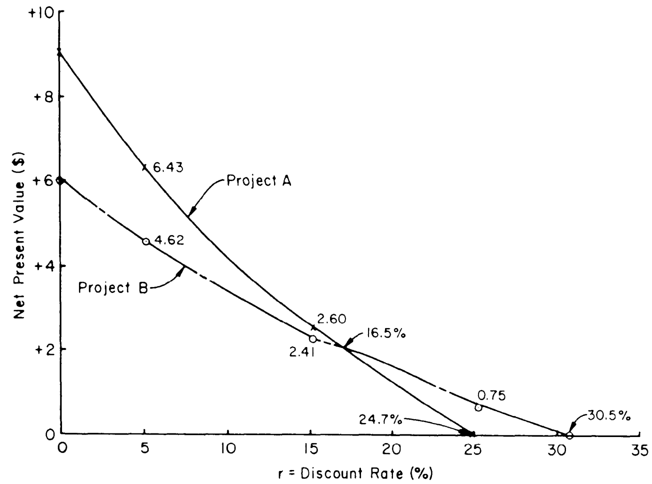

Fig. 6.5.3. Present value profiles for two projects, showing NPV and IRR values.

IRR vs. Payback Period: The internal rate of return and the payback period continue to be two of the more popular evaluation techniques utilized in the minerals industry. Although some relationships do exist between these criteria, they occur under some rather unique circumstances. For instance, where projects have long lives, substantially in excess of the payback period, and where income streams are uniform each year, the payback period is a good approximation to the reciprocal of the internal rate of return. In the unique case where $n \to \infty$, this relationship exists precisely.

IRR vs. NPV: The internal rate of return and the present value methods provide identical accept/reject answers for a single investment proposal. However, it is important to recognize that these two discounted cash flow techniques can give contradictory results when comparing mutually exclusive projects (only one can be selected). Consider two mutually exclusive investment proposals, A and B, which are expected to generate annual cash flows as follows:

	Annual cash flows ($1,000s)	
Year	Project A	Project B
0	$−10	$−10
1	1	7
2	5	5
3	6	3
4	7	1

Fig. 6.5.3 illustrates the net present value profiles for each of these projects. Notice that the IRR for projects A and B are 24.7 and 30.5%, respectively. Therefore, project B is considered superior to project A. However, if the required rate of return were 15% the NPV of proposals A and B is 2.60 and 2.41, respectively, and project A is preferred to project B because it contributes more wealth to the firm. At a required rate of approximately 16.5%, there is virtually no difference between the two projects.

The conflict between these two criteria is the result of different implied assumptions for reinvestment rates for funds generated by the projects. The internal rate of return method assumes that received funds are reinvested at the IRR over the remaining life of the project. The present value method assumes reinvestment at a rate equal to the required rate of return used as the discount rate.

In view of this conflict, which method is best for evaluating investment proposals? The answer to this question depends upon what is considered to be the appropriate reinvestment rate for the intermediate cash flows generated by the project. If one must choose between these two techniques, most believe the present value method is superior—at least from a theoretical standpoint.

When using the internal rate of return to rank projects, a high reinvestment rate is assumed for a proposal having a high IRR whereas a proposal having a low IRR carries with it a low reinvestment rate assumption. Only by coincidence will the IRR calculated be the same as the reinvestment rate actually available to the firm for intermediate cash flows. Also there is seldom any sound reason to assume that reinvestment opportunities following one project would be more favorable than those following another project.

In contrast, the present value method assumes the required rate of return is the reinvestment rate which remains the same for *all* proposals. This rate is intended to represent the minimum return on opportunities available to the firm. This may introduce some conservatism into the calculation, since actual reinvestment rates may exceed these minimum rates. Nonetheless, it has the advantage of being applied consistently to all investment proposals. For these reasons the present value method is preferred over the IRR method for ranking projects, remembering that only in the case of mutually exclusive projects can this conflict arise.

6.5.4 EVALUATING ALTERNATIVES

The previous discussions dealt primarily with the problem of evaluating individual investment projects from the standpoint of measuring proposal acceptability to the firm. The evaluation criteria discussed are simply methods for helping the firm make the appropriate accept/reject decision for a given investment proposal. However, the final investment decision is not based solely on the outcome of the accept/reject decision resulting from evaluation of the investment proposal. Rather, an investment decision also must be based on determination of which of the acceptable investment proposals are best in terms of meeting the objectives of the firm. Consequently, the problem is not one of simply asking, “Are projects A and B acceptable to the firm,” but also, “Is project A better or worse than project B?” It is this problem of ranking or evaluating investment alternatives that is discussed briefly here.

Since these issues are essential to the problem of *capital budgeting,* it may be beneficial to place the problem in proper perspective. Briefly, capital budgeting is the process of allocating available capital in an optimal manner to investment proposals, the benefits from which are to be realized in the future. In a general sense, capital budgeting encompasses:

1. Generation of investment proposals.
2. Estimation of annual cash flows for the proposals.
3. Evaluation of the cash flows.
4. Selection of projects based on an acceptance criterion.
5. Continuous reevaluation of the proposals after acceptance or for investment.

This final selection of projects for investment is based on determination of which projects are acceptable to the firm and, ultimately, the relative desirability of these investment proposals. This latter aspect of evaluating investment alternatives is critical to the capital budgeting process.

Most financial managers view the capital budgeting problem as one concerned with the choice of a group of investment projects from a larger suite of *acceptable* proposals. This problem of *budgeting* capital stems from one or more constraints which prohibit the funding of all acceptable projects available to the firm. If there were no capital constraints imposed, the firm would simply invest in all *acceptable* proposals. Unfortunately, few firms have this luxury, and the general case finds constraints of one form or another imposed on the organization that affect the ultimate investment decision. Typically, these constraints result from a shortage of available capital for new investment proposals, although restrictions of materials and supplies, limited labor availability and management talent, and the mutual exclusiveness of investment proposals also may exist.

If the financial objective of the firm is—it should be—to maximize stockholder wealth, then capital budgeting decisions should be based on the following basic principles (Stevens, 1979, p. 157):

1. Every increment of capital expenditure must justify itself, and

2. An acceptable investment proposal today is better than the speculation that a better proposal will become available in the future.

Dependent vs. Independent Projects: An investment proposal is said to be *independent* when the acceptance of this proposal from a suite of proposals has no effect on the acceptance of any of the other proposals contained in the suite. It is doubtful if very many proposals in a firm are truly independent, but they generally are considered to be independent if they are functionally different. For example, proposals concerning the purchase of a new rotary drill rig, air conditioning the corporate office building, and undertaking a new marketing campaign would generally be considered to be independent proposals.

The capital budgeting problem associated with choosing between independent investment proposal alternatives generally is quite easy. Under these conditions, an appropriate evaluation criterion is used to make the accept/reject decision. These criterion values can then be ranked and the relative proposal desirability determined.

A potential problem with ranking independent investment proposals arises when the IRR and present value methods are used. As pointed out earlier the IRR method will give consistent results with the present value method for accept/reject decisions. However, these criteria can provide inconsistent ranking of independent investment proposals, as previously discussed and illustrated in Fig. 6.2.3.

The more general case involving investment proposals is where they are not independent but are related to one another in such a way that the acceptance of one proposal from within the suite of proposals will influence the acceptance of others. The most common case of investment proposal interdependencies is that of *mutually exclusive* proposals. Mutually exclusive proposals refer to the situation where a group of proposals are related to one another in such a manner that the acceptance of one proposal precludes the acceptance of any of the other proposals. An example of mutually exclusive investment proposals might be the situation where a mine requires an additional piece of primary loading equipment. The investment proposals to be considered might include an electric shovel, hydraulic shovel, and front-end loader. The selection of any one of these proposals would preclude any of the other options from further consideration, since only one alternative is necessary to perform the job. Mutually exclusive investment proposals are a special case of the capital budgeting problem. The appropriate method for analyzing and ranking mutually exclusive proposals is covered later in this chapter.

It is perhaps important to note that whenever there is *capital rationing*—constraints on the amount of capital available for investments—and the aggregate investment cost of all acceptable proposals exceeds the capital available for investment, financial interdependencies are introduced between investment proposals. This may occur, for example, when one project that ranks lower than another is accepted so that a higher return on the entire capital budget is achieved. These interdependencies can occur whether the proposals are inherently independent, contingent, or mutually exclusive. Although these interdependencies are often not obvious, they are, nonetheless, introduced whenever budget constraints are imposed and amplify the importance of the capital budgeting problem.

Three particular situations that may be encountered with mutually exclusive projects are discussed in the following.

6.5.4.1. Projects Having Unequal Lives

The comparison of mutually exclusive investment proposal alternatives having different lives is a common one and represents a special case in the capital budgeting exercise.

It should be noted that the capital budgeting decision in this case concerns investing in two courses of action, not just investing in two projects. The fundamental consideration centers on the total economic impact generated by each of the two courses of action at a given point in the future. Also these opportunities need not be mutually exclusive. For instance, under the constraints of capital rationing, one may be faced with the classic problem of investing in a large, long-life mine promising a modest rate of return vs. small, short-life mines promising high rates of return. An important question here is, "Can the firm continue to generate small-mine reinvestment opportunities (i.e., what are the reinvestment assumptions)?"

When comparing investment proposal alternatives with unequal lives, the basic principle that all alternatives under consideration must be compared over the same time span is fundamental to sound decision making. Equal time spans for investment alternatives must be assumed if the effect of undertaking one alternative is to be compared directly with the effect of undertaking any other alternative.

The basis for comparing mutually exclusive investment proposal alternatives with unequal lives is generally based on one of the following three common assumptions regarding future alternatives (Stevens, 1979, p. 161):

1. Assume that money generated (cash flows) by each alternative will be invested by the firm in other assets that will earn the minimum or required rate of return for a period of time equal to the life of the longest alternative.

2. Assume that each investment alternative will recycle for a period of time equal to the least common multiple of the alternatives' lives. When alternatives are recycled under this assumption, the initial investment, life, salvage value, and annual disbursements are assumed to be identical to the estimates used for the first life cycle.

3. Make specific assumptions (estimates) about future investment opportunities for a period of time equal to the life of the longest alternative.

The appropriate assumption to use will depend upon the type of problem and the assumption which is believed to be the most accurate representation of the future. The following example illustrates assumption 1.

Example 6.5.8. Suppose the following cash flows represent two mutually exclusive investment proposals which have to deal with expanding a mine's production. If the required rate is 12%, which alternative should be selected?

Solution.

Year	Annual Cash Flows ($1000s) Project A	Project B
0	$– 150	$–200
1	55	60
2	55	60
3	55	60
4	55	60
5	55	60
6		60
7		60
8		60
IRR	24.3%	24.9%

The IRR indicates that project B is slightly more profitable than project A. Calculating the NPV of both projects using a required rate of return of 12% yields the following:

$$\begin{aligned} NPV_A &= 55(F/A,12\%,5)(F/P,12\%,3)(P/F,12\%,8) - 150 \\ &= 55(6.353)(1.405)(0.4039) - 150 = 48.29 \\ NPV_B &= (F/A,12\%,8)(P/F,12\%,8) - 200 \\ & \quad 60(12.300)(0.4039) - 200 = 98.06 \end{aligned}$$

The NPV analysis suggests that project B will maximize the value to the firm, and therefore project B should be accepted under assumption 1. Obviously, standard NPV discounting could have been used in the example rather than the circuitous route of forward compounding followed by discounting.

Assumption 2 is often applied to situations where mutually exclusive investment proposal alternatives are measured in terms of negative cash flows. These are typically investment proposals common to most operating divisions and pertain to cost comparisons or equipment replacement alternatives. The following illustrates this procedure.

Example 6.5.9. Assume the following two machines can perform a given job equally well. If the initial investment and annual disbursements are as given and the required rate of return is 10%, which machine should be selected?

	Machine X	Machine Y
Initial investment ($1000s)	150	200
Annual operating disbursements ($1000s)	18	10
Life	5 yr	10 yr
Salvage value ($1000s)	2	0

Solution. Under assumption 2, comparison of the two machine alternatives may be represented as follows:

Year	Annual Cash Flows ($1000s) Machine X	Machine Y
0	−150	−200
1	− 18	− 10
2	− 18	− 10
3	− 18	− 10
4	− 18	− 10
5	− 18, +2, −150	− 10
6	− 18	− 10
7	− 18	− 10
8	− 18	− 10
9	− 18	− 10
10	− 18, +2	− 10

The NPV calculations are as follows:

$$
\begin{aligned}
NPV_x &= -18(P/A,10\%,10) + 2(P/F,10\%,5) - 150(P/F,10\%,5) - 150 + 2(P/F,10\%,10) \\
&= -18(6.1446) + 2(0.6209) - 150(0.6209) - 150 + 2(0.3856) \\
&= \$-351.73
\end{aligned}
$$

$$
\begin{aligned}
NPV_y &= -10(P/A,10\%,10) - 200 \\
&= -10(6.1446) - 200 \\
&= \$-261.45
\end{aligned}
$$

Expressed in equivalent uniform annual costs (EUAC) the machines would have the following costs:

$$
\begin{aligned}
EUAC_x &= (-351.73)(0.1628) = \$-57.26 \\
EUAC_y &= (-261.45)(0.1628) = \$-42.56
\end{aligned}
$$

Based on this analysis, machine Y promises the firm an annual cost savings of $14,700 (57,260 — 42,560) per year over the 10-year interval, if it is selected over machine X. Therefore, machine Y is the preferred alternative.

The reader is referred to Gentry and O'Neil (1984) for a further discussion of the strengths and weaknesses of the three approaches described for evaluating projects with unequal lives.

6.5.4.2 Projects Having Unequal Investment

When comparing mutually exclusive investment proposal alternatives, there are two main principles that should apply. These are as follows (Canada, 1971, p. 62):

1. Each increment of investment capital must justify itself.
2. Compare a higher investment project against a lower investment only if the lower investment project is justified.

Based on these principles, the criterion for choosing between investment proposal alternatives then becomes, "select the proposal that requires the highest investment for which each increment of invested capital is justified."

It is this concept of "bigger is better" that is discussed in this section. Obviously, if two proposals have the same indicated rate of return but different initial investment requirements, the project requiring the larger investment will generate the largest magnitude of total benefits or wealth to the firm. In essence, the problem is one of maximizing use of the investment dollar.

Optimizing use of the investment dollar is really not a troublesome issue in the situation where a firm has adequate investment capital available to undertake all investment proposals that promise returns in excess of the firm's required rate of return. Under these conditions the wealth of the firm will, in theory, be maximized by simply investing in all projects that surpass the cutoff rate. However, where capital rationing does exist, the problem of optimum utilization is an important one.

Under the capital-rationing constraint, all investment proposals that exceed the firm's required or cutoff rate may not be chosen for investment. Additionally, the firm may generate more wealth by selecting several smaller, less profitable proposals that fully utilize the capital budget than to accept one large investment proposal that results in only partial utilization of the budget. The following illustrates this concept.

Example 6.5.10. Suppose that the following investment proposals were available to a firm. If the capital budget constraint is $500,000 for the period, select the optimal investment portfolio.

Proposal	Profitability Index	Initial Capital Investment
7	1.14	$400,000
3	1.13	200,000
5	1.11	300,000
4	1.05	250,000

Solution. The objective is to find that combination of investment proposals that provide the highest net present value to the firm. There are three primary combinations:

Alternative No. 1

Proposal 7: $400,000(1.14 – 1.0) = $56,000 = NPV

Alternative No. 2

Proposal 3: $200,000(1.13 – 1.0) = $26,000
Proposal 5: $300,000(1.11 – 1.0) = 33,000
NPV $59,000

Alternative No. 3

Proposal 3: $200,000(1.13 – 1.0) = $26,000
Proposal 4: $250,000(1.05 – 1.0) = 12,500
NPV $38,500

This solution shows that alternative No. 2 (proposal 3 and 5) should be chosen since the NPV to the firm is maximized with the selection of these proposals. The reason is that more of the available budget is utilized with this combination of proposals, even though a more profitable individual proposal was available to the firm.

This example also illustrates the importance of initial capital outlays when functioning under the constraints of capital budgeting. Implied in the foregoing example is the assumption that uninvested capital has a NPV = 0. This is the same as assuming it is placed in an investment that has a yield equal to the required rate of return. If it cannot be reinvested such that NPV is equal to 0, then full utilization of the investment capital available is even more important.

6.5.4.3 Incremental (Marginal) Analysis

In the preceding section it was noted that one of the main principles that should apply when comparing mutually exclusive investment proposal alternatives is that each increment of investment capital must justify itself. This is an aspect that is often overlooked in many analyses, but one that is fundamental to the capital budgeting problem—particularly under capital-rationing constraints.

Incremental or marginal analysis is a technique that can help the evaluator choose between mutually exclusive projects having unequal investments. The concept is to

1. Calculate the differential investments and annual cash flows between the projects.
2. Compare the calculated rate of return on the differential cash flows with the required rate of return.

If this rate exceeds the required rate, the additional incremental investment is justified.

Example 6.5.11. Suppose the following cash flow estimates represent four mutually exclusive investment proposals. If the firm's required rate of return is 15%, which proposal should be chosen?

Year	Cash Flows Proposal A	Proposal B	Proposal C	Proposal D
0	$ — 12,000	$ — 15,000	$ — 19,000	$—21,000
1	3,000	3,700	4,200	4,600
↓	3,000	3,700	4,200	4,600
	3,000	3,700	4,200	4,600
	3,000	3,700	4,200	4,600
	3,000	3,700	4,200	4,600
10	3,000	3,700	4,200	4,600
IRR (%)	21.4	21.0	17.8	17.5
NPV (at 15%)	3,056	3,569	2,079	2,086

Solution. The IRR for all the proposals exceeds the 15% required rate of return, and, therefore, each would be acceptable to the firm. However, if only one proposal is required, the IRR criterion suggests that proposal A is superior.

At this point it is necessary to perform a rate of return calculation on each increment to determine if the incremental proposal investments can be justified. A comparison between proposals A and B shows:

$$\text{B/A:}(P/A,r\%,10) = \frac{15{,}000 - 12{,}000}{3700 - 3000} = 4.2857$$

$$r = 19.36\%$$

This indicates that proposal B is preferred to proposal A because the return on the incremental investment of $3000 exceeds 15%. The next comparison is between proposals C and B.

$$\text{C/B:}(P/A,r\%,10) = \frac{19{,}000 - 15{,}000}{4200 - 3700} = 8.00$$

$$r = 4.2870$$

This comparison indicates that proposal C should be eliminated because the return on the incremental investment of $4000 is less than the required rate of 15%. The last comparison is between proposals D and B.

$$\text{D/B:}(P/A,r\%,10) = \frac{21{,}000 - 15{,}000}{4600 - 3700} = 6.6667$$

$$r = 8.14\%$$

The final comparison indicates that the rate of return on this incremental investment is also less than the required rate, and therefore proposal B is the final choice.

In part 6.5.3.3 of this chapter, a comparison was made between the IRR and NPV criteria with respect to inconsistent rankings of investment proposals. The simple example used to demonstrate this feature was as follows:

Year	Annual Cash Flows ($1000s) Project A	Project B
0	$–10	$–10
1	1	7
2	5	5
3	6	3
4	7	1
IRR (%)	24.7	30.5
NPV (15%)	2.60	2.41
Required rate of return:	15%	

If an incremental analysis were performed on this example, the following would result:

Year	Incremental Cash Flows, A-B ($1000s)
0	0
1	−6
2	0
3	3
4	6

The IRR that equates $–6 at the end of year 1 with $3 and $6 at the end of years 3 and 4, respectively, is 16.54%. Because this rate exceeds the required rate of return of 15%, project A should be selected, even though project B has the larger IRR.

It is interesting and important to note that both of these examples illustrate situations where the proposal with the largest IRR is not necessarily the best proposal, when mutually exclusive proposals are being considered. Proposal choice in the mutually exclusive case is, of course, dependent upon the required rate of return and the associated reinvestment rate assumption discussed earlier in this chapter. The incremental analysis illustrated in both examples resulted in choosing investment proposals with the highest net present values. These proposals could have been selected simply by comparing NPV values initially. Therefore, it is possible to generalize and state that the internal rate of return and net present value methods give the same results in capital budgeting problems, if *incremental analysis* is used on mutually exclusive projects.

6.5.5. HANDLING RISK

Although the preceding sections of this chapter describe the type of evaluations frequently performed in the mining industry, it has been assumed that input data are known with certainty—clearly an erroneous simplification. In reality, estimates of ore grade, mining cost, metal price, etc., are subject to varying degrees of uncertainty due to the inability to predict the future with much precision.

Risk, in the context of this discussion, is the unforeseen deviation of individual cash flows from expected values for a capital project. For a mining venture, the source of this uncertainty could be any number of factors relating to items such as ore grade, ore reserve tonnage, operating costs, product prices, etc. With conventional deterministic evaluations, a point estimate of each of these factors is made. Subsequent operating results usually reveal these estimates to be in error, thus giving rise to different cash flows than expected.

A firm can take uncertainty into account in many ways. A discussion of some of the more common traditional methods follows.

6.5.5.1 Simplified Approaches

One method used to compensate for risk is to demand shorter payout periods for riskier projects. Aside from the shortcomings of the payout period as a capital investment criterion, the technique encounters additional problems here. Payout period does not measure risk directly; in fact, it is a rather poor measure of profitability. Risk in the early years of a project (when cash inflows are most important) is not affected by the payout period. Risk is considered only by ignoring cash flows after the payout period. By adjusting the maximum acceptable payout period, risk may be resolved sooner, but it will not be diminished.

Difficulty arises with risk-adjusted payout periods as to how much adjustment to make for a particular project. One might agree that the maximum acceptable payout period for a large new mine in a politically turbulent part of the world should be less than for a similar investment in the United States. The question is, of course, "How much less?" for which there is no valid, objective answer. Subjective decisions are required and considerable arbitrariness can creep into the decision. The decision maker can select any course of action he or she pleases simply by assigning prohibitive maximum acceptable payout periods for all other alternatives.

As supplementary information, the payout period is useful in studying the uncertainty in an investment project. Risk-adjusted payout period, however, is a poor primary method for accounting for risk in capital projects.

Risk Adjusted Discount Rate: It is very common for firms to require risky projects to have higher rates of return on investment than safer ones. For example, a firm might establish three classes of capital projects and associated rates of returns as follows:

Class 1: Replacement of equipment in an ongoing operation. Market is known, technology is proven, so that risk is fairly low. Maybe a discount rate of 10% is acceptable here.

Class 2: Expansion of present mine or plant facilities. Many technical problems have already been solved, but there may be a question in marketing. Can the additional output be sold at pre-expansion prices? The added risk here may indicate that a higher discount rate is in order, say, 15%.

Class 3: Opening of a new operation, entering a new market, etc. Here the sources of uncertainty are many, justifying, perhaps, 20% as the minimum acceptable rate of return on such projects.

The magnitudes of the discount rates used in the foregoing example are not important; only the concept of risk-adjusted discount rates is being demonstrated.

The major drawback of this method is the same as with risk-adjusted payout period—subjective establishment of hurdle values. There is no method for assigning acceptable risk-adjusted rates of return to individual projects in an objective manner, so inconsistency is inevitable. Nonetheless, the method is easy to apply and is not likely to disappear soon.

Risk-Adjusted Input Parameters: When considering the extensive uncertainty encountered in evaluating new copper mining ventures, industry executives often compensate for this risk by using very low prices for copper. Similarly, conservative values for other input parameters, such as mining costs, ore grade, etc., during the evaluation stage can screen out all but the best projects. Of course, by being excessively conservative, all projects can be rejected, and this is a real danger with risk-adjusted input parameters. When more than one variable is adjusted, compounded conservatism can easily reject nearly any project. Also subjectivity in quantifying the risk-adjusted variables causes the same type of problems as discussed previously.

6.5.5.2 Sensitivity Analysis

The term "sensitivity analysis" simply describes the process of determining the sensitivity of the results of the project to changes in any one input variable. For example, how sensitive is the project NPV (or IRR, etc.) to changes in, say, ore grade? Only one variable is changed in each iteration to isolate the impact of that variable. In the subsequent section on probabilistic risk analysis, this restriction is relaxed, as simultaneous changes are permitted in several variables.

Sensitivity analysis is a very simple approach to studying project risk. It is intuitively appealing and is, therefore, very widely used. Often, after identifying the most important variables, many firms extend the method by combining changes in two or three variables at once to study the combined impacts. Another common extension of the method is to compute an "upside case" combining favorable values for the key variables, and—more important—a "downside" case where adverse values of several important variables occur simultaneously.

Sensitivity results are frequently plotted in simple graphical form to illustrate the risk attributable to each variable.

A number of sensitivity analyses have been calculated for the Bullion Mining Co. example, discussed in 6.5.6. The reader should work through one or two annual cash flows to become comfortable with the procedure.

6.5.5.3 Probabilistic Risk Analysis

In capital intensive mining projects where investments exceeding $100 million are common, treating uncertainty exclusively with one of the arbitrary methods or with sensitivity analysis may be unsatisfactory. More quantitative data relevant to the sources and magnitude of risk can be developed, and, with such large sums at stake, additional analysis is certainly justified. Rather than using "ignorance factors" to discount risk, an attempt should be made to identify specific sources and estimate the magnitude of this risk. By quantifying risk in this manner, it is possible to estimate the chance of achieving a given level of profitability with a specific project. Management then has much more information at its disposal, so that an intelligent assessment can be made between risk and expected profits.

Stochastic Risk Analysis Models: The general method by which the level and magnitude of risk associated with capital projects in mining is determined is to establish probability distributions for the input variables rather than treating each of these parameters as being known with certainty. For example, in the deterministic case, an ore grade of 3.4% Zn may be used, when it is clear that in reality, the ore grade may be as low as 2.5% Zn or as high as 4.7% Zn. This possible range of values for ore grade can now be recognized by establishing a probability distribution for ore grade. Furthermore, with recent advances in geostatistical methods of ore reserve estimation, analysts no longer need rely on subjective probability estimates by experts to establish these distributions.

The probability distributions for the input variables can be assumed to be either discrete or continuous, the latter usually being the more realistic case.

In the case of *discrete probability distributions,* and with the aid of a computer, the analyst can easily compute an exhaustive set of all possible outcomes, along with the respective probability of occurrence for each. This probability distribution of outcomes can then be examined to quantify the level of risk associated with the project. For example, if the firm's minimum acceptable IRR is 15%, the distribution of outcomes can provide the probability that the IRR > 15%, or the Pr[IRR < 12%], and so forth. For an example illustrating the use of discrete probability distributions for risk analysis, see Gentry and O'Neil (1984).

With *continuous probability distributions* for input variables, the process is directly analogous. That is, values for each input variable are selected from individual continuous probability distributions and the IRR, or some other investment criterion, is determined. Here, however, because continuous probability distribution functions are used, there are an infinite number of possible outcomes, so an exhaustive set of outcomes can not be calculated.

To overcome this obstacle, a system of random, or Monte Carlo, sampling can be employed to choose values for individual variables in calculating a value for the investment criterion. If this process is repeated many times, a continuous probability distribution of possible outcomes will emerge. In all but the most trivial examples of this type of risk analysis, computer assistance is required due to the large number of iterations usually required before a stable distribution of outcomes is created. Once the distribution is created, however, project risk is again quantified, and probabilities of exceeding various levels of profitability can be easily determined. (Again the reader is referred to Gentry and O'Neil, 1984, for further discussion and examples of probabilistic risk analysis.)

The field of risk analysis of capital investments has advanced far beyond the fundamental approaches described in the foregoing. In particular, greater attention has been given to interdependent variables and events. That is, the system described above assumed independence of input variables and either independence or perfect dependence of project performance over time. For example, ore head grade and metallurgical recovery are two variables that frequently are interdependent. Therefore, independent random samples of each variable might yield incorrect results. Furthermore, project performance from period to period also rarely is independent. O'Hara (1982) has suggested one approach to solving this problem in mining evaluations. However, although an active research area, risk analysis in general has not gained wide acceptance in the mining industry, and therefore further discussion here is not warranted.

6.5.5.4 Competitive Cost Position

Generally, the most crucial variable in a mining evaluation is the one least amenable to accurate prediction: commodity price. Many mineral products are priced daily in world markets, and such price movements are influenced by a wide variety of unpredictable stimuli. Most mine owners have learned a painful lesson with mines constructed in the anticipation of rising commodity prices. Unless those mines are low cost producers, cyclical troughs in the price cycle can produce disastrous results even though the long-run price trend may be upward.

As a consequence of the relative unpredictability of mineral prices, many mining firms stress competitive cost analysis as the most important risk analysis technique. The basic theory is that only by being a relatively low-cost producer can a miner withstand market downturns. Other higher cost mines should, therefore, be forced to curtail production earlier in a falling market, thus reducing supply and helping restore satisfactory prices.

Most major companies pay great attention to the position a prospective new mine occupies on the world competitive cost curve. If its production cost per unit of salable product is in the upper 50% of all production, it is best to reexamine the project carefully, even if the current spot price for the commodity suggests a satisfactory ROI. Unless marketing arrangements assure a commodity price sufficiently high to protect the project in down markets, mine investors should strive for production costs in the lower 50% of all producers.

6.5.6 EXAMPLE EVALUATION

A prospective gold-mining operation, Bullion Mining Co., has been constructed to illustrate the evaluation techniques described in Chapter 6.5. This hypothetical but realistic example should be studied closely by the reader to understand the investment analysis methods demonstrated.

The following financial model (Example 6.5.12) represents the evaluation of a gold deposit. The assumption is that the deposit will be open pit mined. The gold ore then would be put through a crushing and grinding circuit before treatment in a carbon-in-leach (CIL) process to extract the gold. The final product would be gold doré, which would be shipped to a gold refinery where pure gold bars are produced for sale on the gold bullion markets.

The model assumes a mining reserve of 3.6 million tons @ 0.133 oz/ton gold comprising three separate deposits, A, B, and C. The reserves of the deposits are A: 1.1 million tons @ 0.145 oz/ton gold and stripping ratio of 9.2:1 (waste:ore); B: 1.7 million tons @ 0.136 oz/ton gold and stripping ratio of 12.3:1; C: 0.8 million tons @ 0.110 oz/ton gold and a stripping ratio of 13.5:1. It is assumed that 25% of the waste overlying the ore would be prestripped before any ore is mined. The cost of this is capitalized. The CIL process recovers 93% of the gold from the ore. The mining and processing operating costs are detailed in the model, along with the capital costs of constructing the mine. The mining of ore is assumed to be performed by a contract mining company so that the capital cost of development is reduced, while the cost of mining is comensurately increased. The mine is assumed to be isolated and requires a small camp to house the personnel in single accommodation while working at the mine and transporting the personnel to a town for their days off. The fixed administration cost allows for the cost of general administration (e.g. accounting, purchasing, and stores) and the operating cost of a single accommodation camp and transport of personnel in and out of the site.

The sensitivity analysis considered three variables: gold price, capital cost, and tax rate level. The project's viability is most sensitive to the price of gold, which relates directly to the revenue of the operation. A similar effect on the viability is caused by variations in the grade of the ore, tonnage throughput, and recovery of gold. The revenue equation is

$$\text{Revenue} = \text{Throughput} \times \text{Ore Grade} \times \text{Recovery} \times \text{Gold Price} \tag{6.5.6}$$

The other major variables in the evaluation are operating costs and the total mining reserve. The effect of a change in stripping ratio will change the quantity of material mined and so directly affect the operating cost.

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Example 6.5.12. Bullion Mining Co. Preliminary Economic Analysis, CIL Gold Plant, 450,000 tpy (All figures in $1000s unless stated)

			$1000s
Net present value	@ 10.0%		17038.6
	15.0%		11184.8
	20.0%		6932.6
	25.0%		3790.7
Benefit/cost ratio	@ 15%	=	1.145
Rate of return	33.9%		
Payout period	2.35 years		

	Total	$/oz	Total gold recovered	445,214 oz
Capital cost	24904	56		
Operating cost	107437	241		
Total	N/A	297		

Assumptions

Production rate	450,000 tpy (000s)		
Mining costs	Ore, $/ton	Waste, $bcy	Mine capacity (000 tpy)
Deposit A	4.00	2.10	300
Deposit B	4.00	2.10	250
Deposit C	4.00	2.10	350
Processing costs	12.75 $/ton		
Administration costs-fixed	1500.0 $/yr (000s)		
Plant recovery	93.0%		
Refinery recovery	99.05%		
Gold price	400 $/oz		
Taxation rate	34%		
Depreciation	Unit of production method		
Waste density	2.0 tons per bcy		

Example 6.5.12.—cont.
Bullion Mining Co. Preliminary Economic Analysis, CIL Gold Plant, 450,000 tpy (All figures in 1000 tons unless stated)

Production Schedule

Year	0	1	2	3	4	5	6	7	8	9	10	Total
Deposit A												
Tons	0	300	300	300	200	0	0	0	0	0	0	1100
Grade, oz/t	0.0	0.161	0.151	0.138	0.122	0.0	0.0	0.0	0.0	0.0	0.0	0.145
Contained gold, oz	0	48,226	44,533	41,474	24,435	0	0	0	0	0	0	159,468
Waste mined, 1000 bcy	0	1,468	1,104	983	493	0	0	0	0	0	0	4,048
Deposit B												
Tons	0	150	150	150	250	250	250	250	250	0	0	1700
Grade, oz/t	0.0	0.138	0.135	0.132	0.145	0.148	0.135	0.132	0.122	0.0	0.0	0.136
Contained gold, oz	0	20,737	20,255	19,773	36,170	36,973	33,758	32,954	30,543	0	0	231,164
Waste mined, 1000 bcy	0	982	738	900	1,230	1,230	1,230	1,230	824	0	0	8,364
Deposit C												
Tons	0	0	0	0	0	200	200	200	200	0	0	800
Grade, oz/t	0.0	0.0	0.0	0.0	0.0	0.113	0.103	0.109	0.116	0.0	0.0	0.110
Contained gold, oz	0	0	0	0	0	22,506	20,576	21,862	23,149	0	0	88,093
Waste mined, 1000 bcy	0	0	0	0	356	1,080	1,080	1,080	724	0	0	4,320
Total												
Tons	0	450	450	450	450	450	450	450	450	0	0	3,600
Grade, oz/t	0.0	0.153	0.146	0.136	0.135	0.132	0.121	0.122	0.119	0.0	0.0	0.133
Contained gold, oz	0	68,963	65,587	61,247	60,604	59,479	54,335	54,817	53,692	0	0	478,724
Waste mined, 1000 bcy	0	2,450	1,842	1,883	2,080	2,310	2,310	2,310	1,548	0	0	16,732

Example 6.5.12.—cont.
Bullion Mining Co. Preliminary Economic Analysis, CIL Gold Plant, 450,000 tpy (All figures in 1000 tons unless stated)

Ore Reserve Depletion Schedule

Year	0	1	2	3	4	5	6	7	8	9	10	Total
Deposit A												
Beginning reserve	1100.0	1100.0	800.0	500.0	200.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Ore mined	0.0	300.0	300.0	300.0	200.0	0.0	0.0	0.0	0.0	0.0	0.0	1,100.0
Mine capacity	0.0	300.0	300.0	300.0	300.0	300.0	300.0	300.0	300.0	300.0	300.0	
Remaining reserve	1100.0	800.0	500.0	200.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Waste mined, 1000 bcy		1468.3	1104.0	982.6	493.1	0.0	0.0	0.0	0.0	0.0	0.0	4,048.0
Prestripping, 1000 bcy	1012.0											1,012.0
Deposit B												
Beginning reserve	1700.0	1700.0	1550.0	1400.0	1250.0	1000.0	750.0	500.0	250.0	0.0	0.0	0.0
Ore mined	0.0	150.0	150.0	150.0	250.0	250.0	250.0	250.0	250.0	0.0	0.0	1,700.0
Mine capacity	0.0	250.0	250.0	250.0	250.0	250.0	250.0	250.0	250.0	250.0	250.0	
Remaining reserve	1700.0	1550.0	1400.0	1250.0	1000.0	750.0	500.0	250.0	0.0	0.0	0.0	0.0
Waste mined, 1000 bcy		981.5	738.0	900.4	1230.0	1230.0	1230.0	1230.0	824.1	0.0	0.0	8,364.0
Prestripping, 1000 bcy	2091.0											2,091.0
Deposit C												
Beginning reserve	800.0	800.0	800.0	800.0	800.0	800.0	600.0	400.0	200.0	0.0	0.0	0.0
Ore mined	0.0	0.0	0.0	0.0	0.0	200.0	200.0	200.0	200.0	0.0	0.0	800.0
Mine capacity	0.0	350.0	350.0	350.0	350.0	350.0	350.0	350.0	350.0	350.0	350.0	
Remaining reserve	800.0	800.0	800.0	800.0	800.0	600.0	400.0	200.0	0.0	0.0	0.0	0.0
Waste mined, 1000 bcy		0.0	0.0	0.0	356.4	1080.0	1080.0	1080.0	723.6	0.0	0.0	4,320.0
Prestripping, 1000 bcy					1080.0							1,080.0
Total												
Beginning reserve	3600.0	3600.0	3150.0	2700.0	2250.0	1800.0	1350.0	900.0	450.0	0.0	0.0	0.0
Ore mined	0.0	450.0	450.0	450.0	450.0	450.0	450.0	450.0	450.0	0.0	0.0	3,600.0
Mill capacity	0.0	450.0	450.0	450.0	450.0	450.0	450.0	450.0	450.0	450.0	450.0	
Remaining reserve	3600.0	3150.0	2700.0	2250.0	1800.0	1350.0	900.0	450.0	0.0	0.0	0.0	0.0
Waste mined, 1000 bcy	0.0	2449.9	1842.0	1882.9	2079.5	2310.0	2310.0	2310.0	1547.7	0.0	0.0	16,732.0
Prestripping, 1000 bcy	3103.0	0.0	0.0	0.0	1080.0	0.0	0.0	0.0	0.0	0.0	0.0	4,183.0

Example 6.5.12.—cont.

Bullion Mining Co. Preliminary Economic Analysis, CIL Gold Plant, 450,000 tpy (All figures in $1000s unless stated)

Capital and Operating Cost Schedule

Year	0	1	2	3	4	5	6	7	8	9	10	Total
Operating costs												
Mining ore	0.0	1,800.0	1,800.0	1,800.0	1,800.0	1,800.0	1,800.0	1,800.0	1,800.0	0.0	0.0	14,400.0
Mining waste	0.0	5,144.7	3,868.2	3,954.1	4,367.0	4,851.0	4,851.0	4,851.0	3,250.2	0.0	0.0	35,137.2
Milling	0.0	5,737.5	5,737.5	5,737.5	5,737.5	5,737.5	5,737.5	5,737.5	5,737.5	0.0	0.0	45,900.0
Administration	0.0	1,500.0	1,500.0	1,500.0	1,500.0	1,500.0	1,500.0	1,500.0	1,500.0	0.0	0.0	12,000.0
Total Op. Cost	$0.0	$14,182.2	$12,905.7	$12,991.6	$13,404.5	$13,888.5	$13,888.5	$13,888.5	$12,287.7	$0.0	$0.0	$107,437.2
Capital Expenditure												
O/C contractor mob.	120.0											120.0
O/C prestripping	6,516.3	0.0	0.0	0.0	2,268.0	0.0	0.0	0.0	0.0	0.0	0.0	8,784.3
Dewatering	150.0											150.0
Plant constn.	10,900.0											10,900.0
Infrastructure	3,100.0											3,100.0
Exploration	100.0	150.0	200.0	200.0								650.0
Replacement capital			100.0	250.0	250.0	250.0	250.0	100.0				1,200.0
												0.0
Total Cap. Exp.	$20,886.3	$150.0	$300.0	$450.0	$2,518.0	$250.0	$250.0	$100.0	$0.0	$0.0	$0.0	$24,904.3
Working capital		1,016.4	−49.8	−64.0	−9.5	−16.6	−75.8	7.1	−16.6	−791.3	0.0	0.0

Note: Working capital is calculated on 4% of the yearly change in gross revenue.

Revenue Statement

Year	0	1	2	3	4	5	6	7	8	9	10	Total
Gold Sales												
Oz	0	64,136	60,996	56,960	56,362	55,315	50,531	50,980	49,933	0	0	445,214
Price $/oz	400	400	400	400	400	400	400	400	400	400	400	400
NSR %	99.1	99.1	99.1	99.1	99.1	99.1	99.1	99.1	99.1	99.1	99.1	99.1
Gold Revenue, $	0.0	25,410.6	24,166.8	22,567.5	22,330.6	21,915.9	20,020.5	20,198.2	19,783.6	0.0	0.0	176,393.6

Example 6.5.12.—cont.

Bullion Mining Co. Preliminary Economic Analysis, CIL Gold Plant, 450,000 tpy (All figures in $1000s unless stated)

Depreciation and Amortization Schedules

Year	0	1	2	3	4	5	6	7	8	9	10	Total
Unrecovered capital	5,000.0	25,886.3	22,781.8	19,784.4	16,862.0	15,504.0	11,815.5	8,043.7	4,071.8	0.0	0.0	
Capital expenditure	20,886.3	150.0	300.0	450.0	2,518.0	250.0	250.0	100.0	0.0	0.0	0.0	24,904.3
Deprec. cap. base	25,886.3	26,036.3	23,081.8	20,234.4	19,380.0	15,754.0	12,065.5	8,143.7	4,071.8	0.0	0.0	
Deprec. & amort	0.0	3,254.5	3,297.4	3,372.4	3,876.0	3,938.5	4,021.8	4,071.8	4,071.8	0.0	0.0	
Cum. capital exp.	20,886.3	21,036.3	21,336.3	21,786.3	24,304.3	24,554.3	24,804.3	24,904.3	24,904.3	24,904.3	24,904.3	

Note: Unrecovered capital in year 0 is exploration expenditure incurred in previous years.

Profit & Loss and Cash Flow Statements

Year	0	1	2	3	4	5	6	7	8	9	10	Total
Revenues	0.0	25,410.6	24,166.8	22,567.5	22,330.6	21,915.9	20,020.5	20,198.2	19,783.6	0.0	0.0	176,393.6
Operating costs	0.0	14,182.2	12,905.7	12,991.6	13,404.5	13,888.5	13,888.5	13,888.5	12,287.7	0.0	0.0	107,437.2
Operating income	0.0	11,228.4	11,261.1	9,575.9	8,926.1	8,027.4	6,132.0	6,309.7	7,495.9	0.0	0.0	68,956.4
Deprec & amort	0.0	3,254.5	3,297.4	3,372.4	3,876.0	3,938.5	4,021.8	4,071.8	4,071.8	0.0	0.0	29,904.3
Royalty	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Depletion	0.0	3,557.5	3,383.3	3,101.7	2,525.0	2,044.5	1,055.1	1,118.9	1,712.0	0.0	0.0	18,498.1
Taxable income	0.0	4,416.4	4,580.3	3,101.7	2,525.0	2,044.5	1,055.1	1,118.9	1,712.0	0.0	0.0	20,554.0
Tax rate	34%	34%	34%	34%	34%	34%	34%	34%	34%	34%	34%	
Taxes due	0.0	1,501.6	1,557.3	1,054.6	858.5	695.1	358.7	380.4	582.1	0.0	0.0	6,988.4
Net income	0.0	2,914.8	3,023.0	2,047.1	1,666.5	1,349.3	696.4	738.5	1,129.9	0.0	0.0	13,565.6
Cash flow	−20,886.3	8,560.4	9,453.5	8,135.2	5,559.0	7,098.9	5,599.1	5,822.1	6,930.4	791.3	0.0	37,063.8
Cum. cash flow	−20,886.3	−12,325.9	−2,872.4	5,262.9	10,821.9	17,920.8	23,519.9	29,342.0	36,272.4	37,063.8	37,063.8	

Note: Tax treatment of working capital has been simplified in this example as it has been considered as a nontax deductable item.

Bullion Mining Co. Preliminary Economic Analysis, CIL Gold Plant, 450,000 tpy (All figures in $1000s unless stated)

Sensitivity Analysis

Sensitivity 1			Sensitivity 2			Sensitivity 3		
Gold Price	NPV @15%	IRR	Income Tax	NPV @15%	IRR	Investment Capital	NPV @15%	IRR
350	2,237.9	19.1%	29.0%	11,760.4	34.9%	85.0%	13,793.3	41.8%
360	4,110.0	22.4%	30.0%	11,645.3	34.7%	87.5%	13,360.5	40.3%
370	5,939.7	25.5%	31.0%	11,530.2	34.5%	90.0%	12,927.6	38.9%
380	7,733.4	28.5%	32.0%	11,415.0	34.3%	92.5%	12,494.7	37.6%
390	9,462.9	31.2%	33.0%	11,299.9	34.1%	95.0%	12,061.8	36.3%
400	11,184.8	33.9%	34.0%	11,184.8	33.9%	97.5%	11,624.4	35.1%
410	12,878.2	36.5%	35.0%	11,069.7	33.7%	100.0%	11,184.8	33.9%
420	14,560.4	39.0%	36.0%	10,954.5	33.5%	102.5%	10,745.2	32.8%
430	16,241.8	41.5%	37.0%	10,839.4	33.3%	105.0%	10,305.7	31.7%
440	17,889.9	43.9%	38.0%	10,724.3	33.1%	107.5%	9,866.1	30.7%
450	19,537.9	46.2%	39.0%	10,609.1	32.9%	110.0%	9,426.5	29.7%
						112.5%	8,987.0	28.7%
						115.0%	8,547.4	27.8%

Chapter 6.6
MINE FINANCING

C. Richard Tinsley

For a mining engineer absorbed in technical and day-to-day managerial responsibilities, the world of mine finance often seems remote and somewhat mystical. This chapter illustrates that the factors requiring consideration in financing are stunningly simple, although variations, combinations, and synthetic constructions evolve each year and must be checked to ascertain if any offers unique qualities that make financing cheaper and/or easier.

The market sources for money vary from seed investors and tax-shelter monies gambled on the potentially high returns from minerals exploration through to banks and insurance companies that only want to deal with ongoing, cash-rich companies where they feel secure about repayment. An important concept to always remember is that the financier's expected return is proportional to the riskiness of the investment.

Much of this chapter draws heavily from the book, *Finance for the Minerals Industry,* edited by Richard Tinsley, Mark Emerson, and Randy Eppler and published by SME (1985). This reference volume is recommended for further reading on the topics presented in this chapter.

6.6.1 FINANCIAL OBJECTIVES OF A MINING COMPANY

For simplicity, a mining company is assumed to evolve through three stages: an exploration company or syndicate, a development company, and an operating company.

The *exploration company/syndicate* continually seeks risk monies, while the promoters try to retain as much of the equity and as little of the future financing obligations as possible. In general, there are two types of exploration companies: those dedicated to finding and developing a mine and those, usually managed by geologists, that simply develop mineral prospects to an appealing stage and then sell out to reinvest in other exploration.

The *development company* usually has one or two interesting exploration successes, each of which requires financing for further drilling and feasibility studies before actual mine development can be financed. Some junior companies specialize in redeveloping defunct mines or restructuring troubled operations as a means to finance mine development.

Operating companies often fund exploration from internally generated cash flow or they use their financial clout to joint venture the development of a property. An operating company normally has a very different set of financing objectives. Revenue needs to be hedged in the volatile commodity and currency markets. Flexible working capital financing for labor and supplies—and perhaps also to cover stockpiles, freight, and distribution costs—has to be available. Capital costs require servicing, including interest and principal on debt obligations, dividends paid to the suppliers of equity, and the accounting for depletion to reflect replacement of the original ore body. In addition, the government's share via taxes and infrastructure charges, together with fixed costs of administration and most of a mine's labor charges, combine to diminish the financial margin available for future exploration, plant expansions, and replacement equipment.

6.6.1.1 Funding Decision: Capital Structure

A number of theories exist on the best capital structure for a company. Van Horne (1977) summarizes the necessary balance of equity and borrowed money as follows:

"The optimal capital structure for a company includes both equity and debt. Since the interest on borrowed money is deductible from profits for tax purposes, using someone else's money creates an equivalent lower capital cost for the same item purchased."

Besides tax reduction, debt leverages the return on equity which is usually the objective of those in control of the company (i.e., the shareholders and the management team). However, leverage cuts both ways. Van Horne goes on to state that:

"As the ratio of borrowed money to equity money increases, the interest payments and the rate of interest itself become so large that profits may disappear entirely and bankruptcy threatens. The latter situation has an associated cost since assets may have to be sold at distressed prices and lawyer's fees may be substantial in any bankruptcy proceeding."

The exploration company has a capital structure consisting largely of equity or tax-sheltered funds. *Equity* can be raised from individuals, companies, the stock market, occasionally governments and multilateral agencies, investment funds and partnerships, and through the provision of services. *Tax-shelter monies* usually pass the company's tax deductions from exploration and drilling to the investors, either directly or through share/trust structures.

Table 6.6.1 Exploration Company Balance Sheet

	$000
Current Assets	$250
Current Liabilities	(10)
Working Capital	$240
Mineral Properties	2250
Plant and Equipment	50
Shareholders Equity	$2540

When looking at the general formula for a company's balance sheet, one can see that an exploration company's *assets* are basically equal to the equity invested to date, as illustrated in Table 6.6.1. Langdon (1985) offers the following representation of this relationship:

$$\text{Assets} = \text{Liabilities} + \text{Equity}$$

Liabilities are the creditors' rights to assets, and equity is the owners' rights to those assets. When a company begins to the develop mineral projects, the relationship of liabilities to equity (the capital structure) starts to assume increasing importance. The effect of leverage is shown in the following table, in which 50% tax applies, and the interest cost on debt is 10% per annum:

	No Debt				50% Debt			
Return on assets	5%	−10%	−25%	−50%	−5%	−10%	−25%	−50%
Return on equity	2.5%	−5%	−12.5%	−25%	−0%	−5%	−20%	−45%

Carraghiaur (1982) divides up the mine predevelopment and financing process into two stages: delineation drilling and underground exploration/bulk sampling.

Table 6.6.2 illustrates the transition during the development phase towards the use of financial leverage in a company's capital structure. The first critical decision is when to finance a mine development. It is often said that "mines are made, not found" and the development financing is often the reason. However, the availability and relative cost of each financing source changes according to economic conditions. In times of a bear stock market, the effective use of a public stock offering becomes extremely difficult as a vehicle for raising money. At the trough of commodity price cycles, major mining companies often withdraw completely from joint venturing and slash their own exploration budgets. The challenge is to identify which sources of investment financing are currently or prospectively available and the information and development structure that appeals to each.

A mining engineer's generalist background can often be an advantage in this regard since he or she has (or should have) a better idea than most of the value of the project or company (i.e., the intangible or potential worth that cannot be represented adequately in a financial pro-forma projection). He/she also knows, or should know, the technical risks involved. Two projects may look the same on paper, but a good financier must have an accurate sense of when to take whatever development financing can be obtained, based on the availability of funding sources, and of when to hold out based on his/her evaluation of future developments (Cruft, 1985).

These mine development financing decisions are important to the genesis of the capital structure of the operating company. This is summarized in Table 6.6.3.

A theoretical relationship between asset structure and overall capital structure is called the *matching principle,* which simply states that one should finance short-term needs with short-term sources and long-term needs with long-term sources (Langdon, 1985). As can be seen in Fig. 6.6.1, the entity is deemed to be a growing concern, and the growth in current assets requires permanent financing. Also the variations in current assets are those matched with short-term financing.

It is these variations in future cash flows that all financiers attempt to model. Bankers want to see that interest and principal repayments can be met during troughs in project earnings, and investors want to see that adequate dividends will emerge in better times. As a rule of thumb, miners producing materials tradable on a commodity exchange can expect a capital structure of 50% equity:50% debt for very cyclical commodities, unless the debt itself is correlated to that commodity (e.g., a gold loan for a gold mine). A mine with highly predictable cash flows—one keyed to a strong sales contract with take-or-pay provisions (now rare) and escalation clauses—perhaps can move its leverage up to 90 to 100%. A mine with cash flows that are difficult to predict, as with many industrial minerals, can expect to attract very little traditional debt financing.

Financial theorists often disagree on whether an optimal capital structure exists for an operating company. Langdon (1985) and Gentry and O'Neil (1984) summarize the three ap-

Table 6.6.2. Financing Alterations during Predevelopment

Finance Sources	Delineation Drilling	Underground Exploration Bulk Sample	Mine Development
Investment/tax partnerships	X		
Venture Capital	X	X	
Major Mining Companies	X	X	X
Public Markets		X	X
Banks			X
Leasing Companies			X

Source: Carraghiaur. 1985.

Table 6.6.3. Components of Capital Structure

Assets	=	Liabilities	+	Equity
Current Assets Cash Marketable securities Accounts receivable Inventory		Current Liabilities Accounts payable Notes payable Current portion of long-term debt Taxes Lease payments		Common Shares Preferred shares Share premium above par Retained earnings
Long-Term Assets Mine/plant Buildings Equipment Land Mineral rights Ore reserves		Long-term liabilities Bank loans Bonds Debentures Long-term notes Leases Subordinated debt		Subordinated debt?

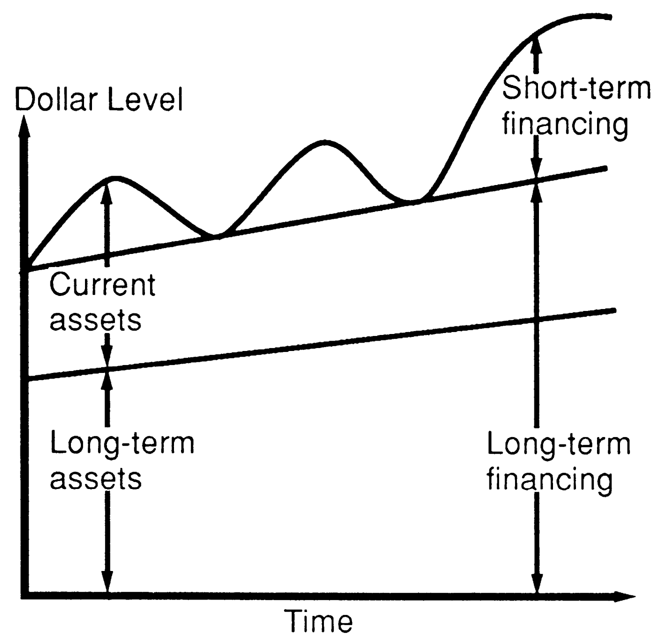

Fig. 6.6.1. Matching principle for financing assets.

proaches normally advocated on this topic—net income, net operating income, and traditional.

The *net income* and *net operating income* approaches assume that debt proceeds are used to repurchase common shares. Most mining companies seem to have better uses for their debt proceeds although major companies have recently adopted this approach as a defensive measure against corporate raiders awash with leveraged buy-out funds. The *traditional method* assumes that the optimal structure occurs just before the leverage level starts to raise the corporation's cost of shares and debt instruments because a threshold of perceived risk has been crossed—usually at or near the industry average debt:equity ratio.

6.6.1.2 Cost of Capital

The corporate cost of capital is discussed in detail in Chapter 6.2 and by Schenck (1985) in the same breadth as discount rates for project valuation and selection. Schenck gives a good example for calculating the cost of capital for an operating company.

Example 6.6.1. Calculation of weighted cost of capital.

Finance Source	% of New Capital	cost, % per annum	After-Tax cost
Bank loans, unsecured	15%	16%	9.6%
Mortgage bond	20%	14%	8.4%
Equipment leases	15%	16%	9.6%
Preferred shares, new	4%	17%	17%
Common shares, new	5%	22%	22%
Internal funds	41%	19%	19%
	100%		16.2%

The weighted average cost of capital is much lower than the cost of new shares. Why? The answer results partially from the deductibility of the interest expense and lease payments from the company's income tax obligation. Therefore, the income tax actually paid is reduced, and thus more money is available to the organization on an after-tax basis.

But such an approach does not measure the impact of different risk factors inherent to the project upon the capital structure or whether or not the cost of capital has any real importance in mine financing. Leverage improves the rate of return, especially in an inflationary environment (Cole, 1985), and the amount of leverage that can be obtained from the financial marketplace varies widely according to the risk profile, especially the mineral/metal product price volatility.

Some of these risks inherent in mine financing are in Table 6.6.4. (Tinsley, 1985).

In the extreme, past equity may be ignored in a financing decision using all debt. In general, the various cost-of-capital measures used in evaluations (e.g., NPV, IRR, ROI and payback, and the methods described in Chapter 6.5) do not provide an adequate means to compare or select a project for development if it is financed.

How then can one judge financing costs? One method is to measure the future cash flow coverage of debt service. This ratio is the annual net cash flow after tax (adjusted for the tax reduction due deduction of interest) divided by the annual principal plus interest charges, all done on a conservative base-case projection. Example 6.6.2 shows a simple start-up project financing with $15 million debt at 10% interest.

Example 6.6.2. Debt service coverage calculation, $ Million.

Year	1	2	3	4
Net revenues	$15	$20	$30	$32
Total cash costs and interest	10	12	14	16
Net Tax Paid	0	3	5	6
Net cash flow	$ 5	$5	$11	$10
Principal repayment	2	2	4	4
Interest Charges	1.5	1.3	1.1	0.8
Debt service	$ 3.5	$ 3.3	$ 5.1	$ 4.8
Debt service coverage ratio	1.42	1.52	2.16	2.08

Table 6.6.4. Assumption of Risks in a Mine Budget Financing

Risk Category	Absorbed by	If "Risky" by
Reserve	Financier	Reserve warranty
Operating; technical		Management agreement, Company, warranty
Cost	Financier	Contract
Management	Financier	Contract
Infrastructure	Financier	Consumer, government
Environmental	Financier, bonding	Company
Market	Financier, contract	Buy-back, take-or-pay
Political	Financier	Insurers, government
Force majeure	Financier, insurers	Company
Foreign exchange	Financier, company	Hedging
Funding	Financier	Swaps, hedging
Participant Agreement, contract	Financier	Joint venture
Engineering	Financier	Insurers, guarantee
Completion	Company	Construction, financier
Syndication	Financier (underwritten)	Company (best efforts)
Legal	Financier	Insurers

If this ratio is around 3 to 4:1, then venture capitalists usually can be attracted. If the ratio is 2 to 3:1, then mining companies often will assume the role of project financiers. If the ratio on the downside case is 1.5:1 or more, then most banks and leasing companies will be participants, but on a secured basis. If a producing project is being acquired by a leveraged buy out, the ratio needs to be a minimum 1.6:1 for the base case, although lower ratios may be tolerable for a year or two (during the turnaround). Strangely, the public markets do not seem to make any judgments about this ratio, preferring instead earnings per share calculations based on net income (not net cash flow) as established by the accounting profession. This further reinforces the notion of some that cost of capital is not critical for mine financing.

6.6.1.3 Changing Market for Funding

A single source of funding is seldom sufficient for all stages of a project or for all the requirements of an operating company. With some 30 to 40 different sources of funding, how can an individual or company make the right selection?

A short historical review will illustrate the changes that have occurred in the financial marketplace. In the mid-l800s, bonds and stocks were major sources of capital for the mining industry, the latter witnessing some famous investments and frauds. Europe, and in particular Britain, was a major source of these monies. In the 1930s, the petroleum industry thrived on production payment financing, especially in Texas. In the 1950s and 1960s, most mining industry financing came from internally generated funds with the notable exception of the aluminium industry, which required debt due to the sheer magnitude of the capital programs (Robinson, 1985).

In the 1960s, commodities with long-term sales contracts, such as iron ore (Australia) and coal (United States), began to attract large-scale bank project financings. The West Germans introduced a tax deduction for investment in mineral exploration overseas to help secure supplies of strategic minerals. Some 600 million deutschemarks (US$200 million) poured into US and Canadian exploration in the late 1960s to early 1970s. West German investments in overseas mine developments also became significant, especially in nickel and copper. The 1970s saw a wave of bank project financings that crested in the mid-1980s.

Canada introduced flow-through share financing in the early 1980s that passed the tax deduction of exploration to the investor in a company's shares. This reached its zenith of some C$l billion per annum in 1988 just prior to its termination in 1989. After the stock market crash of October 1987, any other equity funding sources for exploration essentially dried up.

In the early 1980s, gold loans became widely available in Australia. While still relying on bank project-finance-style guarantees, it nevertheless raised the debt level for new gold projects from 50 to 100% in many cases. At the same time, bonds convertible into gold or shares became popular in Europe, even for junior mining companies in minimum $20 million amounts. The 1987 crash and declining gold prices combined to halt further bond issues.

The US market for high-yield bonds (amounting to $180 billion in 1988), also known as junk bonds, was tapped in 1988 for a privatization of Magma Copper via a leveraged buyout using a $200 million copper-indexed junk bond. Such a subordinated structure would have been unthinkable in 1981 when commodity-indexed bonds briefly came into vogue.

Finally, in the 18 months leading up to the October 1987 stock market crash, billions of dollars in equity were raised by mining companies large and small, which provided the former an opportunity for the retirement of debt that had spiralled with depressed commodity prices prevalent in the first half of the 1980s.

One needs to invest time in assessing when and how to approach the financial markets. As a general rule, unless a finance professional such as a merchant/investment banker or stockbroker is used, it will take one to two years to obtain a feel for the market. Besides establishing some broad parameters such as the amount of further equity/dilution desired, percentage leverage, legal security issues, covenant structures, and the overall timetable, one will determine quickly that future flexibility assumes preeminence among financial objectives, and the cost of funding is pushed to a distant third place after availability of funding.

Early and continuous contact with the financial marketplace also will highlight structural aspects critical in the exploration and development/feasibility process (e.g., minimum reserves, maximum financing size, requirement for offtake contract or hedging mechanisms). This can save embarrassment when premature requests for financing are declined. All too often, the whole mine development process gets stalled while attention is turned to securing the financing mix that may require changes in the feasibility study itself.

6.6.2 EXPLORATION FUNDING

New and improved versions of exploration finance seem to emerge willy-nilly but are usually related to satisfying the changing objectives of the risk capitalist. The following sources of funding are often used in combination, but the principle remains the same. Normally, the earlier the grubstake stage, the greater the percentage of the action to the grubstaker.

6.6.2.1 Private Companies

Exploration funds are often raised privately, either as syndicates of individuals or in private companies established to take advantage of the tax deductions associated with exploration activities. Some tax regimes specify a maximum number of investors in such a company (e.g., Subchapter S corporations in the United States must have 25 or less investors).

6.6.2.2 Tax-loss Funding

Limited partnerships are another form of syndicate where the entrepreneur/geologist or fledgling miner acts as general partner putting up a small proportion of the money and charging a management fee while the limited partners put up the bulk of the funds.

These partnerships are the most prevalent means of tax-loss funding in the United States, although recent rule changes restrict the loss to the amount of money actually at risk. This has reduced the availability of such funds, but it remains tax effective.

In Canada, a means of a tax effective investment is provided by flow-through financing whereby shares could be offered at up to an 80% premium to the present share price, provided these monies were spent on bona fide minerals exploration. The flow-through shareholder was entitled to a tax deduction of 150% or more of the funds invested.

In Australia, trust structures have been developed, most notably for petroleum exploration, where the investor gets a progressive tax deduction as funds are spent on exploration. The trusts are listed on the stock exchange to provide some liquidity to the trust holders and to record how much tax deduction remains.

The disadvantage with tax-loss funding is that the investors tend to focus on the tax angle alone. These investors can be awkward to deal with in the transition to a development company.

6.6.2.3 New Listings/Floats

The public is often tapped as a source of funds, usually through the issuance of shares or trust units, which are listed on stock exchanges as a means of providing trading liquidity. All stock exchanges have regulations that are designed to require adequate disclosure of activities, financial and technical audits, and to restrain insiders from abusing their position.

Before proceeding to the prospectus stage, the exploration company should have obtained proper title to its tenements and claims. It is rare when an initial public offering or float succeeds based only on a geological concept.

The information requirements for a prospectus are summarized in the following. Because it is a legal document, it is fairly expensive to develop. Preparation costs for a simple prospectus may be as low as $50,000, but $150,000 to $250,000 may be closer to the mark for a medium-sized issue. If underwriting fees and expenses of 5 to 15% are added, then one can see that public offerings of much less than $1 million are not worthwhile.

The purchasers of junior mining/exploration company securities are, for the most part, unsophisticated investors with limited financial resources. Generally, junior companies doing public offerings should present several properties for which exploration funds will be devoted, so as to offer the investor some spread of risk in light of the great odds against the ultimate economic viability of any one relatively untested prospect (Chender, 1985). Alternatively, a substantial percentage of the offering proceeds may be dedicated to new properties, in which case the public (and the underwriters) will need to be convinced that the company's management is particularly capable of undertaking this activity successfully.

The main aspect to consider in a public offering is how much equity will be sold to the public, and a range of 20 to 45% seems prevalent. Beyond the equity yielded initially, a thoughtful share issue also will provide for future financing if the exploration is successful. This can be accomplished by issuing options or warrants to the original investors to buy shares at a premium of 20 to 100% over the next few years. Although this procedure represents further dilution, it is an automatic and inexpensive way to raise finance. Sometimes these options are listed, which gives the investor (but not the issuer) additional liquidity if liquidation of this entitlement is desired.

6.6.2.4 Private Placements

Before going public, a company may arrange an offering of unregistered securities or, where debt can be sustained by the balance sheet, by a private placement of debentures. A *debenture* is an unsecured bond. In fact, one should consider requiring an underwriter of a share offering to do a small private placement as a precursor to a public issue (1) to cover the issue costs and (2) to test overall support and placement power. Public companies also can do private placements usually within limited bands of discounts and premiums, around 15 to 20% from the present share price. Such shares have trading restrictions and usually cannot be sold for at least one year.

The advantages with private placements are speed and the much lower cost of the selling document in contrast to a full-blown prospectus. This results because the placees are deemed to be professional investors who should know the worth of the information presented rather than the gullible public.

Shares placed privately to initial seed investors may be registered along with a subsequent public offering (again with selling period restrictions) and the return to the original investors may be quite substantial. It is this process that provides the promoter with his return.

The term promoter should not necessarily be a dirty word according to Brock (1985) who goes on to point out that although junior explorers spent only 28% of the total funds, in this case in Ontario between 1951 and 1974, they were responsible for 62% of all economic discoveries (Freyman, 1978).

6.6.2.5 Joint Venture

Given the equity yielded in private placements, public offerings, and tax-loss funding, the exploration company ultimately faces the dilemma of diluting itself out of existence if major drilling or land payments are required or, indeed, if a mine has to be developed. The alternative is to dilute the property itself by giving up a piece to a larger source of funds, typically a major mining company.

There are dozens of nuances in attracting such financing but several generalities apply.

1. An option is granted for a sufficient period of time to whet the major company's appetite, usually 60 to 90 days. A small up-front payment may be made for this option, say $10,000 to $50,000, and if the major company wishes to continue the arrangement, then a further payment on exercise of the option may be required, say an additional $20,000 to $200,000.

2. Progressive exploration commitments must be made within specified time limits, usually the first 12 months, and escalating annually for several years.

3. The major company covers all property and government payments.

4. The major company either earns its position progressively or else has a one-shot position to its base percentage after all monies have been spent, a certain defined exploration success achieved (say, 100,000 oz or 3100 kg of gold in proven reserves), or a positive feasibility study/development decision reached.

5. If the major company is to finance and operate any mine development, it will usually require a majority interest. At this stage, the junior company can either contribute development costs pro-rata or be diluted to a carried interest or a royalty, such as a net smelter return (NSR) interest or a net profits interest (NPI).

6. Sometimes all of the foregoing is substituted by escalating lease payments and a royalty, perhaps up to a ceiling dollar amount.

7. The junior company may reserve a right to back into a percentage equity by paying a premium or matching the major's funding at some given time benchmarks.

Variations on these themes are legion, and the mining engineer is advised to track the progress of recent similar deals and to be open to negotiations with a number of parties before settling the fate of the joint venture. Corporate objectives such as managing an eventual operation and funding other projects, often impinge on this process and require care so that the exploration company cannot be squeezed out by the major. Brooks (1985) recommends tailoring the financing proposal to address the corporate philosophy of each major company.

6.6.2.6 Farm-in/Farm-out

The principles of joint venturing apply here for a party to earn in or venture out an interest in an individual prospect. The difference is that a farm-out may only be an interim exploration step with both the farmee and the original holder knowing that major financing will still be required if a discovery is made. In a sense, this preserves the option to go to the public markets rather than having one's destiny in the hands of the major company in a joint venture arrangement.

6.6.2.7 Venture Capital

Venture capital, sometimes referred to as mezzanine financing, is made available by sophisticated financial institutions or funds that believe a high return can be achieved by using their marketing, financing, and management skills to convert a promising prospect into a mine. By bridging to the development stage, they can extract a high fee from the high added value from their money and advice (Boettcher, 1985).

The venture capitalist seeks high returns, as follows:

Seed capital	100–300%	Equity
Second or third stage	30–45 %	Intermediate
Fourth stage/bridge	25–35 %	Mezzanine

These returns can either be in equity, or, more usually, loans convertible into equity, or with equity kickers such as options/warrants or royalties (e.g., NSR, NPI, or production payment interests).

Venture capital is an alternative to a public offering, but one must anticipate stronger management oversight instead of the stock exchange regulations. At the back of every venture capitalist's mind lurks the question of how it would manage the venture if the present management fails. Some mining companies with separate operating arms act as venture capitalists with the knowledge that, in a default, they will step in and operate the development.

6.6.6.8 Government Funding

One should be aware of government programs, especially aid grants to foreign countries, as a means to provide seed capital to exploration programs overseas. The United Nations Development Program is active in regional exploration, and the World Bank sometimes will fund more advanced projects. Naturally, these funds are not without some disincentives such as local involvement, bureaucratic delays, and more substantial paperwork. But international agency involvement may open more land and information sources than would otherwise be available. Government export-credit programs can sometimes be used to finance the technical services required to complete a feasibility study, again with ties to that country's professionals.

6.6.3 MINE DEVELOPMENT FUNDING

Once exploration funding has brought forward a project suitable for development, the mining engineer needs to change his/her geological/financial hat for one closer to the green eyeshade of the banker, accountant, lawyer, and tax advisor, since all these skills are required. Before proceeding, it is important to review four fundamental financial measures:

1. *Cash flow* is the actual quantum of cash in and out and ignores such accounting items as depreciation and depletion, except to determine the actual tax payable. No accruals or provisions are accommodated to reach the proverbial bottom line. A good way to look at the concept of cash flow is to view it as a source and application of funds. The determination of project cash flows is discussed in detail in Chapter 6.2.

2. The impact of *taxation* must be thoroughly understood, especially the impact of interest, dividends, and cross-border payments— not just at the federal or central government level but at the state, provincial, and local levels as well. One should also be aware of indirect taxation on imported equipment, hidden costs in rail freights and port charges, and infrastructure contributions. And, finally, a basic understanding of the lease-vs.-buy decision necessitates a thorough analysis of taxation issues.

3. The question of *net income* requires an understanding of depreciation and depletion rules. It remains the main measure for determining earnings per share, although this measure is being supplanted now by the realization that cash flow is a more direct measure of corporate health.

4. *Coverage* is a measure used to examine the health of debt repayments. It is the annual net cash flow coverage of either the debt service (principal plus tax-adjusted interest) or just the unadjusted interest charges (see Example 6.6.2). Other coverages are worth noting, such as (a) residual coverage after the financing has been repaid and (b) NPV coverage compared with the outstanding debt.

In today's financial world, it is unusual not to have a financial model built on a computer that can be changed readily to accommodate different technical and financial assumptions. A base-case projection uses tested and conservative assumptions. Downside sensitivities examine the individual and combined impacts of such factors as lower grades, variable commodity prices, higher operating costs, delays in start-up, inability to reach capacity, or deficiencies in reserves. As discussed previously in 6.6.1.2, the risk profile for a mine development should now be at a stage where coverage levels of 1.5 to 2:1 are achievable. If the project is better than that, has good reserves, and is in the lower quartile of operating costs worldwide, then one can command the mine development financing process. Money is always available for good projects.

Many of the principles of exploration funding hold for mine developments, but a wider spectrum of sources now is available—chiefly due to the willingness of debt providers to come aboard the capital structure because of the lower risk profile. The capital structure for a company that has just built a new project with its development capital is given in Table 6.6.5.

6.6.3.1 Equity Financing

Major mine developments in South Africa are all equity financed by hundreds of millions of dollars. This is the exception rather than the rule, since most companies seek leverage of their hard-earned equity dollar. New equity is usually derived from an issue of shares to the public, or it may be created by the issuance of shares for a property or in exchange for services.

Shares are either common shares or preferred shares. *Common shareholders* are the owners of the company and entitled to vote on its destiny. *Preferred shareholders* have a preference to the company's net income for a dividend, but if there is no income, they receive no return. Sometimes these accrued deficiencies have to be repaid before common shareholders see a return. Preferred shares often are convertible into common shares or redeemable by the company at a given time or at a predetermined price or ratio. Given this quasi-debt character of preference shares, it is sometimes argued that preferred stock is neither pure equity or debt. Gentry and O'Neil (1984) provide more discussion on this topic.

6.6.3.2 Tax-driven Financing

The main tax-driven financing mechanism for a mine development is through leasing of plant and equipment, whereby the lessor owns the equipment and claims the relevant tax deductions. Lease payments include amortization of capital and are fully deductible from income for tax purposes. It is this concept that makes financial leasing work effectively (Ravenscroft, 1985).

Most mine developments have a surfeit of tax-loss carry forwards, investment tax credits, and the normal depreciation and depletion deductions from taxable income; therefore, it is often preferable to pass these tax deductions on to someone else. However, in any lease-vs.-buy analysis, one soon realizes that even though the effective interest rate on the lease seems to be advantageous, in fact, it is an expensive source of funds. Generally, it is preferable for a mining company generating profits to own its equipment instead. Leasing is also relatively inflexible, and early prepayments are severely penalized.

Leveraged leasing has evolved for long-term financing of big-ticket capital equipment. Here the owner borrows 70 to 85% of the cost of the equipment and includes loan interest in its tax calculations. These are so complex that sophisticated computer programs have evolved to handle the lease rate optimization process. Safeguards are built in to cover defaults, changes in tax laws, and to provide sinking funds to cover the negative cash flows later in the project (Ravenscroft, 1985). While seeming to offer a terrifically low interest rate, in fact this financing is fairly expensive on an after-tax basis, due to the relatively high return (well in excess of 20% per annum), expected by the equity provider in a leveraged lease. This disadvantage is often outweighed by the large dollars available and very long-term character of leveraged leases.

Like all financings, it is worthwhile to shop around, as the mix of debt and equity and repayment structures causes significant differences in lease costs. One variation sometimes in vogue is double-dip leveraged leasing where the lender or equity player may be able to claim a tax deduction in two countries, thus enabling it to quote a lower effective lease interest rate.

Besides leases, trusts are also tax-effective in some countries especially if they can eliminate one level of taxation (viz. the trust itself is not taxed whereas a company would have been). In essence, the trustholder is deemed to be in the mining business and directly entitled to the income deductions individually. Certain royalty trusts in the United States have this feature.

6.6.3.3 Project Financing

The premise upon which project finance is based is that future cash flows from the project alone are sufficient to repay the loan raised to build the project. The assets so constructed are held as collateral security. The company, aside from initial completion guarantees and perhaps support of the sales offtake, is not otherwise liable to repay the loan. Banks are the primary sources of these funds, usually for projects in excess of $5 million but potentially up to the hundreds of millions of dollars.

The banks assess the risks involved (Tinsley, 1985) and run numerous cash-flow sensitivities to test their favorite downside cases, the results of which are used to size the amount of debt they are willing to advance, the length (term) of the loan, how loan repayments are made (e.g., tied to a percentage of net cash flow), and the cost of their money (inevitably a margin or spread above a floating interest rate such as the US prime rate or the popular London InterBank Offered Rate, LIBOR). Up-front cash costs of project financings are usually 1/2 to 1%, and legal/documentation costs are the primary expense. Sometimes these can be rolled up into the loan itself. Ongoing margins range from 3/8 to 2% above the floating benchmark interest rate depending on the project and parent company risks.

The structures and supports are surprisingly flexible, with the level of debt closely related to the volatility of commodity prices and the certainty of offtake/sales (Tinsley, 1985). A level of 70 to 80% debt is quite common; a few cases reach and even exceed 100% debt. In the latter case, banks have become comfortable with the achievability of capital cost estimates, compared with the huge cost overruns that plagued the industry in the late 1960s to 1970s, and are reducing corporate support precompletion. Loan terms are also stretching to 10 to 14 years.

US and Canadian banks dominated the business until the mid-1980s. Now European, Japanese, and Australian banks have learned risk-evaluation techniques and are becoming active in this process. Although primarily used by development companies, some operating companies use project finance to isolate a major new development or to finance one or two projects in an environment where money is required for a number of developments (Wightman, 1985). Sometimes project financing is part of an acquisition package.

Many variations exist, such as production payments, advanced sales, buy-back, or throughput agreements, and may have large differences in recourse (access to the parent to repay the project financing), force majeure flexibility, and repayment methodologies. One should set clear objectives internally before approaching the banks and ask the banks to suggest the most suitable structure.

6.6.3.4 Commodity-linked Financing

As pointed out before, the market risk is the most difficult to accommodate in mine financing. The development of silver-indexed bonds for Sunshine Mining (Schreiber and Neuhaus, 1985) in 1980 started a trend of precious metal-related financings culminating in very sophisticated gold loans and hedging structures. Some petroleum-indexed financings have come and gone.

Three varieties of *commodity-linked financing* exist. The first is in an international currency with a monetary interest rate that rises and falls, usually not below a set level, with the commodity price. Magma Copper's high-yield US dollar bond has an interest rate (coupon) linked to the price of copper.

The second is where the loan and interest payments are denominated in the commodity itself. Gold loans are the classic here where, say, 100,000 oz (3100 kg) is advanced to build the mine with the loan to be repaid over five years at an interest rate of 3% per annum. Although monetized for financing purposes, it is obvious that if the price of gold doubles, the loan principal amount and the interest rate have doubled in monetary terms. Because of this feature, gold loans usually only rely on 15 to 30% of a mine's annual output for repayment. Gold loans still require all of the risk-assessment due diligence for a project financing with an added complication of cash margin calls, should gold prices exceed the agreed margin-free limit of credit. Hedging of production by selling forward is recommended to cover mine operating costs (together with a currency hedge if costs are not in US dollars), and floor-price programs are used to ensure basic economic survival by the use of put options to ensure a minimum sales price is received. Each bullion source has its own favorite structure, but these are typical:

Margin-free limit:	$50 to $100/oz above the strike price (the day the loan is advanced)
Hedging:	Not more than 50% of output
Floor prices:	$50 to $75/oz below the strike price.

The third style of commodity-linked financing is a public issue of a bond, note, or certificate that is convertible into a given quantity of the metal by a certain date. The interest rate (coupon), usually fixed, is reduced because the investor retains an upside from the strike price that, in this case, is usually at a premium to the spot price. Gold and silver are the most popular with Europe, particularly Switzerland, the main source of these funds.

This third style of commodity convertible issue has witnessed various security structures to ensure that the metal is actually there upon conversion or that the issue can be repaid in cash if it is not converted. Originally, the metal or its equivalent was held by a trustee (defeasance) and progressively released. Mining projects were used next as collateral security followed by cash-collateral by defeasance or deep-discount bonds. It can now be accomplished unsecured and unsubordinated for good names, project developments, or acquisitions.

6.6.3.5 Contract Mining

Contract mining may be an indirect means of financing since the contractor may pay for all the equipment. A classic example is where the contract-mined ore is delivered to trucks to be hauled to a nearby custom mill that charges, say, 35% of the metal content as its tolling charge. In some instances the contract miner may also be paid as a percentage of the metal content thus requiring attention to dilution. The intricacies of contract mining are discussed in detail in Chapter 6.4.

6.6.3.6 Advanced Sales

Advanced sales are often used within project financing, but in some cases, notably uranium, it may be a stand-alone source of financing from a consumer. The consumer usually will receive some sort of security, such as a mortgage/lien on the mine and development, a guarantee from the mining company, or a percentage of the equity possibly leading to control in the event of a default. Naturally, operating and capital costs need to match the advanced sale price and quantity, as usually such a consumer will not cover cost overrun or other elements of operating risks. A production payment is a type of advanced sale. The miner sells a production payment interest in the minerals to be produced repayable up to a given dollar amount. Banks or companies are the buyers/lenders. While a recognized structure under US law, it is more difficult to reproduce elsewhere because of legal differences in the security of tenure to minerals.

6.6.3.7 Supplier/Buyer Credits

Equipment suppliers often have government financial incentives driven by the need to support the national manufacturing industries. Individual manufacturers may arrange preferential financing terms for their whole range of output, which they parcel out to individual mines or companies. And many governments grant low-cost export credits if the equipment is, obviously, exported (Rides, 1985).

When the feasibility study has been prepared and quotes are being sought, one should canvass a wide range of suppliers and ask for their medium to long-term availability for financing. It is anomalous that the best terms are most usually available from overseas, due to export incentives rather than the local market that may have the added ignominy of sales taxes on equipment. However, import duties and freight often reduce the overseas advantage.

The advantage to the supplier is that these programs often are linked to export insurance. The advantage to the buyer is access to fixed-cost, and medium-term financing at a lower rate than market. The disadvantages include the need to deal with government bureaucracy, the requirement for credit support (perhaps via the project-finance bankers), and the relatively inflexible terms and conditions. However, for projects with large equipment needs such as an open pit mine or a dredging project, this source of funds will be hard to ignore (deGavre, 1985). It is important not to order the equipment until this finance commitment is secured.

6.6.3.8 Governments

Some governments, including those in less developed countries (LDCs), have pools of funds available to stimulate their nationals into investing in overseas minerals projects. The strategy is either the diversity of minerals supply sources or to sell equipment. Sometimes it is political, as with the French in central Africa.

These funds, usually debt but sometimes equity, are for moderate-to large-scale projects and are provided on surprisingly advantageous conditions. The Ok Tedi copper mine financing in Papua New Guinea is typical of the desire of governments to cofinance. Banks ($150 million), government export-credit agencies ($488 million), and the US government agency OPIC ($50 million) were joined as lenders by an agency of the West Germany government, KFW, for $100 million (McGill, 1985). This combination also is a powerful consortium to counterbalance political risk.

Once the doyen of large mine project financings, the multilateral World Bank has now decided instead to concentrate on agricultural and more labor-intensive projects. However, it is still providing funds to assist national governments in minerals policy formulation together with some exploration and feasibility studies. Furthermore, its sister agency, the International Finance Corp. (IFC), is active as a provider of seed equity in LDC. The IFC conducts its own mine reviews and also likes to be a coinvestor and cofinancier in project development. Its multilateral nature again diminishes political risk for a project, since most national governments are loathe to attack/nationalize a project in which it is involved.

A two-year minimum time period should be allowed to brief and negotiate with government agencies. Early dialogue is a must and the assistance of a bank coordinator probably essential. One must also be sensitive to the internal and external politics surrounding these entities. The key advantage, besides a political risk shield, is the very long-term nature of the debt, sometimes subordinated to the bank project financing and paid out last.

6.6.3.9 Public Offerings

The approach to the share markets for an exploration company was discussed in 6.6.2.3. For a development or operating company, however, access to the share markets has to be on an underwritten basis. Brokers and their analysts need to be involved early so that when the feasibility study is finalized, it is the last step rather than the first step in seeking their support (Hodge et al., 1985). It is important to distinguish between dealing with the bankers, who want comfort on the downside, and dealing with brokers/underwriters who live for the sparkle of the upside.

Underwriting agreements are reasonably simple and usually have triggers due to the outbreak of war, or declines in commodity prices or stock market indices that can cancel the underwriting commitment. Lead underwriters (who charge an up-front fee of 5 to 15% for their services) often syndicate their commitment to subunderwriting brokers, but sometimes a company may be a subunderwriter. The agreed issue price for the shares relates to valuations of the company (plus its new development), stock market conditions, and the size of the issue relative to the total shares outstanding (dilution). In the end, judgment matching the skill of the underwriter with the perception of the company/project to boost the underlying share price will determine the pricing and conditions. Once priced, share issues proceed quite quickly.

Bonds are another form of public finance where a company has an acceptable credit rating or, more recently where the bond is convertible into shares, a given amount of both. A bond is a publicly traded financial instrument, usually with a fixed-interest rate and relatively long term (maturity) and having a general security over a company by a debenture or guarantee. The purchasers are usually large financial institutions in the United States or Canada or more widely held in Europe (e.g., the so-called “Belgian dentist” who invests in Eurobonds).

Up-front fees are high, 2 to 3%; registration procedures are difficult and expensive, $200,000 to $500,000; and it is difficult to link to a mine development (Ulatowski, 1985). However, the emerging marketplace for high-yield securities means that unsecured bond issues are now possible for medium- to large-sized miners.

The Vancouver Stock Exchange is the mecca for exploration floats, followed closely by the Australia Stock Exchange. The Alberta Exchange permits blind pool offerings where no specific purpose is required. The Toronto, Montreal, and Denver exchanges come next, but usually for more established and medium-sized issuers. The London Unlisted Securities Market and the London Stock Exchange also see successful junior floats, especially for European minerals prospects. Local stock exchanges in Asia are starting to provide funds, albeit for exploration and development within their respective countries or the nearby region.

Warrants can also be used to raise money for a mine development. Canadian gold producers have issued gold purchase warrants, which are a right to purchase usually 0.02 oz (0.62 g) of gold at a given price. In this way they have raised $2 to $3 million from essentially forward-call options while providing a listed "gold play" for their shareholders. The warrant or option for either shares or to purchase product may be a sweetener to a share or bond issue as a means to increase the size of the financing or to lower its cost. Large amounts ($20 to $40 million plus) can be raised for large gold mine developments as cofinancings with a project financing (Tinsley, 1988). This is the reverse of the miner's call option, used in conjunction with a gold loan.

Commercial paper has been in vogue for 100 years in the United States, usually for very short terms, 30 to 90 days (Yablonsky et al., 1985). It has now been used as a funding source in large Australian project financings by using the banks to substitute for the credit risk by a letter of credit (LC), either jointly or by a fronting bank's LC. Although a semi-public and very liquid market it is highly ratings sensitive. Unfortunately, most mining companies do not have good credit ratings.

6.6.3.10 Joint Ventures

Special considerations must be given to joint venture financings, especially unincorporated ones (Cockburn, 1985). Problems with security, control, and management can complicate the structure, recourse, and legal security for a project financing. Instead of solving these problems and fighting the coventurers, one should look for synergy to help bolster the financing, such that one company provides the management, one the market offtake, one most of the financing, and another the equity, and readjusting joint venture interests later according to the success and contribution of each.

International joint ventures need special attention to jurisdiction cross-border and taxation impacts. And the management agreement, development agreement, joint venture agreement (JVA), and the sales agency agreements take time to settle, unfortunately, sometimes a few years.

6.6.3.11 Institutions/Insurance Companies

Insurance companies, pension-plan investors, mutual funds managers, and investment advisors all comprise a class of investors known as *institutions.* Institutions may be major buyers of shares and private placements or provide the equity in leveraged leases. Their risk tolerance is low, so they become funders only of established operating companies or good mine development situations.

In addition to general funds, institutions have special financing niches. They will sometimes provide very long-term money, say, 15 years plus, for projects like coal mines supplying minemouth coal to power stations. Institutions may also be the source of funds for pollution-control bonds used in connection with a tailings project or a stack gas control investment. In general, they need to see a credit rating, but sometimes they will speculate on an emerging situation or act together with banks in a project financing.

6.6.3.12 Off Balance Sheet

Finally, some varieties of financing are designed to minimize the debt: equity ratio as calculated from the company's accounts. The debt portion of the company's capital structure may be disguised by intermediate entities that undertake the debt liability rather than the parent. The methods depend on accounting rules for consolidation, and it is becoming more difficult not to consolidate off-balance-sheet debt, for example, future lease obligations.

Table 6.6.5. Development Company Balance Sheet

	$000
Current assets	$3,250
Current liabilities	(1,050)
Working capital	$2,200
Investments	1,100
Mineral properties	18,000
Plant and equipment	10,000
Long-term debt	(15,000)
Shareholders equity	$16,300

The intermediate entities are generally owned less than 50% or may be trusts over which the parent ostensibly has no control. Sometimes, unbelievably complex ownership webs are used to this end and the financier and operating company are well advised to get full accounting, tax, and legal advice before proceeding. However, if the parent's capital structure is getting top heavy with debt, this approach may be necessary to raise further money or bypass existing financial covenant restrictions imposed by other lenders.

6.6.4 OPERATING MINE FINANCING

In addition to the financing techniques outlined in the foregoing, an operating company has to manage its ongoing payables and receivables. A typical balance sheet for an operating company is given in Table 6.6.6. This moves the financing more into the realm of the corporate treasurer rather than the mining engineer/company manager. A brief description follows showing how financing can be derived from these day-to-day payments.

6.6.4.1 Trade Financing/Letters of Credit

These are the familiar "net 10 days" for payment. A custom smelter gains trade credit by paying for concentrates two months after the month of shipment (Tinsley et al., 1985).

Inventory loans may be via a warehouse financing for 60 to 80% of good quality materials or work-in-progress. Legal security is handled in a variety of ways. Costs range to 2% per annum above a floating rate (e.g., prime).

Accounts receivable can be continuously assigned to a bank at a cost of 3 to 5% to cover the collection risks, for example, converting blister copper into salable form.

Letters of credit (LC) are bank undertakings to pay a given amount against presentation of specific documents (e.g., loading of concentrates into a ship). Various LC types should be examined.

Acceptances are created when a transaction has been completed and the payment process is underway. The promise of payment may be accepted and traded in the market at a discount.

A forfait is a variety of factoring import and export payment streams via bills of exchange usually guaranteed by the importer's bank (Bradbury, 1985). Put simply, it permits early payment while the goods are in transit.

Credit insurance is also important in trade finance and many government and private insurance sources are available, especially to cover political risks.

Banks are the main players in trade finance and letters of credit, and their advice should be sought on tailoring the best program.

6.6.4.2 Hedging

The mining industry has been too conservative when considering hedging. It is seen as a form of gambling. It is also perceived as altering the pure "metal" play underlying the company's share value. In fact, the opposite is true. Not hedging the imbalances in a mine's future income or past obligations is gambling with the company's financial position if not its very existence.

Hedge markets are available in many commodities, that is commodities that are fungible (viz., having the same character regardless of location). Metals, currency, interest rates, and stock market indices are the main varieties. Hedging may be physical, as on the London Metal Exchange (LME), or more financial in character with put and call options, warrants, and synthetic structures.

The need for a currency hedge arises where a borrowing has been incurred in, say, Swiss francs for a coal mine getting paid in US dollars, or where operating costs are in Australian dollars vs. revenue in US dollars. One should be careful to assess what the underlying revenue currency is, even though payment may be physically made, for example, in the Thai baht equivalent.

Currency hedges can be accomplished by formally purchasing forward contracts to match the net currency exposure, usually limited to 12 to 18 months, or by rearranging the loan obligation itself via back-to-back or parallel loans to shift the currency of the obligation, often out 8 to 10 years (Poole, 1985). A formal currency swap agreement may be negotiated between two parties directly or with a bank intermediary with various fees and reexchange arrangements at maturity. The currency swaps market is gigantic and is reaching the stage where it can be quoted directly off a trader's screen.

The interest rate obligation can also be swapped so that a floating interest obligation is converted to a fixed interest. This can also be overlain on a currency exchange.

Commodity hedging is more straightforward. Various exchanges provide a means to directly sell output (go short). A mine by definition is naturally "long" with its production. The main commodities exchanges are the LME, New York's Comex, and the Chicago exchanges. Others around the world sometimes mirror these contract conditions so that 24-hr trading is possible. If not, many metal traders maintain their own 24-hr "book" of long and short positions. The exchanges do not go out much further than 20 to 24 months, so hedging beyond this period requires either a principal to act as a buyer or an intermediary trader to construct the longer-term hedge synthetically. The outside limit is usually three to five years for otherwise exchange-traded metals.

Most companies do not like to hedge 100% of their production in the hope that they can take advantage of the price spikes that occur from time to time. Good rules of thumb for hedging are as follows:

1. Cover 100% of mine operating costs as far forward as possible on the downside production profile and be sure local currency costs are matched in that currency.
2. Cover loan repayments at least 50% by currency hedging if required. Change the percentage hedged progressively by rollovers and perhaps leave unhedged from time to time.
3. Cover forward price volatility with floor-price programs and put options designed to ensure a minimum cash position, after debt repayments and some exploration costs, remains each year.
4. Match debt obligation where possible by commodity-linked structures, e.g., gold loans, especially for volatile commodities.

6.6.5 MERGERS AND ACQUISITIONS

Finance plays a major role in mergers, acquisitions, takeovers, and divestitures. The process is initiated by an analysis of the underlying value or break up realization compared with the share market or balance-sheet valuation of the asset or equity. Investment analysts, merchant bankers, entrepreneurial companies, and strategic planning groups are continually sifting available information, such as annual reports or stock exchange filings, to see if value is not fully appreciated by the owners.

6.6.5.1 Management Buyouts

The share market may place little value in a conglomerate or a struggling unprofitable company. A parent company may not want to continue supporting a subsidiary due to an internal shift of emphasis. Management may be able to arrange high levels of debt, up to 90%, by getting rid of the "moribund" board or parent and by trimming and focusing product lines, or through improved marketing. Many conventions apply to the process but the keys are to provide a comprehensive business plan and the reason, frankly, why management have not been able to perform in the past.

Buyout funds exist to invest in this type of effort and their rewards are high and usually tied to earning a piece of the equity as well, usually 10 to 30%. Because of the high leverage, management's return can be very high or nothing.

6.6.5.2 Divestitures

Sometimes a mining division or an individual mine will be sold off by necessity since the parent is overloaded with debt or else the disposal is of a peripheral operation to what is viewed as the core business. One can either deal with a short list of potential buyers, or one can let a friendly investment banker earn his keep by privately arranging the sale or by a tender process.

The sale of a company takes longer because of the due diligence to ensure no skeletons remain in the corporate closet. An asset sale is often quicker. Sometimes the seller has to help the buyer with financing or provide for term payments. The latter needs very careful analysis, particularly if the buyer is overly dependent on the project it has thereby just purchased.

6.6.5.3 Reconstructions

Mines with excessive debt or technical problems often present opportunities for refinancing and turnaround by a new party. A company in bankruptcy/receivership may also be a candidate for the financier because the entry price may be very low. In general the financier in these circumstances needs especially strong legal and financial analysis support and the ability to renegotiate with past lenders and investors and the ability to bridge the credibility gap if new technology and management are to be applied.

6.6.6 INFORMATION REQUIREMENTS

The information required at one stage of financing can be drastically different from another phase, and each source of funding has its own conventions and idiosyncrasies.

6.6.6.1 Stock Exchange

Each stock exchange has its own rules and regulations for prospectuses. They are usually organized as follows:

Summary/use of proceeds
Technical review (independently checked; see Chapter 6.2)
Qualifications of directors/management
Legal review
Financial statements (audited)
Subscription form

In most cases, a lawyer should be part of the process to ensure compliance with the regulations. The technical review needs to cover (Affleck and Stevenson, 1985; Gentry and O'Neil, 1984):

Property description/tenure
Location/access
History
Geology
Reserves
Conclusion
Recommendations/budgets

6.6.6.2 Placements

Private placements very often require only an information memorandum, which may have little independent technical and financial audit data.

At the exploration stage, the geologist's review and recommendation may be sufficient. Some sources of private placement may simply conduct their own due diligence, and no formal report is required.

6.6.6.3 Bankable Document

The bankable document is a somewhat mythical volume that is satisfactory to bankers or major debt providers. It will inevitably contain a detailed independent technical review of the project (Emerson, 1985; Schreiber, 1985) covering:

Reserve estimation
Run-of-mine/mill-feed reserve
Mining
Processing
Marketing

The independent consultant should also comment on the appropriate risk levels and conduct sensitivity analyses accordingly. Detailed capital and operating cost workups supported by actual quotes and preliminary designed will be incorporated into the base case financial pro-forma cash flow projections.

The bankable document should state why alternatives were rejected over the chosen course but not provide all the backup testing and calculations. Project financing requires the project to stand on its own financially once it is built; therefore, it requires the most comprehensive review of feasibility. While checklists are useful, the complexity in present day minerals project analysis, marketing, and financing means that financing mixes and risk off-sets should also be analyzed in the bankable presentation and presented with a comprehensive series of cash flow projections reflecting these factors (Tinsley, 1985).

The providers of funds are no less human or warm than the people in the minerals industry and will be able to provide valuable guidance if approached correctly and in a timely manner. Knowledge of the different sources of funding will either eliminate (and thus save the time for preparation and approach) or open up new vistas for finance from the gleam in the geologist's eye to the time when the mine has been built and is happily producing from nature's storehouse.

Table 6.6.6. Operating Company Balance Sheet

	$000
Current assets	$15,800
Current liabilities	(12,500)
Working capital	$13,300
investments	13,200
Mineral properties	22,000
Net plant and equipment	56,000
Long-term debt	(32,000)
Shareholders equity	$72,500

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Section 7 Mine Development

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Chapter 7.0
GENERAL PRINCIPLES

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This section of the *Handbook* deals with general principles to be evaluated in the development of a mining operation. It addresses the major issues that must be examined to fully evaluate a mining venture. Of course, one of the main topics to be evaluated is mine planning, which is covered in more detail in other sections (13, 17) of the *Handbook* and should be referenced in mine development projects. The topics to be discussed in this introduction include equipment and supplies, infrastructure, utilities, and project management.

In any of the aspects mentioned, the current existing conditions at or near the mine site must be evaluated. This will include assessment of current use of utilities, infrastructure, housing, equipment suppliers, etc., as well as anticipated growth, both in the supply aspect of these systems as well as projected usage on these systems. After full assessment of both these aspects, an evaluation must be made as to the requirements the project will place on these items. Assessment must be made as to any upgrades required to bring the project on line, what impact the project will have on existing or expanded systems, and anticipated future growth impacts on support systems not only of the project but of other surrounding industry and urban development.

The general manager of any operating company must address all aspects of any project development. In mine development, additional topics not discussed include market analysis and financial analysis. The market must be assessed as to current pricing levels, projected pricing levels, projected consumption demands, projected supply, potential market penetration of the project, projected sales price of the product of the mining operation, projected cycles in the market, etc. In financial analysis, all of the items discussed in this section must be used as well as market information, cost of capital for the corporation, strategic planning of the corporation, and evaluation of the numerous financial indicators available for such projects. Details of financial analysis are covered in Sections 2 and 6.

Of course, one of the main considerations in the development of any mining project is the evaluation of the project in light of the strategic goals and plans of the parent organization. A detailed analysis must be made as to whether the project fits the needs and goals of the organization before preceding into the actual development stage.

The topics discussed in the chapters of this section are the areas that are under the primary control of management, namely, personnel and machinery and the expenditure of money on both.

7.0.1.1 Equipment and Supply Selection

The equipment to be used in the mining operation will be dictated primarily by the mining plan. However, latitude remains as to the actual equipment supplier selected, as long as the specifications required due to physical parameters dictated by the mining plan are still met. An evaluation must be made of all of the heavy equipment concerning items such as price, availability, product support, local vendor capabilities, etc. It is worth the time in large operations to visit other similar operations prior to the selection of equipment in order to gain the experience of others pertaining to specific pieces of equipment.

Product support is very important in the selection of any equipment. This will dictate the maintenance staffing requirements as well as capital expenditures on such things as shops and tools. Depending on the size of the project, the equipment supplier in many cases will make specific concessions as to product support for the particular project. Additionally, all local vendors are not created equal. Some vendors supply very little support while others supply very significant, support in both parts and supply availability as well as the supply of contract maintenance personnel and offsite component rebuild. If located in a remote location, another consideration is integrated infrastructure in regards to parts and supplies. An evaluation must be made on delivery of parts and supplies to the mining operation which may include use of the product transportation systems such as railroads, trucking, etc.

One of the main supply items of any mining operation is the source of fuel and lubricants. Care must be taken to evaluate all possible suppliers of such products in looking at price, supply capabilities, and evaluation of the product supplied by each vendor.

The supply of explosives is another area for evaluation. There are many legal restrictions on explosives manufacture and transportation all over the world. In many locations, the host country actually controls the movement of explosives through government regulations. In other countries, such as the United States, explosives are distributed by private corporations. An evaluation must be made of potential suppliers in regards to purchase price, delivery modes, and evaluation of specific products supplied by the supplier to determine if the products will meet the needs of the mining operation and how the products will be transported to the operation.

7.0.1.2 Infrastructure

One of the most important aspects of any mining operation in the development stage is to evaluate the availability and qualifications of the work force in an around the mining area. Evaluation of the qualifications of potential operators must be assessed, what training must be provided to these operators, what training facilities are available in the area, and if outside labor must be brought in for the more skilled jobs. Of course, the evaluation of such things as union vs. nonunion conditions and the salary levels of the work force must also be assessed to properly complete the financial analysis of the project.

In an ongoing operation, another large component of cost is parts and supplies. This must be addressed in the development stage of an operation. After the major equipment has been selected, the parts supply to service this equipment must be evaluated in light of such questions as point of sourcing and the transportation required to get these parts to the mining operation. Supplies must be evaluated in the same light.

The next area for evaluation in the development stages of an operation deals with the transportation of the actual product of the mining operation. Naturally, the location of the operation is predetermined by the geologic formation; however, potential markets must be identified and thereby dictate the two end points for the transportation system. All transportation systems should be evaluated including railroads, waterways, highways, potential use of conveyors, potential of pipelines, and any other transportation systems that may be available. This is normally a large portion during the development phase of a project and may vary in magnitude depending on the location of the project. In many of the more developed areas, existing transportation systems are available, and a simple discussion with the potential suppliers of this transportation will lead to the proper data needed to evaluate transportation systems and make the proper selection of the mode to be used. However, in the development of a new operation in a remote location, significant capital expenditures may be required for the transport system. In such case, substantial time and effort must be expended to evaluate each and every potential system as mentioned previously with close attention being paid to capital requirements, physical conditions existing in the area, longevity of the project, operating cost of the transportation system, governmental regulations on transportation systems, other potential users of transportation systems, etc. The decision-tree analysis is a good tool to use in selecting the alternative to put in place.

The final aspect to be considered under infrastructure is the topic of facilities. The size of the offices, shops, and warehouses will be dictated by the size of the operation, the compliment of equipment used in operations, the number of personnel required, and availability of existing facilities. In many remote locations, all of these facilities must be built to service each and every need of the operation. However, in many developed areas, existing office space may be used to reduce capital requirements of the project, and existing shop facilities and warehouse facilities may be utilized offsite for major-component rebuild or supply inventories.

7.0.1.3 Utilities

The installation of utilities is a very important aspect of any modern mining development project. Much of the heavy equipment utilized in large mining operations today is powered electrically. As such, electrical supply to the operation is extremely important and must be evaluated in great detail by personnel experienced in today's electrical technology. Involvement of the equipment suppliers is essential to establish the interaction of such equipment with electrical supply systems. Special attention must be paid to modern ac electrical equipment in surface operations and the higher horsepowers and resulting large power requirements in underground equipment.

Additionally, the mine planner must evaluate the water supply and sewage disposal systems to be established for the mining operations. Provisions must be made for potable water for the work force as well as water supplies for such items as steam cleaning of equipment. Also sewage disposal must be addressed for all the facilities on site as bathhouses, restrooms, etc.

7.0.1.4 Project Management

Project management is extremely important from the most minute activity up to the level of development of the overall large mining complex. The most widely used mathematical management tools are PERT (project evaluation and review technique) and CPM (critical path method). Computer software has made these techniques easy to use and available to almost all mining operations. This software can either be PC-based or mainframe-based, depending on the availability of hardware and the size, location, magnitude, and complexity of the development project being pursued.

Project management should be used throughout the design, construction, and exploitation phases of any mining project. This is imperative to control both cost and time of implementation of the project. Most of the software packages available today are easy to use and serve as a tool to control both the timing and cost factors involved in project management. It is also very important to staff this function with a competent individual who understands the process and who can keep the general managers informed of the current status of any phase of such projects. Coordination with the management information services department of the parent organization should result in a review of current software available and the development of a project management and control system best suited for the overall organization.

Chapter 7.1
LAND ACQUISITION

DONALD L. SCHAIBLE

7.1.1 INTRODUCTION TO LAND ACQUISITION

This chapter deals with the subject of land acquisition from a project engineering perspective. Therefore, the working definition of *land acquisition* as used in this context is

> *The activities and planning efforts that are intended to result in the acquisition of the required level of control over those parcels of property that are vital and necessary for the timely development of a particular mineral project.*

The subject is treated from the perspective of a project located in the United States. Many of the planning principles apply to projects in other countries, but specific legal considerations vary. The developer must make the necessary site-specific adjustments for these variations.

During discussion of the subject, references are made to various US governmental agencies, policies, laws, and regulations that may affect the developer. Likewise, there is some discussion of subject matter relative to land ownership, sources of land information, and forms of agreement. These references are included in this chapter as a convenience to the reader and to serve as background information only. For a more inclusive treatment of the subject of mineral law and land management, refer to Chapter 3.2; and for consideration of the role of land acquisition in project operating strategy, see Chapter 6.4.

Historically, land acquisition has not been considered an engineering function. Various methods of dealing with this activity have been used. Many project developers have assigned this function to their legal or real estate departments. Some have assigned the task to outside contractors or consultants. Still others have delegated the land acquisition activity to the person least busy at that particular time! All of these methods have been used successfully at one time or another. But, whatever method is chosen, the most successful land acquisition endeavors are usually the results of a team effort, characterized by the close involvement of the project engineer.

Land acquisition is fundamental to the success of any mineral development project. Indeed, there is no project without the legal right to affect the properties involved. Regulatory permits required for mining projects will not be granted unless the permittee can demonstrate a legal right to mine the properties. The value of the land and the underlying minerals is often the basis upon which project financing is extended.

Poorly planned acquisition activities can result in:

1. Acquiring less than the required property.
2. Acquiring more than required.
3. Paying too much.
4. Early acquisition (cash flow impact).
5. Late acquisition (project delays).
6. Acquisition by competitors.
7. Failure to acquire the required properties.

The first five results would have negative impacts on the financial feasibility of the project. Both of the last two items could result in total failure to develop the project. A well-conceived and coordinated land acquisition program can enhance the project economics and technical feasibility by avoiding, or minimizing, these negative impacts.

Given the pivotal importance of the activity, it is apparent that land acquisition demands careful planning and integration into the overall project development plan. This planning and integration is a function best handled, or at least coordinated, by the project engineer.

In this chapter, we will:

1. Discuss briefly the land ownership situation in the United States.
2. Identify the various levels of land control and the impacts on the project of each level.
3. Review the steps required to develop an orderly, efficient land acquisition plan, with particular emphasis on the role of the project engineer.
4. Discuss the potential roles of contractors and consultants in land acquisition.
5. Illustrate the land acquisition process in a case study of a hypothetical mining project.

7.1.2 LAND OWNERSHIP IN THE UNITED STATES

Land ownership in the United States resides with the federal government (*federal lands*), state governments (*state lands*), or with private individuals and business entities (*private lands*). The project engineer must be concerned with the distinct types of owners since the acquisition potential and procedures for a particular plot of land are largely determined by the ownership.

7.1.2.1 Federal Lands

The vast majority of federally owned acreage is contained in the western states. Acquisition for mineral development on these federal lands is primarily governed by the Mining Law of 1872 and the Mineral Lands Leasing Act of 1920.

Federal lands can be categorized as follows:

1. Public lands.
2. Reserved and withdrawn lands.
3. Acquired lands.
4. Severed lands.

The importance of distinguishing these categories to the project engineer is that the acquisition potential and the laws governing the acquisition vary for each category, as well as for different minerals within each category (again, see Chapter 3.2).

Public Lands (Public Domain): These lands were acquired by the United States through treaty or purchase from a foreign country. Both the surface and mineral rights belong to the United States, and the Mining Law of 1872 governs their acquisition by mineral developers. The mineral developer, upon the discovery of a valuable mineral deposit on public lands, may gain title to the land by staking a claim (*location*), demonstrating the presence of a valuable mineral deposit (*patenting*), and spending a specified minimum amount of money on development each year (assessment work). This process is referred to as acquisition by location. Additionally, the Mineral Land Leasing Act of 1920 specifies several commodities that may only be acquired by leasing. This list of commodities includes, among others, coal, oil, oil shale, and gas. The Common Varieties Act of 1954 further extended the “lease only” status to certain building materials (e.g., stone, sand, gravel, etc.). These “lease only” requirements apply to the specified minerals for both public lands and the other categories of federal land.

Reserved and Withdrawn Lands: Lands in these categories have been set aside by the federal government for special public use. Public uses include Indian reservations, military reservations and national parks, national monuments, and national forests. *Reserved lands* have been permanently set aside, while *withdrawn lands* have been set aside temporarily. The lands are not obtainable by location. Leases may or may not be issued on these lands depending upon the terms of their separation from the public domain.

Acquired Lands: These parcels have been acquired by the federal government from state governments or private owners. Since these lands are not part of the public domain, they are not available for acquisition or leasing except by special provision of the federal government.

Severed Lands: Severed lands are those lands for which the federal government has sold the surface rights while retaining the rights to any minerals underlying the surface. Mineral rights on severed lands may be acquired or leased depending on the site specific details of their severance.

7.1.2.2 State Lands

Lands owned by the various state governments may be owned in fee or the surface and minerals may have been severed. Most of the states have enacted laws governing the sale or leasing of these lands and mineral rights. Each state also has certain lands which have been set aside for public use, and they may or may not be available for mineral development.

Since mineral land laws and regulations vary considerably from state to state, the developer must become familiar with the situation in each state in which he/she is contemplating acquisition activities.

Acquisition procedures for state lands vary from state to state, but most include some form of prospecting permit that carries with it a priority to lease or purchase the permitted land. Some states have laws that loosely follow the federal government’s acquisition by location procedure.

7.1.2.3 Private Lands

Private land ownership, as with government-owned lands, may be in fee or consist of separate ownership of the surface and mineral rights. In some cases the mineral ownership may be further subdivided based upon the type of mineral or designations of specific lithologic units (as in the case of multiple coal seams).

Again, since state laws regarding the ownership and transfer of mineral rights may vary, the developer must be familiar with the laws of the particular state involved. It is not uncommon for the laws of a state to reflect considerations peculiar to the dominant mineral resource of that state. Texas, Oklahoma, and Louisiana, for example, are heavily influenced by the petroleum industry while Kentucky, West Virginia, and Pennsylvania are coal-oriented states. These influences often have an impact on the requirements for securing, and particularly for holding, mineral rights.

Acquisition activities on privately owned parcels may vary slightly depending on the nature of the owner. We will discuss the impact of dealing with private owners as opposed to corporate entities.

The term *private owners* includes not only individual persons but also trusts that may be land trusts in those jurisdictions that allow them, or estates administered by banks and/or trust companies. Private owners are generally subject to different influences and often possess different objectives from businesses. Since much of our mineral-bearing land occurs in rural settings, it is quite common for the individual owner to be a farmer or rancher. If a rancher or farmer is considering issuing a mineral lease, he/she will be very interested in the length of time the land is not available for agricultural production. The condition of the land when it reverts to him/her will also be an important consideration. Postmining agricultural productivity could become an issue to be addressed in the lease agreement.

If the project area is owned by a large number of private individuals, then the developer must be aware of the danger that he/she may drive prices up by settling too quickly with reluctant landowners. The developer must assume that all of the landowners in the project area will be knowledgeable of the terms and conditions of other purchases or lease agreements. It is generally desirable to acquire the most critical parcels first in this situation.

Corporate land owners of mineral properties quite often fall into one of the following categories:

1. Mining companies.
2. Transportation companies (railroads).
3. Forest products companies.

Corporate entities in the mining business routinely acquire prospects for development, and therefore, it is not uncommon to encounter them as landowners. Since mining companies are usually interested in developing the resource themselves, they may not be inclined to sell or lease to a competitor. There are, however, exceptions to this scenario. Occasionally, a mining company will decide to generate some cash by selling or leasing a property for which they have no immediate development plans. Another possibility for acquiring land from a mining company is by trade. If two mining companies each have partial holdings in several areas, they may be able to mutually benefit by a trade that will consolidate the positions of both parties. In these dealings, the developer may be sure that he/she is dealing with someone conversant in the mining industry and aware of the important issues involved in mineral-land acquisition.

Railroad companies and their subsidiaries control a vast acreage of mineral and surface rights, particularly in the western United States. Historically, the railroad companies have been prohibited from commercial exploitation of the minerals underlying these properties. Therefore, they have entered into lease agreements with mining companies in return for production royalties. Companies that own large tracts of land usually have a real estate management subsidiary handle mineral-land leases and sales. The mineral developer will be dealing with full-time land management professionals who will be working within the framework of company policy.

Another major class of corporate mineral land owners is the companies of the wood-products industry. This includes timber and paper companies. These companies typically will lease out mineral rights as long as the mining operations are compatible with their own use of the forest products. Often they will harvest the timber and/or pulpwood from the site just prior to mining operations. They are usually quite sensitive regarding the length of time the property is tied up by mining and also to the reclamation of surface mined lands to promote reforestation. They will sometimes require that a commitment to such reclamation be a part of the lease agreement.

7.1.3 FORMS OF LAND CONTROL AGREEMENTS

There are various forms of control by which a parcel of land may be held. These forms may be sorted according to the degree or level of control afforded by each. Three forms of agreements that will be discussed are (in ascending order of control):

1. Options.
2. Leases.
3. Purchases.

Often the acquisition project will feature two or all three of these forms of agreement and may address the control of the surface and mineral separately. The optimum level of control can usually be identified by a carefully conducted cost/benefit study. However, identifying the optimum level of control does not always mean that one can attain that level. When dealing with federal or state lands, for example, the level of control available is normally fixed by laws and regulations. Private lands, on the other hand, often offer a wide range of alternatives to the project developer and the landowner.

This discussion of these land agreements is predicated upon a "grass roots" project. If the land acquisition effort is focused on expansion, or continuation, of a mature project, certain of the comments contained herein will not pertain.

7.1.3.1 Options

During the exploration phase of a project, and prior to determining the full potential of a deposit, the developer often needs an agreement with the landowner to secure access and some level of control of the property. This agreement is required to give the developer the right to complete his/her assessment of the project, and also the exclusive (or first refusal) right to gain more complete control if and when the project is proven feasible. A common form of agreement for this purpose is the *option.*

An option agreement conveys the lowest level of control in that it gives the developer only temporary access to the property and usually restricts the developer's activity to exploration and data collection efforts only. The term of the option must be for a period of time sufficient to determine the feasibility of the project and to make the decision to proceed or abandon the project. The agreement should specify at least the general terms of a subsequent, more permanent agreement (lease or purchase) to be executed at the developer's option. In return for granting this temporary control and option, the landowner will usually receive compensation in the form of cash bonuses or rentals.

From the developer's viewpoint, the option allows time to conduct exploration programs and feasibility studies prior to committing large sums of money for acquisition. If the study results are positive, his/her investment in exploration programs and engineering studies is protected by the exclusivity provisions of the agreement. The landowner, on the other hand, receives some compensation during the time the land is off the market and the promise of more substantial revenues if the project moves forward. If the project does not become a reality, he/she still has the land.

7.1.3.2 Leases

The *mineral lease* is probably the most common form of agreement between developer and landowner in the mining industry. The lease agreement is often preferred by the developer over an outright purchase, because it requires less immediate cash outlay prior to the project becoming profitable. The landowner generally prefers this arrangement because he/she retains title to the property, and may resume possession at the expiration of the lease agreement.

The terms of the lease should have been defined to the highest degree possible and included in the option agreement. During the option agreement period, as the nature and economic potential of the deposit is further defined, the specific final terms of the mineral lease may be negotiated.

Whether negotiated prior to signing the option agreement or during the option period, the developer (lessee) should be sure that the following terms are addressed:

1. *Control*—The lessee needs adequate control of the property so that he/she has the flexibility required to conduct operations in the most efficient fashion.

2. *Term*—A long term provides the lessee with the assurance that he/she can maintain the lease and fully exploit the deposit through the downturns in the market that seemingly inevitably affect a commercial mining operation.

3. *Royalty*—A low primary royalty is obviously in the best interest of the lessee. In addition, he/she should attempt to keep the advance minimum royalties low. This is important in maintaining an attractive cash flow.

4. *Buy Out*—Often, if a very large mineral deposit is discovered, the project developer will desire to purchase the remaining term of the lease and assume title to the land after profitability is attained. If this is likely to occur, the lease should provide for an option to purchase.

5. *Termination*—The lessee should retain the right to terminate the lease upon adequate notice.

6. *Force Majeure*—The lessee requires protection in the event of an "act of God" or other disastrous occurrence beyond his/her control. He/she should be relieved of his/her commitments under the lease until such time as he/she is again able to perform. In many instances, it is recommended that events qualifying as force majeure be listed and also that the responsibilities of the parties during specific force majeure events be specified.

The landowner (lessor) will usually seek to have the following terms inserted in the lease agreement:

1. *Term*—The lessor will not want to tie up the property for an extended period of time due to the desire to accelerate the generation of revenues.

2. *Royalty*—The primary item of interest to the lessor will be the production royalty. He/she will attempt to maximize this payment. It may be based on a set rate per unit of production ($/ton), in which case it should be subject to adjustment for inflation to protect the purchasing power of the royalty dollar. Often the royalty rate will be a percentage of the realization from the sale of the mineral.

3. *Schedule*—The lessor will require diligent development to ensure that he/she begins to receive production royalty revenues at an early stage.

4. *Advance Royalty*—During the time when the developer is not producing the mineral commodity, the lessor will expect to receive advance, or minimum, royalty payments. These payments are intended to further prompt the lessee to put the project into production as soon as possible. Usually, these advance royalties will be recoverable from the production royalty.

5. *Reclamation*—The lessor will, if he/she anticipates the return of the property, probably insist upon a high standard of reclamation of the property. Although reclamation standards are generally prescribed by state and federal regulations, a private landowner may stipulate additional requirements in the lease agreement. This can be particularly likely in the case of prime agricultural or forest lands.

6. *Liability*—The lessor will want protection from third-party suits resulting from lessee's actions, and from such occurrences as increased taxes that may result of lessee's improvements to the property.

7. *Termination*—The landowner should insist on the right to terminate the agreement in the event of a breach by the lessee.

The lease agreement will also contain administrative provisions relative to record keeping, weighing and sampling, payment procedures, and remedies in the event of a disagreement.

7.1.3.3 Purchases

Total ownership is the highest level of control, and it is achieved by a *purchase* of the property. The purchaser in this case has the right to develop the property as desired within the framework and restrictions of all laws and zoning ordinances.

The advantages of total ownership all involve security and include:

1. *Access*—Total ownership is the most secure form of control. Uncertainties as to the ability to gain access are eliminated. Data collection programs may be performed at will. Technical planning for development may be conducted with confidence.

2. *Cost*—The cost of acquisition, once determined, is fixed and not subject to the whims of the marketplace. Financial planning may proceed with secure knowledge that little is likely to change.

Note: A variation of the ownership in fee situation involves a purchase of the property, with the seller retaining a production royalty. In this case, the advantage of an unchanging cost of acquisition will be negated by a royalty rate that is indexed for inflationary changes, or tied to the selling price of the commodity.

3. *Residual Value*—At the end of the primary project life, there may remain some quantity of unmined mineral. If the market value of this particular mineral has increased during the duration of the primary project, it may become a source of additional profits.

Disadvantages of total ownership include:

1. *Cash Flow*—The purchase involves an early commitment of cash, which has a greater impact on the project economics when considering the time value of money. Also there will be holding costs (property taxes) to be paid during the development period.

2. *Risk*—If some occurrence should make the project infeasible or less attractive (such as a sudden, drastic change in the market or regulatory actions), then the buyer may be "left holding the bag" until the project again becomes attractive.

In general, mining companies prefer to purchase only those properties upon which they will be constructing permanent facilities. The lands to be mined are more often acquired by leasing, at least initially.

7.1.4 ACQUISITION PLANNING

Land acquisition must be a team effort. The typical team will consist of field landmen, legal counsel, the project engineer, and a coordinator who has final responsibility.

The acquisition of land for a mineral development project is not a straightforward, simple procedure. Many activities must take place concurrently, and few if any are independent of all other activities. When dealing with private individuals, business representatives, or government administrators, personalities become intertwined with business issues to compound the intricacy of the procedure.

The land acquisition process can be greatly facilitated by the presence of a local populace that is friendly to the project. A generous dose of common sense and sensitivity to public relations will go a long way to secure this friendliness. This is particularly important to the development of a project in an area that is privately owned and does not have a significant history of mineral development.

In the face of initial local opposition, it becomes crucial to promote the positive effects to be enjoyed by the public. Issues such as new jobs, increased payrolls and taxes paid into the local economy should be highlighted for the public. Worker safety and environmental protection plans and requirements should be highlighted. The project engineering staff is the primary resource for the production of this information. The dissemination of the information will usually be made by the landmen who have day-to-day contact with the landowners.

The acquisition process is much enhanced if the public, and especially the local landowners in the project area, are aware of the benefits the project can bring to them as individuals and to the area as a whole. Care must be taken, however, that the competitive status of the project is not compromised by the release of sensitive information. The land acquisition coordinator with assistance from legal counsel should be involved in reviewing all information prior to release for the public. In short, mineral land acquisition planning is not an exact science, and certainly cannot be conducted in a vacuum.

The foregoing recommendations and cautions notwithstanding, a good acquisition plan that is integrated into the overall project development plan will play a key role in the success of the acquisition process. The acquisition plan must be flexible enough to adapt to events as they occur but still provide a framework which rigidly keeps the project objectives in the forefront.

A good acquisition plan accomplishes the following:

1. Identify the project area ownership.
2. Identify project land requirements.
3. Prioritize the acquisition of the parcels.
4. Determine the desired form of control.
5. Establish an acquisition budget.

The order in which these activities are listed is not necessarily the order in which they are accomplished. Rather, they are often advanced in steps on a concurrent basis. For example, the identification of the project land requirements must be done, in general terms, before the project ownership may be established. More precise definition of the land requirements may then be influenced by the results of the ownership search and be subject to change as the plans are developed. Similarly, establishment of an acquisition budget will be based on the desired form of control of each parcel, but the decision as to the desired level of control of an individual parcel may be based on information provided by the acquisition budgeting process. The entire process is a constantly evolving interactive network of information gathering, planning, and action.

7.1.4.1 Identifying the Project Area Ownership

When the overall project area has been determined, it is necessary to conduct an investigation to identify the owners of the properties involved. There are a number of sources of information from which to identify the landowners in the project area. If there are large federal land tracts involved, then the Bureau of Land Management (BLM) records are an excellent source. BLM maintains land offices in the western states as follows:

Office Location	Area of Record
Sacramento, CA	Northern California
Los Angeles, CA	Southern California
Phoenix, AZ	Arizona
Reno, NV	Nevada
Salt Lake City, UT	Utah
Santa Fe, NM	New Mexico, Texas, Oklahoma
Denver, CO	Colorado
Cheyenne, WY	Wyoming, Kansas, Nebraska
Billings, MT	Montana, North and South Dakota
Fairbanks, AK	Northern Alaska
Anchorage, AK	Southern Alaska

Information on federal lands in other states is available in the Washington, DC, office of the director of BLM.

The records in these offices are open to inspection by the public. Information on federally owned properties includes the category of ownership, any tracts that have been disposed of and the nature of such disposal, rights granted, patented mining claims, and other useful data.

Information on state lands must usually be obtained in the capital of the state of interest. State lands that were obtained from the federal government will be identified in the BLM record search. Actions relative to such lands subsequent to the transfer will be contained in the state land records. Since formats for land records vary from state to state, it is advisable to obtain assistance from state land management personnel when inspecting these records. The state records will normally contain similar types of data as the BLM records.

Information on the ownership of private lands is available from several sources. BLM and state records will indicate rights to federal and state lands that have been conveyed to private parties. More detail will be available at the office of the tax assessor of the counties involved. The names of individuals, or trustees who are paying the taxes will be available at this location. The most definitive source for current ownership by private parties will be the county recorders' files.

During investigation of the various sources of ownership information, the investigator needs to focus on the ownership, or other form of control, of the surface, mineral rights, structures and other easements or rights-of-way.

Surface control must be identified. The owner (or owners) of the surface may have issued leases to others for agricultural or other purposes. These leases may include all rights associated with surface ownership, or they may be only for a specified purpose. The surface rights may or may not automatically carry the rights to surface minable minerals. This must be determined by an investigation of the federal, state, or local regulations. In any regard, the controller of the surface must be contacted and legal arrangements made to secure access and rights to develop the properties.

Owners and lessees of those mineral rights that are separate from the surface estate must be identified. It is possible that multiple layers of leases and subleases exist, and it will be necessary to identify and locate all parties with interests in the mineral estate. Interests in all minerals should be identified, including minerals other than those that are the particular objective of the project planners. For example, it is common for oil or gas development to take place on land containing coal or lignite deposits. It is also common for the rights to these minerals to be severed and held by separate parties. Since the development of these minerals may be mutually exclusive, a determination must be made as to which rights are prime.

Owners of structures that exist on the properties must be identified. Homes, barns, fences, roads, pipelines, electrical transmission lines, telephone lines, etc., are examples of such structures. Often, but not always, the owners of buildings will be the owners or lessees of the surface. The market value of these structures should be estimated, preferably by an appraiser knowledgeable in the specific area.

Easements may have been conveyed to other parties for access roads, pipelines, transmission lines, drainage ditches, or utilities. These should be identified as prior rights and the holders of these rights established.

A useful tool for the acquisition process (and beyond) is a computerized property database. This database contains the legal description of the property, the size, ownership, assignments (leases or easements), and a description of all existing structures. Other remarks relative to the project are also committed to the database. These remarks include the purpose for which the parcel is required, the estimated amount of reserves contained, the parcel's priority relative to the viability of the project, its priority relative to the project schedule, and the status of the negotiations with the owner or lessee.

Computerization allows fast access to the current status of a particular parcel and allows the data to be sorted and retrieved according to various criteria such as priorities, size, reserves, proposed use, etc. This sorting and retrieving capability feature allows quick response to questions from the project planners and will facilitate the production of useful status reports. As progress is made on the acquisition, the database is updated. If a parcel is acquired, the terms of the acquisition will be included. Eventually, as the project becomes an operating entity, the information in the database becomes valuable for keeping track of property taxes, depletion of reserves, royalty payments and information relative to the status of regulatory permits and reclamation bonds.

All of the data gathered in the source investigations and included in the computer database should be presented in graphical form on a project land ownership map. This map will likely become the single most useful acquisition planning tool involved in the process. The land map should be available on the same scale as those maps that are used to present engineering plans for the mine and related facilities. The land map will show through the use of colors, symbols, and legend references each parcel of land that possesses unique ownership characteristics. If necessary, acetate overlays will facilitate showing all the data that are pertinent to the project. The acquisition process will be tracked on the land map. As each parcel is controlled, or otherwise changes in status, a notation will be made on the map. In effect, the land map becomes a visual representation of the computer database.

7.1.4.2 Identifying Project Land Requirements

The identification of land requirements is a direct function of the project engineering team. The procedure begins with the determination of the economic mineral resources available to the project. Preliminary exploration programs target a large area of potential interest. At this point, the land ownership identification effort may be started. The area of interest will usually be gradually decreased as the engineering studies zero in on the final targets.

The project planning efforts define the area to be mined, the sequence of mining, and other lands required to support the project.

These include areas required for the following purposes:

1. Mining.
2. Access.
3. Transportation.
4. Facilities.
5. Utilities.
6. Other miscellaneous requirements.

Mining areas include the actual portions of the mineral deposit that will be exploited during the mining operations. In the case of surface mining, the effect on the property is obvious. If underground operations are involved, the effects of subsidence must be anticipated and any required mitigating measures planned.

Access to the project area must be provided for both project personnel and for vendors delivering supplies and equipment to the project. The nearest public road is usually the beginning point. The mining plan identifies the location within the project area to which the access corridor must lead.

Transportation of the raw and finished products must be provided. This may include truck haulage roads, railroads, conveyor belts, aerial tramways, or pipelines. Corridors must be obtained to allow for the installation of these transportation facilities. Transportation routes must be provided both within the project area and for market access.

Facilities requirements include maintenance shops, offices, warehouses, bathhouses, equipment service facilities, outside storage, powder magazines, and processing plants including waste disposal areas. The preferred locations for these facilities will be determined by the project engineers.

Utilities include incoming electrical power, internal power distribution, water supply and distribution, sewage treatment and disposal, and communications lines.

Miscellaneous land requirements could include areas intended to provide a "buffer" between the active operations and the project's neighbors.

As these sites and areas are defined, they should be plotted on the land ownership map. In some instances, it may be desirable to adjust the location of a facility slightly to accommodate the land ownership situation. An example would be the relocation of a shop building in order to deal with a single landowner vs. dealing with two or more owners.

7.1.4.3 Prioritize Acquisition of the Parcels

The project engineering team is the only entity that can place the appropriate level of emphasis on the acquisition of each of the individual land parcels. During the process of creating the project plan, the engineers will determine both the strategic and chronological importance of the various parcels of land.

Strategic importance of a parcel of land is the measure of the economic impact on the project if the parcel is not acquired. A parcel may be strategically important due to the existence of particularly attractive mineral reserves or topographical suitability for the construction of mine structures, or it may provide access to a number of other parcels.

Examples of levels of strategic importance include:

Priority 1—This will include only those parcels that would constitute a fatal flaw to the project if they are not acquired. Priority 1 parcels will normally include the bulk of the mineral reserves properties.

Priority 2—These parcels represent very important, but non-fatal-flaw properties. The economic impact on the project without these parcels will be strongly negative, since alternatives will be much more costly, but the project will still be viable.

Priority 3—Parcels at this level of importance do not represent a threat to the viability of the project. Alternatives are available at similar costs.

Priority 4—The lowest priority. These parcels are not absolutely necessary and may only be desired as slight enhancements or for their "nuisance value" to the project.

Determination of the appropriate level of priority to place on an individual piece of property will often require an economic analysis in which the project economics are forecast with and without the property. Occasionally, this analysis will compare the acquisition of a particular property with the acquisition or use of an alternative property that could serve the same purpose.

Based on the results of the economic analyses, the project engineer should place one of the foregoing priority rankings on each parcel and enter these rankings into the land database and on the land ownership map. The reasons for a particularly high ranking should also be entered in the database.

The chronological importance of a parcel is a measure of the point in time, relative to the project schedule, at which it is desirable or imperative to have control of the property.

Examples of several levels of chronological priorities are listed below:

Priority 1—These parcels must be controlled prior to making the decision to spend any additional money on the project. The parcels are usually those that have a high strategic importance and thus serve to lower the level of risk to the project.

Priority 2—Before the project mining permits may be obtained, these properties must be acquired or otherwise controlled.

Priority 3—These properties are required before the commencement of construction and initial mining activities.

Priority 4—These low-priority parcels are required sometime later on during the project. They may provide for future expansion or extension of the project scope.

The project schedule and land database should reflect these priorities.

7.1.4.4 Identify the Required Level of Control

The project engineer should first determine, for each land parcel, the level of control that best fits the technical and economic needs of the project. Alternative levels of control to be considered are

1. Options to lease or to purchase (usually only considered in the early stages of development).
2. Outright ownership of surface and mineral in fee.
3. Lease of surface and/or mineral.
4. A combination of ownership and leasing.

Results of the project engineer's determination should be transferred to the land ownership map. The final determination will need to be done in conjunction with the landmen, legal counsel, and the acquisition coordinator. The landmen will, through their personal contacts with the landowners, know which forms of control the landowners prefer, and which forms they are not likely to accept. Legal counsel will determine the legal obligations that will be assumed for the various forms of control. The coordinator will consider the overall impact on the project and on the acquisition efforts for other parcels within the project area.

7.1.4.5 Prepare an Acquisition Budget

Preparation of an acquisition budget will involve all the members of the acquisition team. The landmen will appraise the value of the land surface and any structures located thereon. They will also know the landowners' expectations regarding price, bonus levels, and royalty rates. The lawyers will estimate the costs of document preparation, abstract searches, and legal representation. The project engineer will provide the estimates of the costs of various alternatives being considered. An example of such a calculation might be the comparison of the cost of acquiring a given parcel vs. the cost of mining around the parcel.

7.1.5 LAND ACQUISITION CONTRACTORS

At times, it may be preferable for the project developer to hire an independent land acquisition contractor or consultant. This will be true of projects in geographical areas in which the developer has little or no experience. Also a contractor may be preferred for "one-of-a-kind" projects, in which case the developer needs the expertise only once, and it does not make economic sense to hire full-time personnel for the task.

Selection of a contract landman should be based upon a record of proven performance that has been supported by references from other clients. This proven performance should preferably have been in the same general geographical vicinity and ideally on a similar mineral project. When these selection criteria are satisfied, the contract landman can usually be counted on to provide superior services to those that inexperienced in-house personnel could offer.

7.1.5.1 Services Available

Contractors may be hired to perform only a small portion or nearly the full gamut of activities required for the land acquisition program. Normally, however, project engineering services are not included.

A very common practice is to hire contract landmen who will make the landowner contacts, provide appraisals of the value of the property and structures, and make recommendations on forms of agreement. Occasionally, the landman contractor will also provide other assistance in the form of abstract searches and advice on leases and acquisition contracts.

7.1.5.2 Administering the Services of Land Acquisition Contractors

Although the administration of the activities and the contractual agreement with the contract landman will be the responsibility of the acquisition coordinator, it is important that the project engineer understand the relationship completely.

One of the most crucial decisions for the coordinator is to determine the level of authority to be bestowed upon the contractor. This authority may be measured by financial limitations or by the ability of the contractor to commit the project to some particular course of action. The coordinator may elect to provide the contractor with his limits of authority on an overall basis or set them on a parcel-by-parcel basis. The administration of the activities will require the coordinator to provide the contractor with information regarding the acquisition schedule and budget. Since much of this information will be prepared by the project engineer, the coordinator must provide a channel of communication for the exchange of data. The coordinator must insist that the contractor preserve the confidentiality of such data.

A regular system of reporting should be instituted whereby the contractor reports progress on either a regular periodic basis or on a milestone of accomplishment basis.

The coordinator will also oversee the payments made to the contractor. Some contracts specify a fee based on hours spent on the project, while others may provide for a commission type of payment.

7.1.6 CASE STUDY

The "Black Hole Project" is a hypothetical surface mining coal operation, the subject of a case study of a land acquisition project. The Acme Bituminous Co. (ABC) has developed a surface mining operation on approximately 15,000 acres (60.7 Mm^2) in the "Black Hole" area. The Black Hole mine commenced production in 1981 and currently produces 5 million tons (4.5 Mt) of coal annually for consumption at a mine-mouth power generating plant. The plant is owned and operated by the Kandleworth Home Electric Co. (KWH). The case study is a recitation of the history of the development of this fictitious project with emphasis on the land acquisition activities.

7.1.6.1 Black Hole Coal Deposit

In the late 1940s, oil drilling crews reported intersections of coalbeds near the surface in the Black Hole area. In the late 1950s, the state geological survey drilled several holes in a preliminary exploration program that indicated two persistent coalbeds correlatable over a wide area. The results of the drilling and preliminary estimates of the resource potential were published in Bulletin No. 1001, "Coal Resources of the Black Hole Deposit" in 1958. Bulletin No. 1001 referenced approximately 300 million tons (270 Mt) of potentially recoverable coal in two beds underlying 22,000 acres (89 Mm^2) in the Black Hole area.

The land overlying the coalbeds was predominantly privately owned. Small farming operations were conducted on some areas, and Timber Resources Eastern Establishment System (TREES) operated a forest products business on approximately 8000 acres (32.4 Mm^2) in the area of the coal deposit. The local county owned a 40-acre (1.8-Mm^2) tract including a garage that was used for storage of their highway maintenance equipment.

7.1.6.2 Predevelopment Period

In 1963, ABC began reconnaissance drilling on l-mi (1.6-km) centers in the Black Hole area. Their landmen secured landowner permission to enter the properties for drilling. This permission was generally granted in exchange for small sums of money and the promise to repair any damage to the surface that might occur during the road building and drilling operations. In some instances, the landowners requested that the drill site access roads be left in place for their later use during agricultural activities. As the results indicated the potential for commercial coal thicknesses, the landmen secured options to lease on several of the more promising areas. The legal form of these lease-option agreements was prepared by a local attorney who was retained to assist in the acquisition process.

In 1964, ABC hired a consultant to perform a prefeasibility study of the commercial potential for developing a coal mine and a power generating plant in the area. The study indicated that the project could become economically attractive if the price of oil were to increase by 40% above then current levels. During this study, the consultant identified the area of primary interest for mining and processing the coal, support facilities, and the power plant complex. The area of interest comprised approximately 20,000 acres (80.9 Mm^2). Using this initial "buy line," the landmen continued to acquire options to lease within the area. ABC held preliminary discussions with TREES regarding their coal properties and the possibility of leasing them for the project. A nonbinding agreement in principle was reached by the two parties.

By 1966, ABC decided to temporarily suspend its activities in the Black Hole area.

In 1973, the price of oil rose dramatically, and the Black Hole Project became potentially attractive. ABC reviewed all the previously generated information, and revised the cost estimates contained in the consultant's study to more accurately reflect the economics of the current time. This new forecast was sufficiently encouraging that ABC approached KWH with a proposal to develop a mine in the Black Hole reserves, serving a mine-mouth generating facility that KWH would build in an adjacent location. KWH expressed interest, negotiations were held, a coal supply agreement was executed, and the development period began.

7.1.6.3 Development Period

At this time ABC decided to form an acquisition team (the "A-Team"). The team leader was the Black Hole project manager. Other members included the local attorney, the ABC project engineer, and contractor landmen. The number of contractor landmen would vary as the acquisition workload increased or decreased. All the members had other responsibilities, but were expected to make land acquisition a high-priority project in their schedules.

ABC and KWH agreed that each company would be responsible for acquisition of the land required for their respective activities. They also agreed to coordinate their acquisition efforts so as to avoid competing for the same tracts, and to make the most efficient use of the lands acquired. The A-Team decided that its first assignment should be to develop the basis for the acquisition activities.

The first task of the team was to identify the land ownership within the project area. This information was primarily obtained from the county recorders' files. A map of the project area was prepared that included information vital to the acquisition effort. (the A-Map). Preparation of the A-Map was performed primarily by the project engineer with the assistance of the landmen. The map was prepared on a topographic base and included the following information:

1. Landowner property lines (surface and mineral).
2. Homes, barns or other structures.
3. Public roads.
4. Power lines.
5. Pipe lines.
6. Ponds, streams, and diversions.
7. Coal croplines and overburden depth contours.
8. Future mining areas with sequence of mining—preliminary.
9. Planned haul roads and access roads—preliminary.
10. Mine support facilities layout—preliminary.
11. Power plant site layout—preliminary.

A legend was prepared that matched a code number on the A-Map with the following information:

Item	Responsibility
Tract number	Project Manager
Surface owners name, address, and phone	Landmen and Attorney
Mineral owners name, address, and phone	Landmen and Attorney
Number of surface acres in the tract	Project Engineer
Number of coal-bearing acres in the tract	Project Engineer
Estimated tons of coal underlying the tract	Project Engineer
Estimated stripping ratio of the coal	Project Engineer

The second task for the A-Team was to identify the land requirements for the project. This was the responsibility of the project engineer. He used the preliminary mining plans as the basis for determining these requirements. The initial and future mining areas were identified in the mining plan. The locations and sizes of the areas required for support facilities such as offices, shops, warehouses, power lines, haul roads, etc., were selected. These areas were plotted on an acetate overlay of the A-Map. The areas were color coded according to their anticipated use and the acetate was titled the "Requirements Overlay."

The third task for the A-Team in determining the basis for the acquisition activities was to prioritize the acquisition of each tract. Two categories of priorities were to be addressed, namely, strategic and chronological priorities.

The project manager and the project engineer met to review the land requirements and rank the tracts according to their strategic importance to the project. The highest priority was assigned to those tracts that were absolutely necessary to the project. The second priority was assigned to those tracts that were very important to the project, but that would not constitute a fatal flaw if not acquired. The third priority was assigned to those tracts for which alternative areas could be acquired if necessary. Finally, the lowest priority was assigned to those tracts that would primarily constitute a convenience to the project and could be left unacquired if absolutely necessary. Each of the four priority levels was color coded and plotted on an acetate. This acetate was titled the "priority overlay."

The project engineer used the project development schedule to determine the date by which each of the tracts would be required. These dates, which represented the chronological priorities, were then plotted on the priority overlay.

Task number four for the A-Team in their acquisition planning was to determine which form of control would be optimum, and attainable, for each tract. The entire team was required for this effort. The project engineer listed the reason each of the tracts was required and the type of use that was anticipated. The landmen contributed their knowledge of the landowners' preferences regarding the type of agreement (option, lease, or purchase). The attorney then advised the team on the legal ramifications of each form of control relative to the anticipated use for which the project required the tract. The attorney also revised the lease-option form and prepared a standard form to be used for land purchases. The project manager subsequently made the decision as to which form of control should be sought for each tract.

The A-Team then convened to perform task number five, which was to estimate the acquisition costs. The landmen had performed a survey of each tract to determine the current use and the number and type of structures in place. They then made an appraisal of the value of the surface of the property including the value of the structures. The project engineer used the information on the coal reserves underlying each tract to make an estimate of the mineral value of each tract. Using all of this information, a consensus of opinion was then reached as to the most likely acquisition cost for each tract. The chronological priority information was used to forecast when each tract would be acquired. Finally, the project manager estimated the administrative and overhead costs that would be allocated to the A-Team's activities and prepared a schedule of expenditures for the acquisition effort. The result was the acquisition budget. Four months after the A-Team was organized, they had completed the first edition of their acquisition plan.

Beginning in 1974, ABC commenced the phase II drilling program on the Black Hole area. The drill holes were drilled on 1/4-mi (0.4-km) centers. Concurrently, an aerial survey was conducted and topographic maps were made of the area.

Prior to commencing this work, permission was required from the landowners for the access and control required to conduct the drilling and ground control survey activities. The project engineer provided the landmen with maps showing the drilling sites and the access roads that would be required for the drills and the survey crews. The project engineer also gave the landmen a description of the activities that would be conducted on each tract. Using these maps and descriptions, the landmen approached the landowners and negotiated the terms by which access was granted.

It was apparent at this early stage that the 8000 acres (32.4 Mm^2) that were owned by TREES would be required in order for the project to be feasible. TREES was interested in leasing to the project but had two primary concerns. The first concern related to the loss of revenue from their normal forest product operations during the time that the land was being mined. The second concern was the ability of ABC to return the timber land to its premining productivity levels.

The A-Team undertook the tasks of working with TREES to resolve its concerns and bring the properties to the project. The project engineer was the lead player in dealing with both of these matters. Working with the technical and financial personnel of TREES, he prepared an optimum mining plan by which the TREES property would be mined in a sequence that was compatible with its planned schedule for harvesting the timber. The revised sequence minimized the amount of time the land would be nonavailable for timber production. The project engineer then prepared an estimated schedule of production royalty payments that would be received by TREES. These plans and projections satisfied TREES concerns regarding the loss of timber revenues issue.

The concern regarding the postmining productivity of the land was addressed by a joint effort of the A-Team and the technical staff of TREES. Greenhouse-level studies were designed and conducted by the school of forestry of the state university. These studies were augmented by field studies on the Black Hole site. The results of these studies (which were completed in 1977) were sufficiently encouraging for TREES to commit their property to the project under the provisions of the standard Black Hole leasing agreement.

In 1975 at the conclusion of the phase II drilling program, ABC performed the final feasibility study on the Black Hole Project. KWH had refined their studies and submitted a revised fuel requirement schedule. In response, the project engineer had to change the initial mining areas from the previous studies, and to increase the maximum annual production requirements. All of these adjustments were also required to be compatible with the commitments that had been made to TREES. By this time, the drilling programs had identified 275 million tons (250 Mt) of commercially recoverable coal underlying an area of approximately 13,500 acres (54.6 Mm^2).

The changes in the mining plans and schedules made it necessary for the A-Team to revise the A-Map and the acquisition budget. The project engineer provided a new “buy line” for the acquisition plan and the A-Map and its overlays were revised accordingly. In addition to the 13,500 acres (54.6 Mm^2) of coal-bearing lands, the project engineer identified an additional 2500 acres (10.1 Mm^2) that would be required for access and support facilities. Based upon the revised “buy line” the landmen began acquisition of the newly identified tracts. Options on some tracts which were no longer included within the “buy line” were dropped.

In 1975, ABC began collecting environmental baseline data for the purpose of submitting a mining permit application. This data-collection effort was conducted over a two-year period, ending in 1977. Legal rights of access to the tracts required for the data collection efforts were negotiated and executed by the landmen.

During a similar period, the A-Team continued to acquire the required tracts according to the acquisition plan (with constant updates). The attorney and the landmen worked together to begin the process of “curing titles” for the tracts for which acquisition terms had been concluded. This process basically confirms, or identifies, the ownership of the tracts and provides the information required for royalty payment allocations.

Another major accomplishment of the team during this period was the development of a computerized land-management database. The system was developed to serve both the acquisition effort and the management of tracts after they had been leased or purchased.

The basic building block of the database system is the individual tract data base. Information compiled for the target tracts included the following:

Ownership Information	Tract Description
Name(s)	Legal description of the tract
Addresses	Surface acres
Telephone numbers	Coal acres
Percent of ownership	Coal tons
	Stripping ratio
	Current land use
	Structure inventory and values
	Postmining use
	Special reclamation requirements

The land-management database system was designed to be of use during the acquisition effort and also during the active mining phase of the project. During the acquisition process, the database management system was used for keeping a record of the acquisition status of each tract. Offers made by ABC were recorded, and counter offers from the landowner were also recorded. The status of the negotiations were thus monitored, and the next required activity was known. When lease or purchase agreements were executed, the conditions of the agreements were recorded in the database. These conditions included data on purchase price and payment terms (if applicable), bonuses, advance royalties and terms of recoupment, production royalties, taxes, length of the lease term, etc.

During the active mining phase of the operations, the database system was used for managing the leases and the purchased tracts. The system calculates payments required for taxes, advance royalties, rentals, production royalties, and the status of the recoupment of advance royalties. Routine correspondence with lessors such as requests for current addresses are automatically processed and printed by the system. The status of each tract in the mining plan area is monitored as the operation proceeds. The database is updated monthly to show the status of the following activities:

Clearing—acres (square meters) cleared
Topsoil removal—acres (square meters) stripped
Overburden stripping—cubic yards (cubic meters) removed
Coal removal—tons (tonnes) mined, in inventory, and remaining
Reclamation regrading—acres (square meters) graded
Topsoil replacement—acres (square meters) replaced
Revegetation—acres (square meters) initial seeding, acres (square meters) secondary seeding
Reclamation bond status—percentage released.

Reports on the status of each tract could be printed out routinely or upon special request of the landowner.

During 1978 and 1979, ABC performed the development drilling on 660-ft (200-m) centers in the initial mining areas and drilled closely spaced holes to define the limits of the cropline of the coal. This drilling information was required to make the detailed mine design and to plan the equipment specifications. Application was made to the state agency for a mining permit.

Land acquisition continued with concentration on the initial mining area. A situation arose during this time that required the A-Team to make an economic comparison of alternative courses of action. Prior to ABC securing leases in the area, a natural gas producer had completed a well on one of the tracts within the initial mining area. Production from the well was scheduled to continue for at least six years beyond the time when the tract was scheduled for mining. The owner of the well offered to sell the well to ABC for an amount that represented his estimate of the net present value of his projected profits. The ABC project engineer checked the well owner’s calculations of net present value of profits and basically concurred with his figures.

Then the project engineer made two alternate mining plans for the area in the vicinity of the well. In the first alternate plan, the mining operation would by-pass the well. This required leaving attractive coal reserves in the ground and also involved extra operating costs because of a loss of efficiency while mining around the well. The second alternate plan involved completely rescheduling the mining operations to delay the mining of the area around the well until the well had ceased to produce. The costs of the two alternate plans were estimated and compared with the costs involved in purchasing the well and mining through the area as originally planned. The comparison was made on a net present value basis, and showed that the offered purchase price of the well was less than the additional cost and the value of the coal that would be left unmined. Based on that analysis, the A-Team recommended that ABC negotiate with the owner of the well to purchase it and take it out of service shortly ahead of the time when the area was scheduled for the first mining activities. The ABC board of directors accepted the recommendation, and the agreement was executed.

In 1979, the state approved the Black Hole permit application. The approval was contingent upon the securing of the legal right to mine on all of the tracts within the permit area. The A-Team then continued to concentrate on leasing or purchasing the tracts within the initial mining area. This area was secured by the end of 1979, the permit was issued, and construction of the mine and facilities was commenced. Two years later, in 1981, ABC delivered the first coal from the Black Hole mine to the Kalcutta power plant of KWH. By mid-1982, both the mine and the power plant were operating at full commercial levels.

During this time, the A-Team continued its activities. Its efforts were required to choose between alternatives, schedule the acquisitions to be consistent with mining operations, and maintain the commitments to the lessors.

7.1.6.4 Case Study Summary

This study of the hypothetical Black Hole Project illustrates the value of planning land acquisition efforts and the need for close involvement of the project engineering personnel. The A-Team for the Black Hole Project was successful in their endeavors because they had a logical plan and the project engineers' participation ensured that the acquisition efforts were always consistent with the technical and financial goals of the project.

Chapter 7.2
PLANT SITING AND CONSTRUCTION

JOHN E. CAFFREY AND MARKUS J. LADD

7.2.1 PLANT SITE SELECTION

Selecting the optimum plant site location is an iterative process that requires the reduction of various geographic, technical, business, legal, and political considerations to economic terms. Once the lowest-cost location has been identified, then the incremental cost of alternative locations that are desirable for various technical, business, or political reasons can be determined.

One starts this process by assembling all of the known information concerning the project and general plant location. This information would include such items as the following:

Geographic Considerations

1. Topographic maps
2. Planimetric maps

Technical Considerations

1. Geologic maps
2. Ground stability
3. Mine(s) location(s)
4. Water sources
5. Utility sources
6. Transportation systems and capabilities
7. Climate data
8. Plant specifications
9. Refuse disposal requirements

Business Considerations

1. Competitive environment
2. Royalties
3. Tax laws
4. Materials availability
5. Contractor availability
6. Labor pool
7. Housing availability
8. Medical care availability

Legal Considerations

1. Property ownership
2. Surface rights
3. Regulatory requirements
4. Statutory requirements
5. Contractual requirements

Political Considerations

1. Development incentives
2. Environmental limitations
   a. Air
   b. Water
   c. Noise
   d. Subsidence
   e. Aesthetics
3. Employment rates
4. Business posture

As a general rule, it is most economical to have the plant located as close to the mine(s) as possible. This is especially true when run-of-mine material requires a high degree of concentration as transportation costs are normally reduced then. Following is a suggested process for the first iteration of the optimum plant site location.

7.2.1.1 Geographic Considerations

Using a topographic map as a base, and knowing the general plant area requirements, potential plant sites close to the mine(s) are identified. These potential sites should generally be on level ground and situated to maximize the natural advantages available. Flood plains, steeply sloping ground, and highly populated areas should normally be avoided. Surface features including railroads, rivers, streams, highways, roads, power lines, gas lines, and surface structures should be identified from planimetric maps and confirmed through onsite observation. There is no substitute for early and frequent field visits during the site selection process. Rough estimates of the total transportation costs from mine to plant and plant to market should be prepared at this point.

7.2.1.2 Technical Considerations

Using the topographic base maps that have been prepared, each potential plant site should be evaluated technically. Potential sites should be checked for geologic suitability, including proximity to future mining areas, soil and ground stability, and potential water availability. The sites of previous, present, and future mine works should also be located.

Water requirements, including quality and quantity, need to be determined and water sources and costs itemized. Other utility needs, such as electrical power, gas, sewage, and telephone, also need to be determined and estimated costs and lead times for these services established.

Alternative transportation systems from mine to plant and plant to market need to be evaluated for initial cost, operating cost, capacity, and reliability. A comparison of the cost of transportation alternatives for a hypothetical coal mine appears in Table 7.2.1.

Climatological data concerning prevailing wind direction, temperature, and precipitation should also be considered. Oftentimes, all of the potential sites fall under the same category, and site consideration based upon this parameter is not required. Sample precipitation frequencies (Table 7.2.2) and a map of maxima (Fig. 7.2.1) are provided.

Plant specifications should also be reviewed and alternatives considered in the event that one or more sites are eliminated unnecessarily due to falsely rigid physical constraints.

Lastly, the requirements for refuse, tailings, and wastewater disposal must be considered. Often, in the case of high-concentration processes, the disposal of tailings is a larger problem than product handling to the market. Requirements for disposal sites need to be evaluated on a life-of-reserve basis.

7.2.1.3 Business Considerations

Potential plant sites that meet necessary geographic and technical considerations can now be evaluated on a business basis. One's strengths and weaknesses compared to those of the competitors are very important. Transportation advantages to market can make the difference in winning a sale. How the sites rank with each other and the competition needs to be evaluated on a business basis. Parameters to be considered include contrac-

Table 7.2.1. Comparison of Transportation Alternatives

		Plant	Market A	Market B	Market C
Operating Costs, $/ton, destination					
Origin	*Mode*				
Mine	Belt	0.35	—	—	—
	Truck	2.10	—	—	—
	Slurry	0.30	—	—	—
Plant	Rail		6.80	15.90	15.50
	Truck		6.50	17.00	15.00
	Rail/water		—	15.60	14.00
Capital Costs, $ (millions), destination					
Origin	*Mode*				
Mine	Belt	5.5	—	—	—
	Truck	—	—	—	—
	Slurry	6.8	—	—	—
Plant	Rail		—	—	—
	Truck		—	—	—
	Rail/water		—	—	—

Table 7.2.2. Average 24-Hr Precipitation Frequencies, West Virginia Counties

	Frequency in Years				
County	1	5	10	25	100
Barbour	2.36	3.50	4.22	4.79	5.77
Berkeley	2.46	3.95	4.80	5.50	6.75
Boone	2.38	3.48	3.96	4.65	5.45
Cabell	2.38	3.42	3.90	4.52	5.25
Calhoun	2.32	3.37	3.88	4.56	5.40
Doddridge	2.30	3.33	3.90	4.57	5.40
Fayette	2.38	3.54	4.10	4.75	5.70
Greenbrier	2.45	3.75	4.40	4.92	6.00
Hampshire	2.45	3.85	4.70	5.20	6.55
Harrison	2.32	3.37	4.00	4.65	5.55
Jackson	2.32	3.35	3.82	4.45	5.17
Jefferson	2.50	4.20	4.95	5.70	7.00
Kanawha	2.35	3.44	3.93	4.60	5.40
Lewis	2.35	3.40	4.10	4.68	5.60
Lincoln	2.38	3.44	3.93	4.58	5.35
Logan	2.40	3.50	3.98	4.67	5.50
McDowell	2.43	3.64	4.15	4.79	5.70
Marion	2.30	3.36	3.99	4.63	5.50
Marshall	2.25	3.25	3.81	4.44	5.10
Mason	2.34	3.37	3.83	4.45	5.12
Mercer	2.45	3.60	4.25	4.87	5.85
Mingo	2.42	3.54	4.00	4.69	5.52
Monongalia	2.30	3.37	4.00	4.65	5.50
Nicholas	2.39	3.50	4.20	4.75	5.75
Ohio	2.24	3.22	3.79	4.39	5.00
Pendleton	2.46	3.85	4.70	5.30	6.55
Pocahontas	2.44	3.75	4.55	4.97	6.20
Putnam	2.34	3.40	3.86	4.51	5.25
Raleigh	2.40	3.56	4.10	4.75	5.68
Ritchie	2.29	3.31	3.82	4.50	5.30
Summers	2.43	3.61	4.30	4.87	5.90
Tyler	2.28	3.28	3.80	4.46	5.20
Upshur	2.37	3.49	4.22	4.79	5.77
Webster	2.39	3.55	4.30	4.82	5.85
Wetzel	2.28	3.28	3.85	4.50	5.25
Wood	2.28	3.29	3.76	4.37	5.07
Wyoming	2.41	3.56	4.07	4.75	5.65

Source: Soil Conservation Service, US Department of Agriculture.

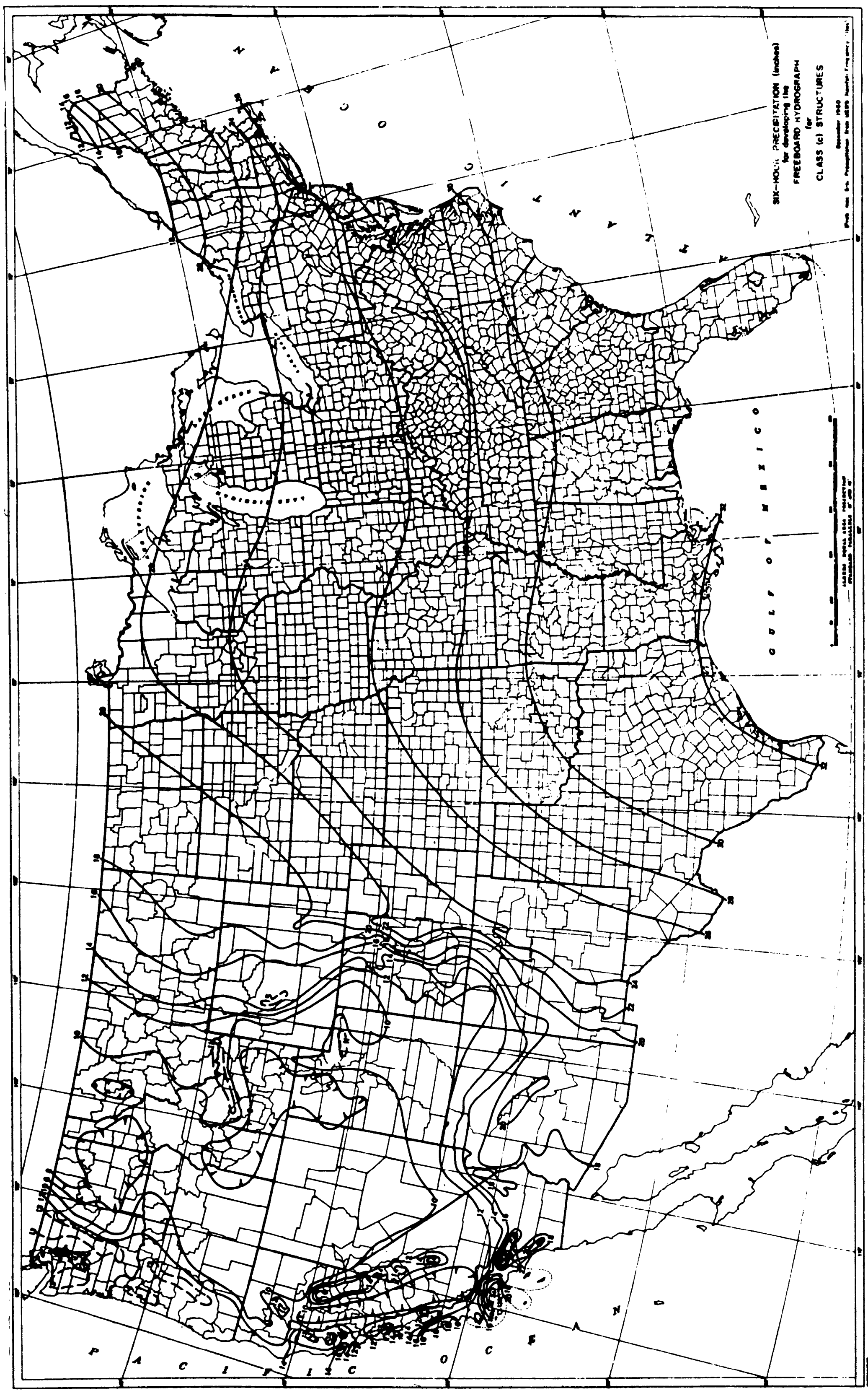

Fig. 7.2.1. Maximum precipitation map. *Source:* Soil Conservation Service, US Department of Agriculture.

tual requirements, royalties, wheelage, property taxes, income taxes, availability of contractors, quality of contractors, availability of labor, quality of labor, availability of materials, and quality of materials. The availability of community services, including school, housing, and medical care, needs to be determined. A final consideration that requires thorough evaluation is the availability of incentives from state and local governments, such as tax credits, municipal financing, and subsidized industrial development programs.

7.2.1.4 Legal Considerations

Potential plant sites that have been evaluated for their geologic, technical, and business characteristics now must be evaluated on legal issues. The starting points for these issues are the property maps showing ownership of surface, mineral, and surface rights for the target sites. Adverse property that is needed for the sites or access must be valued and rated for potential of acquisition. Statutory requirements such as zoning need to be researched as well.

The most significant legal requirements for most mining operations are environmental laws and regulations (see Chapters 3.4 and 7.3). Significant time, effort, and money are required for the preparation of environmental impact statements and regulatory plans. Prior to beginning the permitting process, one should evaluate potential sites for conflict with prohibitions concerning archeological areas, historic sites, endangered species presence, and wetlands protected areas. Statutory and regulatory requirements concerning employee health and safety and public safety may affect structural design requirements and certain construction activities such as blasting. Surface and groundwater protection requirements can be very stringent and should be reviewed as soon as possible for potentially fatal flaws.

7.2.1.5 Political Considerations

The prospective plant site list is in all probability quite short now, and those remaining need to be evaluated on a political basis. The attitude of local governments toward business is very important. One's success or failure often depends on the ability to operate in harmony with local governments and regulatory authorities. The limitation of sites must be assessed, especially concerning legal yet socially unacceptable levels of air, water, and noise pollution. For example, a residential suburban setting is probably not a good location for a copper smelter regardless of the strength of other considerations. Creating employment in high-unemployment areas and improving the tax base can be large assets to ensure continued community support (see Chapter 3.1).

In summary, it cannot be too strongly stressed that site selection is an iterative process. If potential sites are dismissed prematurely, the optimum site location may not be selected. It is important to test the economic valuation of various considerations for accuracy sensitivity. A poor plant location is a decision that must be suffered for the project life.

7.2.2 PLANT CONSTRUCTION

Once the general requirements and facilities for mining, transportation, run-of-mine storage, beneficiation, product storage, and product loading are defined and the general plant site selected, then preliminary engineering for the plant commences.

Reports and discussions concerning the objectives and parameters of the project lead to the preliminary engineering phase of construction. Preliminary engineering is the evaluation of

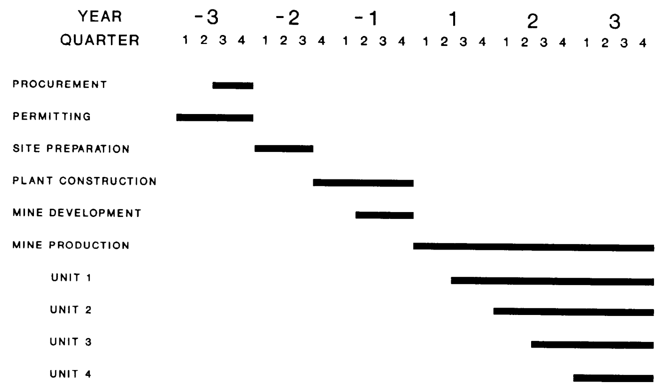

Fig. 7.2.2. Construction and development schedule, mine and plant complex.

potential alternatives leading to feasible solutions that meet the objectives and parameters of the project. Elements that are considered in this process include the following:

1. Local practices for mining, storage, transportation, beneficiation, and refuse disposal.
2. Competition analysis of neighboring operations including operating, transportation, and market advantages and disadvantages.
3. Identification and definition of key success factors.

During the preliminary engineering phase, it is very important that alternative solutions to project requirements are identified and evaluated. Potential alternative solutions are often best identified by a combination of internal engineering efforts and contracted engineering efforts. The former group normally has much greater knowledge of the specific objectives, requirements, and limitations of the project. The latter group, however, can apply a broader perspective to the problems and not be constrained by accepted parameters and assumptions made to date.

Refinement of the preliminary engineering ideas and plans becomes the preliminary design for the project, and a construction and development schedule is prepared (Fig. 7.2.2). The preliminary design usually includes the following:

1. Site selection and supporting documentation.
2. Preliminary facility layout.
3. Preliminary general specifications for plant and supporting infrastructure.
4. Estimated costs of construction and equipment.

The preliminary design becomes the basis for authorization to commit funds for construction and equipment purchase or, in the case of more complex projects, to commit funds for more refined engineering to bring the project feasibility and design within more accurate definition limits (i.e., ± 10%).

The preliminary design also becomes the basis for definitive design, specification preparation, and preparing invitation-to-bid packages for project design, or more commonly, turnkey project design and construction.

If the turnkey approach is selected, an invitation-to-bid package is prepared. Tables 7.2.3 through 7.2.5 outline typical sections of the package. These packages can vary dramatically in complexity, but almost all need to accomplish the following:

1. Ensure that the contractor understands the scope and intent of the project.
2. Ensure that the contractor understands that the preliminary design is a suggested approach, but that he will be held accountable for the feasibility of the design as bid and constructed.

Table 7.2.3. Table of Contents, Request for Proposals, Coal Handling Facility

Part	Description
I	Invitation-to-bid and instructions to bidders
II	Contract
	Exhibit A — Contractor insurance requirements
	Exhibit B — Work requirements
	Exhibit C — Contractor bid
	Exhibit D — Work progress schedule
	Exhibit E — Schedule of payments
	Exhibit F — Federal procurement regulations
	Exhibit G — Independent contractor requirements
	Exhibit H — Cost basis for change in contract price
	Exhibit I — Addendums to contract
III	Drawings and attachments

Table 7.2.4. Sample Bid Sheet, Request for Proposals, Site Preparation, Coal Handling Facility

Item No.	Units	Unit Cost	Total Cost
1. Mobilization-demobilization	LS		$
2. Sediment control structures			
a. Pond #1	LS		$
b. Pond #3	LS		$
3. Clearing and grubbing			
a. Clearing	29.9 ac	/ac	$
b. Clearing and grubbing	16.9 ac	/ac	$
4. Topsoil removal	9,800 bcy	/bcy	$
5. Excavation			
a. Haul road and truck dump	193,800 bcy	/bcy	$
b. Coal storage area	63,000 bcy	/bcy	$
c. Railroad embankment	7,200 bcy	/bcy	$
6. Roadways: gravel	4,500 tons	/tons	$
7. a. Road relocation	1,140 ft	/ft	$
b. Limestone gravel	226 tons	/tons	$
8. Drainage culverts			
a. Roadway culverts	1,380 ft	/ft	$
b. Railroad culverts	360 ft	/ft	$
9. Stream relocation excavation	1,728 bcy	/bcy	$
10. Riprap			
a. Railroad embankment	1,350 sy	/sy	$
b. Other	1,930 sy	/sy	$
11. Concrete collars	14 collars	ea	$
12. Revegetation	16.0 ac	/ac	$
13. Subballast	3,800 tons	/ton	$
Grand Total			$

Contractor: __________

Representative: __________

Title: __________

Date: __________

Table 7.2.5. Drawings List, Request for Proposals, Site Preparation, Coal Handling Facility

Drawing No.	Title
# 1	Plan view of coal handling facilities site
# 2	Embankment sediment dam—1 Riffe branch
# 3	Interim plan plan view—Pond No. 4
# 4	Sediment pond No. 4 cross sections
# 5	Sediment pond No. 4 details
# 6	Interim plan plan view pond Nos. 3 & 6
# 7	Sediment pond No. 3 details
# 8	Sediment pond No. 6 cross sections
# 9	Clearing and grubbing areas
#10	Topsoil removal area and existing utilities
#11	Permit No. 0-26-85 Interim plan baseline a Cross sections 1 of 3
#12	Permit No. 0-26-85 Interim plan baseline a Cross sections 2 of 3
#13	Permit No. 0-26-85 Interim plan baseline a Cross sections 3 of 3
#14	Permit No. 0-26-85 Interim plan-haul road profile and cross sections
#15	Haulageway grade profile permit No. S-5029-86
# 16 (Roll)	Norfolk and Western Railway Co. proposed coal loading facilities (cross sections)
#17	Norfolk and Western Railway Co. proposed coal loading facilities (plan view)
#18	Proposed road and stream changes cross sections
#19	Proposed road and stream changes plan view
#20	Road relocation state Rte. 10, typical cross section for new road
#21	Placement of riprap, typical cross section
#22	Revegetation map
#23	Typical concrete collars

3. Ensure that corporate objectives concerning areas such as discrimination, employment practices, safety, procurement, and insurance are met.

4. Provide sufficient detail in bid breakdown and supporting documentation so that proposals can be evaluated on an objective and professional basis.

5. Provide a schedule of construction that is recommended and the alternative schedule and cost thereof to meet current project completion goals.

During the invitation-to-bid package preparation time, considerable field effort is required to visit and evaluate potential contractors, visit similar installations to better define suggested equipment and specifications, and visit equipment manufacturers to learn as much as possible about the best available technology.

Consider, for example, the evolution during the mid-1960s of underground power centers for coal mines. Equipment was changing from primarily dc-driven to a mixture of ac- and dc-driven equipment. This combination required two electrical distribution systems and an outlay of $80,000 or more for the power center facilities. One company was offering a power center that provided both ac and dc power from one manageable unit using dry rather than fluid-filled transformers, and silicon rather than mercury arc rectifiers. This proprietary technology enabled the purchasers to obtain an ac/dc power center for $20,000 vs. the traditional $80,000 solution. The purchaser could not and did not reveal the proprietary technology to competing suppliers.

After the invitation-to-bid package has been prepared and qualified bidders determined, proposals are solicited from the bidders. The bids received must then be evaluated in an objective manner. Tabulation of the bids is a common method of evaluation and allows ready comparison of facilities, equipment, costs, and schedules. A contract is then drawn and awarded to the recommended supplier. At this time performance requirements are finalized and payment schedules prepared. Retainage or payment for work and/or equipment completed and received less a percentage, commonly 10 to 15%, is the most common method of encouraging performance. Bonding, a more conservative approach, is substantially more expensive, a cost that ultimately is borne by the purchaser.

Project management from the purchaser's viewpoint has the following principal requirements:

1. Ensure that work performed meets specifications.
2. Ensure that equipment delivered meets specifications.
3. Ensure that schedules are followed. Fig. 7.2.2 presents a typical construction and development schedule.
4. Ensure that, if modifications or changes are required to meet project requirements, they are cost effective and documented.
5. Ensure that record-keeping requirements for deliveries, safety, and work performed are adequate to support claims should disputes arise.
6. Verify that progress payments are appropriate.

Samples of a typical contractor's invoice and field inspector's log are shown in Tables 7.2.6 and 7.2.7. Above all, the project manager must ensure that the completed plant will achieve the objectives of the project.

Articles concerning current construction practices and methods may be found in many of the trade and professional journals. Following is a list of some of those that might be consulted:

Engineering News Record (ENR), McGraw-Hill, Inc.

Coal, Maclean Hunter.

Mining Engineering, Society for Mining, Metallurgy, and Exploration, Inc.

AMC Journal, American Mining Congress.

Engineering and Mining Journal (E&MJ), Maclean Hunter.

Table 7.2.6. Sample Contractor Invoice Showing Retainage Calculation

INVOICE

ABC CONSTRUCTION CO., INC.
ANYWHERE, USA

TO: XYZ MINING COMPANY, INC
SOMEWHERE, USA

CONTRACT NO.: OU12345
INVOICE PERIOD: 6/1-6/30/89

ITEM NO.		UNIT COST	INVOICE AMOUNT
1. Mobilization-demobilization	50%	66,000	$ 33,000.00
2. Sediment control structures			
a. Pond # 1	50%	50,000	$ 25,000.00
b. Pond # 3	75%	20,000	$ 15,000.00
3. Clearing and grubbing			
a. Clearing	23 acres	1,500/ac	$ 34,500.00
b. Clearing and grubbing	14 acres	2,500/ac	$ 35,000.00
4. Topsoil removal	4,800 bcy	$3.00/bcy	$ 14,400.00
5. Excavation			
a. Haulroad and truck dump	105,000 bcy	$5.00/bcy	$525,000.00
b. Coal storage area	53,000 bcy	$4.50/bcy	$238,500.00
		TOTAL	$920,400.00

Total Complete through 6/30/89	$920,400.00
Less 10% Retainer	$ 92,040.00
Less Previous Invoices	$571,560.00
Net Amount Due	$256,800.00

Table 7.2.7. Sample Daily Contractor Log

Date: ____________ Day: ____________

Contractor: ____________ Job Name: ____________ Job. No.: ____________

Work Performed Today ____________

Problems—Delays ____________

Subcontractor Progress ____________

Special Assignments ____________

Extra Work	Authorized By	Approx. Price

Equip. Rented	Rented From	Rate

Supervisor's Signature ____________

Weather ____________

Temp. AM ______ PM ______

Safety Meeting ____________

Work Force	No.
Superintendent	
Clerk	
Bricklayers	
Carpenters	
Cement Masons	
Electricians	
Iron Workers	
Laborers	
Operating Eng.	
Plumbers	
Pipe Fitters	
Sheet Metal	
Truck Drivers	
Total	

Equipment	Hrs.

Material Purchased

Chapter 7.3
ENVIRONMENTAL PROTECTION AND PERMITTING

DEAN K. HUNT

7.3.1 INTRODUCTION

7.3.1.1 Importance of Environmental Planning

The increasingly complex regulatory regime has brought environmental planning to the forefront of considerations integrated into the mine development process. In today's world, the integrity of any mining project often depends as much upon the soundness of measures proposed to mitigate environmental consequences, and the operator's ability to negotiate the regulatory maze he is confronted with in procuring required permits, as it does on the miner's traditional ability to extract and transport the mineral resource at the lowest cost.

As regulations for the protection of the environment from the impact of mining change and develop, they have brought with them changes in the methods of mining, in types and utilization of equipment, and in mineral processing and beneficiation techniques, along with changes in the procedures for planning and permitting the mining operation. The overall magnitude of the impact due to environmental restrictions has been dependent on the type of mineral involved, the geographical and topographical conditions at the mine, and the environmental requirements involved. Strict rules often mean not only that the mine will have to take additional measures such as preventing discharges of polluted water, but also that the operation may have to develop totally new concepts of mining in order to remain both in compliance and competitive.

Some examples of areas where changes in mining and related operations have been caused by environmental restrictions are

1. *Topsoil Removal.* Environmental regulations may require that topsoil be removed from the mineral reserve in a separate operation from the removal of other overburden horizons. Additional equipment and labor are needed to remove, segregate, and preserve topsoil.

2. *Runoff Control.* Runoff from mines must typically meet specified effluent limitations. Often, water treatment and sediment control facilities must be constructed prior to mining, and the mining layout and configuration must be designed to facilitate the control of runoff and the prevention of pollution of surrounding waters.

3. *Stream and Wetlands Protection.* Buffer zones to protect streams and wetlands are often required to prevent intrusion of mining into protected areas. Since the geologic location of mineral reserves can conflict with these surface features, difficult questions of stream relocation, wetlands preservation, and reserve loss can result.

4. *Blasting.* Restrictions on blasting limit the times of blasting as well as the blast design. Buffer zones around occupied dwellings and other protected structures are often imposed.

5. *Waste Disposal.* Historically, mine waste was often simply dumped at the nearest convenient location. Today virtually all mine waste must meet standards for the designed and controlled disposal of the waste material.

6. *Overburden Handling.* Overburden handling at one time was as simple as dumping the overburden material on the outslope or in a spoil pile. Most reclamation laws now require consideration of reclamation of the mined area in the handling of overburden.

7. *Processing and Beneficiation.* Discharge of water and air pollutants from processing and mineral beneficiation are today carefully controlled under a variety of environmental programs. Significant revisions have been required in these processes to ensure that the environment is protected and pollutional discharges minimized, while maximizing mineral recovery.

8. *Revegetation.* Revegetation of areas disturbed by mining was at one time left to mother nature. Today most reclamation laws require revegetation of mined areas and restoration of disturbed areas to an acceptable postmining land use.

The attitude that mining operations may be conducted without regard for the environmental consequences of these activities is as foreign to mining development in the United States today as perhaps the idea of environmental regulation itself likely was to miners at the turn of the century. Today every mining operation must contend with a broad range of laws and regulations dealing with a host of environmental concerns including water pollution, air pollution, waste disposal, mine reclamation, and other laws dealing with the protection of the environment.

Occasionally, these environmental programs will impose severe, even prohibitive restrictions on mining since mineral reserves cannot simply be moved to another location. Thus legislation such as the Endangered Species Act,[1] the Bald Eagle Protection Act,[2] the National Historic Preservation Act,[3] wetlands protection under the Clean Water Act,[4] and similar enactments can impose insolvable conflicts with the logical development of a mineral property.

Even where the development of a mineral reserve is not prohibited by environmental legislation, added environmental costs imposed on mineral production can have a direct and significant impact on the domestic industry's viability, particularly where foreign competitors are not subject to the same or similar economic burdens. Environmental regulation can have a dramatic adverse effect on the mineral reserve base. Mineral resources are fixed and finite. Recoverable reserves are that fraction of the resource base that can be mined economically. As environmental prohibitions place additional lands off limits to mining and increase the costs of mineral production, the economically minable fraction of the resource base is reduced.[5]

Nevertheless, our society has made it clear that environmental protection in the United States is an essential and important goal that will be achieved even at the expense of the development of some mineral reserves, and recognition of this basic fact must be included as an important aspect of all modern mine planning.

7.3.1.2 Compliance Strategy: The Continuous Nature of Environmental Planning

The environmental planning process must begin well before the commencement of mining activities and continue through the entire life of the mining project. Once underway, the future viability of a mining project can largely depend upon an effective compliance strategy that combines mineral recovery objectives with regulatory performance goals.

Careful planning at the permitting stage can help ensure that information is gathered in an efficient and effective manner,

1. References in this chapter are listed numerically.

that the application meets all of the minimum requirements for permit approval, that the mine planning is conducted in such a manner as to minimize redesign expenses imposed by regulatory agency review, and that the permit application is reviewed and approved on a timely basis. Failure to fully research permitting requirements and to effectively plan prepermit data gathering can significantly disrupt the start-up of the mining operation and potentially lead to the rejection of an insufficient application.

One of the most serious deficiencies encountered by many operators in the permitting process is to fail to adequately plan the time required to collect baseline data and to provide for public and agency review of the applications. As a practical matter, the preparation of the permit application should begin as soon as the decision is made to begin exploration and prospecting of a specific mineral reserve. Often geologic and hydrologic data can be collected at the same time that exploration drilling and other studies are being conducted to prove out the reserve. Other major tasks and activities that should take place during this early phase include:

1. Identification of the regulatory agencies that will be responsible for permitting the proposed mining operation.
2. Acquisition of all necessary permitting forms and copies of applicable regulations.
3. Identification of possible legal or technical restrictions that may make approval difficult or require special attention in the permitting process.
4. Identification of environmental resources information that will be required and the development of a plan and schedule for collection of the required baseline data.
5. Identification of any special requirements that will be imposed on the operations plan by the environmental protection and permitting regulations.
6. Consultation with regulatory agency personnel to solicit their early comments on the proposed operation and to involve them to the extent practical in the early planning process.

In practice, an operator's greatest aid in obtaining necessary permits will be open and regular discussions with the regulatory staffs involved. Although some agency staff may be hostile to mining, in most cases, government employees are helpful and will work cooperatively with an operator to ensure that applicable regulatory standards are met. Communication with the agencies involved and early planning are more and more becoming necessities if an operator is going to avoid unnecessary problems in the permitting process.

Obtaining necessary permits to open a mining operation, however, is only the first step in full compliance with the underlying regulatory requirements. It is the nature of mining that, as mineral recovery proceeds, changes in the mining operation will be proposed and implemented by the operations staff to improve efficiency and maximize mineral recovery. While such changes may be fully desirable from an operations viewpoint, it must be remembered that permits are generally issued based upon a specific plan for operations submitted to and approved by the regulatory agencies involved. Thus the modern mining engineer must remain ever vigilant to ensure that if operations are to be revised, the permits associated with those operations must also be revised and made consistent with the ongoing operations.

Another major factor that will mandate continued environmental planning throughout the life of most mining operations is the ever-changing nature of the regulatory programs involved. Environmental law is, in fact, a relatively new development, and as such, Congress, state legislatures, and regulatory agencies are in a seemingly never-ending process of reevaluating and revising the regulatory standards involved.

Mines are required to keep abreast of these changing requirements and to ensure that mining operations conform to all applicable standards. Again this can require revisions of the original permits and changes to the ongoing operations to ensure continued compliance with applicable requirements.

Thus it is clear that modern mine engineers must include a new set of skills in their arsenal of mine planning and design techniques. The best mining operation today is one that not only extracts the coal or mineral efficiently and profitably, but is one that is also conducted in full compliance with all permitting and environmental requirements while accomplishing its major objective.

For other discussion of environmental regulations and consequences, the reader is referred to Chapter 3.4.

7.3.2 SCOPING THE REGULATORY REQUIREMENTS

7.3.2.1 Identification of Applicable Regulatory Regimes

Mine permitting requirements in the United States can vary widely depending upon the type of mineral, the method of mining, property ownership, the state or locality in which the mine is to be located, the administrative structure and the laws and regulations of that state, the size and location of the mine or other facility, the type of operation planned, the type of support facilities that will be constructed, the environmental resources affected, and a variety of other factors. One of the confusing aspects of any effort to permit a new mining operation is the multiple levels of government and governmental agencies that may concern themselves with the operation.

As a practical matter, most mining operations are required to obtain at least one and often several permits from either a state or federal permitting authority. Typical permits that may be required for a particular mine could include, for example:

1. Mine reclamation permits.
2. Water quality and discharge permits.
3. Air quality permits.
4. Permits for the construction of dams and impoundments.
5. Waste disposal permits.
6. Waterway encroachment permits.
7. Water withdrawal and water rights permits.
8. Highway encroachment permits.
9. Mining licenses and mine safety permits and approvals.
10. Sanitary sewage permits.
11. Drinking water permits.

As can be seen by this listing, permitting requirements for a particular mine may be complex and varied. In order to simplify the permitting process, some states have attempted to develop a "one-stop" permitting process, designed to ease the burden of multiple permit applications. However, care must be exercised in relying on "one-stop" permit procedures, since there are often permits and approvals required that are not covered by the "one-stop" permit packages.

In identifying applicable permitting requirements, it is essential to understand the regulatory framework at each level of government—federal, state, and local—and to identify the applicable regulatory regimes and permit requirements at each level.

Under the United States Constitution,[6] the federal government has superior authority over any state or local governmental body. Thus, where the federal government determines to pass a law covering a specific area of environmental regulation, such as is the case with the federal Clean Water Act[7] and the federal Surface Mining Control and Reclamation Act,[8] states must either conform their state laws and regulations to the federal mandates or surrender regulatory control to the federal government. Similarly, where, as occasionally exists, an operator seeking a permit is caught between conflicting requirements of state and federal agencies, it is the federal agency and the federal permitting requirements that will prevail.

In any area where the federal government has not passed a law imposing specific regulation, or where the federal law leaves certain authorities and responsibilities to the states, the states retain the residual "police power" to enact appropriate laws to protect the general welfare.[9] This "police power" includes the authority to enact environmental legislation and regulate the environmental effects of mining operations.[10]

In environmental law today, most regulatory programs have evolved into two-level programs. Under this system, a federal agency will typically set minimum nationwide standards, but include provisions allowing the states to have direct regulatory responsibility under state laws that must be consistent with the federal requirements. Under the Clean Water Act (CWA), for example, the federal law imposes minimum standards for the review and issuance of National Pollution Discharge Elimination System (NPDES) permits.[11] Each state has the option, however, of developing an individual state program and receiving approval for the state to exercise NPDES authority within the state's boundaries. Where a state fails to adopt an NPDES permitting program, or where the state program fails to meet the minimum standards of the CWA, the federal Environmental Protection Agency (EPA) is required to assume responsibility for issuing NPDES permits for that state.[12]

Below the federal and state levels of government exist all city, county, and local levels of government. Local governmental bodies are generally considered to have limited authority to adopt environmental restrictions in areas where the federal or state governments have acted. Local governments do nevertheless retain significant land use planning authority through zoning laws, and these local laws can have major impacts on mineral development proposals.

The following discussion addresses the environmental regulatory regime from the perspective of the federal, state, and local governmental agencies involved.

7.3.2.2 Federal Regulatory Regime

The regulatory regime of the federal government will normally come directly into play in one of several circumstances:

1. Where the federal government is the permitting authority under an applicable federal law and there is no state program in effect. For example, the federal Office of Surface Mining Reclamation and Enforcement (OSMRE) will be the permitting authority for coal mining operations under SMCRA where there is no state program for the state in which the proposed coal mine is to be located;[13] and EPA will be the permitting authority for issuance of water discharge permits in states without NDPES authority.[14]

2. Where the federal government is required pursuant to federal law to be the permitting authority for the proposed mining activity. For example, EPA has responsibility for the issuance of Underground Injection Control (UIC) permits under the Safe Drinking Water Act.[15]

3. Where the federal government has permitting authority because there are federal lands involved. For example, the US Forest Service is the permitting authority for issuance of special use permits for any mining operations proposed within the boundaries of the National Forests.[16]

Whenever the federal government is the permitting authority, consideration of additional environmental requirements specifically applicable to federal activities will also necessarily be required. For example, the National Environmental Policy Act (NEPA)[17] requires that an Environmental Impact Statement (EIS) be prepared for every "major federal action" that may have a "significant impact" on the environment. Federal permitting actions have been found to be subject to the EIS requirements of NEPA for proposed mining operations. Where the permitting agency determines that the proposed permitting action is not a major federal action having a significant impact on the environment, a Finding of No Significant Impact (FONSI) will normally be issued with the permitting decision.

In addition to NEPA, the federal government is subject to a host of statutory mandates to protect particular environmental resources. Such mandates include, for example, the Endangered Species Act of 1973,[18] the Bald Eagle Protection Act of 1940,[19] the Fish and Wildlife Coordination Act of 1934,[20] the National Historic Preservation Act of 1966,[21] and the Archaeological and Historic Preservation Act of 1974.[22] Guidance on compliance with these requirements will typically be provided by the responsible federal permitting agency.

Additionally, one-third of the nation's land is owned by the federal government. Any activity on federal lands will necessarily have to comply with the requirements of the appropriate land managing agency.[23] Thus the US Forest Service, the Army Corps of Engineers, the Bureau of Land Management, and other federal agencies exercising jurisdiction over federal lands will play important roles in evaluating the impacts of proposed mining operations on public lands.

Both the Bureau of Land Management and the Forest Service, the two primary land management agencies, have programs for the review and approval of plans of operation for mining. These programs include review of proposed mining plans for their effect on the environmental resources, timber resources, wildlife, recreation, and other resources of the area, along with consideration of appropriate land use planning in effect for the area.

Typical of the broad scope of authority provided to a land management agency are the broad powers granted to the Bureau of Land Management to require specific actions by a mineral lessee on federal lands. The Mineral Leasing Act of 1920 (MLA), for example, controls leasing of federal coal, oil and gas, oils shale, and fertilizer and chemical minerals, and specifically empowers the Bureau of Land Management and the Secretary of the Interior to place conditions in leases to safeguard the "public welfare."[24] Similarly, the Federal Land Policy and Management Act of 1976 (FLPMA) provides the Secretary additional powers to impose lease conditions and requirements for environmental protection.[25] FLPMA contains one of the most sweeping empowerments in federal law:

"In managing the public lands the Secretary shall, by regulation or otherwise, take any action necessary to prevent unnecessary or undue degradation of the lands."[26]

The courts have been protective of the broad power of the federal government to impose lease conditions on federal lands because such conditions and stipulations are an "important mechanism for minimizing the environmental impacts" of mineral extraction.[27]

7.3.2.3 State Regulatory Regime

Despite the broad reach of the federal government, day-to-day mine permitting responsibilities typically rest primarily with the states. Pollution control was, during the first half of the 20th century, viewed as nearly exclusively a state responsibility. The first steps toward the federalization of environmental programs was during the late 1940s and 1950s, when Congress decided to allocate funds for federal research on air and water pollution and to implement measures to deal with some interstate pollution problems. During this period, mining activities were regulated almost exclusively by the states under state statutory and common law.

During the 1960s, however, the general public's perception was that the states had failed to deal adequately with environmental problems. First, many people felt that the standards of pollution control that were enacted by the states were not only insufficient, but also that industries were able to exercise political pressure on state and local governments to keep environmental regulation at a minimum. Secondly, states were faced with almost unprecedented population and industrial growth after World War II, which, in turn, led to an increase in the production of environmental pollutants.

Out of this general perception arose, throughout the 1970s, a comprehensive federal involvement in each area of major environmental concern. However, nearly all of these programs provided that the states would have the opportunity to assume the primary enforcement and permitting responsibility, provided the state program met the standards of and was as stringent as the federal program.

Today the actual permitting of mining operations under nearly all of the major federal environmental laws has been delegated to the states. For example, virtually all states have some form of approved air quality control program, most states with active coal mining operations have assumed primary responsibility for issuance of permits under SMCRA, and most states have assumed responsibility for issuance of NPDES permits under the Clean Water Act.

Depending upon one's view, this state/federal partnership can be seen as either dual regulation, or as a working partnership that forms a foundation for effective implementation of environmental laws. Under either view, it is clear that the mine operator today must normally look to the state first, and then to the local and federal governments to determine the applicable regulatory requirements and standards.

7.3.2.4 Local Requirements

The primary tool of local government in controlling mining development is through the local zoning authority.[28] Significant land-use issues are raised when mining operations are proposed for a particular tract of land. Conflicts between mining and other potentially competing uses are particularly apparent where the mine will be located near municipal and residential areas and where the mineral estate has been severed from the surface estate.

Where the state or federal government has created a regulatory regime for permitting the particular mining operations involved, local zoning may or may not be preempted by the superior state or federal law.[29] In Pennsylvania, for example, the state coal surface mining law specifically preempts all local ordinances purporting to regulate coal mining, "except with respect to local [zoning] ordinances."[30] Additionally, where federal lands are involved, local zoning may not be used to veto a federal program to develop the minerals on the federal lands.[31]

Where the local zoning authority has not been preempted, local governments may adopt ordinances controlling mining. Ordinances that would likely be permissible might include restricting mining in residential districts, providing regulations for the use of roads or highways, and conditioning the use of the land in certain districts.[32] Nevertheless, in practice, few local land-use planning controls and/or zoning restrictions have been developed to deal specifically with mining operations. In fact, in most rural areas, where mines would generally be located, there are no local land-use zoning restrictions at all.

7.3.3 PERMITTING PROCESS

7.3.3.1 Preparing the Permit Application

Permits today are one of the primary mechanisms for enforcement of environmental laws. The purpose of the permitting process is to ensure in advance of mining that the proposed operation will be in compliance with the applicable environmental standards. For the mining engineer, preparing the permit application is perhaps the most complex part of the permitting process. The application will usually identify the permittee, provide information on the proposed activity, and describe how the operation will be conducted in compliance with applicable regulatory requirements. The length of time and complexity of permit preparation can vary greatly depending on the type, location, and size of the proposed mine and the complexity of the applicable regulatory program.

The first step in preparing a permit application is to assemble the necessary permitting forms, guidelines, and the regulations that the permit must satisfy. The best source of the requirements normally is the state environmental and natural resources regulatory agencies. Personnel within these agencies are able to supply information on currently applicable permit requirements, premining data collection requirements, and appropriate forms and analyses.

State and federal permitting programs for mining have progressively become more complex and sophisticated. This is particularly true with respect to the collection of background information on the environment prior to mining, and to the analysis of the potential impacts of mining. Surface water and groundwater quality and discharge analysis, overburden analysis, and revegetation studies are several areas where current regulatory standards may require the mine operator to develop detailed plans, where historically little premining analysis may have been required.

Permit requests are initiated by filing a permit application on the form prescribed by the regulatory agency along with required narratives, maps, cross sections, and auxiliary forms. Typically, the application must be accompanied by a filing fee in an amount prescribed by law and specify whether the application is for a new permit, a revision of an existing permit, or for renewal of an existing permit.

The mine operator should verify that all applicable permitting requirements have been addressed. The key to a successful application is generally a demonstration that the proposed mining plan will be adequate to meet the permit's regulatory requirements. In addition to the potential need to obtain a mining and/or reclamation permit, permits and approvals may be required for safety, water discharges, air pollution control, zoning, transportation, etc.

7.3.3.2 Permitting Check List

Any permitting check list can only be representative of the types of information that are required for a specific permit application. Variations in type of mining, type of permit, state and location requirements, and site characteristics help assure that requirements for one permit will vary from those of another. The check list that follows provides a framework for understanding the scope and breadth of requirements that may be applied to a specific proposed operation.

A. *Applicant and Administrative Information.* The objective of the administrative section of a permit application is to provide the regulatory agency with information on the persons and companies responsible for the proposed mining operation and to address various criteria for permit approval related to the applicant and to the proposed mine generally. A check list of information that typically may be included in the administrative section of an application is as follows:

1. Name, address, and telephone number of the applicant and operator.
2. Officers, partners, directors, and major shareholders of the applicant.
3. Statement of the type of business entity applying for the permit.
4. Statement of current and previous permits held by the applicant.
5. Statement of the mine name.
6. Information on applicant's past violation history and any previous bond forfeitures or permit revocations or suspensions.
7. Proposed term for the permit.
8. Newspaper advertisement, proof of publication, and location where a copy of the application will be available for public inspection.

B. *Property and Right of Entry Information.* Although environmental agencies will not typically render decisions about a mine operator's property rights, many applications today require that the applicant describe the various property interests involved and the documents upon which the applicant relies to conduct mining on the property. Notice of the application and the right-to-file objections will be provided to other property owners potentially impacted by the mining operation. The following provides a check list of documents typically required relating to property and right of entry:

1. Names and addresses of the surface and mineral property owners.
2. Names and addresses of all lease holders.
3. Description of the documents that give the applicant the right to enter the property and conduct mining.
4. Map of the property boundaries.
5. Written consent of the surface owner, if required.

C. *Liability Insurance.* Nearly all mines are required by law to have liability insurance with an approved insurance carrier. Minimum coverage of $300,000 per occurrence and $500,000 aggregate for both bodily injury and property damage is typical. In some states, companies that meet minimum financial requirements may qualify to be self-insured. Information required related to liability insurance may typically include the following:

1. Personal injury and property damage insurance information.
2. A certificate of insurance from a licensed insurance carrier.
3. Where allowed, information for self-insurance approval.

D. *Hydrologic Information.* Surface water and groundwater information is typically required to allow identification of potentially impacted aquifers, surface waters, and water users in the area of the mine. Water quality and quantity data may be required. The following check list indicates the types of hydrologic data often required for a mining application:

1. Surface hydrology background water quantity and quality data.
2. Location of all streams, lakes, ponds, and springs.
3. Description of surface drainage systems.
4. Groundwater hydrology background water quantity and quality data.
5, Depth and horizontal extent of water table and aquifers.
6. Aquifer recharge and storage information.
7. Identification of water supplies potentially impacted by mining.
8. Climatological data.

E. *Information on Environmental Resources.* Collection of environmental data is often required for planning mining activities so that potential impacts on the environment may be identitied and minimized. The level of detail and extent of environmental resources data that will be required for a particular mine may vary significantly depending upon the type, location, and method of mining involved. The following check list indicates the types of environmental data that may be required for a new mining operation:

1. Information on surface water and groundwater resources potentially impacted by the proposed mine.
2. Information on archeological and historic sites around the mine area.
3. Information on fish and wildlife resources and their habitats in the area potentially impacted by mining.
4. Land-use information.
5. Map of buildings, cemeteries, and other structures near the mining operation.
6. Map of public roads near the mining operation.
7. Location of any gas or oil wells in or near the mine plan area.

F. *Soils and Geologic Information.* Minimum required geologic descriptions normally consist of a geologic cross section, drillhole logs, and physical characteristics of overburden strata. Soils information be required to allow identification of topsoil layers to be segregated and protected and to allow planning of the postmining reclamation. A check list of geologic and soils information frequently required with a mining application follows:

1. Map of premining topography.
2. Geology of the mining area.
3. Drillhole logs.
4. Chemical analysis of each stratum to be disturbed by mining.
5. Soil map and description of soils.
6. Location and extent of previous mine workings.

G. *Operations and Reclamation Plan.* A description of proposed mining operations and projected production methods is usually required for every permit application. The application generally contains a statement of the mining method, a statement of the engineering methods to be used for mineral extraction, a description of any mineral processing or beneficiation proposed to be used, and information pertaining to the design, modification, use, maintenance, and removal of mine facilities. Usually, the descriptions must demonstrate how the mining operation will be conducted to minimize the impacts upon the environment and comply with applicable regulatory requirements. The following check list is typical of the types of information that may be required for an operations and reclamation plan:

1. Plan for mining and methods to be used.
2. Size, sequence, and timing of the proposed mining operation.
3. Description of equipment to be used in the mining and processing operations.
4. Description of mine facilities.
5. Plan for overburden handling.
6. Plans for spoil and waste disposal.
7. Plans for all dams, embankments, and impoundments.
8. Plans for surface water and groundwater monitoring.
9. Description of water and air pollution control facilities.
10. Plan for blasting.
11. Plan for reclamation of the mine site, soil stabilization, grading, and revegetation.

H. *Environmental Mitigation Measures.* For each specific type of permit, a prediction of the environmental impacts of the proposed mining operation, along with plans for environmental mitigation measures that will be incorporated into the mine design, will usually be required to ensure that the anticipated impacts are within the regulatory limits. The following check list is typical of the types of information that may be required for specific environmental mitigation measures:

1. Prediction of environmental impacts of the proposed mining in relation to the requirements of the permit application.
2. Plans for mitigation of any prohibited impacts.
3. Analysis of the projected impacts of mining after application of mitigation and environmental protection measures.

I. *Transportation Plan.* Transportation to and from and within the mine site often has additional significant impacts on the public and the environment. Mining operations today are frequently required to submit a plan for the use of any public roads to transport mineral from the mine and designs for any transportation facilities constructed with the mine. The following list depicts the type of information often required for transportation facilities:

1. Plans for transportation routing, including the description of proposed routing over public roads.
2. Plans and approvals for relocation of any public roads.
3. Plans for roads to be constructed or used within the mining area.
4. Plans for conveyor systems and rail facilities to be constructed for the mining area.

J. *Bonding.* Prior to the issuance of a permit, many programs require that the operator post a bond to ensure the completion of measures required to protect the environment and to reclaim the land. The following depict the type of information often required in relation to bonding requirements:

1. Determination of the amount of bond required.
2. Description of the type of bond that will be provided upon permit approval, e.g., cash bond, surety bond, letter of credit, self-bond, etc.

K. *Confidential Information.* As permit applications and the permitting process become more complex, the risk of exposure of proprietary information related to the mineral reserve and proposed mining processes increases. Most regulatory programs provide a mechanism for the applicant to specially mark confidential information in order to prevent its disclosure to competing companies.

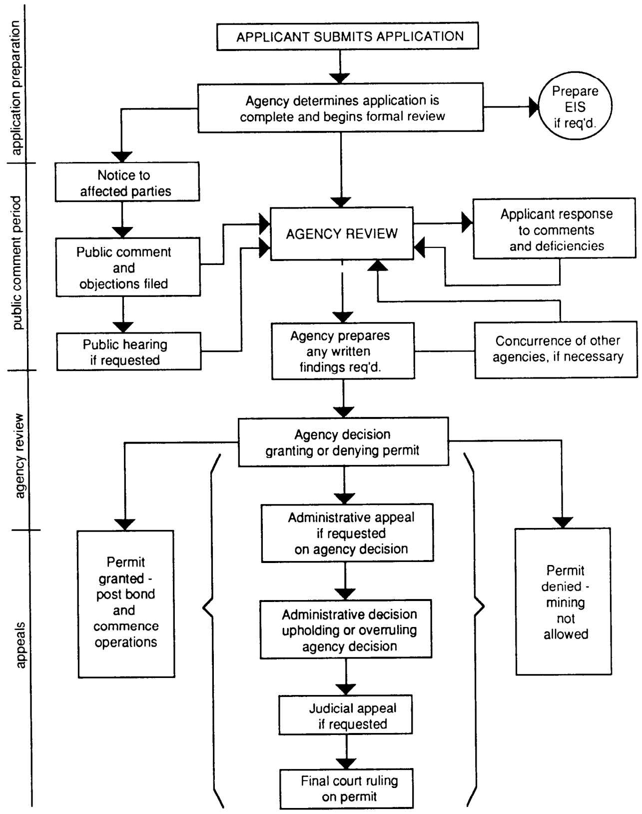

Fig. 7.3.1. Typical permitting process.

7.3.3.3 Agency and Public Review of Permit Applications

The processing of a permit application begins with the submission of a completed application for a new permit, permit revision, or permit renewal to the regulatory authority and ends with the final decision to approve or deny the application. A flow diagram for a typical permitting process is shown in Fig. 7.3.1.

Agency review begins when the applicant officially files the permit application. Once the agency determines that the application is complete, they begin the formal review process, notify affected parties, and begin a public comment period. In most cases, members of the public will be able to both submit written comments and request a hearing on the merits of the particular application. If the permit is granted, the operator may be required to post a bond. Upon approval of the bond, mining operations may commence.

Most agencies will conduct a detailed technical review of each application received to ensure that all applicable regulatory requirements are met. As part of this technical review, the applicant is notified of any deficiencies identified in the application as submitted and provided an opportunity to respond to the agency comments and to correct deficiencies. Failure to provide an adequate response or to correct any permit deficiency could result in denial of the application.

Where the permitting action involves an action by a federal agency, additional reviews may be required to ensure that the environmental impacts of the proposed activity are fully assessed under the applicable federal laws. For example, each federal permitting action will require consideration of the requirements of the National Environmental Policy Act. NEPA compliance will usually result in either the issuance of an environmental impact statement or a finding of no significant impact for the proposed activity.

7.3.3.4 Public Participation

Virtually all environmental laws require opportunity for public participation in the permitting process. Notice of the submittal of the permit application is usually advertised in a local newspaper, with information provided on the schedule for public comment, the availability of the application for public review, and the location where comments may be submitted and hearings requested.

People who have an interest that may be adversely affected have the right to submit comments and objections to the issuance of the permit during the public comment period. Requests for hearings or conferences on the application must also be submitted within the time provided for public comment.

Affected persons who filed comments on the application, and the applicant if the permit is denied, may appeal the agency's decision through an administrative or judicial hearing. In an administrative hearing, evidence will be accepted, and the administrative law judge or review panel will render a decision to uphold or overrule the agency determination. Additional appeals from a final administrative decision can be brought in appropriate state or federal courts if desired by an adversely affected party.

7.3.3.5 Permit Terms and Conditions

Each permit typically is issued for a specified term, with the permittee having the option of applying for a permit renewal prior to expiration. Once approved, the permit can conceptually be viewed as an agreement between the permittee and the regulatory authority to conduct the permitted activity in accordance with the terms and conditions of the permit and the regulatory program.

In addition to the general requirement that activities conducted under the permit be in compliance with the permit and regulatory requirements, a regulatory agency may also impose special conditions on the proposed mining operation to ensure compliance with specific requirements. Such conditions, unless appealed by the permit applicant, will have the force and effect of a regulation when applied to the particular mining operation.

Failure to conform mining operations to the terms and conditions of the permit can result in citizen complaints and/or enforcement action by the regulatory agency involved. If an operator continues in violation of an agency order or has developed a pattern of noncompliance, the more serious steps of permit suspension, revocation, and bond forfeiture are possible. Additionally, an operator with outstanding violations or a history of noncompliance may under some programs be denied the opportunity to receive approval of future permit applications.[33]

7.3.3.6 Overview of EPA's Permit Review Process

A typical permit review process is provided under the EPA's NPDES, RCRA, UIC, and PSD permit processes.[34] In each case, the permit process begins with the submission of an application for a permit, a permit modification, or the reissuance of a permit.[35] In some cases, however, EPA may decide to take action on its own initiative to require a change in a permit.

EPA may handle the permit request in either of two ways. First, it may tentatively decide to grant the request and issue a draft permit for public comment. Second, if it is clear that the application does not meet regulatory requirements, the application may be denied prior to preparation of a draft permit or public comment.

If the permit application is accepted for review, EPA will prepare a draft permit. Along with the preparation of the draft permit, EPA will also prepare a "statement of basis,"[36] a "fact sheet,"[37] and compile an "administrative record."[38] After the draft permit has been prepared, public notice and opportunity for public comment must be provided.[39] The minimum public comment period is 30 days after notice has been issued.[40]

When requested by public comment, and if there is a significant degree of public interest, a public hearing will be held.[41] A public hearing may also be held at the discretion of the local EPA administrator, when he or she determines that it would be helpful to the clarification of issues.[42]

After the close of the comment period,[43] EPA will complete its review for each individual permit requested and render a decision either issuing or denying the permit, or requesting additional information from the applicant.[44] The agency's final decision must be in writing, must include a response to comments,[45] must be based on the administrative record,[46] and must be sent to the applicant and each person who submitted written comment on the permit or requested a copy of the decision.[47]

Any person adversely affected by the decision who filed a timely comment or participated in a public hearing on the application may file an appeal of EPA's decision.[48] Appeals are taken on the formal record of the decision.[49]

The exact form of the appeal will depend upon the type of permit involved. EPA has both a conventional administrative review process,[50] and a special nonadversary panel hearing process that may be used for certain initial licensing decisions involving NPDES permits.[51] Fig. 7.3.2 depicts schematically EPA's conventional permitting procedures.

For RCRA hazardous waste permits, UIC permits, and PSD permits, a request for administrative review must be filed within 30 days after notice of the permitting decision, unless a longer time is allowed. The petition for review must include a statement of the reasons for review, including a demonstration that any issues being raised were raised during the public comment period. The petition must also show that the decision on the permit was based upon clearly erroneous findings of fact or conclusions of law or upon an exercise of discretion that is so important that it should be reconsidered.[52]

Once such an appeal has been filed, the petition for review will be referred to an administrative law judge who will establish a briefing and argument schedule and render a decision on the administrative appeal. Appeals from the administrative law judge can be made through judicial review.

The process of review for NPDES permits vary slightly from other EPA permit decision appeals. An appeal of a decision on an NPDES permit begins with the filing of a petition for review, stating the legal and factual issues involved and the basis for claiming error in the permitting decision. Except for "initial issuance" permits qualifying for the nonadversary panel review procedure, the NPDES review hearing will be a formal evidentiary hearing with formal procedural requirements for filings, service of notice, rules of evidence, etc., being followed.[53]

Again the petition for review will be referred to an administrative law judge who will render an administrative determination on the appeal. Subsequent appeal from the decision of the administrative law judge may be made to the Administrator[54] and then to judicial courts.

7.3.3.7 Overview of SMCRA's Permit Review Process

The permit review process for coal mining operations under SMCRA is similar to the EPA review process and begins with the submission of an application for a permit, permit modification, permit revision, or permit renewal.[55] Additionally, the surface coal mining permitting agency must also review existing permits and may require reasonable revisions of a permit during its term.[56]

The first step in permit review is to determine whether the application is administratively complete. An "administratively complete" application is one that addresses each requirement of the regulatory program under SMCRA and contains all information necessary to initiate processing and public review.[57]

Once the application is deemed administratively complete and accepted for technical review, the applicant will be notified to proceed with public advertisement of the application. The advertisement must be placed in a local newspaper of general circulation in the locality of the mine and be published at least once a week for four consecutive weeks.[58] A public comment period must then be provided that lasts a minimum of 30 days from the last date of advertisement.[59] When requested by public comment, an informal conference with the regulatory agency may be had.[60]

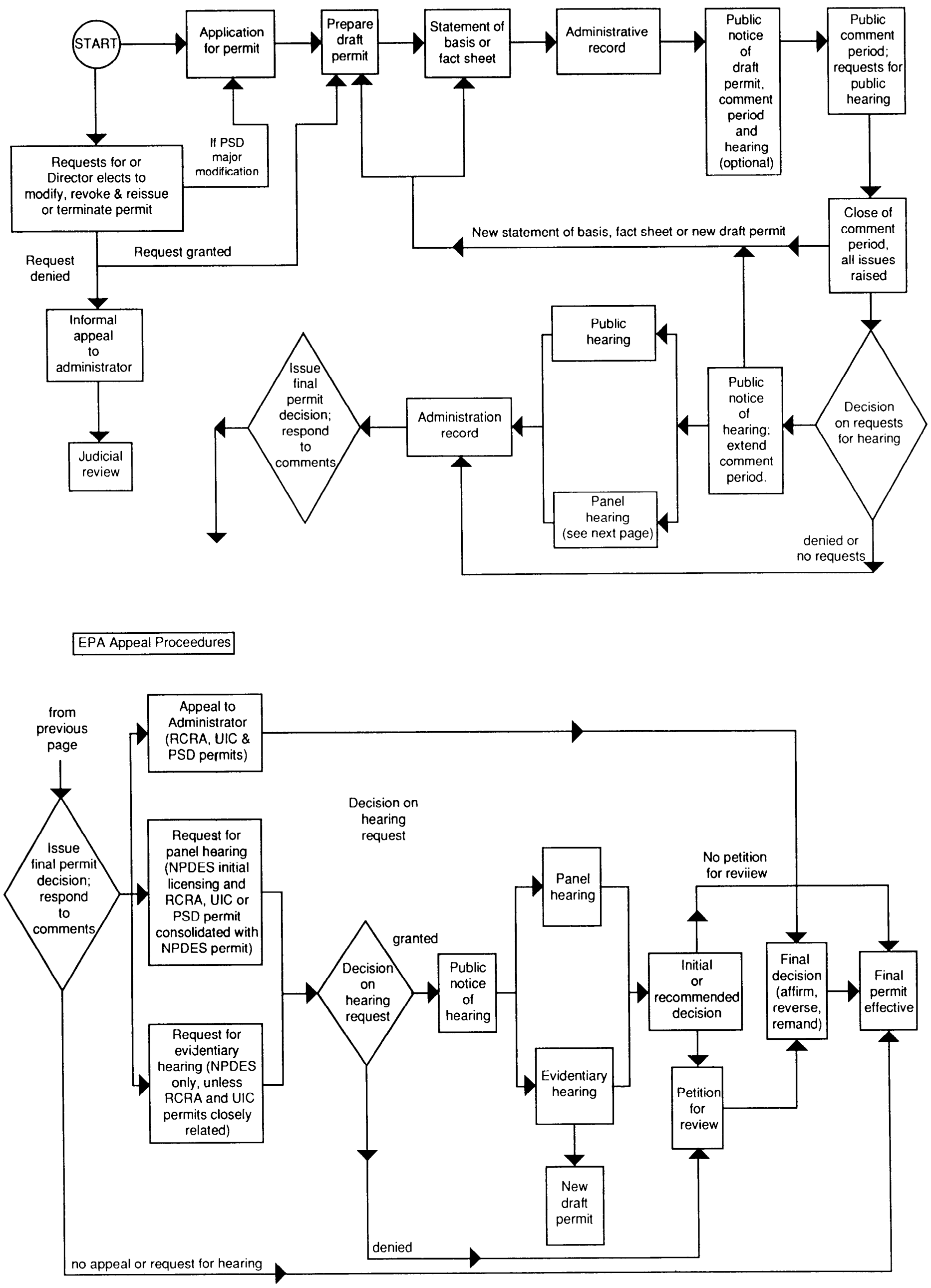

Fig. 7.3.2. Conventional EPA permitting procedures. *Source:* 40 CFR, Part 124, Appendix A.

After close of the comment period,[61] the permitting agency will complete its technical review and render a decision either issuing or denying the permit. The agency's final decision must be in writing, must include certain specific written findings,[62] and must be sent to the applicant, to each person who submitted written comment on the permit, and to local government offrcials.[63]

Any person adversely affected by the decision may file an administrative appeal of the agency's decision.[64] The appeal process is a conventional administrative review, which must be on the record and adjudicatory in nature.[65]

A request for administrative review must be filed within 30 days after notice of the permitting decision is provided to the applicant. Once the request is filed, the agency must schedule a hearing within 30 days of the request and must issue a final decision within 30 days after the close of the hearing record.[66] An appeal from the administrative hearing decision can be made to judicial courts.[67]

7.3.4 OVERVIEW OF SPECIFIC PERMITTING AND REGULATORY PROGRAMS

This segment discusses several of the major permitting and regulatory programs that control various aspects of mining and mining-related activities.

7.3.4.1 Water Pollution Control Programs

Water pollution from mining operations is generally associated with three major sources: ongoing mining operations, abandoned or orphaned mines, and processing and beneficiation plants. Control of these pollution sources is provided for, to varying degrees, under the federal Clean Water Act (CWA)[68] and associated state water quality control programs.

For mining operations, the backbone of the CWA lies in the requirements under Section 402 for obtaining National Pollution Discharge Elimination System (NPDES) permits.[69] Under the NPDES program, any operation that has a "point source"[70] discharge of "pollutants"[71] into "waters of the United States"[72] must obtain a permit and comply with specific requirements and standards.

The basic test of a point source is whether the discharge is a "discernible, confined and discrete conveyance."[73] In mining operations, point source discharges include pumped or gravity drainage from mines that pass through a pipe, ditch, or channel; discharges from ponds; discharges resulting from processing or beneficiation operations;[74] discharges from sanitary wastewater facilities; discharges from other treatment facilities associated with the mining operations; and channeled discharges from mining pits and surface work areas.[75]

Prior to permit issuance, applicants must demonstrate that they will comply with specific effluent limitations intended to protect the nation's waters. The effluent limitations applicable to any source are based on two factors. First, the source must comply with any technology based effluent limitations established for that industry.[76] Second, the discharge must conform to the water quality standards for the receiving stream and to the terms of any area or basin-wide water quality management plan.[77] Limits on the discharge of any toxic pollutants will also be incorporated in the permit if applicable.

Applicants for NPDES permit must submit data on the expected outfall location, the expected discharge dates, the expected flows, sources of pollution and treatment technologies, production, effluent characteristics, available technical evaluations of the wastewater treatment, and any other information the applicant wishes to have considered in the review process.[78] After review of the information submitted by the applicant and consideration of public comment, an NPDES permit containing specific effluent standards applicable to that discharge point may be issued. The permit will be issued for a specific term, with the permittee having the option of applying for reissuance of the permit prior to its expiration.[79]

Once an NPDES permit is obtained and site-specific discharge limits established, compliance with the requirements of the permit during its term constitute compliance with the CWA for enforcement purposes.[80]

An additional consideration under the CWA is the requirements applicable to "nonpoint" sources of pollution governed by Section 208 of the CWA.[81] Mine discharges not collected into a point source, such as unchannelled surface flows and discharges to groundwater, may be governed by a Section 208 plan for the region encompassing the proposed mine. Although Section 208 does not establish a permitting program, the requirements to collect and treat such unchannelled flows are often included in other permits for a mining facility.

7.3.4.2 Corps of Engineers' Section 404 and Section 10 Programs

A special category of permitting under the Clean Water Act (CWA) is the requirement to obtain a permit from the Corps of Engineers under Section 404 of the CWA for dredging and filling of waters and wetland areas.[82] Under Section 404, a permit must be obtained from the US Army Corps of Engineers before any dredge[83] or fill[84] activity can occur in waters of the United States.[85]

Special restrictions are imposed for any activities that are proposed to encroach upon a wetland area. The implementing regulations define a wetlands very broadly and can be extended to include swamps, marshes, bogs, and any other area possessing aquatic-type vegetative growth.[86] The Corps' decision whether to grant a permit for such activities is based upon an evaluation of the probable impacts of the proposed activity and its intended use on the public interest. A permit will be granted if, based upon an evaluation and balancing of the public interest, the District Engineer determines that issuance would not be contrary to the public interest, and that the proposed activity will comply with guidelines established by the EPA under Section 404(b)(l) of the CWA.[87]

Under Sections 404(b) and (c) of the CWA, EPA retains authority to develop guidelines for activities that will be allowed in wetlands and to veto any Corps of Engineers' permit that the EPA feels does not comply with the standards of the guidelines.[88] Before any permit can be issued under EPA guidelines, two general conditions must be met: (1) the applicant must demonstrate that there is no economic alternative to the proposed wetlands disturbance, and (2) measures must be taken to minimize and possibly to mitigate the damage.[89]

In addition to the Corps' permitting authority over dredge and fill activities under Section 404 of the CWA, the Corps also has permitting responsibilities under Sections 9 through 14 of the River and Harbor Act of 1899.[90] Section 9 prohibits the construction of any dam or dike across any navigable water of the United States without Congressional consent and a permit issued by the Corps.[91] Section 10 prohibits the unauthorized obstruction or alteration of any navigable water of the United States.[92] Section 11 allows the Corps to establish harbor lines and prohibit construction of piers, wharves, bulkheads, or other works channelward of the harbor lines. Section 14 provides the Corps with authority to allow temporary occupation or use of any sea wall, bulkhead, jetty, dike, levee, wharf, pier, or other work built by the United States.[93]

By far the most frequently encountered of the Corps' permitting requirements under the Rivers and Harbors Act is the requirement to obtain a Section 10 permit. Under Section 10, the Corps requires permits for any structure in or over any navigable water of the United States, for the excavating from or depositing of material in such waters, and for any other work[94] affecting the course, location, condition, or capacity of such waters.[95]

Each application for a Corps' permit must include a complete description of the proposed activity; the location, purpose, and need for the proposed activity; scheduling information; the names and addresses of adjoining property owners; the location and dimensions of adjacent structures; and a list of approvals required from other agencies. If the activity would involve dredging or the discharge of dredge material, the application would involve dredging or the discharge of dredge material, the method of dredging or disposal, and plans for the activity.[96]

Once the application is determined to be complete, the Corps will issue public notice and conduct a review of the permit application.[97] Permits for permanent structures will normally be issued for an indefinite duration with no expiration date. Permits for construction work, discharge of dredged or fill material, or other activity will normally be issued for a specific time period, with extensions of the permit possible upon approval of the District Engineer.[98]

7.3.4.3 Air Pollution Control Programs

Air pollution from mining activities can stem both from fugitive dust emissions from mine pits, haul roads, and transfer points, and emissions from processing and beneficiation plants. Regulatory control over these pollutants is based upon the requirements of the Clean Air Act (CAA)[99] and state and local laws, rules, and regulations that implement the programs established by the CAA.

The CAA establishes a complex system of interlocking regulations covering air pollutant emissions in all areas of the United States. Clean air areas[100] are governed by a detailed system of area classifications and air quality increments designed to prevent significant deterioration of air quality.[101] Dirty air areas[102] are governed by another set of rules intended to permit some industrial growth while moving toward attainment of acceptable air quality.[103]

Each new major source of air pollution must receive a permit from the state in which it is located.[104] In some cases, EPA retains permitting authority and will issue the permit itself. Additionally, some cities and counties have additional air pollution control requirements that must be met on a local basis.

To implement the CAA, each state develops a state implementation plan (SIP), which upon EPA approval,[105] sets forth the basic strategy by which the state will enforce air pollution standards to meet limitations for the ambient air quality.[106] Under SIPs, each new or modified "major source"[107] of air pollution must apply for a state permit and demonstrate how the source will meet "applicable portions of the control strategy" or how the course will be in accordance with "attainment or maintenance of a national standard."[108]

A review of typical requirements of state program SIPs indicates that most states provide the applicable state air pollution control agency with authority to control fugitive dust at mine sites. Possible control measures include application of water or other dust suppressants and control of dust on roads. Mineral processing is also typically the subject of discussion under most SIPs where mining operations will be located. These standards usually include limits on emissions from specific stationary sources of pollution by application of a variety of pollution control equipment designed to prevent uncontrolled emissions.

For clean air "attainment" areas, the prevention of significant deterioration (PSD) program establishes a detailed system of air classifications and air quality increments limiting the growth of "major emitting sources" or "major modifications."[109] Facilities that qualify as major sources in clean air areas are subject to PSD permitting requirements. An applicant for a PSD permit must demonstrate that it will meet all applicable standards including those applicable to ambient air quality and to the PSD increments.[110] Additionally, a source impact assessment is required to determine if emissions from the proposed source will impair visibility or adversely impact soils or vegetation.[111]

Approval or disapproval of a major source permit will depend primarily on the direct emissions of that source. However, when a new source will cause specific, well-defined, and reasonably quantifiable indirect or secondary emissions, these emissions must also be considered in the PSD increment evaluation and air quality impact assessments.[112]

In addition to standards imposed under the PSD program and under SIPs, the CAA also imposes performance standards on certain new sources of air pollution. These new source performance standards (NSPS) apply to particular categories of sources for which the EPA has established standards and for which construction is commenced after the date on which the EPA issues the proposal for the new standard.[113] These sources require permits regardless of the amount of their emissions, and include such mining-related activities as aluminum ore reduction, copper smelters, ferroalloy production, lead smelters, zinc processing, phosphate rock preparation, rock, gravel and sand quarrying, and certain coal preparation plants that process more than 200 tpd (180 t/day) of raw coa1.[114]

A final category of air emission covered under the CAA is the emission of certain hazardous pollutants. These hazardous air pollutants for which no national ambient air quality standards exist are regulated under the National Emission Standards for Hazardous Air Pollutants (NESHAP) program. NESHAPs apply to pollutants that the EPA believes may result in an increase in mortality or an increase in serious irreversible, or incapacitating irreversible, illness.[115]

NESHAPs are emission limitations that are applied like new source performance standards at both existing and new sources and cover such pollutants as asbestos, beryllium, mercury, and vinyl chloride.[116]

7.3.4.4 Hazardous and Solid Waste Programs

In 1976, Congress passed the Resource Conservation and Recovery Act of 1976 (RCRA).[117] RCRA established a federal program to control the environmental impacts of solid waste management (including mining waste), established minimum standards for solid waste disposal, and established programs to encourage resource conservation and recovery.[118] Included in RCRA is a complex program for the approval of state programs for the permitting and regulation of solid waste facilities, and a federal permitting program applicable to hazardous wastes.

Solid waste under RCRA is defined very broadly and includes "garbage, refuse . . . and other discarded material, including solid, liquid, semisolid, or contained gaseous material."[119] Even though mining wastes are classified as solid wastes under RCRA, Congress also adopted certain exemptions for mining waste from full regulation until studies could be completed by the EPA and determinations made with regard to the most appropriate approach to regulation of the mine waste (the Bevill amendment).[120] The EPA currently exempts 20 categories of waste under the Bevill amendment[121] and has ruled that five categories of mining waste should be regulated as hazardous wastes under RCRA.[122] Further, EPA's regulations list two other important exemptions from RCRA's hazardous waste permitting provisions. These include:

1. "Mining overburden returned to the mine site"[123] that is defined as "any material overlying an economic mineral deposit which is removed to gain access to that deposit and is then used for reclamation of a surface mine;"[124] and

2. Disposal of coal mine wastes or overburden for which a permit is issued or approved under SMCRA.[125]

Under these exemptions, certain mining operations are not required to obtain a hazardous waste disposal permit under RCRA. However, the exemptions do not prohibit EPA or the states from regulating mining waste disposal under nonhazardous solid waste management programs or mean that mining operations are totally exempt from RCRA. A state may regulate mining waste under state solid waste programs, and there are often materials used and discarded at a mine site or processing facility that will not fall within the exemptions. Such waste may include mine-site garbage, oil and machinery fluids, and any other material not generated as part of the mineral extraction and processing. Such materials must be disposed of in accord with state solid waste program requirements, and in accord with hazardous waste management requirements, if they exceed standards established by EPA for ignitability, corrositivity, EP toxicity, or reactivity.[126]

RCRA requires a permit for the "treatment," "storage," and "disposal" of any "hazardous waste."[127] If a hazardous waste management permit is required, the operator must submit a detailed permit application that demonstrates compliance with all of the specific performance and environmental protection standards established by the EPA.[128] Included among these requirements are detailed standards for facility construction and design, groundwater monitoring and protection, and waste material storage and treatment.[129] Once issued, compliance with the terms of the hazardous waste permit constitutes compliance for purposes of enforcement with RCRA.[130]

Requirements for nonhazardous solid waste disposal permits are determined on a state-by-state basis and depend upon the requirements of the particular state program. State program requirements for nonhazardous waste have varied greatly, both from state to state, and within individual states for different industry segments. However, under EPA's guidance, each state program will include minimum requirements for permitting, public participation, and environmental protection.[131] Under RCRA, EPA's guidelines are intended to achieve that level of protection necessary to ensure that there is "no reasonable probability of adverse effects on health or the environment from the disposal of solid waste."[132]

Most states have permitting programs and requirements to control mine-site development, operation, and reclamation. Often, where a state has a comprehensive program to specifically regulate and permit mining operations through a natural resources or mining agency, mine waste disposal is regulated under the specific mining permit and not under the solid waste disposal permit program. Under such programs, mine waste disposal considerations are included as part of the review of the mining permit.

7.3.4.5 SMCRA's Coal Mine Permitting Requirements

An exception to the EPA's extensive authority in the environmental area is the area governing the reclamation of coal mining operations under the federal Surface Mining Control and Reclamation Act (SMCRA).[133] Under SMCRA, the responsibility for regulating coal mine reclamation operations is delegated to the US Department of the Interior, Office of Surface Mining Reclamation and Enforcement (OSM). Under SMCRA, OSM developed minimum national standards creating a nationwide program for coal mine permitting applicable to both surface and underground mining operations.

As in many of the other environmental programs, each state is given the option of either developing a program that meets all of the federal standards and obtaining approval for a state "primary program" or having the federal government assume the primary responsibility.[134] Under state primacy, the state, not the federal government, takes the lead responsibility for issuing coal mining permits and enforcing coal mining regulatory requirements with federal oversight by OSM. In oversight, the role of OSM is that of an overseer, monitoring state programs to ensure continued compliance with the minimum federal standards.

On federal lands, on Indian lands, on lands within a state that chooses not to assume primacy, and where a state's primacy program has failed to meet the federal requirements and is rejected, OSM assumes directly the responsibility of issuing coal mining permits through a federal or Indian lands program.[135] Where a federal program exists, states will be preempted from issuing reclamation permits, but may still have other permitting responsibilities, such as the issuance of National Pollution Discharge Elimination System permits. In each case, before preparing an application, the operator should check with OSM and the state regulatory authority to ensure that the application will address the appropriate regulatory requirements.

The permit application under SMCRA must address the environmental impacts of the proposed coal mining operation, including detailed information on the type and method of coal mining and the engineering techniques and equipment to be used, background information and hydrologic data, a description of the geologic and physical characteristics of the coal and the mining area, a determination of the probable hydrologic consequences of the mining and reclamation, a complete operations and reclamation plan, and a demonstration that minimum performance standards for protection of the environment will be met.[136] Among the comprehensive environmental standards are requirements that all surface areas disturbed by mining, with limited exception, be returned to the approximate original contour, offsite impacts minimized, and the land be restored to as good or better condition as existed prior to mining.[137]

The reclamation plan must describe the condition of the land before mining, including its existing and potential land uses, its productivity, and its average yield or extent of cover. The plan also must specify the proposed postmining land use and describe in detail how this use will be achieved, including the mining techniques and equipment to be used.[138]

Once the SMCRA permit application has been approved, the permittee must post a bond covering the estimated cost for reclamation of the mined area.[139] Each permit is issued for a term of up to five years unless special conditions for a longer term are met. The permittee must commence mining within three years of permit issuance and may apply for renewal of the permit prior to the end of its term.[140]

7.3.4.6 Noncoal-mined Land Reclamation

There is no single nationwide program for the reclamation of noncoal mines like there is for coal mines under the federal Surface Mining Control and Reclamation Act. Rather, the reclamation standards that apply vary from state to state, and from mineral to mineral. As a general rule, most states regulate to some extent the reclamation of land affected by mining operations. These laws typically address the disposal of mine waste and the restoration of the area disturbed by mining, and typically require a state permit prior to commencement of mining.

To regulate the impacts of mining operations, most states supplement any requirements under state reclamation laws with application of the standards established under other programs such as the Clean Water Act and the Clean Air Act to ensure that the range of environmental impacts associated with mining are addressed.

7.3.4.7 Underground Injection Control Program

The underground injection control (UIC) program under the Safe Drinking Water Act[141] regulates any activity that uses a well to inject fluids into the earth.[142] Under the UIC program, any injection that will cause or allow movement of fluids into underground sources of drinking water is prohibited.[143]

The UIC program regulates underground injections by five classes of wells.[144] All owners or operators of any class of injection well must be authorized by either permit or by rule. All mine backfilling operations (active and those undergoing closure) are regulated by rule as Class V underground injection control (UIC) wells (backfilled mines are mines whose voids are filled with mine waste).[145] In situ oil shale and coal gasification retorts and wells used for solution mining and conventional mines, such as stope leaching, are also regulated as Class V wells. These activities are regulated by rule and a general performance standard that prohibits movement of the waste into underground sources of drinking water. Unless required by the state, no UIC permit is required for such Class V wells.[146] Nevertheless, injection that may cause a violation of primary drinking water regulations can result in either a closure action or a requirement that the facility obtain a UIC permit. Additional permits for underground injection may also be required at the state level.

Injection wells for in situ extraction of minerals such as copper, uranium, other metals, sulfur, and salt are regulated as Class III wells. Class III wells must be permitted, with the permit stipulating certain construction, operating, monitoring, and closure requirements to protect underground sources of drinking water.[147] Class I, II, and IV wells also require permits. Class I and IV wells are wells associated with hazardous waste management facilities. Class II wells are wells associated with oil and natural gas production.[148]

The UIC permit application must include information on the applicant, the principal products or services provided by the facility, other permits required for the facility, maps of the facility, and information on the proposed underground injection activity.[149] In addition to complying with the conditions of the permit, operators of UIC wells may also be required to comply with certain financial responsibility requirements for plugging and abandonment of the wells.[150]

7.3.4.8 Mine Safety and Health Act

The federal Mine Safety and Health Act of 1977 applies to almost all mining activities (Chapter 3.3).[151] It is not designed to be an environmental statute and does not impose broad permitting requirements on mine operators. Nevertheless, within the Mine Safety and Health Act's broad parameters are certain requirements for the mine operator to obtain premining approvals and to allow the Mine Safety and Health Administration (MSHA) to review certain mining plans prior to commencement of mining.

As a general rule, each mine operator must notify MSHA of the proposal to open a new mining operation, describe the type of mine or mines proposed, obtain an identification number for each mine, develop and implement mine safety training programs, and either incorporate mandatory safety standards into mine design or qualify for a modification of the standards. Plans for certain mine-related structures such as large impoundments and coal refuse embankments must be submitted to MSHA for approval prior to mining to demonstrate compliance with mandatory safety standards.[152]

7.3.4.9 Uranium Mill Tailings Radiation Control Act

Under the Uranium Mill Tailings Radiation Control Act of 1978 (UMTRCA),[153] the Nuclear Regulatory Commission (NRC), in conjunction with EPA,[154] has regulatory jurisdiction over the opening of any uranium processing facility.[155] The permitting requirements under UMTRCA are comprehensive and cover all aspects of the facility related to radioactive materials. Radioactivity standards and limits for disposal and cleanup of uranium mill tailings wastes are imposed as conditions of the permit.[156]

7.3.4.10 Areas Where No Mining is Allowed

A recent development in the area of environment regulation of mining is the establishment of areas that, because of their environmentally sensitive nature, are precluded from being disturbed by mining. As Congress continues to take steps to preserve natural resources, more and more land areas and in particular lands owned by the federal government may become off limits to mineral production.

Under the 1964 Wilderness Act alone, tens of millions of acres were set aside for study as areas from which any intrusion, including mining, may be permanently banned. Additionally, National Parks,[157] National Wildlife Refuges,[158] National System of Trails,[159] National Wilderness Areas,[160] and Wild and Scenic Rivers[161] for example, all can impose significant restrictions on mineral development. The US Forest Service as well is increasingly under pressure to restrict mining within the boundaries of the National Forests.[162]

Under SMCRA, certain areas are also deemed to be "Designated Unsuitable for Mining." SMCRA provides two ways an area can be declared off limits to coal mining. First, areas can be designated unsuitable for mining by Congress,[163] and second, a process is established by which the state or federal regulatory authority can designate additional areas unsuitable for mining because of the environmentally sensitive nature of the area.[164]

Areas designated by Congress as unsuitable for coal mining include lands within the boundaries of the National Parks, National Wildlife Refuges, National System of Trails, National Wilderness Areas, Wild and Scenic Rivers, National Recreation Areas, and National Forests,[165] and land that will adversely affect a publicly-owned park or a place on the National Register of Historic Places.[166] The Congressional designation also mandates that a buffer zone of area unsuitable for mining be created around publicly-owned parks (300 ft or 90 m), public road rights-of-way (100 ft or 30 m),[167] occupied dwellings (300 ft or 90 m),[168] public buildings (300 ft or 90 m),[169] and cemeteries (100 ft or 30 m).[170]

A discretionary designation as unsuitable for coal mining may be made for any area if the regulatory authority determines that the mine would be incompatible with existing state or local land-use plans or programs, or would adversely affect fragile or historic lands, renewable lands, or natural hazard lands. A designation as unsuitable for mining must be made if the regulatory authority determines that reclamation is not economically or technologically feasible. Designations under the discretionary criteria can be reversed if changing economics or technology make reclamation feasible in an area where it was not feasible before. However, first an applicant must go through a proceeding to "undesignate" the area.[171] Additionally, once a permit application has been submitted for an operation, a subsequent effort to have the area designated as unsuitable for mining will not prevent mining from continuing under the permit.

7.3.5 ENVIRONMENTAL AUDIT

7.3.5.1 Background

In conjunction with permitting, today's regulatory standards mandate some consideration of long-term environmental liabilities associated with most mining activities. Environmental audits are thus increasingly becoming a standard procedure for many mining companies.

Environmental audits have the potential to reveal sensitive environmental data, such as the presence of hazardous waste, the absence of required permits, environmental violations, environmental contamination, and the like; hence, there is occasionally reluctance on the part of mine management to complete a thorough audit. However, most of the information that could lead to environmental enforcement actions could be discovered by others under appropriate conditions, and many types of environmental problems could lead to mine closure if not properly addressed. Thus, as a rule, it is ultimately preferable for a company to take the appropriate steps to identify potential environmental issues and take corrective action prior to any regulatory enforcement action or offsite environmental damage.

Environmental laws are also increasingly acknowledging efforts to identify environmental issues as part of the normal due diligence review for a mining facility. For example, revisions to the Comprehensive Environmental Response, Compensation, and Liability Act (CERCLA) officially recognize preacquisition property audits as a potential defense to Superfund liability.[172] Also, amendments to the Clean Water Act allow a variance in applicable effluent limitations for coal mines where there is a premining bad-water discharge.[173]

The following broad objectives can be stated for an environmental audit:

1. Evaluate the compliance status of all relevant facilities.
2. Ensure the existence of necessary permits, registrations, approvals, or certifications.
3. Identify current and pending or threatened regulatory, administrative, or judicial environmental actions and claims.
4. Identify postmining environmental liabilities.
5. Review the status of waste disposition, emissions, and discharges, past, present, and future.
6. Evaluate the personal exposure of employees, invitees, and the public.
7. Evaluate the potential for catastrophic incidents or releases.
8. Identify potential exposure of management and affiliated companies to liability.

7.3.5.2 Environmental Audit Check List

Based upon these broad objectives, a site "Environmental Audit Check List" can be prepared touching upon the major items that should be evaluated at a particular facility. The following outline provides an abbreviated review of major items that could appropriately be covered by an environmental audit for a mining facility. The outline is geared specifically to mining situations and the goal of providing a management framework within which a more detailed evaluation can be conducted of specific potential problem areas.

Permits:

1. Provide a list of all permits existing, applied for, and in preparation for the mine. Indicate the current status of each permit.
2. If the mine has NPDES permits, air permits, waste permits, water withdrawal permits, or other permits, include a listing of each applicable permit.
3. Are there any additional permits required for the facility but not already obtained?
4. Were any citizen protests or comments filed on any of the permits? If so, describe the issues raised.
5. What is the status of the bond on each permit? Indicate the amount of bonding liability under each permit.

Surface and Mineral Rights Information:

1. Is the surface property owned or controlled by the mining company?
2. Does the mine have a specific written agreement with each surface owner pertaining to and providing applicable rights to the surface?
3. If the mine is an underground mine, do the surface rights specifically include language that says the company has the right to subside?
4. Has the mine every received any complaints from a surface owner or any of the surrounding landowners? Indicate the nature of any complaint received.
5. With respect to mineral rights, has there ever been a claim made suggesting that the mining company does not have complete and full mineral rights?
6. Have any other entities been granted access rights to the property that may conflict with the rights to extract the mineral (e.g., oil wells, pipelines, utility easements, etc.)?

Ownership and Violation Information:

1. Is the mine permit held by any company not owned and controlled 100% by the operator?
2. Is the mine permit being held by an independent contractor?
3. Is the mine operation being conducted by any company not owned and controlled 100% by the permittee?
4. Is the mine being operated by an independent contractor? Give brief summary of applicable relation.
5. If the mine is permitted by or being operated by an independent contractor, has a review been completed of the contractor's violation history with regard to any other mines potentially affiliated with the contractor?
6. If the mine is permitted by or being operated by an entity only partially owned, has a review been completed of the contractor's violation history with regard to any other mines potentially affiliated with the other parties?
7. How is the relationship between mine personnel and the mine inspectors (excellent, good, poor)? If poor, please describe the problems encountered.
8. Have there been any citizen complaints filed on the mine? If so, please describe the complaints filed.

Information on Archaeological Sites, Cemeteries, etc.:

1. Within the permit area and adjacent areas, including the area over existing, past, and proposed underground workings, does the mine contain any of the following features?
   a. Federal lands? Indicate the type of federal land involved.
   b. Any archaeological or historical site that has been previously identified? Indicate the type of site involved.
   c. Public roads where there is not a specific written approval for mining through or under the public road by the agency that has jurisdiction over the road?
   d. Occupied dwellings where there is not a specific written waiver from the owner of the occupied dwelling?
   e. Any public buildings, schools, churches, community, or institutional buildings?
   f. Any public park?
   g. Any public or private cemeteries?

Hydrology:

1. Has the mine obtained all necessary water discharge permits?
2. Is the mineral being mined associated with potential pollutants, such as high sulfur-bearing strata?
3. Has the mine ever experienced any difficulties with poor-quality mine drainage? Briefly describe the situation encountered.
4. Does any of the mine water need to be treated to meet effluent limitations or other applicable standards?
5. Has any poor-quality mine drainage been observed on any part of the permit area, in any of the surrounding areas, whether associated with the mine or not?
6. Have any of the surrounding land owners or occupants ever complained about loss of water or the mine contaminating their water supplies?
7. Is the water used by surrounding landowners of poor quality? For example, is the water discolored, does it have a bad odor or bad taste, or are there other problems with water quality in the area?
8. Has the mine ever had any complaints about the water level in streams increasing or decreasing, or the fish population in streams increasing or decreasing?

9. Are there any in-stream treatment facilities or in-stream sediment ponds associated with the mine?

10. Are there any wetland areas, wet areas with cattails or similar aquatic vegetation that could be considered wetlands, within the vicinity of the mine?

Air Pollution:

1. Has the mine obtained all the necessary quality permits?

2. Have there been complaints about dust or air quality discharges from the mine?

3. Are mine haul trucks covered with tarps to minimize dust and to prevent material from falling off?

4. Is there a program for controlling dust and air pollution from the mine?

Explosives:

1. Has the mine had any complaints from land owners pertaining to blasting or the use of explosives?

2. Is the mine access limited due to topographic conditions or is it possible for local people to obtain unauthorized access to the mine site?

3. If it is possible for unauthorized persons to obtain access to the site, what measures are currently taken to control this access?

4. Are any personnel other than trained mine personnel allowed within the mine pit or the mine property in an area that may ever be subject to blasting?

Waste Disposal:

1. Does the operation include a facility for the disposal of mine waste?

2. Have there ever been any discolored or bad-quality water seeps from the waste pile?

3. Does the groundwater indicate any levels of pollution from the waste pile?

4. What is the worst-case chemical analysis for the waste material being disposed of at the mine?

5. Have there ever been any signs of slumping or instability of the waste pile?

6. Does the waste pile have any public roads, homes, or occupied dwellings, etc., downstream that could be affected by failure?

7. Is the waste pile routinely inspected and certified by a professional engineer to verify that it is being built in accordance with the approved plan?

8. Are there any abandoned mine waste piles within the vicinity of the mine, including the area overlying underground workings?

9. Do any of the abandoned mine waste piles show any signs of instability or poor mine drainage?

10. Does the disposal of any domestic or other nonmine waste occur within the property boundaries, including the disposal of garbage by mine or nonmine personnel?

11. Has a survey been done to determine whether there is any nonmine waste on the property? Are there any abandoned nonmine waste piles within the mine property?

12. What does the mine do with trash from the mine?

Impoundments:

1. Are there any large impoundments within the vicinity of the mine or included in the mine permit?

2. Are there any public roads, facilities, dwellings, or other such facilities downstream of an impoundment that could be affected by the impoundment's failure? If so, briefly describe.

3. Are all impoundments routinely inspected and certified by a professional engineer to verify that they are being built in accordance with the approved plans?

4. Have any of the impoundments ever experienced overtopping?

Backfilling and Slides:

1. Has the mine ever experienced a slide or instability of backfilled material or of outslope material?

2. Does the mine have an excess spoil fill? Does the mine have a durable rock fill? Were any of the fills constructed in a location where there was a stream or running water prior to mining?

3. If the mine is a surface mine, what is the maximum size pit and the size of the total pit currently open for the mine? How many cubic yards (cubic meters) of material would be required if the mine was closed today and the pit had to be backfilled?

4. What is the maximum amount of highwall currently open on the operation, including total length and height? How many cubic yards (cubic meters) of material would be required if the mine was closed today and the highwall had to be completely eliminated?

Subsidence:

1. Does the mine include longwall or pillar removal mining?

2. Has the mine ever identified any substance subsidence?

3. Have there ever been any complaints by surface owners about surface subsidence?

4. Does the mine have any significant problems with roof or floor control?

Mine Fires:

Have there ever been any mine fires, including outcrop fires, identified on the mine property?

Underground Injections:

1. Does the mine have any injection wells or pump any fluids or other material of any type underground?

2. If the mine is an underground mine, has an evaluation ever been completed to determine what will occur after the mine is closed and the mine workings become flooded? If yes, briefly describe the resulting conclusion.

Previously Mined Areas:

1. Does the area affected by the mine include a previously mined area?

2. Are there any of the following problems on the previously mined area?

a. Bad water drainage?

b. Unstable spoil?

c. Spoil on the outslope?

d. Refuse material on the outslope?

e. Unreclaimed areas where revegetation will be difficult?

f. Unreclaimed highwall?

Miscellaneous:

1. How close is the nearest house?

2. Have there been complaints about the use of roads in the area?

3. Has the mine ever received a closure or cessation order?

4. Are there cleaning or other chemical solvents located at the mine? Have all necessary forms been filed and provided in regard to "right to know" requirements for chemical exposure?

5. Have all transformers or other electrical equipment been checked for PCB contamination?

6. Is oil or fuel stored on the property? What type of facilities are provided for oil or fuel storage? Are there any underground storage tanks? Have all such tanks been tested?

7. Are there any oil or gas wells, or similar facilities on the property that could conflict with mining?

REFERENCES

1. 16 U.S.C. §§ 1531 *et seq.* (1988).
2. 16 U.S.C. §§ 668 *et seq.* (1988).
3. 16 U.S.C. §§ 470 *et seq.* (1988).
4. 33 U.S.C. §§ 1251 *et seq.* (1988).
5. See, e.g., "Applied Research on the Benefits and Costs of Public

Regulation of the Copper Industry," National Science Foundation, National Academy of Sciences, 1981. (This study found that the additional burden on the copper industry of being required to recover 90% of the sulfur from sulfur smelters would shift 70% of the remaining mine reserves to economically unrecoverable status).

6. U.S. Constitution, Art. VI, Cl. 2. (This section is commonly known as the "Supremacy Clause.")
7. 33 U.S.C. §§ 1251 *et seq.* (1988)
8. 30 U.S.C. §§ 1201 *et seq.* (1988).
9. See, e.g., *Commonwealth v. Perkins,* 21 A.2d 45, aff'd 314 U.S. 586, 62 S.Ct. 484, 86 L.Ed. 473 (1941).
10. See generally, *Keystone Bituminous Coal Association v. DeBenedictis,* 480 U.S. 470, 107 S.Ct. 1232, 94 L.Ed.2d 472 (1987).
11. 33 U.S.C. § 1342 (1988).
12. *Id.*
13. 30 U.S.C. § 1254(9) (1988).
14. 33 U.S.C. § 1342 (1988).
15. 42 U.S.C. § 300(f) *et seq.* (1988).
16. 16 U.S.C. §§ 471 *et seq.* (1988).
17. 42 U.S.C. §§ 4321 *et seq.* (1988).
18. 16 U.S.C. Section 1531 *et seq.* (1988).
19. 16 U.S.C. Section 668 *et seq.* (1988).
20. 16 U.S.C. Section 661 *et seq.* (1988).
21. 16 U.S.C. Section 470 *et seq.* (1988).
22. 16 U.S.C. Section 469 *et seq.* (1988).
23. Whenever federal lands are involved, a variety of special permits or approvals may be required from the land management agency. For example, access across federal lands may require approval of rights-of-way from the land management agency, special use permits may be required for incidental surface activities, permits for cutting of timber, and permits for disturbance of any historical or archaeological resource may be required, among others.
24. 30 U.S.C. § 187 (1988).
25. 43 U.S.C. §§ 1701 *et seq.* (1988).
26. 43 U.S.C. § 1732(b) (1988).
27. *North Slope Borough v. Andrus,* 13 BNA Env. Rep. Cas. 2169, 2178 (D.C.Cir. 1980).
28. Local zoning authority was recognized by the US Supreme Court in the landmark decision, *Euclid v. Ambler Realty Co.,* 272 U.S. 365 (1926). The *Euclid* case involved a local zoning ordinance that placed height, area, and use restrictions on land use within an Ohio municipality. See generally, MacKenzie, *The Constitutionality of Municipal Ordinances and Administrative Regulations,* 13 Urban Lawyer 774 (Fall 1981).
29. A number of cases have recognized the validity of local and state regulation, even where federal lands were involved. See, e.g., *Wallis v. Pan American Petroleum Corp.,* 384 U.S. 63, 67-72 (1966); *Texas Oil and Gas Corp. v. Phillips Petroleum Company,* 406 F.2d 1303 (10th Cir. 1969) *cert denied* 396 U.S. 829 (1969).
30. Pennsylvania Stat. Ann. Ti. 52, § 1396.17(a) (Purdon, 1982 Supp).
31. In the case of *Ventura County v. Gulf Oil Corporation,* 601 F.2d 1080 (9th Cir. 1979), *cert. denied* 48 U.S.L.W. 3622 (1980), Ventura County had designated the federal lands as "open space" under a zoning ordinance, a category that prohibited oil exploration and extraction unless a permit was granted. Gulf Oil Corp., which had a federal oil lease, refused to apply for the zoning permit. The US Court of Appeals ruled that the zoning ordinance conflicted with a major Congressionally sanctioned program and ruled that Gulf could not be required to obtain the local permit.
32. A total ban on mining within the confines of a municipality would likely be ruled improper, unless it can be shown that there are specific harms that may result from mining and that there are no alternative means that can minimize or prevent that harm. See, *e.g, Spillerdown v. Prewilt,* 21 Ill.2d 288, 171 N.E.2d 582 (1961) (ordinance prohibiting strip mining within village limits).
33. For example, under the federal Surface Mining Control and Reclamation Act (SMCRA), no coal mining permit may be issued to an applicant where information available to the regulatory agency indicates that any coal mining operation owned or controlled by the applicant is in violation of SMCRA or other law pertaining to air or water environmental protection, unless the violation is in the process of being corrected or is being appealed by the operator. 30 U.S.C. Section 1260(c)(1988).
34. EPA's procedures for issuing permits allow for the consolidation of the review of NPDES, RCRA, UIC, and PSD permits (except RCRA and UIC emergency permits). The procedures for processing each of these categories of permits are substantially identical and are contained in 40 C.F.R. Part 124 of the federal regulations.
35. The procedures for an NPDES permit are set forth in 40 C.F.R. Section 122.21, for a UIC permit in 40 C.F.R. Section 144.31, and for a RCRA permit in 40 C.F.R. Section 270.10 and 124.3.
36. 40 C.F.R. Section 124.7 (1989).
37. 40 C.F.R. Section 124.8 (1989).
38. 40 C.F.R. Section 124.9 (1989).
39. 40 C.F.R. 124.10 and 124.11 (1989).
40. 40 C.F.R. Section 124.10(b) (1989). EPA regulations require that all arguments and factual materials that a person wishes to be considered by the EPA in connection with a particular permit must be placed in the record by the close of the public comment period. 40 C.F.R. Section 124.13 (1989).
41. 40 C.F.R. 124.12 (1989).
42. *Id.* Where a public hearing is held, the comment period will be extended through the close of the hearing.
43. Although the EPA regulations do not specifically address it, it has been the custom of EPA to allow applicants time after the close of the public comment period to file responses to the comments submitted.
44. 40 C.F.R. Section 124.15 (1989).
45. 40 C.F.R. Section 124.17 (1989).
46. 40 C.F.R. Section 124.18 (1989).
47. *Id.*
48. 40 C.F.R. Sections 124.19 and 124.74 (1989).
49. 40 C.F.R. Section 124.18 (1989).
50. See 40 C.F.R. Section 124.19 and Section 124 Subpart E (1989).
51. See Appendix A to 40 C.F.R. Part 124. NPDES permits that involve "initial licensing" may be appealed in a special nonadversary panel hearing. The EPA regulations also provide that if such a hearing is granted for an NPDES permit, consolidated RCRA, UIC, or PSD permits may also be reexamined in the same proceeding.
52. 40 C.F.R. Section 124.19 (1989).
53. 40 C.F.R. Section 124.71–124.91 (1989).
54. 40 C.F.R. Section 124.91 (1989).
55. The procedures for permit review may vary among states based upon the specific organization and provisions of the state surface coal mining program. However, each state must meet the minimum federal standards in order to have an approved state program. 30 U.S.C. Section 1253.
56. 30 C.F.R. Section 774.11 (1988).
57. 30 C.F.R. Section 701.5 (1988).
58. 30 C.F.R. Section 773.13(a) (1988).
59. 30 C.F.R. Section 773.13(b) (1988).
60. 30 C.F.R. 773.13(c) (1988).
61. The permitting agency will typically notify the applicant of any deficiencies identified in the permit application and provide the applicant with an opportunity to respond to the deficiencies and any public comment.
62. 30 C.F.R. Section 773.15 (1988).
63. 30 C.F.R. Section 773.19 (1988).
64. 30 C.F.R. Section 775.11 (1988).
65. 30 C.F.R. Section 775.11(b) (1988).
66. 30 C.F.R. Section 775.11 (1988).
67. 30 C.F.R. Section 775.13 (1988).
68. 33 U.S.C. Section 1251 *et seq* (1988).
69. 33 U.S.C. Section 1342 (1988). See also, 40 C.F.R. Part 122 (1989).
70. A "point source" is defined as follows:

    Point source means any discernable, confined and discrete conveyance, including, but not limited to, any pipe, ditch, channel, tunnel, conduit, well, discrete fissure, container, rolling stock, concentrated animal feeding operation, vessel or other floating craft from which pollutants are or may be discharged.

    33 U.S.C. Section 1362(14) (1988). Return flows from irrigated agriculture are not included in this definition. *Id.* See also, 40 C.F.R. Section 122.2 (1989).
71. *Pollutant* means dredged spoil, solid waste, incinerator residue,

filter backwash, sewage, garbage, sewage sludge, munitions, chemical wastes, biological materials, radioactive materials (except those regulated under the Atomic Energy Act of 1954, as amended [42 U.S.C. 2011 *et seq.* (1988)], heat wrecked or discarded equipment, rock, sand, cellar dirt and industrial, municipal, and agricultural waste discharged into water. It does not mean (a) sewage from vessels; or (b) water, gas, or other material [associated with oil and gas production.] 40 C.F.R. Section 122.2 (1989).

72. *Waters of the United States* or "waters of the U.S." means:
   (a) All waters which are currently used, were used in the past, or may be susceptible to use in interstate or foreign commerce, including all waters which are subject to the ebb and flow of the tide;
   (b) All interstate waters, including interstate "wetlands;"
   (c) All other waters such as interstate lakes, rivers, streams (including intermittent streams), mudflats, sandflats, "wetlands," sloughs, prairie potholes, wet meadows, playa lakes, or natural ponds the use, degradation, or destruction of which would affect or could affect interstate or foreign commerce including any such waters;
      (1) Which are or could be used by interstate or foreign travelers for recreational or other purpose
      (2) From which fish or shellfish are or could be taken and sold in interstate or foreign commerce; or
      (3) Which are used or could be used for industrial purposes by industries in interstate commerce;
   (d) All impoundments of water otherwise defined as waters of the United States under this definition;
   (e) Tributaries of waters identified in paragraphs (a) through (d) of this definition;
   (f) The territorial sea; and
   (g) "Wetlands" adjacent to waters (other than waters that are themselves wetlands) identified in paragraphs (a) through (f) of this definition. 40 C.F.R. Section 122.2 (1989).
73. See 40 C.F.R. Section 122.2 (1989).
74. A processing plant that is a "closed system" is not necessarily exempt from NPDES requirements unless the system can be guaranteed *never* to discharge water. *Sierra Club v. Abston Const. Co.,* 620 F.2d 41, 45 (5th Cir. 1980).
75. The case law has supported findings that mining can constitute a point source. *United States v. Earth Sciences, Inc.,* 599 F.2d 368 (10th Cir. 1979) (gold mining); and see also, *Sierra Club v. Abston Const. Co.,* 620 F.2d 41 (5th Cir. 1980).
76. Effluent limitations consist of numerical limits on the amount of pollutants that may be discharged by a facility, regardless of the location or condition of the receiving stream. 33 U.S.C. 1311, 1314, 1316, and 1317 (1988). See EPA Effluent Guidelines and Standards for Mineral Mining and Processing, 40 C.F.R. Part 436 (1989).; EPA Effluent Guidelines and Standards for Ore Mining and Dressing, 40 C.F.R. Part 438 (1989).; EPA Effluent Guidelines and Standards for Ore Mining Point Sources, 40 C.F.R. Part 434 (1989).; EPA Effluent Guidelines for Non-ferrous Metals, 40 C.F.R. Part 421 (1988).; and EPA Effluent Guidelines and Standards for Phosphate Manufacturing, 40 C.F.R. Part 422 (1988).
77. 33 U.S.C. Sections 1311, 1312, and 1342 (1988).
78. 40 C.F.R. Section 122.21(h) and (k) (1989).
79. 40 C.F.R. Section 122.41 (1989).
80. 33 U.S.C. Section 1342(k) (1988).
81. 33 U.S.C. Section 1288 (1988).
82. 33 U.S.C. Section 1344 (1988).
83. The EPA and the Corps define dredged material as "material that is excavated or dredged from waters of the United States." 33 C.F.R. Section 323.2(j) (1989).; 40 C.F.R. Section 232.2 (1989).
84. The dividing line between the Section 404 Corps' permit and the NPDES permit program is based on the type of material to be placed or discharged into the water. If the material is dredged or fill material, Section 404 applies. If the discharge consists of any other pollutant, the NPDES program applies.
85. The Corps regulations for issuing dredged and fill permits are contained in 33 C.F.R. Parts 320–323 (1989). States may be granted authority to issue Section 404 permits by the EPA for some waters of the United States. The Corps always retains jurisdiction over waters presently used, or susceptible to use, as a means to transport interstate commerce, including tidal waters and adjacent wetlands.
86. 33 C.F.R. Section 323.2 (1989).
87. 33 C.F.R. Section 320.4(a) (1989).
88. 42 U.S.C. Section 1344(b) and (c) (1988).
89. 40 C.F.R. Section 230.10 (1989).
90. 33 U.S.C. Sections 401–408 (1988).
91. See 33 C.F.R. Part 321 (1989).
92. See 33 C.F.R. Part 322 (1989).
93. Instead of a permit, the Corps will issue a real estate instrument for such activities.
94. The term "work" includes dredging or disposal of dredged material, excavation, filling, or other modification of a navigable water. 33 C.F.R. Section 322.2(c) (1989).
95. 33 C.F.R. Part 322 (1989).
96. 33 C.F.R. Section 325.1 (1989).
97. 33 C.F.R. Section 325.3 (1989).
98. 33 C.F.R. Section 325.6 (1989).
99. 42 U.S.C. Sections 7401 *et seq* (1988).
100. Areas that have air quality better than the national ambient air quality standards are known as "attainment areas." 42 U.S.C. Section 7407 (1988).
101. 42 U.S.C. Sections 7470–7491 (1988).
102. Areas which do not meet the ambient air quality standards are defined as "non-attainment areas." 42 U.S.C. Section 7407 (1988).
103. 42 U.S.C. Sections 7501–7508 (1988); and see 40 C.F.R. Part 50 (1989).
104. In making a decision as to whether a facility is a "major" source or not, EPA has determined to exclude the consideration of fugitive dust emissions from surface coal mines. This exclusion is significant because it means that few coal mines would be classified as major sources and thus subject to the CAA major source permitting requirements. See 54 Federal Register 48870 (November 28, 1989).
105. See 40 C.F.R. Part 52 (1988).
106. 42 U.S.C. Section 7410 (1988); 40 C.F.R. Part 51 (1989).
107. A "major stationary source" means:
   (1) Any stationary source of air pollutants which emits, or has the potential to emit, 100 tons/year or more of any pollutant subject to regulation under the Clean Air Act; or
   (2) Any physical change that would occur at a stationary source not qualifying under paragraph (1) above as a major stationary source, if the change would constitute a major stationary source by itself. 40 C.F.R. Part 51, Appendix S (1989).
108. 40 C.F.R. Section 51.18 (1989).
109. 42 U.S.C. Section 7470–7491 (1988). A "major emitting facility" and "major modification" are defined in 42 U.S.C. Section 7479 (1988).; and in EPA's regulations at 40 C.F.R.Section 51.24(b) (1989) and 52.2 l(b) (1988). As discussed earlier, EPA has excluded fugitive dust emissions from surface coal mines from the determination of whether a facility is a major facility for purposes of PSD permitting. See 54 Federal Register 48870 (November 28, 1989).
110. 42 U.S.C. Section 7475 (1988).
111. 40 C.F.R. Section 51.24, (1989); 40 C.F.R. Section 52.21 (1988).
112. 40 C.F.R. Section 51.24 (1989); 40 C.F.R. Section 52.21 (1988).
113. 42 U.S.C. Section 7411 (1988).
114. See 40 C.F.R. Part 60 (1989).
115. 42 U.S.C. Section 7412 (1988).
116. 40 C.F.R. Sections 61.01–61.71 (1989).
117. 42 U.S.C. Section 6901 *et seq* (1988). A companion law to RCRA is the Comprehensive Environmental Response, Compensation, and Liability Act (CERCLA), also referred to as the "Superfund Act," which establishes a program to deal with releases and potential releases of hazardous substances, including hazardous wastes. CERCLA sets up a program for remedial measures and enforcement at existing sites, but is not a regulatory program and does not contain permitting requirements.
118. 42 U.S.C. Section 6902 (1988).
119. 42 U.S.C. Section 6903(27) (1988); 40 C.F.R. Section 2561.2 (1988). The definition of solid waste specifically refers to waste from "mining operations." *Id.*
120. 42 U.S.C. Section 6982(f) (1988). This exemption is commonly referred to as the Bevill amendment after Rep. Tom Bevill (Ala).

EPA initially interpreted the Bevill Amendment to apply to all solid waste from the exploration, mining, milling, smelting, and refining of ores and minerals. 45 Federal Register 76118 (Nov. 19, 1980). The list of waste excluded has since been reduced to 15 categories. The mining waste study was to include, among other things, the sources and volume of discarded material per year, present disposal practices, potential dangers from surface runoff or leachate, and estimates of air pollution by dust. *Id.*

121. Currently, five types of mining waste resulting from the extraction, beneficiation, and processing of ores and minerals are excluded from the hazardous waste permitting and regulatory requirements. These are:
(1) Slag from primary copper processing.
(2) Slag from primary lead processing.
(3) Red and brown muds from bauxite refining.
(4) Phosphogypsum from phosphoric acid production.
(5) Slag from elemental phosphorous production.
See 54 F.R. 36592 (Sept. 1, 1989). Also, 15 types of mining waste are being studied by EPA and are conditionally excluded from hazardous waste permitting and regulatory requirements. These include:
(1) Treated residue from roasting/leaching of chrome ore.
(2) Gasifier ash from coal gasification.
(3) Process wastewater from coal gasification.
(4) Calcium sulfate wastewater treatment plant sludge from primary copper processing.
(5) Slag tailings from primary copper processing.
(6) Fluorogypsum from hydrofluoric acid production.
(7) Process wastewater from hydrofluoric acid production.
(8) Air pollution control dust/sludge from iron blast furnaces.
(9) Iron blast furnace slag.
(10) Process wastewater from primary magnesium production by the anhydrous process.
(11) Process wastewater from phosphoric acid production.
(12) Basic oxygen furnace and open hearth furnace air pollution control dust/sludge from carbon steel production.
(13) Basic oxygen furnace and open hearth furnace slag from carbon steel production.
(14) Chloride process waste solids from titanium tetrachloride production.
(15) Slag from primary zinc processing.
55 F.R. 2322 (Jan. 23, 1990).

122. The five categories of mining wastes that have been ruled to be subject to the hazardous waste program requirements include:
(1) Furnace off-gas solids from elemental phosphorus production.
(2) Process wastewater from primary lead processing.
(3) Air pollution control dust/sludge from lightweight aggregate production.
(4) Sulfate process waste acids from titanium dioxide production.
(5) Sulfate process waste solids from titanium dioxide production.
55 Federal Register 2322 (Jan. 23, 1990.)

123. 40 C.F.R. 261.4(b) (3) (1989).
124. 40 C.F.R. Section 260.10 (1989).
125. 42 U.S.C. Section 6905(c) (2) (1988). The exemption for SMCRA permits was added to RCRA in May 1980. Solid Waste Disposal Act Amendments of 1980, P.L. 96-482 (1980).
126. 40 C.F.R. Part 261 (1989). Even if a waste is classified as a hazardous waste under RCRA, it may still be exempt from the hazardous waste program requirements if it falls within the small quantity exemption (40 C.F.R. Section 261.5), if the waste will be stored for a short period of time (40 C.F.R. Section 262.34), or if the material will be beneficially used, reused, recycled, or reclaimed.
127. The terms "treatment," "storage," "disposal," and "hazardous waste" are defined in 40 C.F.R. Section 270.2 (1989).
128. 40 C.F.R. Part 270 (1989).
129. 40 C.F.R. Section 122 *et seq* (1989).
130. 40 C.F.R. Section 270.4 (1989).
131. Subtitle D of RCRA provides guidance and federal financial assistance for the development of state and regional solid waste management plans and implementation of state solid waste programs. Although RCRA does not give EPA authority to enforce a permitting program for specific nonhazardous wastes, EPA still has authority to develop guidelines for implementation by the states which provide guidance on the development of such programs. 42 U.S.C. Section 6941 (1988).
132. 42 U.S.C. Section 6944(a) (1988).
133. 30 U.S.C. Section 1201 *et seq.*, P.L. 95-87 (1988).
134. Although SMCRA was passed by Congress in 1977, the permitting requirements of the federal law became applicable in a state when that state's permanent regulatory program was adopted and approved by the OSM. In states that chose not to develop a state permanent regulatory program, the permitting requirements of SMCRA became applicable upon the promulgation of a federal program by OSM. All states with potential for coal mining currently have either an approved state program or a federal program under SMCRA.
135. Indian programs are operated in conjunction with the individual Indian tribe involved.
136. 30 U.S.C. Sections 1257, 1258, 1260 (1988); See also 30 C.F.R. Parts 773–785 (1988).
137. 30 U.S.C. Sections 1265, 1266 (1988); See also 30 C.F.R. Parts 816, 817 (1988).
138. 30 U.S.C. Sections 1257, 1258, 1260 (1988); See also 30 C.F.R. Parts 773–785 (1988).
139. 30 C.F.R. Part 800 (1988).
140. 30 C.F.R. Section 773.19 (1988).
141. 42 U.S.C. 300(f) *et seq* (1988).
142. See 40 C.F.R. Section 122.32 and 40 C.F.R. Part 144 (1989).
143. 40 C.F.R. Section 122.34 (1989). The definition of "underground source of drinking water" includes "an aquifer or its portion:
(i) which supplies any public water system; or
(ii) which contains a sufficient quantity of groundwater to supply a public water system; and
(A) currently supplies drinking water for human consumption. . . ." 40 C.F.R. Section 146.3.
144. 40 C.F.R. Sections 144.3 and 144.6 (1989).
145. Coal processing wastewater injection wells have generally been interpreted to be Class V wells and allowed by rule unless there would be a potential for contamination of an underground source of drinking water (USDW). See *Re: Robin Reed; John and Helen Newsome v. US EPA et al.*, Civ. No. 84-3511 (6th Cir., 1984). In that case, EPA agreed to review all existing coal processing wastewater injection wells in Kentucky and take enforcement action against any wells causing a violation of primary drinking water standards for a USDW.
146. 40 C.F.R. Section 144.24 (1989).
147. 40 CFR Section 144.31 (1989).
148. See 40 C.F.R. Section 144.6 (1989).
149. 40 C.F.R. Section 144.31 (1989).
150. 40 C.F.R. Section 144.51 (1989).
151. 30 U.S.C. 802 *et seq* (1988).
152. 30 C.F.R. Parts 55–57, 70, 71, 74, 75, 77, 90 (1989).
153. P.L. 95-604, as amended (1978). Incorporated in the Atomic Energy Act of 1954, 42 U.S.C. 2022 (1988).
154. EPA's regulations containing health and environmental protection standards for uranium mill tailings are provided at 40 C.F.R. Part 192 (1989).
155. Wastes from the processing of uranium ores are regulated by the NRC under the Atomic Energy Act of 1954. 42 U.S.C. Section 2011 *et seq* (1988).
156. 40 C.F.R. Section 192.12 (1989).
157. The National Parks and Recreation Act of 1978, 16 U.S.C. §§ 1 *et seq.* (1982), establishes the National Park Service (NPS) within the US Department of the Interior and gives the NPS authority over National Park Lands.
158. The National Wildlife Refuge System Administration Act of 1966 allows the US Fish and Wildlife Service to establish and administer National Wildlife Refuges. 15 U.S.C. §§ 330 *et seq* (1982).
159. The National Trails System Act provides for the establishment and administration of national trails. 16 U.S.C. §§ 1241 *et seq* (1982).
160. The Wilderness Act establishes wilderness areas that are required to be preserved in their natural state. 16 U.S.C. §§ 1131 *et seq* (1982).
161. The Wild and Scenic Rivers Act provides that construction that would have a direct, adverse effect on the values for which a

national wild and scenic river was established can be prohibited. 16 U.S.C. §§ 1271 *et seq* (1982).

162. See 16 U.S.C. §§ 472 *et seq* (1982).
163. The Congressional designation does not apply to mines that were existing on Aug. 3, 1977 or which had valid existing rights to conduct the mining as of Aug. 3, 1977. 30 U.S.C. § 1272 (1988); see also 30 C.F.R. Part 761 (1988).
164. *Id*.
165. Mining may nevertheless be permitted within the National Forest if the mine is determined to be "compatible" with certain aspects of the National Forest and either: (1) the mine is an underground mine, or (2) the mining is west of the 100th meridian and is approved as meeting certain criteria by the Secretary of the Agriculture. However, no mining is allowed within the boundaries of the Custer National Forest. 30 U.S.C. § 1272(e) (1988).
166. Exceptions are allowed to the exception for parks and Historic Places if the mining is approved jointly by the regulatory authority and the federal, state, or local agency with jurisdiction over the park or place. 30 U.S.C.; § 1272(e) (1988).
167. Exceptions are allowed (1) where the mine access or haulage roads join the right-of-way line, and (2) after public notice, if the agency with jurisdiction over the road makes a written finding that the interests of the affected public landowners will be protected. 30 U.S.C. § 1272(e) (1988).
168. Exception is allowed if the owner of the occupied dwelling provides a signed written waiver consenting to mining closer than 300 ft (90 m). 30 U.S.C. § 1272(e) (1988).
169. Public buildings include schools, churches, community, and institutional buildings. 30 U.S.C. § 1272(e) (1988).
170. *Id.*
171. 30 U.S.C. § 1272(c) and (d) (1988); see also 30 C.F.R. Parts 762, 764, 769 (1988).
172. 10 U.S.C. § 2701 *et seq* (1988).
173. 33 U.S.C. § 1342 (1988).

Chapter 7.4
IMPOUNDMENTS AND DAMS

PAUL D. NESBITT

The planning and design of impoundments in conjunction with mining projects takes into account many engineering and environmental considerations. Under the environmental scrutiny of today, any new major facility will come under public review. Whether the public itself is involved in an item-by-item review, the regulatory agencies will ensure that a careful review will take place.

Initial efforts need to focus on determining the exact use and purpose of the impoundment. Commonly, impoundments in conjunction with mining projects are used for sediment control, fine slurry settling ponds, and freshwater sources. There are other purposes, but those named are the most common. Based upon the proposed use of the facility, the design standards employed may vary. As with any design, it is not possible to design a structure without knowing the end use. A clear definitive statement of the end use coupled with referral to such goals as the design/permitting/construction process proceeds will ensure a usable and functioning end product.

For fundamental coverage of soil mechanics, see Chapter 10.1; and for related topics, see various chapters of Sections 10 and 12 (e.g., Chapter 12.1, Water and Sediment Control Systems).

7.4.1 PRELIMINARY STUDIES

In the preliminary study phase, the site must be examined from all aspects, using as much existing information as possible. It is important for the designer of an impoundment to study *all* available information.

Using the stated end use of the project, certain design considerations may appear. Obviously, if the purpose of the impoundment is to settle out fines, either slurry or sediment, then the volume and consistency of such fines must be examined. Obviously, the finer the slurry/sediment is, the longer the detention time must be in order to achieve the desired settling rate and subsequent water quality. If the purpose of the impoundment is for water supply, then the quantities of the water needed must be determined.

US Geological Survey (USGS) topographic maps then can be utilized to determine if the volumes that were estimated to be necessary in the impoundment can be achieved on the potential sites. It should be emphasized that these maps are used for planning purposes only. Coupled with these, many areas of the country also have geologic quadrangles that map the type and extent of the geology of a region. Also published in many regions are soil surveys. All these items need to be analyzed to see if there are any environmental or physical factors present that would impact on the functioning impoundment.

Today more than ever, the designer is faced with a massive amount of regulations and regulatory agencies to deal with in any project. A survey of the agencies that might govern minimum design or performance standards for such a facility needs to be made immediately. All the regulations that apply should be indexed. Obviously, the most stringent regulations for any aspect of a project need to be followed. The agencies could include, but are not limited to, the Office of Surface Mining (OSM), the Mine Safety and Health Administration (MSHA), the US Fish and Wildlife Service, and the Environmental Protection Agency (EPA) (all federal) as well as the state surface mining agencies, the state water agency, and any other agency that might impact this project. It is always helpful to meet with the agencies, clearly state intentions of the project, and ask for their input at the beginning of the process rather than after the design is complete. Any stumbling blocks that they might reveal can be addressed prior to work proceeding past the point where such items need to be utilized in the design/construct process.

Several recent additions to the regulatory list should be pointed out. It is normal now to require such items as fish and wildlife surveys, vegetative surveys, archeological surveys, and an inventory of wetlands. If any negative findings of these agencies' requirements are uncovered, it is easier to deal with in the beginning of the project rather than after an exhaustive amount of work is completed.

Using USGS topographic maps for crude volume calculations, preliminary hydrologic studies can proceed. The design storms that are proposed from the agencies, or more stringent design storms if desired by the designer and/or client, need to be identified. It should be noted that the design storm is specified for facilities given a possible chance of failure. If the client wishes, the designer can demonstrate by statistics what the probabilities are that the capacity of a particular impoundment hydrologic system will be exceeded within a certain time frame. Obviously, the longer the facility is proposed to be in service, the greater is the statistical chance of this happening. Also, based on a hazard classification for the structure, the greater the potential for loss of property and/or life in case of a failure, the greater the margin of safety that needs to be built in to the project. However, the design standards should be arrived at after a review of the minimum regulatory standards and the owner's specifications.

From the preliminary hydrologic studies, design flows for both mid-range storms and the maximum design storms can be arrived at. Computer programs are readily available for such computations.

7.4.2 HYDROLOGY

7.4.2.1 Introduction

The primary emphasis of mining hydrology is placed on storm water hydrology. The entire hydrologic cycle is depicted in Fig. 7.4.1, where the boxes represent storages and the arrows represent flow. Fig. 7.4.2 indicates the parameters related to storm water hydrology that need to be addressed in the analysis of any single precipitation event.

Hydrographs are developed using the unit hydrograph approach. A *unit hydrograph* is a graph depicting the quantity of runoff resulting from a unit of rainfall excess occurring at a uniform rate, uniformly distributed over a watershed in a specified duration of time, plotted against time. This approach is empirical and is based on several assumptions:

1. Uniform distribution of rainfall excess over the watershed.

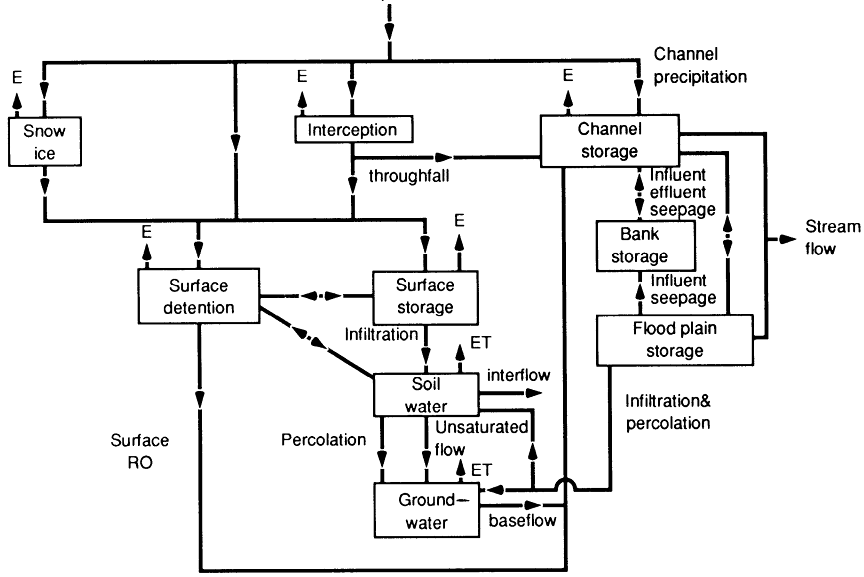

Fig. 7.4.1. Hydrologic cycle.

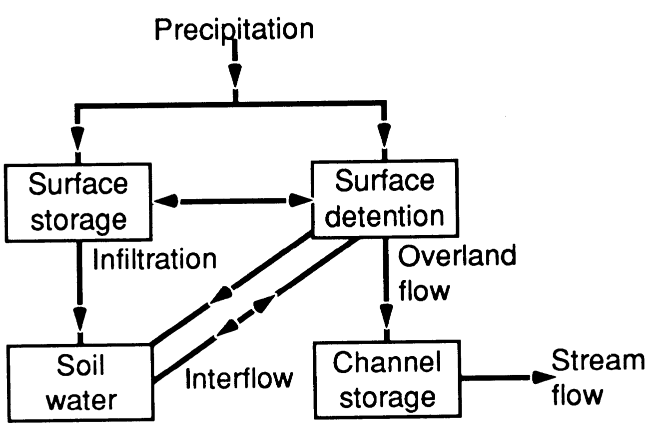

Fig. 7.4.2. Storm water hydrology.

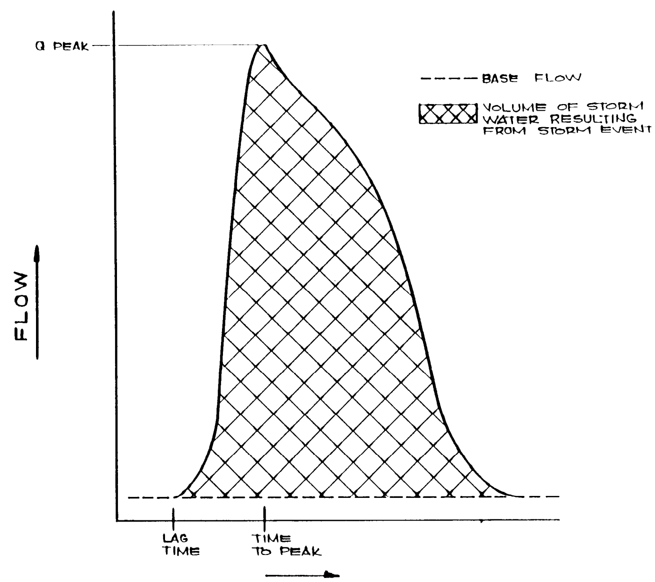

Fig. 7.4.3. Typical hydrograph.

2. Uniform rate of rainfall excess.
3. Superposition or linearity.

The last assumption states that the unit hydrograph reflects all basin characteristics to the degree that the runoff rate is proportional to the runoff volume for a rainfall excess of a given duration (Barfield et al., 1981). Fig. 7.4.3 is a typical hydrograph resulting from a storm event.

7.4.2.2 Design Considerations

If the impoundment is for sediment control or settling fine coal particles out of a slurry, then adequate volume must be available for this purpose. Using the best estimating techniques, such volumes must be estimated.

Any design storm must be specified utilizing both the duration of the storm and the return occurrence of the event. For example, the 10-yr/24-hr storm event is the amount (inches or millimeters) of rainfall that fall in a 24-hr period every 10 years. These numbers are statistical derivations and published by various agencies.

Under regulations, a dam must "contain or treat a precipitation event ('design event') to be based on regional diversity, terrain, climate, and other site specific conditions." Effluent limitation may vary depending on the size of the design storm. At present, during base flow conditions, the 35 to 70-mg/L (0.35 to 0.70-lb/1000 gal) total suspended solids effluent limitation must be met by all dams. If a structure is sized to the 10-yr/24-hr precipitation event, effluent limits for flows caused by rainfall events change to 0.5 mg/L (0.005 lb/1000 gal) settleable solids, and a pH from 6 to 9 is required. Only the pH standard applies during events larger than the 10-yr/24-hr storm. If a dam is

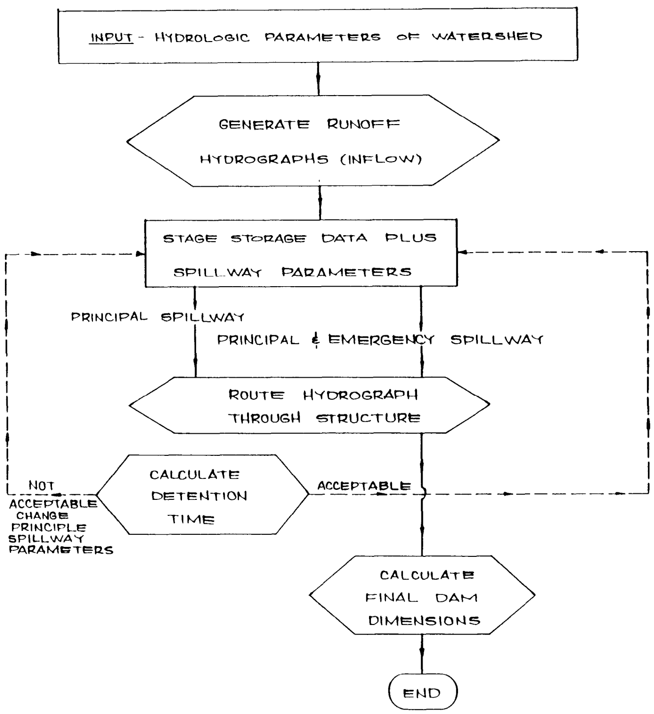

Fig. 7.4.4. Design flow chart.

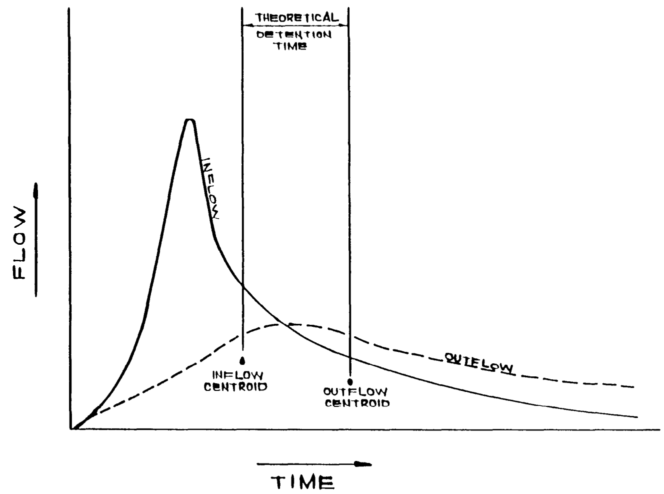

Fig. 7.4.5. Theoretical detention time.

sized to a smaller storm, the EPA effluent limitation of 35- to 70-mg/L (0.35- to 0.70-lb/1000 gal) total suspended solids will still be in effect during any precipitation event.

The design event, typically a 10-yr/24-hr storm event, is analyzed for the principal spillway. (The design procedure is outlined in Fig. 7.4.4.) This event is routed through the dam and the flood crest determined. The emergency spillway invert is set at this crest elevation. This is done in order to prevent flow through the emergency spillway during the principal spillway design event.

The combined principal and emergency spillways are required by the regulations to safely pass a larger storm, typically a 25-yr/24-hr precipitation event. This storm is routed through these spillways and the flood crest determined. To calculate the top of dam elevation, free board is added to the flood crest elevation, typically 1 ft (0.3 m), and a percentage of the dam height (the difference in elevation between the upstream toe of the dam and the emergency spillway crest) is added for settlement, typically 5%. The difference in elevation between the upstream toe of the dam and top of the dam is referred to herein as the total embankment height.

Fig. 7.4.5 illustrates the definition of *theoretical detention time.* This time is defined as the time difference between the centroid of the inflow hydrograph and the centroid of the outflow hydrograph for the design event. The greater the detention time, the better the effluent quality will be. Care must be taken to prevent short-circuiting that would invalidate detention time calculations.

Once a dam exceeds a certain height or reservoir storage, the design criteria become more severe. Also several other government agencies, such as MSHA or the state water division, may want to review the design.

There are many available numerical procedures for flood routing hydrographs through reservoirs. Computer models are modified numerical adaptations of the graphical Puls method (Warsaw et al, 198 1). This method solves the continuity equation in the form,

$$S_2 - S_1 = \frac{(I_2 + I_1)\,t}{2} - \frac{(O_2 + O_1)\,t}{2} \qquad (7.4.1)$$

where S is storage, I is inflow rate, O is outflow rate, and T is time interval over which flow rates are averaged.

This equation states that the net change in storage, be it positive or negative, is the difference between the volume entering the pond and the volume leaving the pond over any given time period. All routing procedures, graphical or numerical, are nothing more than solutions to this equation over a small period of time.

7.4.2.3 Types of Spillways

Three different spillway systems can be utilized. The first system is a trickle tube-emergency spillway system (Fig. 7.4.6). This consists of a single pipe running through the dam embankment. The invert of this pipe is set at the sediment pool elevation. An emergency spillway should be used in conjunction with the trickle tube, and can either be cut in soil or rock or be a lined channel.

The next system is a riser-emergency spillway system (Fig. 7.4.7). This consists of a vertical pipe in the pool connecting to a smaller pipe that runs under the dam. The same emergency spillway installed with the trickle tube should be installed with the riser.

A combination trickle tube, riser, and emergency spillway system (Fig. 7.4.8) can be utilized in order to increase detention times while bringing down the height of the dam.

The methodology used to compare the data generated by the reservoir routing program is relatively straightforward. To explain the rationale involved, a discussion of the hydraulics involved is essential.

Several different types of flow are possible through the spillway systems analyzed. In the riser-principal spillway systems, the following types of flow are possible:

1. Weir controlled.
2. Orifice controlled (riser).
3. Orifice controlled (principal spillway).
4. Pipe flow controlled.

At very low heads, the riser crest acts as a weir and weir flow controls. As the head increases, the inlet begins to act like an

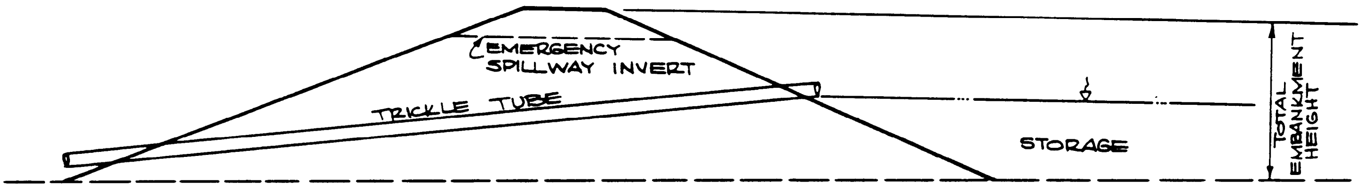

Fig. 7.4.6. Trickle tube—emergency spillway dam.

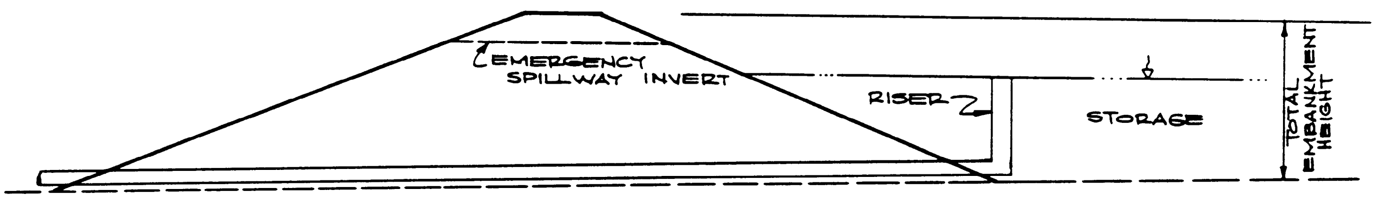

Fig. 7.4.7. Riser—emergency spillway dam.

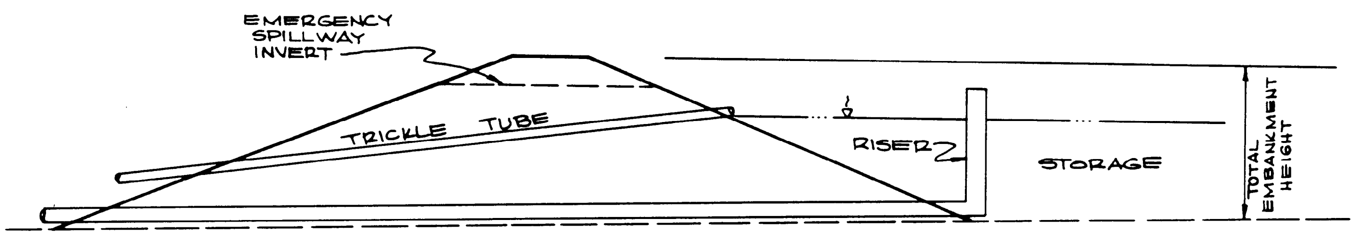

Fig. 7.4.8. Combination trickle tube—riser emergency spillway dam.

orifice with the riser crest still controlling the flow. When the head and flow rate reach a certain point, the principal spillway controls the orifice flow through the system. As the head continues to increase, the principal spillway pipe begins to flow full, and pipe flow will control. In order to determine which type of flow is controlling at any one point in the routing procedure, the flow rates for each control are calculated. The control allowing the smallest flow controls the system.

Flows possible in the trickle system are:

1. Weir controlled.
2. Orifice controlled.
3. Pipe flow controlled

The behavior for this system is similar to that of the riser-principal spillway system. Flow transitions occur from weir flow to orifice flow to pipe flow, depending on the amount of head available at any certain time. The controlling flow at any one point in the routing procedure is determined by computing the flow rates for each control. The one allowing the smallest flow controls the system.

When analyzing the combined trickle tube and riser-principal spillway systems, the same procedures outlined previously are used. Since the pipe systems are not dependent upon each other, flow-rate calculations for the two systems are done independently and their outflows added to find the total outflow at any point in the routing procedure.

In most cases, weir flow is sustained for a comparatively short amount of time in all systems. Orifice flow controlled by the riser is also relatively short term. Orifice flow and pipe flow, controlled by the principal spillway and trickle tube, are responsible for the majority of flow out of the dam. Since the trickle tube and principal spillway are controlling most of the flow out of the dam, results will emphasize comparison of these two entities. Therefore systems with the same size pipe controlling outflow are compared to see how different types of systems affect detention time.

Typically, the detention time for the trickle tube system is the longest of any of the systems. Detention times for the riser-principal spillway systems varied little with the size of the riser and are much lower than that of the trickle tube. As the riser diameter increases, a slight decrease in detention time occurs. This is due to the increased weir and orifice capacity of the larger riser. Since the riser controls flow for just a short period to time, very little change is detected in detention time.

When combining a trickle tube with a riser-principal spillway system, effects on a detention time can be predicted. By adding a riser-principal system to a trickle tube system, the flow rate from the dam would increase thereby decreasing detention time from the trickle tube system. Detention time for the riser-principal spillway system would increase with the addition of a trickle tube below the riser elevation. This would allow the initially slower trickle tube flow rate to control until the water surface reached the riser elevation. The centroid of the outflow hydrograph would be shifted to the right and detention time increased.

7.4.2.4 Hydrographs

Any hydrograph represents the response of a watershed to a particular storm event. Fig. 7.4.3 is such a hydrograph. The hydrograph graphically represents the flow at a particular spot in the watershed plotted against a time axis.

The base flow of the stream is represented in Fig. 7.4.3 as a dashed line. The lag time is the time the watershed takes to respond to a certain storm event, that is, the time it takes after a storm event before flow resulting from the storm occurs. Of course, lag time depends upon many factors including but not limited to soil moistures, types and extent of vegetations, topography, soil types, geography, etc.

The area under the curve but above the base flow represents the volume of the storm that ran off as surface water. The response of a watershed can then be estimated by examining the hydrograph for lag time, time to peak, the maximum flow, and the quantity of water discharged.

7.4.3 EMBANKMENT DESIGN

The design of the embankment itself and the configuration of the spillway system is the basis for implementing the information obtained by the hydrologic analysis. We discuss here the various materials and design considerations that need to be taken into account.

7.4.3.1 Construction Materials

Generally, the materials to be utilized in the construction should be taken from an area adjacent to the embankment itself in order to achieve the best economics possible. These materials may either be earth or rock. Additionally, in conjunction with many mining projects, the reject material from mineral or coal processing may be utilized to construct the embankment. In this case, it is very important to obtain representative samples of the material to be utilized, and to analyze not only their physical characteristics but also the potential environmental dangers that could result from utilization of the materials.

An exploration plan must be undertaken in order to determine the characteristics of the material so that the designer can utilize these characteristics in his/her analysis. Both disturbed and undisturbed samples should be taken in the soil exploration phase. The undisturbed sample can be utilized to classify materials "in place," and the disturbed samples can be utilized to determine the common characteristics that could be achieved during the construction process.

As in any program, the exploration phase must be documented well. Survey controls should be utilized to locate the exact placement of the samples. Any drilling and/or classification that is to be done should have logs prepared that detail at a minimum the following information:

1. Job name, boring. Sample number, sample location. Surface elevation.
2. Degree of saturation, observations.
3. Complete classification of the material, whether it be soil or rock.
4. All samples correctly and completely labeled immediately after taking.

In the exploration phase, care should be taken to totally classify the types of material that can be utilized. Many variations of a design can be prepared, depending upon the type of material to be utilized. If adequate descriptions of the materials are available to the designer, he/she can evaluate the most effective combination to utilize in his/her design. It is possible that additional sampling might have to take place after the designer completes his/her evaluation.

In evaluating the sources, the haul distance, access, location of groundwater, and the moisture content of the materials must be considered.

7.4.3.2 Foundation

Drilling must take place in the area that the dam is to be constructed in order to determine the foundation requirements for the embankment. The underlying rock needs to be pressure tested to make sure there are no cracks or fissures that would allow water to seep under the embankment. At times, by moving the embankment a small distance, such cracks can be avoided. A small amount of analysis and testing at this point will insure a proper working embankment.

The soils underlying the proposed embankment must also be analyzed as to their permeability and makeup. The size of the cutoff trench to be utilized will be determined by this analysis. Fig. 7.4.9 is a common cross section showing the normal design of such a cutoff trench. This cutoff trench insures that the seepage under the embankment will be kept to a minimum.

7.4.3.3 Embankment

The materials to be utilized in the embankment should have been classified during the exploration phase detailed previously. The embankment can be constructed of earth materials, rock materials, mine reject material, or possibly even concrete. For the purpose of this discussion, it is assumed that a concrete embankment would not be utilized.

The materials to be utilized in a rock fill dam should be strong, hard, durable, and relatively dense. Gradation limitations are not restrictive for such an embankment. The materials placed on the upstream and downstream slopes of such an embankment must be capable of resisting weathering. Usually, it is most economical if the materials to be utilized in the rock fill dam are taken from the area to be inundated by the reservoir.

If the material to be used to construct the embankment consists of earth fill, the natural materials and their ability to be utilized or compacted in the construction process will be examined. Since these types of embankments usually have various zones within them, the material may need to be separated. This requires close control during the construction process. As shown in Fig. 7.4.10, seepage control through the embankment, or making sure that the saturated zone stays small, is important. This is achieved by placing drains into the embankment or impervious cores to greatly decrease the amount of seepage through the embankment itself. The placement of these drains and cores is not done lightly by the designer. It is important in the construction process to insure that these are accurately placed.

7.4.3.4 Spillways

Spillways are designed based upon the anticipated flows as analyzed by the hydrologist. If the spillways are to be cut into the natural rocks and soils, care must be exercised to insure that the fracture systems in the rock structures would not cause the area to be unstable. If the spillways are to be cut into earth materials, the natural slopes adjacent to this spillway need to be studied with great care.

It must determined whether the spillway needs to be lined or not. The need for lining is determined by the geology of the region and the velocity of the spillway discharge. Obviously, materials that will slake over time need to be lined. In fact, it may be cheaper to line the channel with either concrete or asphalt initially rather than worry about the long-term maintenance of such a channel.

During the embankment design process, decisions need to be made as to the materials to be utilized, to what degree they need to be worked during the construction process, the place-

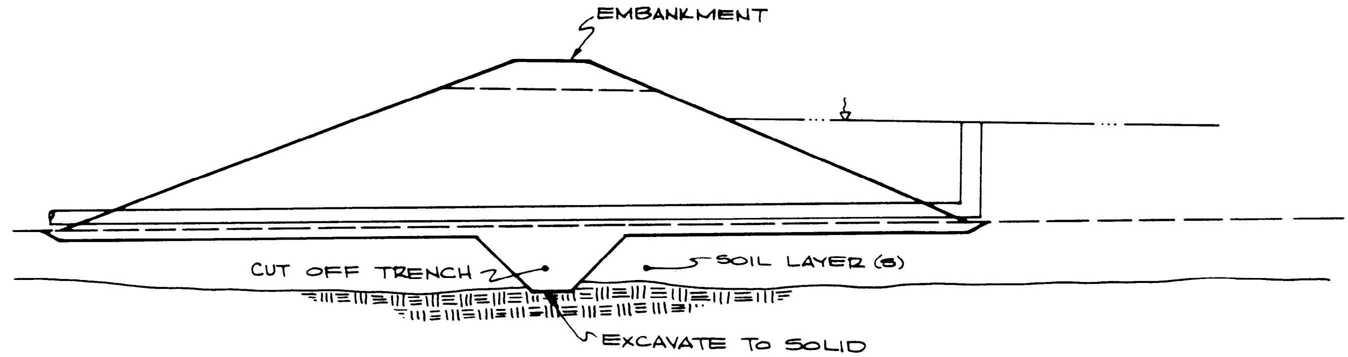

Fig. 7.4.9. Cutoff trench.

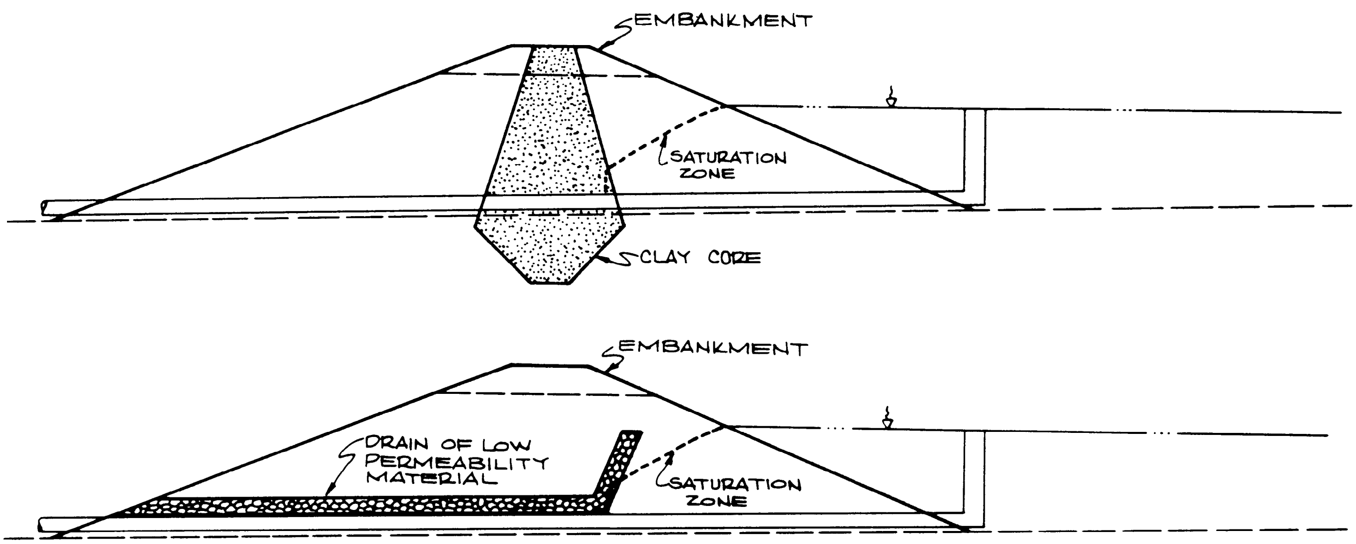

Fig. 7.4.10. Seepage control.

ment of the embankment to insure its proper functioning, and the materials to be utilized in the spillway construction.

7.4.4 CONSTRUCTION

The best studies, planning, and engineering design processes will be totally invalidated by poor construction practices. A definitive quality control and quality assurance plan must be prepared and adhered to during construction practice.

7.4.4.1 Quality Control/Assurance

Quality control and quality assurance are often confused. These terms, as defined by the Canadian Standards Association, are as follows:

1. *Quality control:* those actions that provide a means to measure and regulate the characteristics of an item or service to contractual and regulatory requirements.

2. *Quality assurance:* a planned and systematic pattern of all means and actions designed to provide adequate confidence that items or services meet contractual requirements and will perform satisfactorily in service.

What this means to the designer of an impoundment is that the quality assurance plan must be prepared along with the design that will systematically map out the actions that need to be taken to insure that the impoundment is properly constructed. The quality control program is then the program that carries out those actions and institutes the controls that are designed in the quality assurance program. A quality assurance program must be developed in order for a quality control program to be effective.

The person onsite observing the construction practices should be someone who realizes the consequences of deviations from the plan. Often, during the construction process, items are encountered that will necessitate a change in the construction practice. All such changes need to be noted. The usefulness of daily inspection logs and either pictures or videotaping of the construction cannot be emphasized enough.

7.4.4.2 Construction Practices

The following identify several different aspects of construction that need to be emphasized.

Survey Controls: During the design process, mapping was accomplished using either aerial or ground methods. Regardless of which method was utilized, permanent points that can be recovered both for horizontal and vertical needs should be placed in areas where construction processes will not disturb them. These survey points, or bench marks, should be referenced on the drawings. Multiple bench marks should be placed so that, if some are disturbed, the others can be utilized to replace one of the points. From that point onward, all actions at the scene of the construction should be referenced to these points. A baseline should be established in such a way that a single person using tapes or hand levels could reestablish approximate locations of any actions. All this information should be entered in the daily log for later reference.

The topography of a region is mapped and is critical to the design of an impoundment. The amount of material to be stored behind the impoundment changes as the topography is changed. Often during the construction of an impoundment, it is planned that excavation take place in the impoundment pool itself. While this excavation is taking place, a greater or lesser amount of material than planned may be taken. After the area is excavated and regraded, another survey needs to take place so that the designer can examine the topography of the pool to make sure that it does meet the minimum storage standard required. Also it is possible that if a greater storage area is behind the dam than planned, then additional regulatory and design requirements may be instituted. This needs to be examined as the construction process proceeds.

As construction proceeds and quality control tests take place, as will be described later, these need to be referenced in the daily log, along with their specified location. If a test fails, the location should be noted and a retesting noted at the same location at a later date. One cannot emphasize enough that this material must be organized and taken in a systematic manner.

Material Quality Control: All materials that are utilized in the construction of an impoundment should have specifications for that material. Obviously, in the design process, materials that are onsite to be utilized will be analyzed. Any soils, rocks, etc., that will be utilized have been tested. It is important that the test be representative of the type of materials to be encountered. If the inspector notes that new materials have been encountered, it is important to do new tests on this material. All of the design processes assume certain parameters. If these parameters are not achieved, the design process is voided.

Often the embankments will be constructed of earth materials, and compaction is specified. It is important that compaction testing be performed often and documented well. Prior to placing the material, the ability of the material to be compacted should be analyzed using laboratory tests, which are referred to as the proctor curves and modified proctor curves. What these curves represent is the ability of a material to be compacted. Fig. 7.4.11 represents these curves.

Note that when a material has very little moisture within it, it can be compacted only poorly. As the moisture content rises, the ability of the material to compact increases. Ultimately, it reaches a maximum compaction ability, and as more moisture is added, the ability for it to compact decreases. The top of the curve is called the "100% proctor," and it is normal practice for designers to specify that the material compacted reach a given percentage of the proctor.

Since construction practices take place in the environment, the moisture present will vary depending on the season and the weather. The moisture content of the material to be compacted has to be checked often to make sure that it does meet the degree of compaction specified. Also present in Fig. 7.4.11 is a curve that illustrates the conductivity of a material in relationship to moisture content as it is compacted. Note that the conductivity is much less, meaning that less seepage would occur, as compaction is achieved on the "wet" side of the proctor curve. Less seepage through an embankment is desirable; therefore, construction practice should try to achieve the desired compaction by utilizing the wet side of the curve.

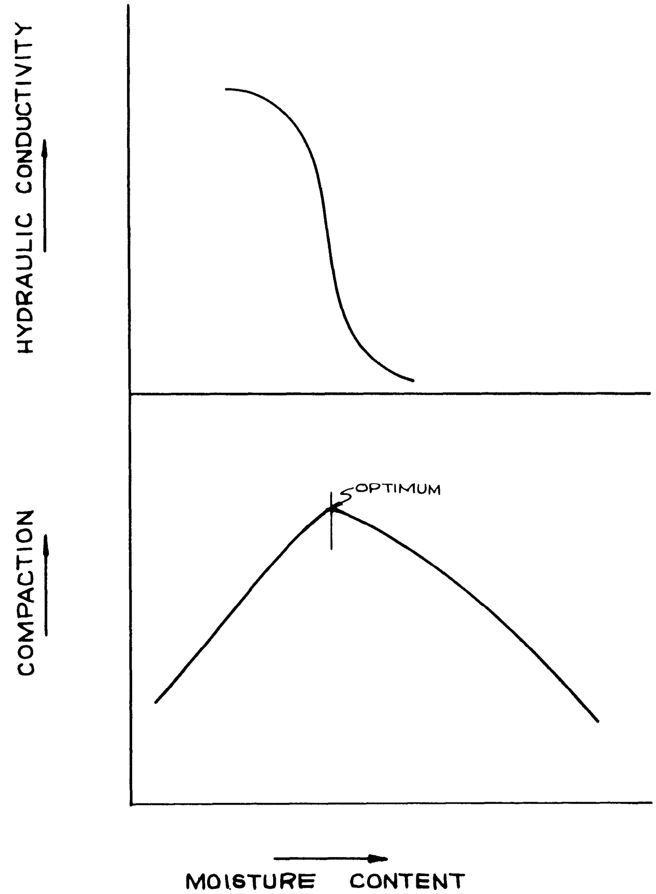

Fig. 7.4.11. Proctor curve.

Compaction testing can be achieved using many methods, but the most convenient and modern would be testing using nuclear density instruments. These are readily available from manufacturers. This allows for instantaneous readouts concerning compaction and moisture contents. Other methodologies such as the balloon or sand cone methods are available, but results generally take hours to achieve.

Piezometers: As construction proceeds, the installation of piezometers may be specified. These are for the protection of the owner and tell him where the saturated zone within the embankment lies. As mentioned previously, it is desired that the seepage remain low. There will be some degree of saturation within the embankment, and the designer allows for this. After construction and during the inspection process, the saturated zone needs to be checked against the design standards. Care should be taken during the construction process to insure that these piezometers are placed in the proper locations, and that any compaction around them does not damage the piezometer.

Cutoff Trenches: Normally, an impoundment is placed on top of a valley floor where soils have accumulated. It is possible that as the permanent pool establishes itself within the impoundment, water will seep into the soil layer and under the impoundment itself. To insure that this does not happen, a cutoff trench needs to be constructed. Fig. 7.4.9 shows such a cutoff; it is a cross section of an embankment illustrating the cutoff trench. Construction equipment should be utilized to excavate as deep as possible in this area and material compacted back in the void. This will insure that seepage under the embankment does not occur.

Anti-Seep Collars: Anti-seep collars may be specified along the pipe. The purpose of anti-seep collars is to prevent seepage along the interface between the material in the dam and the pipe itself. Because of fears of the damaging pipe, maximum compaction may not be achieved. Additionally, if bedding materials are placed around the pipe, at time the permeability of the bedding material is less than the embankment itself, it provides an easy route for seepage to occur. It is of utmost importance that these anti-seep collars be installed and that as much compaction as possible be achieved around the pipe. Normally, this is achieved by either hand compacting or using a mechanical compactor commonly referred to as a "jumping jack." Large compaction equipment should not be utilized around a pipe itself.

In summary, construction practices are extremely critical. Experienced personnel should be utilized to monitor the construction to insure that the assumptions used in the design are met in the construction process.

7.4.5 INSPECTION

After an impoundment is constructed and placed into service, the long-term maintenance of such a facility is of concern to the designer. Periodic inspections are mandated by regulations, but in the best interest of both the owner of the facility and the engineer in charge of operation of the facility, periodic inspections should be made. As with construction quality control, the inspector should be someone who is knowledgeable as to what impact various deviations from approved plans could make on the functioning of an impoundment. Following is a discussion of various aspects of inspection.

Short Circuiting: In the design of the impoundment, the designer assumes the principle of last-in/last-out. In other words, the inflow into the impoundment will flow out of the impoundment only after the rest of the volume of the impoundment is displaced. Obviously, in real life, this does not happen. Short circuiting is a phenomena that occurs where a path of flow exists through the impoundment. There will be areas of the impoundment where water is stagnant and is standing, and there will be other areas where flow through the impoundment occurs almost as if it was a stream flowing in another body of water. This is very simple to check for.

If a nontoxic dye is placed in the upstream inflow, any areas where short circuiting may appear can be easily seen in the surface of the impoundment. In addition, the length of time it takes for the dye to appear in the outflow can be noted. In a low-flow situation with an impoundment that should have had a two-day retention time, the author has observed dye flowing through a facility within 2 min.

There are several aspects within the design process that will minimize short circuiting. Obviously, the inflow must be placed as far away from the outflow as possible. In addition, barriers can be constructed in the impoundment either using wood or fabric material. Filter cloth can also be utilized to diffuse the flow. This forces the water to seek paths other than a straight path through the impoundment to the outflow.

Phreatic Surface: The *phreatic surface* within the impoundment is the surface area at the junction between the unsaturated and saturated zones. Since there is no material that is totally impermeable, the water stored behind an impoundment will force water through whatever structure is constructed. As noted in a previous section, the permeability of the material is the key factor. Designs will be prepared in order to make sure that the phreatic surface or saturated zones remain at a minimum within the embankment itself. The strength of a material is closely related to whether the material is saturated or not. Obviously, the material is weaker when saturated; therefore, if the phreatic surface is higher than the embankment, the factor of safety for failure is lowered. It is safe to say that most failures of embankments and slopes are connected to a higher saturated zone, or greater zone than was anticipated from the design.

There are several ways to insure that the phreatic surface does not rise within an embankment. It was noted in 7.4.4.2, that piezometers need to placed in the embankment itself. These piezometers need to be checked for the water surface, or the phreatic surface, and these comparisons made with design assumptions. If the water is higher than the design assumption, then the design needs to be verified to ensure that the embankment has a factor of safety that might be desired by the regulatory agencies or the owner.

Vegetation: It is extremely worthwhile to step back from the embankment during the inspection process and observe the vegetation growing on the outslopes of the embankment and downstream of the embankment. The vegetation that is growing in a saturated area will be different from that growing in an unsaturated area. Since the phreatic surface tends to assume a constant elevation (since water seeks its own level), a different type of vegetation can be noted at a specific point on the embankment. Usually in spring, because of the moisture, the saturated zones will be greener. Later on, in fall, the vegetation will be taller and a different type of species. This is important because normally it is not anticipated that the phreatic surface will actually outcrop on the downslope side of the embankment. Such observations could represent the saturated zone appearing on the downslope of the dam. When this is observed, it must be noted and reacted to immediately.

Walking downstream, it is important to notice the types of vegetation as well. If the cutoff trench is not constructed properly, water may seep through the soil layer next to the surface underneath the embankment and rise to the surface some distance downstream from the embankment itself. If water is observed coming upward out of the ground, it would be wise to check its source. Obviously, it cannot assumed that it is always from the embankment, but if it occurs immediately downstream from the impoundment, this needs to be investigated. Again, the use of dyes or tracers to verify the source of the water is the easiest way to verify this aspect.

Spillways: Spillways are designed for specific flows given normal functioning. Obviously, any restriction to the operation of the spillway impacts on the amount of water that can flow through a spillway.

Emergency spillways are often cut into rock. If the rock is friable and weathers easily, such as occurs with certain shales, this rock through the years might fall and restrict the flow in the emergency spillway. Obviously, if the slopes are unstable next to the emergency spillway, a slide may occur and partially or totally block the emergency spillway.

Examination of the principal spillway pipes is of utmost importance as well. In a riser-type design, brush and debris often may clog the inlet into the spillway system. This needs to be noted and removed from the inlet. Often a boat must be kept adjacent to the embankment in the designs where the spillway barrel may be in the middle of the embankment itself.

Pipes must be inspected to insure that if the inlet and outlets are not protected by headwalls, machinery and/or persons have not bent the ends of the pipe inward. When corrugated metal is used, often the ends can become bent downward. This, in turn, reduces the cross section available for flow and will restrict the amount of flow through the pipe.

Sediment Load: Any impoundment will have a certain amount of fines settle out in the pool. Obviously, in the case of a sediment pond, this is the reason for the embankment being constructed in the first place. In other areas, any inflow will be carrying suspended solids and settleable solids, and they will be deposited in the pool. In the design process, there is allowed a certain amount of load for this. If the volume allotted for this sediment is exceeded, the hydrology of the design is invalidated. Inspections must be made to examine the volume of the sediment placed in the embankment itself and the embankment cleaned out if necessary.

All of the foregoing items need to be inspected in order to insure that the quality control achieved in the design and construction process is not invalidated due to improper operation of the impoundment itself.

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Section 8 Mine Exploitation

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Chapter 8.0
INTRODUCTION

SCOTT G. BRITTON

Mine exploitation is a broad subject whose focus is the technical skills or efforts needed for the extraction of mineral from a known ore body. Mine exploitation can be considered the very core of the mining engineering discipline. It begins at the conclusion of the exploration and development stages of mining and ends when the ore body is exhausted and the mine (or mines) closes and is reclaimed.

Section 8 on mine exploitation is comprised of seven chapters: (1) classification of mining methods, (2) mine surveying and mapping, 3) systems engineering, (4) computer methods, (5) labor relations and training, (6) management and administration, and (7) mine closure, sealing, and abandonment.

Each chapter focuses its efforts towards providing detailed and useful information for the engineer on basic mine exploitation principles. The information can be applied to a broad range of mining problems without regard to mineral type or mining method.

The first chapter, classification of mining methods, is a rational and systematic approach to the organization of mine exploitation methods. A unique tree structure of mining elements is presented in support of the engineering-operations interface of mining. The reader is able to clarify the engineering problems he/she faces while obtaining a broader understanding of the relationship of these problems to the overall mining process. Prior problems or elements contributing to the current problem(s) can be readily identified, and the reader as well gains some insight into problems downstream of the current problem.

The second chapter concentrates on mine surveying practices, particularily on aerial mapping practices. Traditional mine surveying receives less emphasis in favor of emerging aerial surveying techniques. The reader will be able to understand all opportunities to use aerial mapping as well as the elements needed to manage such a program or the results of such a program.

The third chapter covers the diverse elements of mine systems engineering. This subject area encompasses the principles of industrial engineering as they are applied and refined for mining operations. The reader benefits from a discussion on improving operating functions through systematic study and refinement.

The fourth chapter covers the use of computer models to manage the exploitation function of a new or existing mining operation. The reader obtains a thorough discussion of the advantages and disadvantages of using the computer in mine exploitation applications. Specifications for computer use, as well as detailed examples, are provided.

The fifth chapter discusses the impact of improving the human element of mine exploitation through labor relations and training. Specific discussions of labor-management practices and the use of training as a means of improving the labor element are presented for the reader.

Management and administration is the sixth chapter in the exploitation section. The discussions in this chapter cover the structure and purposes of a variety of management systems used in mining operations today. Specific position descriptions, responsibilities, and authorities are covered within the confines of the chapter. The reader will obtain a complete understanding of the structure of the mining business and why it exists as it does.

The final discussion of the mine exploitation section is one covering the closure, sealing, and abandonment of a mining operation. The authors have compiled and presented current information on the legal, engineering, and public relations process for closing and abandoning a mining operation. The reader derives specific regulatory authority for closing activities on a federal and state basis. He/she is able to plan and budget the closing process as well as determine all related public relations elements needed for successful mine closure.

In reviewing the entire mine exploitation section, the efforts of the authors have been to provide the reader with hands-on information of interest for today's mining engineer.

Chapter 8.1
MINING METHODS CLASSIFICATION SYSTEM

L. ADLER AND S.D. THOMPSON

8.1.1 INTRODUCTION

The purpose of a classification system for mining methods is to provide an initial guideline for the preliminary selection of a suitable method or methods. Its significance is great as this choice impinges on all future mine design decisions and, in turn, on safety, economy, and the environment.

The choice of a mining method assumes a previous but cursory knowledge of the methods themselves. It also assumes a brief understanding of ground control and of excavating and bulk handling equipment. In the formal mine design procedure, the choice of mining methods immediately follows ground control and equipment studies, and feeds directly into the crucial *milestone diagram* where regions of the property are delineated as to prospective mining methods (Lineberry and Adler, 1987). This step in turn just precedes the subjective, complex, and critical *layout and sequencing study*.

To develop the proposed classification system adopted here, many existing ones, domestic and foreign, were examined and incorporated to varying degrees. The result is deemed more systematic, inclusive, and understandable than its predecessors (see references, e.g., Stoces, 1966).

Subsequent sections (13, 16, 17, 21, 23) elaborate on the selection and comparison of mining methods.

8.1.2 BACKGROUND

8.1.2.1 Input Statement

A comprehensive statement has been developed to provide a rapid checklist of the many important input parameters (Adler and Thompson, 1987). The three major areas are (1) natural conditions, (2) company capabilities, and (3) public policy (Table 8.1.1). Those parameters appearing early are generally the most important. Natural conditions, which are almost invariant, require that a dual thrust be maintained concerning resource potentials and engineering capabilities. An additional basic distinction occurs between geography and geology. For company capabilities, fiscal, engineering, and management resources must be recognized. This includes the scale of investment, profitability, and personnel skills and experience. Public policy must be considered, particularly as to governmental regulations (especially safety and health and environmental), tax laws, and contract status. Some of the latter input factors are held in abeyance until near the end of the investigation, and then considered as modifying factors. This organization duplicates but tightens others (Hartman, 1987a).

Table 8.1.1 Input Statement Categories

Primary Categories (Dependency)	Secondary Categories
Natural conditions (invariant)	Geography Geology economic engineering
Company capabilities (variant)	Business administration Monetary aspects Management aspects
Public policy (semi-variant)	Regulations Taxes Contracts Incentives
State of art (mining engineering)	Salient distinctions *Total Systems* (design/control), *Encumbered (and Regulated) Space* *Full Spectrum Practice* (manage/evaluate) Professionalism

8.1.2.2 Spatial Description

Most mineral deposits have been geometrically characterized as to an idealized shape, inclination, size, and depth. Complex or composite bodies are then composed of these elements.

Ideal shapes are either tabular or massive, with chimneys (or pipes) being subordinated. Tabular deposits extend at least hundreds of feet (meters) along two dimensions, and substantially less along a minor dimension. Massive bodies are approximately unidimensional (cubic or spherical), being at least hundreds of feet (meters) in three dimensions. A modification is recommended later to achieve closure with tabular deposits. For tabular deposits, the inclination (attitude or dip) and thickness are crucial. Inclinations range from flat to steep (Table 8.1.2) (Hamrin, 1980; Popov, 1971a).

In surface mining, the inclination limits the advantageous possibility of being able to cast waste material nearby, as opposed to hauling it a distance and then storing it. For flat deposits, especially when fairly shallow (see the following), an area can be successively opened up and the waste then be cast into the previously mined-out strips, a substantial economic advantage. Casting in its normal sense is not restricted to the use of rotating excavators; broadly, it means relatively short-distance hauling of waste, which can also be done with mobile loaders and/or trucks or with mobile bridge conveyors. For steeper (and deeper) deposits, stable pit slopes become important (Table 8.1.3) (Hartman, 1987b; Popov, 1971b). Where the deposit inclination exceeds that of the stable slope, both the hanging wall and footwall must be excavated and the increased waste then handled and placed.

Table 8.1.2 Tabular Deposits Classified by Attitude and Related to Bulk Handling and Rock Strength

Class	Attitude or Dip	Bulk Handling Mode	Rock Strength
Flat	≤ 20°	Use mobile equipment (and conveyors)	Weak rock (surficial)
Inclined	20–45°	Use slushers (also metal plate can vibrate—as gravity slides)	Average rock
Steep	≥ 45°	Gravity flow of bulk solids	Strong rock (at depth)

Table 8.1.3 Surface Pit Slopes Related to Rock Strength and Time

Rock	Maximum Pit Slope Short Term	Maximum Pit Slope Long Term
Strong	40–45°(– 70°)[a]	18–20°
Average	30–40°	15–18°
Weak (soils too)	15–30°	10–15°

[a]Note: Infrequently up to 70°

Table 8.1.4 Underground Deposits Classified by Thickness

Class	Deposit Thickness Coal	Deposit Thickness Ore	Comments
Tabular			
Thin	3–4 ft (0.9–1.2 m)	3–6 ft (0.9–1.8 m)	Low profile or narrow mine equipment
Medium	4–8 ft (1.2–2.4 m)	6–15 ft (1.8–4.6 m)	Post and stulls ≤ 10 ft (3.1 m)
Thick	8–15 ft (2.4–4.6 m) pillar problems	15–50 ft (4.6–15.3 m) can cave (steep dip)	Small surface equipment Crib problems
Massive	≥ 15ft (4.6 m)	≥ 50 ft (5.3 m)	Pillar problems or poor recovery Benching necessary Caving considered

Table 8.1.5 Deposits Classified by Depth

Class	Deposit Depth: Underground (a measure of overburden pressure) Coal	Ore	Surface
Shallow	≤ 200 ft (61 m) slope entries possible	≤ 1000 ft (305 m)	≤ 200 ft (61 m)
Moderate	400–800 ft (122–244 m) pillar problems	1000–1500 ft (305–457 m)	200–1000 ft (61–305 m)
Deep	≥ 3000 ft (915 m) bumps,burst,closure	≥ 6000 ft (1830 m)	≥ 1000–3000 ft (305–915 m) open pit

For both surface and underground mining methods, the inclination cutoff values nearly coincide (one for pit slopes, the other for face bulk handling mechanisms, whether mechanical or by gravity). While not identical, they are close enough to use similar values (20º and 45º; see Table 8.1.2).

The thickness of a tabular deposit is also important (Table 8.1.4), with reference primarily to underground work (Popov, 1971c). When several (≥ 3) benches are required, the deposit tends to be treated as massive. Primarily in flat underground deposits, thickness governs the possible equipment height (low profile), and in steep ones its narrowness. Also, in underground mining, the deposit thickness becomes a support problem, especially if effective pillars become so massive that recovery is compromised. When the upper limit of any of these concerns is reached (e.g., benching, equipment size, and pillar bulk), closure with massive deposits occurs for all practical purposes. Pillar size vs. recovery can dictate caving.

Finally, the depth below the ground surface is important (Table 8.1.5) (Popov, 1971d; Stefanko, 1983). For surface deposits, even flat ones, this can obviate casting and require increased waste haulage and expanded dump sites. For underground mining, earth pressures usually increase with depth, consequently raising the support needs. The ground surface location above a deposit must be clearly identified to evaluate other parameters (see 8.1.2.1, Input Statement).

Table 8.1.6 Deposit Classified by Geometry and Type

Geometric Class	Deposit Type	Comments
Tabular	Alluvium (placer)	Near surface—weak
Flat & Inclined	Coal (folded too)	Weak country rock—an erosion surface
	Evaporites (domes too)	
	Sedimentary	Good country rock, thicker
	Metamorphic (folded too)	
Steep	Veins	Can be weakened or rehealed (gouge and alteration)
Massive	Igneous (magmatic)	Strong
	Disseminated ores	Can be weakened

Table 8.1.7 Rocks Classified by Strength

Class	Compressive Strength	Examples
Weak	≤ 6000 psi (41.3 MPa)	Coal, weathered rock, alluvium
Moderate	6000–20,000 psi (41.3–137.9 MPa)	Shale, sandstone, limestone, schist
		Evaporites, disseminated deposit
Strong	20,000–30,000 psi (137.9–206.8 MPa)	Metamorphic, igneous, veins, marble, slate
Very strong	≥ 30,000 psi (206.8 MPa)	Quartzite, basalt, diabase

8.1.2.3 Correlating Deposit Types

The inclination (dip), discussed previously, can be roughly related to the deposit type (Table 8.1.6). Rocks can also be related to strength (Table 8.1.7) (Hartman, 1987c). The strength of the deposit and its envelope of country rock can then be related to its type (Table 8.1.8). For determining pit slopes, (surface mining) and support requirements (underground mining), these relationships become important. Some variations are noted, especially for veins and disseminated deposits. Near the ground surface, a typical geologic column occurs (Table 8.1.9). It involves weathering, jointing, the water table, and stress relief.

8.1.3 CLASSIFYING SURFACE MINING METHODS

8.1.3.1 Depth Related to Inclination

The surface mining classification, although based on the crucial ability to cast waste material rather than to haul it, has other features. These are primarily based on the depth of the

Table 8.1.8 Deposits Related to Geometry, Genesis, and Strength (In Order of Induration)

Deposits Type	Geometry	Genesis	Strength and Stiffness, Deposit/Country Rock	Examples
Alluvium (placers)	Tabular-flat	Surface-stream action deposition (fans, deltas, meanders, braids)	Poor/poor	Sand and gravel Precious metals and stones (tin)
Erosion surface (swamps)	Tabular-flat and thin (possible folding)	Swamps (possible dynamic metamorphism)	Poor/poor to good	Coal
Disseminated	Massive	Underground channels, and multi-faceted advance	Poor/poor	Hydrothermal ores (porphyry coppers and sulfides)
Vein (can be rehealed)	Tabular-inclined (pipes, chimney shoots)	Major underground channels (fissures) gouge, alteration (reheal)	Poor to good/good	Hydrothermal ores (above)
Evaporites	Tabular-flat-thick	Interior drainage	Good/good	Salt, phosphates
Sedimentary (bedded)	Tabular-flat-thick	Shallow seas	Good/good	Limestone, sandstone
Metamorphic	Tabular-flat-thick	Dynamic and/or thermal	Good/good	Marble, slate
Igneous (magnetic)	Massive	Plutonic emplacement	Good/good	Granite, basalt, diabase

Table 8.1.9 Normal Sequence of Near-surface Geologic Column (Related to Ground Water and Rock Stress)

Rock Column	Ground Water	Rock Stress
Ground surface		
Soil (alluvium) mantle	NA	NA
Blocky & seam rock		
Decomposed rock (weathered)	Water table	Stress relief zone
Jointed fresh rock		
Tight rock	—	Constrained zone

NA—not applicable

deposit being a function of its inclination. Flat seams tend to be shallow, and casting is possible; steep and massive deposits trend to depth. From this a number of relationships result.

8.1.3.2 Depth Related to Excavating Technique and Stripping Ratio

Because of the effects of weathering and stress release (Table 8.1.9), excavating becomes more difficult and expensive with depth, following a continuum from hydraulic action and scooping through to blasting (Hartman, 1987d).

As a matter of definition, the stripping ratio (ratio of waste to mineral) usually increases with depth. However, the relatively inexpensive handling of waste near the surface by casting tends to mitigate this increase, permitting higher ratios. The use of mobile, cross-pit, high-angle conveying allows greater pit depths and, along with the mineral value, also influences this ratio.

8.1.3.3 Surface Mining Classification System

Based on the foregoing factors, a surface mining classification has been developed (Table 8.1.10). The classification incorporates information dependent on the intrinsic characteristics of the geometry of the deposit. Quarrying appears to be anomalous because of (1) relatively steeper pit slopes, (2) specialized means of excavating and handling, and (3) less critical amount of overburden. Glory hole mining or its equivalent is making a comeback in very deep open pits using inclined hoisting.

In contrast to the underground classification, the surface one is not formed into a matrix. This is because depth and therefore excavating technique, waste handling, and stripping ratio are all functionally related to the deposit geometry, particularly the seam inclination. No preceding classification recognizes this relationship (Hartman, 1987e; Lewis and Clark, 1964a; Morrison and Russell, 1973; Stout, 1980; Thomas 1973a).

8.1.4 CLASSIFYING UNDERGROUND MINING METHODS

8.1.4.1 General

Normally, two major independent parameters will be considered that form a matrix, unlike for surface methods. These two parameters are (1) the basic deposit geometry, as for surface methods, and (2) the support requirement necessary to mine stable stopes, or to produce caving, a ground control problem (Boshkov and Wright, 1973; Hamrin, 1980; Hartman, 1987f; Lewis and Clark, 1964b; Thomas, 1973b).

8.1.4.2 Deposit Geometry

The deposit geometry employs the same cutoff points for tabular deposits as in the surface classification, but as noted for different reasons. Flat deposits require machine handling of the bulk solid at or near the face; steep ones can exploit gravity (Table 8.1.2), with an intermediate inclination recognized. If stopes are developed on-strike in steep seams, as "large tunnel sections," a new descriptive term, or "step room" (Hamrin, 1980), machine handling can still be used. The resulting stepped configuration causes either dilution or decreased recovery, or

Table 8.1.10 Classification of Surface Mining Methods

Shape, Attitude (Dip)	Deposit Characteristics	Stripping Ratio	Excavation: Waste Handling	Excavation: Excavation	Mining Method
Tabular					
Flat	Near surface	Low	Onsite	Hydraulic, scoop, dig	Placers—hydrosluicing, dredging, solution—at depth
	Shallow	Moderate	Cast	Scoop, dig, light blast	Open cast (strip)—area, contour, mountain top
Inclined		Moderate (remove hanging wall)	Need highwall	Auger	Auger
	Moderate		Haul (to waste dump)	Blast	Open pit
	Deep	High (remove both hanging wall and footwalls)	Haul (to waste dump)		Open pit
				Saw, jet pierce (joints)	Quarry
Massive	Full range	Depends on depth	Haul (to waste dump)		Open pit Glory hole

Note: In situ mining is always possible.

Table 8.1.11 Structural Components Located and Described for Underground Mining

Component (time dependent)	Location & (Material)	Loaded by	Supported by	Comments
Roof (can deteriorate, slough, slake—dry and crumble)	Back and hanging wall (envelope)	Main roof—all esp. overburden (cap rock)	Pillars and fill, also arched (1/5)	Spans ≈ 10 ft (3 m) for coal to 100 ft (30.5 m) for rock
		Immediate roof—body	Artificial supports can remove	Spans ≈ 10 ft (3.1 m) (stand up time)
Pillars & walls (can deteriorate—slough, slake)	Sides, deposit and waste (horses-mainly deposit)	All—esp. overburden	Floor	Critical: 1) stiffness: [slenderness ratio: ≈ 10/1 (coal) to 1/3 (rock)] 2) strength [material] 3) % recovery
Floor (can settle and heave)	Footwall (envelope)	All—through pillar watch water	Country rock can be compacted, removed, drained	Critical: 1) stiffness 2) strength (bearing capacity esp. if water) 3) heave (deep-seated)
Fill (for permanent stability)	Crushed waste, sand, water	All—esp. as pillars are removed	Footwall and floor	Good mainly to support hanging wall. Requires greater than angle of slide and confinement
Artificial support (limited time)	External: Timber (props, sets, cribs, stulls, posts) Concrete gunite (mesh)	Mainly immediate roof	Floor	Deterioration (chemical and stress)
	Internal: Bolts (headers), trusses, cables, grout, cementation	Mainly immediate roof	Anchorage in roof, etc.	Anchorage a concern

both. Since this face can also be benched (≤ 3), stope mining simply reproduces tunneling.

8.1.4.3 Ground Control

Ground control requires knowledge of the structure (opening), material (rock), and loads (pressures). The structural components (Table 8.1.11) are detailed for the strength and deposits (Tables 8.1.5 and Table 8.1.7). The roof, pillars, and fill are of primary concern.

Main Roof: The main roof (sometimes the hanging wall) is distinguished from the immediate roof by being the critical load-transferring element between the overburden and pillars. The immediate roof can be removed (mined out) or supported artificially and lightly. The main roof is defined as the first close-in, competent (strong) seam. If it is only marginally competent, heavy artificial support may keep it stable; if not, then caving can be expected. For a flat seam, the vertical (perpendicular) loads on the main roof are largely due to the overburden and its own body load. Horizontal (tangential) loads or pressures will tend to be uniformly distributed, resulting in a low stress concentration. If bed separation occurs above the main roof, this stress uniformity is enhanced; but at depth, overburden loading tends to decrease separation. Body loads are invariant, while edge loads particularly those due to the overburden can be shifted (pressure arching). The main roof is often sufficiently thick so that it can be arched ($\leq$ 1/5) to increase stability. For coal, stable spans usually are 3 m, while for rock they are generally less than 30 m.

For an inclined seam the main roof is the hanging wall, and the results are similar to a flat seam. Pressures perpendicular to it are more significant then tangential ones, and bed separation due to gravity is less likely.

Pillars: Pillars serve to support the main roof and its loads, primarily the overburden acting over a tributary area. Pillar material consists mainly of the seam itself and sometimes of waste horses in it. Pillars must not only be sufficiently strong but also must be sufficiently stiff, a frequently overlooked requirement. If pillars are not adequately stiff, but still adequately strong, the roof will collapse about the still free-standing pillars, especially when differential pillar (and floor) deflection occurs. The minimum slenderness ratio for pillars to avoid this crippling is inversely proportional to the recovery. The mining of flat, thick seams of coal dramatically reflects this relationship and is a factor in classifying seam thicknesses (Table 8.1.4). For massive deposits, even in strong rock, this makes free-standing pillars of doubtful value. Upper slenderness ratios range from about 10/1 for coal to 1/3 for rock. Continuous vertical pillars are used to separate vertical stopes in hard rock that employ steep, tabular stoping methods. Even with stable ground, these are usually filled soon after mining for long-term stability. When massive deposits along with their cap rock are weak, caving is necessitated, usually performed as horizontal lifts or as block caving. Caving always requires a sufficient span ($\geq$ 30 ft or 9 m), good draw control, and also risks dilution and/or poor recovery. Soft or nonuniform floors (footwalls) act the same as do soft and irregular pillars.

Fill: Fill, often a sandy slurry consisting of crushed waste, cement, and water, can be readily introduced into confined (plugged), inclined, and steep tabular stopes. When drained and dried, this hardened slurry provides permanent resistance to ground movement, especially for the walls or pillars. It is widely used in all but the caving methods. It is either run in progressively as a stope is mined out or done all at once at the end of stope mining. Because of settlement and shrinkage away from a flat back, it is marginally useful for flat deposits.

When timbering is densely placed, especially with square sets, it rivals pillars. It, too, is usually filled as stoping progresses (overhand mining). These relationships are summarized (Table 8.1.12) and lead into the formal classification.

8.1.4.4 Underground Mining Classification System

Based on an understanding of bulk handling and ground control, the underground classification system closely follows earlier ones (Table 8.1.13). The primary difference is that sometimes shrinkage stoping is considered self-supported rather than supported. However, although the broken mineral provides a working floor, it is still supporting the hanging wall (roof). On the other hand when the stope is drawn empty, it remains substantially self-supported until fill is introduced. The disadvantages of the shrinkage method are unique: (1) an uncertain working floor, (2) dilution due to plucking and falls, (3) possibly adverse chemical effects, and (4) tying up about two-thirds of the mineral until the stope is drawn.

Vertical crater retreat (VCR) mining is included in the classification between sublevel and shrinkage stoping (Hamrin, 1980).

8.1.5 OTHER FACTORS

While subordinated, there are additional factors that must be closely evaluated. These deal with the broad impacts on the environment, health and safety, costs, output rate, and others. They are usually evaluated on a relative basis although numbers may also be employed (Table 8.1.14) (Bohskov and Wright, 1973; Hartman, 1987g).

In addition, innovation is always occurring and some is currently of proven value. These include rapid excavation, methane drainage, underground gasification, and retorting (Hartman, 1987h). Some methods are being automated and even robotized.

8.1.6 CONCLUSIONS

Existing mining methods classifications were used and modified whenever possible. However, for surface mining the classification was drastically reworked for greater effectiveness, based on the relationships of the inclination of tabular deposits to depth, so that casting could be emphasized.

Earlier underground classifications were largely retained but now structured to underscore mechanical vs. gravity handling. It also emphasized an understanding of the roof, pillar, and fill aspects of ground control to assist in the stope support evaluation.

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Table 8.1.12 Deposit and Structural Components Related to Underground Mining Methods

Deposit Geometry	Structural Main Roof & Floor	Components Rated[a] Pillars, Walls	Underground Mining Methods	Type
Tabular				
Flat (and inclined)	Good	Good	Room & pillar (spans ≤ 20 ft) (6 m) Stope & pillar (spans ≤ 100 ft) (31 m)	Self-supported
	Good	Poor	Room & pillar Stope & pillar	Supported
	Poor (roof collapses about free standing pillars)	Good	Longwall Pillaring	Caved
	Poor	Poor	Immediately above	Caved
Steep	Good	Good	Sublevel stoping (spans 20–100 ft) (6–31 m) Large tunnel section	Self-supported then filled
	Good	Poor	Hydraulicking—coal (spans 20–70 ft arch) (6–21 m) Shrinkage	Supported then filled
	Poor	Good	Cut & fill	
	Poor	Poor	Sublevel caving & top slice spans ≥ 20 ft (6 m) (for gravity flow)	Caved
Massive	Good	Good	Vertical slices[b]	Self-supported
	Good	Poor	(as above)	Supported then filled
	Poor (cap rock)	Poor	Block caving (spans ≈ 110 ft (34 m) active—end stope used)	

[a]Rated as to strength (and stiffness of pillar).
[b]Horizontal slices can introduce the many problems associated with multiple seam mining.

Table 8.1.13 Classification of Underground Mining Methods Based on Deposit Geometry and Support

Deposit Shape, Attitude (Dip)	Degree of Support: Unsupported (Open Stopes)	Supported	Caved
Tabular			
Flat (mobile bulk handling)	Room and pillar Stope and pillar	Some degree of artificial support for room and pillar and stope and pillar	Longwall (shortwall) Pillaring (esp. room and pillar)
Inclined (mixed bulk handling)	Above with scrapers	Above with scrapers	Longwall (difficult)
	Large tunnel section (on-strike)	Large tunnel section with artificial support	
Steep (gravity bulk handling)	Coal hydraulicking	Shrinkage stoping cut and fill stoping	Sublevel caving
	Sublevel stoping Vertical crater retreat Gravity fill as needed	Timbered stoping (square sets, stulls, gravity) Fill as needed Gravity fill as needed	Top slicing (control dilution and recovery)
Massive	Immediately above mine in *vertical* slices fill—gravity placement. To remove pillars, can mine and then fill *Horizontal* lifts[a]		Immediately above in horizontal lifts block caving (bulk mining)

[a]Ground control problems, especially for coal, see thick seam and multi-seam mining. As pressure increases (esp. with depth), or as rock strength decreases, shift to right for suitable method (towards supported and caved).

Table 8.1.14 Secondary Factors to be Considered in Selecting a Mining Method

Method	Relative Cost	Flexibility/ Selectivity	% Recovery/ % Dilution	Environment	Safety and Health	Output (TPH) and Productivity (ton/employee)	Miscellaneous
Surface							
Placers and dredging	0.05	Low/high	High/low	Difficult Water pollution	Fair	Moderate	Need water Watch weather
Open cast	0.10	Moderate/ moderate	High/low	Blasting can Can reclaim frequently water pollution	Fair	High	Flat topography Watch weather
Open pit	0.10	Moderate/ moderate	High/low	Ground disturbance Waste pile Some water problems	Slope stability (slides)	High	Watch weather
Quarry	1.00	Low/high	High/high	Ground disturbance Waste pile	Slope stability	Very low	Skilled workers Watch weather
Underground							
Room and pillar (coal)	0.30	High/high	50–80/20	Subsidence Water pollution	Ground control Ventilation	High	Pillaring common
Stope and pillar	0.30	High/high	75/15	Good	Ground control Ventilation	High	Benching common
Sublevel stope	0.40	Low/low	75/15	Fill to avoid subsidence	Less, blast from longholes	Moderate	Fill common
Shrinkage	0.50	Moderate/ moderate	80/10 plucking during draw	Fill to avoid subsidence	Poor floor (collapse) Stored broken mineral[a]	Low	Tie-up 2/3 of ore
Cut and fill	0.60	Moderate/ high	100/0	Fill to avoid subsidence	Some	Low	Sort in stope
Timbered Square set	1.00	Moderate/ high	100/0	Fill to avoid subsidence	Smoulder Fall (personnel)	Very low	Sort in stope
Longwall	0.20	Low/low	80/10	Subsidence Water pollution	Good	Very high	High capital ≤ 12° dip ≤ 8 ft (2.4 m) thick
Sublevel caving (Top slicing)	0.50	Low/low	90/20	Severe subsidence disruption	Fair Stored broken mineral[a]	High	Cave width ≥ 30 ft (9.2 m)
Block caving	0.20	Low/low	90/20	Severe subsidence disruption	Air blasts Stored broken mineral[a]	High	Tie-up mineral

[a]Can pack (cement), oxidize, and smoulder.

Chapter 8.2
MINE SURVEYING AND AERIAL MAPPING

BERNARD W. SOLOMON, JAMES MORGAN, SCOTT G. BRITTON, AND CHARLES O. FRUSH (deceased)

Mine surveying is as basic to prudent mining operations as is the employment of skilled labor to win minerals from the earth. The technical expertise needed to accomplish this task varies with the complexity of the mining operation (whether underground or surface, the size of the mining method, etc.) and the desires of mine management to comply with regulations and good mining practice. This chapter looks at two major components of mine surveying. The first is *operational surveying,* whose objective is to tie active mining efforts to spatial and physical control points. The second is *exploratory surveying,* which primarily focuses on aerial surveying, and whose major objective is to locate and define spatial and physical control points prior to commencing mining operations. Exploratory surveying can be done either in new areas or as support for expansion projects.

To accomplish reviews of these two components of mine surveying requires separate discussions in this chapter. The first reviews fundamental operational surveying techniques generally found at operating mines. This discussion is necessarily short because of the ready availability of instructional information elsewhere.

The second segment discusses the relatively new and emerging practice of aerial mapping techniques used for exploratory mapping and base surveying control points. Aerial mapping is the least expensive and best means for large-scale surveying. The use of aerial mapping was pioneered by the US Geological Survey and has been quickly adopted by the mining industry for nearly all major projects.

8.2.1 UNDERGROUND AND SURFACE MINE SURVEYING*

SCOTT G. BRITTON AND CHARLES O. FRUSH (deceased)

Mine surveying practices at underground and surface mines are based on plane surveying methods dating back to the time of the ancient Egyptians. These practices were refined and improved as new equipment was developed and the demand for accuracy increased.

It is not the goal here to completely and fully discuss the principles of mine surveying— there are many texts now available to the reader that address this subject more extensively than this *Handbook* can in the space available. However, the practice of mine surveying is evolving as new instruments and technology are introduced. Therefore, it is important the reader be aware of advancements in this area of mine engineering.

Of major importance in the field of mine surveying is the introduction and use of computer and laser technology for common surveying practices. Theodolites and electric distance-measuring (EDM) instruments are rapidly replacing the transit and tape measures. Computers with plotters and CAD (computer-aided design) capabilities are replacing the old manual methods of planimetering and contouring. These advancements have not fully displaced all older methods, but their use is becoming more commonplace as the technology becomes more cost competitive, better accuracy is demanded, and more personnel are trained to use such technology.

8.2.1.1 Uniqueness of Mine Surveying

The functions of the *mine surveyor* are to (1) provide a network of points of accurately known position, covering the entire mine property, (2) obtain the required data for the preparation of useful maps and sections, (3) provide position and directional control for the guidance of the operations, (4) measure the progress of the work, (6) measure the movement of rock masses, and (5) make miscellaneous surveys as needed.

Mining properties range from very small to extremely large, and from those involving only surface operations to those involving extensive underground workings. Active mines undergo constant change, necessitating continual updating of the surveys. The differences in the character of the deposits, the scale of mining, and the mining methods result in large differences in the surveyor's activities and techniques, but many of the problems involved are common to most mines.

The surface work of the mine surveyor is very similar to that of the plane surveyor in that control points are established by triangulation, trilateration, and traversing. Detail is taken by direct measurement, by stadia, and by aerial photography. The equipment used, the techniques, and the methods of processing the resulting data are likewise very similar. However, underground work is quite different:

1. The lines of sight frequently must be carried through constricted openings, often involving short lines of sight and awkward setups.
2. The lighting generally is poor, requiring illumination of the backsights and foresights, and the crosshairs.
3. The ambient conditions often are difficult, including falling water, high temperatures, poor visibility, and heavy traffic through the area being surveyed.
4. The surrounding rock may be unstable, resulting in movement or loss of the surveying stations, as well as hazard to the surveyors.
5. The points to be measured often are difficult or impossible to reach.
6. Steep vertical sights often are necessary, requiring the use of special equipment.
7. Many working levels commonly are involved, requiring the transfer of position and orientation to each with a high level of precision.

Because of these uniquenesses, this segment concentrates on surveying problems peculiar to the mine surveyor, chiefly in underground work.

8.2.1.2 Mine Coordinate System

A system of coordinates is essential for all permanent mining operations. It is very desirable that all mining operations in a given area be tied into the same system, as this minimizes problems of boundaries and connections. Wherever possible, this system should be tied into and made part of the state or regional

* Segments 8.2.1.1 through 8.2.1.9 are based on the original chapter by Charles O. Frush in the *SME Mining Engineering Handbook* (Cummins and Given, 1973).

grid system. It is desirable to orient the coordinate grid on a true-north line and to position the origin of the coordinates so that all of the work will be in the "northeast" quadrant, making the north and east coordinates always positive. This can be done by subtracting suitable constant values from the north and east coordinates of the regional system. The residual values are of more convenient magnitude and can be used as the local mine coordinates.

If the long axis of the mineralization is not generally north-south or east-west, it may be useful to establish a secondary coordinate system oriented parallel to the long axis. This makes it possible to depict the mine workings more conveniently on the working maps.

Commonly the elevations will be based on sea level, as taken from established stations. In most cases, all the mine workings will be above sea level so that all the elevations will be positive. If this is not the case, a different reference elevation may be necessary. Frequently, this is chosen so as to be above any possible mine working, making all elevations have a negative sign. This can be ignored if the measurements are considered to be "down" from the plane instead of "up."

Most mining operations are concentrated within relatively small surface areas. Thus it is possible to ignore most of the problems introduced by the curvature of the earth and by the convergence of the meridians, except in the most precise work. A level surface is considered to be a plane, and the meridians are considered to be parallel and perpendicular to the lines of latitude. These simplifying assumptions are entirely satisfactory for compact mining operations, but may not be adequate for distances exceeding several miles. In such cases, the principles of geodesy must be invoked.

8.2.1.3 Underground Traversing

The traverse is of paramount importance in underground surveying. Since mine workings provide the only access to a given point, all lines must be carried through them. If these workings are narrow and tortuous, the traversing will be difficult and less accurate. However, modern mechanized mining commonly requires more space, making surveying operations easier. Even so, traversing is the only practical means of carrying line and grade to the headings.

Stations: Underground traverse stations should be located in the back or on the roof, behind the working faces wherever possible. This prevents their being disturbed or destroyed by current mining operations. The positions of these stations should be selected with care so as to afford convenience in setting up under them and good lines of sight in all directions. A variety of means is available for their establishment.

Plug and Spad—A short hole is drilled into the rock at the desired location. A wooden plug then is driven into the hole, with a tight fit. A hook then is inserted into the plug, from which the plumb bob is to be hung. Specially designed hooks, called spads, are available from surveying equipment suppliers. These are recommended for their durability and ease of recognition. The size of the hole and of the plug is immaterial. Stub-blast drillholes can be used, or the hole can be made with a small star drill or electric hand drill for softer rock. In any case, the spad should be oriented as nearly as possible along the line of sight.

Simple Spads—Sometimes it is necessary to set stations in timber sets. Such stations are to be avoided wherever possible because of their impermanence and the ease with which the timber can be dislodged or shifted, thus destroying the line. Where they must be used, however, the spad, driven into the wood, makes an excellent temporary station.

Sometimes the spad can be driven directly into a crack on the rock, or the rock itself, if it is sufficiently soft. Such stations are rarely permanent enough to justify their use for important points.

Power-driven Studs—Some types of rock will accept and hold permanently a threaded hardened steel stud driven into place with an explosive charge by some variety of stud gun. A threaded hook to serve as a spad then can be attached to make the permanent station. In good rock such a station will be very durable, but the studs cannot be used in hard siliceous rock or in soft crumbly material. However, they work very well in steel-lined mine openings, as the stud can be driven directly into steel webs, flanges, or liner plate to make an exceptionally permanent installation.

Adhesive Stations—The use of durable, quick-setting, adhesive plastic cements, such as the epoxies, simplifies the establishment of underground survey stations in difficult places. The surface is cleaned and dried before application of the adhesive. The spad or hook is embedded before it has hardened completely.

Clamps—Clamps to support the hook may be useful when the line is being carried through an opening supported by steel sets. This technique is particularly useful when the line must be established in a specified position. The clamps can be removed and reset as required. The design of the clamps depends on the conditions.

Station Identification: Each station should be identified by a permanent marker. This can be of metal, usually brass or plastic. If of metal, the tag should not be in contact with the spad, particularly in wet or damp locations, as galvanic action can cause accelerated corrosion that will destroy the station. A wide variety of tags in different sizes, shapes, and compositions is available. Monel metal strips marked with a portable embossing tool make excellent tags.

Various systems of nomenclature for identifying the stations have been devised and are in use. Each has certain advantages. Some of the principal systems of nomenclature are as follows.

Sequential—This system gives each succeeding station a sequential number, regardless of where it may be located. It is best adapted to one-time surveys. An index of stations, showing the name, location, and coordinates of each, should be kept.

Position-coded Sequential—This system incorporates a code related to the station position. The complexity of the code is a function of the complexity of the workings and of the need. A simple example would be "4-N-172," which could indicate the 172nd station in the north part of the 4th level. Variations can easily be devised to identify specific entries, levels, drifts, or crosscuts on each level.

Position-coded Distance—This system names each point according to its distance from a reference point in the working in which it is located. Thus "22N-327L" could indicate a point in the 22nd entry north of the starting point approximately 327 ft (99.6 m) left of the intersection with the reference line.

Each system has numerous variations, which depend on the complexity of the mine and the purpose of the survey. The method adopted should be clear and easy to learn and use.

Station Occupation and Use:

Elevations—Elevations are carried to the stations either by differential leveling or by slope-distance traversing. For stations set in the ground or floor, the elevation is considered to be that of the point on which the level rod would rest. The elevation of points set in the back (roof) is considered to be that which the inverted rod would contact—the lowest point of the hook or spad. When traversing to the point, the horizontal crosshair is set on the top of the nail or the bottom of the spad and the vertical angle measured to that point. Often it will be difficult or inconvenient to sight directly to the spad. In such cases, the crosshair may be set on some other easily identifiable point on the plumb bob string and the distance up or down to it measured. This measurement is known as the *height of station* HS. If the line of sight passes over the station, HS is considered to be positive; if it passes under, negative. (This convention is that employed by several widely used computer programs for reducing survey notes. It may have to be modified to fit others.)

Setting Up Under a Station—In setting up under an underground station, it is convenient to first hang a plumb bob from it, slightly higher than the expected top of the transit. The telescope of the transit is set to the horizontal position, perpendicular to the vertical axis of the instrument. The legs then are positioned conveniently so that each is on solid footing, and their length is adjusted to bring the head directly under the plumb bob. The plate is leveled and shifted laterally as necessary to bring the vertical axis of the instrument directly under the point of the bob. Usually, it will be necessary to readjust the length of the legs, or their position, before a satisfactory setup can be obtained. The instrument is correctly set when the plate is level and the vertical axis is directly under the point.

Occasionally, it may be necessary to set up in extremely difficult locations where the previously described technique cannot be used. In such cases, it may be necessary to replace the tripod with a bracket that can be fastened to a post, or a stull or column to achieve the requisite position for the transit. Once in position, the adjustments proceed as before.

Setting up over a station is identical to the similar operation on the surface.

The *height of instrument* HI is measured as the distance from the point to horizontal axis of the telescope. The sign is positive if the instrument is over the point, negative if it is under it.

Illumination—Typically, mine surveying must be done in the dark, the only illumination coming from the cap lamps carried by the surveyors (and any stationary lighting in the vicinity). In such cases, it is necessary to illuminate the backsights, foresights, crosshairs, and the horizontal and vertical circles.

To illuminate the crosshairs on the standard mining transit, the beam of the cap lamp is directed so that a small part of the light enters the objective lens at an angle. The internal reflections make the crosshairs easily visible. The horizontal and vertical scales are illuminated directly by the lamp, holding it for maximum clarity.

Modern instruments also use internally illuminated horizontal and vertical circles and crosshairs. Light enters through an aperture on the side, passing along an intricate path through the glass circles and the vernier through a small telescope to the eye of the observer. A small mirror can be rotated to direct part of the beam onto the crosshairs. Auxiliary bulbs, mounting on the instrument, and a battery pack can be used as the illumination source if the surveyor so desires. Some instruments now have LED displays and readouts, making accurate reading easier.

Several techniques are used for illuminated the sights:

1. The chainman's cap lamp can be held behind the junction of the plumb bob and its string, directed at an angle toward the telescope so as not to blind the observer. The instrument man uses the light as a sighting point, seeing the string and the bob as a shadow against the light. This is a preferred method.

2. The chainman holds his lamp as before, but places a translucent paper or screen between it and the string. This provides a larger, but less bright, target for the instrument man, who puts his crosshair on the shadow.

3. Various kinds of self-illuminating sights can be used. One easily fabricated type is made from an ordinary two-cell flashlight, a piece of string and a bead. A hole is drilled in the center of the flashlight lens to receive the end of the string, which is knotted in place. The bead is slipped over the string and can be slid up or down as required. In use, the flashlight illuminates the lower side of the bead, to which the measurements are made.

Double-angle Traverses: Double-angle traverses are used to extend the network of control points in mines, both surface and underground. Where greater precision is required the angle should be observed more than two times, but always an even number. Double-centered instruments commonly are used, but pointing-type theodolites often are seen.

With the conventional mining transit, the zero points of the inner and outer horizontal circles are brought into exact coincidence. The telescope, in the erect position, then is centered on the backsight. The upper motion then is released and the telescope rotated to the foresight, the crosshair being brought to exact position with the tangent screw. The angle from the backsight then is read in a clockwise direction ("angle right") and the vertical angle is recorded. With the upper motion remaining clamped to the lower, the telescope then is rotated to the inverted position and, with the lower motion loosened, again is centered on the backsight. The lower motion is clamped, the hairline set exactly with the tangent screw, the upper motion loosened, and the telescope turned to the foresight and set. The doubled angle then is read from the horizontal circle and recorded, as is the vertical circle. The distance from the horizontal axis of the telescope to the foresight point is read and recorded, as is the HI and the HS. The instrument then is ready to be moved to the foresight station, unless there are other points to be observed from the same station.

The technique with the pointing-type theodolite is basically the same, except that there is only one horizontal circle instead of two. With the telescope erect, the crosshair is set on the backsight and the horizontal scale is read and recorded. The telescope then is pointed at the foresight and the horizontal scale read and recorded. The difference is the observed horizontal angle. The vertical angle to the foresight also is read and recorded. The telescope then is inverted and the process repeated. The two horizontal angles are averaged, eliminating any slight instrumental errors, as are the vertical angles. The inclined distance to the foresight and the HI and the HS are read as before.

If the traverse is carried in a loop back to the point of beginning, it is said to be "closed." It also may be closed by terminating it on another known point of equal accuracy. Closed traverses should produce the expected coordinates and azimuths within the desired degree of precision, as determined by the care and skill used in the field work, the applied corrections and the calculations. Any acceptable discrepancies can be distributed among the various courses by any of several means. Closed traverses should be done on a regular basis (i.e., monthly) to ensure rapid error correction within any one traverse.

Traverses that do not close back on their starting point or on other known points are said to be "open," meaning that there is no automatic check on the validity of the work. Such traverses can be checked in approximate fashion by using a protractor and a scale to plot the observed angles and the horizontal distances. If the coordinates of the plotted points check the calculated coordinates, within normal plotting accuracy, it may be assumed that the calculations are essentially correct.

Azimuth Traverses: By *azimuth traverse* is meant that procedure by which the instrument always is kept oriented on map north so that the pointings to the foresights also are the forward azimuths. The angles are not doubled, so azimuth traverses are of lesser precision. However, they are very useful for controlling the advance of mine workings between periods of advancing the main network with the double-angle traverse.

The traverse is started at a point on a line of known azimuth. The horizontal circle is set on the azimuth of the line from the backsight to the occupied station. The backsight is observed with the telescope in the inverted position. The telescope then is plunged to the erect position, the upper motion is loosened, and the instrument is rotated to the foresight. The observed reading on the horizontal scale also is the forward azimuth of the line from the station to the foresight. The vertical angle is read, as are the slope distance, the HI and the HS.

The calculations in an azimuth traverse also are simplified in that the azimuth already is known.

Compass Traverses: Ordinary surface compass traverses are very similar in theory to the azimuth traverse. The compass in the transit may be used to provide orientation, or an instrument such as the Brunton compass may be used to find the bearings and vertical angles. The calculations are identical with those of the azimuth traverse. The accuracy of the method depends on the constancy of the ambient magnetic field and on the accuracy with which the instrument can be read. Unless there are local magnetic influences, the ambient magnetic field will remain sufficiently constant for this technique to be useful for many applications requiring low to moderate accuracy.

Underground, however, there are likely to be many disturbing magnetic fields. These may result from the proximity of steel pipe and rails, machinery, power lines, and perhaps magnetic minerals in the ore and country rock. The possibility of the existence of such influence restricts the utility of the simple compass traverse as described.

These extraneous magnetic influences can be canceled out if the compass is held in the same position while both backsights and foresights are taken. The difference between the foresight and the backsight is the *horizontal angle*. This can be used in the ordinary traverse calculation to carry the bearings forward and to obtain coordinates. However, the greater difficulty in obtaining exact angles, as compared with the transit survey, introduces greater uncertainty into the results.

Distance Measurement in Traversing:

Steel Tapes—The standard device for measuring distance in surface or underground traverses is the steel tape. In mine surveying, the distances commonly are measured along the line of sight from the horizontal axis of the telescope to the sighting point. This is known as “slope chaining.” The technique is preferred to horizontal chaining because of the frequent steeply inclined lines of sight and because of its higher accuracy.

The tapes used are limited to 200 ft (60 m) in length for underground use, partly because of the frequently limited lines of sight but chiefly by the problem of compensating for sag in the longer unsupported lengths. The required pull becomes excessive as the length becomes great and the accuracy of the work diminishes as errors in the applied tension increase. The preferred tape is graduated to 0.01 ft (3mm) along its entire length. The tape should be carried on a reel for safety and convenience, and should be carefully cleaned and oiled after each use, particularly if it has become wet or dirty.

Ordinary traversing work seldom requires corrections for temperature or precise control of the pull. These factors must be taken into consideration for very precise work, such as in determining the exact length of a base line for triangulation. The tapes used for such work should be checked against tapes that have been calibrated by the National Bureau of Standards.

Subtense Bar—The subtense bar is a device for the indirect measurement of distances by measuring the horizontal angle subtended by the sighting points at the ends of a horizontal bar. Knowledge of the exact length between the marks and of the angle permits the distance to be calculated as the product of the cotangent of half the subtended angle and half the bar length. A typical bar is 7 ft (2 m) long in extended position, the sighting points being fastened to invar wires under spring tension. A theodolite is used to measure the subtended angles to the nearest second.

The accuracy of the determination is very high for intermediate distances up to perhaps 400 ft (120 m) per reading, but diminishes with increasing distance. The bar has the advantage of making it possible to determine the length of an inaccessible course and permits the measurement to be done by an individual, should this be necessary. With it, a theodolite, three tripods, and a set of targets, one person could run a traverse by himself, although this is not desirable.

The subtense bar is not widely used underground, but could be employed to advantage where the openings are wide enough to accommodate the bar and straight enough to provide a convenient length of sight.

The distances obtained with the subtense bar are horizontal distances, since the subtended angle is measured in the horizontal plane.

Electronic Distance Measuring Devices—The development and widespread availability of electronic devices for the direct measurement of distance has evolved rapidly (8.2.1.10). The apparatus best adapted to mine surveying problems is light in weight, easily portable, utilizes a beam of infrared or visible light reflected to the transmitter by a suitable reflector, and produces a distance reading directly without the need for data conversion by the operator. The equipment emits a narrow carrier beam, modulated at a succession of frequencies. Comparison of the phase difference of the outgoing and returning signals provides the basis for the distance determination. Corrections for air density and temperature and index constants for the transmitter and reflector must be supplied by the user. The equipment is made to be mounted on the same tribrachs or mountings used by theodolites, so that the horizontal and vertical angles can be determined from the same setup.

For short distances, up to perhaps several hundred feet (tens of meters), the reflectors can consist of simple reflecting tape or inexpensive plastic reflectors. These are useful in monitoring the position of inaccessible walls or back, as in large underground rooms.

For longer lines, it is necessary to use corner prisms set up over or under the desired point. These prisms have the property of reflecting a light beam back along the incident path, no matter what the angle of incidence may be within the acceptable range.

The distance measured is the inclined point-to-point distance. The error is a fixed quantity, essentially unrelated to the distance, except as the temperature and density of the intervening air may vary. The height of transmitter and height of reflector are essential data in work involving elevations.

The greatest application of electronic distance measuring devices is in surface work, where their long range and high accuracy make it possible to carry traverses rapidly over long distances. They make trilateration possible as a substitute for, or an adjunct to triangulation.

In underground surveying, it appears that their greatest application will be in measuring the movement of rock masses on the periphery of large excavations, and for traversing in mines where long lines of sight are involved.

Detail: In addition to establishing the point-to-point traverse, the surveyor frequently must locate the boundaries of the mine workings, the position of important structures, and other detail. Methods for doing so include the following.

Angle and Distance Measurements—The instrument is set up under or over a station and backsighted on a suitable neighboring station. The upper motion is released and the instrument pointed at successive significant points, to which slope distances are measured and recorded, as are the horizontal and vertical angles. The accuracy required is a function of the purpose of the work and of the scale to which the data is to be plotted. For most work, it is sufficient to read the horizontal angle to the nearest 15′ and the distance to the nearest 0.1 ft (30 mm). If elevations are not required, it often is possible to measure horizontally and thus ignore the vertical angle.

Ordinary detail taken in this fashion is best plotted by use of a protractor and scale. The more important points can be treated as points on the traverse, the coordinates computed, and the points plotted by the coordinates. This type of plotting is easily done when the coordinates are calculated by use of the digital computer.

Lefts-and-Rights—This technique requires that a tape be stretched from the instrument station to the foresight. Significant points are located by measuring the distance perpendicular to the tape and noting the distance along the tape at which the measurement is made. The technique is best adapted to rather narrow workings, as otherwise failure to measure perpendicularly may result in noticeable errors in the resultant plot. Measurements of this type usually are made at waist level for convenience and because this corresponds to the usual maximum width of the drift.

Plotting is done by laying a scale along the indicated line and marking off the taped distances. The scale then is held perpendicularly to the line and the corresponding widths plotted.

Stadia—The use of the stadia board as a means of obtaining detail is not appropriate to narrow workings, particularly if of a rounded cross section. However, it is useful in the larger workings, such as stopes and rooms where it is necessary to obtain not only the periphery, but also numerous points on the back and on the floor. When used, the length of the stadia board must be chosen to fit within the available vertical height. The board is held vertically over or under the point, the horizontal crosshair of the transit brought to a convenient number, usually the HI, and the vertical angle read. The lower crosshair then is moved slightly to an even marking or fraction and the intercept of the upper crosshair noted. The difference gives the stadia interval. The horizontal angle, vertical angle, and stadia interval are noted as the basis for subsequent reduction to horizontal distance, vertical distance, and elevation. This information then may be plotted with protractor and scale. If coordinates are desired, it is better to process the raw information with a suitable computer program, regarding the point as a point on the traverse, but the same calculation can easily be done by hand if necessary.

Range Finder—The range finder is not widely used in underground mines, but under the proper conditions, it is a very useful instrument. Its chief application is in obtaining cross sections of large stopes or rooms.

The range-finder instrument is similar in many respects to those used by the military. It is binocular in character, each line of sight being reflected to a prism or mirror at opposite ends of a horizontal tube and from there to the target. The mechanical adjustment required to bring the images into coincidence is used to provide a numerical indication of the distance.

In use, the range finder is positioned at the desired point along a line, backsighted, and the proper horizontal angle turned to obtain the desired orientation of the vertical plane passing perpendicularly through the center of the tube. A point of light is directed onto the stope wall, upon which the range finder is focused. The inclined distance is read directly from the instrument, as is the vertical angle. This information is used to plot the profile of the stope or room on the cross section with a protractor and a scale.

Steeply Inclined Lines: Occasionally traverse lines must be carried through openings so steeply inclined that the telescope cannot be raised or depressed enough to see the sighting point. Or perhaps it may be necessary to position the transit in a specific location where some obstruction interferes with the line of sight. In such cases, auxiliary equipment must be used, the type depending on the instrument used.

Side Telescope—The side telescope is used with the conventional mining transit, which is equipped with projecting threads at each end of the horizontal axis to receive the side telescope and its counterweight. The side telescope clears the base of the transit so that sights of any inclination are readily made. It is similar in construction to the main telescope, and its crosshairs are adjustable in the same manner. It is essential only that the horizontal hair remain on a point as the instrument is swung from side to side. A vertical adjustment permits the horizontal hair to be made coincident with the horizontal hair of the main telescope. If the situation permits both erect and inverted observations, the averages of the horizontal and vertical angles are the correct values, and no other calculations are needed. Minor maladjustments of the instrument also are compensated for in this case. If direct and inverted observations cannot be made, the instrument must be in excellent adjustment and the effects of convergence and offset of the line-of-sight must be eliminated from the observed horizontal angles through the application of calculated corrections.

Top Telescope—Most mining transits are equipped to use the auxiliary telescope as either a side or a top telescope. Another set of projecting threads extends above and below the main telescope for the attachment of the auxiliary telescope and the counterweight. The adjusting screws readily permit the vertical crosshair to be made coincident with the vertical crosshair of the main telescope. The top telescope can be used in the erect position to see past the instrument base, but not in the inverted one. Therefore, the angles cannot be doubled in the usual way and the effects of minor maladjustments in the instrument cannot be eliminated by manipulation. The effects of offset and convergence of the line-of-sight must be eliminated from the observed vertical angles by the application of calculated corrections.

Pentaprism—The widely used modern theodolites are not designed to permit use of the auxiliary telescope. The same function is served by the use of a pentaprism clamped to the objective end of the telescope, balanced by a counterweight at the eyepiece eye. The pentaprism is a five-sided prism, so cut that an emerging ray of light is offset exactly 90º from the entering ray. The prism is mounted in a tube which permits 360º of rotation about the telescope axis. In use, it is rotated until the vertical crosshair stays on a point as the telescope is rotated about its horizontal axis. The bent line of sight permits the user to see points that normally would be obscured by the base of the instrument. The telescope can be used in the erect and inverted positions by rotating the prism 180°. This has no effect on the angle of bending since the prism bends the light rays 90°, regardless of the direction in which it is pointed. The horizontal angles so measured are correct, but the vertical angles must be corrected for the 90º bend and for the offset.

Side Telescope Calculations: The line of sight through the side telescope will lie in a plane with the line through the main telescope but offset from it, and the lines may converge or diverge. If the averages of the erect and inverted sightings cannot be determined, it will become necessary to find the amount of offset from the line through the main telescope at the instrument and the amount and direction of convergence.

This is done by setting the instrument over a point and reading the intersections of the two vertical crosshairs on scales placed at known distances from the instrument. The offset at the instrument and the divergence then can be determined through the use of similar triangles.

Example 8.2.1.1. Assume that the following data were obtained:

Distance from instrument to scale, in.	Scale reading, main telescope, in.	Scale reading, side telescope, in.
99.86	4.01	8.56
198.57	1.53	6.59

Let x be the offset. Then

$$\frac{(6.59 - 1.53 - x)}{198.57} = \frac{(8.56 - 4.01 - x)}{99.86}$$

from which $x = 4.034$ in. or 0.336 ft offset.
Let alpha be the angle of divergence. Then

$$\text{Alpha} = \arctan \frac{(6.59 - 1.53) - (8.56 - 4.01)}{(12)(198.57 - 99.86)}$$

$$= \arctan(0.0004305541)$$

From which alpha = 0° 01′ 28.8″, angle of divergence.
(Conversion factor: 1 ft = 12 in. = 0.3048 m)

If the side telescope is used for the foresight, with the instrument erect and the auxiliary telescope on the right, then the observed angle will be too small by the sum of the angle subtended by the offset and the angle of divergence. The angle subtended by the offset is computed as the arctangent of the offset divided by the horizontal distance from the instrument to the point. If the instrument were inverted the observed angle would be too great by this sum.

Similarly, if the side telescope in the erect position is used for the backsight, with the main telescope used for the foresight, the observed horizontal angle is too large by this sum.

The correct horizontal angle must be determined before calculating the coordinates of the foresight.

Top Telescope Corrections—Theoffset and the angle of divergence for the top telescope are determined in similar fashion, but using the horizontal crosshairs of the two telescopes. The calculations are similar to those of the side telescope, except that the slope distance is used in determining the angle subtended by the offset. Up angles must be increased by the sum of the angle of divergence and the angle subtended by the offset. Down angles must be diminished by this sum.

Pentaprism Calculations—The determination of the offset and the divergence, if any, for the pentaprism is similar in principle to the technique used for the side telescope but is slightly more difficult to perform. Set the instrument up on level ground, and two other tripods on line, with the tripod heads as nearly as possible on a level line passing through the telescope, the nearer tripod being slightly lower than the farther one. Place a scale on each tripod, level and perpendicular to the line of sight. Measure the distance from the instrument to each scale. Turn the telescope at right angles to this line, attach the pentaprism and level the telescope. Rotate the prism so that the intercept of the vertical crosshair and the horizontal crosshair touch the top edge of the nearer scale. Read the scale. Repeat on the more distant scale. Next invert the telescope and relevel it. Make a similar set of readings. The calculations required to obtain the divergence and the offset are illustrated with these data.

Example 8.2.1.2. See Fig. 8.2.1.1.

OA = 15 ft	D = 10.97″	F = 10.14″
OB = 31 ft	C = 1.20″	E = 0.38″
	9.77	9.76

Diff. = 0.01″

Let alpha be the angle of divergence between the true line and the bent line.

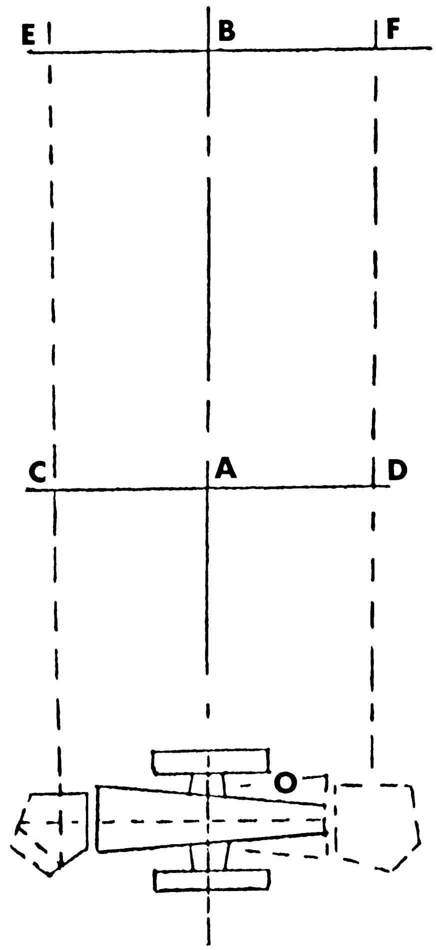

Fig. 8.2.1.1 Divergence and offset-calculations (see Example 8.2.1.1).

$$\text{Tan alpha} = \frac{-0.001}{(12(31 - 15)(2))} = -0.00002604$$

Therefore, alpha is 0° 0′ 05.4″converging.
Let x be the offset:

$$\frac{(9.76 - 9.77)}{(31 - 15)} = \frac{(9.77 - x)}{(15 - 0)}$$

Therefore, $x = 4.89$ in.
(Conversion factor: 1 ft = 12 in. = 0.3048 m)

If the line of sight in the field work is to a point lower than the instrument it will be necessary to add 90º for the bend and to subtract the sum of the divergence and the angle subtended by the offset. If the point is above the instrument it will be necessary to subtract 90º for the bend and to add the sum of the divergence and the angle subtended by the offset. The angle subtended by the offset is the arctangent of the offset divided by the inclined distance.

Calculation of Traverses: Traverse calculations, like most other mine surveying computations, are most conveniently done in the mine engineering office where calculating facilities are readily available. The equipment for such an office is described under a different heading. Such calculations as must be done in the field are greatly facilitated by the use of a pocket electronic calculator that generates trigonometric functions and utilizes several levels of storage registers.

All traverse calculations should be carefully made and recorded. The use of conventional formats for all manual calculations will help minimize errors and will facilitate checking.

Manual Reduction of Data—The averaged values of the field observations are copied into the calculation book or onto the calculation page. Care should be taken to put the data into the proper sequence, so that the foresight of the first line becomes the station of the next. If several foresights were taken from the same station, care must be taken to ensure that the right station coordinates are used in determining the coordinates of each foresight point, and that the right coordinates are used in carrying the traverse forward. It is very helpful to leave at least three blank lines between each line of data to facilitate the calculation of the azimuths, as shown in Table 8.2.1.1.

The first line of data pertains to the starting station. The azimuth shown is that from the backsight to the station and the coordinates are those of the station itself. The remaining spaces are not used. The subsequent lines are those of the successive traverse courses.

The azimuth is carried forward by adding the angle right to the previous azimuth and subtracting 180° from the sum. This compensates for the fact that the azimuth of the line from the station to the backsight is 180° different from that of the line from the backsight to the station as indicated on the previous line. If the difference is greater than 360°, subtract 360°. If the sum of the previous azimuth and the angle right is less than 180°, add 360° and then subtract the 180°, or simply add 180° to the sum. The result is the new azimuth.

If the new azimuth is greater than 9° and 90° or less, the forward line is in the northeast quadrant, so that the sine and the cosine of the bearing both have a positive sign. If the sum is between 90° and 180°, the forward line is in the southeast quadrant. The sine of the bearing will be positive, the cosine, negative. If the sum is between 180° and 270°, the forward line will be in the southwest quadrant so that both the sine and the cosine are negative. If the sum is between 270° and 360°, the forward line will be in the northwest quadrant. The sine will be positive, the cosine negative. Since the cosine of the bearing is used to determine the latitude and the sine is used to determine the departure, it is very important to determine and apply the correct sign to each.

The horizontal distance is computed by multiplying the slope distance by the cosine of the vertical angle; the vertical distance is obtained by multiplying the slope distance by the sine of the vertical angle. The cosine will always have a positive sign. The sine will be positive if the forward line is inclined upward, negative if inclined downward from the horizontal. However, if theodolites are used that have 0° at the zenith and 90° at the horizontal, it is more convenient to use the vertical angles as recorded but to multiply the slope distance by the sine of the vertical angle to get the horizontal distance and by the cosine to get the vertical distance. Angles greater than 90° have negative cosines, but the sine remains positive to 180°.

Multiplying the horizontal distance, which is always positive, by the cosine of the bearing, with the proper sign, gives the latitude of the foresight point. Multiplying by the sine gives the departure. Careful attention to the sign of each is required.

Adding, with due attention to sign, the latitude to the north coordinate of the station will produce the north coordinate of the foresight. Similarly, adding, with corresponding attention to sign, the departure to the east coordinate of the station will give the east coordinate of the foresight point.

The elevation of the foresight point is obtained by taking the elevation of the station, adding the height of instrument, adding (with due attention to sign) the vertical distance, and subtracting the height of the station. This convention works equally well on surface and underground points if it be remembered that the HI and HS are considered to be positive if the line of sight passes over them, negative if it passes under.

It is very desirable to record the trigonometric functions obtained by calculator or standard tables. This saves much time in subsequent checking of the calculations. Clarity is achieved and space is saved by writing these values on the calculation form immediately over the corresponding product, as shown in the example in Table 8.2.1.1.

Computer Reduction of Data—Much time and money can be saved and many errors avoided if the traverse calculations are done with the aid of a computer or with a programmable calculator. Greater accuracy will also be achieved. The ideal situation is for the surveyor to have access to a time-sharing terminal as this not only limits errors to those associated with typing in the data, but also permits immediate return of the results. The cost of time-sharing service is small and is trivial compared to the benefits obtained for any but the smallest mines, or those having poor telephone service.

Many of the programs needed by the mine surveyor can be obtained readily from the companies supplying the service. Others are available from the literature, the US Bureau of Mines, or other agencies. Often the user will wish to modify them somewhat, or devise his own, to better meet his particular needs.

Adjustment of Closed Traverses—The strongest type of closed traverse is one that ties in to one or more known stations and closes back on its point of origin. The tie into the known stations confirms that the azimuths and the quality of the closure into these and into the initial station indicates the quality of the work. The traverse that closed only on itself is almost as strong, but the one that closes only on previously accepted points is no stronger than the quality of the work by which the coordinates of such points were established. High-quality work requires meticulous care in closing the traverse and in eliminating the minor discrepancies that creep in in spite of the best of care in the field work and the calculations.

When closing a traverse on the initial point, it is necessary to occupy the initial point again and to turn the angle back to the original backsight. This provides a check on the quality of the angles-right that were observed. The closing error should be less than the product of the inherent error to be expected in each angle multiplied by the square root of the number of angles involved. In this case, the closing error may be distributed equally among all the angles. If the closing error is greater than this, but could still be acceptable, then it may be possible that it can be attributed to one or two difficult setups. The surveyor may have justification for placing all or most of the correction in these angles, but even so there may be uncertainty as to the quality of this judgment and its effects on the resulting coordinates of all the stations in the loop.

This uncertainty can be minimized, and the cost of correcting obvious errors can be reduced if the outgoing and returning lines of the traverse are made to cross at one or more common stations. It is usually preferable not to have the outgoing and returning lines have all common points, however, as this may result in the making and accepting of identical errors which would not be detected. The loops thus created can be considered as smaller closed traverses and any errors frequently can be localized in one or more loops, in which case additional field work may be indicated.

When the angles have been suitably adjusted so that no closure error remains, the traverse is then calculated. If this is done manually, the work is carried through the determination of the latitudes and departures before further adjustment can

Table 8.2.1.1. Averaged Values of Field Observations Copied Onto Calculations Page

Table 8.2.1.1 Averaged Values of Field Observations Copied Onto Calculation Page

Traverse in A-Left Drift, Experimental Mine, made on July 15, .

BS	STA	FS	HOR. ANGLE	SL. DIS.	VER. ANG.	HI	HS	AZIMUTH	BEARING	HOR. DIS.	VER. DIS.	LATITUDE	DEPARTURE	NORTH	EAST	ELEV.
	M800	A-00						347-16-30						3617.24	8111.67	7624.18
M800	A-00	A191	89-17-30	191.27	+01-10-00	-1.87	-2.00	89-17-30 436-34-00 -180-00-00 256-34-00	256-34-00 -180-00-00 S76-34-00W	0.999793 191.23	0.02361 +4.52	-0.23231380 -44.42	-0.97257327 -185.99	3572.82	7925.63	7628.83
A00	A191	A306	182-51-00	115.97	+00-57-00	-2.15	-2.56	182-57-00 449-25-00 -180-00-00 269-25-00	269-25-00 -180-00-00 S89-25-00W	0.999863 115.95	0.016580 +1.92	-0.01018091 -1.18	-0.99994817 -115.94	3571.64	7809.74	7631.16
A191	A306	A381	232-11-30	75.03	-1-17-30	-2.18	0.00	232-11-30 501-36-30 -180-00-00 321-36-30	359-59-60 -321-36-30 N38-23-30W	0.999746 75.10	-0.022542 -1.69	+0.78378379 +58.86	-0.62103379 -46.64	3630.50	7763.10	7627.29

begin. For a traverse closing back on its initial point, the sum of the latitudes should be zero and the sum of the departures zero. The linear discrepancy is calculated as the square root of the sum of the squares of the latitude and departure discrepancies. If the ratio of this length to the total distance traveled is smaller than that ratio that has previously been determined to be acceptable, it is appropriate to distribute the discrepancies among the various lines in proportion to their length, remembering that the incremental discrepancy corrections must all be applied in the same direction. The correction in latitude or departure for each side is calculated as:

$$\text{Correction} = \frac{\text{closure in latitude or departure}}{\text{total length of traverse}} \times \text{length of side} \tag{8.2.1.1}$$

This approach is known as the *compass rule* and is the one most commonly used.

8.2.1.4 Transferring Meridian Underground

All the underground workings of a mine must be referenced with respect to the same meridian and coordinates. This is easily done if the mine is serviced with a slope so that a traverse can be carried to each level. If it is not, other methods must be used.

Two-Shaft Method: This technique is useful when the level is accessed by two vertical shafts or raises straight enough to allow one wire to be hung in each without contact with the sides or fixtures. Piano wire is ideal for this purpose, as it combines great strength with small diameter. A heavy bob is attached to each wire. The bob may be immersed in water or oil to minimize oscillation of the wire. Care must be taken to insure that the wires hang free and that they are not excessively disturbed by air currents or falling water.

A traverse is then run to each wire from known points on the surface or on adjacent levels and elevations carried to a fixed point on or immediately adjacent to the wire. The coordinates of each wire are obtained and, by calculation, the length and azimuth of the line connecting them.

A convenient permanent station is then established on the new level, as is a permanent backsight. From this station is run a traverse to each of the wires. Arbitrary coordinates are assigned to the station, and an azimuth is assigned to the line from the backsight to the first instrument station. The coordinates of each wire are then calculated, and the bearing and distance of the line between them. The distance should be the same as previously determined, but the bearing of the calculated line will be off. The amount and direction of the discrepancy is determined, and the correction is applied to the azimuth of the assumed backsight line, making it now correct. The traverse is recalculated on the basis of the new bearing, obtaining new coordinates for the wires.

These new coordinates still differ from the true coordinates. If all the work has been well done, the differences will be the same for both wires. If they are not, the work must be redone correctly. These differences are used to correct the assumed coordinates of the initial station on the new level, after which the traverses again are recalculated. The coordinates thus obtained for each wire are identical with the true coordinates, and the coordinates of all the traverse points will be correct with respect to the mine coordinate system.

The elevation for the new level is obtained by careful taping down the vertical shaft.

Shaft Plumbing: When only one vertical shaft is available, the meridian must be carried through it. This is much more difficult because of the much shorter base line involved. Correspondingly greater care must be used. For important work it will be necessary to repeat the work a number of times to obtain the necessary precision.

Hanging the Wires—The wires should be piano wire of small diameter and should be kept on small reels for safe, easy handling. It is convenient to have the wires pass through a movable notch or aperture for easy exact positioning of the upper end. Heavy bobs are attached to the lower end to stretch the wire close to its elastic limit. These bobs may be immersed in water or oil to damp out any tendency to swing pendulum-fashion and thus bring the wire to rest more quickly. Vibration or oscillation of the wires about different nodes and planes will continue. Heavy bobs reduce the resulting difficulty of observation.

Care must be taken to insure that each wire hangs free in the shaft, touching neither the wall nor any fixture. Efforts should also be made to minimize the effect of falling water and strong air currents. If the surrounding rock contains large amounts of magnetic minerals, it may be necessary to use bronze wires to prevent attractive forces that could displace the wires slightly and warp the plane.

For deep shafts, those in excess of 3000 ft (900 m), the effect of other forces in distorting the plane may become noticeable. Gravitational attraction between the bobs and nearby masses or voids becomes increasingly important with depth because the increasing length of the wire reduces the restoring effect of gravity for a given lateral attraction. However, this problem has been eliminated by the introduction of the modern gyro-theodolite, as have most other problems associated with shaft plumbing if this equipment can be used.

Coplaning—This technique is used to put the transit and both wires into the same vertical plane. The transit is set up fairly close to the nearest wire and focused on the more distant one. Then, without turning the telescope on its vertical axis, the scope is focused on the nearer wire. The offset from the crosshair will show the operator which direction to shift the instrument to put the wire on the hair. The instrument is shifted bodily on its plate, releveled, and the process repeated until both wires coincide with the vertical crosshair. No difficulty will be experienced by the near wire obscuring the far one since the near wire will not be visible when the telescope is focused on the far one.

The point where the vertical axis of the instrument intersects the ground is then marked at the horizontal angle from a traverse station used as a backsight to the plane of the wires, is read and recorded, as are the distances to each wire and the vertical angles. The transit is then moved to the backsight station and the traverse carried forward to the temporary station just vacated.

The coplaning process is most easily done on the level from which the wires are suspended, as there is virtually no swing or vibration to interfere with the observations. Further down the shaft, however, difficulty will be experienced in bringing the wires to rest, particularly in the presence of falling water or air currents. Under such conditions it is better to use the Weisbach technique.

Weisbach Triangulation—This technique requires that the transit be set up close to, but not on, the plane of the wires, and as near the closer wire as conveniently possible. The angle subtended by the two wires should be less than 1º. The mean position of each wire must be established if they are moving or vibrating and the horizontal angle between these mean positions measured. The distances between the wires and from each wire to the transit are measured to the nearest thousandth of a foot (three-tenths of a millimeter), but accuracy to within 0.01 ft (3 mm) is sufficient for all normal work. The intersection of the vertical axis of the instrument with the ground surface is carefully marked.

With this information, the angle subtended at the far wire by the near wire and the transit can be calculated by the law of sines. In the range from 0° to 1°, the sine is a linear function of the angle, to 8 decimal places. Thus one may substitute the angle for the sine and solve for the unknown angle directly. If one knows the azimuth of the plane of the wires, it is a simple matter to obtain the azimuth of the line from the far wire to the transit. Knowing the coordinates of the far wire and the horizontal distance to the transit, the coordinates of the temporary station occupied by the transit are easily determined. The traverse from the temporary station to two permanent stations on the level is then carried forward and the task of establishing the meridian and the coordinate system on the new level is done.

The precision required in this work is a function of the accuracy demanded. For short levels, it may be sufficient to estimate the mean position of the wires, particularly if they are moving only over a short range, and to double the angles. The greater the distance between the wires, the more satisfactory this will be. For higher accuracy or for carrying long lines, greater care will be required. For maximum accuracy with the ordinary mining transit, both the interior and the exterior angles subtended by the wires should be read six times, three times erect, three times inverted; and the mean position of the wire must be determined much more carefully.

This can be done by placing a scale behind each wire, perpendicular to each line of sight. For very precise work, there should be another scale parallel to each line of sight. The observer notes a series of extreme positions for each wire, on both sides of the mean. The average is calculated and this average scale reading used for subsequent pointings of the telescope for the angular observations.

The work can be checked by moving the transit to the other side of the plane of the wires and repeating the work.

Triangulation—Sometimes it is impossible either to coplane or to use the Weisbach technique because of space or other limitations. In these cases, it is necessary to use triangulation to carry the azimuth forward from the plane of the wires. The transit is set up at a convenient point such that the triangle formed by the transit and the two wires is approximately equilateral. The distances between the wires and between each wire and the transit are carefully measured and recorded. The mean position of each wire is determined, and the angle subtended at the transit by the two wires is measured. Knowing the three distances, the cosine law can be used to find each of the angles of the triangle, and the observed angle can be compared with the calculated one as a means of estimating the quality of the work. The traverse can be carried forward from the wires through the temporary point to the two or more permanent points on the level as before, thus satisfying the requirements.

The triangulation technique is considered inferior to the Weisbach technique because of the greater inherent errors arising from difficulty in obtaining accurate distances and the role these distances play in the calculations. However, its accuracy can be improved by establishing a quadrangle, of which the wires form two of the corners, and the other two are instrument positions, permanently located. At each station the angles between the wires and between the wires and the other station are measured. The distance between the two stations is also obtained. This information, plus the known distance between the wires, allows the various unknown angles and distances to be calculated and checked. From this, the azimuth of the line between the two setup points can be determined accurately, as can their coordinates.

Use of Multiple Wires—More than two wires may be hung in the shaft. The measurements between them at depth can be used to confirm the fact that none are displaced by contact with the walls or timber. Additional planes of known azimuth are established from which the meridian can be taken and checked by methods already described and by others.

Gyro-Theodolite: The gyro-theodolite is a revolutionary instrument that simplifies the task of carrying azimuth into underground mines. It is a lightweight, self-contained apparatus giving results of great accuracy in a short time. It does not require the use of a shaft, nor does it interfere with normal mine operations if there is an unused heading of sufficient length to a backsight line. It is operated by one instrument person and a recorder. Similar units are supplied by several manufacturers.

The basic unit consists of a very precise gyroscope suspended by a short thin metallic band. The gyro is housed in a metal case that mounts on top of a theodolite. Power is supplied by a portable battery activating a converter supplying alternating current to the gyro motor. The position of the gyro is observed through an illuminated eyepiece. The gyro is clamped in position while being moved and brought up to speed.

When the rapidly revolving gyro is uncaged, its axis is horizontal and pointed toward some particular spot on the celestial sphere. The earth on which the tripod stands, however, is revolving. This, with gravity, produces a force on the gyro, to which it reacts by swinging its north end toward north. The momentum of the gyro causes it to overswing and thus to oscillate about the astronomical north line. The techniques of observation vary somewhat among the different types of instruments, but the basic approach is to find the mean position of the swing from a series of observations.

The physics of the gyro do not permit the theodolite to be used in the erect and inverted positions in order to average out any maladjustments. Consequently, these must be determined by setting up the instrument on a line of known azimuth, obtaining the angle from that line to astronomical north as indicated by the gyro, and using the difference as the correction factor. This work is commonly done on the surface before taking the equipment underground. It is important that this calibration be done within 200 to 300 ft (60 to 90 m) east or west of the point where it is to be used if calculations for the convergence of the meridians are to be avoided.

Mine coordinate systems are commonly regarded as exiting on plane surfaces, with the north-south and east-west lines being everywhere straight and at right angles. In reality, this is not true, except close to the equator. The convergence of the meridians in the temperate latitudes is perceptible to the gyro along east-west lines exceeding several hundred feet (meters) and must be compensated for either by manipulation or by calculation if the full available precision of the unit is to be achieved.

Having determined the correction constant to be used, the surveyor takes the equipment underground to the place where the azimuth is to be determined. Setting up under one permanent point he backsights another, preferably several hundred feet (meters) away. He then brings the gyro up to speed, uncages it, and proceeds to find the exact angle between his line and true north. Applying his correction, he now has the true azimuth of his fixed line. He will normally repeat the operation from the other end as a check.

It must always be remembered that when the gyro is revolving at full speed, it has stored in it a very large amount of kinetic energy. If the theodolite is revolved rapidly beyond the limits of free travel, a force will be exerted on the gyro that will cause its axis to rise out of the horizontal plane. Serious damage to the equipment may result. In extreme cases, there may even be some hazard for the operator. Therefore, the greatest of care is required. In spite of this, however, the gyro-theodolite is a rugged instrument and, when not in operation, can be handled like any other surveying instrument.

Coordinates can be brought down to the new level by means of a single wire hung in the shaft. The elevation is brought down by vertical taping. Small oscillations of the wire will not result in any error affecting normal mining operations. If greater precision is required, however, the laser can be used to establish a fixed vertical line.

Use of the Laser in Shaft Plumbing: Modern surveying in both mining and tunneling makes extensive use of the laser to extend the line of sight (8.2.1.10). In shaft plumbing, it is useful in transferring the meridian.

The helium-neon laser is a device which emits a beam of coherent red light. With simple optics, this beam can be narrowed into a slim pencil of intense light with only minor divergence. Such a beam is capable of traveling thousands of feet (meters) without loss of usable intensity. Further, commonly used laser units emit light beams containing only a few milliwatts of energy; these beams are too weak to cause permanent injury to any one looking directly into it momentarily.

If such a unit is mounted so that it projects its beam straight down a shaft onto a mirror of liquid mercury having an undisturbed surface, and the beam returns to the point of origin, it is apparent that such a beam will be truly vertical, being perpendicular to a level surface. Two such beams would define a vertical plane. By measuring to one of the beams at the surface and again underground, the coordinates could be transferred to any depth within its range.

Problems arise from density variations in the air, falling water disturbing the mercury surface, and, conceivably, from unequal masses near the mercury mirrors producing slight differences in the level surfaces. Also, it is difficult to define a sufficiently narrow point of light or shadow to define the plane accurately.

Air density variations, due to moving air currents and changing temperature and humidities, produce wavering of the light beam. Any disturbance of the mercury surface produces excessive wavering. Techniques for the successful use of laser surveying have been developed and are available from the manufacturer.

8.2.1.5 Triangulation and Trilateration

Triangulation and trilateration are surface surveying techniques for extending precise control over large areas at relatively low cost. Typical mining applications include the establishing of primary control points over the mining area and local control points within the operating areas. This control is in the form of a network of points characterized by triangles and quadrilaterals.

Triangulation is much the older of these techniques, having become feasible with the development of accurate optical instruments for the measurement of horizontal angles. Trilateration did not become a practical field technique for the measurement of large triangles until the fairly recent advent of electronic devices capable of very accurate measurement of line-of-sight distances. It is now becoming widely used.

Triangulation: The first step in the creation of a triangulation network is to establish a baseline of precisely known length and orientation, the ends of which are permanent stations in the network. Often permanent stations established by the government geodetic survey agency can be used; otherwise, it is necessary to establish a new base line, measuring the distance between the ends with very great care. Other permanent stations then are established to the sides in convenient positions to build up the desired network of triangles. The triangles should have no angles less than approximately 30° or larger than 120°. If possible the triangles should form quadrilaterals, the corners of which are all mutually intervisible, since a network of quadrilaterals is much stronger than a network consisting only of triangles.

Each corner of the triangle or quadrilateral is then occupied in succession for the measurement of the angles. If at all possible a modern theodolite should be used, graduated to seconds, as this produces much better work at lower cost than does the ordinary transit. If the transit must be used, it is necessary to increase the precision of the measurement by repetition. Six repetitions will determine the angle to ± 10″; 12 will reduce the uncertainty to ± 5″, at the expense of a much longer time for observation. The nature of the job will determine how much precision is required. Most mine surveys will not require or benefit from angular precision greater than 1″, which is readily obtainable with the theodolite.

It is always desirable to measure both the interior and the exterior angles, the sum of which should be 360°, plus or minus no more than the inherent uncertainty in the measurement of each angle multiplied by the square root of the number of angles. This test should be made before leaving the station.

Calculations—Each triangle should be solved using the following steps:

1. Balance the quadrilaterals to comply with the geometric requirements. Each can be divided into four overlapping triangles consisting of two quad sides and a diagonal. The interior angles of each triangle must total 180°. The diagonals cross each other, creating equal opposite angles. Thus, the sums of the remaining angles in the smaller triangles so created must also be equal. Lastly, the sum of all the interior angles of the quadrilateral must be 360°. Adjustment of the angles to meet these conditions does not necessarily produce exactly correct angles, but if the theodolite was used for the initial measurements, the discrepancies will be trivial for most applications.

2. Using the corrected angles, work out the azimuth of each of the lines, starting from the known line, going around each triangle, and coming back to the known line with no discrepancy.

3. Use the sine law to find the lengths of the unknown sides, starting with the known side:

$$\frac{\text{Sine A}}{\text{Side a}} = \frac{\text{Sine B}}{\text{Side b}} = \frac{\text{Sine C}}{\text{Side c}} \tag{8.2.1.2}$$

where side a is opposite angle A, b opposite angle B, etc.

4. Compute the latitude and departure of each line from its length and azimuth.

5. Determine the coordinates of the unknown point by adding the latitude to the north coordinate of the known point, the departure to the east coordinate. The coordinates of the unknown point should be determined twice, using each side line. If the work has been well done, using eight-place trigonometric tables, the coordinates thus determined should ordinarily check within about 0.02 ft (6 mm).

6. If a quadrilateral is involved an additional check is available. Compute the coordinates of each unknown corner, using a triangle based on the known line. Then compute the coordinates of one of the corners from the coordinates of a known station and the computed coordinates of the other unknown station. If these check, it is proof that the field work and the calculations were all well done.

If the project is so extensive or requires such precision that the foregoing steps are not sufficiently precise, the user should use the more elaborate methods of balancing described in texts on plane surveying and geodesy.

Trilateration: Trilateration, as the term is used here, requires the use of a properly calibrated electronic distance-measuring instrument and accessories. This will commonly involve a laser or infrared carrier-beam transmitting unit and one or more cube-corner reflecting prisms as a means of determining vertical angles accurately. The transmitting unit and reflectors are used with standard surveying tripods and tribrachs interchangeably with theodolites and targets. Time will be saved if several reflecting units and tripods are available so that one can be set on each point to be observed from a given station. Two-way radio transmitters are a most useful accessory for rapid communication.

In practice the transmitter and the reflectors are set up and centered over the points. The height of the emitted beam and of the reflecting point should both be read and the vertical angle determined. Using an altimeter, the elevation of the transmitter station should be obtained, as well as the air temperature, and corresponding adjustments made to the equipment. The transmitter is then pointed at the reflector and adjusted for maximum return signal. The distance is then measured by proper manipulation of the controls. This measured inclined distance is corrected by application of the calibration constants for the emitter and the reflector. Each line should be measured several times and the average used.

Calculations—The coordinates of each point are found as follows:

1. Reduce the inclined distances to the corresponding horizontal distances using the proper vertical angle.
2. Use the cosine law to solve for each of the internal angles of each triangle.
3. Work out the azimuth of each line and the latitude and departure of the unknown points.
4. Add a latitude and departure of each line to the north and east coordinates of the known end to get the coordinates of the unknown point. Each unknown should be computed from at least two known points and the coordinates should check closely.

Trigonometric Leveling: With suitable care the levels of the new stations can be approximated quite closely by trigonometric leveling. The vertical angles forward and backward from each station to the others in a triangle are measured with the theodolite. The height of instrument is measured each time, as is the height of the sighting point. The difference in vertical elevation between the instrument and the sighting point is determined from the inclined or horizontal distance and the vertical angle. From this and the HI and HS measurements the elevation of the new point is obtained. If the work is properly done, there will be two forward and two backward lines converging on the same point. The elevations indicated by these lines should agree closely, the average being used as a close approximation of the true elevation of the new point.

8.2.1.6 Field Notes

The importance of taking legible field notes and of keeping comprehensive notebooks cannot be overemphasized. The surveyor's work is commonly varied, involving many aspects of an operation that may have a long life. His measurements may be the only record of an original situation that is later affected by subsequent mining operations. It is frequently necessary to consult old notes for any of many different reasons. Hence the importance of making the notes taken so clear and explicit that others, years hence, will have no difficulty in reading them and in knowing exactly what was done. Completed notebooks should be preserved.

Different organizations utilize different means of taking and keeping notes. Some use hard-bound notebooks; others prefer loose-leaf pages that can be filed according to the level or section to which they pertain. The format of the data recorded also varies somewhat. The important thing is that notes for each job record the nature of the work, the place, the date, the people involved, the equipment used, and the conditions under which the work was done. The data kept must be clear, complete, and consistent. They should be accompanied by a neat sketch showing the situation. References should be made to related notes that may pertain to the same work or problem. No erasures should be made; errors should be canceled by being crossed-out and the data reentered on a new line. The data should be taken with a sharp, medium-hard pencil, and care should be taken to avoid smudging as far as possible. A table of contents should be kept in the front of the notebook, listing all the work.

8.2.1.7 Calculations

The mine surveyor should develop the habit of making all his calculations in a bound workbook, except for those that are required in loose-leaf form for convenient filing. As these books become filled, they should be indexed and filed for easy back reference.

Each page should be captioned with the name of the calculation, the purpose, the situation to which it refers, the name of the person doing the calculation, and the date. If the work involves computer printouts or calculator tapes, it is wise to attach these in appropriate fashion. The goal is to facilitate the work, to be meticulously accurate, and to leave a record so complete that subsequent readers will have no difficulty in understanding what was done and in checking the work. Neatness and clarity are essential attributes of a good calculation book.

8.2.1.8 Maps and Sections

Formerly all mine maps were hand-drawn. Now, however, with the ever-increasing use of computer graphics, many maps are produced by plotters. Even so, there is continual need for updating maps manually. One of the best practices is to put the grid and all permanent detail on mylar sheets or other high-quality translucent stock. This then can easily be reproduced or copied on paper or vellum. Secondary tracings made in this way can be completed in various ways to accentuate certain types of detail and can themselves serve as masters for subsequent families of tracings and prints.

Great care should always be taken in laying out the original grids to make sure that the horizontal and vertical lines are exactly perpendicular and that the lines are exactly spaced. Only if this is correctly done can the subsequent work be plotted properly.

Each map should be numbered and the master tracing indexed and filed for easy reference and recovery. Storage should be in a fireproof vault. Very important maps should also be microfilmed and the films stored in a different location for greater security. Working copies are easily reproduced or copied as needed.

8.2.1.9 Computer Calculations

A wide variety of computer programs for solving surveying problems is available from computer manufacturers, computer service organizations, and proprietary sources. A similar variety is available for use with the smaller desk-top stored-program calculators. These programs are easy to use, fast, and accurate, and make obsolete the calculation of traverses and other routine problems by manual means except in emergencies. Programs of this kind should be used as much as possible to reduce errors and increase productivity.

8.2.1.10 Innovations in Instrumentation

New technology has kept pace with the surveying profession as well as other aspects of mining. Surveying accuracy, distance, and recording features have been enhanced with the introduction of both electronic and laser technology. These features are especially useful on surface projects such as refuse piles, roads, ponds, plants, land and property traversing, and pipelines.

Electrical transits, capable of high degrees of accuracy, are now readily available for use in rugged environments. These transits carry batteries for improving optical prism angles and often combine with straightforward numerical keypads for transit control. Keyboard commands give the operator access to on-board software designed to provide the broadest range of capabilities within cost limits. The software can include selection of angles, slope distance, horizontal angles, height measurement, missing line measurement, remote elevation measurement, northing and easting, and setting-out functions. These transits very often carry RS-232 output ports to interface with office computers.

Electronic distance measuring instruments (EDMs) are electronic instruments that either attach to the transit (or theodolite) or stand alone for one purpose, and that is to measure the horizontal distance between two stations with remarkable accuracy (8.2.1.3). Using infrared light beams, one type of attachable EDM instrument bounces the light between stations and measures the time required to send and receive the signal. Another type of EDM is free standing and uses the principle of phase difference comparison between the emitted-to-received modulated light beam of a xenon gas lamp. From either application, distance can be reduced to a high degree of accuracy. The EDM is usually battery operated and is designed for at least 4 to 6 hr of continuous work.

Laser based technology is widely accepted for high-precision work such as vertical shaft sinking or driving inclined or horizontal slopes or tunnels. Lasers are also being used as intermediate sight instruments for face advances up to 1000 ft (300 m). An infrared laser beam is projected from a preconstructed, battery-operated tube hung from a string attached to a station and driving spad. The tube and battery are considered permissible, and a magnetic switch is used to turn the beam on and off.

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8.2.2 AERIAL MAPPING

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The purpose of this chapter segment is to assist engineers in the development and acquisition of aerial mapping and mapping control surveying services. It further covers geodetic, monumented, vertical, and horizontal control networks required of new field development and expansion or routine development of mapping for operations such as disposal site development, coal storage monitoring, design and construction of surface support systems, etc.

This segment is concluded by an appendix that discusses contract specifications for aerial mapping.

8.2.2.1 Aerial Camera Types and Evaluation Criteria

There are three types of camera systems currently used in aerial mapping production:

1. Older type cameras with metrogen lens or planigon conversions.
2. Newer precision cameras with planigon lens.
3. Image motion precision cameras that limit motion distortion under low altitude and limited light conditions.

Camera manufacturers in each category include:

1. Fairchild K17 and T11 series (T-12 is a planigon conversion).
2. Wild RC 8, 10, & 10A; Zeiss RMK 15/23, RMKA 15/23; Jena MRB.
3. Wild RC 20; Zeiss RMKA 15/23 IMC; Jena LMK.

Older type cameras are cameras with metrogen lens developed, manufactured, and modified during the 1930s, 40s, and 50s. They have limited aperture and shutter characteristics, which limit their ability to produce quality imagery over the entire photograph. This is due to shutter-design lens distortion and lens resolution qualities. The shutter design causes uneven lighting characteristics over the negative area. These cameras should not be used for high-quality mapping requirements. Even cameras that have been retrofitted with new planigon lens still have resolution and shutter limitations that make them inferior as compared to current-day-technology cameras.

The newer precision cameras manufactured since the 1960s have the f 5.6 aperture lens and even newer f 4 aperture lens. They have superior distortion and resolution characteristics as well as effective shutter designs. These cameras are better suited for quality mapping than their predecessors. Lens distortion is reduced to a minimum, and resolution is increased to provide extremely sharp photography translating to more accurate maps. The new shutter designs also provide even lighting distribution over the negative that enhances image quality.

Image motion cameras are the newest generation in camera development. They have been designed to reduce the image motion inherent in aerial photography because of the forward motion of the aircraft camera platform. Though this camera has the ability to produce better quality imagery under very low-altitude and limited-light conditions, it is doubtful that the quality is noticeably enhanced over the f 4 lens cameras under good lighting conditions.

Regardless of the camera system that is used, the camera should meet the following minimum standards as reported in camera calibration reports, issued by the USGS Calibration Laboratory, under the following headings.

Focal Length: 153 + 3 mm (6-in. lens).

Radial Distortion: No reading shall exceed 10; one-half of all readings shall be less than 6.

Area-weighted Average Resolution: Minimum 70.

Model Flatness: The highest plus reading and highest minus reading combined shall not exceed 24; no reading shall exceed 15.

Fiducials: Cameras are equipped with at least 4 and a maximum of 8; only 4 are required for the newer analytic control programs.

Aerial cameras are also designed with different focal lengths for various applications. Longer focal lengths such as 8¼, 12, 24 in. (210, 304, 610 mm), etc., are designed to reduce terrain distortion when only photo imagery is desired. Also the super wide angle, 3½-in. (89-mm) lens can be used by some mapping instruments for topography generation. This camera configuration should only be used in flat to rolling terrain and where there exists a limited number of tall structures. This is due to the large distortion effect this lens has on any vertical feature such as steep grades and tall structures.

8.2.2.2 Film Types and Uses

Should the user require other film uses beside basic mapping, the following film types and basic characteristics are provided.

Black and White Film: Black and white (Kodak 2405) Double X Aerographic film is the most popular film medium in the industry and is used for both mapping and photointerpretation projects.

Color Film: Color film (Kodak 2445) can be processed as either a positive or negative and is used for cosmetic and photointerpretation reasons where black and white photography will not suffice. It is a matter of dispute among stereo compilers whether or not color photography assists or hinders the stereo mapping process. Color photography requires better lighting conditions for exposure, and along with color processing and material costs, makes color photography more expensive to provide.

Infrared Film: In addition to color and black and white photography, two other film types are available. They are used basically for photointerpretive purposes. Color infrared (IR) films (Kodak 2424) are used mainly in vegetation and timbering types of photo interpretation projects and also can be processed for mapping purposes.

Black and white IR is also useful in determining shallow or minor water areas and swamp type conditions, in that water appears very black in the imagery.

Image Motion Photography: Two other types of black and white film, with finer-grained emulsion characteristics, are used in mapping in conjunction with the new image motion cameras. They are Plus X and Panatomic X films.

8.2.2.3 Photo-scale to Map-scale Ratios

In the competitive commercial world of aerial mapping, it is common—more so in the United States than in any other country—to stretch the vertical resolving limits of the stereo compilation equipment by flying higher in order to be more competitive. This is done, with disregard to the resultant decrease in map accuracies, in the interest of cost savings.

There are two primary factors in determining the proper photo-scale to map-scale ratio. One factor is the vertical resolving power of the stereo plotting instrument and the other is the ability to properly discern the physical features of the planimetric detail to be shown on the map. Because photogrammetry is a visual science, the better the stereo operator can see the ground and its features, then the more accurately the details can be measured and plotted. A camera is similar to the human eye (in the respect that the closer it is to an object, the better it sees the object), thus lower-altitude photography will produce more accurate and more detailed maps.

Certain constraints should be adhered to when using current technology optic train stereo plotters and aerial cameras with 6-in. (150-mm) nominal focal length for aerial mapping work. Basic map accuracy standards can be readily achieved if the following limitations are observed.

For horizontal accuracy and detail completeness, the ratio between the photo scale and map scale should not exceed five or six diameters. Example: 1 in. = 100 ft mapping equals 1 in. = 500 ft or 1 in. = 600 ft photography (1 mm = 1.2 m mapping equals 1 mm = 6.0 m or 1 mm = 7.2 m photography.) A smaller physical feature can be more readily seen and accurately measured by a smaller ratio.

For contours and vertical measurements, a C (contour) factor of 1500 to 2000 is recommended. This ratio is a factor of aircraft altitude. The altitude of an aircraft can be determined by multiplying the photo scale times six. The correct contour interval is determined by dividing the aircraft altitude by the C factor. Example: photoscale = 1 in. = 300 ft × 6 = 1800 ft, which is the altitude, divided by 1800 C factor = 1 ft contour.

The basic photo scale to contour convention involves the following:

l-ft (0.3-m) contours: 1 in. = 250 ft to 1 in. = 300 ft scale photography (1 mm = 3.0 m to 1 mm = 3.6 m).

2-ft (0.6-m) contours: 1 in. = 500 ft to 1 in. = 600 ft scale photography (1 mm = 6.0 m to 1 mm = 7.2 m)

5-ft (1.5-m) contours: 1 in. = 1000 ft to 1 in. = 1200 ft scale photography (1 mm = 12.0 m to 1 mm = 14.4 m).

Where minimum planimetric features exist in the project area, photography as high as 1 in. = 1500 ft (1 mm = 18 m) could be used for 5-ft (1.5-m) contours.

8.2.2.4 Seasonal Affects on Aerial Mapping

Photogrammetry is the science of vision and film imagery. Precision measurements require sharp and distinct images. Therefore, any impediments to sharp images, such as shadows, obstructions, poor light, and parallax, will cause accuracy problems wherever they occur.

Let us review the problems, one at a time, to understand their impact on accuracy. Such an understanding can lead to preplanning to minimize the negative effects to the overall project.

Shadows are a direct result of sun angle. The lower the sun angle, the longer the shadows and the deeper are their intensity. Unfortunately, the best sun angle occurs (June 22) during the foliage time of the year where deciduous trees and plants exist. Therefore, photography must be obtained as soon after leaf fall or just before foliage regrows to receive the best sun angles. Since the sun angle is already quite low when the leaves fall, the best time of year for good results is in the spring.

When the sun angle is lowest (December 22) or when it drops below 30º to the horizon, there is limited residual light for good sharp photography. During this period, shadows are extremely deep so features and ground surfaces in wooded areas cannot be seen properly. Shadows are a definite problem, for example, in rough terrain, where large buildings exist, in evergreen forested areas, etc.

Obstructions are another problem in attaining sharp, clear imagery. They include such things as foliated trees and plants, evergreen trees, snow, weeds, brush, buildings, overpasses, etc. Obstructions such as ground foliage and oak tree stands is another reason that spring photography is better. Rain, snow, and wind action better defoliates and compacts foliage obstructions during the winter months. Depending on map details and accuracy requirements, a certain amount of field survey completion may be required due to such obstructions.

In conclusion, close scrutiny of the aerial photos will provide a good idea of map accuracy and completeness. If ground features on the photo cannot be seen for any of the foregoing reasons, the chances are neither can the stereo compilation technician see them. Such areas would thus be prime places for field inspection and checking.

8.2.2.5 Control Placement Procedures

Ground control is required to scale and level each stereo model that comprises a project. The ground control for each stereo model within a project must be tied to the same horizontal and vertical network. As with good surveying practices, it is recommended that all circuits (vertical and horizontal) begin and end on different benchmarks and monuments. Ground control can be preestablished and pretargeted prior to flight or run after the flight to photo-identifiable points. Pretargeting is more positive and accurate, but properly identified physical features can be just as accurate and usually established with less cost and planning effort.

There are two types of control procedures. One is conventional full-model control that requires the field survey crew to occupy each of the numerous points in each stereo model. The other is analytical control that requires a limited number of points to be occupied in each group of models by the field survey crew. By using these limited number of points and specialized office instruments and computer programs, the values of the remaining required points can be computed.

Each stereo model is made up of the overlap area created by two adjacent stereo photos; the size of the area is basically 6.3 in. (160 mm) wide (perpendicular to the line of flight) and 3.6 in. (90 mm) long (parallel to the line of flight). This rectangle location is controlled by the nadir (center) point of each photo. The model lays between the nadir points of adjacent photos and 3.15 in. (80 mm) (perpendicular to the flight line) on each side of the nadir points. Photo control for each model must fall within these limits. The minimum points per model are as follows.

Vertical control requires four points per stereo model. They must be located as close to the model corners as possible. When adjacent stereo models are involved, it is recommended that two points controlling the adjacent edges of each model be visible in each model. In multiple model projects, the addition of two extra points per extra model edge placement requirement is met.

Horizontal control requires a minimum of three points in each single model project and two additional points in each additional model of the project. These points can be located in any portion of the model area but should be as far apart as possible. By locating the points along model edges, the points can be used for adjacent models so that if one model requires three points, two models will only require four points, and so forth.

The location of photo control points within each model is flexible to some degree. If points are not obtainable at a certain location, it may be possible to relocate the point without harm to its value to the model. Such movement should be performed only by a qualified person.

Preproject planning with the objective of deciding whether or not to pretarget or use photo-identifiable points is important. Photo-identifiable points are more efficient in areas where such points exist and can be readily identified. A project would not be recommended for this procedure if it were in a heavily forested area where one tree looks similar to another on the photo. Also one would not hope to find points in large open grasslands or agriculture areas where no distinguishable features exist. Such areas would require pretargeting. Pretargeting is also recommended for existing ground control that is in or adjacent to the project area.

Photo-identifiable horizontal points must be sharp features, such as intersecting white lines, sidewalk intersections, utility poles, etc. Vertical control points can be less distinct, but must be in relatively flat or in gently sloping areas, such as center of road intersections, edges of pavement, opposite poles, etc.

Misidentification is the most significant problem of photo-identifiable points. This means the survey crew has not located the exact point that the photogrammetrist intended. This requires skill on the part of the crew to occupy points only that they believe will not be misinterpreted by the stereo compiler who has not been to the site.

For instance, for horizontal points, if one utility pole in a long line of poles is being located, care must be exercised so that the correct pole is selected. Also, if it appears the pole has been freshly set, advise the photogrammetrist since it could have been relocated since the photo was taken.

For vertical points, in selecting the center of an intersection, if it so happens that at that point there is a deep pothole, do not locate the point at the bottom of the pothole. Obtain the elevation of the point at the average normal pavement surface adjacent to the pothole.

8.2.2.6 Targeting for Photo Control

Although pretargeting is the most positive and accurate procedure, it is not without drawbacks. The procedure is more expensive since additional employee-hours are involved to set and monitor the points prior to the flight and to retrieve the targets after the flight. Targets are subject to wind, vandalism, and other disturbances that can render them unusable unless they are monitored just prior to the flight.

Target materials can be large plastic sheeting, paper, cloth, wood, lime, rock dust, etc. Aluminum foil or any other reflective material should not be used. The best colors are white in contrasting areas of grass, black top, forest floors, etc., and black for contrasting areas such as dry earth, etc. Fluorescent orange should not be used in grass areas for black and white photography since this color is in the same grey spectrum as grass.

The best shapes for control are chevrons, Ts, and crosses. Circles can be used if the point is only for vertical control. When targeting to locate a feature such as a manhole, the best approach is to outline the feature with a 6- to 12-in. (150- to 300-mm) outline, depending on the photo scale.

The following information on target sizes and shapes will be of assistance:

Negative Scale	Thickness of Leg	Length of Legs
1 in. = 150 ft to 1 in. = 300 ft (1 mm = 1.8 to 3.6 m)	1 ft (0.3 m)	2 to 3 ft. (0.6 to 0.9 m)
1 in. = 350 ft to 1 in. = 500 ft (1 mm = 4.2 to 6.0 m)	1 ft (0.3 m)	3 to 4 ft (0.9 to 1.2 m)
1 in. = 500 ft to 1 in. = 800 ft (1 mm = 6.0 to 9.6 m)	2 ft (0.6 m)	4 to 5 ft (1.2 to 1.5 m)
1 in. = 1000 ft to 1 in. = 1200 ft (1 mm = 12.0 to 14.4 m)	2.5 ft (0.8 m)	8 to 10 ft (2.4 to 3.0 m)
over 1 in. = 1200 ft (1 mm = 14.4 m)		

Target leg length should be 1% of photo scale, and the width of each leg should be 30 to 36 in. (760 to 1080 mm).

1. The targets are placed so that the point being targeted is at the center intersection of the panels, with the exception of the

L or chevron panel. On the L or chevron panel, the point being marked is the intersection of the inside edges of the material.

2. Plastic flagging materials should be avoided for paneling purposes in grazing areas. Grazing animals eat certain types of plastic flagging.

3. In wooded areas, the length of the target panels should be doubled.

4. Any panel being used as a vertical target must be placed in a relatively flat or gently sloping area.

8.2.2.7 Ground Control Surveying Procedures

Photogrammetric ground control is accomplished with standard surveying equipment and procedures. It involves the systematic completion of both horizontal and vertical networks that provide scale and elevation points to strategic locations for the stereoscopic model scale and leveling.

Horizontal scale can be accomplished by measured horizontal distances between chosen control points or the establishment of coordinates using various accepted procedures. Closed traverse coordinate control is the preferred procedure. Coordinate networks can be established by conventional traverse, triangulation, inertial guidance, or satellite positioning (GPS) instrumentation.

Vertical control can be established by differential leveling, vertical angle, barometric, inertial guidance, or satellite positioning instrumentation.

Both vertical and horizontal networks can be tied to existing state plane coordinate systems and NGS vertical datums or independent arbitrary systems. It is very important that each total project is based on only one network or datum and that the correct equipment and procedures are used that yield the required accuracy of the map scale and contour interval.

Accuracy requirements for aerial mapping ground control are as follows:

1. Horizontal control is to be accurate to $^1/_{100}$ of the map scale.

2. Vertical control is to be accurate to $^1/_{10}$ of the contour interval.

Both of these accuracies can normally be achieved by third-order, class-one procedures.

8.2.2.8 Stereoscopic Compilation

There are three basic generations of stereoscopic (aerial mapping) equipment. They are (1) the anaglyphic analog (Kelsh type), (2) the optic train analog, and (3) the analytic. Although types 2 and 3 are the rule in most mapping firms and other mapping-related organizations, there are still a few type-1 instruments in use today. The following contour factor and scale ratio for each instrument type will better assure accurate mapping, even though many maps are being prepared from higher factor and ratio photography:

Type	Scale Ratio	Vertical Factor
1	4× to 5×	1200 to 1500
2	5× to 6×	1500 to 1800
3	6× to 7×	2000 to 2800

All stereo mapping instruments require periodic recalibration. Any prudent, quality-control-conscious mapper will make recalibration checks by both internal personnel and manufacturer personnel a routine matter. Internal checks should be made monthly, and manufacturer checks should be made every three years, or when internal checks indicate the instrument needs professional checking.

Normal map-scale to contour-interval ratios fall into the following basic convention:

Photo Scale	Map Scale	Contour Interval
1 in. = 200 ft to 1 in. = 300 ft (1 mm = 2.4 to 3.6 m)	1 in. = 20 to 50 ft (1 mm = 0.24 to 0.60 m)	1 ft (0.3 m)
1 in. = 500 ft to 1 in. = 600 ft (1 mm = 6.0 to 7.2 m)	1 in. = 100 ft (1 mm = 1.2 m)	2 ft (0.6 m)
1 in. = 1000 ft to 1 in. = 1200 ft (1 mm = 12.0 to 14.4 m)	1 in. = 200 ft (1 mm = 2.4 m)	5 ft (1.5 m)
1 in. = 2000 ft to 1 in. = 2400 ft (1 mm = 24.0 to 28.8 m)	1 in. = 400 ft (1 mm = 4.8 m)	10 ft (3.0 m)

Due to user needs, it is not uncommon for variations of the foregoing to be used. It is wise to remember that the photo-to-map conventional plotting ratio determines the map accuracy, and any enlargement or reduction of the nominal plotting scale will affect the size of the map but will not change the accuracy.

The basic scale and contour interval accepted standards follow. Although these are taken from the US Department of Transportation's *Guidelines for Aerial Mapping,* they are very close to and considered by some users to be "national map accuracy standards."

1. *Contours.* Ninety percent of the elevations determined from the solid-line contours of the topographic maps shall have an accuracy with respect to true elevation of one-half contour interval or better, and the remaining 10% of such elevations shall not be in error by more than one contour interval. This accuracy shall apply only to the contours that are on each map. Thus, in each particular area where the intermediate contours have been omitted because of the steepness of the ground slopes, and only the index contours are delineated on the maps, the accuracy stipulations apply to the contour interval of the index contours. Wherever the intermediate contours are not omitted, the accuracies are applicable to the contour interval specified for the topographic maps. In densely wooded areas where heavy brush or tree cover fully obscures the ground, and the contours are shown as dashed lines,* they shall be plotted from the stereoscopic model, while making full use of spot elevations measured photogrammetrically in places where the ground is visible.

* Author's Note: Dashed contour lines are used to indicate the contour placement is the best judgment of the map compiler and the foregoing accuracies may not apply.

2. *Coordinate Grid Lines.* The plotted position of each plane coordinate grid line shall not vary by more than $^1/_{100}$ in. (0.25 mm) from true grid value on each map manuscript.

3. *Horizontal Control.* Each horizontal control point shall be plotted on the map manuscript within the coordinate grid in which it should lie to an accuracy of $^1/_{100}$ in. (0.25 mm) of its true position as expressed by the plane coordinates for the point.

4. *Planimetric Features.* Ninety percent of all planimetric features that are well-defined on the photographs shall be plotted so that their position on the finished maps shall be accurate to within at least $^1/_{40}$ in. (0.64 mm) of their true coordinate position, and none of the features shall be misplaced on the finished map by more than $^1/_{20}$ in. (1.27 mm) from their true coordinate position.

5. *Spot Elevation.* Ninety percent of all spot elevations placed on the maps shall be accurate to one-quarter of the contour interval, and the remaining 10% shall not exceed one-half the contour interval.

8.2.2.9 Graphics and Drafting

Graphics or the finished map presentation comes in many forms. They include (1) pencil manuscript, (2) ink drafted, (3) scribe drafted, (4) laser or electrostatic drafted, and (5) digital formatted. The current media can be paper, vellum, mylar, tapes, or floppy disks.

Descriptions are as follows:

1. *Pencil Manuscript.* The end product is usually on one- or two-sided, matte-finished mylar and is in pencil or ballpoint ink. The graphics are the freehand result of the stereo operator, with little or no attempt at finish drafted quality.

2. *Ink Drafted.* Usually entails drafting with cartographic ink equipment on mylar with mechanical lettering and the finished qualities of a skilled cartographic draftsperson.

3. *Scribe Drafting.* Presents the same type of finished drawing as ink drafted, with the exception that all line weights are standardized by the designated scribe points and cutters. Scribing insures standard line weights, regardless of the drafter. Also scribe line work then becomes a photographic image and is part of the reverse side of the mylar emulsion, rather than ink being applied to the emulsion surface.

4. *Laser or electrostatic drafted.* These are the most recent generation of plotters. The laser plotter provides a solid line image, while the electrostatic plotter provides a screened line image of various dot density, depending on the plotter design.

5. *Digital Data.* These data are provided in user software language. It is the responsibility of the user to clearly designate the exact design language or standard interchange format that their software requires.

8.2.2.10 Digital Mapping

With the advent of computer-aided drafting (CAD) came a new need for aerial mapping output. Due to various advanced data collection systems that are available for stereo plotters, it is now possible to provide maps in digital form for computer-aided drafting and design (CADD) requirements.

A popular characteristic is the ability to layer features. This permits the land features to be stored in separate layers so the CAD user can display or plot individual features in an overlay fashion. Coding of lines for different line weights, symbols, and colors can also be provided. Digital mapping data can be structured for both the mainframe or personal computer (PC) systems.

Since there are many more types of host systems available than collection systems, it is necessary to do file conversions. Although there are some good conversion software available for the more prevalent systems, file conversion is a very real problem that needs to be resolved prior to starting into a digital mapping project. File structure differences between mainframe and PC systems can also present problems between collection and use systems. A sample disk or tape for trial use on the intended system is recommended.

File structures can also be designed to include 2D, Z and one-half D (2D tagged Z), or 3D data. The differences are as follows:

1. 2D files will provide feature data with X and Y values only. This will provide only the ability to recreate lines, but will give no intelligence for vertical data.

2. 2 and one-half D or 2D tagged Z will provide files with both vertical and horizontal intelligence, but the vertical intelligence will be limited to visual numeric titles for all contours and spot elevations.

3. 3D will provide feature data with X, Y, and Z intelligence so the database can be used directly for such things as earthwork volume computations and grade design information with available CAD software.

It should be noted that 3D file structures can be burdensome to PC systems and should be carefully considered. A successful alternate would be to use 2D tagged Z files and to upgrade small areas of use to 3D by windowing and enhancing the data on the user system.

8.2.2.11 Field Completion and Checking Procedures

There are many reasons that ground features can be obscured from view on an aerial photo. These include foliage, shadows, overhanging structures, etc. All these conditions cause the stereo instrument operators to use their best judgment or leave out data altogether. This causes the need for field completion and field checking.

The field completion is accomplished by using field surveying techniques and locating the missing features by coordinate and elevation to be plotted on the final map. This works best when accomplished after the map is complete. The use of autorecording "total station" surveying equipment and "digital terrain model" software has greatly improved both the cost and accuracy of this procedure.

Since errors or omissions can occur in any science, it is prudent to field check all maps before they are relied upon for engineering and constructions computations. It is wise to assume that any map that is not checked is wrong.

Field checking for physical features includes the establishment of a traverse to the features to be checked to obtain true coordinate positions. Checking for contours includes the establishment of a cross section to locate each contour that crosses as perpendicular as possible to at least 10 contours.

All control checks must start and end on control points that were established for the original mapping.

Field checking and field completion are normally options in any mapping project and must be requested. This practice is due to the normal requirement of a nonbiased checking procedure.

8.2.2.12 Volumes

Utilizing aerial mapping for volume determination is a proven and reliable procedure, relied on by many. The two basic methods that are employed are (1) the horizontal slice using an equal planes computation, and (2) the vertical slice using an end area computation.

The horizontal slice is used more often in that it involves the preparation of a contour map normally required as an end product. With current equipment and technology, the stereo plotter is enhanced with a digital collection device attached to a small computer which reduces human error. Each contour is compiled by conventional means, and the digital collection system collects and stores the area of each contour slice as it is completed. A simple program then computes the volume of the pile. A base computation is then required to provide quantitative values to the underside of the pile that is not visible by aerial photography. The procedure requires the digital input from a base contour map or base map with grid elevation points. Care must be exercised to guarantee both the contours produced in the quantity collection phase and the base map are on the same datum and grid system.

One- and two-foot (0.3-and 0.6-m) contours are recommended for accuracy requirements of valuable material such as coal and limestone. One-foot (0.3-m) contours should be used in gentle slope areas, limiting 2-ft (0.6-m) contours to steep slopes.

The vertical slice procedure differs only in that the computations are done by an average end area method based in elevation data obtained by cross sections. If the cross-sectional method is employed, the closeness of sections impacts the accuracy. Ten-foot (3-m) intervals between cross sections plus added sections at major breaks provide more reliable results than using sections with greater intervals.

A cross-sectional collection system replaces the quantitizer of the horizontal procedure, but all other automated aspects should remain the same including the base map treatment.

Both procedures yield basically the same accuracies when properly constrained.

To eliminate one major cause of discrepancy in comparing repetitive volume studies of the same pile, it is important to include all material that exists above the base in each inventory. It is normal for material to exist beyond the toe of slope in any material storage area. Mostly this material is considered unusable. If this material is ignored in one inventory and the pile enlarges to cover it in the next inventory, there will exist a quantity error when comparing the aerial volumes to those obtained by other sources such as scale readings, production reports, etc.

Aerial inventory procedures have been proven to be reliable within 2% on the average pile configuration. The accuracy is affected by the ratio of volume to pile surface. Piles that have a large surface and a shallow depth provide a lesser accuracy percentage than compact piles of greater depth.

The vertical slice procedure does not produce a contour map, and an additional step must be accomplished. This would include the compilation of a map by conventional aerial methods or using the cross section data with digital terrain model (DTM) software to create a topo map.

8.2.2.13 Ortho Photography

Ortho photography is the product of a procedure that corrects the distortions that exist in all aerial photography. These distortions exist due to the instability of the camera platform (aircraft), the terrain relief, and the angle of the light rays entering the camera lens.

The ortho instrumentation attached to a stereo plotting instrument is designed to semi-automatically relieve these distortions. The equipment is monitored by a stereo operator as it transfers the photo image from the original aerial negative to a new negative. The function of the equipment is to rectify the image in this transfer process so as to reposition it orthographically in its correct plane position. The resulting negative now has the total image restored to its correct coordinate position to "national map accuracy standards" (see 8.2.2.8).

The negative can now be processed to either a paper or mylar media in positive form. Photo scales can be adjusted by enlargers and copy cameras to any desired scale.

Since the photo has been rectified to the accuracy of a map, contours can be applied by drafting in ink on the mylar, creating a black line or by a positive overlay method creating a white contour. Since both black and white images occur on a photo, some problems of contour legibility will exist with either procedure. A third process involving an overlay lab procedure will produce a black contour with a white halo so the contour can be seen in either condition.

Two other shortcomings of the process are as follows:

1. The rectified negative is not as sharp as the original negative since it has gone through a duplicating process.

2. The procedure involves the transfer of only one of the two images making up the stereo model. This causes the displacement of the top of any building or feature that projects above the earth's surface. Due to such lean, features that are hidden by such lean will not be seen on the final photo, and there will be edge match inconsistencies due to the lean along model ties.

8.2.2.14 Cross Sectioning and Profiling

Automated cross-section and profile collection, by aerial mapping procedures, is a common practice. The stereo instrument is equipped with hardware and software systems that lock the stereo instrument on a given section. The equipment is designed to automatically collect and store the station, offset, and elevations that are input by the operator. The station is manually inserted, the offset is automatically inserted by digital scalers, and the elevation is automatically inserted once it is manually observed by the operator.

The accuracy of the data is twice that of a contour map and greatly reduces the task and human error possibilities over manual compilation from a topo map.

The output can be a magnetic tape or disk with the cross-sectional data, a hard copy printout of the section value, offset distance and elevation, or a hard copy plot that would be digitally plotted by a mechanical plotting device to any desired horizontal and vertical scale.

8.2.2.15 Photo Lab Products

The photo lab is an important attribute to the aerial mapping science since it provides many useful byproducts. Some of the user products are as follows:

1. Contact prints are 9 × 9-in. (229 × 229-mm) direct reproductions of the original negatives and can be provided in color or black and white depending on the negative.

2. Photo indexes are composites of the 9 × 9-in. (229 × 229-mm) contact prints that are reduced to one-third or one-quarter of their size so they can fit on one normal sheet size for use in identifying the location of any one print in a project.

3. Photo enlargements are direct enlargements of the original aerial negative to attain a large feature scale of the area. This can be done on a simple enlarger that will also correct for tip and tilt distortions introduced into the photo by aircraft instability.

4. Direct positives are reproductions of original drawings made on a vacuum frame to eliminate any distortions such as those created by standard print machines.

5. Mosaics are a composite of either aerial photos or drawings.

All of the foregoing products can be produced on either resin-coated waterproof paper (RC paper) or mylar. Mylar material is convenient for photo use in that it can be reproduced on a standard print machine.

8.2.2.16 Project Cost Factors

Photogrammetry has the ability to accurately map large tracts of land more cost- and time-effectively than any other procedure.

Cost is the predominant question when it comes to planning a project and one that is asked usually before definitive data is gathered.

The cost is related to the project as a unique entity, and it is difficult to respond to a project fee request without detailed information about the site. Since photogrammetry has a number of fixed procedures that have tightly controlled parameters, cost evaluations must address some of the following questions:

1. The number of miles from the project to the aircraft base.
2. The scale and contour interval of the map to be produced.
3. The size and shape of the tract of land.
4. The location of existing ground control to the site.
5. The road pattern or open areas available for field survey crew mobility.

6. The relief and its specific elevation differences and complexity.

7. The amount of cultural features, on the site, to be shown on the map.

8. Whether or not the map is to be field checked to verify compliance with the national map accuracy standards.

9. Whether the map is to be field completed to verify non-photo visible features.

These are a majority, but not all, of the factors that individually and significantly impact cost. Since the answers to all of these questions cannot be the same for each site, the cost cannot be the same by any evaluation criteria.

Basic cost rules are functions of scale and contour intervals. Basic map scales and contour intervals are

1 in. = 400 ft (1 mm = 48 m scale) with 10-ft (3-mm) contours

1 in. = 200 ft (1 mm = 2.4 m) scale with 5-ft (1.5-m) contours

1 in. = 100 ft (1 mm = 1.2 m) scale with 2-ft (0.6-m) contours

1 in. = 50 ft (1 mm = 0.6 m) scale with l- and 2-ft (0.3-and 0.6-m) contours

As the scale and contour interval decreases, the cost increases at the rate of over double the preceding category.

Minimal project sizes can be equated as follows. If the project would take three days in the field to collect the necessary data to construct a map by standard survey methods, then it has reached the area where it can probably be done as cheaply by aerial methods.

APPENDIX A. CONTRACT SPECIFICATIONS FOR AERIAL SURVEYING

JAMES MORGAN

This segment generalizes the basic criteria needed to establish or implement a successful aerial mapping project. It contains specifications for both managing a project and evaluating the results of a project. These specifications were extracted from a series of general contracting guidelines from generic public sources.

8.2.2.A1 General

1.01 Definitions:

a. The contracting officer is the official representative of the client.

b. The contractor is that firm, company, or organization to which the mapping or service contract has been let.

1.02 General Mapping Specifications:

a. The contractor will supply the contracting officer with a sample format of each map and/or overlay type showing the placement and content of all border information for approval prior to final map and/or overlay production.

b. All deliverable maps and/or overlays shall contain the following statement:

This map/overlay is correlated to the State Plane Coordinate System, 1983 North American Datum or the local mine control network, whichever applies.

c. All lettering on all deliverable maps and/or overlays shall be done by mechanical means only—freehand lettering will not be accepted.

8.2.2.A2 Aerial Photography

2.01 Project Area and Contract Map:

The location, size, and boundaries of the areas to be mapped should be outlined on a USGS 7½-in. (190-mm) quadrangle.

2.02 Conditions During Photography:

Vertical photography should be flown during the period when deciduous trees are barren, between the hours of 10:00 am and 2:30 pm Eastern Standard Time, and when the sun angle or elevation is not less than 30° above the horizon.

Photography will not be undertaken when the ground is obscured by snow, haze, fog, or dust; when streams are not within their normal banks; or when the clouds or cloud shadows will appear on more than 5% of the area in any one photograph. Photographic targets should be used to mark horizontal control points for aerial triangulation and control of base map compilation.

2.03 Scale of Aerial Photography Negatives:

The altitude above average ground elevation for aerial photography should be such that negatives will be at the scale specified in the following schedule:

Map Scale	Contour Intervals	Max. Negative Scale[a]
1″ = 400′ (1 mm = 4.8 m)	10′ – 20′ (3–6 m)	1″ = 2000′ (1 mm = 2.4 m)
1″ = 200′ (1 mm = 2.4 m)	5′ – 10′ (1.5–3 m)	1″ = 1200′ (1 mm = 14.4 m)
1″ = 100′ (1 mm = 1.2 m)	2′ – 5′ (0.6–l.5 m)	1″ = 600′ (1 mm = 7.2 m)
1″ = 50′ (1 mm = 0.6 m)	1′ – 2′ (0.3–0.6 m)	1″ = 300′ (1 mm = 3.6 m)

[a]Depending on the type of stereo plotters and 'C' Factor.

Negatives deviating from the following scales by more than 5% may be rejected.

2.04 Flight Plan:

The contractor's flight plan shall be drawn on a copy of the 7½-in. (190-mm) quadrangle. Each flight line will be flown continuously across the project area. The principal points of the first two and the last two exposures of each flight strip should fall outside the boundaries of the area to be mapped. All side boundaries should be covered by a minimum of 25% of the photo image format.

2.05 Endlap:

Photography used in the development of orthophoto maps should have an endlap of approximately 80% to ensure that one photograph will cover each map (based on neat image of 25 × 25 in. or 635 × 635 mm).

2.06 Sidelap:

Sidelap should be no less than 25% for topographic mapping.

2.07 Crab:

Crab in excess of 3° may cause rejection of a flight line or any portion thereof in which the excess crab occurs.

2.08 Tilt:

Tilt of the camera from verticality at the instant of exposure should not exceed 3°, nor should it exceed 5° between successive exposure stations. Average tilt over the entire project should not exceed 1°.

2.09 Aerial Camera:

A US Geological Survey (USGS) camera calibration report no more than four years old should be required for each camera used to obtain aerial photography.

a. Camera and lens.

The aerial camera should be a precision aerial mapping camera equipped with a low-distortion, high-resolution lens with the following characteristics:

1. Focal length, 153 + 3.0 mm Universal Aviogon, Pleogon A, or equivalent.
2. Usable angular field, at least 90°.
3. The following table lists the minimum acceptable radial and tangential resolution cycles line pairs per millimeter (measured with type V-F stectroscopic emulsion on micro flat glass plates exposed at maximum lens aperture):

0	7.5	15	22.5	30	35	40
57	57	48	48	40	34	14

b. Filter.

An appropriate glass filter with a metallic antivignetting coating should be used.

c. Shutter speed and efficiency.

1. The camera shall be equipped with a between-the-lens shutter of variable speed. The range of speed settings shall be such that in conjunction with flight height and aircraft speed, the camera will produce negatives that will result in high definition photographs. The shutter should also have a speed of $^{1}/_{200}$ sec or slower for laboratory testing.
2. The effective exposure time and the efficiency of the shutter as mounted in the camera will be measured at maximum

aperture, and the shutter shall have a minimum efficiency of 70% at a speed of $^1/_{200}$ sec.

3. This test shall be made in accordance with "Method I," American National Standard PH3-48-1972 (R1978).

d. Platen flatness and identification.

Cameras shall be equipped with an approved means of flattening the film at the instant of exposure. Data markers that protrude inside the focal plane frame should not exceed 0.25 in. (6.33 mm) in height and 1.0 in. (25.4 mm) in length and shall not obscure any part of the fiducial mark or reduce the usable image area.

e. Fiducial marks.

1. Each camera body shall be equipped with means of recording eight fiducial marks on each exposure, the marks to be located in each corner of the format and at the center of each side.

2. All fiducial marks and other marks intended for precise measuring shall be clear and well defined on the negative and shall be of such a form that the standard deviation of repeated readings of the coordinates of each mark made on a comparator shall not exceed 2µm.

f. Stereomodel flatness.

The deviation from flatness (elevation discrepancy at photography scale) at measured points should not exceed + $^1/_{5000}$ of the focal length of nominal 6-in. (153 + 3.0 mm) cameras. If elevation discrepancies exceed this value, the camera should not be used.

2.10 Aerial Film:

The black and white aerial film should be a fine-grained, high-speed photographic emulsion on a dimensionally stable base.

8.2.2.A3 Horizontal and Vertical Control

3.01 General:

Sufficient horizontal and, if applicable, vertical control surveys shall be established by the contractor for all photogrammetric mapping purposes. Prior to the establishment of the necessary basic horizontal and vertical control, the contractor shall make a thorough search of the project area for existing control of second order Class II accuracy or better, as established by the National Geodetic Survey (NGS, formerly the United States Coast and Geodetic Survey [CGS]). Additional control points established by the contractor will be monumented with permanent monuments as described in subsegment 3.05.

3.02 Horizontal Control Surveys:

All basic horizontal control shall be established by triangulation, traverse, or Global Position System (GPS) (see subsegment 3.08). The traverse error of closure of base control shall not be greater than one part in 10,000, and the allowable angular error shall not exceed 5 sec per angle.

3.03 Vertical Control Surveys:

All basic vertical control shall be extended from existing NGS or local mine datum bench marks and referenced to the National Geodetic Vertical Datum 1929 Adjustment. The vertical control survey shall be established according to NGS standards and procedures, and the survey shall be of third-order accuracy or higher.

3.04 Photo Control Contact Prints:

Control points will be pinpricked and symbolized on the face of the appropriate aerial photograph, and the locations will be precisely described on the back of the photo.

3.05 Permanent Monuments:

A permanent monument is defined as a shaft of concrete, metal, or precast concrete with metal reinforcement in or on which a survey disk or cap is imbedded or affixed, and which is capable of being detected by commonly used magnetic or electronic locators; is buried or driven nearly vertically in the earth in such a way so as to possess horizontal stability; and is extended below the normal frost line or a minimum depth of 30 in. (750 mm) below the ground surface unless a subsurface obstruction dictates a lesser depth. A survey disk or cap with a stem set flush in a drillhole in bedrock, rock outcrop, or concrete structure shall also be defined as a permanent monument. Monuments shall be ½-in.(13-mm) inside diameter or greater iron pipes, ½-in. (13-mm) diameter or greater solid metal rods, or 4 × 4 in. (100 × 100 mm) or greater concrete shafts. If iron pipes or rods are used without concrete, they must be flush with or below the ground surface. Iron pipes or rods projecting 1 in. (25 mm) or more above the ground surface shall be encased in concrete from the disk or cap to 8 in. (200 mm) below the ground surface. The disk or cap of each monument established by the contractor shall be stamped with the name and a sequential number obtained from the traverse. In the event that both vertical and horizontal control are to be established along the same route the vertical control should be established on horizontal control monuments that are accessible; no additional vertical control permanent monuments will be required along such routes.

3.06 Survey Records:

a. Field notebooks and data collector printouts.

Field notebooks shall be carefully and neatly prepared, identified, indexed, and preserved. All data regarding the establishment and extension of horizontal and vertical control, including descriptions of all established and recovered monuments, shall be recorded. Where existing control points are recovered by the contractor in extending the basic control required by the contract, the field notebooks should contain:

(1) Information as to the general condition of the recovered mark

(2) Original description

(3) Exact letter and numbers stamped on (not cast in) the mark

(4) Amended description, if applicable

(5) Additional tie data, if any

(6) Sketch of the location to facilitate future recovery

b. Computations.

All computations and adjustments of control data shall be referenced to the field notebooks by book and page number.

c. Control diagram.

This diagram will indicate all horizontal and vertical control pertinent to this project and will be shown on a copy of the index of deliverable maps. This schematic diagram shall show all existing and established control points properly identified in their approximate location. It will also show all traverse lines with their designations, including the beginning and ending points.

d. Control Data.

The contractor shall provide the information listed below for all monumented control points established and/or recovered:

1. Information on control points established by the contractor. This information shall be collected on monument description cards and shall include the following information:

(a) Designation of station.

(b) Date of establishment.

(c) Horizontal and/or vertical control data.

(d) A complete description of the nature and location of the point to include a "to reach" description referenced to nearby landmarks (e.g., proceed from the intersection of Aker Road [SR 1010] and Stagecoach Road [SR 1012] . . .; proceed from US 64 bridge over Neuse River . . .) and identified by field survey ties (bearing and distance) to three or more definable photo image points in the immediate vicinity.

(e) The location of each marked horizontal control point, symbolized on the face of the appropriate photograph by a triangle and annotated on the back with reference to its station designation as in (a).

3.07 Feet/Meter Conversions:

The US Survey Foot (1 meter = 3.2808333333 feet) shall be used in all conversions of State Plane Coordinates from meters to feet or feet to meters. All final control data shall be in feet, and the datum used (e.g., NAD 83) will be noted on any sheets bearing coordinates.

3.08 Global Positioning System (GPS):

GPS satellite surveying is a three-dimensional measurement system based on observations of the radio signals of NAVSTAR Global Positioning System. The GPS observations are processed to determine station positions in cartesian coordinates (X,Y,Z), which can be converted to geodetic coordinates (latitude, longitude, and height-above-reference ellipsoid). When using GPS to establish control surveys, the contractor shall adhere to the specifications for "traditional" surveys delineated in this section; additionally, the contractor shall follow the specifications for second-order surveys in the "Geometric Geodetic Survey Standards and Specifications for Geodetic Surveys Using GPS Relative Positioning Techniques" as incorporated into the Federal Geodetic Control Committee's (FGCC) Standards and Specifications for Geodetic Control Networks. Reduced GPS data submitted to the contracting officer should include baseline vector components in the cartesian system (X,Y,Z), variance-covariance data (or variance-correlations), number of observations available/used in the solution, and standard deviation of the "range" residuals. When the "single" baseline processing method (two-station solution) is employed, this data will be provided for all possible baselines dependent and independent. For example, if there were three receivers used during an observing session, three baselines will be processed; if there were four receivers, six baselines will be processed; and so on. When the "session" processing method (three or more station combined solution using all data from each observing session) is used, *r*-1 independent base line solutions will be provided where *r* is equal to the number of receivers collecting data simultaneously during an observing session. When the "network" processing method (two or more sessions combined solution using all data from each observing session) is used, *s*-1 independent baseline solutions will be provided where *s* is the total number of stations in the network solution. All digital data submitted will contain adjusted coordinates whenever they appear (latitude, longitude, ellipsoid height). The adjusted coordinates must be the result of a three-dimensional, least-squares, minimally constrained adjustment in the reference system for the satellite ephemerides (WGS 72 or WGS 84).

a. General Requirements for a GPS Survey:

1. The monumented point for a proposed GPS station must be in a location suitable for occupation with a survey tripod, stand, or tribrach. Monumented points located on vertical surfaces such as bridge abutment walls, building walls, and retraining walls, etc., are not suitable GPS stations. If it becomes necessary to connect the GPS survey to a station that is not acceptable for direct occupation, a permanent offset station should be established and monumented and tied to the existing network station using traditional surveying methods.

2. All GPS points must be established in pairs. Only one coordinate position will be established; the second position will be used as an azimuth mark.

3. All stations established using GPS will be monumented using permanent markers as detailed in subsegment 3.05.

4. The antenna must be located where there are no obstructions overhead.

5. The antenna should be located where there will be minimum radio interference. Areas with radio interference for the 1227.6 and 1575.42 Mhz frequencies, medium-frequency radar and high-power transmission antennas; high-voltage power lines and transformers, microwave relay towers, and excessive noise from automotive ignition systems must be avoided.

6. The antenna should be positioned at a height so as to avoid ground reflection of the satellite signal.

7. The antenna must not be placed near metal structures.

8. The antenna phase center will be plumbed over the survey point using an optical plummet, collimator or similar instrument. The plumbing for antennas set up on tripods shall be checked with a plumb bob at least once during the survey. This check is for the purpose of determining gross plumb errors of 0.4 in. (10 mm) or more. When back-to-back sessions are observed and an antenna occupies the same station for both sessions, the tripod will be reset and replumbed between sessions.

9. The receiver assembly, antenna, and other equipment must be protected at all times from theft, vandalism, and other types of disturbances.

10. All GPS receivers should have a signal input port for an external frequency standard.

11. Receivers must have the capability to track a minimum of four GPS satellites simultaneously.

12. The "master" receiver will occupy two or more local first- or second-order NGS or NCGS marks during a project. Some vertical data must be obtained via GPS to obtain a model of the geoid in the project area; therefore, a minimum of two vertical control monuments must be occupied.

8.2.2.A4 Analytical Triangulation

Fully Analytical Aerial Triangulation (FAAT) can be used to extend the horizontal ground control subject to the requirements stated in the following. Techniques of high-to-low aerial triangulation may be used, but shall not exceed a two-times ratio. Drop points established from FAAT performed on 1″ = 2000′ (1 mm = 24 m) photography may be used for scaling of 1″ = 1200′ (1 mm = 14.4 m) aerial photography. These drop points may be used in the production of maps from the 1″ = 1200′ (1 mm = 14.4 m) photography. Maps produced from 1″ = 500′ (1 mm = 6 m) photography may be produced from drop points established by FAAT on 1″ = 1000′ (1 mm = 12 m) aerial photography. The FAAT on 1″ = 1000′ (1 mm = 12 m) photography must use horizontal control established from the second order horizontal control points indicated on the control map. FAAT may not be used with aerial photography at scales smaller than 1″ = 2000′ (1 mm = 24 m). Maps at a scale of 1″ = 400′ (1 mm = 4.8 m) can be developed from FAAT performed on 1″ = 2000′(1 mm = 24 m) aerial photography. Maps at a scale of 1″= 200′ (1 mm = 2.4 m) can be developed from FAAT performed on 1″ = 1200′ (1 mm = 14.4 m) photography or using passpoints and drop points established from FAAT performed on 1″ = 2000′ (1 mm = 24 m) aerial photography. Maps at a scale of 1″ = 100′ (1 mm = 1.2 m) can be developed using analytical triangulation blocks on 1″ = 600′ (1 mm = 7.2 m) aerial photography or using passpoints and drop points established from an analytical triangulation block performed on 1″ = 1200′ (1 mm = 14.4 m) photography.

4.01 Nominal Scales for Triangulation:

Aerial triangulation may be used for horizontal scaling to produce maps of 1″ = 400′ (1 mm = 4.8 m), 1″ = 200' (1 mm = 2.4 m), and 1″ = 100′ (1 mm = 1.2 m). The nominal scale of the mapping photographs will be 1″ = 2000′ (1 mm = 24 m) for 1″ = 400′ (1 mm = 4.8 m) maps, 1″ = 1200′ (1 mm = 14.4 m) for 1″ = 200′ (1 mm = 2.4 m) maps, 1″ = 600′ (1 mm

= 7.2 m) for 1″ = 100′ (1 mm = 1.2 m) maps, and 1″ = 300′ (1 mm = 3.6 m) for 1″ = 50′ (1 mm = 0.6 m) maps.

4.02 Ground Coordinate Systems:

All ground positions determined by aerial triangulation will be in the State Plane Coordinate System or local mine datum whichever pertains. Elevations of monumented positions will be used in FAAT computations when they are available. If these elevations are not available, spot elevations derived from USGS 7½-in. (190-mm) quadrangle maps may be used.

4.03 Standards:

a. Positional accuracy.

The root-mean-square error (vector of both northing and easting coordinate errors) of passpoints established by aerial triangulation shall not exceed $^1/_{50}$ of the denominator of the finished map scale.

b. Elevation accuracy.

The elevation accuracy of passpoints does not have to meet the required accuracy for topographic mapping.

4.04 Passpoints:

a. General requirements.

Passpoint locations will be manually selected by reviewing the control photographs with a pocket stereoscope or other suitable stereo-viewing device. Selected passpoints shall be located, symbolized, and labeled on the image side of the control photographs. All selected passpoint locations will lie on unobscured level ground whenever topographic conditions permit.

b. Distribution of passpoints for fully analytical aerial triangulation.

Individual frames will carry a minimum of nine passpoints, with the exception of end frames of flight lines that will carry a minimum of six passpoints. One (1) point will lie near the corner of each neat model, and one point will lie near the nadir position of each neat model. It is recognized that deviation from the ideal distribution may be necessary for those photographs covering bodies of water or areas of heavy ground cover. Tie points between strips will occur with a frequency of at least one per frame. As a general rule, wing passpoints within lines of flight will also serve as tie points between strips.

4.05 Drop Points:

Drop points used to control lower altitude flights for direct compilation from photographs of larger scale will be located and labeled on the control photographs. These points should be marked, measured, and carried as extra passpoints in the aerial triangulation of the higher altitude photography.

4.06 Checkpoints:

Checkpoints are horizontal control points that have been established by ground control procedures throughout the block for accuracy checking purposes and should not be used in the analytical adjustment.

4.07 Diapositives:

a. Materials used.

All diapositives will be printed from original aerial photography negatives (i.e., not from duplicate negatives). Glass diapositives will conform to Kodak Ultra Flat Glass specifications for aerographic positive plates.

b. Printing and processing.

The printing and processing of all diapositives including development, fixation, washing, and drying must produce diapositives free from defects. Extreme care should be exercised to prevent lint from collecting on both the original negatives and the diapositives.

4.08 Point Marking:

a. General requirements.

All point marking (pugging) will be performed on the diapositives. Under no circumstances should any marking be performed on the original negatives. As a general rule targets will be marked, except targets which are exceptionally well defined on the diapositives.

b. Fully analytical aerial triangulation with mono-comparator.

All passpoints, checkpoints, and drop points (if any) should be marked stereoscopically on every frame with a Wild PUG point transfer instrument or equipment of equal or greater capability .

4.09 Aerial Triangulation Computations:

a. Computations will include ground control data, triangulated ground point residuals, triangulated ground points, and corrections applied to ground control.

b. Computed coordinates of all control points, passpoints, checkpoints, and drop points will be labeled on a computer printout with their field and/or computer designations.

The printout should include project name, date of computations, the name of the analytical program that was used to perform the computations, and the block or subblock designations. Root-mean-square (rms) error summaries should be given for bundle adjustment photographic measurement residuals or strip tie point residuals and misclosures at control points. All FAAT computations should be delivered to the contracting officer for review prior to any stereo compilation.

c. Narrative.

The report must include a brief narrative, including descriptions of laboratory equipment, procedures, and computer programs used. Of particular importance is a listing of field control points not used in the survey, any codes used in the computations, and a full description of significant misfits encountered at control points.

8.2.2.A5 Preparation of Base Maps

5.01 General:

The contractor should produce base maps in strict accordance with accepted stereophotogrammetric procedures, as published by the American Society for Photogrammetry and Remote Sensing, for each map scale. The maps (whether drawn, cartographic, or photographic image, orthophoto base maps) shall be compiled for each area at a scale equal to or larger than the scale required. Map compilation scales for the areas to be mapped will be limited to an approximate five-time enlargement over the scale of the aerial photography for each respective area. Example: a photo scale of 1″ = 2000′ (1 mm = 24 m) is the maximum scale allowed for a compilation scale of 1″ = 400′ (1 mm = 4.8 m). Only precision stereoplotting instruments of recognized proven accuracy shall be used in compilation work.

a. Coordinate grid ticks.

The plotted position of each plane coordinate grid tick shall not vary by more than 0.01 in. (0.25 mm) from true grid value on each manuscript map. These manuscript maps are used in scaling the stereo model to ground control. This is not to imply that the grid ticks plotted on the finished base maps (cartographic, orthophoto, or digital) are accurate to 0.01 in. (0.25 mm).

b. Horizontal control.

Horizontal control points and monuments shall be plotted to an accuracy of 0.01 in. (0.25 mm) of their true position as expressed by the plane coordinates for the control points.

c. Planimetric features.

Ninety percent of all planimetric features shall be plotted so their positions on the finished map will be accurate to within $^1/_{40}$-in. (0.64 mm) of their true coordinate positions; and none of the features shall be displaced on the finished map by more than $^1/_{30}$-in. (0.84 mm) from its true coordinate position.

5.02 Grid Lines and Values:

Grid line intersections of the State Plane Coordinate System are to be plotted at 5-in. (127-mm) intervals. These grid line intersections ½-in. (13 mm) in length, shall be shown throughout the neat image area. The State Plane Coordinates of the grid lines shall be printed in the margin along the image area border at the west (left) and north (top) sides of the map.

5.03 Match Notes:

"Match Notes" indicate the map numbers of the adjoining maps. These notes will be placed in the margin outside the image area.

5.04 Index of Deliverable Maps:

An index of maps to be delivered shall be prepared on a reproducible, polyester-based material (matte both sides) having a minimum thickness of 0.004 in. (0.10 mm). This map will show the individual map boundary outlines, map numbers, and existing and established monumented ground control.

8.2.2.A6 Orthophoto Base Maps

6.01 Final Orthophoto Base Maps:

Final orthophoto base maps shall be prepared by differential rectification. Mosaicking should not be permitted in the development of the orthophoto base maps. The outside edge dimensions of all orthophoto base maps shall be 30 in. (762 mm) high by 42 in. (1067 mm) long. The neat image area shall be 25 × 25 in. (635 × 635 mm) and the limit of the imagery shall extend 1½ in. (38 mm) beyond the neat image area on all sides of the maps. All ground level features on the final orthophoto maps should not be displaced from their State Plane Coordinate positions by more than $^1/_{30}$ in. (0.84 mm). When adjacent maps are aligned by grid ticks, imagery will not be misaligned by more than $^1/_{30}$ in. (0.84 mm), and on the majority of the maps, the imagery will be expected to exceed this accuracy.

6.02 Compilation:

Each stereo model to be scanned should be free of residual parallax and shall be leveled and scaled to plotted control. Ninety percent of all ground level features should be located so their positions on the finished map are accurate to within $^1/_{40}$ in. (0.64 mm) of their true coordinate positions. None of the features should be displaced on the finished map by more than $^1/_{30}$ in. (0.84 mm) from their true coordinate positions. Each stereo model should be scanned with a slit width compatible with the terrain and at a speed which achieves a good photo image and the accuracy referred to above. The Contractor should develop an orthophoto negative at least a two-times enlargement of the original aerial film negative.

6.03 Orthophoto Negatives:

For purposes of these specifications, "original orthophoto negatives" are defined as those negatives made directly from the diapositives used in the stereocompilation. The negatives should be free of scan lines, double exposures, out-of-focus images, mismatched imagery and inconsistencies in tone and density from one negative to an adjacent negative that may interfere with the interpretability of ground features or that are aesthetically objectionable. Original orthophoto negatives should not be cut in the process of mosaicking or for any reason. The corner grid positions of the neat image area of each orthophoto negative should be located and marked by pugging, pin pricking, or permanent ink; a minimum of two corner grid positions should be labeled with their State Plane Coordinates, outside of the image area.

6.04 Orthophoto Positives:

One unscreened reproducible positive of each formatted orthophoto map printed from the original orthophoto negative should be provided on polyester material with a minimum thickness of 0.004 in. (0.10 mm). All positives must be right-reading. Grid ticks on the positives will appear as white lines. The orthophoto positives shall be free of chemical stains; scratches; dust marks; and streaks, stains, and blemishes of any kind. Tone and density of the imagery shall be consistent from one map to the adjacent map. The photographic emulsion of the orthophoto positives shall be on the back side of the maps. Labeling such as match notes and map number shall be on the front side of the maps.

6.05 Border Data:

The following should be shown on all orthophoto maps: (1) title of map, (2) north arrow, (3) map number, (4) scale, (5) horizontal datum, (6) accuracy statement, (7) date of aerial photography, (8) name of mapping contractor, (9) match notes, (10) grid ticks, and (11) grid coordinate values (north coordinate values labeled across the west side of map and east coordinate values labeled across the north side of map).

8.2.2.A7 Topographic Maps

7.01 General:

A *topographic map* is an orthophoto or cartographic base map upon which contour lines and points of spot elevation have been depicted to show vertical datum changes in elevation of the ground surface. Elevations are normally based on the National Geodetic Vertical Datum, 1929 Adjustment. The topographic information may be depicted on an overlay which is registered to a base map.

7.02 Compilation:

The topographic map manuscripts may be compiled utilizing the techniques of Fully Analytical Aerial Triangulation (FAAT). For aerial photography at scales of either 1″ = 1200′ (1 mm = 14.4 m) or 1″ = 600′ (1 mm = 7.2 m), vertical ground control must be established in order to utilize FAAT, and no high-to-low bridging may be employed to obtain vertical control at these scales. When using FAAT for extension of vertical control, field positions should be established at a minimum of one point in every other stereo model along each flight line. Additional criteria include:

a. Contour lines.

Contour lines should be shown at a vertical interval of 2 ft (0.6 m) on 1″ = 100′ (1 mm = 1.2 m) scale topographic maps and at a vertical interval of 4 ft or 5 ft on 1″ = 200′ (1 mm = 2.4 m) maps. Every fifth contour line will be an index contour, indicated by a line heavier than that used for the intermediate contours. Index contours will be labeled inside a break in the contour line at a frequency that will permit ease of interpretation and that is aesthetically acceptable. All contour lines will be solid and unbroken except where they pass through dense ground cover, in which case dashed lines will be used. In those areas where vegetation prohibits accurate plotting of contour elevations, the contour line elevations will be interpolated as accurately as possible from spot elevations measured photogrammetrically in places where the ground is visible.

b. Spot elevations.

Spot elevations will be determined by photogrammetric procedures from the aerial photography that was used to produce the contour lines. The spot elevations will be placed on topographic maps at water levels of lakes, reservoirs, and ponds; hilltops; saddles; bottoms of depressions; intersections of principal streets and highways; and ends of bridges. Where contour lines are more than 1 in. (25 mm) apart, additional spot elevations can be shown to more accurately portray the slope of the land in such areas.

c. Accuracy requirements.

Vertical ground elevations generated from FAAT shall be third-order accuracy or higher.

d. Planimetric features.

The names of cities, towns, villages, rivers, major lakes, airports, streets, roads, railroads, and highways will be labeled on the topographic map to aid in orientation of the user.

e. Grid lines.

Grid lines at 5-in. (127-mm) intervals, running continuously across the topographic map, will be utilized rather than grid ticks and will constitute the outer boundaries of the map neat image area. Grid lines will appear as black lines on the finished (positive) topographic map.

f. Accuracy statement.

The following accuracy statement shall appear on each topographic map:

"Map complied by stereophotogrammetric methods in accordance with National Map Accuracy Standards."

g. Horizontal and vertical datum.

A note will be placed in the border of the map that will read as follows:

(Indicate proper zone) Zone 1983 North American Datum Vertical Datum 1929 Adjustment

h. Border data.

The following data shall be shown on the topographic map: map title, north arrow, grid coordinate values (labeled on all sides of map), match notes, horizontal and vertical datum, map scale, contour interval, map number, name of mapping contractor, date of aerial photography, accuracy statement, and map legend.

Chapter 8.3
SYSTEMS ENGINEERING

STANLEY C. SUBOLESKI, ROBERT E. CAMERON, AND ERIC K. ALBERT

8.3.1 INTRODUCTION

As Mutmansky (1973) has noted,

"The terms 'systems engineering,' 'systems analysis,' and 'operations research,' common in today's technical literature, often are interchanged because of their similar meaning. However, some differences exist . . ."

He then defines each term as follows. *Systems engineering* is the conception, planning, design, and engineering of any system of interacting elements so that the objective of the system is automatically optimized. Thus systems engineering normally is concerned with systems not yet activated. *Systems analysis* may be described as a scientific method of making decisions based on a quantitative or other objective evaluation of all action alternatives. It is very similar to systems engineering, but usually is not considered to be restricted to problems of design prior to use. While the previous two disciplines are somewhat vague and can encompass a great variety of engineering functions, *operations research* can be more specifically defined as the application of mathematical models to the problem of optimizing the objective of any predefined system.

Mutmansky further listed the techniques of operations research. This is largely reproduced as Table 8.3.1.

The techniques of operations research and systems engineering most widely used in practice within the mining industry are as follows:

1. Stochastic simulation.
2. Linear programming.
3. Network analysis/CPM (critical path method)/PERT (project evaluation and review technique).
4. Optimum pit limit techniques.
5. Nonstochastic modeling or simulation.
6. Structural/regression models.
7. Control charts.
8. Mixed-integer programming.

This chapter covers the principles of these systems used most widely in practice, along with applications and limitations of each. Examples of applications are also included in appropriate chapters of this *Handbook,* for example, 8.4 (Computer Methods), 9.4 (Cycles and Systems), and 12.6 (Monitoring, Control, and Communications).

Table 8.3.1. Compilation of the More Common Operations Research Techniques

Lagrange multipliers	Game theory
Mathematical programming methods;	Utility and decision theory
Linear programming	Inventory theory
Parametric programming	Response surface methods;
Nonlinear programming	Search methods
Stochastic programming	Evolutionary operation (EVOP)
Simulation methods	Simplex self-directing evolutionary operations (SSDEVOP)
Multistage optimization methods:	Sequencing theory
Markov processes	Network analysis:
Dynamic programming	Project evaluation and review technique (PERT)
Maximum principles of Pontryagin	Critical path method (CPM)
Queuing theory	
Expert systems	

Source: After Mutmansky, 1973.

8.3.1.1 General Method of Solution

Scientific problem solving comprises the following general steps:

1. Definition of the problem.
2. Construction of a model of the system.
3. Gathering data.
4. Testing of the model under known conditions.
5. Solution of the problem using the model.
6. Implementing the solution.

Often systems engineering problems are ill-defined or of such complexity that classical methods of solution cannot readily be applied. Direct optimization may not be possible; aid for making intelligent decisions may be the best that can be hoped for.

8.3.2 DATA COLLECTION

Often the most difficult part of systems engineering is the collection of data. Data are of four types (Manula, 1979):

1. *Static data* that do not change with time and space, e.g., length, weight, and capacity of a vehicle.
2. *Kinematic data* that deal with the motion of objects in time and space, e.g., speeds of trucks, shuttle cars, locomotives, etc.
3. *Dynamic data* that may change over time but do not necessarily involve space, e.g., methane emission, airflow rates in ventilation, etc.
4. *Statistical data* that deal with factors that follow no predictable pattern but can be studied using probability and statistical theory and by particular arrangement of measurements.

The most difficult problem lies in collecting statistical data. These problems concern determining:

1. What method to use.
2. When enough samples have been collected.
3. Whether to alter the raw data.
4. How to validate the data.

8.3.2.1 Collection Method

Generally the method of collection of kinematic or statistical data involves time studies or work sampling. The latter (also known as a "ratio-delay" study) is a sampling technique used to study elements that occur infrequently, thus making continuous observation impractical. Here the machines are observed at random intervals over a long period of time, and the percentage of time that the infrequent event (such as a specific type of breakdown) occurs is determined statistically.

Note that often historical records, such as breakdown reports, can be used to collect these data as well. At times, these records may be of doubtful accuracy, and work sampling may be used to validate the data.

Table 8.3.2. Time Study for Data Collection at the Loader in Room and Pillar Mining

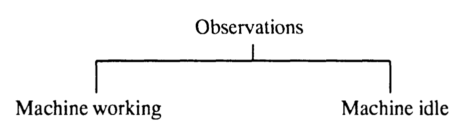
SC No.	Arrive Loader Empty	Start Load	Stop Load	Leave Loader Loaded	Load Time	Travel Loaded Time	Travel Empty Time	Delay Time	Change Out Dist.	Cut. No.: Start: Stop: Date: Obs: REMARKS
2		14.49		16.68					Start @ 115′	Start-#2 Entry
1	17.47		18.50							Tubing Fell
		19.76		20.32						Down: Gas Check
2	21.06			22.67						
1	24.97			26.62						
2	27.45	28.65	30.21	30.21						Extend Tubing
1	31.33			32.93						
2	34.60	35.11	36.72							Gas Check
1	38.52			40.05						
2	41.00	42.25		44.02						Extend Tubing
1	44.85			46.31						
2	48.95			54.84						Clean Up
2	59.39			62.52					End @ 134′	Finish Cut
										Clean Up ⅔ Car

Time Studies: *Time studies* are used to collect data on events that occur frequently enough that continuous, direct observations are practical. Time study forms for collecting and processing data for cable reel shuttle cars behind a loading machine (or continuous miner) in a room and pillar mine are shown in Tables 8.3.2 through 8.3.4. The hauling pattern and terminology are shown in the mining plan (Fig. 8.3.1). Tables 8.3.5 through 8.3.7 illustrate the completed calculation of rates, speeds, and capacities for the cut shown.

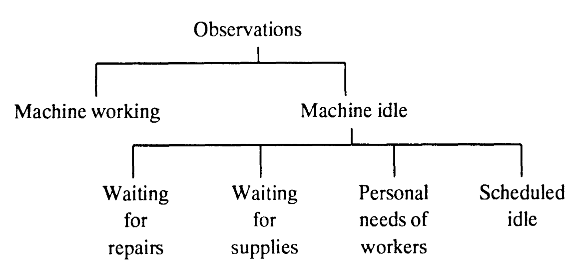
If the user is gathering data for a model, these data may be sufficient. However, if the purpose is to analyze the actual expenditure of time, then a summary sheet is ordinarily prepared showing the amount of time spent in various major time categories. In addition to a form showing the division of time, a second observation form is often used to help the user understand the special conditions encountered, as well as the qualitative factors at the production face.

An example of a very basic summary form is shown in Table 8.3.8. Fig. 8.3.2 shows a section observation and summary form, and Table 8.3.9 illustrates a summary form with additional section information that might be used as a supplement to the first two forms. Ordinarily, the time study would also include a map of the production section, showing the location of the loader and the dump, the routes of the load-haul-dumps (LHDs), trucks, or shuttle cars, and any other pertinent information helpful to the user of the time study.

Work Sampling: The simplest type of *work sampling* study is that of determining whether a given machine is idle or working. In such a case, the observations aim at detecting one of two possibilities only:



This simple model can be extended to determine the cause of the stoppage of the machine:



The objectives to be reached by the study will therefore determine the design of the recording sheet used in work sampling, as can be seen from Figs. 8.3.3 and 8.3.4.

Table 8.3.3. Time Study Form for Data Collection at the Change Point in Room and Pillar Mining

SC No.	Arrive CP Empty	Leave CP Empty	Arrive CP Loaded	Leave CP Loaded	Standby Time	Dist. Ch. Pt. to Dump	Cut No. Start: Stop: Date: Obs.: REMARKS
							Loading From #2 Entry
2	—		17.10			335′	
1	—	17.10	20.71				
2	20.46	20.71	23.08				
1	24.60		27.02				
2	26.85	27.02	30.83				
1	29.96	30.83	33.38				
2	34.15		37.25				
1	38.05		40.55				
2	40.10	40.55	44.49				
1	43.61	44.49	46.67				
2	48.45		55.41				
1	49.97	—					Cable Blown, Car Down
2	58.16	59.03	3.00				Operator Helps Pull Cable
							Last Car In Cut-⅔ Car

8.3.2.2 Number of Samples

For time study elements, it is seldom necessary to calculate the minimum number of samples needed since these are usually exceeded. This can be checked using several formulas based on normal distributions (i.e., the central limit theorem), large sample sizes, and the concept of confidence limits. One such formula, based on 95% confidence limits, is

$$N = [40\ (s/\bar{x})]^2 \tag{8.3.1}$$

where N is sample size needed, $\bar{x}$ is sample mean, S is sample standard deviation, and $S/\bar{x}$ is the coefficient of variation (the standard deviation expressed as a percentage of the mean). For 99% confidence limits, this becomes

$$N = [60\ (s/\bar{x})]^2 \tag{8.3.2}$$

The standard deviation of the sample may be calculated from the formula for sample variance:

$$S^2 = \frac{\Sigma\ (x_i - \bar{x})^2}{n - 1} \text{ or } S^2 = \frac{\Sigma\ x_i^2 - [(\Sigma x_i)^2/n]}{n - 1} \tag{8.3.3}$$

or may be estimated approximately as the range R divided by 4, or

$$S = R/4 \tag{8.3.4}$$

In general,

$$\sqrt{N} = \frac{t}{e}\left(\frac{S}{\bar{x}}\right) \tag{8.3.5}$$

where t is the number of standard deviations corresponding to the desired confidence limits (for 95%, t = 2; for 99%, t = 3), and e is the percentage error from the mean desired or allowed.

The chart shown in Fig. 8.3.5 may be used in lieu of the formula.

Work Sampling: In work sampling, the number of samples is more critical (Manula, 1979). Here the number of samples is based on the binomial distribution and its approximation of the normal distribution. The technique is to first take a (large) number of preliminary observations and determine the percentage of time the event being studied occurs. Then using the nomogram in Fig. 8.3.6, and the error desired, the final number of observations needed may be read.

Example 8.3.1. (Manula, 1979). Suppose a study is conducted to determine the downtime on a machine. Assume that some 100 observations were recorded as a preliminary study, and that these showed the machine to be idle in 25% of the cases (p = 25). In order to determine the value of n at the 95% confidence level with a ± 5 % margin of error, about 310 observa-

Table 8.3.4. Time Study Form for Data Collection at the Discharge Point in Room and Pillar Mining

SC No	Arrive Dump	Start Dump	Stop Dump	Leave Dump	Travel Loaded Time	Travel Empty Time	Dump Time	Delay Time	Cut No.: Start: Stop: Date: Obs: REMARKS
									Belt Feeder Stalling
									Dumping at Low Rate
2	18.38		19.60						
1	22.80		23.70						
2	24.60	24.93	26.04						Talk to Observer
1	27.97		29.00						
2	32.38		33.30						
1	34.34		35.50						
2	38.33		39.28						
1	41.52		42.62						
2	45.56		46.45	47.38					Operator Stops For Water
1	47.50	48.15	48.93						Delay-Wait On #2 Car
2	56.35		57.10						
2	40.08		5.11	9.23					Operator Cleans Spillage

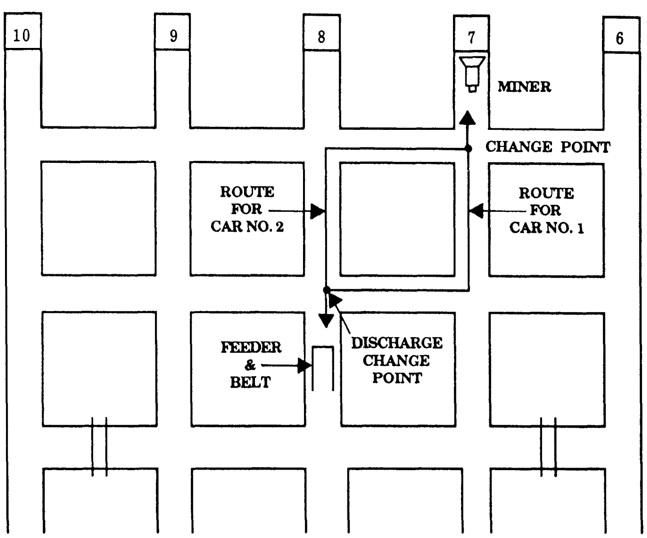


Fig. 8.3.1. Standard hauling pattern with cable reel shuttle cars.

tions are needed (Fig. 8.3.6). Note that a 10% error requires 75 observations.

8.3.3 ANALYTICAL TECHNIQUES

Analytical techniques are defined here as those methods that do not result in optimization as a *direct* result of calculations. In general, these methods tend to have less restrictive conditions and are able more realistically to approximate a greater number of operational problems. *Stochastic simulation* is the most widely used technique of this type today. Other methods used fairly widely are control charts, nonstochastic models and statistical (usually regression) models. The application of each of these is covered in the sections following.

8.3.3.1 Statistical (Linear Regression) Models

Linear regression analysis has become one of the most widely used statistical tools for analyzing multifactor data. It is appealing because it provides a conceptually simple method for investigating functional relationships among variables. The standard approach in regression analysis is to use sample data to compute an estimate of the proposed relationship, and then evaluate the fit using statistics such as t, F, and R^2 (Chatterjee and Prince, 1977).

For example, in mining, productivity depends to some degree on the dimensions of the mining plan, the seam thickness, top and bottom conditions, equipment performance characteristics, and possibly other physical factors. If equations could be developed to predict productivities in advance of mining based on these simple measurements, the problems of mine planning, design, and economic justification would be greatly simplified.

Thus the solution would be proposed as a linear equation of the form,

Table 8.3.5. Processed Data from Tables 8.3.2 and 8.3.3

SC No.	Arrive Loader Empty	Start Load	Stop Load	Leave Loader Loaded	Load Time	Travel Loaded Time to CP	Travel Empty Time From CP	Delay Time	Change Out Dist.	Cut. No.: Start: Stop: Date: Obs: REMARKS
2		14.49		16.68	2.19		—		Start @ 115′	Start-#2 Entry
1	17.47		18.50			0.42	0.37	1.26	116.6	Tubing Fell
		19.76		20.32	1.59	—	—			Down: Gas Check
2	21.06			22.67	1.61	0.39	0.35		118	
1	24.97			26.62	1.65	0.41	0.37		120	
2	27.45	28.65	30.21	30.21	1.56	0.40	0.43	1.20	121	Extend Tubing
1	31.33			32.93	1.60	0.62	0.50		123	
2	34.60	35.11	36.72		1.61	0.45	0.45	0.51	125	Gas Check
1	38.52			40.05	1.53	0.53	0.47		126	
2	41.00	42.25		44.02	1.77	0.50	0.45	1.25	128	Extend Tubing
1	44.85			46.31	1.46	0.47	0.36		129′	
2	48.95			54.84	5.89	0.36	0.50		131′ 132.4	Clean Up
2	59.39			62.52	3.13	0.57	0.42		End @ 134′	Finish Cut
						0.48			134′	Clean Up ⅔ Car

$$Prod = A_1 + A_2 * THICK + A_3 * TOP + A_4 * BOTTOM, \text{ etc.} \qquad (8.3.6)$$

where A_1 is a constant, and A_2, A_3, etc., are the coefficients determined from the regression analysis; *THICK, TOP, BOTTOM,* etc., are the values for the thickness, top condition, bottom condition, etc., for the mine being evaluated; and *Prod* is the resultant productivity. Data would then be collected on these factors and the resultant production from as many operations as possible. Regression analysis would then be conducted to find the linear coefficients that best fit the data, that is, minimize the sum of squares error.

Serious regression analysis cannot be undertaken without the aid of computer programs, many of which are now available for personal computers. The role of the engineer is then to insure that the problem is adequately stated (i.e., that the significant independent variables are included), and that no serious errors mar the analysis. Likely error sources are:

1. Nonlinear relationships between the independent and dependent variables.
2. Outliers which significantly alter the regression results.
3. Collinearity among independent variables.
4. Autocorrelation within one or more independent variables.
5. In general, any nonrandomness in residuals caused by problems in the underlying model or data.

Often a secondary problem is to then choose an equation with the fewest number of independent variables to adequately explain the variation in the dependent variable. This involves both tests of significance and judgment on the part of the model builder. In addition to standard techniques of selection, such as forward and backward stepwise regression, several computer packages also offer the user information about all combinations of the variables, so that he may apply his practical knowledge as much as possible. For a more complete discussion, see Chatterjee and Prince (1977) or Draper and Smith (1966). Regression models have these general uses:

1. Description and model building: to help in the understanding of complex processes or interactions.
2. Estimation and prediction: to predict future values.
3. Control: to determine the change in independent variables needed to obtain a specified value of the dependent variable.

8.3.3.2 Control Charts

SQC or *statistical quality control* charts are used fairly widely in the mining industry to measure and control product quality. The primary purpose of a statistical control chart is to determine whether or not a stable system exists, that is, whether or not the system is regularly meeting statistical expectations or is in a state of statistical control. A second use is as the basis for making real-time decisions. Corrective action may be taken on the basis of the following:

1. Point outside the control limits.
2. Run of seven or more observations in a continuous trend (up or down).
3. Run of seven or more observations, all of which are either above or below the central line.

Table 8.3.6. Processed Data from Table 8.3.3

Arrive CP Empty	Leave CP Empty	Arrive CP Loaded	Leave CP Loaded	Standby Time	Dist. Ch. Pt. to Dump	Cut. No.: Start: Stop: Date: Obs: REMARKS
						Loading From #2 Entry
—		17.10			335′	
—	17.10	20.71				
20.46	20.71	23.08		0.25		
24.60		27.02		—		
26.85	27.02	30.83		0.17		
29.96	30.83	33.38		0.87		
34.15		37.25		—		
38.05		40.55		—		
40.10	40.55	44.49		0.45		
43.61	44.49	46.67		0.88		
48.45		55.41		—		
49.97	—			—		Cable Blown, Car Down
58.16	59.03	3.00		—		Operator Helps Pull Cable
						Last Car In Cut ⅔ Car

Two major types of statistical quality control charts are used. These are charts for means and charts for ranges. They are usually used together and are often constructed on the same page. An example of a control chart form used by Ford Motor Co. is shown in Fig. 8.3.7. (Anon., 1987).

Control Charts for Means: With a control chart for means, the vertical scale is the observed value of $\bar{X}$, and the horizontal scale indicates the time at which the sample was taken. A solid line is drawn through the standard mean, and two parallel dashed lines are drawn above and below the solid line, indicating the statistical control limits. If the observed $\bar{X}$ lies between the dashed lines, and no strong trends are evident, the production process is in control, and no action is required. If the point or points indicate the process is out of control, an attempt is made to find the factor that is causing these extreme observations. In Example 8.3.2, discussed later, the process was out of control on the fifth day.

Control Chart for Ranges: A second control chart to record dispersion is usually kept simultaneously with the chart for means. The dispersion is most often measured by the sample range R. If $\bar{R}$ is the expected range, then the control limits are $D_L\bar{R}$ and $D_U\bar{R}$, the lower and upper control limits, respectively. These multipliers are usually computed to give a 1% chance of the sample range being less than D_L or exceeding D_U (i.e., a 0.5 and 99.5 percentile of the sampling distribution of the statistic R).

Construction of Control Charts—Control charts are based on the central limit theorem and are usually constructed at the $3s$ limits.

To construct a control limit for means, a small number (n = 5 is typical) of samples is taken at close time intervals when the process is in control. Each five samples forms a subgroup, and N (N = 20 to 25 is typical) subgroups are collected.

When m and s are not known, the usual case, they must be estimated by sampling to establish the control limits.

Let i equal the sample (subgroup) number, i = 1,2,3. . . .N, x_i equal to the mean of the ith sample, and R_i equals the range of the ith sample. Then the

$$\textit{grand mean} = \overset{*}{x} = \frac{\sum_{i=1}^{N} \bar{x}_i}{N} \tag{8.3.7}$$

and

Table 8.3.7. Processed Data from Tables 8.3.3 and 8.3.4

SC No.	Arrive Dump	Start Dump	Stop Dump	Leave Dump	Travel Loaded Time	Travel Empty Time	Dump Time	Delay Time	Cut No.: Start: Stop: Date: Obs: REMARKS
									Belt Feeder Stalling
									Dumping At Low Rate
2	18.38		19.60		1.28	0.86	1.22		
1	22.80		23.70		2.09	0.90	0.90		
2	24.60	24.93	26.04		1.52	0.81	1.11	0.33	Talk to Observer
1	27.97		29.00		0.95	0.96	1.03		
2	32.38		33.30		1.55	1.15	0.92		
1	34.34		35.50		0.96	2.55	1.06		
2	38.33		39.28		1.08	0.82	0.95		
1	41.52		42.62		0.97	0.99	1.10		
2	45.56		46.45	47.38	1.07	1.07	0.89	0.93	Operator Stops For Water
1	47.50	48.15	48.93		1.03	1.04	0.78	0.65	Delay-Wait On #2 Car
2	56.35		57.10		0.94	1.06	0.75		
2	4.08		5.11	9.23	1.08	—	1.03	4.12	Operator Cleans Spillage

SAMPLE CALCULATIONS:
1. TRAVEL LOADED: (ARR DUMP − LV CP LDD)
or 18.38 − 17.10 = 1.28 MIN
2. TRAVEL EMPTY: (ARR CP EMPTY − LEAVE DUMP)
or 20.46 − 19.60 = 0.86 MIN
FOR SPEEDS DIVIDE DISTANCE TO DUMP BY TRAVEL TIME:
335 FT ÷ 1.28 MIN = 262 FT/MIN

$$\textit{expected range} = \bar{R} = \frac{\sum_{i=1}^{N} R_i}{N} \tag{8.3.8}$$

are calculated. These are then used to construct the control charts. For the control chart for mean (**$\bar{x}$-chart**):

$$\text{Centerline} = \overset{*}{x} \tag{8.3.9}$$

$$\text{Upper control limit} = \overset{*}{x} + A_2\bar{R} \tag{8.3.10}$$

$$\text{Lower control limit} = \overset{*}{x} - A_2\bar{R} \tag{8.3.11}$$

For the control chart for ranges (R-chart):

$$\text{Centerline} = \bar{R} \tag{8.3.12}$$

$$\text{Lower control limit} = D_3\bar{R} \tag{8.3.13}$$

$$\text{Upper control limit} = D_4\bar{R} \tag{8.3.14}$$

Values for A_2, D_3, and D_4 are shown in Table 8.3.10.

It should be noted that:

1. "Out-of-control" readings should be excluded in calculating control limits.
2. A control chart assumes "normality" of data; other techniques are available for nonnormal data.
3. Charts and limits should be updated at regular intervals as more experience is accumulated.

The control limits on the mean define the *capability* of the process. Capability can then be compared with the *requirements* to determine if these can be met when the process is operating in statistical control.

Example 8.3.2. (Buffa, 1972). A study is undertaken to determine the time required to perform a certain repetitive task. Among other data gathered for the work measurement study are

Table 8.3.8. Basic Time Study Summary Form

Operational Study on Section No. xx

	FREQUENCY	ADJ. MIN. (Whole Nos.)	% AVAIL. TIME (Carried to 1/10)
Productive Time			
Load	105	168	37.9
Car change	98	128	28.9
Maneuver with car	14	12	2.7
Total		308	69.5
Necessary Time			
Safety meeting	1	7	1.6
Maneuver miner to face	1	3	.7
Timber with car	4	13	2.9
Extend line brattice	1	3	.7
Handle cable	3	10	2.2
Handle supplies	5	11	2.5
Service miner	1	8	1.8
Maneuver miner from face	1	3	.7
Place change	7	54	12.2
Total		112	25.3
Unnecessary Time			
Replace cover on ripper head	1	3	.7
Let roof bolter pass	2	3	.7
Miner lost power	1	3	.7
Rock on head of miner	1	6	1.3
Rehang miner cable	1	2	.5
Move timbers out of roadway	1	6	1.3
Total		23	5.2
Mantrip—In		12	
Out		17	
Prepare for Production		8	
GRAND TOTAL		480	100%

100 stopwatch readings of the actual time required for the task. Table 8.3.11 shows these 100 readings in minutes, divided into the subsamples of $n = 5$ by which they were gathered. Each subgroup of five readings was taken at random times in the order of the sample numbers as shown.

Solution. Table 8.3.12 shows the sample means for each of the 20 samples, the grand mean of which is $\bar{\bar{x}} = 2.01$ min. Table 8.3.13 shows the range in minutes for each of the 20 samples, and the average of the ranges is calculated as $\bar{R} = 0.43$ minutes. The preliminary control limits for the $\bar{X}$ chart are then calculated as follows (the value of $A_2 = 0.58$ is obtained from Table 8.3.10:

$$UCL = \dot{x} + A_2\bar{R}$$
$$= 2.01 + (0.58 \times 0.43) = 2.26 \text{ min}$$
$$LCL = \dot{x} - A_2\bar{R}$$
$$= 2.01 - (0.58 \times 0.43) = 1.76 \text{ min}$$

These preliminary control limits and the preliminary centerline for the grand mean are plotted in Fig. 8.3.8a together with the 20 sample means shown in Table 8.3.12. The chart seems to indicate that there is a stable data generating system with the exception of sample No. 18 that falls below the lower control limit line. An investigation reveals the fact that the operator had been following a nonstandard method at that time because of a material shortage. Since it was not in control and an assignable cause was determined, sample No. 18 was eliminated from the data, and new revised grand mean and control limits computed as shown. The elimination of sample No. 18 also affects $\bar{R}$, which enters into the calculation of revised control limits. The control limits for the R chart are determined in a similar way:

$$UCL_R = D_4\bar{R}$$
$$= 2.115 \times 0.43 = 0.91 \text{ min}$$
$$LCL_R = D_3\bar{R}$$
$$= 0 \times 0.43 = 0$$

Fig. 8.3.8b shows the preliminary control limits for the R chart as just calculated. The centerline for the R chart is the average of the sample ranges, $\bar{R} = 0.43$ min. The 20 sample ranges are shown in Table 8.3.5, and we see throughout the 20 samples that variability within the samples remained stable within the control limits. Nevertheless, since sample No. 18 has been eliminated on the basis of the $\bar{X}$ chart, it must also be eliminated to calculate revised control limits for the R chart. The revised centerlines and control limits for the two charts of Figs. 8.3.8a and 8.3.8b now represent reasonable standards for comparison of future samples. The basic calculations for determining the control limits and centerlines for $\bar{X}$ and R charts remain the same, regardless of the variable being measured.

8.3.3.3 Mathematical Models of Loading and Hauling

Deterministic models of loading and hauling in underground or surface mines can be used to predict *changes* in these systems

Date________ Shift________ Mantrip Time: (In)________ (Out)________
Mine________ Prepare for Production: (Start)________ (End)________
Section________ First Trip Loaded: (Time)________ (No. Cars)________
Type Mining________ Seam Height________ Type Roof________
Foreman________ No. of Men________ Observer________

SECTION EQUIPMENT AND COMPANY NUMBER

Continuous Miner	Shuttle Cars	Roof Bolter	Car Spotter	Haulage
________	________	________	________	________

PRODUCTION DATA

No. of Mine Cars Loaded________ Tons/Mine Car________ Tons Loaded________
No. Shuttle Cars Loaded________ Tons/Shuttle Car________ Loading Rate________

LOADING GROUP

1.	Loading Time/SC	____	____	____	____	____	____
2.	Sump Depth	____	____	____	____	____	____
3.	Place Change Time	____	____	____	____	____	____

			(GOOD)	(FAIR)	(POOR)
4.	Miner operator bulldoze spillage during car change?	4.	____	____	____
5.	Miner operator maneuver, break coal down during SC change?	5.	____	____	____
6.	Communications - miner operator and shuttle car operator.	6.	____	____	____
7.	Condition of cutter bits.	7.	____	____	____
8.	Voltage.	8.	____	____	____
9.	Coordination of trip change and place change.	9.	____	____	____

			(YES)	(NO)
10.	Was preparation made to handle & suspend cable prior to place change?	10.	____	____
11.	Did operator take too much cable to face?	11.	____	____
12.	Were cable hooks on miner used during place change?	12.	____	____
13.	Was pre-examination made by other personnel prior to miner entering?	13.	____	____
14.	Were safety jacks available in place?	14.	____	____
15.	Did the miner have adequate water sprays and quantity of water?	15.	____	____
16.	Was line brattice maintained the prescribed distance from the face?	16.	____	____
17.	Were centerlines marked in advance for the miner?	17.	____	____
18.	Were safety jacks installed upon completion of one side of the cut?	18.	____	____
19.	Were supplies pre-delivered for roofbolting, timbering, etc.?	19.	____	____
20.	Were supplies delivered so as to minimize delay to the miner?	20.	____	____
21.	Did miner operator always know which place was to be mined next?	21.	____	____
22.	Were supplies for servicing miner delivered in advance?	22.	____	____

23. Service Miner: No. of men utilized________ Total time required________
24. How many splices in miner cable? ________

COMMENTS: ________

____

Reviewed with: ________

Fig. 8.3.2. Time study observation and summary form.

with considerable accuracy. Prediction of absolute productive capability depends not only on the loading and hauling capacity but also the equipment and manning availability, as well as other factors which are largely a function of management and mining conditions.

Underground Mining: In room and pillar operations using either a loader and trucks, loader (which may be a continuous miner) and shuttle cars, or LHD units, the haulage system is usually as shown in Fig. 8.3.9. (Note that if the haulage vehicles can pass in a roadway, the haul distances are always equal.) As noted previously, management variability makes it difficult to predict the production that may be expected from a given plan and set of equipment. However, the production *rate* may be predicted readily for any given cut (or series of cuts) if average values of elemental operations are used. If it is assumed that the loader will complete any maneuver during the time the shuttle cars and trucks are changing out, the loader cycle time will consist of only five elements (Prelaz et al., 1964):

1. *Loading* (*TLT*). The time spent in loading a cut. Loading time begins at the instant the shuttle car (or truck) arrives at the back of the loader, and ends when the car is fully loaded.

2. *Changeout Time* (*COT*). The time spent waiting for the cars to travel over the common changeout distance. For systems with sufficiently wide entries, this may equal zero, equal the loader maneuver time, or equal the spot time of a truck.

3. *Waiting on a Shuttle Car* (*WSC*). The time during which a loaded car has cleared the change point and an empty car is not available at the change point.

4. *Tramming* (*TT*). Traveling from the face of one cut to the face of another, including all delays, such as hanging cables or changing ventilation controls.

5. *Waiting on a Place* (*WC*). Waiting because the next support operation has not completed preparation of the face for the loader.

Note that a sixth factor, *In Cut Delays* (*ICD*) may need to be added if the loader cannot complete maneuvers between cars. Measured individual *ICD* times should be decreased by the changeout time(s) before adding them to the cycle time. To predict total production, *Abnormal Delays* and *Fixed Delays* (Unavailable Time) must be accounted for. As noted earlier, these are site specific and, to a large extent, a function of management.

Table 8.3.9. Supplemental Time Study Summary Form

Loading Data Shift		Actual
1. No. Shuttle Cars		105
2. No. Mine Cars		64
3. Tons Loaded		512
4. Loading Rate TPM—High		4.80
	Low	1.91
	Avg.	3.05
5. No. of Men		7
6. Tons per Man		69.71
7. Tons/S.C.		4.88
8. Tons/Mine Car		8.0
9. Condition of Roadway		Muddy and rough with pot holes
10. Type Roof Support		4′ & 7′ bolts on 4′ centers with 2x8x24″ boards
11. Type Rib Support		None

Section Conditions

Place Name	Loading Sequence	Condition Beginning of Shift	Condition End of Shift
No. 3 Room Pillar	1 - 2 - 3	Roof bolted	Finished
No. 2 Room Pillar (Push)	5	Timbered	Finished
No. 1 Room Pillar (Pocket & Wing Rt.)	4 - 7 - 9	Roof bolted	Push (Timbered)
No. 1 Room Pillar (Pocket Left)	6 - 8	Roof bolted	Roof bolted

Avg. Height of Place 58″

Avg. Width of Place 17′

HOURLY SC LOADING

	1st Hour		2nd Hour		3rd Hour		4th Hour		5th Hour		6th Hour		7th Hour		8th Hour	
	Std.	Act.	Std.	Act.	Std.	Act.	Std.	Act.	Std.	Act.	Std.	Act.	Std.	Act.	Std.	Act.
Sub Total		8		15		13		11		9		19		20		10
Running Total		8		23		36		47		56		75		95		105

Date: Observer: Study No.:		
Number of observations: 75	Total	Percentage
Machine running: 卌 卌 卌 卌 卌 卌 卌 卌 卌 卌 卌 卌 ||	62	82.7
Machine idle: 卌 卌 |||	13	17.3

Fig. 8.3.3. Example of a simple work sampling record sheet (Manula, 1979).

Date: Observer: Study No.:				
Number of observations: 75			Total	Percentage
Machine running		卌 卌 卌 卌 卌 卌 卌 卌 卌 卌 卌 卌 ||	62	82.7
Machine idle	Repairs	||	2	2.7
	Supplies	卌 |	6	8.0
	Personal	|	1	1.3
	Idle	||||	4	5.3

Fig. 8.3.4. Work sampling record sheet showing machine utilization and distribution of idle time (Manula, 1979).

Fixed delays include:
Travel In and Out
Lunch
Scheduled Machinery Servicing
Prepare to Mine
Prepare to Leave

Thus, if Abnormal and In Cut Delays are disregarded, the loader cycle time for a cut may be stated as follows:

$$LCT = TLT + COT + WCS + TT + WC \qquad (8.3.15)$$

To calculate these terms, the following are used:

$$TLT = TONS \div LR \qquad (8.3.16)$$

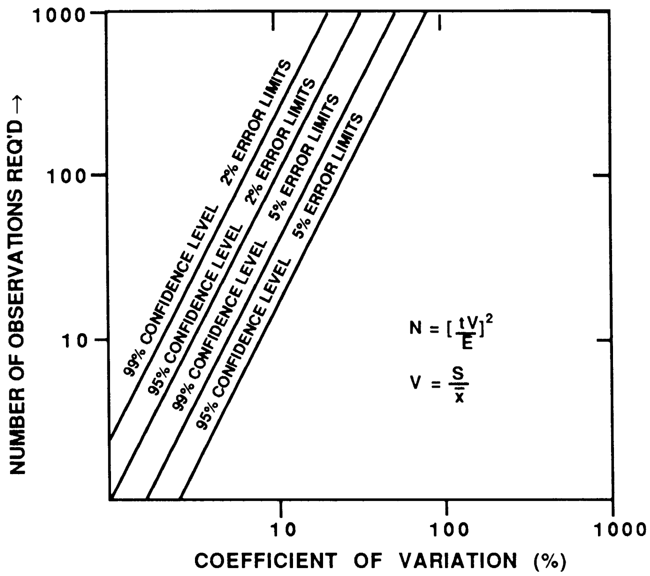


Fig. 8.3.5 Logarithmic plot of the relative dispersion to determine sample size (Manula, 1979).

$$COT = (n - 1)(O + I) \qquad (8.3.17)$$

$$W = TL + DT + TE - (N - 1)(L + O + I) \qquad (8.3.18)$$

$$WSC = \left[\frac{n - 1}{N}\right] W \qquad (8.3.19)$$

$$TT = (TD \times TR) + TC \qquad (8.3.20)$$

where *TONS* = tons in the cut, *LR* = loading rate (or mining rate), *COT* = changeout time, *O* = time to travel from the loader to the change point, *I* = time to travel from the change point to the loader, *L* = time to load one shuttle car, *n* = number of shuttle-car loads in the cut, N = number of shuttle cars used in the analysis (in this case = 2), [ ] = denotes the enclosed value should be truncated to an integer value (i.e., [3.83] = 3), *TE* = time to travel from the dump to the change point, *TL* = time to travel from the change point to the dump, *TR* = tram rate of the loader, *TC* = nontram delay time encountered which during tramming, and *W* = wait time for a shuttle car to reach the change point.

Note that it is also assumed that all haulage units have the same payload and speed characteristics.

While this equation may be used to quickly predict the production from a cut, it does not include the Wait on Place time that depends on the relative speeds of the support operations (i.e., "bottleneck" delays) and thus affects the overall production from a system. In addition, with a deterministic model, the effects of randomness are lost, and any waiting that may occur at the discharge change point, usually minor, is ignored.

Application—In ore mines or with long hauls in general, the speeds must either be time studied or estimated from rimpull-speed-gradeability curves (Chapter 13.1.2.). Table 8.3.14 lists average speeds for LHD units in underground mines. For short hauls (< 500 ft or < 150 m) or when electric-powered vehicles are used, Table 8.3.15 may be used to estimate speeds in firm, dry roadways. LHDs, trucks, and ram-type cars must usually turn at the change point and dump as part of their haul cycle. In coal mines, the "turn-car" time for battery cars in good conditions is 0.15 to 0.25 min per event.

Loading rates for front-end loaders may be calculated from cycle-time charts (Chapter 13.3). LHD loading times will vary with the material being loaded and the condition of the shot. A study of LHDs in underground lead mines (Bullock, 1975) showed an average load time of 1.16 min, including nonhaulage face delays. Stevens and Acuna (1982) note a range of 0.25 min for single-pass loads in good condition to 2 min for three-pass loads in poor conditions. Coal-mine loading machines typically load at 10 to 15 tpm (9 to 14 t/m), except for cleanup cars, which average 2 to 4 tpm (1.8 to 3.6 t/m). Continuous miners can load at similar rates but usually perform closer to their mining rate, which typically range from 5 to 10 tons/min (4.5 to 9 t/m), except for cleanup rates of 2 to 4 tpm (1.8 to 3.6 t/m).

Dump times are typically 30 sec for LHDs and trucks and 30 to 45 sec for chain-bed shuttle cars with unrestricted dumping. Turn and spot times must be added to these where appropriate. Table 8.3.16 shows recommended load and dump times for LHDs.

Payloads for trucks must be determined case by case, based upon the opening dimensions, material density, and truck dimensions. LHD payloads likewise must be estimated for specific cases. Table 8.3.17 lists rated payloads and dimensions for some LHD units. Fig. 8.3.10 shows the results of a study of payloads for shuttle cars with 48-in. (122-cm) conveyors in 15 coal mines. In general, ram-dump cars will have payloads 20 to 40% larger than this. Payloads for shuttle cars with wider conveyors should be increased proportionately.

Example 8.3.3. The following example illustrates the application of the cycle time model in coal. It is desired to calculate the expected miner cycle time for a cut assuming the vent tubing advances, etc., can be accomplished during haulage delays to the miner. Data regarding the cut and equipment are as follows:

seam thickness *H*	5.5 ft (1.6 m)
entry width *W*	18 ft (5.5 m)
cut depth *D*	18 ft (5.5 m)
coal density *d*	85 lb/ft^3 (1362 kg/m^3)
number of cars in use *N*	2
shuttle car payload *SCP*	5.0 ton (4.5 t)
loading rate *LR*	6 tpm (5.4 t/min)
average car speeds *SE SL*	300 fpm (91 m/min)
change out distance *CPD*	100 ft (30 m)
haul distance *HD*	300 ft (91 m)
dump time *DT*	0.7 min
tram time *TT*	20 min

Solution.

$$TONS = 18 \times 8 \times 5.5 \times 85/2000 = 75.7 \text{ ton/cut}$$
$$L = 5/6 = 0.83 \text{ min}$$
$$TLT = 75.7/6 = 12.6 \text{ min}$$
$$I = COD/SL = 100/300 = 0.33 \text{ min}$$
$$O = COD/SE = 100/300 = 0.33 \text{ min}$$
$$N = TONS/SCP = 75.7/5 = 15.1, \text{ say } 15$$
$$COT = (N - 1)(O + I) = 14(0.67) = 9.38 \text{ min}$$
$$TE = HD/SE = 300/300 = 100 \text{ min}$$
$$TL = HD/SL = 100 \text{ min}$$
$$WSC = \left[\frac{15 - 1}{2}\right](1.0 + 0.7 + 1.0 - ((2 - 1)(0.33 + 0.33 + 0.83))$$
$$= (7)(1.2) = 8.40 \text{ min}$$

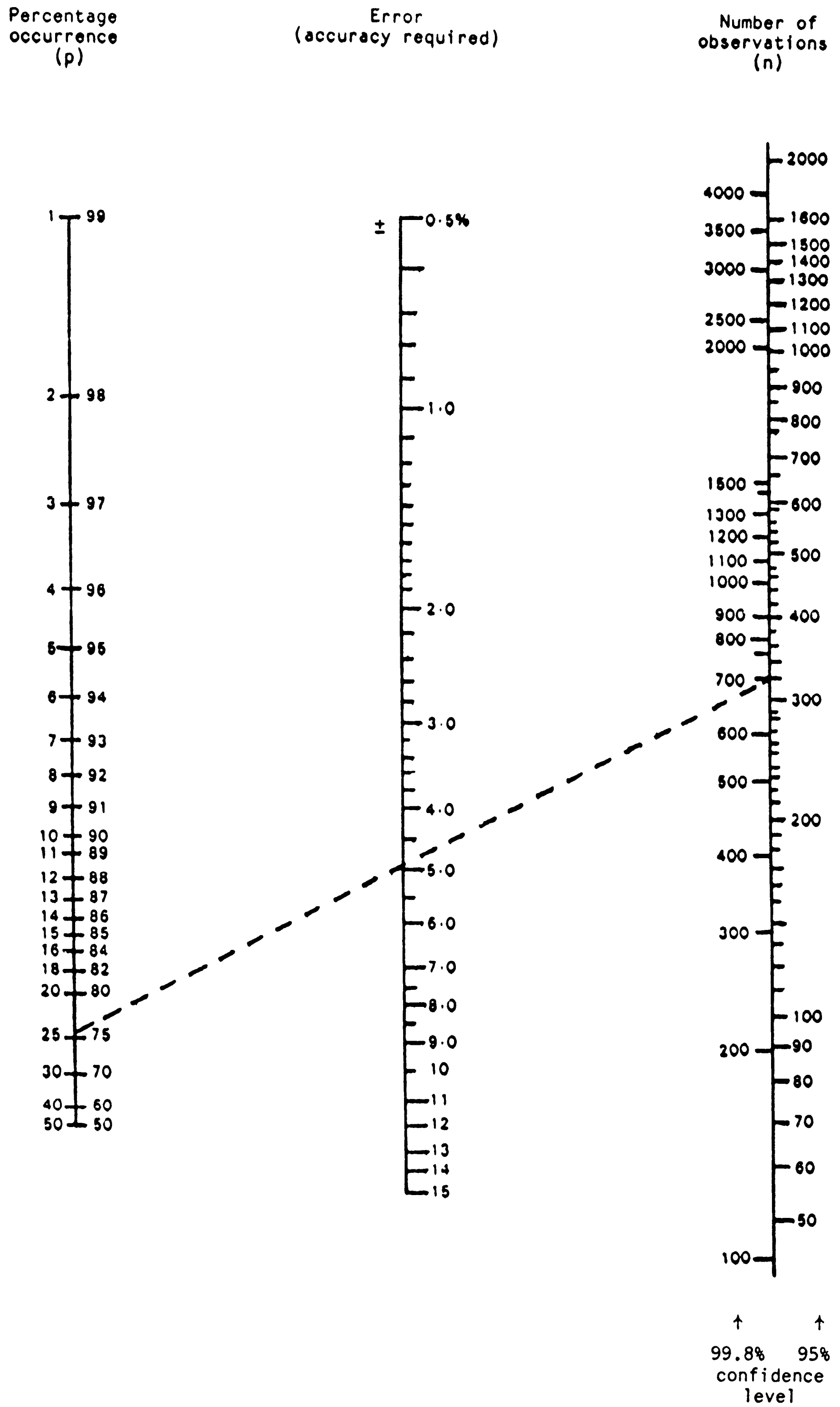


Fig. 8.3.6. Nomogram for determining number of observations (Manula, 1979).

$$LCT = 12.6 + 9.38 + 8.40 + TT$$
$$= 30.38 \text{ min}$$

Surface Mining: The previous model is applicable to analysis of truck-shovel operations in surface mining, with the following change in terms: I = spot time of the truck, and O = time to clear the loader area such that the next truck to be loaded may spot.

The model may be used to estimate average production rates from a cut just as in underground cuts. Alternatively, the number of trucks needed to insure the loader does not wait may be solved for. Since

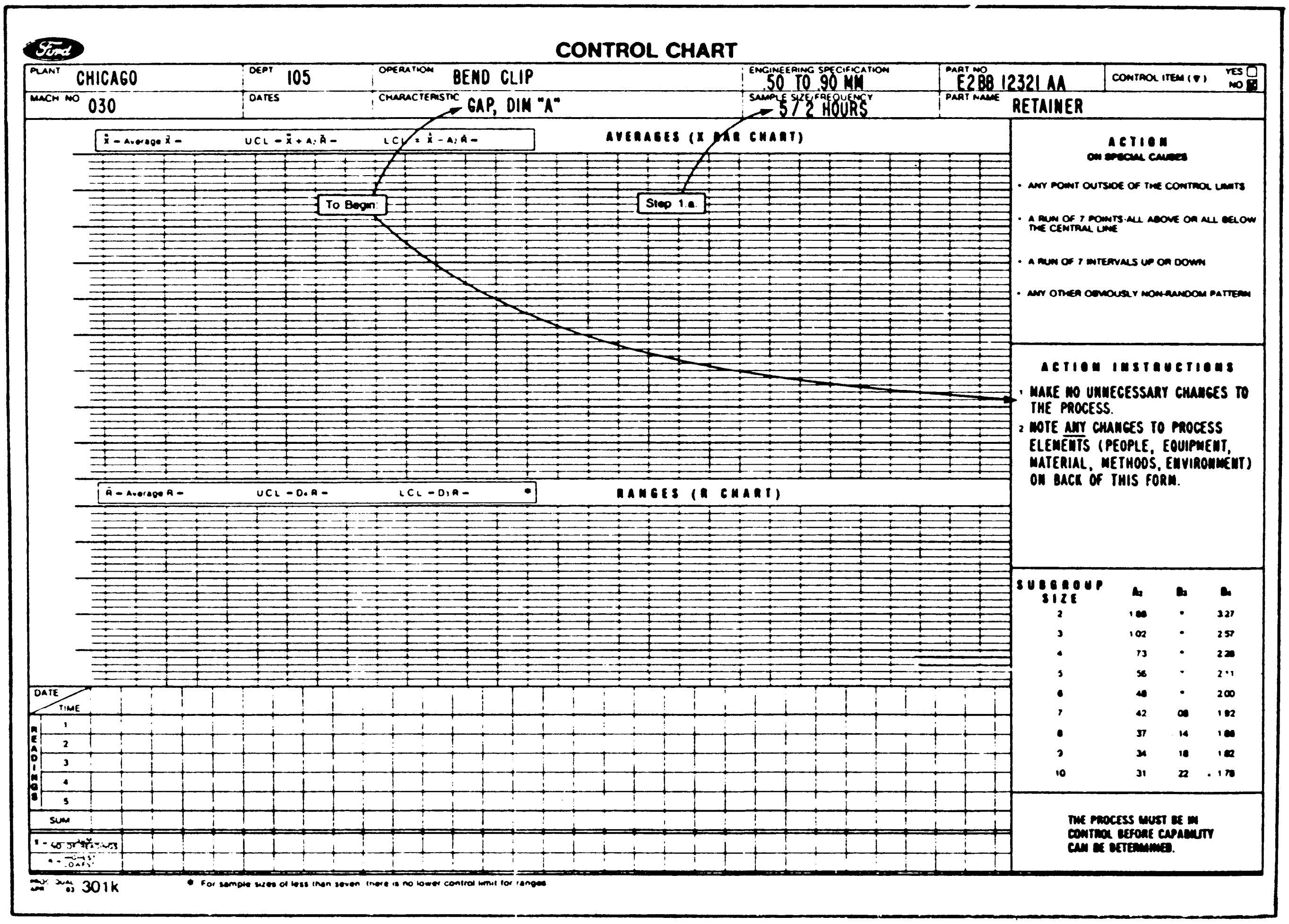


Fig. 8.3.7. Example of a control chart (Anon., 1987).

Table 8.3.10. Factors for the Construction of Control Charts

Number of Observations in Subgroup, n	Control Limit for Mean, A_2	Lower Control Limit for Range, D_3	Upper Control Limit for Range, D_4
2	1.88	—	3.27
3	1.02	—	2.57
4	0.73	—	2.28
5	0.58	—	2.11
6	0.48	—	2.00
7	0.42	0.08	1.92
8	0.37	0.14	1.86
9	0.34	0.18	1.82
10	0.31	0.22	1.78

Source: ASTM Manual on Quality Control of Materials.

$$W = O = TL + DT + TE - (N - 1)(L + O + I) \tag{8.3.21}$$

then

$$N = 1 + [(TL + DT + TE) / (L + O + I)] \tag{8.3.22}$$

Care must be taken if a single dump is used by trucks from several loaders (faces) since dump congestion may affect the dump time. In this case, simulation may be necessary.

8.3.4 STOCHASTIC MODELS

Stochastic models as related to mine systems engineering are generally viewed as discrete event Monte Carlo simulations, although the notion of stochastic models also include topics such as Markovian processes, dynamic programming, and other nondeterministic theories. Monte Carlo simulations are based on the utilization of Monte Carlo sampling for the duplication of mine systems containing stochastic or probabilistic elements. The technique was proposed by Von Neumann and Ulan in the 1940s and has become almost synonymous with the term simulation for systems engineering, although many simulations may not employ the method. The Monte Carlo approach is one in which the stochastic or probabilistic elements of a system are randomly determined through the use of randomized streams of variates that duplicate the expected probabilistic distribution of those elements.

The underlying assumption for employing Monte Carlo techniques is that the average or expected performance of a mining system that consists of many interacting stochastic variables can

Table 8.3.11. 100 Stopwatch Readings for a Repetitive Task, with Cycle Times in Minutes

Sample Number									
1	2	3	4	5	6	7	8	9	10
1.98	2.24	1.95	1.83	1.94	2.12	1.79	2.16	2.21	2.26
1.75	2.09	1.72	1.91	1.42	2.38	1.86	2.12	1.72	1.84
2.01	1.84	2.04	1.68	2.08	2.19	2.06	2.01	2.01	1.87
2.09	2.25	2.13	1.94	2.26	1.98	1.70	1.96	2.05	1.82
2.04	2.09	2.08	2.02	1.88	2.30	2.12	2.12	2.04	2.29

Sample Number									
11	12	13	14	15	16	17	18	19	20
1.81	1.76	2.17	2.03	2.43	2.55	2.10	1.78	1.63	2.18
2.10	1.79	1.99	1.92	1.84	2.17	2.26	1.88	2.23	1.92
2.19	2.06	2.25	2.03	1.87	2.00	1.87	1.57	1.71	1.98
2.06	1.82	2.05	2.07	2.20	2.31	1.89	1.84	2.08	1.99
1.84	2.44	2.08	2.08	2.14	2.14	1.90	1.62	2.02	1.99

Source: Buffa, 1972.

Table 8.3.12. Sample Means for Twenty Samples of Stopwatch Readings (*n* = 5, *x* = 2.01) (Sequence of samples is by rows.)

1.97	2.10	1.98	1.88	1.92	2.19	1.91	2.10	2.01	2.02
2.00	1.97	2.11	2.02	2.10	2.24	2.00	1.74	1.93	2.01

Source: Buffa, 1972.

Table 8.3.13. Ranges for Twenty Samples of Stopwatch Readings (*n* = 5). (Sequence of samples is by rows.) *R* = 0.43 min

0.34	0.41	0.41	0.34	0.84	0.40	0.42	0.28	0.49	0.47
0.38	0.68	0.26	0.16	0.59	0.55	0.39	0.31	0.60	0.26

Source: Buffa, 1972.

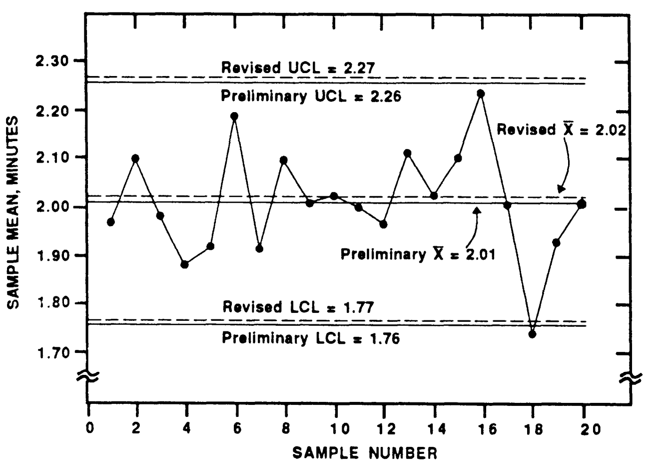


Fig. 8.3.8a. $\overline{X}$ chart for the data of Table 8.3.11 showing preliminary control limits and revised limits after data for sample No. 18 have been eliminated. Values plotted from Table 8.3.12 (Buffa, 1972).

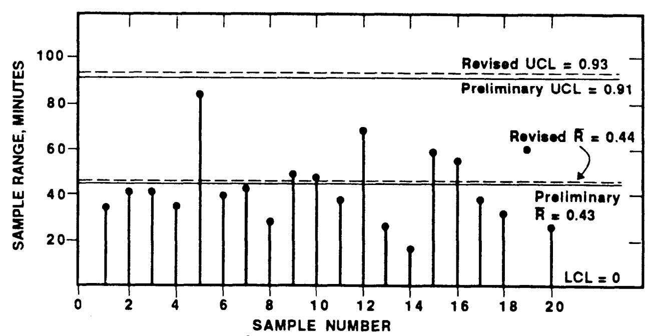


Fig. 8.3.8b. *R* chart for data of Table 8.3.11 showing preliminary control limits and revised limits after data for sample No. 18 have been eliminated because of out-of-control mean. Values plotted from Table 8.3.13 (Buffa, 1972).

be approximated through determining the average performance of repeated solutions of the system using randomly selected discrete values for each probabilistic variable that conforms to predefined probabilistic distributions. For example, the average performance of a shovel loading an 80-ton (72-t) truck can be approximated using Monte Carlo simulation techniques by randomly determining 100 load times that are variates of the probabilistic distribution inferred through time studies and performance statistics. Problems with using Monte Carlo simulation occur when there are too little data to adequately determine

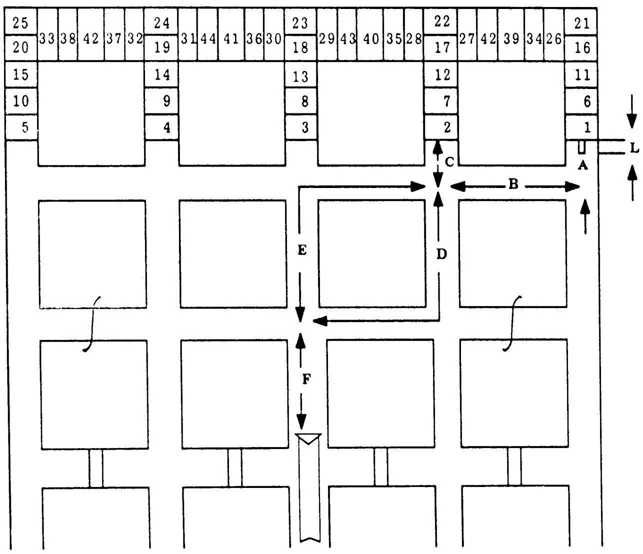


Fig. 8.3.9. Loading and hauling variables used in developing the mathematical model (Prelaz et al., 1964). L = Loader length; A + B = Change out distance; D = Haul distance – first unit; E = Haul distance – second unit; F = Discharge change out distance; A + B + C = Tram distance to next cut.

Table 8.3.14. Average LHD Tramming Speeds on Level Surface

Job Conditions	Units Smaller than 3 m^3 (4 yd^3)				Units of 3 m^3 (4 yd^3) and Larger	
	Hydrostatic Drive		Converter Drive			
	km/h	mph	km/h	mph	km/h	mph
Excellent	See specs. or use 13 km/h (8 mph)				16	10
Average[a]	8	5	8	5	13	8
Severe[a]	5	3	5	3	8	5

[a]Slower average and severe speeds for smaller units is a reflection of potentially smaller haulageway clearances.
Source: Stevens and Acuna, 1982.

Table 8.3.15. Observed Average Tramming Speeds for Shuttle Cars and Battery Powered Ram-Dump Cars in Coal Mines

Segment	Car Type	
	Shuttle Car	Battery Car
Change point—loader loader—Change point	300–325	300–325
Change point—dump Dump—change point	350–365	325–350

Conditions: Firm, level bottom. Minimum seam thickness = 4 ft (1.2 m)
Source: Prelaz et al., 1965; Suboleski, private files.

the representative probabilistic distributions of each stochastic element in the system, or when no attention is given to the correlation between stochastic elements and subsequently treated as independent elements.

Table 8.3.16. Fixed Time to Load, Maneuver, and Dump with LHD

Job Conditions	Time, min
Excellent	0.80
Average	1.10
Severe	1.40

Source: Stevens and Acuna, 1982.

Monte Carlo simulation enjoys popularity because of its ease of approximating solutions of highly complex interrelated stochastic systems. In the area of queueing models and waiting-line theory, the addition of interacting stochastic variable rapidly increases the mathematical complexity of the solution that results in an analyst's inability to derive a direct solution to the problem. As a result, Monte Carlo simulation techniques are generally utilized to accurately model the interaction of the probabilistic elements.

Monte Carlo simulation is a generic term used to denote any duplication of the dynamic behavior of a system through the use of a randomized stream of variates whether it be the simulation of financial risks within a mining venture, the probabilistic nature of chemical reactions, the modeling of failure criteria of a rock mass, or the interaction of trucks and shovels in an open pit. In applying Monte Carlo techniques, mine systems engineering usually looks at it from a narrower perspective of discrete simulation involving queues, waiting lines, or wait states. *Discrete simulation* is the branch of simulation that embodies a world view where all changes of a mining system can be described at discrete points in time such as an arrival of a truck at a shovel, the dumping to a bucket of ore into a truck, or the arrival of a surge of coal at a belt transfer point. In contrast, the term *continuous simulation* refers to the branch of simulation in which changes of a system are best described by defining a set of differential equations such as modeling the response of an electrical system to a ground fault.

Typical terminology used in model formulation of discrete change simulations involving the possible formation of wait states or queues (Shannon, 1975) includes the following.

1. Entity: An item which flows through the simulation. These can be trucks, shovels, ore cars, computers, maintenance personnel, or any other item within the simulation. They can also become available.

2. Attribute: A modification or characteristic of an entity. A truck might be characterized by its size, model, serial number, color, or maintenance records. These all would be attributes of the truck.

3. Activity: Time-dependent action within the simulation such as the activity of loading a truck, driving the truck from the pit to the crusher, or dumping the truck.

4. Event: A discrete change in the state of the system. The arrival of a truck at the crusher would be an event since the system changes states, that is, one of the trucks is no longer traveling. Events usually mark the beginning and ending of activities.

5. Process: A time-ordered sequence of events that may involve several activities. The cycle of a truck being loaded, driving to the crusher, dumping its load, and driving back to the shovel could be defined as a process.

The development of discrete simulation models involves the engagement of entities or items in a system in a manner which duplicates the functionality of the models and behavior of the system. Three primary modeling approaches are generally employed and according to Shannon (1975) include:

Table 8.3.17. LHD Vehicle Capacities and Sizes

Rated Payload Capacities				Operator Height		Overall Width		Approx. Turn Radius	
m^3	yd^3	t	tons	mm	in.	mm	in.	m	ft
0.8	1.0	1.4	1.5	1830	72	1220	48	2.4	8.0
1.5	2.0	2.7	3.0	1930	76	1550	61	3.7	12.1
2.7	3.5	4.8	5.3	1730	68	1830	72	4.1	13.3
3.8	5.0A	6.8	7.5	2110	83	2440	96	4.7	15.5
3.8	5.0B	6.8	7.5	2140	84	2140	84	4.8	15.9
3.8	5.0D	6.8	7.5	1980	78	2510	99	5.0	16.4
6.1	8.0	10.9	12.0	2260	89	2490	98	6.1	19.9
9.9	13.0	17.7	19.5	2540	100	3050	120	5.8	19.1

Source: Stevens and Acuna, 1982.

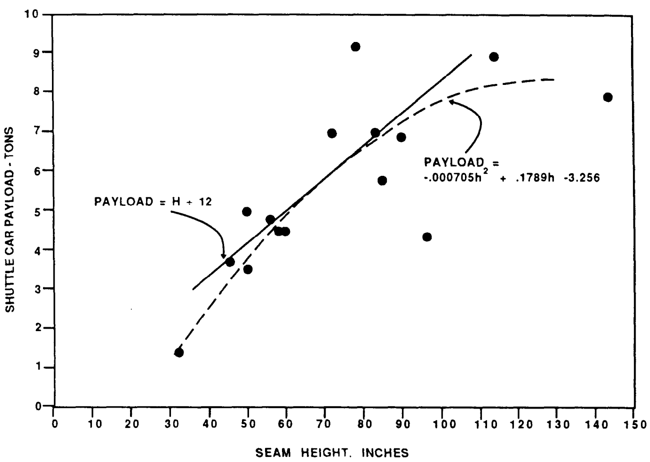


Fig. 8.3.10. Shuttle car payload vs. seam thickness in coal. All cars have 48-in. conveyors (Suboleski and Manula, 1979). Conversion factors: 1 in. = 25.4 mm, 1 ton = 0.9072 t.

1. Activity-oriented modeling.
2. Event-oriented modeling.
3. Process-oriented modeling.

Activity-oriented computer simulation involves representing the system as a set of activities. Entities within simulation engage activities and prescribe the conditions in which activities either start or end (Pritsker, 1985). Activities are initiated when a set of conditions is satisfied and therefore require the entire list of activities be evaluated when time is advanced. This type of simulation modeling is generally viewed as inefficient for most types of simulation work and is usually presented in most simulation texts for academic reasons. Pritsker (1985) gives a set of conditions in which the activity-oriented modeling framework should be considered. This approach would involve describing the activities in which entities are engaged and the set of conditions that cause a start or stop of the activity. Activities are initiated or terminated when a set of conditions is met.

The second major type of computer simulation involves representing the mining system as a set of events. *Event-oriented simulation* views the system from a perspective that the entire logic of a system can be represented as a set of time-ordered events. State changes within a system occur only at discrete points in time that are defined as events. The concept of activities occur in event-oriented simulation, but only as the abstraction that defines the point in time when events (the stop or start of the activities) take place.

For example, a truck-shovel cycle can be defined as a set of arrivals and departures of the truck. Every change of this system happens in conjunction with either an arrival or departure of a truck to or from some discrete point within the system. A queue at the shovel can only expand when a truck arrives, the next truck can only be loaded if the previous truck departs for the crusher or dump, and a truck can dump its load only after it arrives at its dumping point.

Process-oriented simulation models the flow of entities through the system. The distinction between event-oriented and process-oriented approaches is somewhat unclear since the notion of a process is defined as a sequence of events associated with the behavior description of the system. The main differentiation usually developed by general-purpose simulation languages is whether the events are directly defined and actually logically scheduled by the modeler (event-oriented simulation), or the events are logically implied through a process description and the simulation language translates the process into an appropriate sequence of events and handles the scheduling of the events in the sequence.

Process or event simulation approach is usually recommended by simulation experts and can be found applicable to a variety of mining problems. These areas include performance optimization (queuing problems), production scheduling and sequencing optimization, equipment selection, and production control.

It should be understood, however, that computer simulation itself is not a direct optimization technique. Simulation only provides a predictive model of the system. The analyst uses the predictive model as a tool for investigating alternative solutions and their economic impacts to the company. Optimization is a matter of selecting the best alternative within the framework of the alternatives explored.

There are a number of generic computer packages available to assist with problem definition and simulation for some of these optimizational areas. For example, there exist generic truck-shovel simulators, belt capacity simulators, and rail haulage simulators. These packages typically allow a user to input descriptions of a specific system and then provide a prediction of the operational characteristics of that system.

Several general-purpose simulation computer languages exist to aid in the development of stochastic simulations when an existing modeling package proves to be unavailable or inadequate. These include languages such as SLAM II, Simscript, GPSS, GASP IV, SIMAN, and DYNAMO to name just a few. Most of these languages are designed to limit the amount of code that a user must generate in order to simulate discrete processes or events and are usually better suited for model development than general purpose programming languages such as FORTRAN or C.

The design philosophy of simulation languages recognizes that if a system is modeled as parallel processes, a set of ordered events, or as a group of interrelated activities, certain generic operations must be done regardless of whether the simulation involves a small maintenance shop or the entire truck fleet of a large open pit mining operation. These generic operations include keeping track of time, accumulation of operational statistics, queue or idle time management, and the generation of the random variates of the defined probabilistic distributions. The advantage of the general-purpose simulation languages is that the codes to accomplish these tasks are generated for the user.

8.3.5 PIT LIMIT OPTIMIZATION

Optimum pit limit techniques or the delineation of recoverable ore reserves is an important area of application of systems engineering. Feasibility analyses, long-range planning, and the assessment of capital exposure and corporate risk can be significantly impacted by the results of the optimum pit limit determination. It is becoming increasingly necessary for companies to base even initial engineering studies of a mineral deposit on good estimates of the size shape and extent of a potential ore reserve (see 13.1.1 and 13.2.1).

The ultimate pit limit problem can be defined simply as determining the ultimate or final mining limits of a mineral deposit such that some set standard of maximum value or profitability is obtained from its extraction. By most pit optimization techniques, the *standard of profitability* is defined as maximizing the difference between the profit obtained from extracted ores and the cost incurred in removing associated waste materials. The standard most generally is set using a single period (life of mine) production schedule technique where profitability or lack of during subsets of that single period are ignored.

The use of advanced mathematical techniques in designing ultimate pit limits has become more practical in recent years due to the extension of various pit optimization methods along with advances in automated computational technologies. Optimization techniques can be briefly classified into five categories:

1. Heuristic techniques.
2. Dynamic programming.
3. Linear programming (integer programming).
4. Network flow theory.
5. Graph theory.

Heuristic optimization techniques have been one of the most widely used methods for ultimate pit limit analysis; however, these methods many times fail to generate true optimal designs. The most common heuristic technique used is the moving cone algorithm. *Dynamic programming* generates good results in two dimensions; however, three-dimensional extensions produce erratic results. The last three techniques—linear programming, network flow theory, and graph theory—actually employ the same problem formulation. Network theory and graph theory are alternate solution techniques for the linear programming problem.

The moving cone method (Pana, 1965; Pana and Carlson, 1966) is one of the most widely accepted techniques in the design of ultimate pit limits because of its rapid execution speed and easy conceptualization. Programming logic included in the method is patterned after conventional cross-sectional methods. It utilizes a break-even stripping ratio as the basic optimization criterion. The main difference from manual techniques is that it uses a 3D moving cone concept for removal increments instead of vertical sections to generate the final pit geometry.

A pit is generated and analyzed by constructing an upward cone and moving its vertex from one ore block to another. The cone shape is defined such that it conforms with pit slope design constraints in the various areas of the deposit. The computer is employed in generating 3D conical configurations and in the calculation of net value of each cone by summing the values of all ore and waste blocks enclosed within the cone. Finally, a 3D pit limit is obtained by removing the frustrums of all cones with net positive value.

From the point of view of finding a mutual support between overlapping cones, the moving cone method is significantly restrictive due to a large amount of computational effort required (Barnes, 1982). Indeed, the determination of a mutual support between overlapping cones is not usually required by most implementations of the moving cone algorithm. The method usually terminates after cones having their vertices located on all positive ore blocks have been evaluated. For this reason, the heuristic moving cone method many times fails to generate true ultimate pit limits.

In attempts to consider the mutual support between cones, more efficient techniques applying sophisticated mathematical programming or systems analysis concepts have been developed.

The original dynamic programming algorithm was demonstrated in the design of the optimal configuration of blocks to be removed in a two-dimensional (2D) cross section by Lerchs and Grossmann (1965). Given a 2D cross section of a block model and the defined slope constraints, the algorithm proceeds by calculating the column sum of original block value for each block. This column sum value represents a cumulative value realized in extracting a single vertical column from the top of the block model to each individual block. Next, column-by-column starting from any end column of the cross section, a pit value representing the maximum value of the potential 2D pit is computed for each block. This pit value is calculated from the column sum value of the block and the predetermined pit value of an overlying block in the previous column. This new value is the maximum possible contribution of the starting end column to the column containing that block to any feasible 2D pit that contains the block on its contour. An arrow (or pointer) is used to indicate the overlying block that provides the maximum value to the calculation of the pit value of a particular block. The ultimate pit limit is then determined by tracing back the arrows from the block in the uppermost level that has the maximum pit value. Fig. 8.3.11 shows an example of the 2D dynamic programming method.

The dynamic programming approach as originally defined by Lerchs and Grossmann is able to generate the optimal pit contours in given 2D cross sections. A final 3D pit geometry is then determined by merging the geometry determined by multiple cross sections through the pit. A true optimum may not be obtained by the assemblage of these 2D cross sections, and typically one finds that the final limits may also violate maximum allowable pit slopes.

In the past several years, many attempts have been made to extend the original 2D algorithm concepts to the design of truly optimum 3D ultimate pit limits. The algorithms developed by Johnson and Mickle (1970), Johnson and Sharp (1971), Barnes (1982), and Koenigsberg (1982) are still close to the original 2D algorithm, except they also consider block value information from adjacent cross sections. These 3D approaches are restricted in that the pit slopes must be consistent throughout the block model. The configuration of the generated pit generated with the algorithms may still fail to follow the general pit slope constraints, and a process called pit reaming is usually required as a final stage to correct inconsistencies. In contrast to other pit limit generation techniques, the dynamic programming algo-

	1	2	3	4	5	6	7	8
1	−1	−1	−1	−1	−1	−1	−1	−1
2	−2	−2	+1	−2	+2	+1	−2	−2
3	−3	−3	+3	+4	−1	+4	−3	−3

a) An example cross section of a block model with original block values and slope 1:1.

	1	2	3	4	5	6	7	8
1	−1	−1	−1	−1	−1	−1	−1	−1
2	−3	−3	0	−3	+1	0	−3	−3
3	−6	−6	+3	+1	0	+4	−6	−6

b) The tableau shows the column sum values obtained from a).

	0	1	2	3	4	5	6	7	8
0	0	0	0	0	0	0	0	0	0
1		−1	−1	−1	−1	−1	0	0	+1
2		−3	−4	−1	−4	+1	+1	+2	−1
3		−6	−9	−1	0	0	+5	−1	−4

c) The final tableau shows the pit value of each block and an ultimate pit limit.

Fig. 8.3.11. Ultimate pit limit problem formulated and solved by using the 2D dynamic programming algorithm.

rithms do not utilize a cone generation routine for obtaining the set of overlying blocks. Thus the method may yield final pit limits that are far from optimal.

The ultimate pit limit problem can be formulated as a simple zero-one integer programming problem with the following structure:

$$\max Z = \sum_{n=1}^{N} C_n X_n \qquad (8.3.23)$$

Subject to:
$$-X_t + X_n \leq 0 \text{ for all } n = 1 \text{ to } N \qquad (8.3.24)$$
$$X_t, X_n \text{ integer}$$

where N is total number of blocks, X_t is an overlying block that must be removed before mining block n, and C_n is cost or profit associated with mining block n.

The computer time required to generate the solution of this *integer programming* problem is proportional to the number of block sequence constraints. The sequence constraints required for each block amounts to one constraint equation for each overlying block that must be removed as defined by the pit slope constraints before that particular block can be mined. This means that unless the block model is fairly small, the integer zero-one programming technique cannot be practically applied due to the large amount of computer resources required for model description and resolution. However, this large binary integer programming problem may be transformed into an equivalent bipartite network flow problem that can be solved by highly efficient solution algorithms.

The *network flow* model for determining ultimate pit limits is based on the network theory of maximum flow and minimum cut. The solution technique was originally developed by Johnson (1968). The problem is formulated such that the nodes in the network are equated to blocks in a 3D block model, and arcs are generated to represent the required pit slope constraints for feasible mining sequences. That is, each node is defined as an indicator variable X_n of the integer programming problem. Network arcs connect positive valued nodes (potential ore) to overlying negative valued nodes (waste) that must be removed in order to mine the ore node. These arcs therefore represent the precedence constraints.

The basic network formulation has been shown by both Johnson (1968) and Barnes (1982) to be a bipartite network that can be easily solved by maximum flow theory. The basic set up procedure of the network is graphically shown in Fig. 8.3.12 with these constraints:

1. All positive-valued blocks (potential ore) are set up on the left side of the network and connected to an imaginary source node.
2. All negative-valued blocks (waste blocks) are set up on the right side of the network and connected to an imaginary sink node.
3. Arcs are drawn from each positive block to every waste block that must be mined prior to mining the positive valued block. In 3D space, this insures that pit slope geometry and constraints are maintained.
4. The arc capacity from the imaginary source node to each positive node (ore block) is equated to the financial return of mining and processing the block.
5. The arc capacity of the waste blocks to the imaginary sink is equated to the cost of extraction and removal of the block.
6. The arc capacity of the connections between the positive blocks and overlying the waste blocks is set to infinity.

A bipartite network of this nature can be effectively solved by one of several maximum flow algorithms. The ultimate pit ends up being defined as the removal of all blocks connected by arcs that belong to the maximum flow or minimum cut. If a labeling algorithm is used, this simply means that all labeled blocks in the final iteration are mined.

This approach to the generation of the ultimate pit can be shown to produce a truly optimal solution. The method does require significant computational resources and employs somewhat hard to understand mathematical techniques. Hanson (1986) presents a modification of the method to reduce the required computational resources by assigning priority to flow allocation.

The application of *graph theory* for designing pit limits was creatively introduced by Lerchs and Grossmann (1965). The algorithm formulates the zero-one integer programming problem into network tree. The vertices are equated to mining blocks, and the imposed directed arcs represent pit slope constraints. These directed arcs indicate the relationship between waste

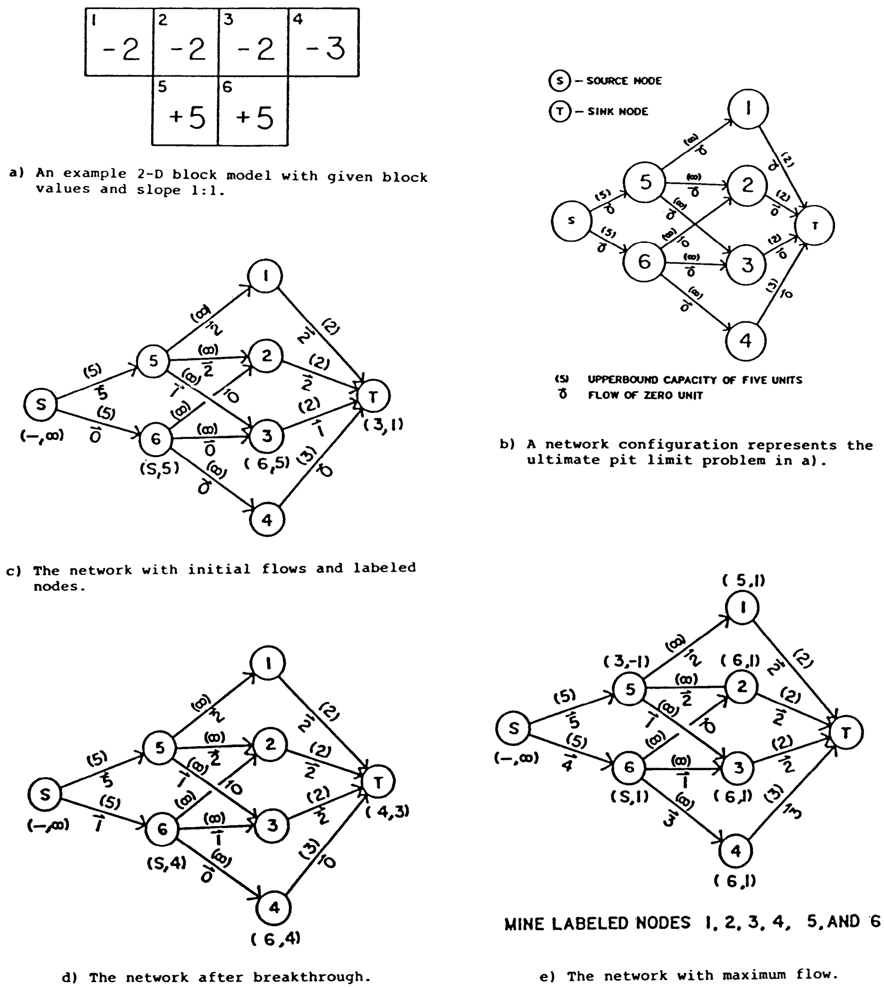


Fig. 8.3.12. Ultimate pit limit problem formulated as the network flow problem and solved by the labeling algorithm.

blocks that must be removed before mining a particular ore block. Since any feasible pit contour is obtained by a closure of a graph, Lerchs and Grossmann recognized that the ultimate pit is a problem of determining the closure of a graph with a maximum total mass.

This algorithm begins by classifying each vertex into either a positive or a negative mode category corresponding to its block value (positive = potential ore and negative = waste). Directed arcs are generated that represent the slope constraints from a positive node to its overlying negative node. In general, the algorithm starts constructing the initial graph tree from the blocks in the uppermost level of the block mode and proceeds down level by level. The initial tree is constructed by having a set of dummy arcs connecting the referential dummy node to all vertices. The tree will then be transformed into successive trees following set rules. The transformation process proceeds until no further transformation is possible. The algorithm terminates in a finite number of iterations. The graph method is shown in Fig. 8.3.13.

The directed graph problem can be transformed into a network flow problem due to the special structure of the network flow programming model (Lerchs and Grossmann, 1965). Barnes (1982) shows the graph concept is mathematically equivalent to the network flow programming concept previously discussed.

The graph algorithm has been shown to be a practical method for determining the true ultimate pit limit with variable pit slopes in 3D. The main drawback of the technique is that in its original form, it requires large quantities of computing resources to determine maximum graph closure. Many researchers have looked at techniques to reduce the computer resources required. These include bounding techniques (Rychkum and Chen, 1979), limiting the number of directed arcs (Whittle, 1987), and the modified tree algorithm (Huttagosol, 1988: Huttagosol and Cameron, 1989).

8.3.6 LINEAR PROGRAMMING

Linear programming (LP) is considered one of the most powerful tools in operations research. The technique is a mathematical structure for the analysis of normative, linear models formulated to allocate limited resources such as manpower, haulage capacities, or contract constraints among various activities (ore production) competing for those resources in a manner to achieve the most desirable result. The "most desirable result"

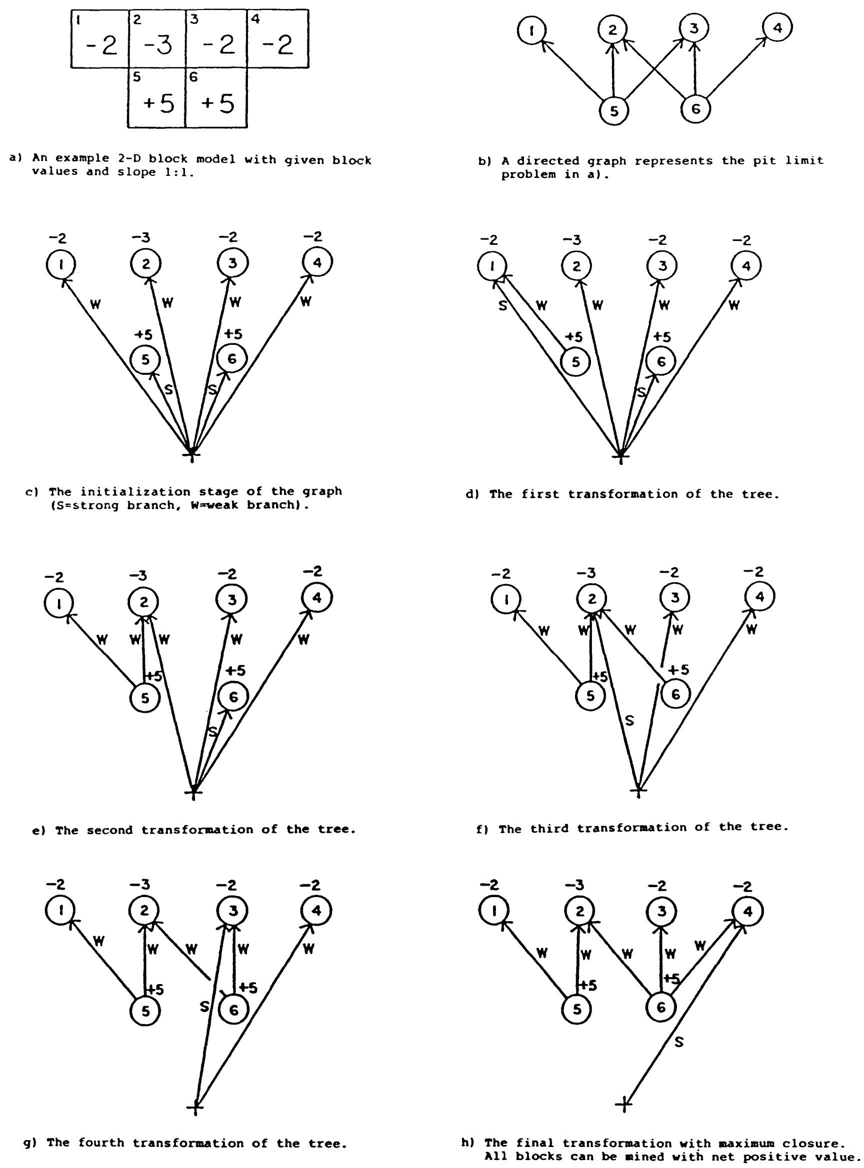


Fig. 8.3.13. Ultimate pit limit problem formulated and solved by using the directed graph algorithm.

typically is defined as the allocation scheme with maximum profits or minimum production costs per salable unit and is understood to be the objective for optimization.

Utilization of LP technique requires that all mathematical functions within the modeled system be linear. Both the objective function (maximizing profits or minimizing productin costs) and the limited resources (constraints) are expressed by linear equations. Hence problems are stated typically in a general mathematical form shown in Eqs. 8.3.25 and 8.3.26, in which n is the number of decision variables and m is the number of constraint equations.

$$\text{Max } Z = \sum_{i=1}^{n} c_i x_i \qquad (8.3.25)$$

Subject to:

$$\sum_{i=1}^{n} a_{ki} x_i \leq b_k \qquad k = 1, \ldots, m \qquad (8.3.26)$$

$$x_i \geq 0 \qquad i = 1, \ldots, n$$

$$b_k \geq 0 \qquad k = 1, \ldots, m$$

The solution technique used in linear programming to set production levels of competing activities is called the *simplex tech-*

nique. The simplex technique commonly imposes two fundamental requirements with respect to the structure of the system being analyzed:

1. All variables can assume any real nonnegative value.
2. All relationships are continuously linear.

The first condition states that variables are not restricted to whole values. A variable must be able to assume any value within the positive number space. If the quantity of labor for a certain mining section is to be determined, this assumption makes it equally valid for the solution to be either 4.3567 or 5.0 employee-hours of labor. Should it be necessary to limit a solution strictly to whole or integer numbers, then linear programming in its basic form is inappropriate and a mixed linear integer programming analysis should be considered.

The solution space is strictly limited to the positive number system by the first condition. In many cases it makes little sense to discuss negative activities or production. (Ore may be produced from a mine; however, rarely are we able to mine a negative quantity of ore!) When it is physically possible that a variable can assume a negative value, this restriction does not hinder the use of linear programming since any number, positive or negative, can be expressed as the algebraic difference of two nonnegative numbers. For example, when working with inventories it is possible to buy and sell goods. A positive number usually represents the quantity to buy while a negative number usually represents the selling or the consumption of inventory. However, if two separate variables are used, one positive number can quantify the amount to buy, and another positive variable can represent the quantity of inventory to sell.

The second condition is necessary for the simplex solution method to work. Implied in this condition by the term linear are proportionality and additivity assumptions over the entire range of the mathematically defined system being analyzed. Proportionality states that every increment of production utilizes an equal increment of the available resources. This is important since this restriction precludes equations that may take into account the economies of scale. The additivity property mathematically states that there are no cross-product terms that arise between activities. Given any activity level, the total usage of a resource is equal to the sum of the individual activities if considered separately. This precludes the use of LP where the marketability of one product is dependent of the production of another product.

Continuously linear conditions prohibit the use of functions that are stepwise or undefined at any point within the solution space. This restriction makes it more complicated to accurately model incremental production facilities typically represented by discontinuous functions.

Problem formulation for linear programming (LP) can be illustrated by the following highly simplified example.

Example 8.3.4. A small mine produces ore from two working stopes. The ore in stope A yields a profit of \$4.30/ton (\$4.74/t) and contains 0.36% arsenic. The ore in stope B yields a profit of \$5.00/ton (\$5.51/t) and contains 0.65% arsenic. The smelter requires that the average arsenic content of ore shipped cannot exceed 0.50% and the mine haulage system makes it impossible to haul more than 400 tons/shift (360t/shift). The objective of the problem is to maximize the profit per shift.

Solution. The decision that must be made is setting the quantity (or activity level) of ore to be produced from stopes A and B resulting in maximum profits. Two variables X_1 and X_2 can be defined: X_1 = tons/shift produced from stope A. X_2 = tons/shift produced from stope B.

Hence the following equation can be written to represent the objective of our linear programming analysis:

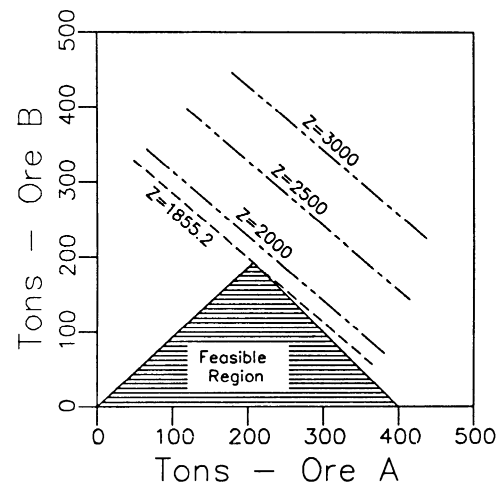


Fig. 8.3.14. Graphical solution of Ex. 8.3.4.

$$\text{Max } Z = 4.30X_1 + 5.00X_2 \tag{8.3.27}$$

The problem also has two constraints that bound the solution space. These constraints are a limited haulage capacity and a limit on the average arsenic content. In linear programming terminology these two constraints are considered the limiting resources. The mathematical form of the haulage constraint is easy to derive and can be expressed as

$$X_1 + X_2 \leq 400 \tag{8.3.28}$$

The average arsenic constraint requires a maximum average amount and can be expressed in mathematical terms as:

$$0.36X_1 + 0.65X_2 \leq 0.5(X_1 + X_2) \tag{8.3.29}$$

A little algebraic manipulation is needed to get the equation into the form specified in Eq. 8.3.26. Rearranging the terms, the arsenic constraint can be specified,

$$-0.14X_1 + 0.15X_2 \leq 0 \tag{8.3.30}$$

The result is the following formal mathematical formulation for the example linear programming problem:

$$\begin{array}{llrl}
\text{Max Z} & = & 4.30X_1 + 5.00X_2 & \text{objective function} \\
\text{Subject to:} & & X_1 + \quad X_2 \leq 400 & \text{haulage} \\
 & & -0.14X_1 + 0.15X_2 \leq \quad 0 & \text{arsenic} \\
 & & X_1 \geq \quad 0 & \\
 & & X_2 \geq \quad 0 &
\end{array}$$

The optimal solution of this highly simplified example can be easily seen by a quick 2D graphical analysis as shown in Fig. 8.3.14 since there are only two decision variables of interest. This figure shows that a maximum profit of 1855.20 can be obtained if 206.9 tons (187.7 t) of ore from stope A and 193.1 tons (175.2 t) of ore from stope B are produced. It illustrates the feasible solution space along with contours of equal and increasing values for the objective function (Z contour lines).

An important feature to note in the graphical solution is that the optimal solution lies on one of the extreme corner points of the feasible region as defined by the constraints. Formal proof that this will be true for any LP problem is shown by Wehl

(1934). Should the coefficients of the variables in the objective function change such that the optimal Z contour is parallel to a boundary of the feasible region, the optimal solution will be nonunique. However, one of the optimal solutions will still correspond to a comer of the feasible region.

Since the system of equations is also required to be continuously linear, the feasible region is guaranteed to be convex (Hillier and Lieberman, 1986). Convexity of the space and the linearity of the Z contours mandates that every corner will have at least one adjacent comer whose corresponding Z value is greater unless it is the optimal corner point. Therefore, the simplex method is a simple search technique that uses the foregoing two properties. It begins the process by starting at the origin where the Z contour value is zero, looks for an adjacent corner where the Z value is better, and iterates until it can make no more improvements to the objective function.

The simplex algorithm is covered in most texts on linear programming or operations research (Hillier and Lieberman, 1986; Daellenbach and George, 1978) if the exact mathematical manipulations involved in this solution technique are of interest. Many computer programs are available for a small charge to solve LP problems and are easier to obtain and use than working through the algorithm by hand.

The mathematical form of linear programming problems previously expressed with Eqs. 8.3.25 and 8.3.26 does not contain provisions for ≥ or = constraints on the resources. Yet production contracts typically require use of these types of constraints. For example, a contract may require delivery of an exact tonnage with minimums set on the quality of the product. Most texts dealing with linear programming derive the necessary algebraic manipulations and modifications required to use the simplex algorithm to derive a solution when these types of constraints are present. Commercially available computer software designed to solve linear programming problems automatically performs these manipulations should ≥ or = constraints be present. Therefore, a general formulation for LP problems, shown in Eqs. 8.3.31 and 8.3.32, can be expressed:

$$\text{Max } Z = \sum_{i=1}^{n} c_i x_i \qquad (8.3.31)$$

$$\text{Subject to:} \quad \sum_{i=1}^{n} a_{ki} x_i \begin{Bmatrix} \leq \\ \geq \\ = \end{Bmatrix} b_k \qquad k = 1, \ldots, m \qquad (8.3.32)$$

$$x_i \geq 0 \qquad i = 1, \ldots, n$$

$$b_k \geq 0 \qquad k = 1, \ldots, m$$

Example 8.3.5. The previous example can be modified slightly to illustrate problem formulation when dealing with the more general form. For example, assume that the silver in the ore in stope A averages 6 oz/ton (188 g/t), the silver in the ore in stope B averages 3.5 oz/ton (109 g/t), and the following two additional contractual constraints are imposed:

1. Tonnage in stope A must equal 200 tons/shift (180 t/shift):

$$X_1 = 200.$$

2. Average silver must be at least 4 oz/ton (125 g/t):

$$6X_1 + 3.5X_2 \geq 4.0(X_1 + X_2)$$

Solution. The modified mathematical formulation of the problem becomes:

$$\text{Max } Z = 4.30X_1 + 5.00X_2 \qquad \text{Objective Function} \qquad (8.3.33)$$

$$\text{Subject to:} \quad X_1 + X_2 \leq 400 \text{ haulage} \qquad (8.3.34)$$

$$-0.14X_1 + 0.15X_2 \leq 0 \text{ arsenic} \qquad (8.3.35)$$

$$X_1 = 200 \text{ tonnage} \qquad (8.3.36)$$

$$2X_1 - 0.5X_2 \geq 0 \text{ silver} \qquad (8.3.37)$$

$$X_1 \geq 0$$

$$X_2 \geq 0$$

The optimal solution of the modified example results in a maximum profit of 1793.50 by producing 200.0 tons (181.4 t) of ore A and 186.7 tons (169.4 t) of ore B. This problem can also be solved with most commercially available linear programming packages.

Simplex methods derive a solution to an associated problem simultaneously while finding the optimum resource allocations of the original problem. The related problem whose solution is obtained by solving the original problem is defined as the *dual* while the original problem is usually referred to as the *primal.* This concept of duality is a very powerful and useful advantage of utilizing linear programming techniques to solve resource allocations problems.

The secondary problem solved (the dual) when dealing with resource allocation is one which deals with the pricing of the scarce resources. The dual give the imputed value of the scarce resources within the problem; that is, the maximum price a company could pay for additional amounts is greater than the incremental profit derived by its use.

Included in most linear programming packages is the ability to display or print the solution to the dual problem. The dual solution of Example 8.3.5 gives the following:

Constraint	**Dual Value**
Haulage	0.00
Arsenic	33.33
Tonnage	8.97
Silver	0.00

The interpretation of the dual for resource allocation problems is that the dual values represent the maximum amount a company should pay to acquire an additional unit of each constraint. Haulage and silver constraints have dual values equal to zero. This is because the optimal solution does not utilize all of the existing capacity for the resources (i.e., 386.7 tons, or 350.8 t, of the 400-tons, or 362.9-t, haulage capacity is being used, and the silver content averages 4.79 oz/ton, or 150 g/t, in contrast to a minimum requirement of 4.0 oz/ton, 125 g/t). Therefore, a company should not be willing to spend any money on increased capacities for these two constraints since, in essence, any more capacity would also go to waste. The dual value for the tonnage in stope A is 8.97. This indicates that if the maximum tonnage mined from stope A can be increased 1 unit, an additional $8.97 can be obtained in profit (notice that increasing production in stope A also will allow additional tonnage to be mined in stope B at optimum performance of the system). The dual value of the arsenic content is 33.33. Interpretation of the dual for average content is a little more complex since the dual value reflects increasing the right-hand side of the final arsenic equation used in the linear program by one unit. Accordingly, that means that the equation,

$$-0.14X_1 + 0.15X_2 \leq 0 \qquad (8.3.38)$$

must go to

$$-0.14X_1 + 0.15X_2 \leq 1 \qquad (8.3.39)$$

or that the original arsenic equation must look like:

$$0.36X_1 + 0.65X_2 \leq 0.50(X_1 + X_2) + 1 \qquad (8.3.40)$$

With the assumed production constraints in the hypothetical problem of about 400 tons (360 t), increased profits of $33.33 can be expected for each 1/400th increase in the average arsenic content. If the average allowable arsenic content is raised to 0.5025, then an additional $33.33 could be expected as a profit if operating the mine at optimum levels. This means additional profit can be realized in the operation if penalty costs to the mine of increasing the arsenic content to that level is less than $33.33 additional profit gained.

Two other analyses that are also given by the simplex solution of the problem concern the sensitivity of the solution of its input parameters. These include what are called *objective row ranges* or *cost coefficients* and *right-hand-side ranges* analyses. The first is simply an indication of how stable the solution is in respect to changes in the coefficients of the objective function, while the second gives an indication of how far one can increase or decrease the value of the constraints before it becomes a fruitless exercise.

Simplex techniques usually express the objective row ranges as minimum and maximum values. In the previous example, the simplex gives:

Objective Row Ranges

Variable	Minimum	Maximum
X_1	None	None
X_2	0	None

This indicates that the coefficient of X_1 could be changed to anything, and the optimal operating point (ore in stope A = 200 tons, or 181.4 t, and ore in stope B = 186.7 tons, or 169.4 t) would not change. Profitability would change; however, the optimal operating point does not. This is not surprising since one of the constraints requires that production in stope A must equal 200 tons (181.4 t) regardless of its profit or loss. Likewise, the previous table shows that the profit coefficient of ore produced from stope B can be anything positive and the optimal operating point would not change. Notice that should the profit coefficient go negative (the company loses money by mining ore in stope B), then the optimal production rates will change. Typically, in real problems, these maximum and minimum coefficients tend to be within a few cents of the given parameters so that careful evaluation may have to be made on the appropriateness of the estimates utilized.

The right-hand-side range analysis is also expressed in terms of the maximums and minimums for the constraints of the problem. As before, they give an indication of how well estimates must be made. Another use of these values is to determine how far one of the constraints can be moved.

Example 8.3.6. To illustrate this concept, the simplex solution of Example 8.3.5 yields a right-hand-side range (RHS) printout as follows:

Right-Hand-Side Range Analysis

Constraints	Dual Value	RHS	Minimum	Maximum
Haulage	0.00	400.00	386.7	None
Arsenic	33.33	0.00	– 28.0	2.0
Tonnage	8.96	200.00	0.0	206.9
Silver	0.00	0.00	None	306.7

Solution. In this problem, haulage and silver have no dual value, so the right-hand-side range analysis indicates how far the RHS can be moved before the constraints would have a dual value; that is, it really indicates how much unutilized resource exists. Arsenic that has a dual value of $33.33 has a minimum RHS of –28.0 and a maximum RHS of 2. Once again referring to the original formulation of the constraint, a value of 1 equates to approximately $^1/_{400}$th of a percentage change in the allowable content. A maximum increase of profit can be realized if the arsenic constraint is increased from 0 to 2 or 2/400th of a percentage yielding increased profits of $66.66. The tonnage produced from stope A is constrained to equal 200 tons (181.4 t). The RHS analysis shows that if this tonnage constraint could be increased to 206.9 tons (187.7 t), increased profits of $8.96 ($9.88/t) for each ton would be realized. The RHS analysis shows that once the constraint exceeds 206.9 tons (187.7 t), the additional increase will not net additional profit. Likewise decreases of the constraints can be examined. They are handled the same as increases with the exception that the profit is being decreased by the dual value amount for each unit of decrease of the constraint.

8.3.7 MIXED INTEGER LINEAR PROGRAMMING

Mixed integer linear programming is a technique very similar to linear programming in its general mathematical form, with the exception that some or all of the variables used in the system can be defined such that they only assume whole or integer values. While the technique is still labeled by most texts as a special linear programming method, the introduction of integral constraints to one or more variables make it actually a nonlinear optimization technique (Beightler et al., 1979). Typical mixed linear integer problems are stated in a mathematical form almost identical to linear programming problems, with the exception that the nonnegativity condition for the integer variable are replaced by the requirement that the variables assume finite integer values, as shown in Eqs. 8.3.41 and 8.3.42:

$$\text{Max} \qquad Z = \sum_{i=1}^{n} c_i x_i \qquad (8.3.41)$$

$$\text{Subject to:} \quad \sum_{i=1}^{n} a_{ki} x_i \begin{Bmatrix} \leq \\ \geq \\ = \end{Bmatrix} b_k \qquad k = 1, \ldots, m \qquad (8.3.42)$$

$$x_i \geq 0 \qquad i = 1, \ldots, \bar{n}$$

$$x_i \text{ integer} \qquad i = \bar{n} + 1, \ldots, n$$

$$b_k \geq 0 \qquad k = 1, \ldots, m$$

The technique is useful in mining analyses since it allows one to accurately model production rates expressed in terms of the number of cars or labor in terms of employee-shifts. Once again, referring to the previous simplified example (i.e., Example 8.3.5), assume that the production in stopes A and B combined must be hoisted in a 7-ton (6.3-t) skip. A third variable that can only assume integer values can be introduced into the problem

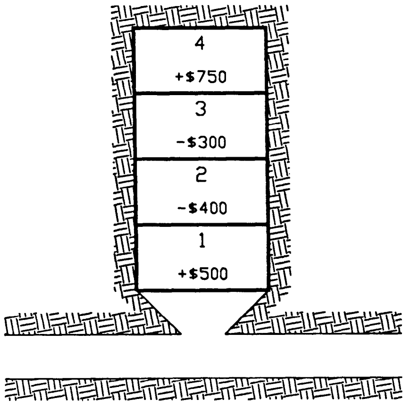


Fig. 8.3.15. Small ore stope with block values

to generate a production constraint that forces the total ore produced to some multiple of 7 tons (6.3 t). The new constraint added to the problem is shown in Eq. 8.3.43.

$$X_1 + X_2 = 7X_3 \tag{8.3.43}$$

where X_3 must assume an integer value of 1, 2, 3, ... *n*.

Rearranging the terms in Eq. 8.3.43 to standard mathematical form, the complete problem formulation then becomes:

$$\text{Max } Z = 4.30X_1 + 5.00X_2 \qquad \text{Objective Function} \tag{8.3.44}$$

$$\begin{aligned}
\text{Subject to:}\quad & X_1 + X_2 \le 400 \text{ haulage} && (8.3.45)\\
& X_1 + X_2 - 7X_3 = 0 \text{ haulage} && (8.3.46)\\
& -0.14_1 + 0.15X_2 \le 0 \text{ arsenic} && (8.3.47)\\
& X_1 = 200 \text{ tonnage} && (8.3.48)\\
& 2X_1 - 0.5X_2 \ge 0 \text{ silver} && (8.3.49)\\
& X_1 \ge 0 &&\\
& X_1 \ge 0 &&\\
& X_3 \text{ integer} &&
\end{aligned}$$

Mixed integer linear programming is used very frequently for scheduling and sequencing problems involving ordering restrictions such as modeling the production of an ore body where ore or waste must be exposed before it can be extracted. With such ordering restrictions, one typically formulates the problem with the introduction of variables that may take on only values of zero or one. This special case of mixed integer linear programming is referred to by most analysts as integer 0-l programming when all integer variables can only assume a value of either 0 or 1. In general, more efficient solution techniques can be applied to 1-1 problems; however, in practice, the general mixed integer linear programming algorithms are often used on these types of problems.

Example 8.3.7. To illustrate the use of mixed integer programming as applied to sequencing problems, assume that a small stope has four ore blocks (Fig. 8.3.15). The monetary value at the bottom of each block indicates either the profit or cost associated with mining and processing each block. In order to mine block 4 in this stope, blocks 1 through 3 must first be removed. If the objective of the problem is to determine the blocks to be mined that maximizes the total profit obtained from this stope, then four decision variables can be defined: X_1 through X_4. Each variable will assume a value of 0 if the block is to be left while it assumes a value of 1 if it is to be mined.

Solution. The following equation can be written to represent the objective of this problem:

$$\text{Max } Z = 500X_1 - 400X_2 - 300X_3 + 750X_4 \tag{8.3.50}$$

Constraints can be written requiring that if block 1 is not mined, then block 2 cannot be mined, and likewise, if block 2 is not mined, then block 3 and 4 are also left. This is accomplished by requiring that each 0-l variable be less than or equal to the 0-l variable representing the preceding block in the sequence:

$$X_1 \le X_2 \le X_3 \le X_4 \tag{8.3.51}$$

This results in the following integer programming formulation:

$$\text{Max } Z = 500X_1 - 400X_2 - 300X_3 + 750X_4 \tag{8.3.52}$$

$$\begin{aligned}
\text{Subject to:}\quad & X_1 - X_2 \le 0\\
& X_2 - X_3 \le 0\\
& X_3 - X_4 \le 0\\
& 0 \le X_1 \le 1\\
& 0 \le X_2 \le 1\\
& 0 \le X_3 \le 1\\
& 0 \le X_4 \le 1\\
& X_1, X_2, X_3, X_4 \text{ integer}
\end{aligned}$$

8.3.8 CPM/PERT

The *critical path method* (CPM) and the *program evaluation and review technique* (PERT) are two prominent procedures developed to aid in successful management of large-scale projects. These two techniques were first introduced in the late 1950s as methods for project management. PERT was developed by the US Navy to coordinate the work of many private contractors, while CPM was developed by the E.I. du Pont de Nemours Co. In recent years, these terms have been used to reference the network approach to project management in a very broad sense with little regard to the actual formal procedures represented by the original techniques.

Successful project management requires good planning, scheduling, and coordinating of numerous interrelated activities. Both CPM and PERT were developed with those goals in mind. These techniques answer the question of what is the minimum duration required for a given project, given proper management and sufficient resources.

While CPM and PERT are not direct optimization techniques, they provide a manager with information to aid project planning and control. These methods help resourceful managers deal with resource balancing and utilization in a manner that better sequences and schedules the use of a scarce resource. In addition, CPM and PERT may be used on an ongoing basis throughout the life of a project to help reduce total project duration, to allocate project funds, to optimally place purchasing requests, and to help allocate project or company resources involved in a project.

Project management as defined by CPM and PERT consists of three primary steps:

1. List and sequence project tasks.
2. Estimate task duration.
3. Evaluate the project.

The fundamental and common characteristic of CPM and

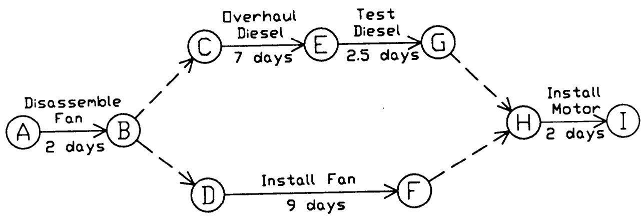


Fig. 8.3.16. Initial CPM network of a simplied fan move.

PERT is the use of a network to graphically illustrate both tasks and their sequence involved in completing a project. The project as a whole must be a set of well-defined tasks, which, in CPM/PERT terminology, are called *activities.* These activities or tasks are ordered such that they must be completed in some set sequence, and they are defined as the branches of the CPM/PERT network. The nodes of the network represents what is termed an *event,* which is defined as the point in time when all activities leading into that node are completed. Another component typically used with CPM/PERT networks is the incorporation of *dummy branches.* These are branches or dummy activities used to indicate interdependencies that cannot be represented correctly solely with the conventional activity and event structure and are useful when doing hand solutions of the network for creating parallel activities or merging several parallel activities. A highly simplified example is illustrated in Fig. 8.3.16 that represents a project where a diesel-powered mine fan is being moved to another site, and the diesel engine is overhauled in the process.

Step two involves the estimation of task duration. This is where CPM and PERT diverge. CPM takes a deterministic approach to the determination of task duration while PERT incorporates uncertainty into task duration.

8.3.8.1 CPM

CPM techniques utilize a deterministic approach to the determination of task duration. In step two, this simply involves estimating the time required to complete each task and using this assigned value.

Proceeding, the third step of evaluating the project involves determining the minimum duration of the project. The key to this step is to recognize that the minimum duration of the project is defined as the sequence of activities that yields the longest path or critical path through the network.

The estimated task durations found in the second step are used to calculate quantities for each node of the network called earliest and latest time. The *earliest time* is defined as the time at which the event will occur if all of the predecessor activities are started as early as possible and can be calculated on a forward pass through the network. Mathematically, the earliest time for any node j (ET_j) can be defined as:

$$ET_j = \max\ (EF_{i_k}, \text{ for all } i_k) \tag{8.3.53}$$

where (i_k) is a task ending at node j, EF_{i_k} is the earliest finish time of task i_k, and $EF_{i_k} = ET_i$ + task duration of task i_k.

The *latest time* for a node or event is the latest possible time that the event can occur without delaying the completion of the project and can be calculated from the earliest times on a backward pass through the network. Mathematically, the latest time for any node i (LT_i) can be defined as:

$$LT_i = \min\ (LS_{i_k}, \text{ for all } i_k) \tag{8.3.54}$$

where i_k is a task starting at node i, LS_{i_k} is the latest start time of task i_k, and $LS_{i_k} = LT_j$ – task duration of task i_k.

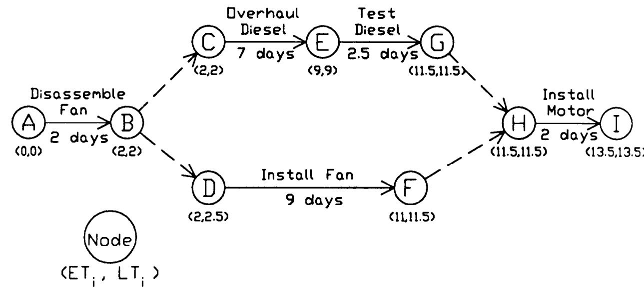


Fig. 8.3.17. Solved CPM network of a simplified fan move.

Once the earliest and latest times have been determined for each node or event in the network, the *critical path* is then defined as those tasks for which the earliest time is equal to the latest time. The critical path or paths (there may be more than one critical path within a network) represents the series of tasks or activities in which no delay in their start or finish can be tolerated without delaying the project as a whole.

Other nodes that are not considered critical have what is termed float or slack time. This *float time* is the difference between latest and earliest times and represents flexibility of task initiation by a project manager. A project manager can use float or slack time to delay initiation of various project tasks in order to better schedule manpower, equipment, or other company resources without delaying project completion.

A CPM analysis of the example illustrated in Fig. 8.3.16 yields the solution shown in Fig. 8.3.17. The figure shows the critical path by heavy lines, with the earliest and latest times of each node indicated.

8.3.8.2 PERT

The PERT technique utilizes a probabilistic approach to task duration. This approach is called the three-estimate approach. CPM assumes that task durations are known with certainty, while PERT in contrast accounts for the possible uncertainty in the activity duration by defining all task durations as a random variable conforming to a probabilistic distribution.

Each task or activity in a PERT analysis requires three estimates for its duration. These estimates include a "most likely" (usually denoted as *m),* an "optimistic" or minimum (denoted by *a),* and a "pessimistic" or maximum (denoted by *b).* PERT then assumes a beta distribution where the expected completion time t_e is approximated as,

$$t_e = \frac{1}{3}\left[2m + \frac{1}{2}(a + b)\right] \tag{8.3.55}$$

Next PERT requires that a variance is calculated for each task duration. This value for the beta distribution is approximated as

$$\sigma^2 = \left[\frac{1}{6}(b - a)\right]^2 \tag{8.3.56}$$

The third step of PERT, that of project evaluation, involves once again determining the minimum duration of the project. The expected values of the task durations t_e found in the second step are the values utilized. The process is then to use the t_e values to determine the critical path of the project, identically as with the CPM technique.

After the critical path has been determined, the probabilistic information is used to give a probabilisitic description of the

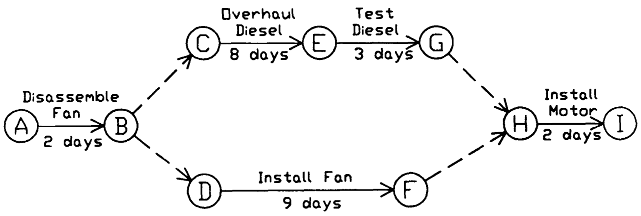


Fig. 8.3.18. Initial PERT network of a simplified fan move.

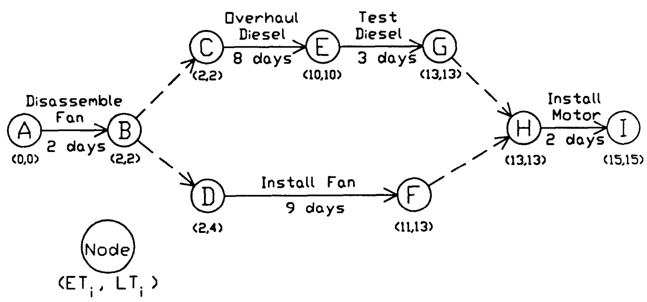


Fig. 8.3.19. Solved PERT network of a simplified fan move.

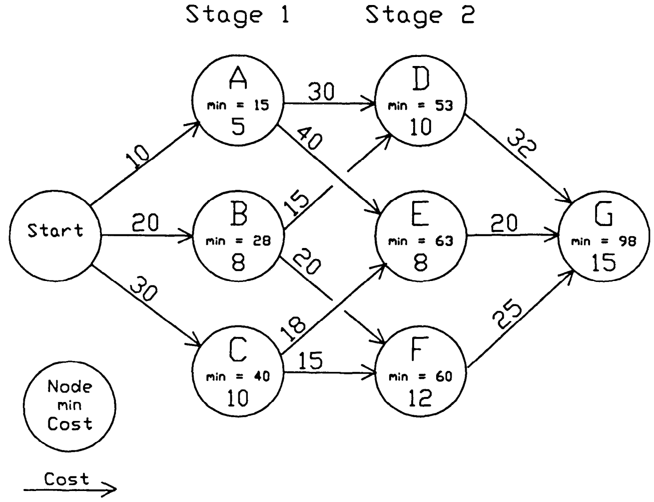


Fig. 8.3.20. Example dynamic programming network.

project completion time. Since any path through the network represents a sum of random variables, and PERT assumes that these random variables are statistically independent, the sum of the variances along each path represents the variance of the path. The critical path represents the project duration so that its mean and variance therefore reflect those of the project as whole. The central limit theorem then implies that such a sum is approximately normal so that, given the mean and variance, it is then possible to find the probability that the project completion time will be less than any given value. Likewise, the same technique can be used to find the probability that any given subevent within the project will be completed before some specific time.

Example 8.3.8. Assume that the following are the estimated time durations for each task using the PERT three-estimate approach of the original project illustrated in the network shown in Fig. 8.3.18.

	Duration				
Task	m	a	b	t_e	σ_e^2
Disassemble fan	2	2	2	2	0
Overhaul diesel	7	6	14	8	16/9
Install fan	9	8	10	9	1/9
Test diesel	2.5	2	6	3	4/9
Install motor	2	2	2	2	0

Solution. The resulting task durations denoted by the calculation of t_e are then used to determine the critical path as shown in Fig. 8.3.18 and yield the solution illustrated in Fig. 8.3.19. The total project duration according to the PERT analysis is increased to 15 days. The next step in a PERT analysis is to determine the expected variance along the critical path. This is simply the sum of the variance terms associated with each activity which lies on the critical path. Numerically, the value is 20/9.

Since the sum of independent ransom variables tend to approximate a normal distribution, the area under curve from negative infinity to some given time duration represents the probability that the project will be completed in that time or less. This is provided that the variability of noncritical paths is ignored. In the previous example (Example 8.3.8), this indicates that there is a 50% probability of completing the task in less than 15 days (mean value), a 90% probability of completing the project in less than 17.45 days (mean + 1.645 standard deviations), and a 95% chance of completing in less than 17.9 days (mean + 1.940 standard deviations).

8.3.9 DYNAMIC PROGRAMMING

Dynamic programming does not have a standard mathematical formulation in which a generalized algorithm can be applied to derive a solution, such as the simplex method for linear programming problems. Instead, *dynamic programming* is a systematic procedure for building and analyzing mathematical relationships that describe the operational characteristics of sequences of interacting decisions, taking into account the interrelationship of multistage systems. Beightler et al. (1979) refers to dynamic programming as conditional or partial optimization schemes.

The most common examples of dynamic programming techniques involve serial multistage systems. These involve a series of decisions in which it is necessary to determine which combination of decisions maximizes the operational performance of the entire system. Dynamic programming recognizes that solutions to this sort of multistage problems do not require total knowledge of every possible path to the states in each stage. Instead, the concept looks at how to get to the current state from the previous stage at a maximum profit or minimum cost.

Dynamic programming as applied to serial multistage systems can be illustrated by a simple network-type decision problem. Assume that a mine is interested in building an overland conveyor system that will be made up of five different belts. It is planned to build four transfer stations, for each of which there are several different site alternatives available along the planned route. The resulting network along with associated costs is shown in Fig. 8.3.20.

The dynamic programming concept simply recognizes that determining the minimum cost of building the belt system to any given node does not require investigating every possible route to that node. It requires only that the cost of the belt link from a predecessor site and the cost of the transfer station be added to the known minimum cost of reaching each predecessor node in

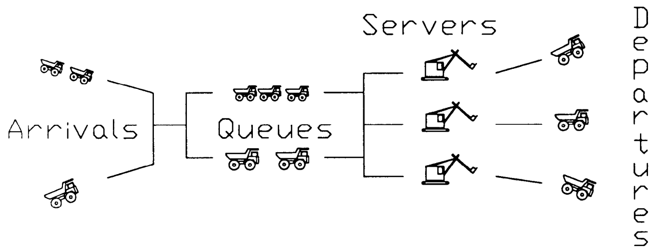


Fig. 8.3.21. Major components of typical queuing models.

the previous stage. The minimum cost of reaching a given site is then the minimum of all possible ways of getting to that state from the previous stage. Thus only partial paths need be examined to determine an optimal value. For example, the minimum cost of constructing the system node G in Fig. 8.3.20 is found by:

$$\text{Min} \begin{cases} \text{Minimum Cost to D} + 32 + 15 \\ \text{Minimum Cost to E} + 20 + 15 \\ \text{Minimum Cost to F} + 25 + 15 \end{cases}$$

In a very small network, such as that illustrated, the computational saving over investigating through complete enumeration many times are minimal. However, in many multistage networks, the computational requirements can soar very rapidly. For example, if the previous belt system had 10 links instead of 5, and there were 10 alternative sites for each of the transfer stations, then exhaustive enumeration would result in 10^9 total paths. The dynamic programming approach would evaluate only approximately 810 partial paths (Beightler et al., 1979).

8.3.10 QUEUING THEORY

Queuing theory is a branch of mine systems engineering that involves the mathematical analysis of waiting lines or queues. The theory primarily deals with the formal mathematical description of the waiting lines that form behind service facilities. In particular, queuing theory is used to determine the crucial characteristics of a waiting line such as average queue length, average waiting time, and average idle time of the service facility, given that both waiting-line formation and service times are stochastic processes. The theory itself is not a direct optimization technique. It only deals with providing a description of the waiting line. The description of the queue provided by the mathematical model is used as input for optimal decision making by an analyst.

Queuing problems tend to deal ultimately with trying to determine the optimum number of servers, efficiency of the servers, and required number of service facilities. In mining, the classic example of a queuing problem is trucks waiting in line to be loaded in a truck-shovel mining operation. With these problems the goal is to evaluate the optimum arrangement of trucks and shovels or to try to assist in the selection of trucks and shovels.

There are four major components common to most queuing models, as illustrated in Fig. 8.3.21. These include:

1. Arrivals.
2. Queue(s) or waiting lines.
3. Service facilities.
4. Departures.

Arrivals may originate from one or several different sources and are usually randomly spaced over time, conforming to some sort of probability distribution. These distributions are usually defined by time studies of an operation. Once an arrival takes place, it is either serviced by one of the servers, or if all servers are busy, it is placed in line (or in one of several lines). Here the arrival waits until its turn for service. The service facilities may be one shovel or several serving a single or multiple lines. The amount of time required for service (e.g., filling a 200-ton, or 180-t, truck) is also somewhat randomly distributed conforming to some sort of probability distribution. When service is completed, the arrival then departs from the system.

The nature of the queues is one of the controllable aspects of any queuing system. For example, a waiting line might be a first-come-first-serve queue; it might be subject to some sort of priority arrangement; or it might be possible for arrival to switch between queues, balk (leave if length is excessive), or renege (leave after joining a queue). The overall performance of a system can sometimes be improved by simply changing how the queues are handled.

The waiting line optimization problem is one of determining the number of service facilities and queue priorities or rules such that a minimum cost is obtained by a company. A company incurs cost if it has servers (such as a shovel) idle. A company also incurs a cost if it has too many customers (trucks) waiting on the shovel. Thus the optimization problem becomes one of minimizing the total cost to the company.

Most mining applications of waiting-line theory tends to be highly complex. Accurate modeling of most mining situations results in complex queuing models that have no direct analytic solution. Usually, even the more complex cyclic queuing models are simple approximations of the actual queue and tend to serve only as a first approximation to the behavior of the system. As a result, stochastic simulation techniques are generally employed as the practical choice of representing the true features of a waiting line or queue. Those interested in the mathematical formulation of these simple queuing models should refer to Daellenbath and George (1978), Panico (1969), or Hillier and Lieberman (1986).

8.3.11 ARTIFICIAL INTELLIGENCE

Artificial intelligence (AI) can be generally defined as “the development of computer systems to perform cognitive tasks that are, at present, more appropriately performed by humans” (Rich, 1983). Four broad AI categories can be identified. These are (1) natural language understanding, (2) machine vision and scene recognition, (3) robotics, and (4) expert systems. AI has its roots in the development of modern computational equipment in the 1950s and so is a very young and emerging area.

Natural language understanding is aimed at creating computers that can recognize continuous speech, and subsequently determine the deep meaning of what is spoken. Speech recognition has developed rapidly, and much progress has been made in this area. However, universal and continuous speech recognition (i.e., speaker and language independent) has not been achieved to date. Natural language understanding is still in its infancy. While much research has been done to advance this area, machines that can truly understand unbounded discourse are far from reality.

Machine vision attempts to duplicate both the ability to record a scene and to incorporate the complex scene processing of human vision. his includes such tasks as depth perception, motion cues, and scene recognition. All these visual tasks come naturally to humans, but represent very complex mechanisms that are not yet fully understood. While there are many general similarities between eyes and cameras, these are largely along the lines of physical optics. Research into how the eye actually works began with Helmholz in the 1880s. However, many questions remain unanswered. We still do not fully understand the retina and the action of neural processes behind the primary visual receptors. Beyond this, we have no firm understanding of how our brains construct an image from the retinal information. Recent work on edge detection models of the retina has proved of value, but only in a limited way. From the standpoint of applications, machine vision holds much promise for mining. Anywhere that human vision is essential to operation could potentially be replaced or improved by machine vision. Coupled with robotics, machines could be designed to operate remotely and in hazardous situations unrestricted by considerations of human safety.

Robotics, while somewhat more familiar to the general public through the media, is nevertheless distinct from that popular view. Robotics is the attempt to provide machines with much the same freedom of motion and mechanical interaction with the environment as is shown by humans. This includes tactile sensory feedback (such as pressure, temperature, and texture), balance, position sensing, and adaptation to obstacles of travel (collision avoidance). Simple robots are widely in use in well-controlled applications (such as assembly line work). Where more complex situations are found and adaptation is needed, robots are still being developed. While machine vision would solve the problem of scene recognition and understanding, robotics seeks to provide mechanical devices capable of responding to the information obtained from the visual processing of the environment. The average person thinks little of driving a car, but this is a tremendously complex problem for a machine. Recent development of an Autonomous Land Vehicle (ALV) shows some promise for the future, but this system is capable of navigating only well-constructed test roads. Again, much work is needed to advance robotics to levels approaching human performance.

The last area is *expert systems.* An expert system is, in effect, a computer counterpart to a human expert in a particular field (Rolston, 1988). Both the knowledge of the expert and the rules of problem solving are captured in an expert system. The role of the expert system is to gather facts about a problem in a directed manner, much as an expert would do, so that only those facts that are pertinent to the analysis are obtained. The expert system then "reasons" to a certain conclusion. This could be a simple statement concluding that the evidence submitted is indicative of an identifiable state of affairs, or that some facts are inconsistent with known parameters. The conclusion is based on the responses provided and the rules present in the rule base. Depending on the "shell" (software that implements the expert system), the entire reasoning process can be traced and inquiries can be made as to why the expert system provided a particular recommendation and what changes in facts or parameters will influence the outcome. This is much like the use of sensitivity analysis in numerical studies. The decision-making process can deal with both certain and uncertain information, thus imitating the reasoning process of a human expert. If uncertainty is included in the reasoning process, then it is possible that the result is a list of ranked conclusions rather than one value for an assertion.

Some examples of expert systems in earth sciences and engineering are worthy of note. An expert system was developed in 1987 to address issues related to natural resource management by researchers at Texas A & M University (Folse and Loh, 1987). GEOTOX, an expert system to analyze hazardous waste disposal sites, was developed at Lehigh University in 1986 (Wilson et al., 1987). The Environmental Protection Agency developed in 1986 an expert system to process effluent discharge permits. This system automatically adjusts the analysis based on the individual states' permit requirements. Prospector, developed by Science Research Associates, is an expert system which duplicated the "common sense" of a field geologist and a prospector (Duda et al., 1979). Its focus was to aid in locating mineral deposits. Other examples of expert systems in mining include blasting round design, ventilation, network analysis, maintenance and diagnostics of underground machinery, electrical system faults, and mining permit evaluation (Albert, 1989a).

Artificial intelligence remains a rapidly developing research field encompassing first computer science and subsequently many related disciplines. While some practical applications have emerged (such as expert systems), most of the fundamental goals of AI research have yet to be realized. Clearly, the nature of mechanized equipment, including that used in mining, will benefit greatly from advances in this area.

Other discussion of artificial intelligence may be found in Chapter 22.2 of this *Handbook.*

8.3.12 EXAMPLE OF A STOCHASTIC MODEL

A *deterministic model* is one in which a given set of inputs always produces the same set of outputs. However, in studying real-world systems, certain measurements or outcomes can only be described by an observed or expected distribution of outcomes—a random quantity. Generally, whenever a model includes random variables, it is considered to be a *stochastic model* (see 8.3.4) (Hillier and Lieberman, 1986).

Simply stated, the term *random* means without plan or order. Thus a random variable cannot be described by an equation, since the ordered rules of mathematics precludes this very situation. Instead, a random variable can be characterized by its distribution—an expression of the probability density for the observed set of outcomes (Emshoff and Sisson, 1970). Despite having constructed such a distribution, this only provides the probability of outcomes, not an explicit prediction of individual outcomes. Indeed, computer algorithms (systems of equations) to generate random numbers are actually pseudorandom.

In constructing stochastic models, variables in the system are recognized (or chosen explicitly) as stochastic. For example, virtually any task involving human labor will demonstrate a random variation that can be described by a distribution (Maisel and Gnugoli, 1972). Time study data plotted as a frequency distribution (or histogram) will almost always have this characteristic.

The goal of modeling is primarily to reflect the essential attributes of a system in a mathematical expression, thus duplicating the behavior of the system without constructing or altering the "real-world" counterpart. Of course, one is not restricted to known physical systems in the construction of a model. In either case, the behavior of the system is the result of the interaction of the entities of the system through the variables that represent the state of nature of each entity. The model is said to simulate the system of interest.

An important issue involved with modeling of any sort is the level of abstraction. A model with a high level of abstraction is one which includes only major and essential variables. Conversely, a model constructed with a low level of abstraction includes all of the major details of a system and many less important ones as well. It is usually the case that a model with a low level of abstraction can provide more detailed (and therefore more accurate) reflection of the modeled system. However, merely including more detail in a model does not guarantee a more accurate or even more realistic simulation.

Example 8.3.9. To illustrate these concepts and demonstrate a practical approach to stochastic modeling, consider a shovel-

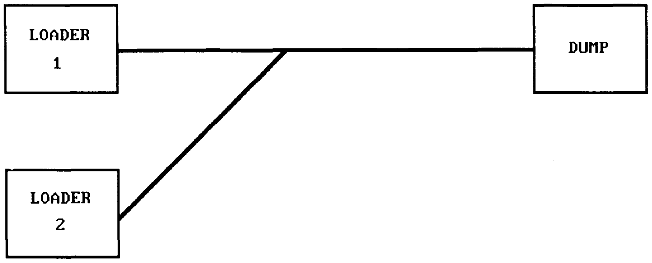


Fig. 8.3.22. Truck/shovel haulage diagram.

truck haulage system operating as in Fig. 8.3.22. At a high level of abstraction, this system embodies the load and haul deterministic model of the previous segment (8.3.4). Viewing this system as a collection of variables (in the load-haul-dump equation), the following time relationship for a truck is derived (Albert, 1979):

$$TCT_{\text{truck}} = LTT + TL + DT + TE + ST \qquad (8.3.57)$$

where TCT_{truck} = total cycle time for one truck load, from beginning of loading to next, LTT = total time to load a truck by shovel, TL = travel loaded time to dump, DT = dumping time, TE = travel empty time, and ST = spot time for truck before loading.

Note that no provision for mechanical delays is explicitly included, and any waiting time is considered separately.

Solution. To simplify the expression further, the variables TL, DT, TE, and ST are lumped together into one variable, namely, TA (truck away time from loader). The model is now formed with only two distributions: one for the load time of the truck and one for the away time of a truck (the time from the dispatch event to the return of the truck for loading again). Now assume that each of these distributions is normally distributed for the sake of demonstration (in real life, they usually are not).

To create the simulation, consider that there are two loaders and six trucks. The mean time to load a truck is 4 min with a standard deviation of 1 min, and the away time of a truck is 16 min with a standard deviation of 2 min. As indicated, the times will be modeled with a normal distribution. Every instance in the model that a time is needed for either a loading time or an away time, the following equation is employed:

$$Ts = MeanTime + (NDRD * StdDev) \qquad (8.3.58)$$

where Ts = time sample for this event, $MeanTime$ = mean time for event, $NDRD$ = normally distributed, random number sample (mean= 0, standard deviation = 1), and $StdDev$ = standard deviation of time for event.

The model proceeds from event to event, creating an event-oriented simulation. This means that the time clock in the model will jump from each loading event to the next based on the times sampled for each sequential event.

The procedure to determine an particular sampled time (the time to load a truck or the time a truck is away from the loader) is to substitute the distribution parameters in the equation, obtain a normally distributed random number *(NDRD)*, and apply the equation. The normally distributed random number stream is created by adapting the uniform random function supplied in the programming language environment.

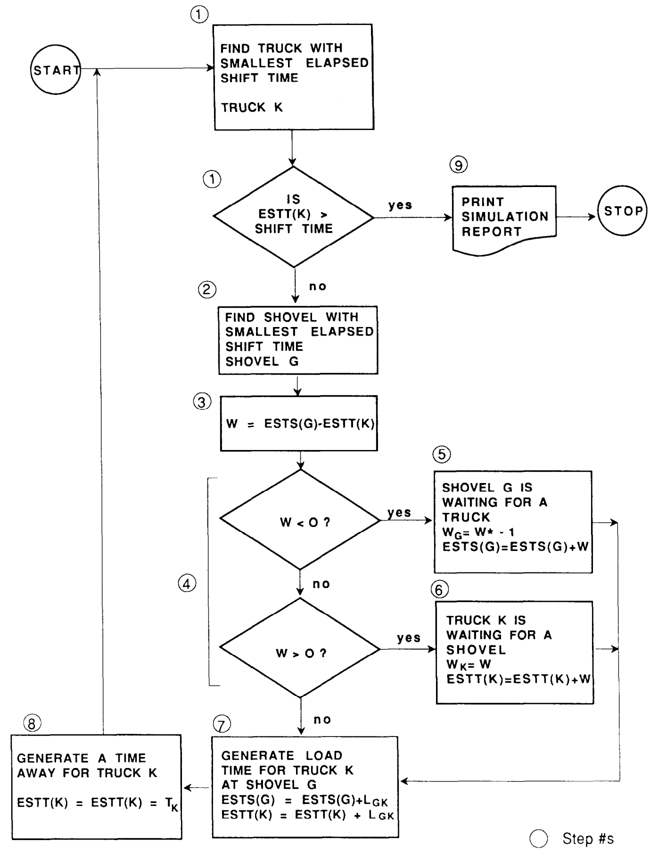


Fig. 8.3.23. Flow diagram for event model.

Fig. 8.3.23 shows a flowchart for the simulation model. It has been generalized to any number of trucks and loaders, so it is not explicitly limited to the two loaders and six trucks used for this demonstration. Refer to the flowchart for the steps described in the following that are a procedural definition of the model.

Next define the following:

M = number of shovels in the system.
N = number of trucks in the system.
$ESTS(i)$ = elapsed shift time of shovel i.
$ESTT(i)$ = elapsed shift time of truck i.
ST = ending time for simulation end.

To perform the simulation, the steps below are executed (Ramani and Albert, 1985). These steps correspond to the labels on Fig. 8.3.23.

1. Choose the truck with the smallest elapsed shift time. Let it be the Kth truck.

If $ESTT(K) \geq ST$ THEN GO TO step 9

2. Choose the shovel with the smallest elapsed shift time. Let it be the Gth shovel.

3. Calculate W as follows:

$$W = ESTS(G) - ESTT(K) \qquad (8.3.59)$$

4. If $W < 0$ GO TO step 5 [Shovel waited for a truck]
If $W \geq 0$ GO TO step 6 [Truck waited for a shovel]
If $W = 0$ GO TO step 7 [No waiting—exact match]

5. Shovel G was waiting for a truck to become available to load. Therefore, W is the waiting time of the shovel.

Update the event time for shovel G:

$$ESTS(G) = ESTS(G) + W \tag{8.3.60}$$

GO TO step 7

6. Truck K was waiting for a loader to become available. Therefore, W is the waiting time for truck K.

Update the event time for truck K:

$$ESTT(K) = ESTT(K) + W \tag{8.3.61}$$

GO TO step 7

7. Generate a load time for truck K at the Gth shovel. Call this time L_{GK}.

Update the status of the shovel and truck by the load time:

$$ESTS(G) = ESTS(G) + L_{GK} \tag{8.3.62}$$

$$ESTT(K) = ESTT(K) + L_{GK} \tag{8.3.63}$$

8. Generate an away time for truck K. Call this T_K.

Update the status of the truck:

$$ESTT(K) = ESTT(K) + L_{GK} \tag{8.3.64}$$

GO TO step 1

9. Produce end of simulation report since the time threshold has been equalled or exceeded by the time of the next event.

Table 8.3.18 is a listing of a program written in Microsoft™ Quick BASIC to perform the simulation just described. Applying the information for time distributions and costs, Table 8.3.19 presents the summary report from running the model.

Table 8.3.18. Truck/Shovel Simulation Program

```
'
' Programmed by Eric K. Albert
'
  RANDOMIZE TIMER
'
' Set up the arrays (tables) to hold the values for
' simulation
'
  DIM loader.wait(20), truck.wait(20), ESTT(20), ESTS(20), trk.load.time(20)
  DIM load.time.mean(20), load.time.dev(20), away.time.mean(20)
  DIM away.time.dev(20), trk.trips(20), ldr.loads(20), away.time.total(20)
  F$ = "####.##"
'
' The following section obtains from the user the pertinent data
' for running the problem
'
begin: CLS
  PRINT TAB(20); "Event Oriented Truck/Shovel Simulator"
  PRINT
  PRINT "Please enter all times in MINUTES"
  PRINT
  INPUT "Enter simulation time = "; total.simulation.time
100   PRINT
      INPUT "Enter number of loaders = "; nbr.loaders
      IF nbr.loaders < 1 OR nbr.loaders > 10 THEN 100
      PRINT
      INPUT "Enter the cost PER HOUR for a loader = "; loader.cost
      INPUT "Enter the revenue per truck loaded = "; revenue.per.load
110   PRINT
      INPUT "Enter beginning number of trucks in fleet = "; NS
      INPUT "Enter ending number of trucks in fleet = "; NM
      IF NM > 20 OR NM < 1 OR NM < NS OR NS < 1 THEN 110
      PRINT
      INPUT "Enter the cost PER HOUR for a truck = "; truck.cost
'
      CLS
      PRINT "Enter (Mean time,Standard deviation)—Away Time in Minutes"
      PRINT
      FOR I = 1 TO NM
      PRINT "For Truck "; I; " ";
      INPUT away.time.mean(I), away.time.dev(I)
      NEXT I
'
      CLS
      PRINT "Enter (Mean time,Standard Deviation)—Loading Time in Minutes"
      PRINT
```

Table 8.3.18. Truck/Shovel Simulation Program—cont.

```
      FOR I =  1 TO nbr.loaders
      PRINT "For Loader "; I; " ";
      INPUT load.time.mean(I), load.time.dev(I)
      NEXT I
'
      CLS
      FOR NX = NS TO NM
      PRINT "SIMULATING "; NX; " TRUCKS"
'
' The next section does the necessary initialization of variables
'
      FOR I = 1 TO 20
        loader.wait(I) = 0
        truck.wait(I) = 0
        truck.wait(I) = 0
        ESTT(I) = 0
        ESTS(I) = 0
        trk.load.time( I) = 0
        trk.trips(I) = 0
        ldr.loads(I) = 0
        away.time.total(I) = 0
      NEXT I
'
' The main body of the simulation begins here
'
next.event:
'
' Find the truck with the shortest elapsed shift time
'
      GOSUB findtruck
'
' Is the next truck (event) time after the total elapsed shift time
' entered by the user? i.e., is it time to stop the simulation?
'
      IF S >= total.simulation.time GOTO print.report
'
' Find loader with shortest elapsed shift time
'
      GOSUB findloader
'
' Calculate W, to determine the type of event we are seeing
'
      W = R—S
'
' Create an event printing on screen to show status
'
      T$ = STR$(K)
      L$ = STR$(G)
      IF LEN(T$) = 1 THEN T$ = " " + T$
      IF LEN(L$) = 1 THEN L$ = " " + L$
      PRINT "EVENT = Truck("; T$; ") =";
      PRINT USING F$; S;
      PRINT " Loader("; L$; ") = ";
      PRINT USING F$; R;
      IF W > 0 THEN PRINT " (truck waited for loader)"
      IF W < 0 THEN PRINT " (loader waited for truck)"
      IF W = 0 THEN PRINT " (exact time—no wait)"
'
' Branch to appropriate section of code based on sign of W
'
      IF W > 0 GOTO trk.waited
      IF W < 0 GOTO ldr.waited
'
' Exact match indicates that nobody waited
'
```

Table 8.3.18. Truck/Shovel Simulation Program—cont.

```
        GOTO load.truck
'
' The loader waited for the truck
'
ldr.waited:
        W = W * ( – 1 )
'
' Add to total loader wait time for this loader (G)
'
        loader.wait(G) = loader.wait(G) + W
'
' Update elapsed shift time counter ESTS
'
        ESTS(G) = ESTS(G) + W
        GOTO load.truck
'
' The truck waited for the loader
'
trk.waited:
'
' Add to total truck wait time for this truck (K)
'
        truck.wait(K) = truck.wait(K) + W
'
' Update elapsed shift time counter ESTT
'
        ESTT(K) = ESTT(K) + W
'
loadtruck:
        GOSUB getrand
'
' Generate a new load time for this loader and truck
'
        new.load.time = load.time.mean(G) + RN * load.time.dev(G)
        ESTT(K) = ESTT(K) + new.load.time
        ESTS(G) = ESTS(G) + new.load.time
'
' Add up the total load time for this truck
'
        trk.load.time(K) = trk.load.time(K) + new.load.time
'
' Generate a new away time for this truck
'
        GOSUB getrand
        TA = away.time.mean(K) + RN * away.time.dev(K)
        ESTT(K) = ESTT(K) + TA
'
' Add to the statistics: total truck away time, total number of trips,
' and the total loads of the loader
'
        away.time.total(K) = away.time.total(K) + TA
        trk.trips(K) = trk.trips(K) + 1
        ldr.loads(G) = ldr.loads(G) + 1
'
        GOTO next.event
'
' At this point, the simulation is over and we print a report
'
print.report:
        CLS
        T2 = 0
        T3 = 0
        T4 = 0
        T5 = 0
        PRINT "Simulation results at elapsed time = "; S
```

Table 8.3.18. Truck/Shovel Simulation Program—cont.

```
        PRINT
        PRINT "Truck Statistics"
        PRINT
        PRINT "Truck", "Loads", "Wt/Loader", "Load Time", "Away Time"
        FOR I = 1 TO NX
          PRINT I, trk.trips(I), ;
          PRINT USING F$; truck.wait(I); : PRINT SPC(7);
          PRINT USING F$; trk.load.time(I); : PRINT SPC(7);
          PRINT USING F$; away.time.total(I)
          T2 = T2 + truck.wait(I)
          T3 = T3 + trk.load.time(I)
          T4 = T4 + away.time.total(I)
          T5 = T5 + trk.trips(I)
        NEXT I
        PRINT "Totals—> ", T5, ;
        PRINT USING F$; T2; : PRINT SPC(7);
        PRINT USING F$; T3; : PRINT SPC(7);
        PRINT USING F$; T4
        GOSUB keypress
        CLS
        PRINT "Loader Statistics"
        T21 = T2
        PRINT : PRINT "Loader", "# of Loads", "Idle Time", "Loading Time"
        T2 = 0
        T3 = 0
        T4 = 0
        FOR I = 1 TO nbr.loaders
          PRINT I, ldr.loads(I), ;
          PRINT USING F$; loader.wait(I); : PRINT SPC(7);
          PRINT USING F$; (total.simulation.time – loader.wait(I))
          T3 = T3 + ldr.loads(I)
          T4 = T4 + loader.wait(I)
          T2 = T2 + (total.simulation.time – loader.wait(I))
        NEXT I
        PRINT "Totals—> ", T3, ;
        PRINT USING F$; T4; : PRINT SPC(7);
        PRINT USING F$; T2
        PRINT
'
' The cost function here is simplistic, but a reasonable starting point
'
' The cost of trucks is considered to be the cost per hour per truck
' no matter what they are doing
'
' The cost for a loader is considered to be the cost per hour, and this
' is offset by the value of the material moved per load
'
        system.cost = (T3 * revenue.per.load)—(S * (truck.cost / 60!) * NX + (loader.cost * S *
                      nbr.loaders / 60!))
        PRINT "Total system revenue (– is loss) per hour = "; system.cost / (S
        / 60!)
        GOSUB keypress
        CLS
        NEXT NX
        INPUT "Simulation complete – Do you want to run another problem (y/n)
        "; A$
        IF LEFT$(A$, 1) = "Y" OR LEFT$(A$, 1) = "y" GOTO begin
        END
'
' The support subroutines follow here
'
findtruck:
        S = 999999!
        K = 0
        FOR I = 1 TO NX
          IF ESTT(I) < S THEN S = ESTT(I): K = I
```

Table 8.3.18. Truck/Shovel Simulation Program—cont.

```
      NEXT I
      RETURN
'
findloader:
      R = 999999!
      G = 0
      FOR I = 1 TO nbr.loaders
        IF ESTS(I) < R THEN R = ESTS(I): G = I
      NEXT I
      RETURN
'
getrand:
'
' Routine returns a normally distributed random number with
' a mean of 0 and a standard deviation of 1 in variable RN
'
      RN = 0
      FOR I = 1 T0 12
        RN = RN + RND(1!)
      NEXT I
      RN = RN – 6!
      RETURN
'
keypress:
      LOCATE 24, 1
      PRINT "Press any key to continue";
      BEEP
      WHILE INKEY$ = ""
      WEND
      RETURN
```

Table 8.3.19. Sample Report for Simulation Program

Simulation results at elapsed time = 484.3102

Truck Statistics

Truck Time	Loads	Wt/Loader	Load Time	Away
1	25	9.86	96.77	390.61
2	24	12.26	91.65	387.64
3	25	7.73	97.68	387.87
4	24	13.62	96.87	389.20
5	24	7.06	93.75	383.50
6	24	15.32	99.19	370.40
Totals	146	65.84	575.91	2309.23

Loader Statistics

Loader	# of Loads	Idle Time	Loading Time
1	71	201.52	278.48
2	75	188.74	291.26
Totals	146	390.26	569.74

Total system revenue (– is loss) per hour = 178975.8

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Chapter 8.4
COMPUTER METHODS

KHOSROW BADIOZAMANI

8.4.1 INTRODUCTION

To provide a description of computer methods used in mining, one must first define the area of interest. The extension of computer usage to the many facets of mining is too numerous and complex to be described in a short chapter. Today computers are used in all areas of mining, from exploration and drilling to mine planning and mine design, mine operation, reclamation, surveying, maintenance, inventory control, accounting, payroll, and marketing, to name a few. For computer applications in mining, the reader is referred to the appropriate sections and chapters of this *Handbook.*

In this chapter, concentration is on the scientific and technical aspects of mining and only on major topics. The plan is to examine the what, where, how, and when aspects of computer usage, that is, what a computer is used for, where it is used, how it is used, and when it should be used. Then the concepts, advantages, and disadvantages of computer usage are expanded to provide a better understanding of the details involved in its usage. In addition, an attempt is made to provide some guidelines on how to select a system of hardware and software. It is understood that rapid changes in the state-of-the-art of hardware and software technology make it difficult to establish long-lasting selection criteria; however, the main ideas and concepts should stand the test of time and provide useful and general guidelines.

8.4.1.1 Where Computers Are Used

Computers are used for storage, retrieval, and analysis of geologic data plus design, simulation, and monitoring of mine plans and mining operations.

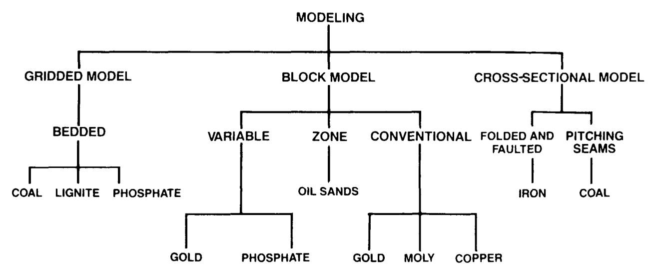
The majority of effort in computerization has been concentrated in the areas of storage, retrieval, and modeling. Early efforts were concentrated on automating manual methods to reduce error and increase efficiency. As the user's acceptance of the automated methods has increased, along with the complexity of ore bodies, the need for more sophisticated techniques has moved computer usage to a new frontier. Geostatistics and complex mathematical modeling provide ore-reserve approximation techniques and simulation approaches that are basically impractical to perform manually.

Blind application of computers to all situations, however, is not advisable. Computers should only be used where they provide an immediate or long-range advantage. For situations with little data and only one time evaluation or calculation, it may be easier and less costly to use manual methods. It should be remembered that computers are another tool in the engineer's tool kit. They have their place and can be useful if used properly and can complicate the situation if not.

8.4.1.2 How Computers Are Used

Computer applications vary with the type of mineral deposit and the complexity of the ore body. The basic ore body modeling concept, however, remains the same even though the method of compositing varies from one type of deposit to another. Table

Table 8.4.1. Decision Tree



8.4.1 provides a simple look at the classification of modeling based on the type of commodity, stratabound vs. disseminated type of deposits, and the locale of mining, that is, surface or underground. From this table, it is obvious that to provide a complete explanation for all available options is not practical within the space allotted to this *Handbook.* As a result, except for conditions common to all deposits and types of mining, our discussion will concentrate on the stratabound type of deposits and mostly on surface applications.

8.4.1.3 Basic Concepts in Use of Computers

The main objective of computer application is to simplify the process of data collection, retrieval, analysis, and modeling. In addition, computer modeling removes the possibility of an undetected error in calculation and allows evaluation of more alternatives in a shorter period of time than is possible by the manual approach. As a result, computer modeling will assist in achieving the ultimate goal of mine planning, which is to provide the "best" strategy for developing and mining a deposit. The best may not necessarily be the optimum and will be dependent on the goals set by each corporation. The reason is that to obtain an optimum solution, one may need to spend a great deal of computer resources, and at the end the optimum solution may not be practical.

As easier deposits are exhausted and the need for designing more complex mine plans become commonplace, computers can assist in reducing tedious tasks and allow the specialist to concentrate on his/her area of expertise, which is mine design.

Computers can be regarded simply as devices that are fed raw information. These raw data are processed, analyzed, and modeled, and the result is an output in the form of reports, maps, and graphs. In other words, the operation is based on three simple activities: input, processing, and output (Fig.8.4.1). The main task is to make sure that the correct information is contained in the input and the right type of process is selected to assure correct results as output.

A major benefit of computerization is compulsory collection of all data in advance of modeling. This process in actuality forces many users to collect all data, maps, and information from various individuals involved in the project in one area rather than having them scattered in different file cabinets and drawers.

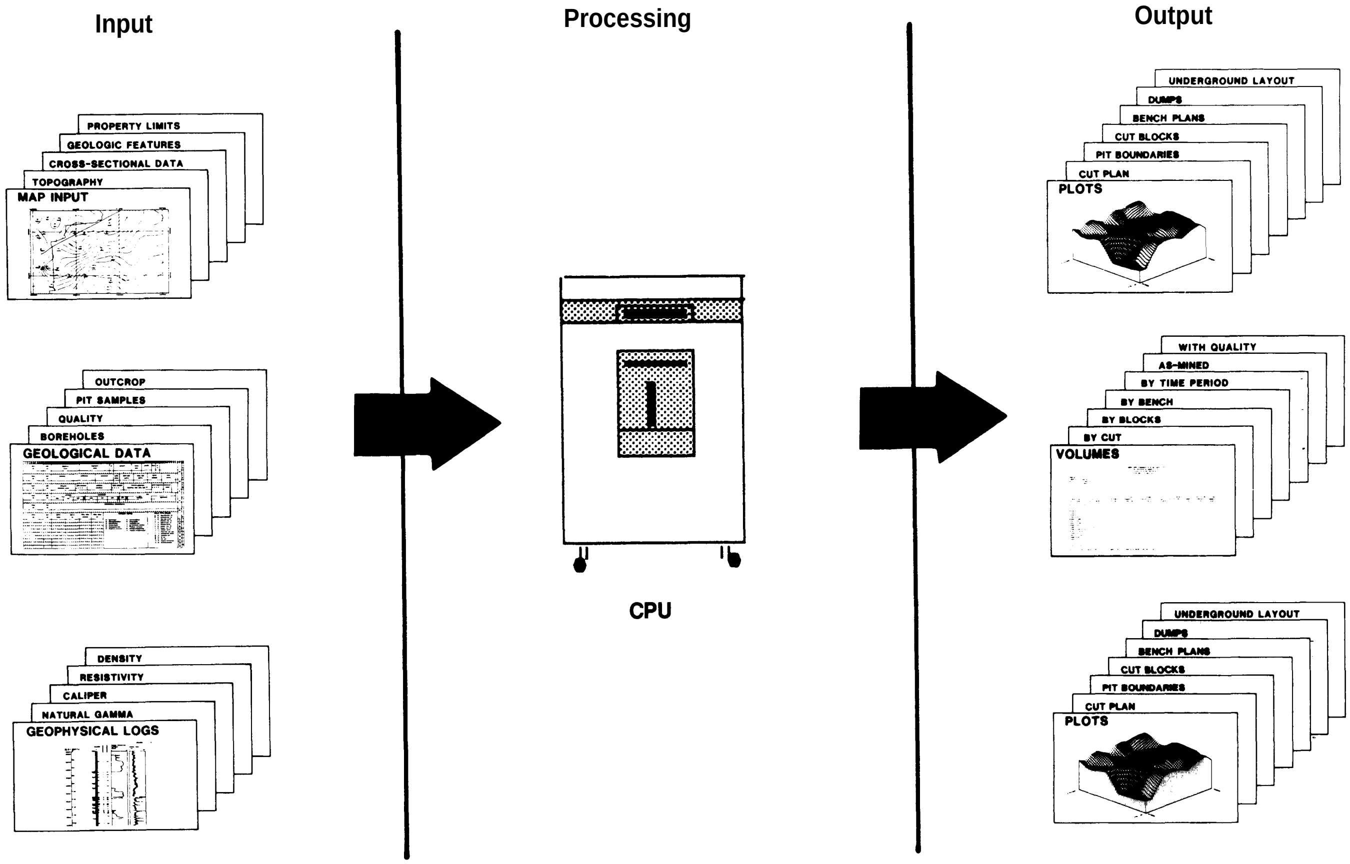


Fig. 8.4.1. Stages of computer processing.

The initial collection and input of data into computers forces purging of invalid and incorrect data or simply forces cleansing of information before use. In fact, a sign of a well-designed system is its ability to assist users in differentiating between valid and invalid data by processing the information before permitting its transfer to the database.

8.4.2 RESOURCE MODELING

8.4.2.1 Data Acquisition

Resource modeling starts with the gathering of information, that is, drillers' data, core-hole information, geophysical logs, topographic maps, cross sections, and survey data. In some cases, detailed drillhole information is not available, and only previously generated structures or isopach maps are provided.

The initial task is to organize the data in appropriate categories, because each set of data is handled differently. Drillhole information is either transferred to preprinted forms or it is directly entered into the computer using formatted screens.

Even though it is possible for the geologist to input data directly into the computer, if more than 50 to 100 drillholes need to be processed, it is advantageous to have data entry clerks enter the data. After all the information is entered, the geologist can check the results for inconsistencies or possible data-entry errors.

As mentioned earlier, a good system provides automatic checking of the data as much as possible. In the following pages, for the sake of demonstration, examples from EAGLES, a comprehensive geologic and mine planning system developed at Morrison-Knudsen Co., are presented. There are many other systems available on the market that are suitable for modeling and mine planning of various types of deposits. These models vary in their scope and the range of mining options they cover. Their functions mainly reflect the area of expertise of the primary developers and the type of mining the model was intended for. One should examine the functionality of a system carefully to ensure that it can adequately perform the functions specified. A detailed discussion of the hardware and software selection is provided at the end of this chapter to assist readers in the selection process.

8.4.2.2 Minimum Data Requirements and Database Concepts

To analyze and model a property, one has to have a minimum set of data. The basic data consist of a set of information that defines drillhole location and geologic as well as assay or analytic data. As a minimum, the database should contain:

1. Drillhole name, collar elevation, X-coordinate, Y-coordinate, drillhole deviation, seam/unit ID, seam/unit top intercept, seam/unit bottom intercept, grade/assay, analytic/quality, lithology/rock type.

2. Additional information, such as property name, driller's logs, geophysical logs, special codes for geologic conditions, water table, and many other such items of information may be also added to the database.

A database may be set up with two different concepts in mind. The first is to set up a database as a repository for all the information. In this type of database, in addition to items specified previously, information such as lithologic characteristics, mineralogical, petrographic, and alteration descriptions are also stored. To accomplish this, a number of text fields are assigned to the descriptions and narrative information provided by the drillers or geologists.

The second type of database is mainly set up for mapping and modeling. Such a database does not contain much text or narrative information, and only data pertinent to modeling will be coded and entered into the database. The majority of available commercial modeling systems are of this type, where they provide for storage of data pertinent to modeling. The advantage of this type of database is that it minimizes the disk space required, and by reducing the size of the database it will expedite updating and retrieval of the data.

A separate database contains property boundaries, physical structures, pipelines, power lines, and so on. This type of data is normally available in the form of maps or survey information. Map data needs to be digitized.

8.4.2.3 Digital Map Data

To transfer plotted information to the computer, a map is positioned on the digitizer tablet, and after initialization the data are digitized. If the X and Y (easting and northing) scales are the same, two separate points digitized on the map can establish the scale. However, if the X and Y scales are different, three points are required to establish the scale. The points digitized for the initialization process should be at the extremities of the map to assure correct scale and to compensate automatically for the stretch or shrinkage in the map. Paper maps stretch and shrink depending on the moisture content in the air; therefore, providing a map scale directly to the system may result in error.

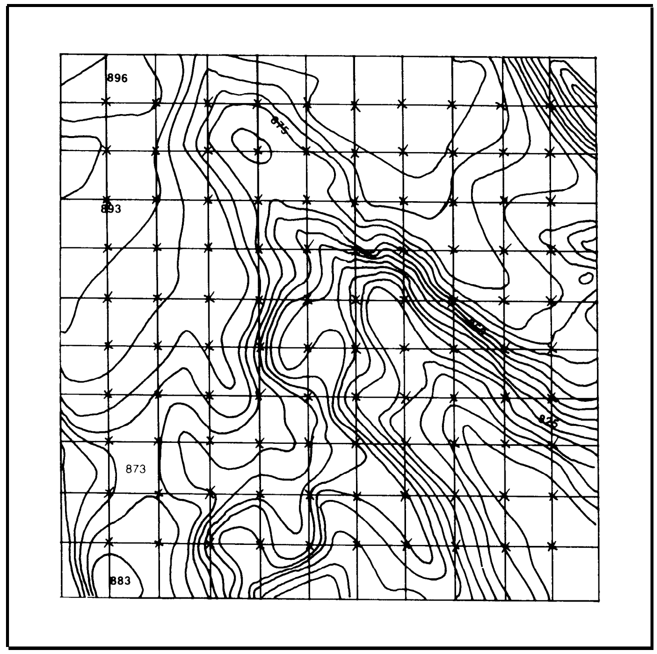
Following establishment of the scale, each individual line is traced either in a continuous or in a point mode. Most systems allow editing capability during digitization or immediately after. Data can be stored as individual files or they can be directly input into a database and tagged with identifiers.

Survey data is regularly provided in a digital form by photogrammetric companies or by survey crews through a total station or survey notes. These data can be added to the system by automatic reading of the digital tapes or by direct input of the data via formatted screens.

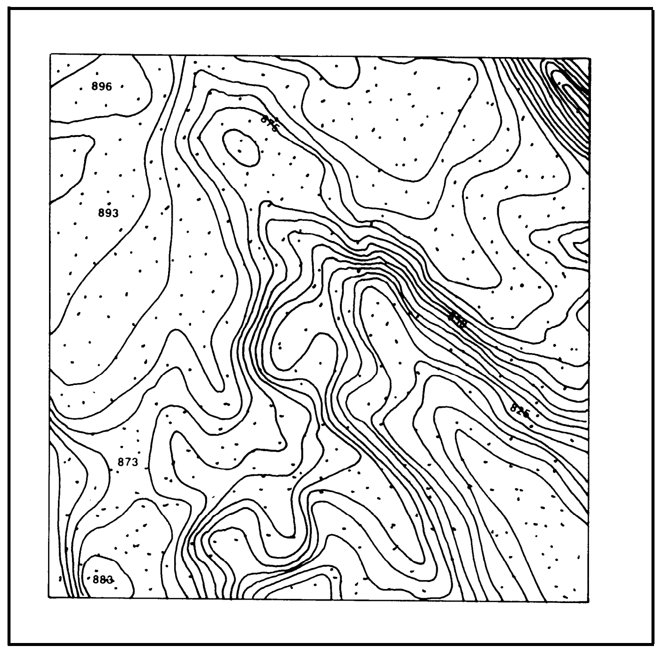
Photogrammetric companies can provide topographic data in digital or map form. Digital data are provided in one of two forms: (1) in rows and columns at a given interval, that is, on a grid pattern; or (2) as a set of irregularly spaced data depicting breaks in elevation taken throughout the region of interest (Fig. 8.4.2). Irregularly spaced data may be provided by way of cross sections parallel to Y (northing) or X (easting) and points taken at the break points greater than specified elevation difference (Fig. 8.4.3).

Each set of data provided by any of the foregoing methods has its advantages and disadvantages. The gridded data is the simplest to handle, because all that needs to be done to it is to read it and store it in the desired format. As long as the same grid increment or multiples of it are used throughout the operation, there is no problem. As soon as a smaller grid interval becomes necessary, however, the data has to be regridded. As a general rule, the accuracy of the regridded data is less than original data, and the grids may be off by a substantial amount from the actual. The difference between the estimated and actual surface elevation is a function of changes in topography, that is, the more rugged the terrain the more deviation from the actual.

The irregularly spaced data option eliminates this problem and allows generation of grids of any size. Grids generated by



Gridded Pattern

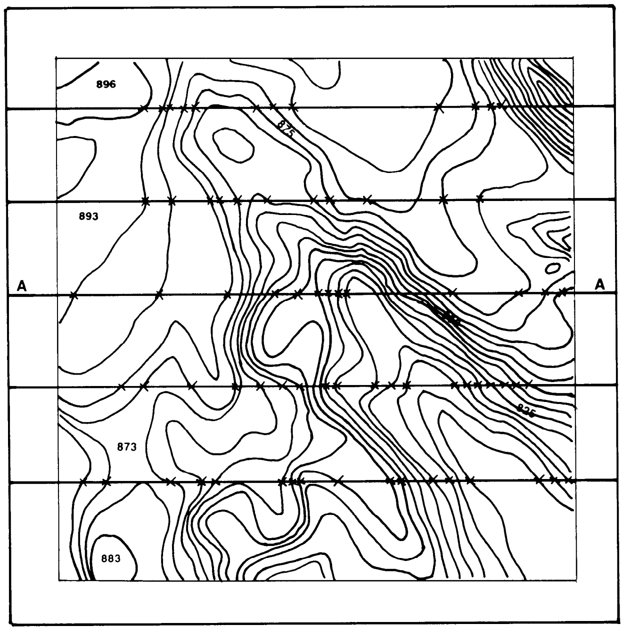


Irregular Pattern

Fig. 8.4.2. Digital sampling of topographic data.

this method represent the actual elevations as closely as possible, the main reason being that during sampling, all major breaks in the topography are picked. The gridding process uses this data to arrive at a value for each node. The main problem with the irregularly spaced data option is the enormous number of data points sampled and the tremendous amount of processing required to generate a gridded surface from this data.

The cross-sectional method is a cross between the two other options (Fig. 8.4.3). It provides the number of data at the break



CROSS-SECTIONAL PATTERN

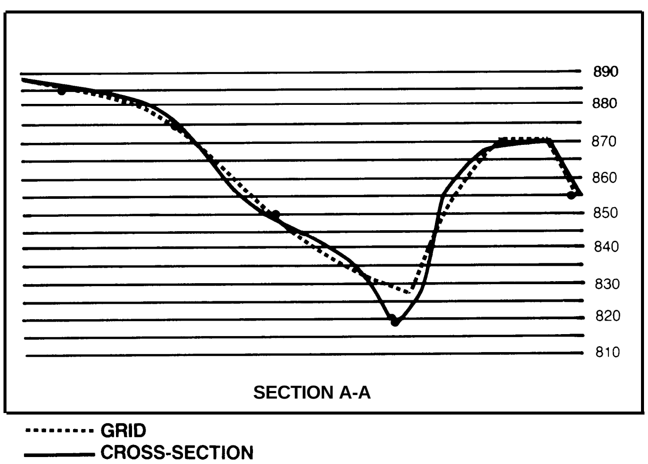


Fig. 8.4.3. Cross-sectional sampling of topographic data.

points in one direction, but it is set at regular intervals in the other direction. The processing requirement is not as intense as the irregularly spaced option, but it is more than gridded data. The accuracy factor, however, is not as good as the irregularly spaced option, because it lacks the necessary detail in one direction. For this reason, the cross-sectional data are not as useful as the other two. If the possibility of regridding is minimal, then the gridded data provide the best and fastest result. In particular, if the initial grid is generated at small enough increments, then larger grid increments can be generated without losing accuracy. If gridding increments are to be changed on a regular basis, or a suitable grid interval cannot be decided upon, then the irregularly spaced option is a better choice. Experience shows that a 100-ft (30-m) grid increment is a satisfactory increment for most operations in the United States, Canada, and most other parts of the world.

8.4.2.4 Geologic Modeling

The most important task in any modeling process is collection and editing of the data. The reason for emphasis on the need for thorough data checking and validation is to minimize wasted time and effort during the modeling process. Discovery of an erroneous data point during the modeling stage is a very costly and time-consuming endeavor because the entire process of data checking, retrieval, and modeling has to be repeated again. If the problem is not discovered on time, it may result in inaccurate interpretation and improper decision, with grave consequences. Following the completion of data entry, checking, and editing, the modeling task begins.

Geologic modeling, in essence, is reconstruction of the three-dimensional geologic picture of a deposit from a handful of sample data. The process combines the power of imagination with mathematical formulations to arrive at a satisfactorily reconstructed model. There are numerous approaches to the problem. In the following pages, the major approaches to the process are described.

To model a property, three pieces of information—the topography, the geologic and assay data, and the properties or other physical information—are required. The data may be obtained through any of the previously mentioned processes.

8.4.3 DATA INPUT

8.4.3.1 Topography

Topographic data are obtained either through photogrammetric means or digitization of existing maps. If data are available from photogrammetric services, the tape provided is loaded into the system, and through usage of a translation program, it is translated into a specific required format.

If the provided information is in the form of irregularly spaced data, it is passed through a gridding program with desired grid increments specified. The resultant grid then is used in the subsequent operations, and the raw data are stored on tape for possible later use. Cross-sectional data may or may not need any processing, depending on the capability of the system. EAGLES provides cross-sectional, gridded, and variable block modeling techniques, and their usage is dependent on the type of deposit and the degree of pitch of the strata. For steeply dipping deposits or for very complex geology, a cross-sectional modeling approach is more appropriate. The cross sections can be used at any time to generate a gridded surface. This data then can be used in the subsequent operations. At the end of this process, a model of the topography is developed for the area of interest.

8.4.3.2 Geologic Input

Geologic data-input requirements vary depending on the commodity of interest, the complexity of the geologic conditions, and the size of the property. As a general rule, the more complex the geology, the more information is required. Also a coal deposit normally requires more data than, say, a gold deposit. In addition to basic information such as the ones specified, quality data for proximate and ultimate analysis, washability, ash-fusion temperatures, and trace-element analyses are regularly provided for coal. Other commodities such as phosphate may require information for the processing plant as well. Regardless of the type of commodity, the data input deserves a great deal of attention. The input determines the usefulness of the model. Companies usually assign their most junior geologists or engineers to this important task and as a result spend a great amount of time rectifying the resulting problems during or after modeling.

Data Verification: After loading the data into the database, various reports should be generated and reviewed for accuracy of input and for elimination of invalid data. Statistical tables showing the range of data and the mean and standard deviations for each field can assist in data verification. It is useful to display the number and the X,Y locations of the drillholes that contain the maximum and minimum values for each field. These reports and statistical tables should be reviewed carefully to ensure that the data extremes are valid.

Another verification method is to generate a contour map of the data and look for "bull's eyes" on the map. They invariably point to the error in the data, that is, either input error or assignment of the wrong X and Y location to the hole. All errors should be corrected in the database, and new reports generated. The process should be repeated as long as erroneous data are found.

Use of Database: The database should contain data in their raw form and without any regard to mining criteria or minability of any unit. Many companies allow their interpretation of the minability of the units to determine which ones will be entered into the database. This practice is unwise, and as mining and economic conditions change, the database becomes useless, resulting in another round of data input. The preferred method is to keep data in the database in their rawest form possible but allow the retrieval programs to combine and sort the data, based on the economic criteria specified at the time of retrieval. This simple concept will ensure the longevity of the database while providing the possibility of unlimited mining and economic combination alternatives throughout the life of the project.

Correlation: Bedded deposits require extensive amounts of time to correlate individual units across the property. This process is not only very time-consuming but repetitive. The geologist has to analyze each drillhole in conjunction with all other holes and for all units down-hole. An on-line correlation program with direct access to the database can save many man-months of efforts. The concept is simple but somewhat hard to implement. EAGLES allows the user to select a set of cross sections from a plan-view location map of the area. Sections can be defined either by a traverse line and an area of influence to either side of the line or by selecting individual drillholes. Following the selection process, all drillholes will be retrieved from the database, and the geologist can examine any of the sections by recalling that section. In addition, geophysical logs for each hole can be displayed, along with the stratigraphic logs, to assist in the correlation process. A comprehensive set of commands provide full correlation capability right on the graphic terminal. Key holes can be set up as the guiding holes for the rest of the data. Modified drillholes will be updated directly into the database, therefore providing an up-to-the minute set of data to all users.

8.4.4 DATA RETRIEVAL

8.4.4.1 Retrieval Process: How and What to Include

A good retrieval system is essential to a good geologic system. Not so obvious is the fact that a strong retrieval system can simplify, and at the same time, enhance the geologist's task. It can simplify a task because the geologist does not have to decide on the details of what should be included in the database; instead, concentration will be on actual data input and correlation. It will enhance the geologist's capability because it will allow retrieval of data in any set of combinations desired. Many factors such as economic conditions, minability of the thin units, and geologic factors such as erosion or pinching seams can be specified at the retrieval time for alternative evaluation. The retrieval process normally is customized for the type of commodity being modeled.

8.4.4.2 Coal

For coal deposits, structural and thickness data are retrieved and modeled first, to delineate the extent of the deposit and its spatial distribution. If borehole deviation or pitching seam data are available, the necessary correction is made to provide for a true-thickness calculation. However, a true-thickness calculation is not necessary if the model is going to be used for volumetric calculation. A majority of the volumetric programs calculate the in-place or as-mined volumes directly from the model.

Following or concurrent with retrieval and modeling of physical data, quality data are also retrieved and modeled. Samples usually need to be combined for a given seam or range of seams. If washability data are available, washability tables are generated to determine the appropriate specific gravity of wash. Quality data for desired specific gravities are retrieved and modeled. Reduction and dilution factors may be incorporated in the data at this time, or they may be postponed and applied to the gridded model at the time of mining. Reduction and dilution can be on a percentage basis or based on a specified thickness from top and/or bottom of each seam. Cumulative information may be necessary if seams are too thin or the parting is less than a given thickness that can be mined separately.

The retrieval system provides for selection of raw or washed data and for conversion of as-received data to dry basis and back. It should be able to use any of the three moisture data in the database, that is, sample moisture, equilibrium moisture, or air-dry moisture for these calculations.

8.4.4.3 Noncoal

For nonstratabound deposits and for disseminated ore bodies, physical data retrieval is combined with retrieval and compositing of the assay data for each mineral and for a given bench height. Correction for borehole deviation, if applicable for dipping seams, is made at this time. Compositing of samples is necessary for various reasons. The sample size, length, and weight of sample vary among drillholes, and to arrive at a common size for a given interval, samples are composited. Depending on the deposit, it is necessary to combine samples into common weight segments representing a segment of ore body for volume or grade calculation. The compositing process is the same for various commodities; however, depending on the type of deposit, care should be taken not to composite among dissimilar geologic zones.

Provided that the sample length and associated assay values are available, compositing is accomplished by summing the products of sample lengths and corresponding assay values, then dividing this sum by the total of sample lengths, to arrive at a weighted average (Barnes, 1980a). The composite interval is usually determined by the bench height at the time of mining. For example, if samples are taken at 10-ft (3-m) intervals, and a 35-ft (11-m) bench is required, then the first three samples and half of the fourth are used to arrive at the composite value for that bench (Fig. 8.4.4). This assumes that the entire bench is part of the same geologic unit. Extreme care should be taken to avoid compositing of dissimilar data. Barnes (1980) suggests the term *draped composite* for a compositing process that uses a surface or multiple surfaces to initiate the compositing. This process allows the use of the surface of the mineralized zone as the starting point for compositing rather than an arbitrary

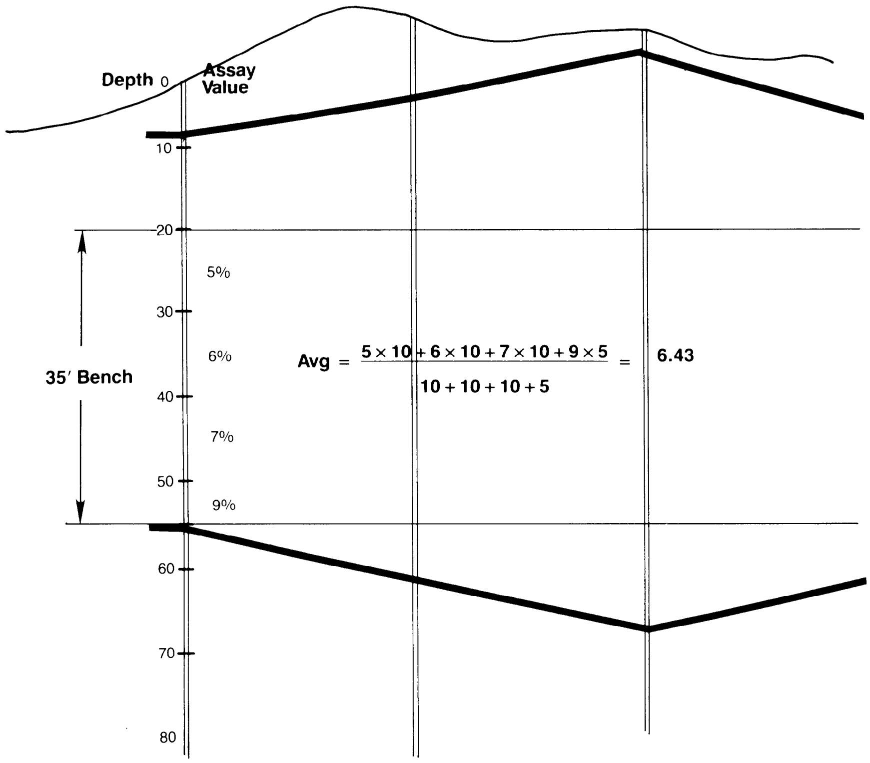


Fig. 8.4.4. Bench compositing.

elevation, resulting in a more accurate representation of the mineralized zone.

Badiozamani et al., (1988a, 1988b) have developed a new compositing and variable-zone modeling process to deal specifically with the problem of inclusion of geologic and depositional environment in the compositing and modeling process. The model was developed to handle complex geology and the variation in ore content of deltaic tar sand deposits. However, the same modeling process can be applied to other mineral deposits. with only minor modification to the compositing routine.

Variable Zone Compositing: This compositing process starts with first definition of depositional environment and correlation of geologic units across the property. The groups and formations are identified, and each group or formation is further divided into their depositional subunits or so-called zones. For example, a group may be divided into marine, transitional, and continental zones. If additional information is available, each of these zones may be further divided into subzones. For instance, a transitional zone may be divided into delta plain, delta front, and prodelta subzones. By using the depositional environment to arrive at the subzones, advantage is taken of the available geologic knowledge to limit the grade estimation across the property to only comparable units. In essence, we take advantage of other parameters contributing to grade variation such as porosity and permeability, especially when secondary mineralization is involved.

Following identification of the zones, the combination of each unique lithology and depositional facies is identified. We refer to this unique combination as *material type* (MT), which is used as the basic unit for interpolation and compositing between the drillholes. For example, a channel sand and a point bar sand constitute two different material types. During modeling, regardless of the algorithm used, only similar material types are used for interpolation within each subzone.

In addition to material type, a zone may be divided into smaller subzones by using sudden grade variation down-hole as another criterion for control of lateral interpolation. Figs. 8.4.5 and 8.4.6 depict the effect of zone, subzone, material type, and grade variations on defining each interpolation region. Fig. 8.4.5 shows the entire region being divided into zones one, two, and three. Then each one of these zones is further divided into four material types. Fig. 8.4.6 shows how each of these material types is subsequently divided into a number of subzones as a result of grade variations of more than 3%.

As indicated previously, this process allows inclusion of maximum geologic and depositional control into interpolation process. After zone and subzone designation, and prior to interpolation, grade and other values need to be assigned to each subzone. Generally, the sample intervals do not coincide with the top and bottom of each subzone, or multiple samples may be available for each subzone. If the subzone and the sample intervals coincide, the value of the sample interval is assigned to the subzone. However, if there is more than one sample per subzone, the samples are composited and the composite value is assigned to the subzone. Composite value may be weight averaged by thickness of the samples or, where there is a noticeable difference in the density of the samples, by the product of the thickness and density. The weight averaging is performed only for those values that are reported as weight percentage such as grade, and not for the values that are reported as volume percent such as porosity.

8.4.5 MODELING AND RESERVE ESTIMATION

8.4.5.1 Modeling Selection

The modeling process starts with results of data retrieval or compositing. Modeling choices are extensive and depend on the type of deposit, the commodity being modeled, the value of interest, the complexity of the geologic deposit, and the user's

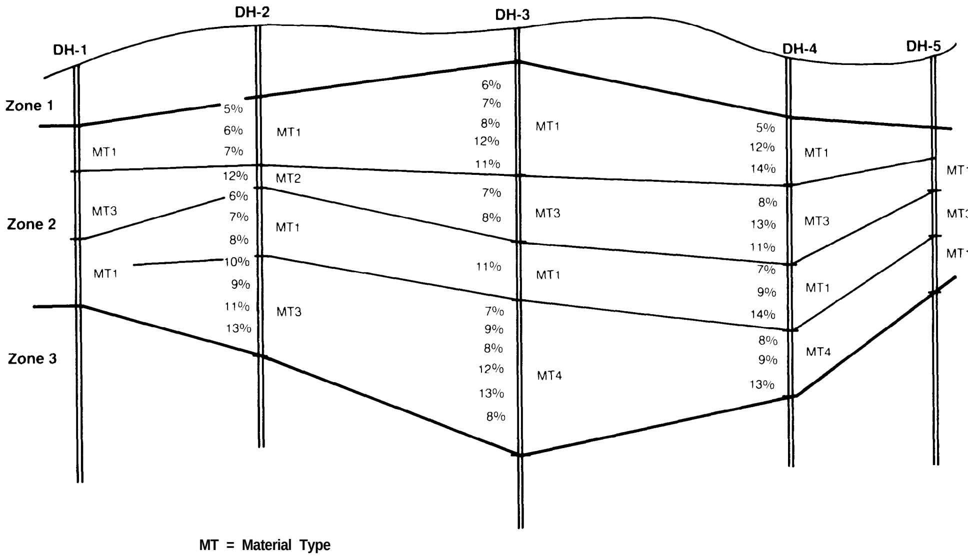


Fig. 8.4.5. Zone and subzone designation based on material type.

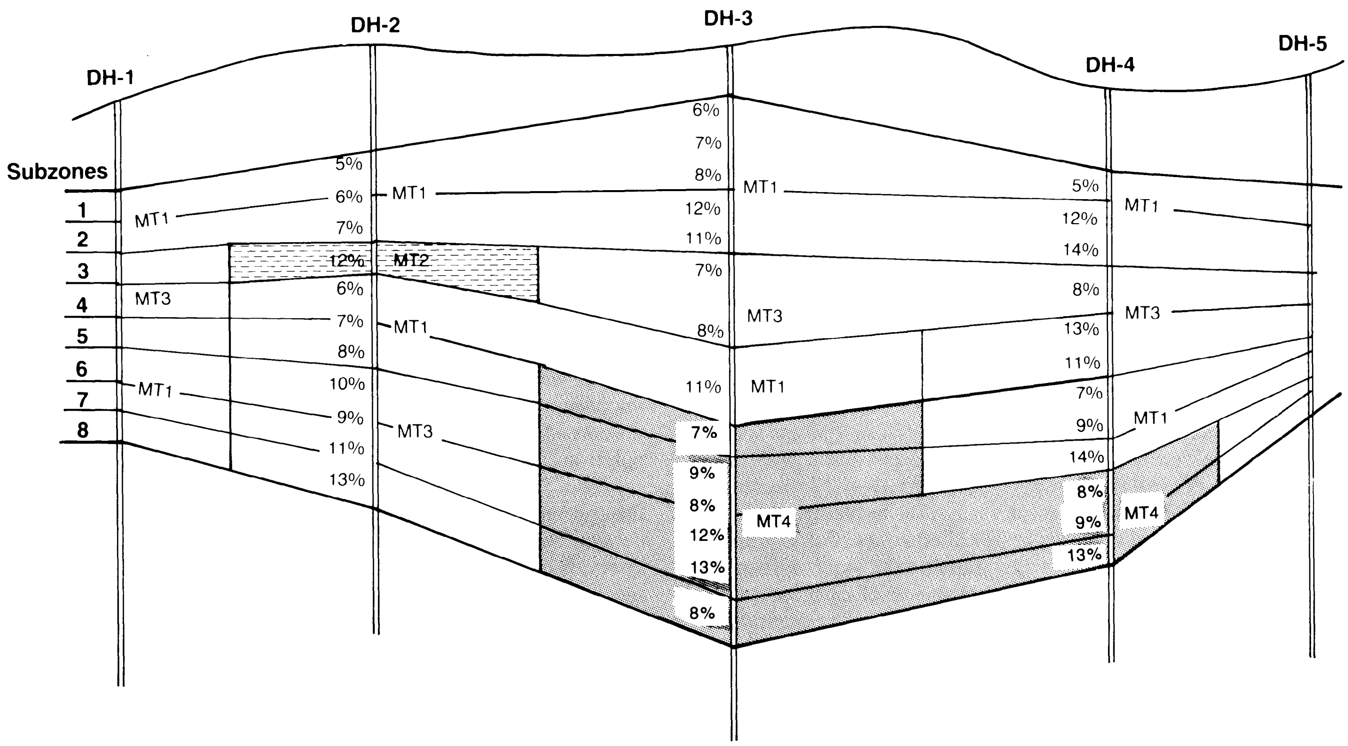


Fig. 8.4.6. Subzone designation based on material type and grade variation.

acceptance and familiarity with various modeling techniques. There are essentially three major modeling concepts and a few other approaches that do not meet the strict definition of modeling. For example, polygonal estimation should not be regarded as a modeling process. The three major modeling processes are the gridded model, block model, and cross-sectional model. Each of these models is used for specific conditions and specific mining operations. The gridded model is normally used for bedded deposits such coal, phosphate, sulfur, limestone, oil shale, and tar sand. The block model is usually used for disseminated deposits such as porphyry copper, uranium, gold, and other nonstratabound deposits. The cross-sectional model is commonly used for complex folded and faulted or steeply dipping deposits. Block modeling techniques have been applied to steeply dipping deposits; however, by using the block model, one loses control over the structural position of the units in a block. If it is important to maintain the structural integrity of the unit, then the cross-sectional modeling technique should be used.

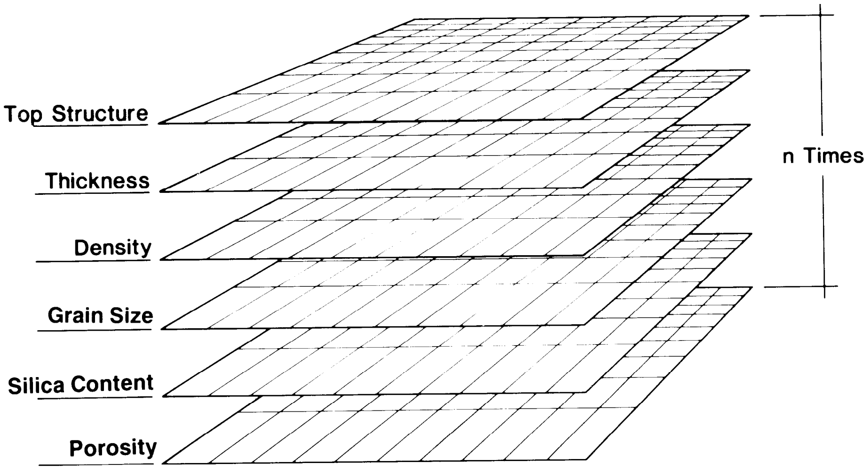


Fig. 8.4.7. Gridded model.

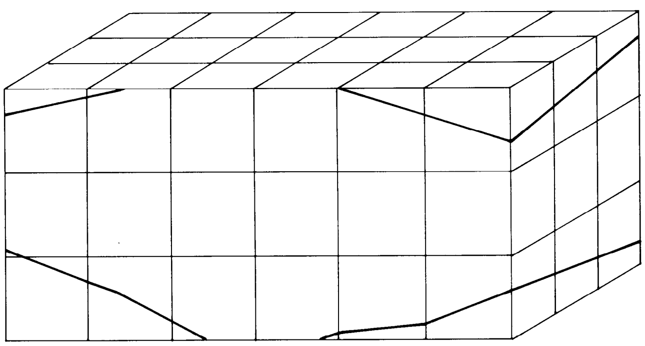


Fig. 8.4.8. Conventional block model with fixed blocks.

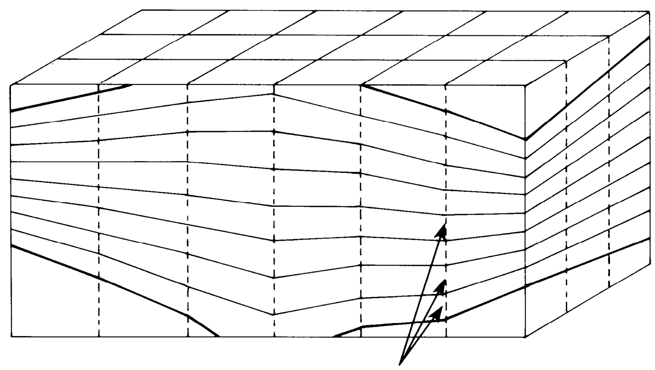


Fig. 8.4.9. Modified block model with variable thickness blocks.

8.4.5.2 Gridded Model

The *gridded model* is a set of two-dimensional matrices, each representing a surface or a value. These surfaces or values are each the result of interpolation from a set of irregularly spaced data to a fixed grid or matrix (Fig. 8.4.7). The major advantage of using a matrix or a grid is ease of use and manipulation of the data. Structures, thicknesses, and other values stacked on top of each other can be easily added, subtracted, multiplied, divided, or logically compared to arrive at other sets of data. For example, the bottom structure of a seam can be subtracted from its top structure to obtain the seam thickness, or the top structure of the first seam can be subtracted from the topography grid to obtain an overburden thickness. Another advantage of the gridded model is a reduction of required space (by two-thirds) by eliminating the need for X and Y coordinates. Given a matrix and the coordinates of its starting position along with the X and Y increments, the location of any other point on the matrix can be easily calculated. This is a very significant reduction of disk space, especially when numerous seams are involved, and each seam may have as many as 20 or more other attributes associated with it.

In a gridded model, only the units of interest are modeled, which means that the model need not necessarily be contiguous. This is the major difference between the gridded model and the block model.

8.4.5.3 Block Model

A *block model* is the collection of a series of blocks of given X, Y, and Z dimensions stacked on top of one another, which represent a model of a property to a given depth (Fig. 8.4.8). Each block is identified by the X, Y, and Z coordinate at the center of the block and contains the percentage values for each item of interest. For example, in a porphyry copper deposit, a block may contain, in addition to the percentage copper, the percentage of zinc, lead, waste, and air, totaling 100%.

There are two other variations to the conventional block-modeling technique called variable block model (VBM) and variable zone model (VZM). In a variable block-model, the X and Y dimensions are fixed and the Z dimension is variable to better approximate the geologic variability of the deposit (Banfield and Wolff, 1979). However, even though the Z dimension may change from block to block, they form rectangular blocks of different size (Fig. 8.4.9). In the variable zone model the Z dimension may vary for each side of the block, resulting in blocks of trapezohedron shapes (Fig. 8.4.10).

8.4.5.4 Cross-sectional Model

A *cross-sectional* model is a collection of parallel or nonparallel sections, dissecting the property of interest. These sections contain the geologic structures of the deposit along the section lines. A model is built basically by connecting sections to one another, by linear interpolation and assuming a gradual change from one section to the next; or by extending each section halfway to the next, reflecting an abrupt change in the deposit. The grade or quality data may be assigned to polygons formed by the top and bottom structures of each seam and the lines halfway between the two adjacent drillholes on the section (Fig. 8.4.11). Another way of assigning grades to these sections is to generate

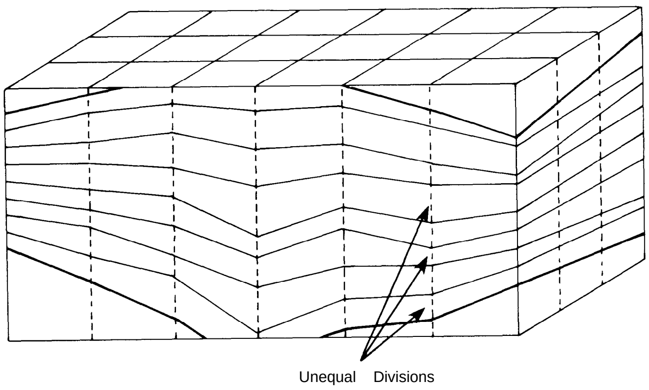


Fig. 8.4.10. Variable zone model with variable thickness subzone blocks.

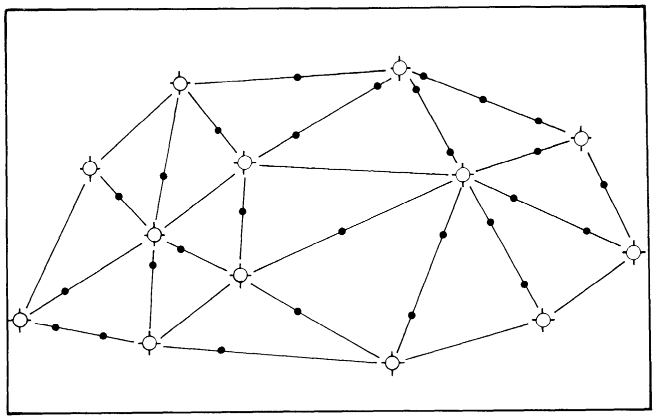


Fig. 8.4.13. Division of triangle sides into contour intervals.

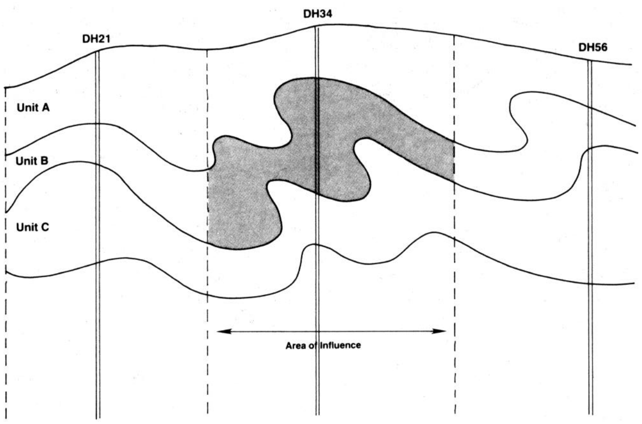


Fig. 8.4.11. Grade assignment using structure and drillhole area of influence.

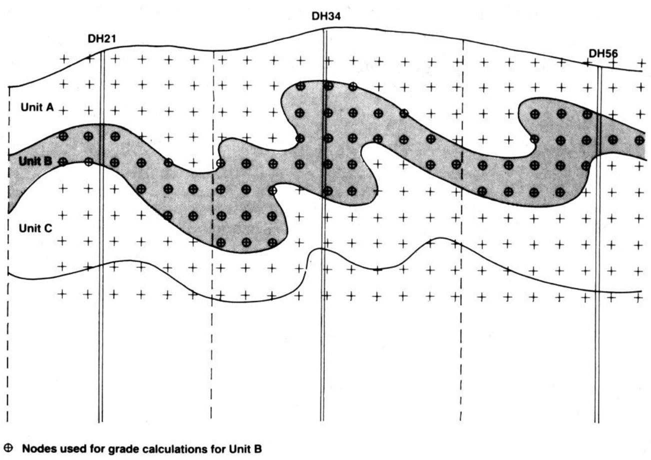


Fig. 8.4.12. Grade assignment using gridded assays.

a gridded model of the grade or other quality data, and then to use the values that fall within the area of influence of each drillhole to calculate the desired grade or quality (Fig. 8.4.12).

8.4.5.5 Estimation Process

The estimation process is common to all models discussed previously. It involves the application of a series of mathematical algorithms to interpolate and extrapolate from a set of irregularly spaced data to a grid node or the center of a block. Estimation techniques vary from a simple triangulation, to polygonal approximation—where an area of influence is assigned to each drillhole—to a much more complex algorithms such as trend-surface analysis, Fourier series, and geostatistics.

The estimation process is nothing new to geologists and engineers. They have been using manual methods, starting at a corner of the property and connecting the three closest holes with straight lines to form a triangle. Then they linearly divide the length of each side into equal units between the values of the two vertices to arrive at desired increments. This process is then repeated over and over again for the next closest hole that forms a triangle with one of the sides of the existing triangle, until all the holes are used and a triangulation network is established. The computer industry started initially by mimicking the manual methods and has built on the same concepts by adding an array of mathematical techniques, which more closely approximate the actual reserve.

Given a set of irregularly spaced data, the user has to develop the model of the deposit in order to be able to calculate the reserve. As mentioned earlier, when performed manually, one uses only three holes at a time. However, using computers, more than three holes at a time can be used to arrive at an estimate for a node. The method used routinely by many companies is to use a circular search radius and as many as 24 holes for estimation. Let us look at few estimation techniques and see the difference in approach by evaluating the pros and cons of each method.

8.4.5.6 Triangulation

One of the earliest methods of estimation was *triangulation.* It is based on generation of a network of triangles connecting all the control points together. The assumption in this type of estimation is that every three points form a plane from which given any X, Y point falling on that plane, a Z value can be estimated, by using the equation of the plane. Triangulation has been used for many applications, but mainly it has been used in finite element calculations and in geodetic surveys. More recently they are being popularized by some companies for quick modeling and contouring. As shown in Fig. 8.4.13, following completion of the network, each side of the triangle is divided into a fixed

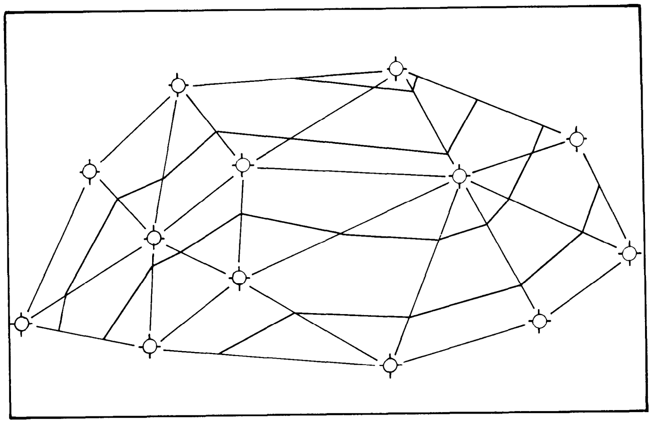


Fig. 8.4.14. Contour generation by triangulation.

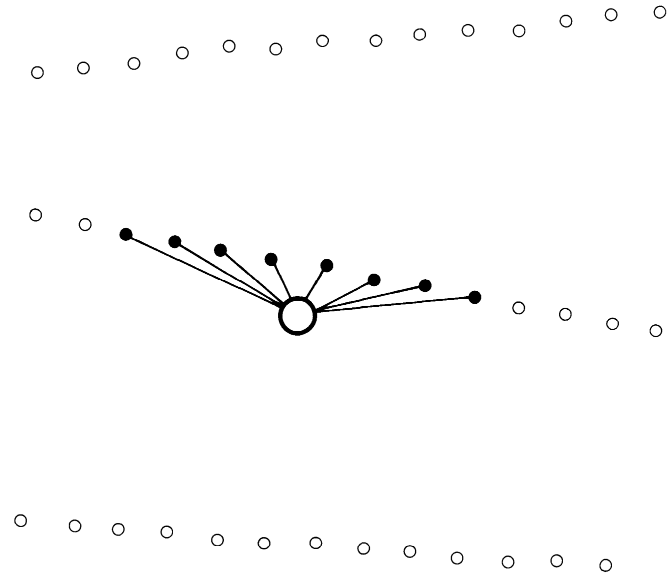


Fig. 8.4.15. Improper data selection (modified after Jones, 1986).

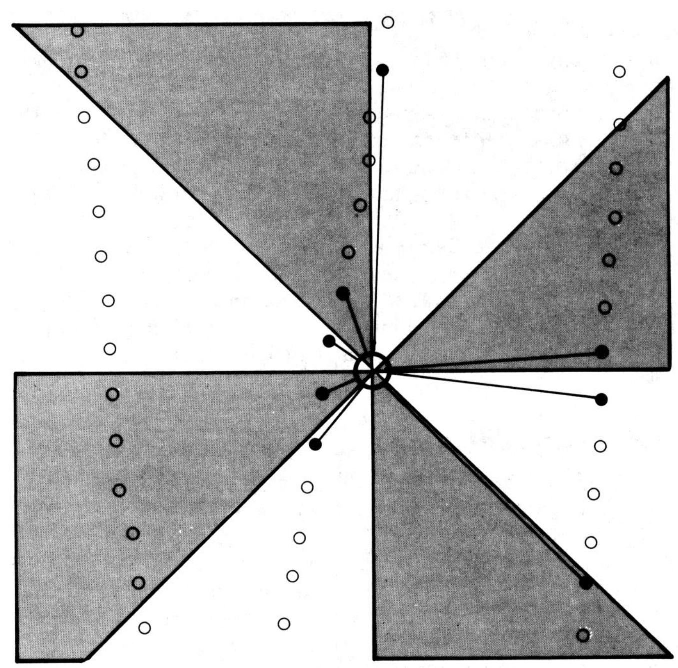


Fig. 8.4.16. Proper data selection using octants (after Jones, 1986).

number of specified elevations, which then can be connected and elevation contours generated (Fig. 8.4.14). It is obvious that in the absence of a standard technique, various triangulation networks can be generated from the same set of control points, resulting in totally different-looking contour maps. Early attempts at solving this problem were not successful, and trial and error attempts were too costly, resulting in complete abandonment of the algorithm (Davis, 1986). Recent development of algorithms (Gold et al., 1977; McCullagh and Ross, 1980; Watson 1981; Bowyer 1981), that ensure almost optimum triangles (referred to as Delaunay triangles) with as equiangular triangles as possible, has revived interest in the triangulation method again.

Davis (1986) provides a good description of the methodology of generating Delaunay triangle networks. Advocates of triangulation techniques refer to the ease of generation of contour maps without gridding, the fact that it will fit a surface more tightly than a square grid, and other inherent mathematical properties of triangulation as the main advantages of this technique. However, along with any advantage, there is a disadvantage. Triangulation suffers from lack of surface manipulation capability, especially where deviated holes are present (Jones et al., 1986).

8.4.5.7 Grid Generation

Initial problems with triangulation and the lack of consistency in contour generation by this method resulted in development of gridding techniques. Using the gridding process, generation of contour maps became a much more consistent process, because instead of using the original values, one could use a set of regularly spaced estimated data for each node. To arrive at the estimated value for each node, two steps are necessary: first, selection of data surrounding a grid node and, second, the estimation process itself.

8.4.5.8 Selection of Control Points

To estimate the value of data for a given grid node, one may use the closest data points to the node. However, this approach may result in selection of data from only one set of drillholes or samples, especially if seismic line information is being used (Fig. 8.4.15) (Jones et al., 1986). To reduce the bias, many programs now require selection of data from within a specified search radius and limit the number of data to one to three points from each quadrant, sextant, or octant (Fig. 8.4.16). The search radius limits the number of data points to be selected for evaluation, therefore minimizing the computer search time. In addition, the reduced number of drillholes reduces the node calculation time. The more recent programs automatically adjust the search radius to further eliminate unnecessary processing time. The initial radius is set to a small window, and if the desired number of control points are found, the estimation process will take place. If in the first pass, the desired number of points are not found, then a pre-set increment is added to the original radius and search continues. This concept is specifically useful where there is an uneven distribution of data, for example, a cluster of data

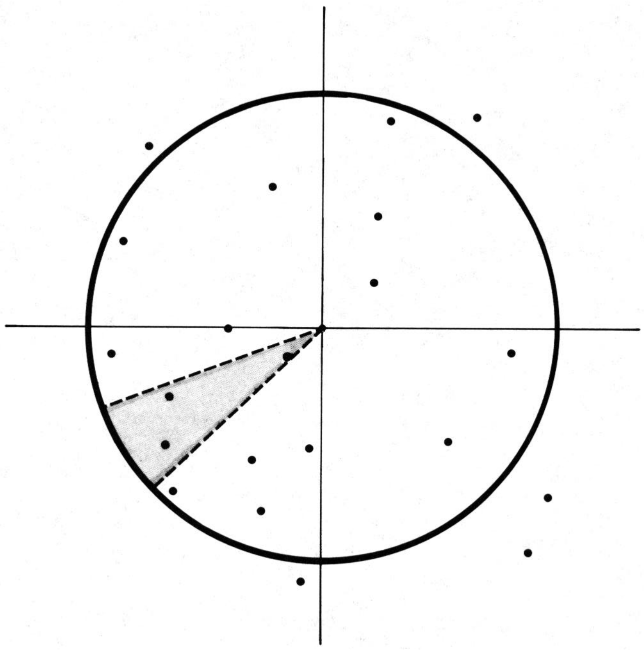


Fig. 8.4.17. Shadow effect concept.

in the active mining area and dispersed information in the rest of the area.

In selecting the points for estimation, it is important to ensure proper representation of data from all directions. By selecting data from all octants, the estimated value more appropriately represents the actual condition. Some programs use the concept of *shadow effect,* to further reduce biased contribution from one direction (Fig. 8.4.17). The shadow effect is specified as a cone around each point, so that any other point falling within the specified cone will not be used in estimation. This is based on the concept that any point falling in the shadow of the previous point should not contribute much to the estimated value.

8.4.5.9 Grid Estimation

Grid estimation is accomplished in one of two ways: (1) by individual node calculation or (2) by fitting a function to the data and then estimating the grid node from the function.

The individual node estimation consists of calculating the average contribution from all the samples to the node by some weight function. The concept is that the data points farther away from the node have lesser effect on the node than the closer ones, resulting in inverse distance functions. The only question, then, is how should the distance be weighted. The linear inverse distance assumes a one to one relationship, in other words, a point 50 units away has 10 times less effect than the one 5 units away. In many situations, it is more appropriate to diminish the effect of the control points at a faster rate. As a result, weight functions of higher degrees are used. For example, an inverse squared distance will reduce the contribution of the control points by the square of the distance. Some software companies provide deterministic, as well as statistical, weight functions. Statistical weight functions are more applicable in situations where an individual sample's contribution to the estimation is not significant by itself. An example is the sulfur distribution in a coal seam, where a high sample value should not be weighted heavily because it may be the result of a nugget in the sample. Under such circumstances, the general trend of the data is more meaningful than an individual sample value.

The functional estimation process fits planes, polynomials of various degrees, or other functions to the data first, and then uses this calculated function to estimate the value of interest at the grid node. The main advantage of fitting a function to the data is generation of a grid that preserves the trend of the data. Grid nodes form continuous and smooth transition from node to node but may have values that are higher or lower than the actual data points. The same advantages may be looked upon as major disadvantages of surface fitting, that is, they normally obscure the sharp variations in the data, and oftentimes they project beyond the limits of the data, to much higher or lower values than the actual data permits, especially at the edges of the property.

Trend surface analysis has been used in *geologic modeling* for many years (Whitten, 1963; Krumbein and Graybill, 1965; Harbaugh and Merriam, 1968; Whitten, 1970; Agterberg, 1974; Davis, 1973, 1986). As Davis points out, the trend analysis is a mathematical method of separating map data into two components, (1) that of a regional nature and (2) local fluctuations. The regional and local trend separation concept has been used by geologists, geophysicists, and petrologists for many years on a manual basis. They have, for example, considered the regional dip of a formation vs. the local structural variations. For a detailed review of usage and application of trend surface analysis, refer to Krumbein and Graybill (1965) or a more recent explanation by Davis (1986).

In trend surface analysis, linear, quadratic, or higher-order polynomials are used with the intent to minimize the squared deviation from the trend. The lower the degree of the polynomial, the lesser the degree of variations from the straight line. For example, the linear model only will display straight-line fitting through the data, whereas the quadratic fit will display the harmonics (Fig. 8.4.18), and cubic fit will even display smaller variations in the trend. The residual data, if gridded, represent the local trends in the data.

The main disadvantage of polynomial fitting is the so-called edge effect at the property boundary, where data are scarce and limited to one side only. Under these conditions the estimated grids display a high degree of deviation from actual.

8.4.5.10 Kriging

Under previous topics, we reviewed the application of trending or deterministic functions (polynomials) to the estimation process. In this segment, we evaluate the concept of Matheronian geostatistics, which is the study of regionalized variables. Regionalized variables are variables which their magnitude is a function of their neighboring values and their position in the three-dimensional space. The previous methods use classical statistics, which are based on random variables and the probability of independent events (Barnes, 1980b). To have an unbiased evaluation means that samples should have been collected in a totally random fashion. Unfortunately, many drilling activities are controlled by physical and practical considerations, which violates the random sample assumption.

Matheronian geostatistics, or simply geostatistics, is the science of regionalized variables as defined by Matheron (1963). The key assumption in geostatistics is that the data display a stationary behavior. Stationarity means that the distribution of the difference for a given parameter, between two sample points, is the same over the entire deposit, and it depends only on the distance and direction of the sample points. In other words, there is no trend in the data.

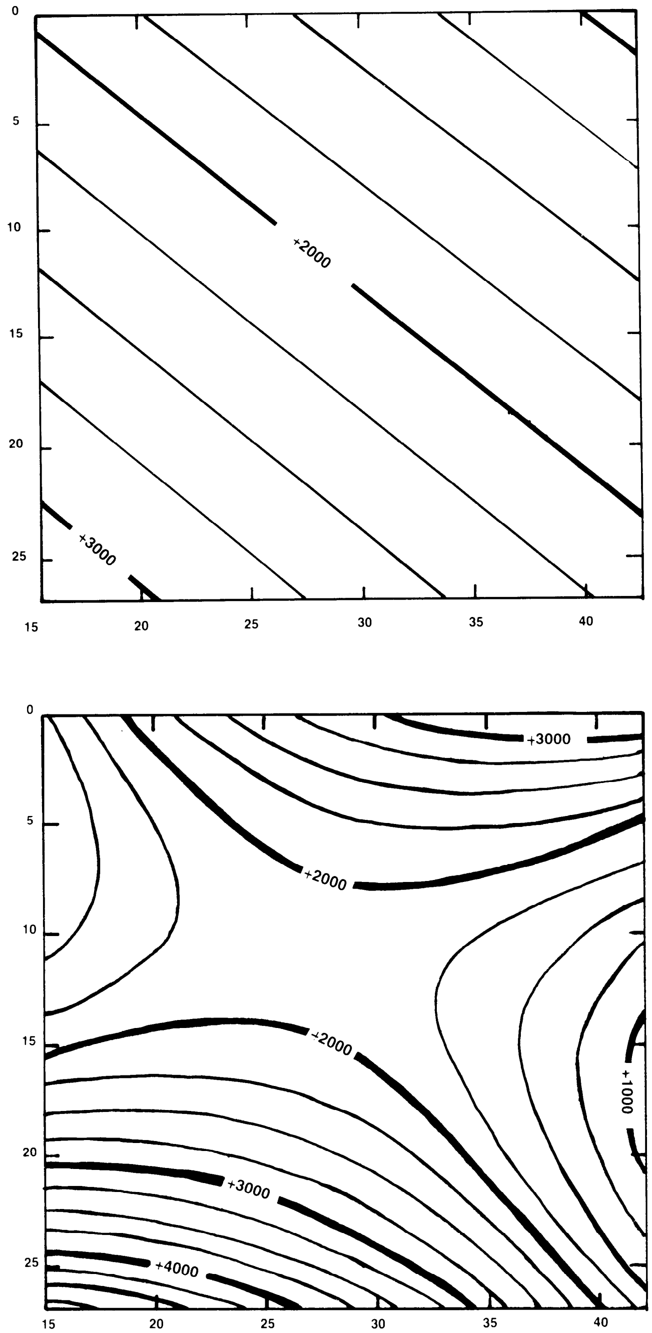


Fig. 8.4.18. First and second degree surface trends (modified after Davis, Wiley, (c) 1986).

It is an intuitive concept that the closer the samples, the less variation in the data, except for a totally randomly distributed deposit. If samples are fully random in their distribution, then we are better off using the standard statistical techniques. However, if there is a correlation between the samples and their spatial position, then geostatistics not only can provide the estimated value for the block, but it can provide us with an estimate of how good our block estimation is. For a discussion of elements of *kriging* and geostatistics, see David (1977), Royle (1980), Clark (1980), Journel (1978), Agterberg (1974), Barnes (1980a), and Davis (1986). Geostatistics provides a formalized and mathematical approach to geologists' long-used methodology for calculating the direction of rapid change. For a more detailed discussion of geostatistics, refer to Chapter 5.6 of the *Handbook.*

8.4.5.11 Grid Modification

To develop a complete model of a deposit, grid or block generation the process is repeated for all units and their associated grade or chemical analysis. Normally, for a single unit with thickness and structures, as many as 15 other kinds of chemical or grade information may be modeled. Depending on the complexity of the deposit, the generated structure grids may crossover each other. The crossover normally occurs because all drillholes do not contain information for all units. Lack of information may be caused by missing data, pinching units, nondepositional, and shallow drillholes where they do not penetrate all units. To eliminate the crossover problem or to incorporate some of the geologist's knowledge of the deposit into the model, a series of tools are made available to modify the generated surfaces and grids. These include arithmetic, logical, and comparative operations. For example, Fig. 8.4.19 shows drillhole 5 penetrating an erosional channel (paleo-erosion). To model such a unit, it is better to allow the straight interpolation across the channel and then to modify or blank the generated grid within the channel area. Grid modification programs are also used to obtain other grids from existing ones. For example, an isopach map can be generated by subtracting the bottom structure from the top structure, or a ratio map can be generated by summing up all the waste units and dividing it by the sum of all the ore units.

8.4.5.12 Block or Grid Size Selection

Selection of appropriate block size or grid spacing is crucial in generation of representative models. It is a balancing act between generating grids that are not too far apart so that they mask variations in the data, or too small a spacing that requires extensive computer time and resources, or too many blocks or grids that exceed the program limits. The grid-spacing selection is more an art than a science; however, a few comments may assist the novice user in a better selection process. As a rule of thumb, the best results are generated when each grid cell contains only one drillhole. This is more easily said than done, because the data are not usually distributed evenly. As a matter of fact, one usually faces the extreme variation of the cluster of closely drilled data in one end of the property and highly scattered data at the other.

The second rule of thumb is to adjust the grid spacing to the type of usage. For example, for a cursory evaluation, a large grid spacing will be acceptable, whereas a detailed mine plan requires a dense grid spacing. A dense grid spacing does not necessarily have to be generated from the original data. If there is not sufficient data to warrant generation of a tight grid from original data, a useful approach would be to generate the initial grid from control points at a larger grid spacing and then regrid it to a smaller spacing at a fraction of the central processing unit (CPU) resources necessary to do so otherwise.

8.4.6 MAPPING AND RESERVE CALCULATION

8.4.6.1 Mapping

Initial contouring programs followed the manual method by using triangulation and then dividing each side of these triangles into desired multiples of the contour interval based on the values

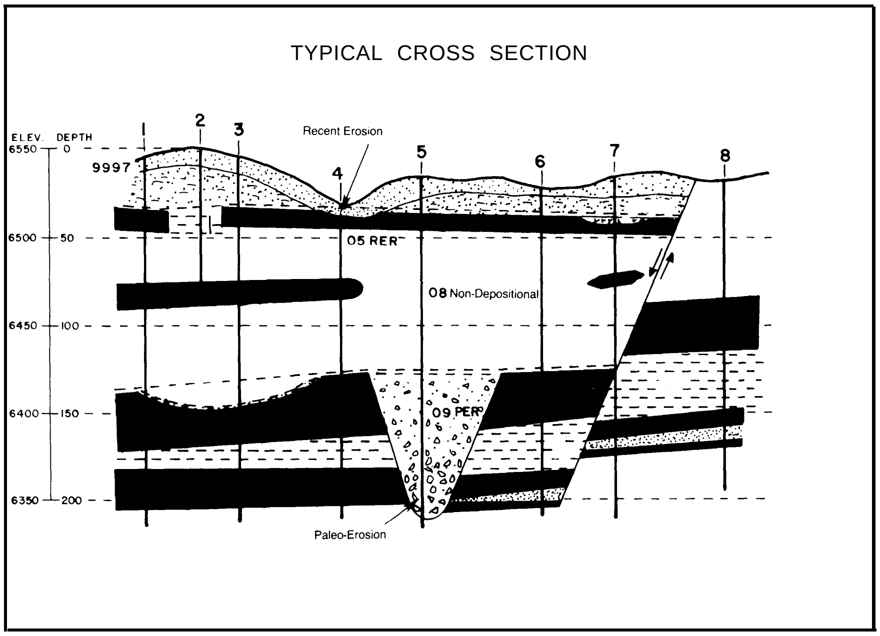


Fig. 8.4.19. Modeling erosional or nondepositional units.

at each vertex. Connection of equal values throughout the area would then generate a contour map. Problems with generation of appropriate triangles resulted in discontinuation of triangulation approach and precipitated gridding techniques. More than three adjacent drillholes are usually used for grid generation. Generated grids are then used for contouring. To generate smooth contours that are appealing to the eye, most contouring programs process the data a second time, where the initial grids and the original control points are used together to calculate newly weighted values at each node. One approach uses values from one or two rows and columns surrounding a node of interest and recalculates a new weighted average value for the node. The process is repeated for all subsequent nodes. Smoothed contours generated by these programs, while more pleasing, may in actuality be less accurate than the initially generated contours.

Once a map is generated and plotted, it must be checked thoroughly for possible problems resulting from modeling or mapping. The maps should be checked for accuracy, to ensure that the correct search radius and grid spacing was selected to generate contours that match the data. In addition, the extrapolated areas need to be evaluated for validity of extrapolation. The undesirable contours may have resulted from incorrect data input, inappropriate grid spacing, improper search radius, unsuitable algorithm, or a combination of the above.

Extrapolation may be contained by providing the cropline or other natural and ownership limits. The newer programs provide on-line editing and modifying capabilities. The geologist can correct or enhance the contour map based on his/her knowledge of the depositional environment, which is not quantifiable. For example, the geologist's knowledge of a sand bar channel can be used to elongate the contours parallel to a paleoshoreline. At the present time, unquantifiable data cannot be provided to the mapping program, but incorporation of expert systems in the future may make this possible.

Lack of data may result in blank regions; these areas need to be evaluated. If the blank areas are out of the region of interest, no action is necessary. However, if they are within the areas of interest, either the search radius should be increased or a new drilling program should be scheduled to fill in the gap in the data. If neither of these are possible or desirable, then the areas can be updated by one of the following actions:

1. Use of outcrop or other information that may not have been incorporated in the database.

2. Use of on-line interactive graphics update capabilities and geologist's interpretative ability.

3. Use of grid extension capabilities by slope or gradient modification.

The final map should include all localized geologic features and physical information such as paleo-channels, fault lines, burn areas, limits of lenticular units, weathered zones, and property, section, or mined-out areas. Almost all mapping packages include a capability for plotting property boundaries along with control point locations, names, and the associated Z values over the contour map. Most of these packages also allow for plotting of various drillhole symbols representing core holes, open holes, oil wells, gas wells, environmental holes, hydrological wells or others.

8.4.6.2 Reserve Calculation

The reserve calculation is the simplest and yet the most elusive part of any mining computer program. It is the simplest because it is a multiplication and summation process. It is elusive because to calculate an accurate reserve, there are many intricate points that need to be considered. Reserves can be calculated by various techniques, that is, by a simple polygonal method where the area of influence around each drillhole is multiplied by the thickness of the unit or the percentage grade. This method of volume calculation is not very accurate, and it is only used when an order-of-magnitude calculation is intended or when very dense drilling is available. Another method of reserve calculation is double-end area calculation from cross sections. This is the same process as the manual method; only a digitizer is used to expedite the calculation. An automatic cross-sectional calculation approach is used for steeply dipping deposits, which is basically the most accurate volumetric method for such deposits. The third method is again a duplication of the manual process where a planimeter is replaced by a digitizer tablet. Here the

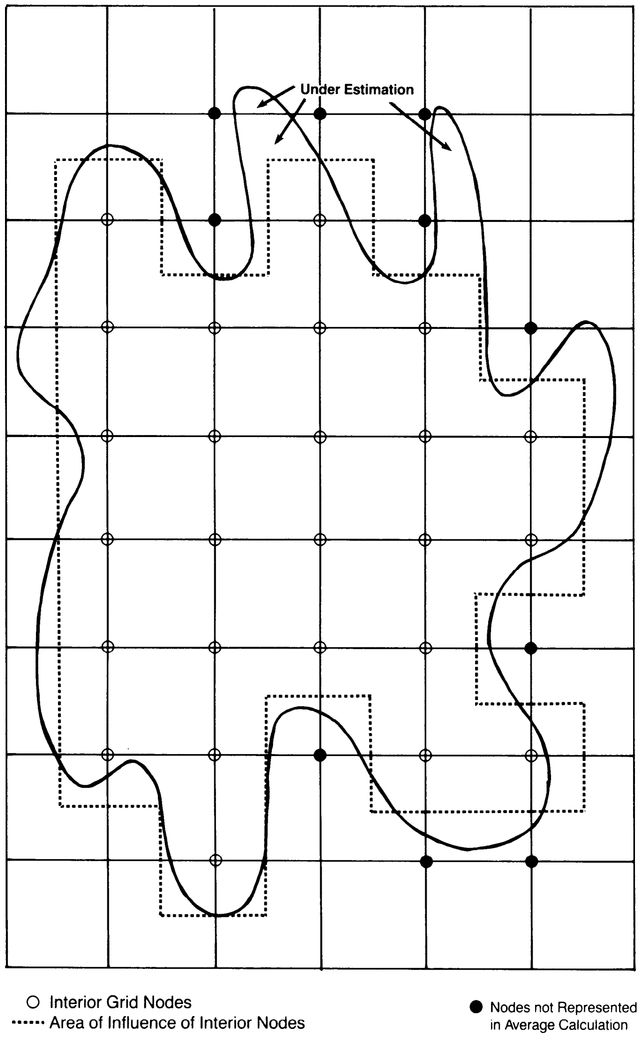


Fig. 8.4.20. Effect of irregular boundary on volumetric calculations.

isopach maps of the ore and waste are placed on a digitizer, and each contour level is traced separately to determine the area of influence under the contour. The volumetric program uses the contour thickness value and the area traced by the digitizer to calculate the volume. If tonnage calculations are required, then a density factor is used to convert the volumes to tons. This type of calculation is equivalent to the manual approach at a much faster speed, however, the computerized version automatically accumulates the results for each unit thickness and provides a final report.

Introduction of grids opened a whole new approach to volumetrics. Using grids, the volumetric program adds the values of all the nodes falling within the digitized area and calculates an average value which, when multiplied by the area, will result in desired volume. This process is fine for flat-lying or uniform deposits and boundaries that are smooth. In situations where the digitized boundary is an irregular shape, the result of the volumetric calculation varies as a function of irregularity. The higher the degree of irregularity the less accurate the result, because many nodes close to the boundary, but not falling within it, will be excluded from the average calculation (Fig. 8.4.20). It should be noted that the volume inaccuracy is also dependent on the grid spacing. As a result, volumetric programs should

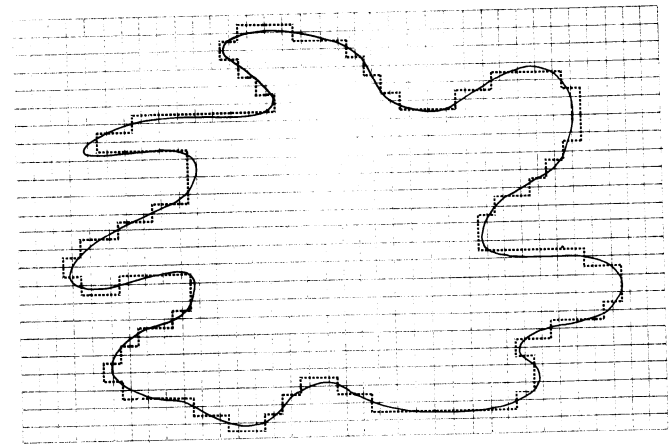


Fig. 8.4.21. Increase of volumetric calculation accuracy using smaller grid size.

provide automatic regridding of the data to smaller grids, at user's option, during the volumetric calculation (Fig. 8.4.21). The smaller the grid spacing, the better the precision of calculation, provided the original data warrants the smaller spacing.

The more advanced version of reserve calculation programs assign an area of influence, that is, a small rectangle equal to the grid spacing to each grid node and then will intersect each one of these rectangles with the property boundary (Fig. 8.4.22). The resulting area is multiplied by the value of the grid and summed up to provide the volumetric result:

$$V = \sum_{i=1}^{n} A_i * T_i \tag{8.4.1}$$

where i is the individual grid node, A is the intersection of the digitized boundary and the area of each rectangular grid, and T is thickness at each grid node.

This is the most accurate approach to in situ volumetric calculations; however, it requires much more CPU and it takes longer to calculate. This approach is fine if a thickness grid is used, but if the program is to calculate the thickness from subtraction of two structure grids, then the problem is somewhat more complex. Fig. 8.4.23 shows a cross section of a deposit with topography and top and bottom structures of two seams. Vertical lines represent the location of the grid nodes. Subtracting the top structure of the first seam from the topography at each grid node will result in a waste grid above the first seam as shown in Fig. 8.4.23. Direct subtraction of one node from the other will result in a series of nodes with positive and negative values. Since the negative thicknesses are not valid, the remaining positive grids result in an overburden thickness grid that is not representative of the pinching portion of the unit. There are two ways to resolve this problem. First, the volumetric program may be set up such that it will automatically extend the area of influence of each grid at the edge of the pinch out by half the grid spacing (Fig. 8.4.24). The second approach is to replace all the negative values with zeros. In essence, this will have the same effect as the previous method. The only disadvantage of these two approaches is that grid spacing will play an important role in the final result, that is, the larger the grid spacing the lower the accuracy of calculation. In other words, these approaches assume that the pinching always occurs halfway between the positive and zero or negative grid nodes.

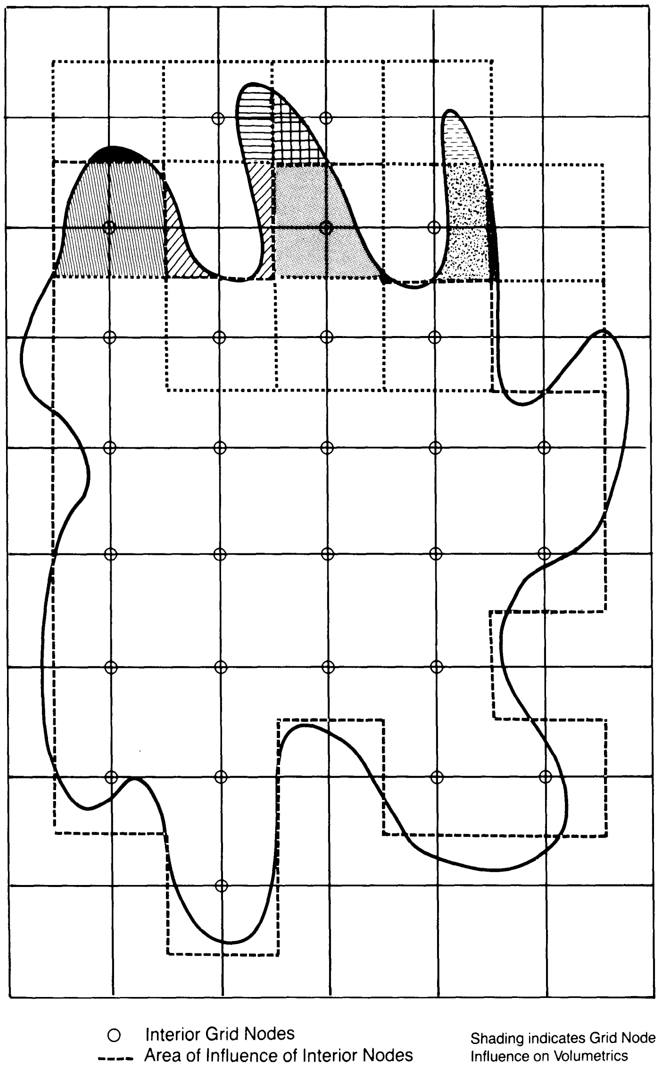


Fig. 8.4.22. Volumetric calculations using actual area of influence.

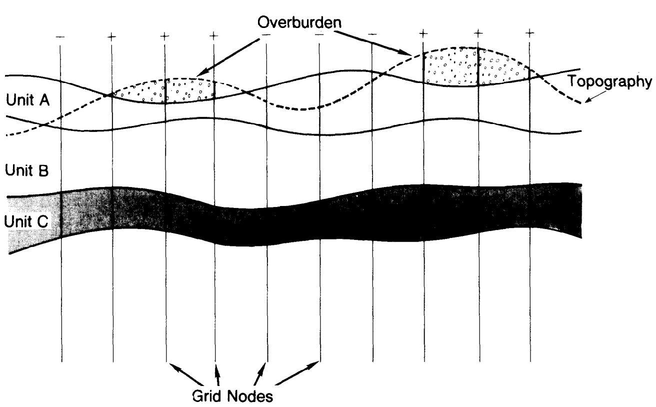


Fig. 8.4.23. Representation of computer generated waste above first seam.

To alleviate the foregoing problem and to improve the accuracy, volumetric programs are designed to request the degree of accuracy desired from the user. If the volumetric is for bulk

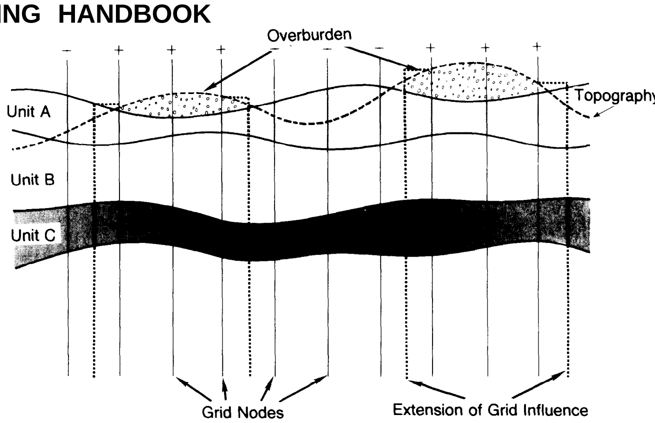


Fig. 8.4.24. Result of grid extension on volumetric calculation.

estimate, then the lower accuracy level is used with a much faster response. However, if higher degrees of accuracies are desired, then the program automatically divides the grid spacing into smaller units, depending on the level of accuracy requested. In essence, the higher the degree of accuracy, the smaller the integration steps. It should be noted that the higher the level of accuracy, the longer it will take to obtain the result.

Computerized volumetric calculations are superior to manual methods in many respects. For one, the calculation speed is much greater, even when the highest degree of accuracy is used, which will result in high levels of CPU usage. In addition, computerized volumetrics provide different accuracy levels and speed, as well as minimizing the calculation error compared to the manual method. Using the geologic model of the deposit, one can calculate and generate a cumulative report for the entire property by merely digitizing the areas of interest. The programs can additionally accommodate dilution, reduction, or minimum thickness considerations in a coal mining operation, or recovery and processing factors in other types of mining. Concurrent with volumetrics, one can also generate individual as well as cumulative seam ratios.

8.4.7 MINE PLANNING

At the end of the modeling process, depending on the number of units or depth of mining considered, tremendous amounts of information and files may have been generated. To simplify access to the latest data available for the property of interest, all the information is combined in a three-dimensional model of the deposit. This model includes the topography, topsoil, structures, thicknesses, grade or analytic data, property, and crop boundaries, fault zones, and other pertinent data all in one place. By collecting all the data in one area and assigning a code to the project or property, the subsequent users will be relieved from bookkeeping chores and can concentrate on the analysis and usage of the data. Data for reclamation, processing, and other factors for each unit may also be added to the list. The volumetric factors may include items such as the unit densities, swell and compaction factors, angle of repose, and abrasion factor.

Mine planning is a complex and involved process. Each mining process is to some degree unique and dependent on the mine location, mine management's experience, economic conditions, and local, state, and federal government regulations. In effect, the uniqueness of a plan is based more on the specific sequence of mining and the equipment used. The entire process can, however, be summarized in a simple statement, that is, the best and the most economical method possible to extract the ore.

In other words, it is the optimum mining process that can result in maximum profit. Given this assignment, the task becomes clear. In order to maximize the profit, one needs to minimize the cost, increase the ore recovery, or a combination of the two. To minimize the cost or maximize the ore recovery, one must be able to evaluate multiple alternatives in a short period of time and have the ability to modify the assumptions used, based on new sets of information. Computer mine planning is the best way to accomplish both objectives, that is, to allow evaluation of multiple alternatives in a short period of time and to be able to change the assumptions quickly.

Computer applications to mining can be categorized under two broad headings. The first includes programs that provide basic analysis of the data and allow a design generation analogous to the manual method. The second consists of programs that provide simulation and optimization capabilities beyond the reach of the manual approach. These two are not always distinct and separate processes; rather they may be integrated in the same program with options to use them or not. Because of the complexity and variation of approach, it is difficult to describe a general system. Instead, major concepts will be provided from EAGLES, and wherever possible there will be discussions of other variations. For discussion of other methods and mining operations, see Pana and Davey (1973), Manula et al. (1977), Mooney et al. (1979), Banfield and Wolff (1979), McMorran (1982), McBride (1988), and Badiozamani (1988b).

Mine planning in EAGLES follows the conventional path for both surface and underground operations. Surface mine planning starts with the definition of the pit boundary and proceeds with the pit slope specifications to the location of safety benches, haulage routes, bench heights, and continues with cut or block generation, mining scenarios, as-mined volumetric calculations, equipment selection and specification, working schedules, production scheduling, plus haulage and other equipment simulations. Underground mine planning starts with building a library of room and pillar or longwall panels (if coal), or stoping designs (if noncoal). These panels will form the basic building blocks from which the entire mine layout is built. Following the layout design, the events follow the path of sequencing, equipment and crew assignment, simulation, and volumetric calculation (Badiozamani et al., 1988c)

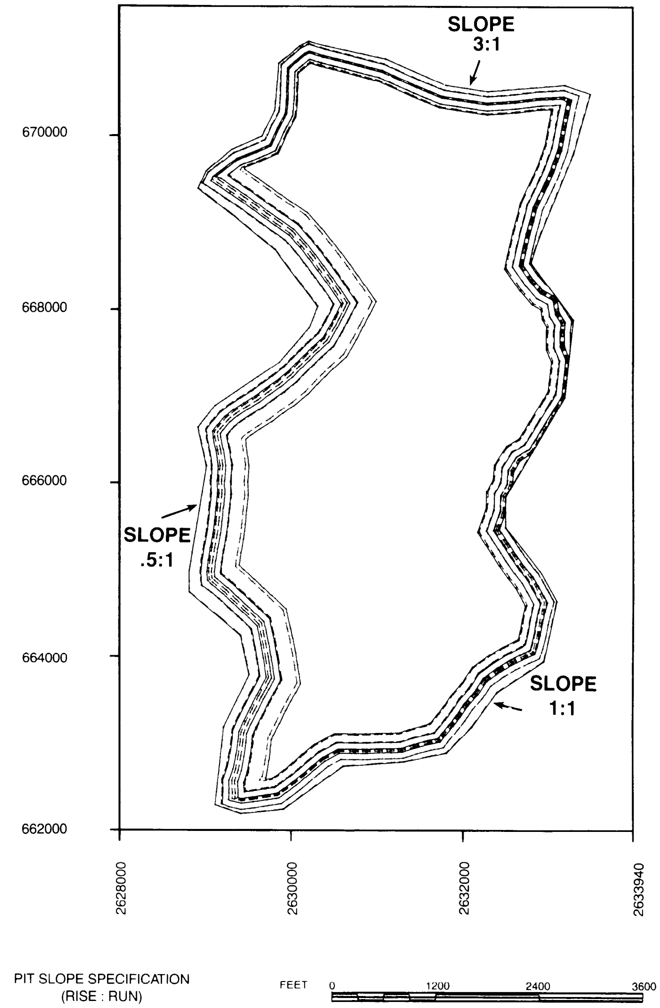


Fig. 8.4.25. Final pit configuration after pit slope specification. Conversion factor: 1 ft = 0.3048 m.

8.4.7.1 Surface Mine Planning

For general coverage of the topic, refer to Chapters 13.1 and 13.2.

Pit Limit Specification: The pit limit can be either generated manually or by computer, given specific criteria. In the manual method, the engineer uses the digitizer or interactive graphic programs to define the pit limits. The pit limit may be a combination of crop boundaries, property line, and the plan-view limit of the ore body. The computer generated limits may be based on volumetric or economics ratio, or limits defined by floating cone algorithm. Using this boundary as the ultimate limit of the pit, one then proceeds to define the pit angles for each lift. The pit angles around the ultimate pit boundary may vary based on geotechnical considerations and the competency of the rocks. Fig. 8.4.25 shows the final result of an interactive graphics screen where the user defines the angles around the pit boundary by specifying the starting and the ending points for each segment. The program then requests the desired angle in degrees or rise to run. The process is continued until the pit angles for all the segments around the pit are defined. The same angles can be applied to all the subsequent benches, or a different set of angles can be defined for any or all benches. The interactive graphic capability allows the engineer to examine and refine the design criteria as the process proceeds. In a matter of minutes, a series of options can be evaluated. By the end of this process, the final configuration of a pit—complete with sloped walls, benches, and haul roads to the bottom of the pit—is generated. This configuration contains the total material, ore and waste, that is to be removed from the pit.

Cut Generation: Results of overall pit configuration are passed to the next program where the actual mining cuts and blocks are defined. Cut definition is based on the type of mining and the type of equipment to be used. In open-cast mining, a dragline, truck-shovel, dozer-shovel, or combination of these are used. Consequently, the width of cuts to be removed in successive advances of the mine is a function of the size of the equipment. The direction of mining and the size of the cuts have considerable influence on the amount of material to be removed at a given time and therefore on the economics of the mine. An interactive graphic program, again, simplifies the generation of multiple alternatives in a short period of time. Fig. 8.4.26 demonstrates the cut generation capability of EAGLES. Other systems may provide different approaches than the one being described here. Simply by specifying two end points of a line, a direction, and cut width, the program generates the entire cut pattern in less than a minute. The cuts can be set up to follow the starting pattern or to be straightened up as they proceed from one end

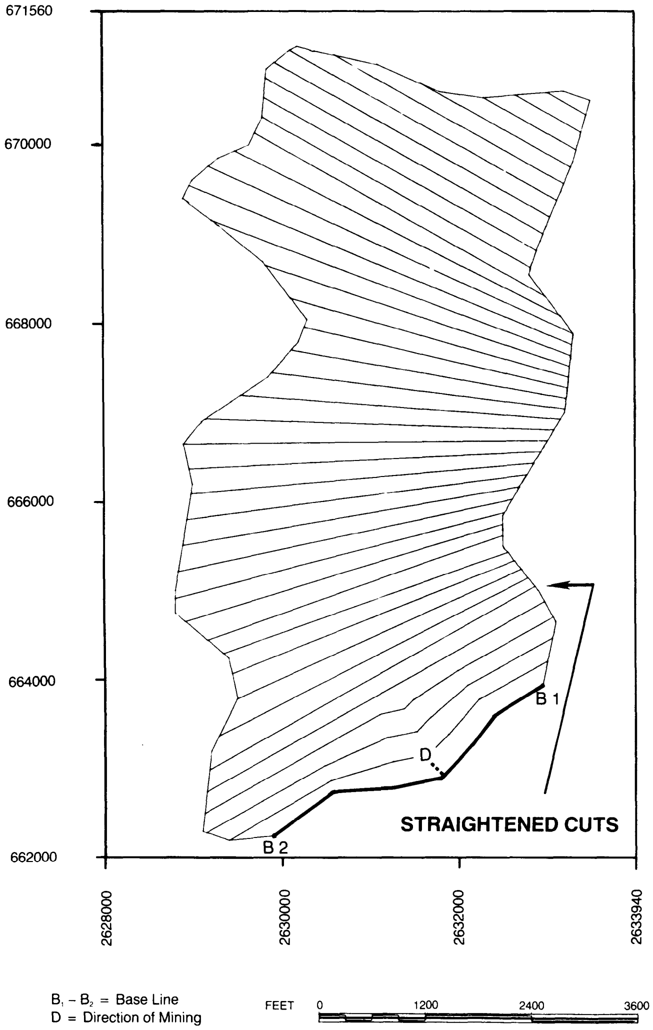


Fig. 8.4.26. Automatic straightening cut generation.

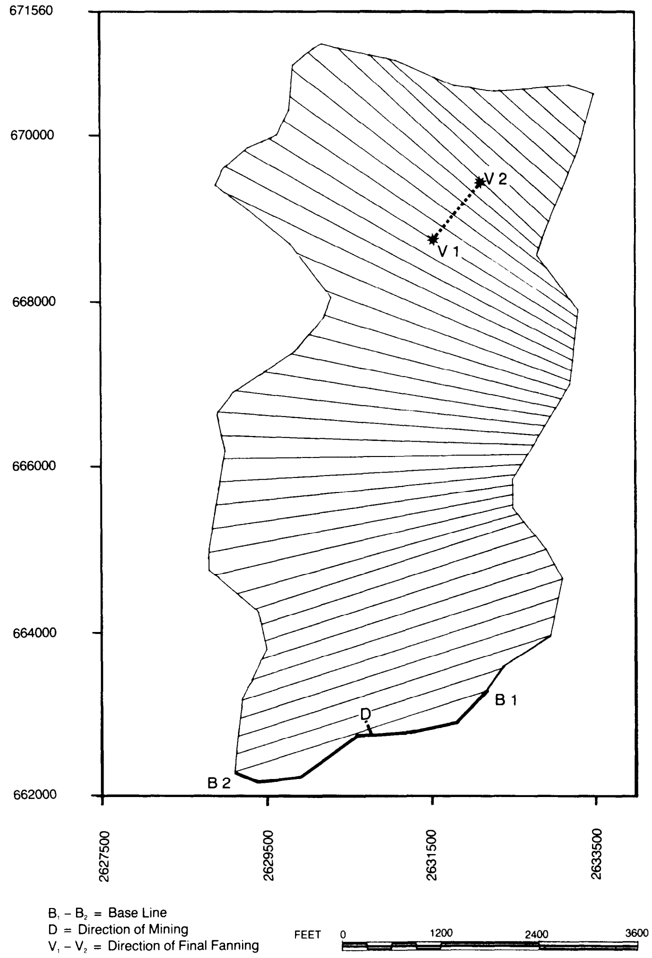


Fig. 8.4.27. Computer-generated fan cuts.

to the other (Fig. 8.4.26). In addition, using a minimum or maximum cut-width specification, cuts can be fanned (Fig. 8.4.27). Changes to the cut pattern or direction of advance is easily accomplished by using the edit function. A range of cuts specifies the limits of modification and eliminates those cuts from the screen (Fig. 8.4.28). New cuts are then designed as before with all the previous functions being available for new cut generation (Fig. 8.4.29).

Upon completion of the cut design, the cuts are projected from one unit to the next, using the specified angles and the benching configuration defined in the previous stage. Each cut is divided into a number of blocks (Fig. 8.4.30), which will form the smallest units to be assigned a volume and grade. Blocks are three-dimensional units defined by cut width, block length, and bench height or stratigraphic units (Fig. 8.4.31). In computerized mining, the block generation is necessary to reduce computation time and space requirements. Otherwise, angle projections, wall intersections, and all calculations have to be repeated each time. Block definition and block generation reduce this process to one time only, and thereafter, it will be mostly straight arithmetic operations.

Mining Scenarios: The preceding steps prepare a three-dimensional model of the deposit with all the associated grades, quality, and analytical information interpolated to each block. These blocks then form the basis for all subsequent volumetric calculation, scheduling, and production sequencing. It should be noted that, unlike the block modeling process, the volumetric or grade calculation in EAGLES is not only based on summation of blocks, but the block values are interpolated to the segments being mined, regardless of their geometric configuration. In other words, the intersection of the pit plane, the bench, the cuts, and the blocks will determine the geometric configuration to be used in volumetric and grade calculations (Fig. 8.4.32).

Mining scenarios are alternate plans to evaluate the most desirable mine plan. These alternatives will consider the effects of different rates of mining from each bench, the trade-offs between a truck-shovel and, for example, a dragline operation. Effects of combining smaller waste zones into the ore zone or thinner ore zones into waste are considered and their merits evaluated. In coal operations, conditional mining scenarios based on variations in the quality parameters can be considered as well. For instance, conditions can be specified, where, if the sulfur content is greater than 1% and the caloric value is less than 4000 Btu (6200 kJ), then that block will be designated as low-grade coal or waste. Or if seam A is less than 2 ft (0.6 m), and seam B is greater than 6 ft (1.8 m), and the thickness of the waste material between the two is less than 2 ft (0.6 m), then A, B, and the waste in between will be considered as ore and will be mined

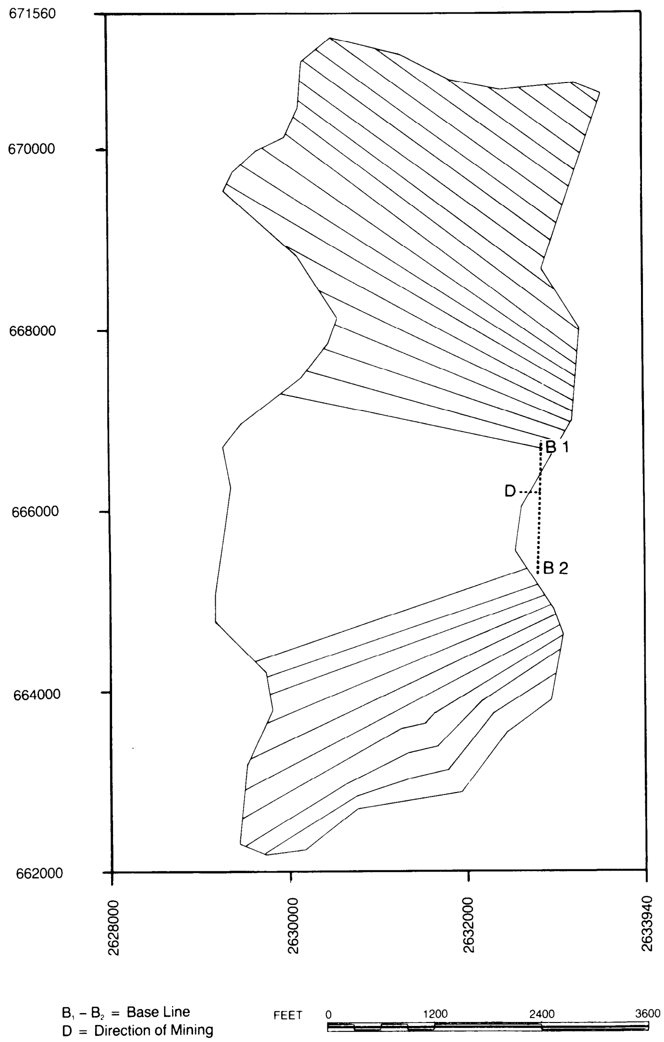


Fig. 8.4.28. Specification of range of cuts for editing.

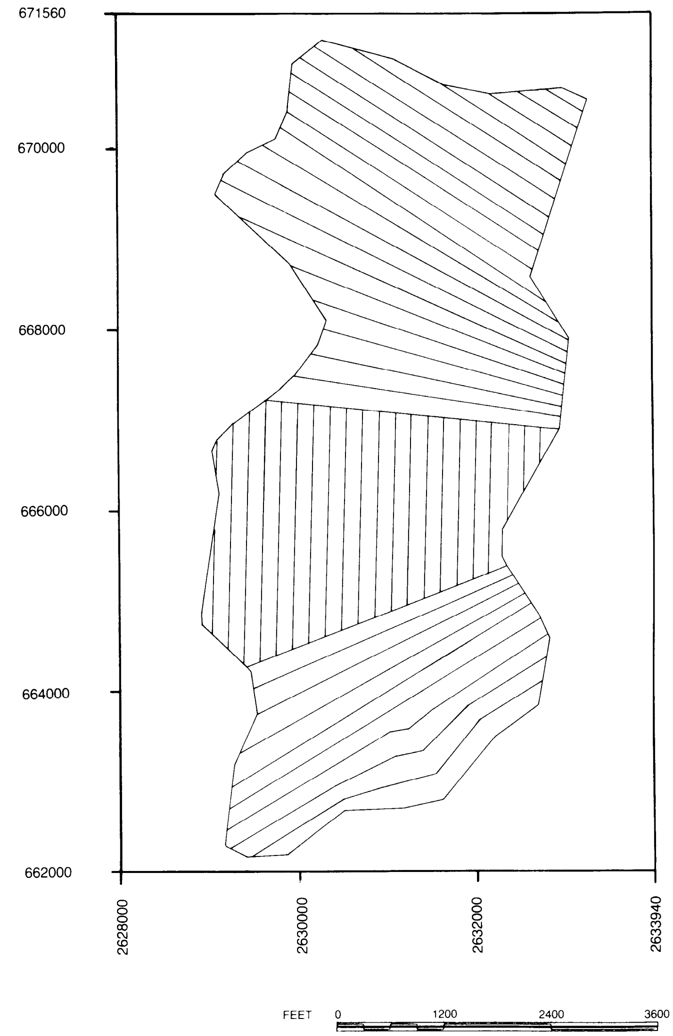


Fig. 8.4.29. Final cut plan after modification.

as one unit. However, if the waste thickness in between is greater than 2 ft (0.6 m), the A seam will be considered as waste along with the waste in between the two seams. These changes in conditional simulations provide a quick answer to different scenarios that an engineer can evaluate readily.

At this point, a bench can be considered as a unit, or it can be divided into smaller units or lifts. A minimum and maximum bench height specification guarantees the division of each bench into the appropriate number of lifts. To generate a new lift, the remaining portion should be equal to or greater than the minimum lift height specified; otherwise, a new lift is not created, and the extra height is kept with the previous lift. The minimum height specification is based on the assumption that stripping equipment cannot operate efficiently on lifts smaller than the minimum height.

Bench specification for advance stripping (benching across the cuts), for operation within a cut (benching within a cut), and for evaluation of different bench heights is also provided. The bench-height specification at this stage has a different purpose than the one specified during pit limit designation. During pit limit specification, the bench-height designation establishes the individual volumetric and reporting units; that is, the volumetric for benches specified at that stage will be always reported as a unit, even though many of them may be combined to form a single bench during mining scenario evaluations. On the other hand, during mining scenario specification, the benches are combined to form a single unit for a specific operation such as dragline bench. When reporting the volume and quality for this bench, each original bench is reported separately along with the total for the combined benches. Prestripping bench configuration is specified by the maximum separation desired for each bench. These separations are defined in multiple of cut widths for the across the cut option and in multiple of block lengths for within a cut. This configuration is used during scheduling to ensure appropriate spacing between the working benches for proper operation (Fig. 8.4.33).

Working Schedule: Operation of a mine depends on the scheduled operating shifts, the number of hours per shift, the equipment fleet, the assignment of each piece of equipment to a given bench, scheduled downtime, holidays, and many other items. These items and data are provided to the mining programs prior to production scheduling. Initially, the number of shifts per day that a mine is planning to operate, the hours that each shift will be operating, the number of days in a year that the

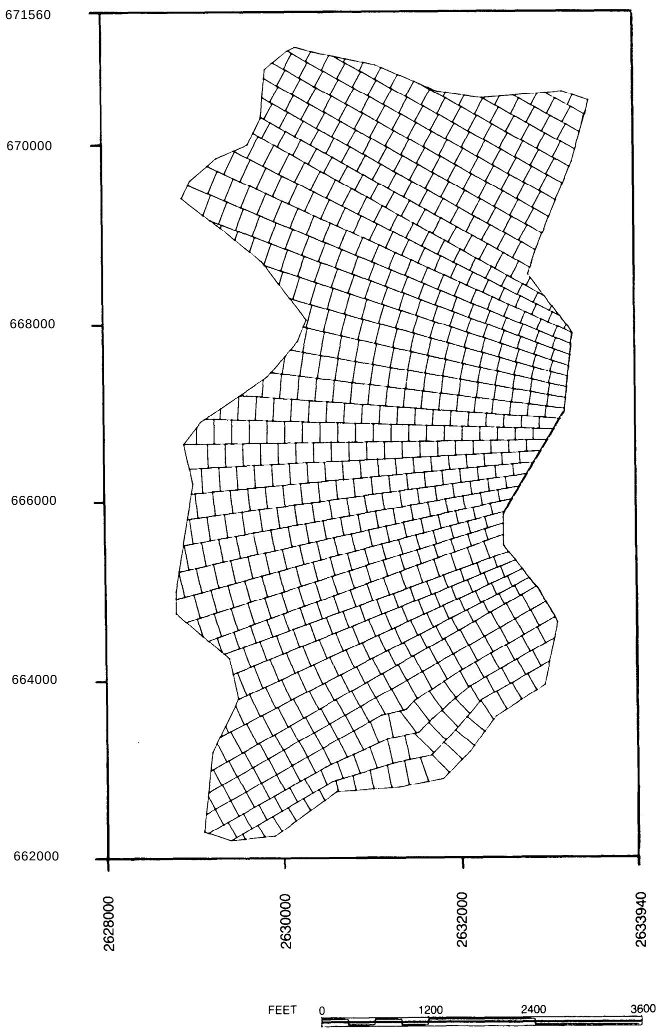


Fig. 8.4.30. Final block generation.

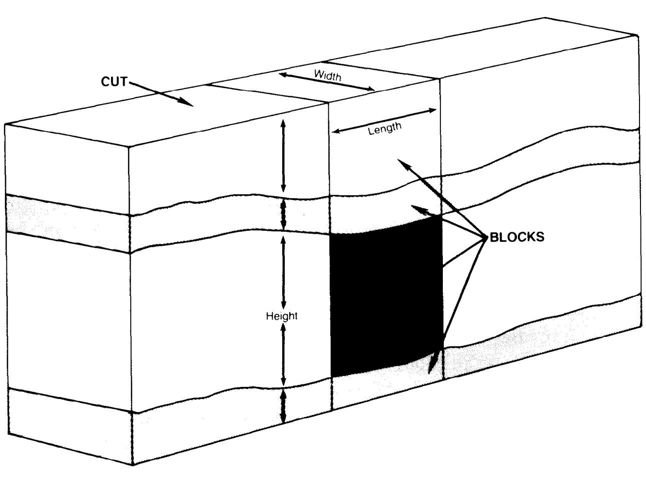


Fig. 8.4.31. Block definition in 3D.

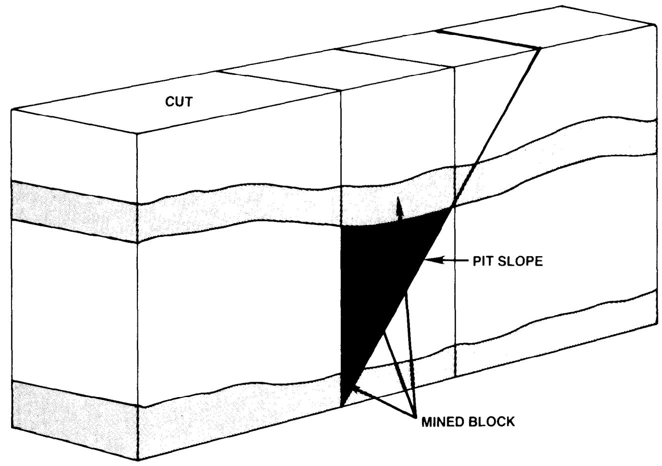


Fig. 8.4.32. Volumetric block definition.

mine will be in production, and the scheduled vacations and holidays will be defined. The schedule can be assigned to any given bench or a series of benches. In addition, the number of Saturdays and Sundays that the mine will be in operation is defined for any given bench desired. These data form the operating calendar for the mine. Following the scheduled days and shifts, the individual equipment units are assigned to a given bench or benches. Also the scheduled downtimes are incorporated in the plan. This information is then used by a production scheduling program to calculate the rate of mining, the tonnage and yardage removed during each shift and throughout the year, and the status of each bench at the end of any given period.

If any overtime is scheduled, the shift differential pay for the overtime and Saturdays and Sundays needs to be specified. The shift differential pay is used to calculate overtime payment schedule.

Production Scheduling: Production targets in a mine may be based on tonnage of coal or ore produced, yardage of waste removed, production capacity of a specific piece of equipment, and the quality or recovery of a product. In each one of these cases the production calculations will be dictated by that specific item. For example, if the production target is based on tonnage of ore produced on a monthly basis, then the program will proceed based on the scheduled shifts per day and number of hours per shift during a given month and will report the tonnage, the yardage, and the stripping ratio for each bench, plus the cumulative values. The calculation is performed by summing the tonnage and yardage for each block along each cut according to the direction of mining specified by the engineer. When the desired target is reached, the total tonnage, yardage, ratio, and weighted quality will be reported for that period and the process repeats for the next period until the entire mine area is covered. In addition, the location of the mined-out area is recorded for each bench so it can be plotted on a plan-view map.

The program uses the mining and benching scenarios established during mine block specification. The calculation at this stage is only a summation and reporting process, according to the factors specified. If the production schedule is based on a specific equipment, then the program uses that equipment's mining rate and scheduled hours to establish the removal process. The operation schedule for this equipment determines the schedule for the rest of the fleet. This is especially applicable to the use of a dragline. Because of the large amount of capital associated with a dragline, mine managers will try to maximize its operation, thus making the remaining equipment subservient to it.

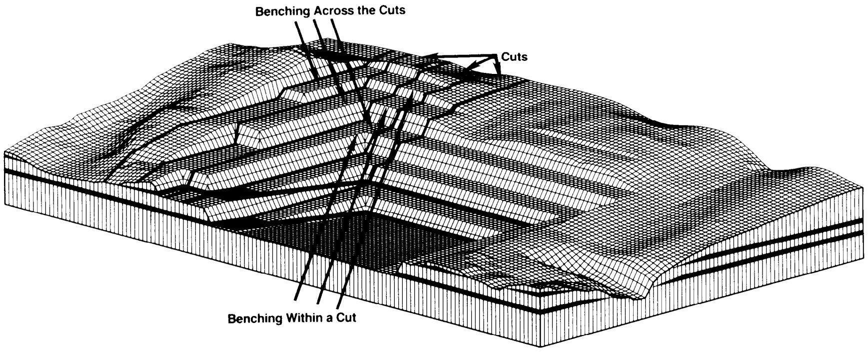


Fig. 8.4.33. Bench specification both within and across the cuts.

In some deposits, such as lignites, the tonnage or the yardage mined is not as important as the millions of Btu (kilocalories) delivered to the plant. As a result, the production target is based on the millions of Btu required. Consequently, the type and number of equipment items selected is tied to the tonnage of lignite that can produce the desired Btu (Badiozamani and Price, 1983).

Production schedules can be defined for long- or short-term operations. Even though the process is similar on the surface, there are many differences in detail. In long-range planning, the main objective is to meet the overall production targets and maximize the life of the mine. In short-range scheduling, the engineer is concerned with the day-to-day operation; and his main goal is to meet specified production targets, maximize the equipment usage, and minimize the cost. The two objectives are not necessarily contradictory, but their emphasis is different. The latter is concerned with using available equipment, moving available equipment to the most critical location to eliminate bottle necks, balancing mining from various pits to meet the grade or quality balance, and minimizing the effect of equipment downtime. The former is concerned with the type of equipment that best suits the long-range planning based on the geological and geotechnical conditions, location and distribution of ore, the balancing and minimizing the rehandled material, location of spoil piles to minimize the cost of reclamation, and so on.

Many procedures and programs have been written to optimize mining operations and sequencing (Lerchs and Grossman, 1965; Lipkewich and Borgman, 1969; Johnson and Mickle, 1970; Reibell, 1971; Manula and Venkatarman, 1973; Davis et al., 1973; Rogado, 1974; Lemieux, 1979). These optimization programs have had more success in hard-rock mining than any other type of mining. The major lack of success may be attributed to three factors. First, many of these programs are limited or restricted in capability so that an engineer does not have the full range of options or flexibility to direct the outcome. Second, the more sophisticated programs are too complex for an engineer without adequate knowledge of operations research and computing to be able to use them. Finally, up to now, the programs did not provide proper interactive graphic tools to help the engineer visualize the sequence of operation and see the effect of a decision on an ongoing basis. With the introduction of PCs and graphics workstations, many of the visualization problems are being resolved. An example of this type of program is provided by Gershon (1988) where he uses a heuristic approach and the concept of a ranked positional weight to arrive at the optimum mining sequence.

As mentioned at the beginning of this segment, computerized optimization, specially for soft-rock mining, may not be ideal or practical. As a result many operations prefer engineer-assisted "optimization" programs where an engineer can evaluate multiple alternatives in a short period of time to arrive at the desired outcome. An example is EAGLES' short-range production sequencing program where an engineer uses an interactive graphic terminal to arrive at the desired sequence. This system is especially useful when one desires to blend various grades of product to obtain a given product. The interactive nature of operation allows the user to select individual blocks, cuts, or benches to be mined. The program marks each unit specified and automatically displays the tonnage, yardage, and quality or grade at the bottom of screen. As the process continues, the results are reported cumulatively. If any of the items exceed the desired target value, the engineer can "un-mine" any sequence and reenter a new sequence. This approach is very useful in an actual mining operation where a quick decision is required to change the mining operation to alleviate a short-term problem in the pit.

Run-of-Mine Volumetric: At this stage, a run-of-mine volumetric is based on the blocks generated by the combination of cuts, benches, block cuts, pit slope, and any other specified haul roads or safety benches. The volumetric programs then use the information for each block and the sequence specified by the engineer, or from simulation, to arrive at the total volumes and tonnages. The program reports in situ as well as run-of-mine data. The data are reported for each seam, each bench, and each cut within the area of interest. Also, if there were any thin-coal units that were considered waste, coal tonnages that did not meet high-grade coal specifications, or burden that was included as ore; they will be reported separately as such for each seam. In addition, quality information for each seam as well as cumulative ratio and product recovery through the processing plant or wash plant is reported. The detailed data are then summed up and reported in the cumulative section at the end of each cut (Table 8.4.2). The cut information is repeated for each weekly, monthly, or yearly period.

8.4.7.2 Underground Mine Planning

General coverage of the topic is contained in Chapters 17.1 and 17.3.

To initiate an underground mine plan, it is desirable to have a computer program that allows freedom of choice to a designer, that is, to start a plan from ground zero or to build upon previously developed panels or layouts. A selection menu provides option for designing a panel, copying panels, or developing a layout. The system then directs the designer's actions according to his selection. For example, if the plan is to start from scratch, then one is asked for general mine-wide information such as property boundaries, location of shafts, markers, etc. In the case of panel design, the system moves to the panel design screen and continues from there.

Design of the Mine Plan: The design phase of the mine planning effort usually has two phases. In the first phase, the panels to be used are designed or copied from an existing plan,

Table 8.4.2. Run-of-Mine Volumetric Report

SURVEY VOLUMETRIC CALCULATION REPORT
ACCURACY LEVEL: 1

AREA MINING METHOD

PROJECT : DEMO
COMPANY :
DATE :
TITLE :

SUMMARY REPORT BY HORIZON

TOTAL AREA IS 97.80 ACRES

HORIZON NO.	HORIZON NAME	IN-PLACE VOLUME BURDEN BCY	IN-PLACE VOLUME COAL BCY	TONNAGE COAL TONS	SEAM STRIP RATIO	CUM STRIP RATIO	CUM BCY/ MBTU	WASHED TONS (000)
9998	TOPSOIL	315,554						
3511	BURDEN	6,452,930						
3510	RIDER		683,051	737,695	9.2	9.2		627
3501	PARTING	2,272,254						
3500	ONE		2,326,030	2,512,113	3.6	2.8	0.22	2,135
3495	PARTING	364,699						
3494	TWO UPPER		2,261,095	2,441,983	3.9	1.7	0.11	2,076
3491	TWO MIDDLE	433,583						
3490	TWO LOWER		629,203	679,539	14.5	1.5	0.11	578
2501	PARTING	13,361,230						
2500	THREE		812,309	877,293	26.4	3.2	0.22	746
TOTAL		23,200,252	6,711,688	7,248,624		3.2	0.22	6,161

HORIZON NO.	HORIZON NAME	SULFUR PERCENT	ASH PERCENT	MOISTURE PERCENT	BTU/ POUND
9998	TOPSOIL				
3511	BURDEN				
3510	RIDER				
3501	PARTING				
3500	ONE	0.67	11.43		8071
3495	PARTING				
3494	TWO UPPER	1.80	11.51		8552
3491	TWO MIDDLE				
3490	TWO LOWER	1.19	12.64		7187
2501	PARTING				
2500	THREE	0.77	10.24	12.62	8148
TOTAL		1.16	11.43	12.62	8170

and in the second phase a layout is generated. The concept of the *panel* as an object means that each panel can be constructed, saved, copied, and modified as many times as required. These objects inherit properties of the parent objects; that is, they maintain the original characteristics of the panel such as heading, crosscut, and pillar configurations, unless modified by the user. The collection of panels for a project is called the *panel library.* Each panel can be placed on the layout or mine plan as many times as required.

Panel Design: Polygons for room and pillar mining using conventional equipment may be designed in three ways. Each method utilizes pillar configuration parameters such as heading and crosscut widths and pillar dimensions to calculate placement within the polygon. Room and pillar mining is chosen as an example because it is the most popular underground method and because it lends itself readily to computer design.

The first method involves a combination of keyboard output and graphical design, using information on overall panel length, width, heading and crosscut widths, and rectangular pillar dimensions to generate a *default* rectangular panel. The generated panel may be sized to the exact length and width indicated, fitting as many full pillars into the panel as possible, or may be automatically adjusted to fit the proper number of headings and crosscuts into the panel.

The second method involves building a panel using only the graphics cursor, in a *free-form* manner. This is particularly useful when irregular shaped polygons must be constructed to meet special circumstances. The panels will be generated to fit as many pillars into the panel as possible.

The third method requires the user to input *exact dimensions* for the panel. This method is very useful for situations where a prespecific panel configuration is established for certain regions of the country, for example coal mines in the Illinois No. 6 seams.

Note that although individual polygons each have their own dimensions and pillar configurations, these polygons may be joined at any angle to form the collection of polygons called a panel (Figure 8.4.34). The panel *object* can then itself be copied and modified as needed.

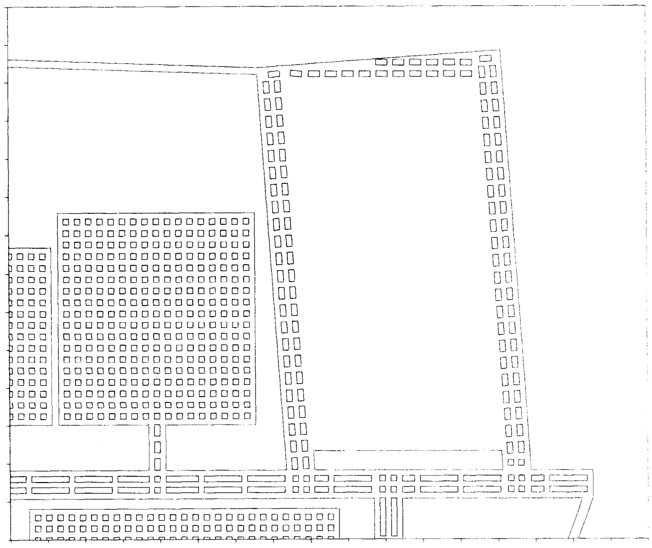


Fig. 8.4.34. Generation of room and pillar or longwall panels.

Panels in longwall mining, another popular underground method, typically consist of many separate polygons, each with its own characteristics. Graphically designing each polygon would be quite tedious, as would entering coordinates into the system. A simple interactive menu requires only that the user supply the dimensions of the longwall block, pillar configurations for the gateroads and back, and the dimensions of the barrier pillar. From these dimensions, the system generates the configuration of the longwall panel (Fig. 8.4.34). The panel itself may then be modified using interactive graphics. It should be noted that this automatic capability includes options for angling the longwall block entry to allow more room for start-up operations, and designating areas for start-up and finishing. These areas will have lower production rates than main longwalling operations.

In each case, the pillars are generated as rectangles according to the input pillar configuration information. The user may modify individual pillars by deleting, cutting, extending, or moving them. In addition, any pillar may be moved to any location within the panel polygon. This allows the user to compensate for local conditions by interactively changing the pillars in that area.

Such localized operations would be quite time-consuming when building chevron or herringbone pillar configurations. To meet these requirements, the program provides the ability to use the first row of pillars in a polygon as a template. All options for modifying pillars (e.g., cutting, shifting, moving) can be applied during the design of the pillar template. Once the first row has been designed, the same configuration will be replicated throughout the rest of the polygon (Fig. 8.4.35). Users can always then modify individual pillars as required. Note that some of the pillars have been modified for local mining conditions.

When panel design has been completed, the results are placed in the panel library. This library is available to the design of the current project and can also be used, through a utility program, in other projects concurrently. Panels may be added to or deleted from the library at any time. The library of panel objects can save considerable amounts of design time and can ensure a high degree of uniformity among various mine design projects.

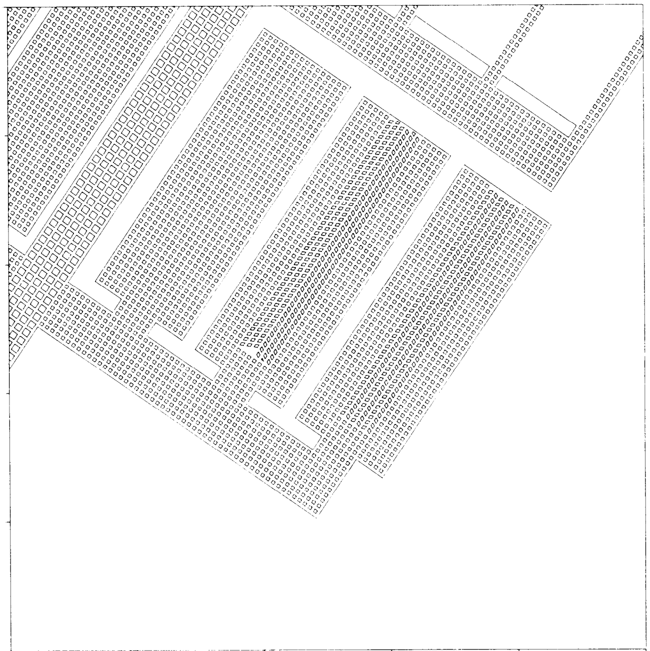


Fig. 8.4.35. Generation of chevron pattern or modified pillars.

Layout Design: The layout is usually entered by combining the digitizing of an existing mine plan with interactive graphics design. The graphics design capabilities indicated in the panel design is largely transferred to the layout design phase as well.

Any submain can be designed in default rectangular form, as a freeform polygon, or can be entered using map coordinates. The same pillar generation and modification capabilities of the panel design phase are present in the layout design phase as well.

The basic steps in layout design follow an intuitive pattern. Initially, basic mains and submains are designed; then panels from the panel library are *snapped* or placed on the layout so they are connected to the submains. Once a panel is selected from the library, it can be placed on the layout as many times as desired, at any orientation to the other polygons on the layout. Additional submains can be added and the process repeated. It should be noted that submains themselves can be copied to other locations on the layout, so that the mine design can be quickly created, cloned, or modified (Fig. 8.4.36).

The layout design phase also contains significant additional capabilities. Any submain can be extended automatically to a specified distance of a boundary feature such as a river, railroad, or inhabited area. This feature allows the designer to be sure that his layout meets required safety criteria.

Once designed, submains can be lengthened or shortened as required. An important feature is that longwall panels can be adjusted in either length or width by adjusting the size of the longwall block. This allows a generalized longwall panel to be used in many different areas of the layout. Once the longwall panel is placed, its dimensions can be adjusted to meet local layout conditions.

Existing Operations: Frequently, alternative mine plans are to be developed for operations already in progress. To accomplish this, the initial step is to digitize the relevant portions of the current mine plan, using specialized digitizing routines. These routines require only that the outside boundaries of each polygon be digitized. Pillar configuration information is entered using an interactive menu system, and the pillars are automatically generated. If desired, individual pillars can also be digitized to reflect specialized conditions.

Another important capability is the use of surveyed information to mark progress on the mine plan. The survey information input can be direct from a total station or by using a forms-driven survey system. The survey data then available for generation of month-end reports or can be plotted on the layout maps.

Sequencing: Sequencing combines the *who*—what mining systems and crews will be working—the *what*—what areas of the mine plan are to be mined and in what order—the *when*—at what times will these crews be working—and the *how fast*—at what rates of advance will the mining systems be moving.

In order to develop the sequence, the system must have the following:

1. The mining systems and crews that will be working.

2. The calendar showing the working schedule for each system. The calendar system contains all information on shift lengths and schedules, scheduled holidays, time off, partial shifts, etc. As such, it reflects the actual work patterns for each crew.

3. The advance rates for each mining system. Advance rates are given in tons recovered or distance traveled each shift. These advance rates may be used interchangeably anywhere in the sequence for any crew.

4. The "relative sequence" showing what polygons will be mined and in what order. The relative sequence would be the actual sequence if only one crew were used. When multiple crews are introduced, the relative sequence provides information on the general order in which areas may be mined.

5. Mining options, such as minimum and maximum mining thickness, and leaving coal roofs and floors. These affect both the rates of mining and the recovered tonnage and quality.

6. Production delays that are required by the nature of the mining systems used. For example, a longwall crew may be required to spend a considerable amount of time setting up before longwalling can begin. These delays must be accounted for by the sequencing system.

To facilitate the entry of this information, a combination of forms-driven screens and graphic inputs are used. The objective is to provide the maximum flexibility of input while minimizing the amount of typing and work on the part of the engineer. Even a complicated sequence can be entered in a short period of time. An expert system is used to ensure that no area of the mine can start production until all required development to reach that area has been completed. Data from the geologic model concerning ore body thicknesses, quality parameters, etc., is automatically included in the calculations. Once the sequence has been completely generated, the engineer can now review the results via an animated sequencing and determine if the sequence meets his requirements.

Feed Back: One key feedback mechanism found in the system is the *animated sequence* feature. This shows the advance by calendar month for all crews and can be generated for any time period in the sequence. The animation demonstrates which crews are working at what particular times and easily identifies delays that may be able removed by revising the sequence, the production rates, or crew schedules. This type of visual feedback is very important because it utilizes the pattern recognition skills of an experienced engineer to determine the occurrence and nature of problems that would be very difficult to determine by simple reports or static plots. For example, delays in finishing longwall development activities can easily be located because the longwall crews will be idle after they have finished their scheduled activities.

Other feedback mechanisms include hard-copy plots of the sequence, showing the areas mined by each crew in a time period, and a production report giving the raw and recovered tonnages

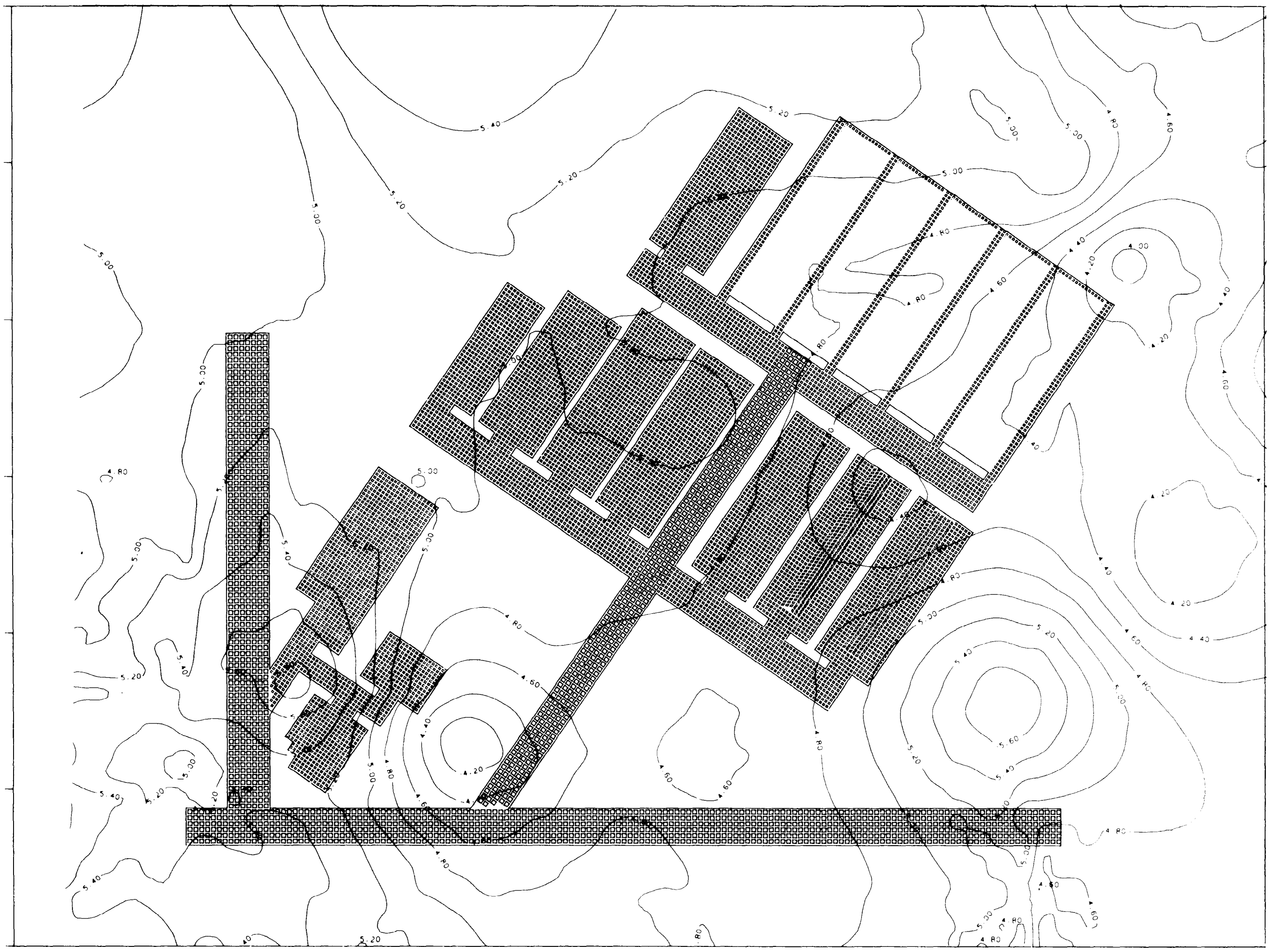


Fig. 8.4.36. Final layout design with superimposed contours.

and quality parameters for selected time periods (month, quarter, year, etc.). In addition, a crew utilization report provides a Gantt chart of crew activities, showing periods of inactivity that may be modified if necessary. With these tools, the engineer can now modify the sequence to more closely meet his requirements. For example, the quality produced in a time period may not meet contractual requirements. This may require higher production from an area of higher quality. One or more crews may be having long periods of inactivity, suggesting that they be shifted to other areas or furloughed.

Finally, the engineer can use current production information to update the sequence and develop projections from this date forward. All production plans are affected by outside events, such as production that occurs too rapidly or too slowly, equipment breakdowns, changes in marketing requirements, etc. The ability to play *what if* games with a sequence, based on currently available knowledge, is a major requisite of any mine planning system.

8.4.8 MINE OPERATIONS

Mine operation programs provide the capability for maintaining records of actual mining operations. These programs can form a set of sophisticated tools for day-to-day monitoring of operations, personnel, equipment, production, and maintenance. In this segment, emphasis will be on the EAGLES' operation system and its application to surface mining, mainly for production, equipment, surveying, and personnel. There will be no discussion of items such as maintenance and inventory.

The system is organized around three databases, that is, daily production, equipment operation, and volumetric (Fig. 8.4.37). It allows daily monitoring of the mine and enables the exchange of data between the planning department and operations. By use of this system, deviations from a plan can be quickly identified and necessary action taken to bring the operation in-line with the plan or to change the plan to reflect the actual operational conditions.

Another function of this system is to allow the resident engineer at the mine to update the plan and the topographic maps to reflect the latest mining advance, the position of equipment, and the location of the pit. Prestripping and reclamation progress is also monitored and mapped. The result of all these activities can be reported on daily, weekly, and monthly reports and maps.

8.4.8.1 Surveying

Surveying systems in recent years have improved to the point where survey data can be captured in the field and transmitted

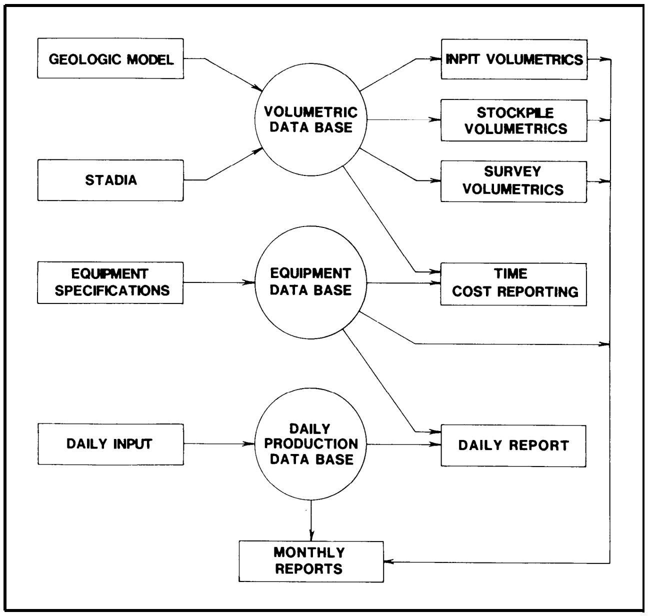


Fig. 8.4.37. Schematic of mine operations monitoring system.

electronically to the computer (Chapter 8.2). The *total station* has eliminated many tedious surveying tasks. Despite the introduction of total stations, many mines still are using the more traditional surveying methods of stadia and chaining. To facilitate and provide for all diverse situations, an interactive screen-driven menu assists the users at the mine site to capture information from any source and incorporate it into the database (Fig. 8.4.38). Following data entry, the coordinates of each survey point are calculated using the information provided for gun site, back site, and each shot point. The program also provides for loop closure and automatic adjustments. Input errors occur especially during the manual input so the program provides for graceful error recovery procedure; and to ensure data input accuracy, it generates a plot of the survey points.

Four types of data are tagged specifically for later volumetric calculations: toe, crest, highwall, and floor. These data are also used to combine the location of the crest of a bench in previous period with the present one to delineate the area mined during this period. One may graphically join these to form a boundary for volumetrics.

Stockpile surveys are captured in the same manner; however, these data are identified separately to allow special interpolation routines for volumetrics. The special interpolation routines are necessary to capture abrupt changes in the stockpile or account for the central pipe. Stockpile volumetrics require special handling because the survey information from the previous period is used to generate a grid of the stockpile surface. The same process is used for the survey data from the latest period. The resulting surfaces are subtracted from one another to determine the relative change during the period. A positive change will indicate an addition to the stockpile, and a negative volume is indicative of extraction. If the total volume of stockpile is required rather than the relative change, then the original ground survey must be available, and therefore the system always retains a copy of the original surface. The same process is followed for maintaining a dump inventory.

8.4.8.2 Mine Production

To keep track of a mining operation, four activities have to be monitored separately: (1) waste production, (2) ore production, (3) reclamation activities, and (4) monthly mapping updates. Waste production is based on the truck counts from the pit, and the volume is calculated from survey data. Ore production is also monitored by truck count, but the weights from the weighing station are also entered into the database to compare with tonnage calculated from the plan. As a general rule, tonnages calculated through multiplication of the mined-out area, times the thickness of the ore, times the density of the ore, never match the tonnage reported by weighing. The sources of discrepancies are numerous. For one, in normal mining operations, the equipment removing the ore is not capable of removing it to the exact interface of the ore and waste. As a result, in certain areas a portion of the ore is left behind, and in other areas a portion of the waste is added to the ore. This mixing of ore and waste, usually referred to as reduction and dilution, contributes to discrepancy between the two measurements. The second source of discrepancy is from the inaccuracy introduced by using only one density value for the entire unit. Experience has shown that coal density varies from sample to sample, and even though an average density is fine for approximating the overall tonnage throughout the mine, it will not compare favorably against localized direct measurements by each truck. The third possible source is from the lack of ability to account accurately for the fractures and gaps in the ore deposit while using the areal extent.

Information gathered from various activities is recorded in a database and is used to update the topographic map from the pit, reclamation, or dump-site survey data. This process eliminates the need for end-of-the-month aerial surveys of the pit except for once or twice a year.

Equipment operating hours, scheduled and unscheduled downtimes, delays, maintenance hours, and other data are re-

SURVEY POINT DATA BASE - EDIT DATA

HORZ NAME : 5200	AREA NAME : AREA 2	DATE : 2/25/89

BACKSITE NAME	:	BSI-A2	GUNSITE NAME	:	GS2-A2			
EASTING	:	2230000.000	EASTING	:	2230000.000	TOTAL STATION	:	X
NORTHING	:	672600.000	NORTHING	:	672400.000		:	
ELEVATION	:	1047.98	ELEVATION	:	1044.12	POINT MEASURE	:	X
BEARING	:	0 0 0 NE	INSTRUMENT HGT	:	0.00		:	

FIELD ANGLE	HORIZ DIST.	VERT. DIFF.	THICK/ PCODE	EASTING	NORTHING	ELEV	PT
70 9 45	825.48	−78.77	6.00	2230776.49	672680.13	965.35	W
82 41 14	777.32	−72.08	7.00	2230771.00	672498.94	972.04	W
91 13 23	693.91	−74.89	8.00	2230693.75	672385.19	969.23	T
96 40 17	567.59	−81.81	9.00	2230563.75	672334.06	962.31	T
108 15 15	544.13	−83.29	10.00	2230516.75	672229.56	960.83	T

Return if form complete

Fig. 8.4.38. Survey data input screen.

corded for each piece of equipment and stored in the equipment database. This information can be used by the operation manager to evaluate the merits of one type of equipment vs. the others, or to check the production variations for different shifts and even for various seasons. The information, once generated, will be useful in developing production rates for the coming year's planning.

Drilling and blasting information is collected for the drilling pattern, the amount of explosives used, the number of holes drilled, and the depth of holes drilled. The summary report provides information with respect to the amount of explosives used for a given period and the drilling footage. Also, drilling data from the blastholes and the ore-waste interface can be passed to the geologic database for incorporation into the modeling. These data are used later as an added control point for interpolation during revisions to the geologic model.

Personnel records contain information with regards to hiring dates, absenteeism, seniority, and other necessary records. These data are available to the foreman to use for assigning appropriate jobs to the crew quickly and effectively. The result of all such information can be reported on a daily, weekly, or monthly basis.

8.4.9 HARDWARE AND SOFTWARE SELECTION

8.4.9.1 Selection Steps

Proper selection of hardware and software play an essential role in acceptance, usage, and satisfaction of computerized systems by the users. Unfortunately, many software packages are impressive and user-friendly on the surface, but on a more thorough evaluation fall short of expectations. To avoid pitfalls and assist the reader in the selection criteria, the following procedure is provided. It should be noted, however, that the computer industry is the fastest growing industry, and some of the generalizations presented may not stand the test of time.

Software and not hardware should be the primary focus of selecting a computerized system. Hardware costs have been dropping steadily, and if the present trend continues, future hardware will be even less expensive. Many companies overlook the long-term and hidden costs associated with new software. For example, for software to get widespread acceptance among users, the computing department has to spend a long time assisting the users and providing free demonstrations and training before expecting user support for their effort. The user's acceptance and willingness to use the system is essential for successful implementation of any system. The computing department should concentrate on the ease-of-use and applicability of software to the user's need. Also those in charge should select the most suitable computer that can accomplish the task, rather than trying to fit software to existing hardware. By trying to retrofit the software to existing hardware, one may spend more than the cost of new hardware while attempting to make the new software work on the existing hardware.

The following approach is recommended to hardware and software selection:

1. Define project scope specifications.
2. Establish a user committee.
3. Define objectives.
4. Define the system.
5. Write a request for proposal (RFP) for the system.
6. Select a short list of candidates.
7. Arrange for demonstration and testing.
8. Score and make final selection.
9. Select hardware.
10. Negotiate a contract.
11. Install hardware and software.
12. Train operating personnel.
13. Implement cooperative projects.
14. Evaluate and provide advance training.

8.4.9.2 Define Project Scope Specifications

The first step in system acquisition is project scope specification. This task involves defining the scope of the project as far as the type of system to be acquired, the overall company needs, and the extent of computerization. If this is the first large system the company is acquiring, questions such as centralized system vs. a distributed computing environment should be decided on. Also it is advantageous to determine which department and the person in that department who will be responsible for maintaining support for the rest of the company. If multiple mines are involved, the method of access by the users in those mines should be decided on at this time.

In essence, project scope should answer who the users are, where the machine will be located, what the system will be used for, how it will serve the users, and who will be responsible for supporting it throughout the company.

8.4.9.3 Establish a User Committee

The next step is to assemble a representative user committee from all the interested departments. This kind of representation ensures participation from all interested parties and at the same time can define their needs by direct participation. A major advantage of early participation by user departments is their involvement in the project, which will make them party to the new development. This will help in breaking the barrier to entry and acceptance instead of being perceived as another concept that is being pushed on them by management. Care should be taken to draft interested and enthusiastic participants rather than the skeptical and ill-informed.

The initial task of this committee is to select a group of three to five individuals who will be responsible for assembling all the input from other users and will draft the final RFP. This group will also participate in demonstration and testing of the products by the short-listed companies. At least one senior geologist, a senior mining engineer, a computer specialist, and a representative from management is recommended.

8.4.9.4 Define Objectives

The committee has to develop a precise and definitive mission objective. This objective must state the reason for computerization and what exactly the system has to be able to provide. For example, the objective may state, "To obtain an interactive, graphic oriented, geologic, and mine planning system to simplify operation, enhance productivity, and improve accuracy. The system should handle shallow as well as steeply dipping seams. The software should have been in use for more than five years and should have at least five installed bases similar to the planned operation."

8.4.9.5 Define the System

System definition provides a detailed description of all the required functions as well as desirable ones. In this study, every area of interest is defined and detailed. For each section, performance criteria have to be defined. For example, for a geologic database, the details may include having the capability of entering borehole names with up to 10 alphanumeric fields, X and Y coordinates with up to 12 numeric fields, surface elevation, lati-

Table 8.4.3. System Selection Weight Factor Example

Geologic Model's General Requirements	Weight Factor
On-line help capability	6
Menu-driven screens	8
Relational database	10
Screen, card, tape input	7
Auto validation process	5
Optional requirements	
Ability to custom design reports	3
Ability to handle English and metric	4
Total	350

Other modules of the system are given similar weight factors and the overall system may be summarized as follows:

Digital terrain modeling	150
Geologic modeling	350
Mapping and contouring	230
Grid manipulation	160
Volumetric calculation	210
Mine planning	400
Total system	1500

tude and longitude, geophysical logs, and hole-deviation information in azimuth and degree. In addition, there should be ability to input data either directly from screen or via card or tape punch. Log date, logger's name, date drilled, and date capped may be defined as desirable items. Other general information may be included for systems operation such as computational speed, input screens, relational database, and range checking requirements.

8.4.9.6 Write a Request for Proposal for the System

Following compilation of the entire set of detailed-functional requirements and system definition, an RFP is prepared and copies distributed to all possible candidates for bidding. This document should contain the objective definition, the detailed system definition, along with a schedule of events and deadlines for RFP submittal plus decision dates. The RFP should also include a short description of the company's primary functions, the overall objective, and location of each mining operation.

8.4.9.7 Select a Short List of Candidates

To compile a short list of the companies that meet specified software requirements, an unbiased weighing system should be developed. The weighing system can be a range of numbers from one to ten for each item, one being the lowest, and ten the highest. The required functions have a much higher weight than the desirable ones. Each function shall be divided into small components and its usefulness to the overall system ranked accordingly. Table 8.4.3 provides an example of weight factors for a geologic and surface mine planning system.

Functional requirements and associated weight factors should be listed and distributed to all members of the selection committee. Copies of received RFPs are also given to each panelist to evaluate and rate each proposal according to the functional requirements and the established weight functions. The result of the committee's evaluation is then summarized and the top three to five respondents selected for further evaluation. It is a good idea to establish the minimum number of points required for a software vendor to make the short list ahead of time. If too many companies make the short list in the first pass, then only the top-ranking companies will be selected, and the lower-ranked ones will be eliminated from further consideration. The short list participants should be notified of the final selection and specific dates set up for actual testing and demonstration. All the other respondents should be thanked for their participation and notified of not making the short list.

8.4.9.8 Arrange for Demonstration and Testing

Arrangements should be made for each company on the short list to demonstrate their software. It is recommended practice to visit all the companies on the short list and to participate in a live demonstration. The reason for this is to allow the selection committee to meet with each vendor, and while evaluating their product to evaluate their staff as well. The number of people available for support, the stability of the company, the length of time that the main support and programming staff have been with the company, and their mode of operation are of primary importance. These items will provide valuable insight to the suitability of the company and its capability to provide training, future support, and the speed by which they will be able to correct software problems.

Companies that look good on paper may not be as impressive in a live demonstration. It is advisable to have a representative set of data which can be given to each company a few days in advance of the visit. This is to eliminate the time necessary to load and convert the data to the vendor's computer system. However, the test data should not be given to contenders too much in advance because they may use it to acquaint themselves with the data, thus obscuring the actual time required to model a property.

The main idea in the collection of test data should not be to discredit the software under investigation, but rather to find out how many of the operations can be automated with each specific software. It is impossible to find on-the-shelf systems that will satisfy 100% of a company's needs. A software satisfying 70 to 90% of a company's requirements is more than satisfactory. The remaining requirements have to be contracted out. As a result, software houses that are willing to modify their software or provide companion programs to satisfy the remaining needs should be sought.

During the visit and demonstration, a similar weighing document will help to rate the companies on their performance. The testing process should include additional items such as competency of the demonstration staff, the ability of the company to respond to questions, the documentation availability and completeness, the length of time to complete the test, and other items of importance to each company.

8.4.9.9 Score and Make Final Selection

Upon completion of the testing process, the result of the evaluation is tabulated and the highest-rated company normally should be selected as the finalist. However, as it is the case with any scoring system, it is usually very difficult to look at the scores alone. There are other factors that cannot be quantified, and care should be taken to somehow consider such factors. For example, the working relation with the vendor's staff may become an issue of concern. The scoring system is fine to quantify the overall performance of the system, but the committee's judgment should also be taken into consideration.

The final vendor should be notified of the decision and the others should be thanked for their participation, and if necessary a debriefing session should be set up to explain why they were not selected. The strengths and weaknesses of their system should be explained in a short meeting.

8.4.9.10 Select Hardware

Hardware selection has become both easier and harder at the same time. It is easier because mining software is available on most major hardware platforms. It is harder because of the availability of too many options. The choices are no longer limited to mainframe and minis but to mainframe, minis, personal computers (PCs), micros, and workstations. At the same time, the number of vendors of hardware has increased extensively. The user not only has to consider the machine type but has to look at the viability of a vendor 10 and 20 years down the road. One has to consider longevity of a vendor and availability of the equipment, consistent with rapidly changing hardware and software technology from the same vendor in the future. The selection of hardware based on equipment capability is also becoming more difficult. Micros have the advantage of cost, ease of use, and user acceptance; however, they do not have the processing power of workstations or larger machines. Workstations provide the power necessary for large computations and interactive graphics, but they have not been in the market long. As a result, not enough software is available for them yet. This problem will most probably be eliminated within a few years.

Software consumes a major portion of a system's cost; therefore, during hardware selection, in addition to capability, the number of users and aggregate cost of software plays a major role in the final decision. If the number of users is small, or if the majority of the users are not in a central location, then PCs or workstations may be more appropriate. However, if there are more than five to ten users in one location, then it is more cost effective to have a central mini or mainframe system.

If possible, the final hardware selection should be based on the recommendation of the vendor. It is important to select hardware that best suits the capability of the software. Some vendors may provide software on many different platforms, but even these vendors have a preferred hardware platform. This reference is based either on the machine that the original software was developed on or the one that they are presently concentrating their development efforts.

With reduction in the cost of workstations, if possible, it is recommended to have a smaller central unit with a number of distributed workstations. Such an arrangement serves two purposes. First, provides a central repository for the entire company, and second, it permits distribution of the processing load. The distributed processing will aid in increased productivity and better utilization of the resources; it will eliminates bottlenecks and transfers the load to the users' sites.

8.4.9.11 Negotiate a Contract

Contract negotiation is as important as the selection process. Among items to be included in the negotiations are training, warranty period, maintenance support, frequency of maintenance release, the unscheduled release in case of emergency, and the availability of source code. The contract should clearly specify the conditions under which the source code will become available to the company in case the vendor decides to stop maintenance or they become insolvent. Some companies prefer having the source code in an escrow account. This arrangement is satisfactory as long as there is a mechanism to ensure that the escrow account always has the latest workable version of the software, a task not easily accomplished. The reason for uncertainty lies in the fact that, as a rule, a bank where the software is kept does not have either the facility or the technical knowledge to test and ensure the workability of the software provided them.

8.4.9.12 Install Hardware and Software

Coordination among the hardware vendor, the software vendor, and the company is the main concern during installation. There are great numbers of items that need to be considered before installation. The location of the hardware has to be identified, and proper wiring for electrical outlets, appropriate electricity such as three-phase 220-V lines, air conditioning, raised floor, patch panel for user terminals, and so on, have to be clearly established and installed before the equipment's arrival. Shipping date for the hardware sets the date for installation. It is advisable to have a member of the software vendor on hand to assist in testing and acceptance of the hardware. Also the software vendor should schedule installation of software concurrent or immediately after hardware installation.

8.4.9.13 Train Operating Personnel

As a general rule, companies do not spend as much time on training as necessary. To have a successful installation, a great amount of time should be devoted to staff training. The payoffs of well-planned training is enormous, and assignment of a certain number of users to the initial training is essential. It is advantageous to have a preliminary training course so that the users will become acquainted with the overall system in a short time. Then they should be given a project to complete using the new system. One of two approaches may be taken at this stage. The first is, after initial training, to let users struggle with the new system on their own, which is usually not very productive, especially if they have to meet a project deadline. A second approach and a much more appropriate one, is to have the users work on a project while a representative of the software vendor is present. The vendor's staff should not perform the tasks; rather he/she should assist in using the correct application programs and keeping the users from taking a wrong turn. Even though the second method is costlier at the start, it will pay for itself in increased productivity and user acceptance in the long run. User acceptance of the system cannot be overemphasized. It makes the difference between a system that will be used extensively and a system that will be forgotten in few years.

8.4.9.14 Implement Cooperative Projects

The best strategy for successful implementation of a new hardware and software system of the magnitude discussed previously is a three-step process. The first step is to have initial training for a limited number of users. These users may be referred to as *super-users,* for lack of a better term. After the initial training, the software company should be asked to complete a project in the client's office, where the super-users will participate as assistants. During this process, the super-users gain first-hand knowledge of the usage of the system and the speed with which a project can be completed using the software. At the same time, they will gain valuable lessons in using the total package at once rather than at a later date when they have forgotten the lessons from the training. In the third stage, the roles are reversed—that is, the super-users perform a project while a member of the software company monitors their usage of the software and interjects guidance when necessary.

At the completion of cooperative project implementation, the super-users are ready to train other members of the company by conducting similar training and cooperative projects throughout the company. The payoff of such a plan is very high, because it can contribute right from the beginning to the correct usage of the software. In addition, the super-users are available at any time to assist new users and to distribute the new technology to the rest of the staff. This approach will reduce the need for continuous dependence on the software vendor as well.

8.4.9.15 Evaluate and Provide Advance Training

To complete the process, a review and advance training session should be set up for the super-users and others, three to six months following the system usage. After six months of usage, the users are more confident of their ability and can question the software vendor on specifics of their applications. The software vendor should be ready to respond to the users' questions. At the same time, they may provide advanced training plus shortcuts to accomplish the more difficult tasks. As mentioned before, on-the-shelf software cannot usually satisfy all the requirements of a company. Additional requirements that the existing software cannot provide may be discussed, and new development can be initiated during the advanced training session. It is a much better approach to wait for at least six months of usage before modifications and additional capabilities are requested. There is a tendency to overreact to functions provided by a software that are different from the standard procedures in a company. Users usually want to maintain familiar routines and resist any changes in approach. However, if they are given the opportunity to use the system as is for a period of time and make their recommendations for change after usage, then they will have a more objective viewpoint. This procedure will eliminate unnecessary modifications and additional cost to the company.

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Chapter 8.5
LABOR RELATIONS AND TRAINING

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8.5.1 MANAGEMENT'S LABOR RELATIONS RESPONSIBILITIES IN LABOR NEGOTIATIONS

8.5.1.1 Employer's "Good Faith" Bargaining Requirements.

Once the status of a bargaining representative for employees is established, the National Labor Relations Act imposes a duty on both the employer and the bargaining representative to bargain "in good faith." Specifically, Section 8(d) states:

"For the purposes of this section, to bargain collectively is the performance of the mutual obligation of the employer and the representative of the employees to meet at reasonable times and confer in good faith with respect to wages, hours, and other terms and conditions of employment, or the negotiation of an agreement, or any question arising thereunder, and the execution of a written contract incorporating any agreement reached if requested by either party, but such obligation does not compel either party to agree to a proposal or require the making of a concession." (Anon., 1959.)

In addition, under Section 8(b)(3), it is an unfair labor practice for a labor organization "to refuse to bargain collectively with an employer, provided it is the representative of his employees. . . ."

What constitutes "good faith" on the part of the employer or the union is difficult to define; indeed, the National Labor Relations Board (NLRB) has described the test of good faith as a fluctuating one, "dependent in part upon how a reasonable man might be expected to react to the bargaining attitude displayed by those across the table" (Anon., 1959).

8.5.1.2 "Good Faith"–What Does it Mean to the NLRB?

Although it is difficult to establish a precise definition of the term "good faith," the basic requirement of good faith bargaining is that both the union and the employer must negotiate with the view of *trying to reach an agreement.* The bargaining obligation is not satisfied merely by going through the notions without actually seeking to adjust differences. In order to determine whether the conduct of the union and employer meets this general requirement, the Board and the courts look to all the relevant facts of the case; the "totality of conduct" of the parties is the standard for testing the quality of the negotiations.

8.5.1.3 "Good Faith Bargaining" Factors

In examining the conduct of a union or an employer, the Board and the courts look to the following, among others, to ascertain whether the parties have bargained in "good faith:"

1. Whether concessions have been made.
2. Whether an adamant position has been taken.
3. What proposals and demands have been made, and when they have been made.
4. Whether counter-proposals have been made.
5. The sufficiency of the company offer.
6. Whether there has been discussion of proposals.
7. The duration of negotiations.
8. Whether there have been inconsistent positions taken.
9. Whether there has been withdrawal of offers or concessions.
10. Whether the employer is willing to take a position on economic issues.
11. Whether either party has attempted to delay the negotiations.
12. Whether either party has used evasive tactics.
13. Whether the employer has made any unilateral changes in working conditions.
14. Whether the employer has attempted to deal directly with the employees rather than with the union.
15. Whether negotiators have been given sufficient authority to carry on meaningful negotiations.

Any one of these factors alone does not constitute "bad faith" in itself. However, if many of them are present, the bargaining will be deemed to be "in bad faith."

8.5.1.4 Management Guidelines in Dealing Directly with Employees During Labor Negotiations

Once the representative status of a union is established, the employer is required to deal with the employees through the union rather than dealing with the union through the employees. Thus any attempts by the employer to bypass the union representative may be held to be evidence of bad faith.

The Board has condemned employer attempts to deal directly with employees in order to *undermine* a certified union bargaining agent. Examples of conduct that the Board has found objectionable are:

1. Offering the employees a wage raise if they will disavow the union.
2. Seeking individual contracts with employees after having recognized a union.
3. Making speeches or statements to employees demonstrating anti-union sentiments.

It should be noted that the employer is free to communicate with his employees both during contract negotiations and during the term of the collective bargaining agreement, so long as his communications do not undermine the employees' bargaining agent. For example, an employer is free to tell his employees about the proposals he has made to the union during contract negotiations, and he may even attempt to convince them of their fairness, so long as his conduct does not amount to an attempt to *bypass the representative.*

The Supreme Court, in *NLRB v. Wooster Division of the Borg- Warner Corp.,* 356 U.S. 342, 42 LRRM 2038 (1958), clarified an employer's duty to bargain "in good faith." The Court divided lawful subjects of bargaining into two groups: mandatory subjects and nonmandatory (permissive) subjects. The Court read Section 8(a)(5) and Section 8(d) in combination and stated

". . . these provisions establish the obligation of the employer and the representative of its employees to bargain with each other in good faith with respect to "wages, hours, and other terms and conditions of employment. . . ." The duty is limited to those subjects, and within that area neither party is legally obligated to yield. . . . As to other matters, however, each party is free to bargain or not to bargain, and to agree or not to agree."

8.5.1.5 Mandatory Subjects of Bargaining

An employer must bargain with the union about "rates of pay, wages, hours, and other terms and conditions of employment." This duty encompasses such topics as bonuses, pension plans, profit-sharing plans, stock purchase plans, merit wage increases, and company meals, discounts, and services.

Mandatory subjects of bargaining are not only topics that an employer is compelled by law to negotiate in good faith, but also include topics where an employer is barred from *unilateral action.* An employer never has a legal duty to concede a mandatory subject during bargaining, but he must bargain over these topics until an agreement is reached or the parties come to an impasse. Refusal to bargain about a mandatory subject prior to *impasse* is a Section 8(a)(5) violation.

Mandatory subjects of bargaining include:

1. *Wage Compensation for Services*
   - Wages
   - Salaries
   - Incentive pay or bonus
   - Shift premiums
   - Overtime premiums
   - Premium pay for work on Sundays and holidays
   - Merit pay increases
   - Equity pay adjustments
   - Cost of living adjustments (COLAs)
   - Premium payments for undesirable schedules of work
   - Red-circle pay
   - Longevity pay or premiums
   - Pay for training
2. *Wage Compensation for Nonworked Time*
   - Holidays
   - Vacations
   - Jury duty pay
   - Funeral or bereavement pay
   - Call-in or call-back pay
   - Report-in pay or daily/weekly minimum pay
   - Stand-by pay
   - Travel pay
   - Pay for time spent on union business
   - Pay for time spent on safety matters
   - Severance pay
   - Pay for lost wages due to contract violation
3. *Non wage Benefits*
   - Holiday bonuses or "gifts"
   - Pension benefits for current employees when they retire
   - Profit-sharing plans
   - Stock purchase or bonus plans
   - Employer-provided or employer-reimbursed housing, meals, and services
   - Medical, life, sickness and accident, dental and vision plans
   - Legal services plans
   - Discounts on company products
   - Leaves of absence (paid or unpaid)
   - Clothing and tool allowances or benefits
   - Company-provided or employer-reimbursed transportation
   - Tuition reimbursement programs
4. *Working Conditions on the Job*
   - Hours of work and work schedules
   - Rest and lunch periods
   - Plant rules of conduct
   - Grievance procedures and arbitration
   - Promotions, transfers, layoffs, recalls, etc.
   - Work loads
   - Discharge and discipline
   - Safety and safety rules
   - Probationary and trial periods
   - Testing of employees, including use of lie detectors
   - Job-required clothing or equipment
   - Job duties
   - Job qualifications and certification requirements
   - Workplace facilities and conveniences
   - Seniority accumulation (loss and application)
   - Work assignments
5. *Management Rights and Union-Management Rights*
   - Subcontracting
   - Management rights clause
   - Effects on employees of plant closure, relocation, sale, etc.
   - Union shop or other union security agreement (unless state law prohibits)
   - Dues checkoff clauses
   - Bargaining unit work
   - No-strike clause
   - Union hiring halls
   - Nondiscrimination clauses
   - Waiver or "zipper" clauses
   - "Most favored nation" clause
   - Arrangements for labor contract negotiations

8.5.1.6 Non-mandatory Subjects of Bargaining

Nonmandatory or "permissive" subjects of bargaining are all legal topics that fall outside the phrase "rates of pay, wages, hours, and other terms and conditions of employment." The law permits, but does not compel, an employer to bargain about these topics. Some of these topics include industry promotion funds, settlement of charges, union labels, and internal union affairs. An employer may never insist to the point of impasse on a permissive subject of bargaining. Insistence on a permissive subject of bargaining is a Section 8(a)(5) violation. (Anon., 1957)

Nonmandatory subjects of bargaining include:

- Definition of bargaining unit
- Conditions affecting supervisors
- Formal parties to the collective bargaining agreement
- Performance bonds
- Legal liability or indemnification clauses
- Identity of either party's bargaining representatives
- Internal union affairs including contract ratification procedures
- Union label
- Industry promotion funds
- Settlement of unfair labor practice charges
- Multi-employer or multilevel negotiations
- Pension benefits for persons presently retired
- Interest arbitration
- Recording or transcribing of negotiations
- Closed shop (illegal)
- Hot cargo clauses (illegal)
- Clauses which are racially or sexually discriminatory (illegal)
- Union shop clauses (in states with right-to-work law)
- Strike insurance or mutual aid plans (MAPs)

8.5.1.7. NLRB's Concept of "Good Faith" and "Bad Faith" Bargaining

In determining whether an unlawful refusal to bargain has occurred, the Board and the courts normally probe the conduct of the parties for evidence of their subjective "good faith." In some situations, the Board will find a "per se" violation of Section 8(a)(5). A "per se" violation occurs when an employer refuses to discuss a mandatory subject of bargaining, insists until impasse on a permissive or illegal subject of bargaining, or makes unilateral changes in proper subjects of bargaining. (Anon., 1957)

What is said by management to employees and actions taken toward them during labor negotiations will be considered by the NLRB to be the acts and opinions of the company. Every supervisor and member of a management team must realize that any word or action by them may give rise to an unfair labor practice charge by the union.

The following few ground rules are offered to help understand the areas of permitted and prohibited conduct by supervisors and all members of management (Anon., 1979). By no means are these statements to be considered complete on the subject. Any questions which arise as to proper legal conduct should be referred to a labor attorney.

What Management Cannot Do or Say.

1. Management cannot bargain individually with employees on even minor matters pertaining to wages, benefits, and working conditions.
2. Management cannot threaten to close or move, or to drastically reduce operations, to avoid the union.
3. Management cannot threaten to discharge employees who engage in union activity.
4. Management cannot threaten, coerce, or in any way intimidate employees in their right to engage in union activity.
5. Management cannot spy on meetings.
6. Management cannot promise employees a pay increase, a promotion, betterment, benefits, or special favors not previously offered to the union in negotiations.
7. Management cannot call individuals or small groups of employees in the bargaining unit in to supervisors' or management offices to make anti-union speeches.
8. Management cannot make false statements about the union or its position on issues in negotiations.
9. Management cannot make a statement or hint that the company will not negotiate in good faith, or that management does not want a contract, or that management will never sign a contract.
10. Management cannot show favoritism between pro-union and anti-union employees.

The TIPS model is an easy and effective way to remember the foregoing things a manager cannot do (Fig. 8.5.1).

Threats

Interrogation

Promises

Spying

Fig. 8.5.1. TIPS: what a manager cannot do.

What Management Can Do and Say.

1. Management can tell or describe to employees the company's position in negotiations and the reasons for it. It must always be completely *accurate and factual.*
2. Management can talk with employees about the status of negotiations and answer any questions they may have on any issues.
3. Management can enforce company rules fairly and impartially in accordance with past practice and published rules.
4. Management can tell employees the company will comply in good faith with all legal requirements in any future dealings with the union.
5. Management can continue to manage its business and supervise its people in all departments according to past and present company operating procedures and performance standards.
6. Management can handle employee complaints and problems on the job without going through any union "shop steward" or union grievance procedure. This will continue to be the supervisor's responsibility in all bargaining unit departments and classifications.
7. Management can tell employees that the company plans to keep operating to serve its customers during labor negotiations and during any union strike if a strike ever took place.

8.5.1.8 Management Guidelines on Making Unilateral Changes in Wages, Benefits, and Working Conditions

Once a collective bargaining relationship is established, an employer is not free to change his policy regarding such things as sick leave, wage increases, vacation time, or any other term or condition of employment without negotiating with the union about them.

Unilateral action by an employer during bargaining that affects the wages, hours, or working conditions of employees is normally indicative of bad faith on the part of the employer. However, the continuation of wage scales already in effect or the implementation of increases already promised to employees is not unlawful.

There are, however, limited circumstances when an employer may institute unilateral changes while a collective bargaining relationship exists. These circumstances exist when the union has waived its right to bargain about a particular subject or where an impasse in bargaining has been reached.

8.5.1.9 Recommended Ground Rules for Labor Negotiations

1. Negotiations are divided into two phases: (a) contract language and noneconomic issues, and (b) economic and money issues. The first area of discussion between the parties is limited to contract language and noneconomic items.
2. In advancing contract proposals and language, both parties have the right during negotiations to substitute proposals for those included in the first draft and to add to or delete proposals as negotiations proceed and the problems arising therefrom become apparent.
3. Both parties present demands or counterproposals in writing whenever possible. There is no tentative agreement reached on anything not reduced to written form, read aloud to

both sides, and agreed upon as the understanding between the parties. As complete articles and paragraphs of each article are agreed upon verbally, they are retyped with copies provided each party, proofread, and then initialed and dated by both the union and the company. These are filed together pending total tentative agreement on the entire contract.

4. As proposals and counterproposals are considered and tentatively agreed to by the parties, it is understood that such tentative agreement will become final only after total agreement on all parts of the contract is reached by the negotiating team for each side, and after approval by the company's board of directors, and after ratification by the local union and the international union, if required.

5. During all negotiating sessions, both parties equally share in the cost of meeting room expenses.

6. Should an employee be asked to serve on the union negotiating committee, and if such negotiating session is held during the employee's scheduled work shift, the employee may be excused to serve on the union negotiating committee. Such excused time will be noncompensable by the company.

7. Both parties agree to meet with each other at reasonable and convenient times for the purpose of negotiating in good faith.

8.5.2 PROGRESSIVE DISCIPLINE: DISCHARGE RESPONSIBILITIES

8.5.2.1 Progressive Discipline

The role of a supervisor is to counsel effectively with an employee and obtain the best possible job performance and job behavior. To the extent that behavior is not consistent with desired work requirements, it is the obligation of each supervisor to counsel with each employee to obtain this maximum job performance. As these discussions take place, each meeting should be documented.

The first verbal discussion with an employee about a job-related problem is not necessarily a formal "write up." However, you may want to write yourself a note and/or route to personnel file. The purpose of this is to "jog" your memory as to when this conversation took place and what was said. If the problem continues, the verbal discussion should be confirmed with a written correction notice. This notice should spell out the inappropriate behavior and a detail of your concerns about corrective action. It is suggested giving the employee a copy of this notice.

If the inappropriate job behavior continues, further progressive discipline should be exercised. With each continuing application of discipline, a confirmation in writing must be made. The number of corrective steps and resulting documentation depends upon the severity of the questionable work performance and the steps outlined in your discipline procedure.

When the time comes to implement a discharge, in most cases, it means that management failed in its counseling efforts with the employee. Accordingly, when the discharge decision is ultimately communicated to the employee, it should not be a surprise, for the last discussion prior to the discharge should have warned of the pending consequences.

Each termination should be fully documented. The exact reasons for separation should be noted on the separation form. It is suggested that the employee sign his separation form, along with you, to verify the information stated. The original separation notice is to be returned to the employee's personnel file and a copy should be given to the employee. See Fig. 8.5.2 for a summary of the discipline process.

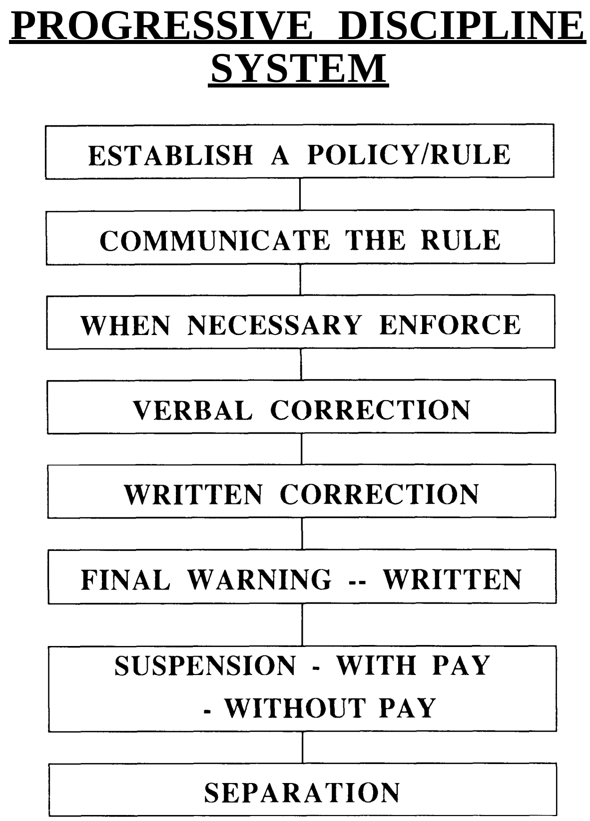


Fig. 8.5.2. Summary of the discipline process.

8.5.2.2 Eight Steps on Correcting an Employee Properly

1. Never criticize, correct, or reprimand your employee before others. Employee discipline and correction are a private matter.

2. Never try to correct or reprimand an employee when tensions are high.

3. Listen. Listen quietly to your employee's point of view of what happened. He/she may see the situation more clearly than you do. Even if he has a distorted view, let him get it off his chest.

4. Share the blame if necessary. Accept your part of the responsibility for the mistake. Perhaps you did not give your employee adequate training or preparation. Or maybe you did not forewarn him/her that this type of problem or situation might arise. Your becoming a "center" with him helps ease the load and assures him that he is not alone.

5. Discuss the problem rather than the employee. Be concerned with correcting a mistake because it is a mistake. Do not focus on the person or his/her personality. To all of us, our person and personality are usually sacred ground.

6. Deal with "why" as well as "how." Many supervisors tell employees what they are doing wrong, but not why they should do something another way. Explain your recommendations fully. Make certain that you and the employee are shooting as the same target with the same kind of gun.

7. Find a better way. A correction or disciplinary interview is not a success unless there is agreement on a better way. No one likes to be told flatly that he/she is doing something wrong. He/she will dislike it even more if he/she is left up in the air

with no solution to his/her problem. Through free give-and-take, try to settle on an approach that you both agree will be better.

8. Finish your correcting and disciplinary interview on a high note. End on a note of optimism and confidence. Don't let your employee feel you have less confidence in him/her because the problem arose (Anon., 1988).

8.5.2.3 How to Avoid Firing Mistakes: A Supervisor's Checklist

When an employee continues to pay no attention to rules and disciplinary action where an offense is repeated, or misconduct is serious enough for discharge on the first offense, decisive action must be taken. However, most discharges can be planned so careful consideration is given to all circumstances.

To help guide you through this area, we suggest you stop and carefully review the following checklist before any employee is ever discharged. Chronic rule violators who continue to menace efficient operations must be dealt with accordingly. On the other hand, every employee represents an investment in time and money—**our employees are our most valuable asset!** Ask yourself these questions, before you fire an employee:

1. Has the policy or rule been properly communicated to the employee?
2. Have I been objective and treated this employee the same as another would be treated for the same offense?
3. Have I accumulated all of the facts accurately?
4. If it is a repeated offense, has the employee been properly reprimanded in the past and have written corrections been issued?
5. Is the employee guilty by his/her own actions or by association with another employee?
6. Is the employee involved in "protected concerted activity," complaining about a job-related problem involving another employee(s)?
7. Am I taking action against the employee because he/she has "challenged my authority"?
8. Does the punishment fit the offense?
9. Was the employee's guilt supported by direct objective evidence, as opposed to just suspicion?
10. Has a top management official reviewed the facts and approved the discharge, or should I suspend the employee first to review all facts?

Remember, this recommended checklist is not very helpful after a discharge. If there is any question about facts or reasons for discharge, suspend the employee instead of firing, during an investigation of the facts (Anon., 1988).

8.5.3 MINE MANAGEMENT GUIDELINES FOR HANDLING WORK STOPPAGES

A misconception held by many is that the National Labor Relations Act only protects employee efforts in organizing, supporting, or joining labor organizations. Employers and employees are often unaware that many other kinds of group actions by employees are also protected by the NLRA, regardless of the presence or absence of any union activity.

The idea that employee group action is protected from employer retaliation emanates from Section 7 of the National Labor Relations Act, which provides in part that employees shall have the right "to engage in . . . concerted activities for . . . mutual aid or protection," 29 U.S.C. Section 157. It is this same Section 7 that affords employees the right to freely engage in union activity. The significance of this is that protected, group employee action is *not* of lesser significance, but is instead afforded the full panoply of rights and remedies granted to employee union activity.

Perhaps the most appropriate way to address group employee activity is to consider it as being a form of union activity. It can be said that, for its duration, protected, concerted activity is in essence *de facto* union activity, in that it is, for that instant, an attempt to bargain collectively over wages, hours, or other conditions of employment.

In order for employee group action to be protected, it must be *both* protected activity *and* corrected activity. Not all employee group activity is protected, concerted activity in that one of the necessary elements will be absent. If it is clearly protected activity, that is reasonable and reasonably related to wages, hours, and conditions of employment, but is not *group* activity, then it is *not* protected, concerted activity. Conversely, if it is clearly group action, but is *unrelated* to wages, hours, and conditions of employment, or irresponsible in its conduct, it is *not protected, concerted activity.*

If the task of determining whether or not particular employee actions are protected and concerted activities seems at times complex, the reader probably has a good grasp of its complexities. At times protected, concerted activity is clearly recognizable—at times it is vague—at times it is only a legal invention.

Nevertheless, following a formula similar to the one proposed hereafter will simply, or at least structure your determination:

1. Determine if the actions are protected
   - Is the object legitimate?
   - Is the conduct permissible?
2. Determine if the actions are concerted
3. If the actions are not clearly concerted
   - Is the employee acting, by admission, on behalf of other employees?

   Or, if not,
   - Do any facts suggest such a representative capacity?
4. Determine if the acts of the sort in which "groupness" is implied by law.

8.5.3.1 How to Handle Protected Concerted Activity

Section 7 of the Act declares that employees have the "right to engage in mutual aid or protection" for themselves through concerted activities on your premises during work time and nonworking time. If your employees engage in protected concerted activity:

- You should meet with them and listen to their complaints about wages, benefits, or working conditions. Tell them you are concerned about their job satisfaction—that you are interested in their well-being—and, that you do want to listen to their complaint, concern, or problems. You have the right to listen to (not negotiate with) them *individually* or in *small group meetings.*
- You should ask "leaders" of employees engaged in concerted activity on a one-to-one basis, *why* they feel the way they do about the particular complaint or problem and, *what* they are asking for in terms of improvements or solutions to these complaints or problems.
- You must not discipline or discharge any employee for engaging in protected concerted activity. This includes verbal warnings, written warnings, or disciplinary suspensions.
- You must not *threaten, coerce, or intimidate* any employee for engaging in protected concerted activity. You are prohibited by Section 8(a)(l) of the National Labor Relations Act from intimidating, coercing, or restraining employees in the exercise of their right guaranteed in Section 7 including the right to

engage in protected concerted activity for their mutual aid or protection.

- You should not commit yourself or agree to making any changes in wages, hours, benefits, or working conditions or to reinstate any discharged employees while the employees are engaged in a work stoppage, strike, sit down, or slow down. Do not let yourself be "blackmailed" by employee threats, ultimatums, deadlines. Make every effort to persuade the unhappy, dissatisfied employees of your sincere and genuine interest in removing any cause of their job dissatisfactions, but you will need time to look into the matter carefully and in depth.

8.5.3.2 If Concerted Activity Does Occur: How to Operate With New Replacements

Employees may refuse to "clock in" and report to work or fail to continue performing their assigned job duties during scheduled work time. If this protected concerted activity occurs, you have the right to inform them of a deadline by which they must report to work and continue their job duties or else they will be subjected to being *replaced with other (new)* employees. The NLRB permits management to hire a *permanent replacement* for any person who engages in any economic strike (a work stoppage involving wages, hours, and working conditions) or any employee engaged in concerted activity even though the activity is "protected." *You cannot discharge or suspend any employee for engaging in protected concerted activity.* But you do have the place to *replace* the employee *permanently* with a new employee yet to be hired.

Your deadline for reporting to work or continuing work must be communicated by a bulletin board notice, mailgram, or letter to the employees' homes as well as verbally to individuals or in small group meetings on the premises. (Caution: it is extremely important that all verbal and written communications from management to employees engaged in concerted activity be screened and approved by a labor relations consultant or labor attorney. The communications must be free of any threats, coercion, or intimidation as a result of their protected concerted activity.)

You may use properly worded advertisements in your local newspaper or on radio and TV to announce permanent job vacancies that are created by employees who are refusing to perform scheduled work by withholding their services. Such advertisements should include a statement to the effect that there presently exists a work stoppage involving several employees. The advertisement may list positive statements about your wages, excellent benefits, and good working conditions to attract as many qualified applicants as possible.

It is recommended that your advertisement include a brief "application for employment" and a telephone number that will be manned during the evening and on weekends. Thus interested applications may "mail in" an application or call for further information without taking time off from their job or coming onto your premises.

Your response to a striking employee or employees will depend, to some extent, on the cause of the strike. If the strike or work stoppage is to protest conduct by the employer that is found to be in violation of the National Labor Relations Act the strikers are said to be unfair labor practice strikers and may not be replaced. If the employees are on strike in support of such bargaining demands as wages, working conditions, or recognition, the strikers are considered to be economic strikers and may be replaced.

8.5.3.3 Document Carefully

To protect yourself from ever-present danger of an Unfair Labor Practice (ULP) charge, you must have a detailed record of every incident involving the work stoppage, sit down, or refusal to perform assigned duties. A conscientious person should be assigned the responsibility for maintaining a "Daily Strike Log" to record every incident, either major or minor. This log should include the following information:

1. Names and job classifications of employees engaged in the concerted activity.
2. Date, time of day, and location of activity taking place.
3. Type of concerted activity.
4. Description of any verbal threats (verbatim if possible) or strike misconduct with the identity of individuals involved—perpetrator, recipient, and witnesses. Take written, signed, and dated statements from recipients of misconduct.

Document all "permanent replacements" for employees engaged in concerted activity. This would include:

1. Name of new employee hired as permanent replacement for the employee engaged in the protected concerted activity, date of employment, first day worked, shift, rate of pay, immediate supervisor.
2. A notation indicating that the new employee has been told that he/she is being hired as a replacement for an individual engaged in the concerted activity. Place a copy of this permanent replacement notice in the new employee's personnel file and a copy in the personnel file of the replaced employee who is engaging in the concerted activity. You need only tell the new employee he/she is being hired as a permanent activity. Do not pinpoint a specific named individual because, while this activity is going on, you may have to shift employees around to cover protection and service needs. You may then wait until the striking employee wants to return to work to determine who specifically has replaced him/her.

8.5.3.4 When Employees Call In or Report Back to Work

When the employees who are engaged in the concerted activity call in by telephone or report in at the end of the work stoppage, wanting their job back, if they have been permanently replaced, tell them so. There is no need to disclose the name of the new person hired as their replacement.

If the replaced employee makes an unconditional offer to return to work, then you are required to place them on a "Preferential Recall" list for the next available position in which they are qualified. *They must be recalled before you hire additional employees off the street.* To provide documentation, you are required to record their name, date of unconditional offer to return to work, and former job classification. You cannot refuse to take them back unless the employee that was permanently replaced finds substantially equivalent employment elsewhere, quits, or resigns, or you abolish their former job for business or economic reasons. Otherwise, should the replacement employee leave the job, the employee replaced must be offered the job now available.

You may hire new employees to replace employees engaged in a work stoppage, strike, or refusal to perform assigned duties at the current hourly or salary rate for that particular job classification. You *cannot* promise new employees who are hired as permanent replacements more pay, benefits, or better working conditions, or to actually give any of these to them in order to induce them to come to work as new replacements.

8.5.3.5 Ground Rules for Managers and Supervisors During Protected Concerted Activity Work Stoppage

What we say to employee strikers and the actions taken toward them during a strike situation will be considered by the NLRB to be the acts and opinions of the company. Every supervisor and member of the management team must realize that any careless word or action by them may give rise to an unfair labor practice charge by the union. We must all prevent this from happening.

The following few ground rules will help you know and understand the areas of permitted and prohibited conduct by supervisors and all members of management. By no means are these statements to be considered complete on the subject. Any questions which arise as to proper legal conduct will be referred to a labor attorney.

8.5.3.6 What Management Cannot Do or Say

1. Management cannot bargain individually with employees on even minor matters.
2. Management cannot threaten to close or move, or to drastically reduce operations.
3. Management cannot threaten, coerce, or in any way intimidate strikers in their right to strike.
4. Management cannot visit strikers' homes to persuade them to return to work.
5. Management cannot take pictures (photographs or videofilm) of *peaceful picketing* or distribution of handbills when it takes place off company property.
6. Management cannot promise strikers anything to try and solicit them to stop striking and return to work.

8.5.3.7 What Management Can Do and Say

1. Management can tell employees we plan to keep operating to serve our customers—strike or no strike.
2. Management can tell employees that federal law gives them the right to strike or not to strike, and we will respect that right.
3. Management can tell employees that federal law gives our company the right to hire a permanent replacement for anyone who engages in an economic strike.
4. Management can enforce company rules fairly and impartially in accordance with past practice published rules.
5. Management can speak to strikers on the picket line so long as no promises or threats are made to solicit them to give up the strike, abandon the union, and come back to work.
6. Management can permanently replace an economic striker unless he has made an unconditional offer to return to work before the time he was replaced with another person.
7. Management can refuse to reinstate an economic striker if he has committed strike misconduct, made threats to other employees, damaged personal or company property, or used force to prevent other employees from crossing a picket line.

8.5.4 HANDLING A WILDCAT STRIKE

8.5.4.1 What to Do Before and During a Wildcat Strike

There is a major difference between a wildcat strike and one that is authorized by the union to enforce demands after a collective bargaining agreement expires. The wildcat is in violation of the no-strike clause of an existing labor contract. MANAGEMENT CAN ASSERT CONSIDERABLE PRESSURE, including discipline up to and including discharge, to put an end to this kind of work stoppage.

1. When a supervisor senses that a WILDCAT STRIKE IS IMMINENT (because of employees' grumblings and hesitation in response to an unpopular management decision for example), the supervisor should immediately alert his immediate supervisor and/or superintendent. The manager will often take over direction of the situation. In many cases, it will be desirable to have the supervisor do the talking to the rebellious rank-and-filers whom he knows and with whom he deals every day. The labor consultant or labor attorney should be called immediately and possibly the union should be called immediately either by the company itself, its labor consultant, or labor attorney.
2. The supervisor SHOULD GET HOLD OF THE UNION STEWARD and advise him of his obligation to tell the employees that the strike violates the contract and subjects them to extreme penalties; that the grievance procedure established under the contract must be invoked to settle their gripes.
3. Supervisors should KEEP AN EYE ON LOCKER AREAS, water coolers, and time clocks where wildcatters are apt to congregate at the outset of their stoppage. All-out efforts should be made to snuff out wildcats at the start.
4. If employees have no reasonable excuse for being off the job, a supervisor should ASK THEM TO RETURN TO WORK and use the established grievance procedure. (Should be in employee handbook or posted on bulletin board.)
5. The supervisor should WARN THE EMPLOYEES that refusal to work violates the contract and subjects them to discipline, perhaps discharge. Supervisors should not leave it to the union steward alone to get the employees back to work. However, the supervisor should not guarantee that if the wildcatters go back all will be forgiven. Supervisors SHOULD BE SILENT ON THIS, not committing the employer before getting explicit instructions on this point.
6. Employees should be reminded that they will NOT BE PAID for time not worked.
7. Supervisors should ASK EMPLOYEES WHY THEY HAVE STOPPED WORK, but they should not "negotiate" a settlement of the grievance or discuss it in any way, until the employees are back on the job.
8. Supervisors must be instructed to be sure to observe the efforts made—or not made—by the stewards and other union officials to end the stoppage.
9. Supervisors should TRY TO PREVENT THE STRIKE FROM SPREADING. Discouraging wildcatters from standing around in idle groups may help. Supervisors in adjacent areas should be advised about the situation at once.
10. It is essential for supervisors to record, accurately and with all relevant detail, all of the facts involved in a wildcat. Each supervisor should try to have a witness with him, if possible, to corroborate what was done and said. An accurate written record can help sustain company disciplinary decisions before an arbitrator; buttress a possible damage suit in court. Corroboration adds that much more. It should be impressed on supervisors that they are the management men immediately on the scene. Supervisors, specifically, should observe and record:

A. WHO STOPPED WORK: WHO STOPPED FIRST (AN INDICATOR OF LEADERSHIP OF THE STRIKE)?
B. WHO WERE THE STRIKE LEADERS?
C. DETAILS OF WHAT HAPPENED, WHEN, AND WHERE.

D. DETAILS OF WHAT WAS HEARD AND OBSERVED: WHAT THE SUPERVISOR SAID, AND TO WHOM.

E. WHO THE WITNESSES ARE.

11. Ask the following questions of every wildcatter as he leaves the job:

"WHERE ARE YOU GOING?"

"WHY ARE YOU DOING THIS?"

"WHO TOLD YOU TO TAKE THIS ACTION?"

12. Get pictures of picket line, work stoppage or slow down, and picket signs.

13. Advise local union and international union officers by telephone, confirmed by wire, confirmation of delivery, that illegal stoppage exists and that positive and affirmative action be taken to halt the stoppage. Please be advised that the local and international union will be held responsible for losses incurred should positive and affirmation action not be taken!

14. NEVER AGREE TO NEGOTIATE THE END OF A "WILDCAT." Agree only to discuss an existing grievance after employees are back to work.

15. Different levels of discipline are acceptable according to degrees of guilt; however, all in each level must receive the same discipline.

16. When in doubt as to appropriate discipline to give wildcat striker—suspend the striker pending a review of the facts and circumstances surrounding the work stoppage and final documentation.

8.5.5 TRAINING AND DEVELOPING HUMAN RESOURCES

8.5.5.1 Today's Training Scene

Training and developing employees and managers is becoming a number one priority in today's workplace. The employer no longer has the option to train or not to train—the question simply is how much and when? The mining industry will experience this need as automation increasingly makes an impact in the workplace and new jobs and employees are required. As the mining labor supply gets smaller, it will be necessary to keep and develop the resources that the industry has. In this segment, the topics to be covered are the people to train, when to train, how to train, how to plan for training, and how to evaluate training.

The training field today is getting more sophisticated and professional. There are more training materials of various types available than ever before. There are better equipped trainers. These developments in turn provide more tools to meet individual training needs.

Professional and business conditions have led to the era of the lifelong learner. Workers will have to continue to learn all of their lives or become obsolete. For example, it is said that the productive work life of an engineer after graduating from college is four to six months before he or she has to start learning new things. This lifelong learning has to be deliberate, planned, and continuous.

8.5.5.2 People to Train

The first step in preparing to train people is to have a clear understanding of the people to be trained. The better the understanding, the more effective the training can be planned and conducted. In most cases, training centers on the adult learner. The adult learner is mature and a different breed. He or she is different from the child (for a comparison of how adults and children learn, see Table 8.5.1). One cannot teach adults in the same way that children have been taught in the past.

Table 8.5.1. Comparison of Child and Adult Learning

	Children	Adults
Concept of learner	Dependent	Self-directed
Role of learner	To be built upon	To be used as a resource
Readiness to learn	Curriculum	Develops from own motivation
Orientation	Subject	Life centered
Motivation	External	Internal
Climate	Low trust formal	Trust mutual respect
Planning	By teacher	Mutually
Diagnosis of needs	Teacher	Mutually
Setting of objectives	Teacher	Mutually
Designing learning plans	Teacher	Learning contracts
Activities	Transmittal	Various techniques
Evaluation	By teacher	By learner

Who is the adult learner? The adult learner can be characterized as follows:

- The **adult** has his or her own set of values, needs, beliefs, attitudes, self-concepts, and past experiences that are brought to the training session.
- The **adult** comes to learn what he or she wants.
- The **adult** comes to learn when he or she wants.
- The **adult** chooses what he or she is going to learn.
- The **adult** has his or her goals for life set.
- The **adult** wants his or her needs met in the learning experience.
- The **adult** is somewhat of a neglected learner when treated as a child.
- The **adult** has limited time for learning.

These factors all affect the working place and the training that takes place there. They must be taken into account when seeking to train a person.

The other part of understanding the individual to be trained is looking at the training and what the individual's needs are at present. Training involves either a new employee or a current employee. New employees come to the training eager to learn because it will mean being able to do the job for which they have been hired. It will mean new skills. What do they want to learn? They want to learn about the company, how to do their job, and how the workplace operates.

Present employees come to the training experience either reluctantly or eagerly. They will be reluctant if they cannot see the advantage of the training, or if they have been sent because of something they have done. They will be eager if they can see the need and the advantages of the training.

What do today's employees want to learn? The American Society of Training and Development and the US Department of Labor recently conducted a study of what employees want to learn in the workplace, of the skills they need to do their job (Carnevale et al., 1989). Today's employee wants and needs these key basic skills:

1. *Learning to Learn*—The ability to learn new skills.
2. *Listening*—The ability to really hear what others are saying.

3. *Oral Communication*—The ability to communicate effectively orally.

4. *Problem Solving*—The ability to solve problems on one's own.

5. *Creative Thinking*—The ability to come up with new ideas.

6. *Self-Esteem* —The ability to feel good about one's self.

7. *Goal Setting*—The ability to set realistic goals.

8. *Personal and Career Goals*—The ability to set clear goals.

9. *Interpersonal Skills*—The ability to relate effectively to others.

10. *Team Work*—The ability to work on a team.

11. *Negotiation —The* ability to build consensus.

12. *Organizational Effectiveness*—The ability to understand the direction the organization is headed.

13. *Leadership*—The ability to assume responsibility.

In planning for training, it is important to have a thorough understanding of the person to be trained and his/her needs. One can then plan the training more effectively.

8.5.5.3 When Does Training Occur?

When are the best times to train? Here are some key times when training can be most effective:

1. When an employee is new.
2. When a promotion is coming or given.
3. When jobs are restructured.
4. When there is a slowdown or shutdown.
5. When the quality of the work needs improvement.
6. When it has been planned.

8.5.5.4 Types of Training

Training in the workplace involves several types of training.

1. Cross Training: Cross training is training persons to do a number of jobs within the company so that they can be shifted from job to job as needed. As fewer people are required in the mining industry, this will become more viable.

2. Retraining: Training of present employees for a new job when their jobs have become obsolete or the procedure for doing the job has been changed.

3. Employee Training: Training designed specifically for the employees. This may include technical safety or human relations skills. It may be done on site, or they may be sent to a seminar or workshop. It focuses on what they need to do the job effectively.

4. Required Training: This is training that is mandated by the federal or state government for the mining industry. An example is training on hazardous materials mandated by the Occupational Safety and Health Administration.

5. Technical Training: Training focuses on the technical skills needed to do the job effectively. It is usually very structured and "hands on."

6. Management Training: Training focuses on the key skills that a person needs to manage people effectively in today's workplace.

7. Executive Development: Specific training for executives to learn to develop their skills to work as team and to manage the organization more effectively.

8.5.5.5 Planning for Training

It is extremely important that training be planned. If not, it may not achieve the stated objectives. If a *helter-skelter* approach is used, the training may end up getting no where, and no one will increase his or her skill level.

The first step in planning for training is to do a training needs assessment. The purpose of the assessment is to determine the present and future training needs of the organization. When these data are gathered, a training gap usually emerges; then plans can be made to bridge that gap. The training needs assessment provides a sense of direction; it points the trainer in the right direction.

How is a training needs assessment conducted? The following steps are suggested:

Step 1. Determine the training needs of the organization.

What kinds of training are needed for the organization to operate at top efficiency? Here one needs to look at the organization as a whole and what kinds of management training, employee training, safety training, and technical training are needed at present. In this way, a complete picture is obtained.

Step 2. Assess present training.

Now that the idea is before management, the next step is to look at what is being done at present. What kinds of training are being offered? How is it being done? Who is doing it? The process normally will turn up a gap between the *real* and the *ideal*

Step 3. What is the present skill level of the people?

Then it is time to look at what is available. What training do the people have, and what training do they need to do the job more effectively? Very likely one will again discover the difference between what is available and what is needed.

Step 4. Bringing the data together.

Now it is time to bring all of the data together and discover the training gaps. At this point, there is a comparison of the ideal with the real, what is needed?, what is available?, and what is needed to bridge the gap. Regarding training needs, conclusions can then be drawn.

With the assessment done for the organization, a master training plan can be developed. The elements in the master plan are (1) the training mission statement, (2) training goals, (3) training strategy, (4) training plan, and (5) an implementation plan.

The *training mission* is the purpose or reason for doing the training. It is what needs to be achieved in providing training for human resources. Take this example of a training mission statement:

"The training mission of XYZ Coal Co. is to provide the finest training available for our most valuable assets, our human resources. We will keep these skills up-to-date so that we are able to perform at top efficiency."

Note how this statement sets both tone and direction for the company.

Training goals are developed next. They are taken from the mission statement. They are the specific objectives to strive for in conducting training in an organization. They are the overall guides that will be aimed for in training. Based on the mission statement, the next step is the training goals of XYZ Coal Co., which are

1. To provide the finest training available.
2. To provide training for the most valuable assets, human resources.
3. To keep the employees' skills up-to-date.
4. To train to perform at top efficiency.

Once the training mission and training goals are developed, a *training strategy is needed.* The strategy is how to do it. It is how to carry out the mission and to accomplish the goals. The training strategy of XYZ Coal Co. is

"Our training strategy at XYZ Coal Co. is to provide our training in-house. We will provide all the training needs to do the job effectively. Training will be planned according to the goals of the company and the needs of the individuals. We will seek constantly to bridge our training gap."

The training strategy is followed by a *training plan.* The training plan outlines the kinds of training that will be included in the process of meeting the strategy, the mission, and the goals. It will outline the kinds of seminars that will be held, the kinds of *hands-on* training that will be conducted, and the kind of individual instruction that will take place. This plan will be reviewed annually as the needs change.

The training plan of XYZ Coal Co. consists of the following:

1. Conduct one seminar for managers each month.
2. Develop technical training guides and train supervisors to use them.
3. Develop an orientation program for all new employees.

As can be seen, XYZ Coal Co. has pinpointed specific areas to focus on.

All this leads to implementation. An *implementation plan* is needed of the exact time that the training will be delivered and who will deliver it. In implementing a training plan, it is good to follow it as closely as possible. XYZ Coal Co. also has an implementation plan.

The implementation plan for XYZ Coal Co. contains these elements:

1. January: Management Seminar: Identify elements of employee orientation training program.

2. February: Management Seminar: Begin writing orientation program; assign persons to develop technical manual.

In addition to the organization plan, individual training plans for the persons in the organization may be developed. This allows focusing the individual's training where it will do the most good. It will show that the individual's training needs are taken seriously. The *training gap analysis* is a method of determining the individual's training needs. An example of a training gap analysis is included in Table 8.5.2.

Table 8.5.2. Training Gap Analysis, Employee

Employees' present skill

↓ ↓ ↓

Training gap

↑ ↑ ↑

Skills needed to do job now and in the future

Training gap analysis

The training gap analysis is a tool for planning training and retraining of managers and supervisors. The training gap is the distance between the present level of training and the desired level to do the job effectively now and in the future. The training gap analysis helps you identify the areas in which you as an employer need to provide training to develop and preserve your human resources.

Name ______________________________

Position ______________________________

Step 1. Developing a job description

The job description outlines the duties of the job and the skills necessary to perform it. This is the foundation of the training gap analysis.

Step 2. Identify present skills

What skills does the employee have now that meet the requirements of the job description. Identify technical and human relations. If you do not have one, now is the time to develop one.

Human relations—interpersonal skills

Technical skills—list particular skills for the job

Step 3. Skills needed to do job effectively now

What skills are needed to do the job now. Identify human relations and technical skills. Look at the job description and list in step two.

Human Relations and Management Skills	Technical Skills

Step 4. Identify training gap

What skills does the employee need training to bridge the gap—Compare step two and three.

Step 5. Future skills needs

What skills will employee need in the next five years to stay current.

Step 6. Your plan to bridge the gap

To bridge the gap, the following training will be provided for this person.

A. In-house training

Type	Date	Trainer

B. Seminars away from site

Seminar	Date

C. One-on-one coaching and counseling

Person Providing	Date

D. Reading and self-study

The other part of planning for training is planning to conduct the individual sessions. Here are some suggestions on how to plan a training session:

1. Master all the material in advance.
2. Plan sequence of training. The following guidelines are helpful:
   a. Start the sequence with materials that are familiar to the trainee.
   b. Proceed from simple to complex.
   c. Place easily learned tasks early in the sequence.
   d. Don't overload one learning time, keep it focused.
   e. Provide adequate practice time between sequences.
   f. Put most complex skills late in sequence.
   g. Plan training so one segment builds on another.
3. Plan training schedules—when and how long?
4. Gather all materials.

The last piece of planning for training is the evaluation of training resources. There are many kinds of training resources on the market, and the following criteria are suggestions for evaluating packaged training:

1. Will this training resource help meet the training mission?
2. Will this training resource help meet the training strategy?
3. Are the skills taught applicable to the workplace transferrable from the classroom?
4. Will this material be understandable to all employees?
5. Can this material be used more than once if it is purchased?

Careful evaluation of the material will permit a more effective job to be done in training. Also unusable or unused materials will collect on the shelf.

Next, how does one evaluate trainers? The following questions will help in evaluating a potential trainer:

1. What is the person's training style?

2. What do others say about this trainer (references)?
3. What kinds of evaluations does this person get?
4. Has the person conducted this seminar before?

8.5.5.6 Methods of Training

There are several ways the training can be delivered and take place within the organization. Each has strengths in certain situations. Each has its limitations. The methods of training are as follows:

1. Self-Instruction: Self-instruction occurs when the employee takes the material and goes through it him or herself. Here the worker may take the material while at the workplace and take it home to study on his or her own time. This type of material may be a workbook, video tape series, computer-based training, or an audio tape series. Here the individual takes responsibility for learning the material and may be asked to pass a written or hands-on test when he/she feels confident with it. The advantage of this is that the instruction can be individualized and it can be self-directed. The disadvantage is that the employee does not receive feedback from the instructor as the learning process takes place.

2. Mentor: Here an older or more experienced employee takes the new employee under his/her wing and trains the person to do the job. The advantage of this is that it is “hands-on” training, and the person can learn the job effectively. The caution or disadvantage here is that the training done by the mentor must be carried out in the way that the company wants its people trained. If the training is done wrong, it is worse than no training at all. Be sure to check the mentor out before turning him/her loose.

3. On-the-Job Training: This is hands-on training. The person normally does not go to any formal sessions but does it all by simply working in the actual work situation. The advantage of this is that the person learns by doing. The disadvantage to it is that the person may not know the reason behind what he/she is doing.

4. Seminar: A seminar is a formalized training setting that can take place onsite or offsite. It ranges from one day to several days and may cover a variety of subjects or one subject. The advantage of this type of training is that it can provide more professional instruction. The disadvantage is that it may not cover the material in the way the employer considers useful, and the person may come back to the work situation frustrated because what he/she has learned is not applicable.

8.5.5.7 Proposed Training Method

Now we need to look at a proposed method of training. There are many methods available, but the most effective method of training is behavior modeling.

Before looking at how behavior modeling works, some background would be helpful. Behavior modeling first appeared as a method of instruction in the 1970s. It is based on the work of psychologist Albert Bandura. He concluded that people learn best by imitating other’s behavior (modeling) and receiving an immediate observable reward. B.F. Skinner expanded this into reinforcement theory. Skinner says positive consequences produce positive results, and negative consequences produce negative results. It has been adopted by many corporations today and is the theory behind a great deal of current training material (Zemke, 1978).

Behavior modeling is based on the principle that if a person sees a task being done correctly, and then has it explained as to why it is being done that way, it will transfer to the workplace more effectively. He/she will not only know how to do it but why it must be done a particular way.

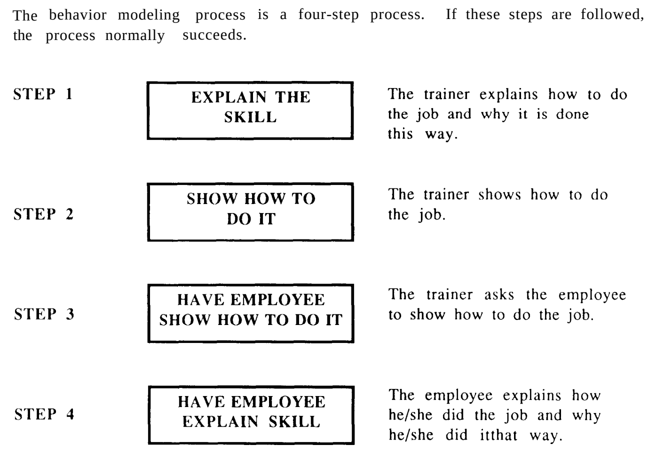


Fig. 8.5.3. Behavior modeling.

Behavior modeling works like this. The trainer does the task and then explains to the person what he/she has done and why he/she has done it that way. Then the trainer asks the employee to perform the task, explaining how to do it and why. The trainer gives feedback as to how the task went. He/she may repeat this process several times until the person has the skill down effectively. This method is highly effective in settings where tasks are taught as *hands on* (Fig. 8.5.3).

Four tools are helpful in supporting behavior modeling. They include coaching, positive reinforcement, counseling, and feedback. (See Figs. 8.5.4 through 8.5.7 for models of coaching, positive reinforcement, counseling, and feedback.)

8.5.5.8 Training Evaluation and Record Keeping

The last part of the training process is evaluation and record keeping. It is very important to keep good training records. The person responsible needs to know what he/she has done and the individuals trained. Record keeping is a simple process that can be done by hand. A sample form appears in Table 8.5.3.

Also there are many computerized training record programs that can be purchased. These make it easier to keep training records by entering each time.

To complete the training process evaluation needs, there are two methods of evaluation that may be used. One method is oral evaluation. Here each of the trainees is interviewed separately; he/she is asked how it went and what the good points and bad points were. Alternatively, this also can be done in a group with all persons sharing their evaluations together.

The second method of evaluation is written. In this method, an individualized form is created or use is made of a prepared one. If the form is especially designed, one should ensure that appropriate data are obtained. It should be custom designed to get the kind of feedback that is helpful both in evaluating the training and in designing future training experiences. A sample evaluation form appears in Table 8.5.4.

The ultimate evaluation of training is successful transfer to the workplace. One can evaluate transfer by observation. The transfer process works as diagrammed as Fig. 8.5.8.

How do skills transfer to the workplace? Skill transfer takes place through observation, imitation, practice, and adaptation. The employee learns the skill by watching someone else perform the task. Then he/she performs the task and practices it. Finally, If an employee has problems during the training process, the coaching technique is a way to plan how to overcome the problem. In the coaching session, the employee and supervisor work together to solve the training problem.

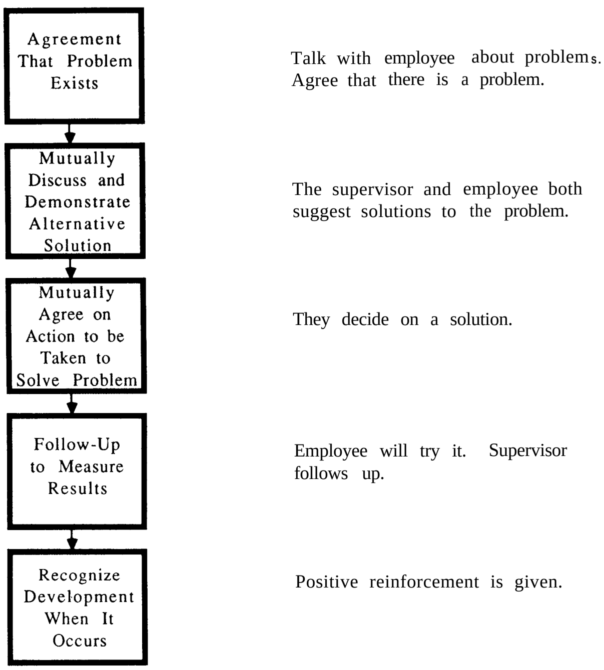


Fig. 8.5.4. Steps in coaching technique.

Positive reinforcement is a powerful tool for management to use during training. It produces the desired behavior, and the employee feels good about what he/she has learned. Positive reinforcement is rewarding the employee with praise when he/she does it right.

STEPS IN POSITIVE REINFORCEMENT - This is how the process works:

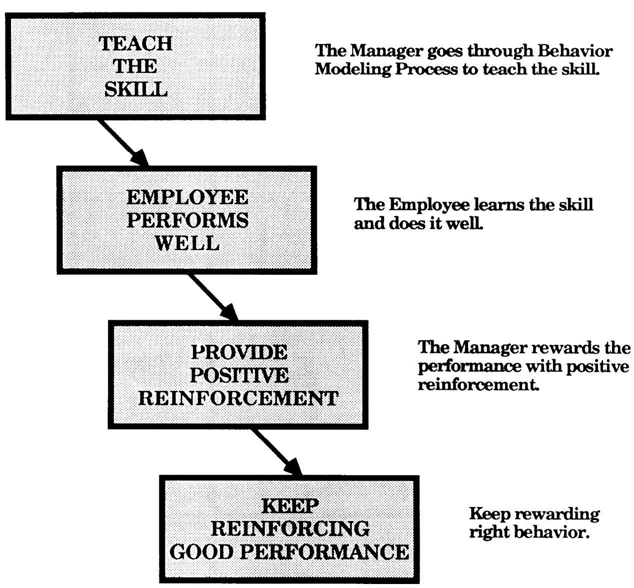


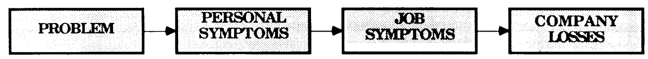
Fig. 8.5.5. Positive reinforcement during training.

During the training process, the employee may experience one or more of the symptoms below. If he/she does, the supervisor should sit down and have a counseling session.

AN EMPLOYEE NEEDS COUNSELING WHEN

1. There is failure/frustration.
2. There is discouragement about progress.
3. There is lack of confidence to do the job.
4. There are personal problems preventing adequate performance.
5. There is a bad attitude toward the job.

COUNSELING CHAIN - If the problem is not treated, the process looks like this:



The following guidelines are offered to assist during the counseling session.

1. State the problem -- Get person to acknowledge it.
2. Listen to the problem.
3. Generate solutions to the problem that are acceptable to both parties
4. Agree on a solution.
5. Implement solution.

Fig. 8.5.6. Counseling.

Feedback is of utmost importance during the training process. The following guidelines are offered for good feedback.

1.	***Give frequent feedback:***	Do it all of the time
2.	***Feedback on the behavior:***	Let the person know his/her behavior is correct.
3.	***Reflect the feelings of the worker:***	Be sensitive to problems with the new skill.
4.	***Perception Checking:***	How does the person feel about the progress being made?
5.	***Give timely feedback:***	Give it when the situation occurs. Don't wait several days.

Fig. 8.5.7. Feedback.

the skill becomes truly his/hers when adaptation is made. This happens when the person makes some adaptations to make it fit him/her. For example, the left-handed person adapts a right-handed procedure to fit his/her need.

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Table 8.5.3. Training Record

Name	Training	Given	Date	Recommendations

OBSERVATION
Watching someone else do it.
The employee watches the task performed through the Behavior Modeling Process.

IMITATION
Does it like the one watched.
The employee does the task in the last steps of the Behavior Modeling.

PRACTICE
Repeats until it becomes habitual.
The employee performs the task until it becomes comfortable on the job.

ADAPTATION
Individual modifies and adapts to suit self and situation
The employee makes it his/her own. He/She adapts the skill to his style of doing things.

Fig. 8.5.8. Transfer of training.

Table 8.5.4. Evaluation of Training

Give us your evaluation of this seminar:

1. What did you like about this seminar?

2. What would you have done different?

3. On a scale of 1–10, with 1 being poor and 10 being excellent, How would you rate the instructor?

4. Additional comments

Please evaluate the seminar on the following aspects:

The Exercises used in the seminar were:	(Poor)	1	2	3	4	5	(Excellent)
The Skills taught in the seminar were:	(Poor)	1	2	3	4	5	(Excellent)
The Instructor was:	(Poor)	1	2	3	4	5	(Excellent)
The Facilities were:	(Poor)	1	2	3	4	5	(Excellent)

Chapter 8.6
MANAGEMENT AND ADMINISTRATION

MILTON H. WARD AND SCOTT G. BRITTON

8.6.1 INTRODUCTION

Management is defined by the well-known business consultant Peter Drucker (1954) as "taking action to make the desired results come to pass." More formal descriptions, however, include "management is the conscious process by which individual and group actions are coordinated to accomplish organizational goals" (Duncan, 1983), or ". . . the attainment of organizational goals in an effective and efficient manner through planning, organizing, leading, and controlling organizational resources" (Daft, 1988).

These ideas indicate a relatively straightforward process, but in reality, managing even a small mining operation may involve a complex series of activities—not necessarily related to the actual mining of ore. These activities involve developing all encompassing plans, controls, and task assignments; allocating scarce resources; managing conflict; and reacting to the unexpected. The following discussion outlines methods for addressing and handling the variety of tasks found in the mining business. However, the reader should keep in mind that, in many instances, no ready-made solution exists. Success requires the manager to utilize guidelines and information contained herein, as well as a willingness to call on his own ingenuity and the experience and talents of his colleagues and employees.

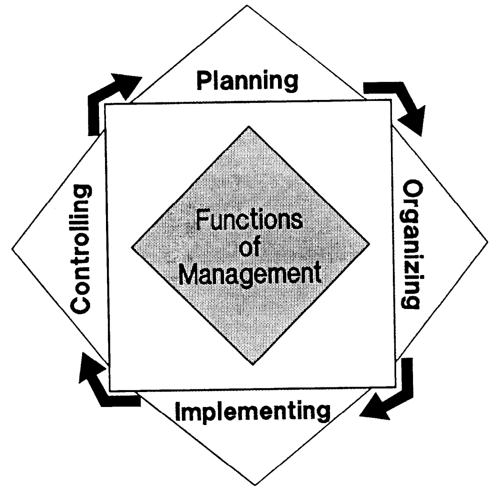

Fig. 8.6.1. The functions of management.

8.6.1.1 Evolution

Management has been practiced since the dawn of civilization, and it can be credited as the catalyst that fostered many of our social and physical accomplishments. Construction of the pyramids of Egypt and the building of our industrial society are often given as examples of well-organized management systems. No doubt, this is correct, but today the demands of society and industry placed on the participants, worker and manager, are changing and tasks are not as repetitive; therefore, even more attention must be given to management and organization.

The Industrial Revolution of the 18th century stimulated the earliest formal interest in management (Flippo, 1978). This interest occurred because better direction was needed as large numbers of workers were brought together to utilize the equipment and machinery of a separate owner, all supposedly working toward a common goal. In time, management began to develop as a recognizable and structured field of study. The analytical approach of scientific management has been credited to an engineer, Frederic Taylor (1911), who practiced from the late 1800s through the early 1900s. Taylor believed waste could be reduced by applying scientific principles to the problems of management. Specifically, his thesis stated that management was a process in which the scientific method of studying operations aided in formulating answers to business problems. His experiments and improvements undertaken for employers supported this position. Subsequently, the idea was espoused by a host of management contributors, including the famous French mining engineer and executive, Henri Fayol (1916). Fayol developed a broader theory of general management that focused on the fundamental functions of planning, organizing, implementing, coordinating, and controlling. Later, in the 1920s and 1930s, a variety of concepts was set forth with emphasis placed on the human factors of management. Another management pioneer, Mayo (1933) a clinical psychologist at Harvard Business School, was given credit for championing this effort that fostered concern for others (the employee) thereby yielding worker effectiveness, reducing labor unrest, and satisfying the human needs of the employee. An amalgamation of portions of these and ideas of other management thinkers make up what we currently consider the functions of management that are shown in Fig. 8.6.1.

8.6.1.2 Management Commitment

An untold number of writings and studies has explored the question of management plans, structure, controls, and other elements of this field, all with the objective of encouraging one to take appropriate action to obtain the desired results. The mining business is replete with risks and unknown conditions; therefore, it is also imperative that management be committed to formulating contingency programs and be prepared to implement these in a prompt manner. If the contingency plan fails, then the quest for a solution, and if necessary formulation of another plan, should be pursued until the desired objective is reached. A commitment to this "success philosophy" should permeate the entire organization—from president to shift boss—all in an effort to achieve the stated goals. Be assured that the mining business will always have problems, and when challenging objectives are attained, management is managing. Success depends on many factors, and these will be reviewed. Key among the initial decisions of the newly created business is the form of the entity.

8.6.2 LEGAL FORM

The typical mining enterprise is now conducted in the corporate form, although many of the earlier companies in our industry transacted business as an individual proprietorship or part-

Table 8.6.1. Comparison of Legal Forms

Pros	Cons
Proprietorship	
Fewer Regulations	Death terminates business
Less reporting	Only owner can act
Quick decision making	Unlimited personal liability
Less risk by others' actions	Equity capital is limited
Owner's total assets recognized	Profits taxed as income
Single taxation	Fewer tax privileges
Partnership	
Fewer regulations than corporations	Unlimited personal liability
Less reporting than corporations	Joint and several liability
Every partner can act	Dissolution by partner's death
Eliminates one tier of taxation	Profits taxed as income
All partners agree to new partner	
Owners' total assets recognized	
Corporation	
Limited personal liability	Owner's personal assets not considered
Unrestricted transfer of stock	No restriction on share ownership
Legal entity that can act	Desires of majority rules
Unlimited owners	Ownership separate from management
Ready access to public trading	Must qualify in state
Survives death of investors	Many governmental restrictions
	More onerous taxation

nership. The most appropriate form for any organization is that which best meets the objectives of the owners, and owners often have a wide disparity of goals. Tax provisions usually influence this choice, but unfortunately, in recent years, US tax laws have changed frequently—and in many instances in a manner that totally circumvents the planned objectives of the organizational form selected. A tabulation of the pros and cons of each form is shown in Table 8.6.1.

8.6.2.1 Individual Proprietorship

As the name *individual proprietorship* indicates, this form is often suitable for the sole proprietor. It is by far the simplest and most common, and it allows the owner to function as owner and worker, as well as an owner that can employ unlimited workers. The primary disadvantage of the proprietorship is that the business terminates on death of the principal, liability is unlimited, and financing and growth can be restricted.

8.6.2.2 Partnership

Formation of a *partnership* takes place when two or more individuals agree to combine their property, labor, and/or capital to function in an activity in which they have a community of interests. Here each partner is to be a principal owner as distinguished from a mere agent, clerk, or creditor, and each is to share in profits and losses because of this relationship. Some would say a partnership is nothing more than a proprietorship with several owners; and while partnerships do have many of the legal characteristics of proprietorships, compared to the corporate form, they eliminate one tier of taxation.

One of the major disadvantages of a partnership is the unlimited personal liability for obligations of the business. This characteristic led to the *limited partnership*, which is a statutory form that allows the investor to limit his liability to the amount invested in the partnership. If this and other protection are to be obtained, scrupulous adherence and compliance with statutes is a necessity. There must be one partner, the general partner, who accepts the risks of the enterprise in this situation, and the limited partners are passive participants who should not exercise control. Another attractive feature of the limited partnership is the assignability of the interest of the limited partners without dissolution of the entity. Use of this form of business has increased in recent years, primarily because the limited partnership eliminates one layer of taxation, and it can be marketed on a public exchange, traded, and operated quite similarly to a corporation.

8.6.2.3 Corporation

As noted earlier, today the *corporate form* is most common for larger businesses and has many attractive features, generally including limited liability of shareholders, unrestricted transferability of ownership by sale of stock certificates, and an unlimited number of owners. As with other forms, often the advantages of a particular entity lead to disadvantages. Among the drawbacks of a corporation are limited credit available (only that justified by the corporation's assets—not total assets of the investors), and taxation requirements are more onerous—especially the double taxation that requires payment of taxes by the entity and then additional payment by the individual investor.

A corporation is most suitable for the large and rapidly growing enterprise, and this is currently the most common form for mining companies. As noted earlier, tax advantages have fostered a revival of partnerships in recent years, but this form remains secondary to the corporate mode. These three entity structures (sole proprietorships, partnerships, and corporations) are the basic forms, although variations of these can be designed to fit the needs of a particular investor or situation. To be most effective, these needs should be identified in the planning stage of the business.

8.6.2.4 Formation Mechanics

The mechanics for establishing the business entity are influenced by the form (proprietorship, partnership, or corporation) and domicile of the enterprise. Regulations are set forth by local, state, and federal governments, and provisions of all these bodies must be followed. Here also, requirements are affected by the type or entity, whether the enterprise is marketed publicly or privately, and how widely traded.

The pros and cons of proprietorships have been listed, and while simplicity of formation and ease of operation are advantages, this structure is rarely selected for a growing business. Exposure to liability, restrictions to funding, and limited life are usually justifications for rejecting. Partnerships may be less onerous and can fill special needs, but since they are often designed to meet a particular situation, they too are infrequently selected for the growth-minded company.

As noted, the corporate form is the most common structure chosen in the mining business, and the manner in which this entity is legally established is well defined. A corporation is created through the joint act of the state and individuals whereby the government grants the privilege of incorporation. The legal concept of a corporation consists in the offer and acceptance of a charter, which is a written instrument that sets forth the terms

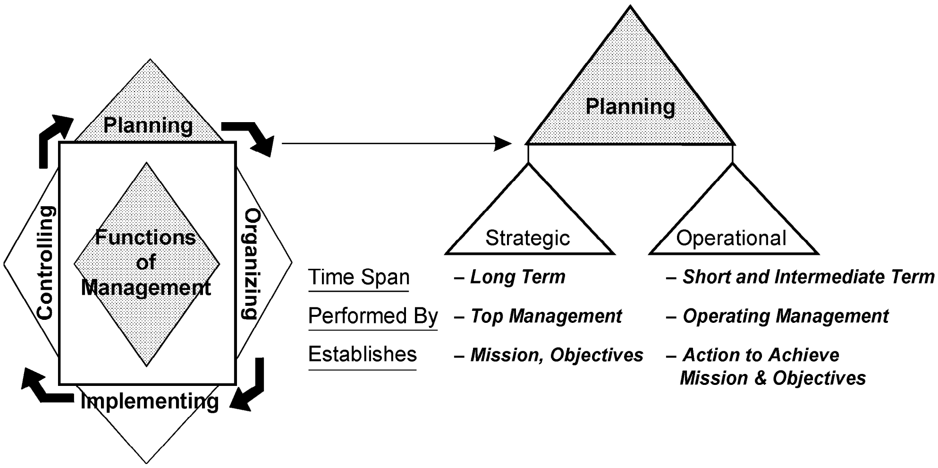

Fig. 8.6.2. Planning—key function of management.

and conditions upon which the stock permits one to exercise the franchise created in the charter. Briefly stated, the steps of incorporation are:

1. Principals arrive at an agreement as to nature and scope of business and broad outline of capitalization and distribution of securities.
2. Select and clear corporate name.
3. Select corporate directors and officers.
4. Describe role of directors and intent with regard to meetings.
5. Note amount and par value of stock.
6. Select the resident agent.
7. Submit the appropriate information from the foregoing and file for a certificate of incorporation with state (secretary of state) and local offices (county clerk of the county wherein the corporation will have its principal office).
8. A few states require a preliminary agreement of association.
9. Prepare bylaws.
10. Obtain any necessary authority for issuance of stock and/or other securities under local blue-sky law.
11. Obtain from law stationer, corporate seal, stock ledger, minute book, and stock certificates.
12. Hold organization meetings.

If public trading of the stock is planned, there will be additional filing requirements with regard to listing on a public stock exchange and a host of related issues. While it is relatively simple to form a corporation by completing standard forms, if the enterprise is to be sold to the public, the complexity increases, and the advice of a competent attorney early in the formation stages can be most valuable.

8.6.3 PLANNING

Planning is the process by which profitable growth is sought and attained in a changing and uncertain world. As shown in Fig. 8.6.2, planning is divided into two major segments, often classified as strategic or operational: *strategic planning* is the broadest, highest level, and longest term effort concerned with defining the objectives of the firm—where the firm wants to go and how it will get there. The other segment, *operational planning,* often relates to the activities of the organization that concern the near term, but its primary focus is controlling the actions to achieve the stated objectives.

Planning may also be further described as long-range, intermediate-range and short-range. The distinctions result from the time frame covered, individuals involved, and activities incorporated in each type. Needless to say, these many classifications of planning may be confusing, because they often overlap, and definitions may vary from company to company.

8.6.3.1 Strategic Planning

Strategic planning involves an assessment of strengths and weaknesses that must be undertaken in order to direct future action. A review of past activities and experience should determine the following:

1. Can management operate in foreign environments?
2. Is the company to be an explorer, developer, and/or operator?
3. Is the enterprise knowledgeable in transportation and marketing?
4. Does it have in-house financial and legal personnel?
5. Is it generating positive cash flow?
6. How large or limited is the borrowing capacity?
7. A host of other key factors.

If the potential business of the company requires new skills, then this inadequacy should be recognized and action plans (called appraisals) formulated to eliminate or minimize these weaknesses. Such appraisals can significantly alter the direction of an enterprise and are the key determinants of restructuring programs implemented by a major segment of American business in recent years. Such reviews have suggested divestment of ill-fitting businesses and have guided many companies back to the core business from which they started.

Appraisal of markets and products starts with the customer. During a strategic planning session, typical questions posed include:

1. Who is the customer, and what are his needs?
2. Where are the customers located, and what type products are or will be in demand?
3. How much of the market do we control, and how can we capture more? Is the demand for our commodity or product expected to change significantly?
4. If markets change, will this influence our divestment or acquisition policies?

Assessment of these and other key factors make up the mission, objectives, and strategies of the mining enterprise and are a part of the total strategic planning process and action flow shown in Fig. 8.6.3.

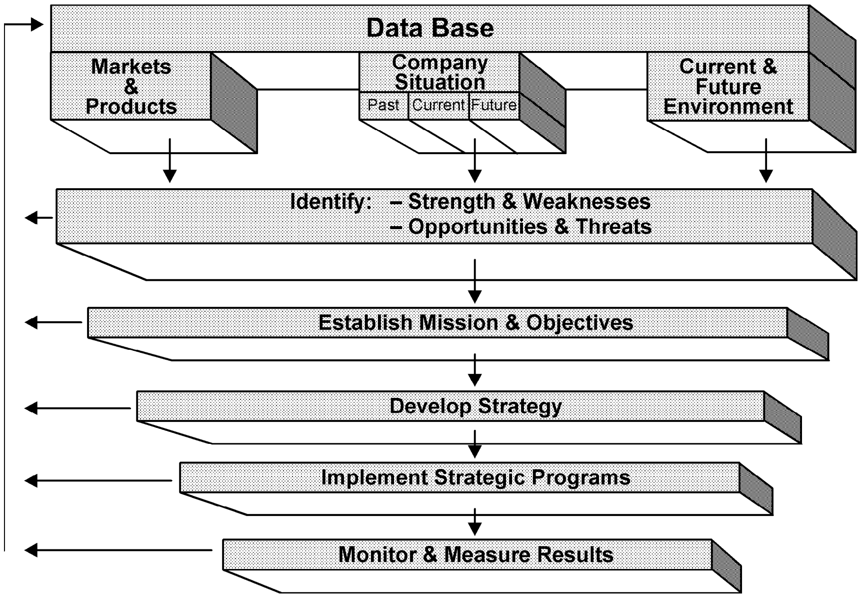

Fig. 8.6.3. Strategic planning—action flow.

8.6.3.2 Mission

The efforts of any organization, be it military, religious, or business, can be better marshaled and directed if they are committed to a guiding principle known as the *mission.* The company's mission statement articulates this principle and outlines the nature of the enterprise's activities and businesses that it will pursue. The mission or justification for the organization's existence should be shared with those involved in attaining the desired results. Someone, ideally top management, must ask and answer: what is our business, and what should it be? Typically, this utilizes steps in the foregoing assessment relative to where the company is, the status of its markets and products, and an appraisal of the firm's strengths and weaknesses.

Many companies prefer not to state their mission precisely, but one can be assured that the goal exists—whether noted in public documents or only etched in the mind of the chief executive officer (CEO).

Cyprus Minerals Co.'s goal that was noted in a recent annual report (Anon., 1989) may serve as an example of the mission statement:

"Cyprus Minerals' goal is to be a premier mining and minerals company. Cyprus aims to profitably grow internally and through acquisitions to increase shareholder value. Cyprus seeks to be a low-cost producer, a technologically superior mining and processing company, a reliable customer-oriented supplier, an effective marketer of quality mineral products and a safe and fair employer. "

Fulfilling the mission of the mining company requires continual success in many areas—profits, debt repayment, discovery of ore bodies, and exploitation of reserves—all with a regard for the constituency of the company. These achievements do not come easily, but they must be pursued and assessment must utilize meaningful measuring sticks.

8.6.3.3 Objectives

In the parlance of strategic planning, objectives are the longer-term goals of the enterprise. Webster defines *objectives* as "something toward which effort is directed, that is, an aim or end of action." This is how the term is used in this writing. Specifically, objectives are the longer-term aims of the natural resource company. Typically, such objectives last for years and relate to the broad issues of growth, profitability, stability, and image—all factors that the company seeks in order to fulfill its mission.

As noted, objectives should be written and distributed to members of the management team to aid in decision making and give direction to operations of the firm. Usually financial objectives speak to the desired growth rate, dividends paid, debt level, and return on investment. Each major segment of the company should formulate its objectives and run its business accordingly. As an example, exploration objectives over the next five years could include maintaining a portfolio of ten new prospects, four advanced-stage prospects, and ongoing definition drilling on two projects—all with the objective of finding an additional gold ore body within a three- to five-year span. Specific objectives of different departments should be tailored to those activities in which their performance and results significantly affect the prosperity of the business. This requires a determination of what is to be measured and what is the yardstick for measuring.

Objectives may be identified and formulated by middle- and senior-level management, but since these guide key action of the enterprise, top management must participate. The CEO or president must not only review and give approval but should also be the person to communicate objectives to all involved. Top management's task is to solicit support of all today for action that will influence results tomorrow.

8.6.3.4 Strategy

Company strategies are made up of action plans and programs necessary to achieve the overall important objectives of the enterprise (Steiner, 1969). *Strategy* is often simply defined as "the action taken to achieve the company's purposes" (Cannon, 1968). Frequently the words strategy and tactics are used synonymously, but in other instances, *tactics* are the more detailed actions taken to attain the strategy. Here we speak only of strategy, and it is indeed action- and results-oriented.

Development and implementation of strategy require identifying the goals of the organization, the activities to be considered, and selecting the most likely action that will yield the expected results. Additionally, responsibilities must be assigned, an allocation of resources determined, and a schedule for completion established. The entire program should be monitored and the results measured by predetermined guidelines.

As an example, consider strategy development to achieve the exploration objectives. As mentioned previously, a company must weigh a number of factors: whether the area of exploration offers the possibility of being a district, region, or country, whether a full geological/exploration staff is to be employed or a portion of the work handled by consultants, and whether drilling is to be undertaken by the company or by contractors. After review, the preferred strategy is selected, responsibility assigned, and work commenced. In many instances, decision making is delayed while newly identified objectives are investigated. In this instance, the review might suggest joint venturing with an exploration company that is seeking a partner. Further study could lead to selection of a totally different option that requires less time and is more cost effective.

8.6.3.4 Operational Planning

Operational planning, as noted earlier, is one of the key functions of management for directing and controlling activities so that the objectives of the enterprise can be attained. While the time frame of action can be long or moderate, most actions involve short-term or in many instances day-to-day tasks. Often unit or departmental programs are classified as operational planning, and typically annual budgets and rolling forecasts may fit this classification. Again the key factor that makes operational planning different from strategic planning is that operational planning is involved with the tasks on a day-to-day basis that are necessary to influence the company's long-term objectives.

8.6.3.5 Long-range Plans

Long-range plans can set the direction of the enterprise and may be analogous to strategic planning. These plans focus on the mission, objectives, and strategies of the company, but also influence and guide intermediate- and short-range operational planning (Steiner, 1963). They are developed, monitored, and, when required, altered by top management. However, since they direct major investments and action, all for an extended period, only infrequent changes are made to these plans. As an example, these can focus on the commodity to be produced, and whether the company will be a miner, processor, and/or smelter of the commodity. They also consider where the entity will operate and with whom. Additionally, long-range plans may also be quite operational in nature. Mine development plans of major ore bodies often consider programs that may not be implemented for decades, and similarly, it is not unusual for financial schemes to extend for at least a decade. The point is that long-range plans, as the words indicate, look to the long-term, regardless of whether the plans are strategic or operational.

8.6.3.6 Intermediate-range Plans

This type of planning provides the means for achieving the mission and long-term goals. The plans, depending on the company's definition of short-term planning, usually have an outlook of two to five years. In the mining business, the definition of *intermediate-range* may cover that time period required to develop and bring into production a deposit that has recently been discovered. Here the term intermediate will depend on the size of deposit, method of mining (underground or surface), degree of processing, and other key parameters of the deposit. If the mine is already in operation, this planning may focus on that period required to plan, develop, and bring into production an expansion of existing operations. At any rate, such planning is often viewed as that ranging from two to five years.

8.6.3.7 Short-range Plans

Short-range plans cover a span of one year or less. Ideally, these would be geared to long-term and intermediate-term programs as well as the overall strategic plan. Here investment funds, revenues, production, and activities can be budgeted with a more reliable degree of accuracy and can be measured on quarterly or more frequent bases.

Summarizing the planning function, as in describing its multifaceted types, objectives, and methods, is no mean task. Definitions are often confusing with several terms used for identical activities. This is because objectives, goals, and tasks may be words that mean the same thing in one organization or writing, while totally different in another organization or writing. The preceding description should clarify the verbage, but more importantly it should aid in leading the organization toward its long-term objectives. This is the purpose of planning—not only to establish the plan, but to assign responsibility and cause the appropriate action to be taken. The best designed plan will be ineffective if it does not generate action and the desired results. Numerous methods of motivating the staff and management to achieve such results are utilized, including the commonly applied practice of management by objectives, an approach that will be described later in this writing.

8.6.4 ORGANIZATIONAL STRUCTURE

8.6.4.1 Method of Design

Design of the organizational structure should focus on enterprise objectives and how the structure can best assist in achieving these objectives. The mining company must be concerned with minimizing capital requirements, but also effectively finding the ore body and efficiently mining, processing, and transporting the mineral product to the customer. How best can this be done? Many organizational design factors come into play—including size of entity, functions required, span of control, reporting relationship, and minimizing duplication while maximizing communication and flexibility. The broader question of the type of structure must be addressed. Which of the forms in Fig. 8.6.4. should be followed?

Should we use functional, divisional, matrix, or a combination of these. Within this issue is the question of how much staff and the relationship of staff to line. Additionally, top management must decide whether the available personnel and the nature of the business suggests a heavy emphasis to centralization or decentralization. This array of questions, large number of issues, and new terms indicates that structure design may be a difficult and complex task, and it may. But paramount to good design is a clear understanding of the strategy and tasks to be achieved and then designing a structure to accomplish these tasks. Chandler (1962), an expert in organizational design, strongly advocates "strategy before structure." Clearly, it pays to know what you want to do, before you start doing it. Success also depends on getting the proper person in the proper position. Many would say that after design, the requirements for each position should be established, and if qualified personnel are not available, they should be recruited and hired. In reality, this is not necessarily the practice. Often such positions are assigned to whomever is available within the organization. While this practice can be criticized, to some extent it has merit—especially if the one assigned is underutilized in his current role; has the proper attitude, desire, basic skills; and is willing to be trained.

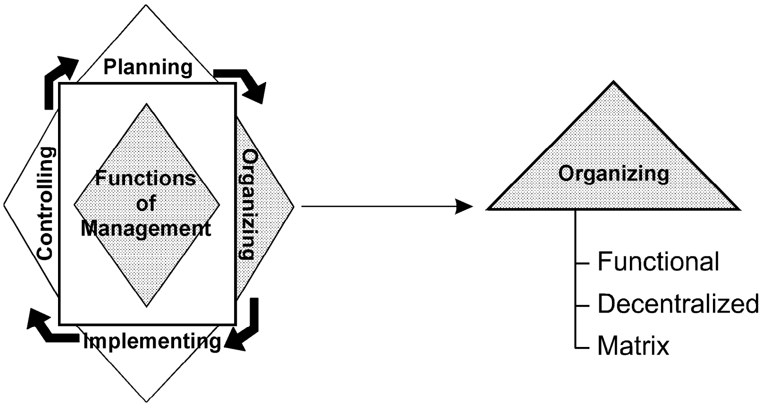

Fig. 8.6.4. Organizing—key function of management.

The key to organizational design is creativity with flexibility. Flexibility is important not only for maximizing performance but also so that one will feel free to redesign organizational structure as frequently as required. Proper design is an organic process in which organizational goals and needs change and management must change the structure to meet the new demands of the enterprise.

8.6.4.2 Centralization - Decentralization

Whether an enterprise is organized on a centralized or decentralized basis depends on where most decisions are made. In centralized organizations, the key decision making takes place where top management is located. Many would say centralization weakens middle management and restricts a manager's development, but others would note that it most effectively conveys the true philosophy and intent of top management. Critics of centralization believe that decision making can be learned, and if one (a younger inexperienced manager) is not exposed to or allowed to make decisions, then his experience will be limited. Decentralization usually moves the administrative and support systems from headquarters to regions, districts, or branches. Decentralization of most activities is almost a necessity for large foreign operations and is frequently used in the larger domestic company. Often, even when decentralization is followed, the policy-making, planning, and financing activities remain with headquarters.

Management's preference for centralization or decentralization changes from time to time and from one CEO to another. Several decades ago, there was a strong shift to decentralization in American industry. This was then followed by a shift back to centralization, with creation of staff specialists and control from headquarters. Recently, hard times in the natural-resource industry have fostered a retrenchment effort with trimming of central staffs, and lean simplified organizations are back in vogue. Proponents of decentralization believe that its advantages far outweigh its shortcomings and that it serves industry best under any condition. Its advantages are thought to

1. Allow daily decision making at the lowest level and thereby aid in development of personnel.
2. Raise morale to a peak if the manager is part of the decision-making process.
3. Enable managers at the work location, who are supposedly best qualified, to make decisions and take action.

8.6.4.3 Matrix

Structures that follow the *matrix mode* are often thought of as temporary because they are frequently created to solve a particular problem. Commonly, the use involves technical organizations in which specialists (mine development, mining engineers, mechanical engineers, geologists, etc.) are brought together to complete a particular project (e.g., an underground machine shop). The organizational structure for this special project team in a smaller company could be as shown in Fig. 8.6.5.

Here the project manager is responsible for the overall program. He/she is assigned personnel from other functional departments to complete this special task, and when the job is finished the personnel return to their former duties, or move on to another project. The next project may have a totally different group of specialists and a new project manager.

Some organizations use the matrix system as a permanent structure with the project manager giving full-time effort to a continuing variety of projects. As an example, this position (project manager) could be responsible for corporate growth and his/her teams made up of personnel from different departments depending on the activity under consideration. Here the project manager could be responsible for several growth projects at any time, with teams dismantled as projects are completed.

The matrix structure allows special projects to utilize the best personnel available for the tasks at hand; it minimizes the number of specialists required in the company, puts them to maximum use, and allows jobs that are often deferred or set aside by the busy functional manager to be completed by the special project manager. This practice, however, is not without problems. Such planning sets the stage for conflict between the functional and project managers since the functional officer may feel that his authority is being preempted. Similarly, matrix planning can create a problem for the specialist since he is often working for two superiors, and there may be questions of loyalty, time allocation, and priority. Also personnel directly involved in the project may be concerned about job security. What job do they return to when the project is completed? Will someone have taken their place in the functional line while they are on assignment? Will their relationship with their functional superior be damaged because of the new job? These questions must be addressed, and senior management should give comfort to all involved so that they will strive eagerly to assist with the project to attain a common goal for the enterprise.

8.6.4.4 Line And Staff

The two structural classifications common to most organizations are line and staff. *Line activities* follow the primary functions of the company and normally include operations, sales, finance, and exploration. *Staff activities* are separate from the primary line, but interface with this group. Staff types are com-

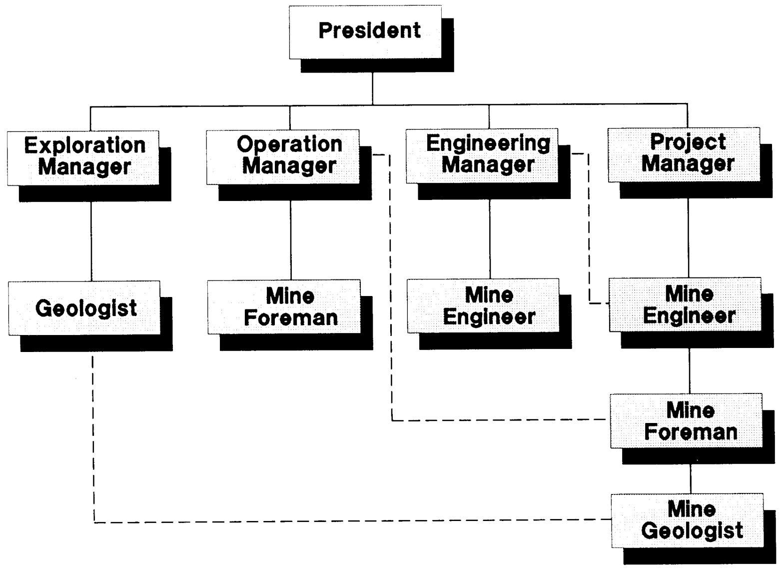

Fig. 8.6.5. Matrix organization.

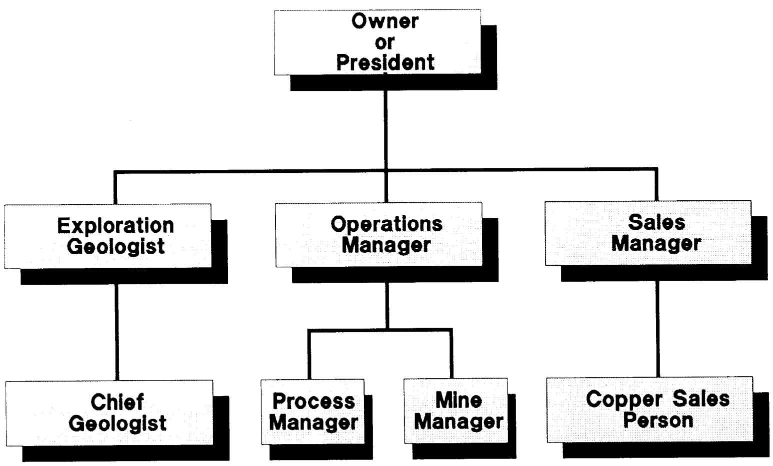

Fig. 8.6.6. Line organization.

monly classified as general and specialized, where general staff serves the needs and duties of a senior officer, as an example, assistant to the president. Specialized staff, on the other hand, commonly concentrate on the more specific part of the business and include cost control engineers and marketing research specialists. The simplified organizational chart in Fig. 8.6.6. illustrates the line tasks. Addition of staff personnel to this structure would show the positions listed in Fig. 8.6.7.

Utilization of staff specialists requires a clear understanding of these duties. Typically, staff personnel lack rights of command, but rather advise and recommend. Cost control engineers collect and analyze data and suggest methods for reducing cost but do not direct action to capture the savings. Similarly, market research staff personnel identify and develop programs that may yield higher revenues but do not implement the new ideas. This practice is often followed in an effort to maintain integration of the line, prevent confusion, and fix responsibility. In a situation such as this, the effective staff person strives to work with and support line personnel rather than take over line duties or impair the activity. As an example, the surveyor picks his time for installing grade or locations points to minimize interruption of production. Similarly, in the spirit of cooperation, the wise grade-control engineer follows the principle of giving prompt advice on dilution to low levels in the line rather than directing the problem to higher levels, that is, to the supervisor of his counterpart .

It is inevitable that from time-to-time conflicts will develop between line and staff. These conflicts occur because objectives of the two groups are perceived as being different. The foreman wants to maximize production while the grade-control engineer's objective is to maximize quality of output. However, conflicts can be reduced by educating all involved with the overall objectives of the mine, and then noting the importance of the line and staff groups working together to achieve joint goals. Occasionally, better appreciation of the tasks of each can be gained by rotating the advisors into line roles and the operating personnel into staff positions.

Clearly, the best solution to conflict is for the participants to solve their own differences, and this approach should be fostered. If a solution cannot be reached, then the supervisors of these personnel should interject themselves into the fray to reach a compromise. Conflicts will develop, but if handled properly and promptly, the impact will be minimal—and, in the process, the enterprise should reap the efficiency and performance that can flow from the advice of the staff specialist.

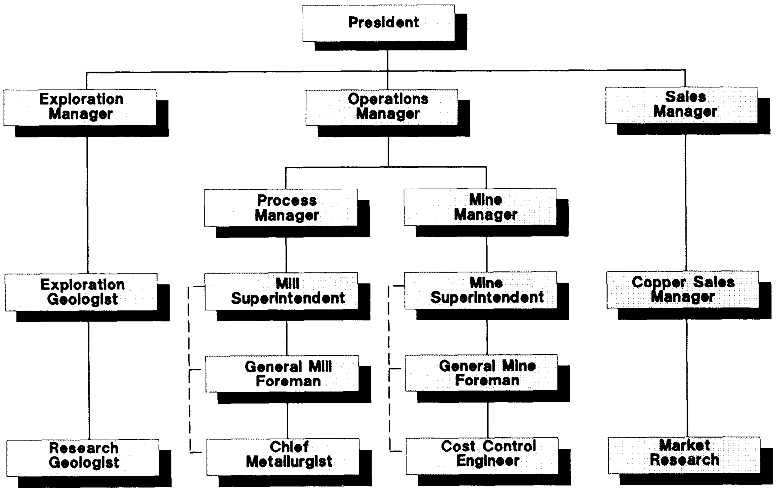

Fig. 8.6.7. Combination of line and staff organization.

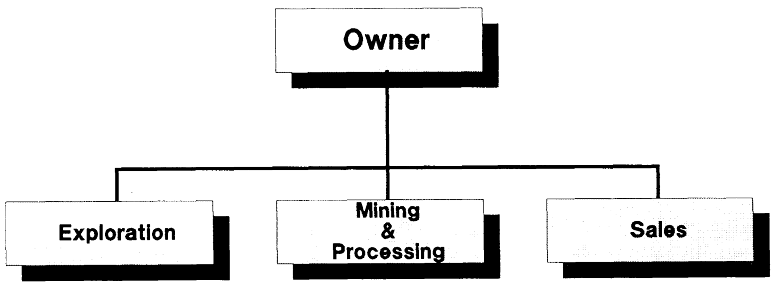

Fig. 8.6.8. Simplified organization.

8.6.4.5 Organizational Size

Normally, the small operation can function satisfactorily without the heavy hierarchy of the typical large institution. The small entity requires no president or CEO or other sophisticated structure. The owner directs some of the functions and actually performs others. Jobs may be combined and often only one person is responsible for financing, accounting, and sales. Initially, part-time personnel may be employed, but eventually, full-time employees are retained. A typical simple structure for the small start-up company is shown in Fig. 8.6.8.

Changes take place as the enterprise grows, and many specialized skills are required. Therefore, there is merit in reviewing the differences in structure and organizational responsibilities associated with size. Governmental agencies and industry associations often precisely describe small, medium, and large entities by the number of employees, dollars of turnover, geographical span, or variety of products. Obviously, the definition of size is a relative statement, and herein the number of personnel involved in the business has been selected as the distinguishing feature, with small being any organization that has less than 100 employees, medium the firm with up to 500 employees, and large the enterprise that contains more than 500 personnel. Often the classification can be better defined by the more sophisticated manner in which the business is organized and managed.

In today's business environment, most moderate-sized companies in the mining industry have chosen to follow an organizational form that includes a presidential, vice presidential, and managerial structure. This organizational form mirrors the large corporation, provides management ego satisfaction, and allows corporate and operating officers to carry the title of their counterparts in larger organizations.

The transition from the small to medium-sized company typically involves a flat management structure that is aligned with functional activities. These might include the structure shown in Fig. 8.6.9. Any complex, hierarchical structure should be avoided since a multi-layered organization can be confusing and costly. Most operating personnel believe that optimal management involves a broad span, and where possible decentralized decision making is preferred. Here "span" refers to "span of control," a term that indicates the number of subordinates reporting to a particular supervisor. As an example, the president in the chart in Fig. 8.6.9 has a span of seven individuals, that is, seven managers performing functions and reporting directly to him. Span limits are often selected arbitrarily. They depend on the superior, the subordinate, and the complexity of the tasks being performed. These like the structure can also change with time.

The middle-sized organization can usually justify management for each functional area, while the large company has a clear-cut assignment of responsibilities, usually with a vice president for each functional area.

The foregoing are hypothetical organizations; however, it is apparent from the actual examples shown in Figs. 8.6.10 and 8.6.11 that the structures described incorporate the key functions. But, as noted at the beginning of this part, organizational structure is an organic process, and while these designs are appropriate for the companies shown at the time of this writing, changes will be made as the organizations change. It has been stated that "The shelf life of any organization's design is probably less than a year" (Sheridan, 1989), and this comment is grounded on the fact that markets, needs, and personnel change—all of which may call for a new structure. Interestingly and supportive

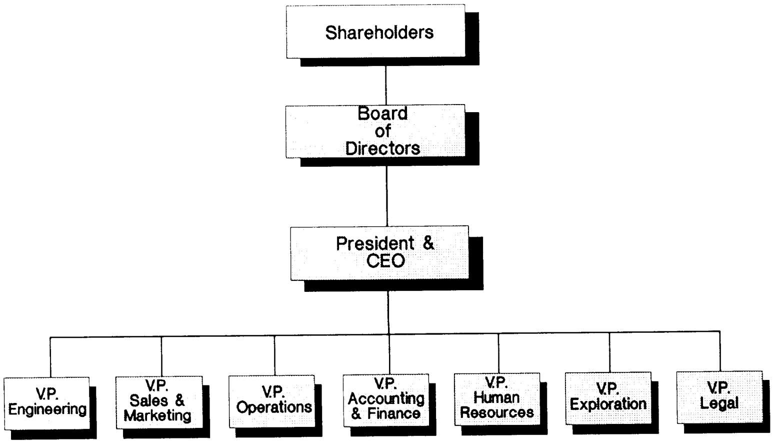

Fig. 8.6.9. Moderate-sized organization.

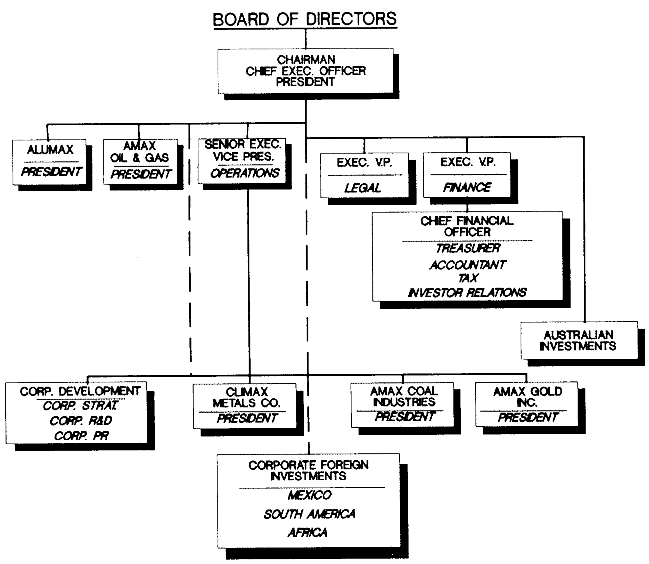

Fig. 8.6.10. Abbreviated organizational structure: Amax Corp.

Fig. 8.6.12. Activities of the organization.

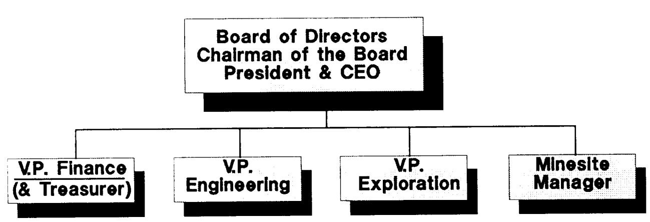

Fig. 8.6.11. Current organization structure: CoCa Mines inc.

of this statement is that the submittor of the organization in Fig. 8.6.11 said that a vice president-operations would probably be added as the company developed additional mines.

8.6.5 FUNCTIONS

8.6.5.1 General

Development and growth of the mining organization often evolve through several stages. Typically, the first stage has a dominant focus on exploration. The initial deposit must be discovered or acquired before self-sustaining funds can be generated, and therefore finding this ore body is of paramount importance. Once in hand, the deposit must be developed, and this requires skills of engineers and operators. Concurrent with, if not before development, come financing, marketing, and many staff and line functions required in the mining business. A brief consolidation of these plus all activities for the mining entity is depicted in Fig. 8.6.12.

8.6.5.2 Exploration

The explorationist must have excellent geological and exploration skills and properly apply these if the mining company is to find ore bodies allowing commencement of operations. Unfortunately, successful exploration requires not only technical skills and good judgment but persistence, long-term funding, and a modicum of luck. The search for a viable deposit is the riskiest and most uncertain challenge facing company management. Statistics show that the odds of success are overwhelmingly negative, with reports indicating that only one out of every 1500 prospects will make a mine. Therefore, the key actors in this facet of the business must have access to and the strong support of top management. Ideally, the person responsible for exploration reports directly to the chief executive officer and works in concert with him. It is imperative that there be agreement about the target commodity, geographical area of interest, budget level, and manner in which the exploration program is to be pursued. For further discussion of exploration management, see Chapter 4.6.

8.6.5.3 Engineering

This facet of the organization typically includes surveying, structural, mechanical, and electrical engineering; mine planning; cost estimating; and other related engineering tasks. Larger companies will have full-time specialists in these areas, while smaller mines will contract or retain certain skills on an as-needed basis. Usually, mine planning and engineering include full-time personnel since this group is involved in all stages of the operation, such as ore reserve computation, predevelopment, development, and operational planning, as well as grade control and production statistics. Selection of the mining method is a critical decision, and where possible, the choice should be designed with flexibility in mind. Unfortunately, ground conditions, water flow, and ore deposition cannot be predicted with precision. Former President Herbert Hoover (1909), a renowned mining engineer, described the situation well:

"The mining engineer's works, on the other hand, depend at all times on many elements which, from the nature of things, must remain unknown. No mine is laid bare to study and resolve in advance. We have to deal with conditions buried in the earth. Especially in metal mines we cannot know, when our works are initiated, what the size, mineralization, or surroundings of the ore bodies will be. We must plunge into them and learn—and repent."

In his excellent book on mining management, Sloan (1983) supports this theme and makes the point that in many areas, mining and mining management are different from that of other industries. One should therefore recognize that mining management systems as well as the work of the mining engineer need to be tailored to our industry.

8.6.5.4 Operations

This usually contains the largest group of workers and can encompass predevelopment, construction, mining, milling, and smelting. Obviously, the makeup of the organization depends on the size of the company and its span of activities. Even the smallest operation commonly has a mining and milling/processing department. These managers may handle their own engineering, construction, and production functions, or they may be limited to only production responsibilities. Here the term mining describes the tasks of development, which includes prestripping (if an open pit) or excavating access and mine openings used in extraction (if an underground mine), and the drilling, blasting, loading, and transporting of ore. Milling or ore dressing describes the process of crushing, grinding, and treatment of the ore such that its valuable constituents are concentrated in a more valuable salable form. Smelting and refining are further stages of processing that change the form of the product, that is, copper concentrates to copper metal.

The operations manager, superintendent, and front-line supervisor must not only have technical expertise but should also be schooled in human relation skills, management techniques, and company policy. Production output and efficiency are influenced by the layout and choice of equipment, but equally by marshaling and motivating the work force. Training incentives and other methods for obtaining commitment of the worker have been emphasized in recent years with gratifying results.

8.6.5.5 Taxes

As noted at the beginning of this chapter, taxes play an important role in the form of the entity, and they also influence the manner in which the business is managed (e.g., see Chapter 2.4). When a mine is determined operational, its method of depreciation and amortization, its depletion rate, and whether costs are expensed or capitalized all affect the amount and timing of profitability. Taxes, as mine development and operations, will be greatly influenced by preplanning.

Many small firms retain consultants to advise, organize, and administer their taxes, and even the largest fully staffed companies frequently call on experts to advise and assist in special tax situations.

8.6.5.6 Marketing, Sales, and Transportation

The purpose of any business is to satisfy the customer since the customer's satisfaction determines whether the enterprise will continue to exist. Marketing skills are necessary for mining companies because competition and customer demands require both an anticipatory and reactive supplier. Therefore, this task encompasses sales, customer research, product planning and diversification, advertising, warehousing, and transportation—all to capture and hold the customer, and thereby justify production. Senior management must be sensitive to these needs and espouse a philosophy tailored to meet the customer's requirements.

Marketing can be a staff or line function, or a combination of the two, and there are a number of successful companies that follow each approach. Traditionally, in the mining industry, this function has not been given the attention and support of top management that it deserves. This attitude is changing as the marketplace becomes a world arena and competition intensifies. Cycles of our business, especially the lengthy down-portion of the cycle, have caused management to attempt to differentiate a portion of the commodities produced in an effort to create a specialty demand that will be countercyclical. Differentiation methods include further processing or modifying the output, or providing cheaper transportation to capture an advantage. The marketing manager is the executive responsible for the marketing task and the individual who should be seeking approaches that will lower costs or make the mine's output more attractive to the customer. It has been noted that mining industry products often are the most important and costly raw material of a manufacturing company, and the customer's chief executive officer may take direct charge of purchasing certain high-cost raw materials (Boyd, 1973). This often urges the top officer of the mining enterprise to deal directly with and indeed build a business and social relationship with the customer. Such practice may be a necessity for the new miner attempting to break into a market. A major effort to gain market share is often pursued at this stage—including granting of discounts on materials sold. In mining, the direct and administrative expenses of moving supplies to, and products from, the mine are one of the major cost items. It is therefore necessary to establish a traffic person or group responsible for finding the lowest-cost method for transporting and receiving incoming supplies and raw materials and shipping the product produced. This requires a professional manager experienced and capable of dealing with transportation companies.

Most commodities and mining products now compete in the world market, and low-cost transportation can be the key to profitability. Experienced assistance is a requisite for this task, especially for low-value, high-bulk minerals and products. Its importance is apparent when one notes that for some materials, freight costs to move the material exceed the value at the mine site. Therefore, swaps of ores, concentrates, and metals are practiced, such that a competitor may deliver for the company in one market while the company may deliver for the competitor in another market—all which can significantly reduce costs. Similar cooperation in using transportation systems jointly, whereby a back haul is identified to eliminate the return of empty vessels can also reduce transportation costs.

8.6.5.7 Human Resources

The term *human resource department* is a relatively new one for the mining industry. Over the years, the activities encompassed herein have often been listed as labor relations, employee relations, and personnel functions. Briefly stated, the human resources department is the department that deals with all the people activities and today this can also include public relations. Traditionally, labor relations has been the primary function of this department, especially where large and powerful unions are in place. This can be a very important and difficult task. In the early years of the union movement, as labor fought for its rights, confrontation was inevitable. This often created an uncooperative atmosphere that adversely affected labor-management relations and therefore hampered productivity and efficiency, which in turn resulted in higher production costs. The union's increased demands not only raised costs but occasionally led to strikes. Unions realized that their most powerful weapon for obtaining the ear of management and achieving their demands was the strike. This is indeed a costly activity, and one that management should strive to avoid. Experience has shown that the best method for preventing such confrontations is to treat the employee so well that the need for a union is eliminated. Achieving or maintaining the goal of a nonunion mine requires a sensitive responsive human resource department and supportive top management.

The human resource management structure for all sizes of companies, but especially the moderate- and large-sized entities, requires close attention and assignment of responsibility to operations. The individual selected to handle mine-site activities is often backed up by a coordinating and monitoring staff position at corporate headquarters. Benefits and work rules may be developed and implemented at the job site, but they should be critiqued and guided by the corporate officer to assure appropriateness and elimination of conflict with established company-wide practices.

8.6.5.8 Safety and Environment

Generally, mining operations of even modest size have assigned a person or persons to be responsible for safety and, with the emphasis now placed on the environment, this area is also receiving constant attention. Both sectors merit professionally trained personnel, with the authority to summarily cease an activity or breach of procedures that may endanger life or violate regulatory guidelines. Governmental regulations and their administration have evolved to the point where it is expensive and embarrassing to the company if citations are issued. The responsible enterprise places protection and safety of its personnel first. This practice not only minimizes suffering to the employee and his family, but it is cost effective. Ideally, personnel responsible for these functions should report to top management, not only to assure that authority is given those involved, but also to show the entire organization that top management is committed to safety and protection of its employees and the environment. Historically, the safety engineer has been located at the mine and has reported to mine personnel, with oversight from a safety director at headquarters. Environmental activities, in many operations, if monitored at all, were handled by an occasional visit from a headquarters person. In recent decades, most mining organizations view these tasks so important that such site personnel report through a matrix structure to safety and environmental directors located in the corporate office.

Table 8.6.2. Traits Required of CEOs—Present and Future. Required Knowledge and Skills

	Now, %	Year 2000, %
Strategy formulation	68	78
Human resource management	41	53
International economics & politics	10	19
Science & technology	11	15
Computer literacy	3	7
Marketing & sales	50	48
Negotiation	34	24
Accounting & finance	33	24
Handling media & public speaking	16	13
Production	21	9

Source: Korn, 1989.

8.6.6 TOP MANAGEMENT

8.6.6.1 Chairman of the Board

The person carrying this lofty title is viewed as holding the highest authority in any corporate entity. This is because the board of directors is responsible for hiring and retaining top management. The person that greatly influences board action is its chairman. The chairman is charged with handling board meetings and related activities, and also serves as the board's spokesman. He is selected by the board and represents the board to management, the public, and shareholders. A review of practice shows that while this is an important position, it is not as influential, nor as critical to the success of the company, as the chief executive officer (CEO) (Townsend, 1984; Geneen, 1984). However, frequently the chairman also serves as CEO, and this position then carries maximum clout.

8.6.6.2 Chief Executive Officer

This position description has evolved over time, as have the words that describe the top decision maker. In the proprietorship, "owner" is still used, while in a partnership "managing partner" is quite fitting. Chairman, CEO, president, or manager is the title given the key person in corporations, and today it is not uncommon for any of these terms to be used regardless of the legal structure—whether proprietorship, partnership, or corporation.

The CEO, subject to the board of directors, usually has general control, authority, and responsibility for the corporation. Clearly, this is the most important role in the organization, and therefore it is critical that the best possible person be selected for this job. Here the position description considers not only the typical requirements of managing the corporation but also the special present needs, skills, and vision that will insure future success for the enterprise. Business, financial, and operating talents are necessities, but equally important is the ability to select, train, and motivate the management team and work force. This is a changing role, and Table 8.6.2 contains a tabulation of the knowledge and skills expected for the year 2000 compared with those of today. As business demands increase, more attention will be given to strategy formulation, human resource management, and international economics and politics.

8.6.6.3 President

Frequently the president of the corporation is also its CEO, and it is not unusual for the president to be the CEO and chairman. However, in today's complex business environment, and especially in a larger corporation, there is often both a CEO and a president. Typically, the president performs the functions delegated to him by the CEO, and this usually involves the day-to-day operations of the business. In a situation such as this, commonly the chairman carries the title chairman and chief executive officer, and the president has the title of president and chief operating officer (COO).

8.6.6.4 Board of Directors

The function of directors varies from entity to entity, depending on the needs and expectations of the enterprise (Drucker, 1972). Their duties can include:

1. Selection of top officers and decision makers.
2. Approval of long-range direction of the enterprise.
3. Determining company objectives and philosophy.
4. Fixing responsibility.
5. Determining executive compensation.
6. Approval of annual operating plans.
7. Attention to public affairs.

Generally, these and other duties are classified under three broad headings: legal requirements, assisting management, and acting as trustees for the owners.

Frequently, the new enterprise and especially the small company, because of limited resources, can gain from a board of directors that has specialized skills. Ideally, it would have one director with experience in the following:

1. Banking.
2. Engineering and construction.
3. Finance and accounting.
4. Legal requirements.

8.6.6.5 Shareholders

Shareholders own the company and are represented by the board of directors. Investors have bought company shares with the objective of obtaining profits, and in fact, increasing profits and rate of return on their investment. In earlier years, management's attention to shareholder interest tended to ebb and wane, depending on the tenor of the times. In recent years, because of reporting requirements, the risk of takeovers, and the attention given by the press, there has been a shift back to serving the interest of the shareholder.

Identifying methods for fulfilling the needs of the shareholder is a challenge that alert management will pursue. Earnings per share is commonly the measuring stick for assessing performance, and its movement will influence share prices. Without doubt, management's expression of a desire to enhance shareholder value has merit and can aid in raising share prices, but action and results must follow if credence is given to this position. The key factor is earnings growth, but responses to provide growth in stock value can also include share buybacks, paying higher dividends, recommending stock splits, and other creative mechanisms.

8.6.7 IMPLEMENTING

8.6.7.1 Description

It is worth repeating that the basic and key functions of management are planning, organizing, implementing, and controlling. The activity that is described as *implementing* is often termed directing, leading, or decision making. Any or all descriptions could fit, but "implementing" is selected because the dictionary specifically defines this as "carrying out" or "accomplishing," and that is the step shown in Fig. 8.6.13 that is required in this stage of the management continuum. All the key activities of management are critical, but little ore is going to be hoisted and processed if well-thought-out plans are not transmitted into action, or implemented. This facet of the process is often the most difficult, and therefore it has received great study and attention. Detailed analysis of leadership, motivation, training, and incentive programs are all pursued in an effort to attain the desired results that flow from proper implementation of well-crafted plans. Traditionally, the mining company has directed adequate if not excessive attention to the mining method or processing flowsheet, while sometimes giving insufficient thought to the overall motivation of the worker and individual employee.

Most are familiar with Maslow's hierarchy of needs shown in Fig. 8.6.14, but in earlier years, only thoughtful management (or, ironically, organized labor) acted to provide a response to some of the needs of the worker (Silber, 1974). Today, however, practically every organization realizes that most if not all these considerations should receive attention. Proper action by management will not only improve performance but can eliminate the justification for a workers' union. Holding and motivating employees require programs that speak to Maslow's entire hierarchy, from physiological to self-actualization.

8.6.7.2 Leadership

President Dwight Eisenhower reportedly said that "leadership is the art of getting someone else to do something you want done because he wants to do it." Leadership is indeed often identified as the key factor that stimulates implementation of action programs, and as noted, action is the critical element. *Leadership* by definition is the capacity to direct or influence others. In any organization, mine included, the leader must have the ability to convey the objectives of the company and to inspire employees to commit to high performance. Leaders may use many incentives, ranging from instilling fear of job loss to providing an opportunity for advancement or profit sharing.

All these approaches have been and, in fact, are still used in mining, but experience shows that in today's environment, positive, encouraging, participative efforts yield the most favorable results. The leader sets the tone, but all managers in the organization should be involved. If the enthusiasm and commitment of the top person can be expanded to other members of the company, then success is more likely.

8.6.7.3 Management By Objective

Like many other management practices, *management by objective* (MBO) was outlined by Drucker (1973) as a method of joint (manager-subordinate) goal setting, measurement of results, and motivation. MBO programs vary from company to company, but all urge employees to concentrate on specific objectives while pursuing ongoing activities of the organization. MBO involves agreement between the manager and his subordinate about the task to be accomplished, action to be taken, time

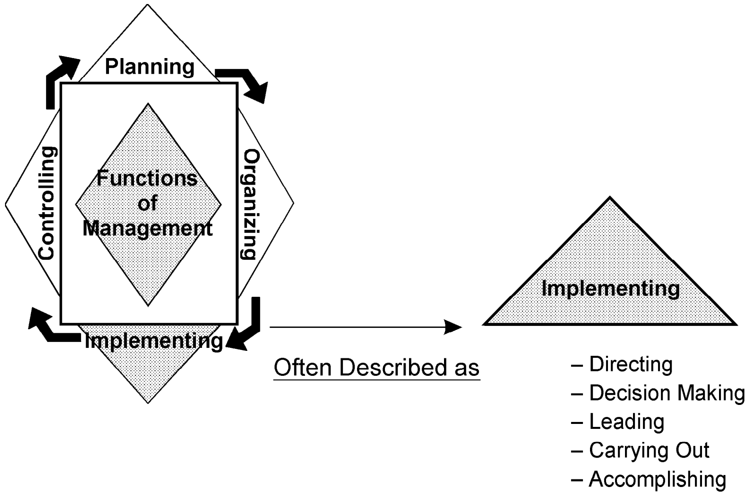

Fig. 8.6.13. Implementing—key function of management.

Fig. 8.6.14. Maslow's hierarchy of needs.

Table 8.6.3. Management By Objectives—Advantages and Problems

Management By Objectives (MBO)	
Advantages	Problems
Emphasizes enterprise's objectives	Developing and maintaining monitoring system
Develops commitment	Reluctance to share authority
Motivates employees	Tendency to focus on mechanics rather than results
Improves quality & output	Time-consuming
Enhances communications	Requires excessive paperwork
Fosters morale	
Aids control	

frame for completion, and method for measuring progress. Such programs have value but require a significant amount of planning, explanation, administration, and commitment. Often this practice, because of its complexity when first implemented, is introduced to one sector or department, such as engineering, and in time is then expanded to other areas such as the mine, mill, and other departments. Typically, the program eventually covers all corporate-wide activities.

MBO is now accepted practice in a large number of North American and foreign operations and, while companies may not have a formalized program, most would recognize both its advantages and associated problems that are shown in Table 8.6.3.

Without doubt, the last requirement in this table, that is, excessive paperwork, is a valid point—MBO does demand an exceptional amount of time, record keeping, and paperwork. Its total benefits, however, by far outweigh the disadvantages of the program. Ideally, it is a system that incorporates and guides individual, department, and division efforts toward the mission of the company, by integrating the objectives and tasks of top, middle, and front-line management. At the minimum, it produces an improvement in the planning and performance of those participating. In an overall sense, MBO can be viewed as an activity that transcends and supports a broader list of the elements of good management, such as planning, organizing, motivating, implementing, controlling, and coordinating.

8.6.7.4 Compensation

Typically, this term, *compensation*, is defined as the benefits provided an employee for services performed. More specifically, it can be described as base pay, variable pay based on individual or group performance, and lastly additional benefits that are tied directly to the job. *Base pay* is influenced by demands of the job and the ability and aptitudes required to perform the tasks assigned. Additionally, competition for employment and location of the work site also affect this amount. Normally, the base pay rate considers all these factors, and job analysis and job evaluation are the formal steps undertaken to numerically quantify the value of the job.

The objectives of variable pay are to increase performance, output, or sales. This form of pay is often termed *bonus* and is based on performance above a fixed level. It is not unusual in underground hard-rock mines for the bonus to range from 10 to 100% or more of base pay, with the higher amounts being awarded to only a limited number or small percentage of the participants.

Additional benefits to that noted are *fringe benefits* which include pensions, insurance, vacations, holiday pay, and other related payments. This form of compensation, similar to the base rate, is often influenced by competition or union practices. Fringe benefit costs vary but can range from 25 to 40% of base pay.

The greatest variance in compensation in the mining industry is in the variable pay or bonus sector of the hourly worker. In some industry branches or types of work (surface, nonmetallics), bonus may not be paid; while in other operations (underground, metallics), the miner may make an excellent bonus. Arguments can be made that all workers should participate, or not participate, in an incentive compensation program, and cases for either situation have been made. Frequently, the location of the work site influences this decision. It is most difficult to refuse to pay a bonus if the local competition has an incentive system. Without doubt, a well-designed bonus program can dramatically improve output and productivity. A difficult problem with such programs is that of establishing a fair assessment of the work required for the pay awarded. This is a challenge, but the incentive value of such a program far outweighs its problems. Studies have shown that the motivational force is greatest if the reward is highly valued, goals can be attained, and one expects the award to be properly allocated. Ideally, the incentive program would be based on the performance of each individual directly involved in a given task and not be dependent on the actions of others. Such a program is often difficult to design, but an effort should be made to minimize the participants grouped in any particular bonus pool. The mining industry is replete with examples of effective, fair, reward systems, but unfortunately, there are many instances of unsatisfactory programs that over or under incent the worker. However, even with such problems, most believe the benefits of such systems are worth pursuing.

The foregoing comments refer primarily to hourly workers, but the same compensation philosophy applies to the staff and executive personnel of the organization. The units of measurement should consider not only specific accomplishments, such as production of the mine or mill, but also overall profitability of the enterprise. The measuring sticks of performance may be different, but the results can be the same—most positive.

Record keeping and paperwork for incentive programs can frequently be time-consuming and complex. Therefore, a constant effort should be made to simplify the program so the participant knows how he is performing and what reward can be expected. Programs of this type are not low cost, but shareholders have consistently supported hourly and executive compensation programs if they are based on performance.

8.6.8 CONTROLLING

8.6.8.1 Controlling

This is often viewed as the last of the four key functions of management, and as reported earlier, this activity follows planning, organizing, and implementing. *Control* is the manner in which an organization influences and monitors activities in an effort to achieve or redirect action to attain planned goals. The control system requires the tasks shown in Fig. 8.6.15.

Performance standards vary from activity to activity and department to department, and are often incorporated in the formal plan or department budget. As an example, if next year's mine plan calls for a production increase from 4000 to 5000 tpd (3600 to 4500 t/day), while at the same time increasing productivity from 8 to 10 tons (7 to 9 t)/employee-shift, these goals should be used as the basis for the standards. If standards are not being achieved as the year progresses, deliberate action must be taken to pull or push the area of shortfall back on schedule. The properly designed plan contains specific measuring sticks or factors that are monitored to determine progress. Daily, weekly, and monthly reports containing the tonnage and grade produced from each stope, section, and mine level, along with the work force involved in such production activities provide the basic performance numbers. These data are recorded, certain statistics computed, and the information reported for comparative purposes, as shown in Table 8.6.4.

Ideally, a simple comparison of actual to expected results will indicate whether corrective action is necessary. If daily mine production is experiencing a shortfall, that is, 3800 tpd (3450 t/day) rather than the budgeted 4500 tpd (4080 t/day), then additional workplaces must be activated. Implementing such changes may appear to be a minor task, but in practice this may be most difficult. Are required resources, extra workplaces, equipment, and manpower available to commence mining in the new production area? Sensitivity to this possible need and adequate preplanning or contingency planning will allow corrective action to be taken in an expeditious manner. This simple example of a control procedure used for monitoring and adjusting to a production shortfall must be developed for the entire operation, and it is performed more effectively if standards are identified in the planning stage.

8.6.8.2 Budgeting

A key element of the control process is the formal budget—either a period or rolling budget. Here, *period budget* describes a process that remains essentially fixed or unchanged for the period under review, possibly a year, while the *rolling budget* is modified on a more frequent basis. Obviously, the rolling forecast represents more current information, and therefore comparisons with actual performance show less variation. Either budget, period or rolling, may utilize a short-term forecast of one year and a mid-term budget of three to five years. If term forecasting is followed, then the one-year plan is often updated at mid-year, and the mid-term (three- to five-year) budget is updated on an annual basis. Rolling forecasts, on the other hand, are normally updated quarterly for one year into the future. Traditionally, most mining companies use term planning and budgeting, but there is merit in the more frequent updating and assessment of forecasts, and recognition of this value is causing increased use of rolling forecasts. The primary disadvantage is that the rolling forecast requires additional analysis, action, and paperwork. However, it should yield a better measuring stick and better performance.

Proper planning requires budgeting of all critical activities including exploration, operations, and sales. These inputs form the foundation of the financial budget, along with project revenues, costs, and profits. Typical performance standards for certain activities of the business are monitored monthly.

8.6.8.3 Establishment of Performance Standards

Performance standards must be defined in a manner that is clear to those involved—conceptually, numerically, and in time. The standard of performance is not articulated only as "raising overall mine production," but more specifically as "raising overall mine production of ore by 25% by the end of next year." Often an even more detailed breakdown of the appropriate factors is

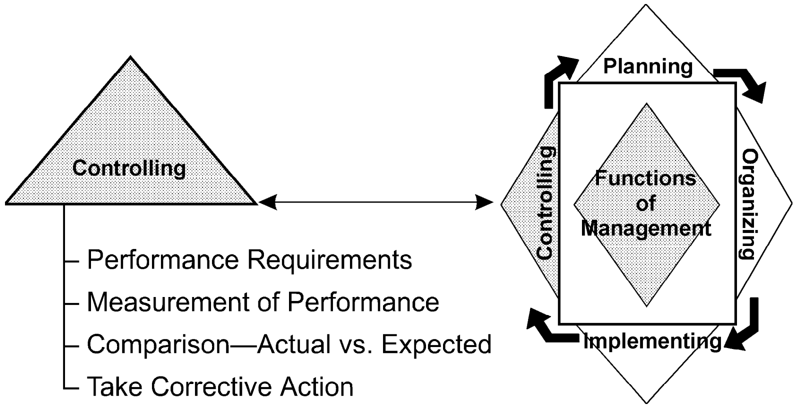

Fig. 8.6.15. Controlling—key function of management.

Table 8.6.4. Control Data—Mill Operations

	Current Mtd	% of Plan	Plan Mtd	Current Ytd	% of Plan	Plan Ytd
Feed dry tons	114,604.000	97.5	117,600.000	713,530.300	94.9	751,800.000
Feed dry tons/day	4,093.000	97.5	4,200.000	3,986.200	94.9	4,200.000
Feed leach ratio	0.010	10.0	0.100	0.010	10.0	0.100
CIP feed ratio	0.215	109.1	0.197	0.198	98.1	0.202
CIP recovery	90.350	99.3	91.000	90.744	99.7	91.000
CIP tails oz/ton	0.019	105.6	0.018	0.019	105.6	0.018
CIP liquor, ppm	0.029	72.5	0.040	0.029	72.5	0.040
Rec. CIP, Au/oz	22,237.650	104.0	21,377.832	128,463.240	91.6	140,216.786
Mill equip. available	95.000	98.5	96.500	95.978	99.5	96.500
Total mill available	94.500	99.5	95.000	94.334	99.3	95.000

established. Mine production may be defined as tons of ore, at or above the required ore cutoff grade delivered to the mill stockpile. The desired production increase of 25% must consider an increase of 25% above a specific level or point in time, say, the average of last year, last quarter, or the last month of last year. These numbers can be quite different depending on whether production is increasing, decreasing or static during the base period.

8.6.8.4 Measurement of Performance

Every key activity in the operation must have *measures of performance* if the activity is to be properly quantified and assessed. Development, construction, mining, and processing are typical functions that are reported in detail on a frequent, often daily, basis. However, other activities such as exploration and research and development are analyzed on a longer-term basis. Generally, the measures for ore processing are relatively standard, focusing on tons of ore milled, percentage recovery, pounds of reagents consumed, and dollars per unit for the cost factors of production. These particular measures are often monitored, not only on a daily basis, but also by shift. Obviously, this reporting requires constant attention and measuring schemes that provided the information desired. However, such data are also required for daily process control, so little additional expense or effort is involved in accumulating this information. These are excellent quantitative measures, and their value will be illustrated in the next section, but control can also be exerted by on-the-spot personal assessment. Personal contact shows an interest in the task, and also allows an immediate assessment so that prompt suggestions or corrective responses can be given if required.

8.6.8.5 Comparison

Actual to expected comparisons serve as the motivator for corrective action, should such action be necessary. Reporting of the comparison between actual and expected results is usually through a variance report. This document frequently contains columnar numbers of actual, planned, and the variance or difference between the two. The report concentrates on the performance results to be controlled including quantity (tons), quality (grade), time (per day or month) and costs (per unit or total). If the variation between actual and expected is significant, an explanation of the cause is justified. Reports, and the appropriate variances, commonly compare the month and quarter with that budgeted, as well as the month and quarter for the previous year. This comparison is helpful in determining differences, trends, and whether immediate action is required, or it indicates whether the situation only requires monitoring for the near term. After studying the variation report, management must determine what action, if any, should be taken.

8.6.8.6 Corrective Action

The primary objective of the control process is to indicate whether corrective action is necessary for a particular program, and if so, whether action must be taken immediately or later. If the degree of variation is just a minor or short-term aberration, no action is required. Experience has shown, however, that if there is a period of significant variation, or a strong negative trend, there can be merit in responding quickly. As an example, if the ore grade is lower than forecast for a given month, then an investigation should be undertaken to determine the cause. One may choose to defer immediate action, assuming that the low grade of one month or one period will be offset by a higher grade the next period. After all, it is the average grade that counts. This assumption may be correct, and the situation may improve. Unfortunately, however, valuable time, production, and profits can be lost if the cause of the low grade is dilution from poor drilling control, caving of waste, or errors of location in the deposit. Prudence would say, identify the cause of the problem as rapidly as possible and formulate a program to get back on budget. Finding the cause is not always easy, and developing a program for correcting the problem may be quite time-consuming and costly. Therefore, it is best to initiate promptly the required changes.

The second phase of corrective action is to determine why the variance developed and what can be done to prevent a recurrence. This review, as well as the search for a solution to get back on schedule, may show that the original plan was faulty. If this is the case, management must decide whether it is better to change the plan or provide additional resources to achieve the original goal. This has been a rather lengthy example of the control cycle, but it illustrates a system that is commonly applied and that most companies believe has merit.

When the company and operations are small and contain few mines and a relatively small number of personnel, the foregoing control system alone may be adequate. But as the enterprise grows, additional mines are developed, and personnel and staff increase, policies and procedures are a necessity. Here policy is defined as a general guideline, allowing one to use discretion in its application. Procedures, on the other hand, tell the user exactly what to do in certain situations. These, plus more sophisticated and more specific control procedures, are utilized and, as might be expected, the complexity of the task may influence the complexity of the system.

8.6.8.7 Communication

Narrowly defined, *communication* is the act of transmitting information, but more broadly described, it is a giving and exchanging of signals or messages. For our purposes, exchanging is emphasized. We not only need to exchange or share information, but we want the message to be understood. The objective is to motivate or influence the attitude of our constituencies, and this can be better accomplished if the intent of the communication is understood. Management deals or makes contact with a variety of individuals and groups, and the methods and tools of communication should be adapted to the particular need or relationship. Management information, as most information, flows downward, upward, and horizontally, and leaders of the organization must develop the channels for establishing this flow. The mix of downward communication is abundant compared with that flowing upward. Normally, the mission, objectives, plans, procedures, practices, and training information are communicated down through the organization. Upward communication, which should be encouraged, is often formalized. Examples include financial and accounting information, complaints and grievances, suggestions, problem, reporting, and exceptions. While upward communication is desired, it often meets with resistance because lower levels of management are reluctant to pass along negative reports, or employees are fearful that their comments will be misinterpreted. Poor morale, efforts to unionize, unsafe working conditions, and discontent can develop in the mine that has faulty communication.

Management must continually foster an environment conducive to good information flow, and should assure lower levels in the organization that their comments will be objectively received and acted on. Horizontal, the third form of communication, is the lateral transmitting of information among peers and departments. This contact is effective in keeping colleagues informed, coordinating task forces, and advising staff departments of the plans and actions of the enterprise. Examples of downward communication in a mining company include approval of the mine production program for the upcoming year, the budget for the period, and more specifically blueprints for developing and producing from a portion of the mine. Some of this information passes through several levels of management in the organization, and when this is the case it is important that the transmitted data and information be clearly presented, and where possible be self-explanatory. This can best be assured if those involved in the proposed program can make inputs into the plan when it is being formulated. Both special discussions held with the user to aid in understanding and personal contact associated with a feedback system (suggestion box, etc.) should yield better results. Personal contact with the worker is a job responsibility and a necessity for mining supervisors, superintendents, and managers. The frequency of contact with the worker varies inversely with the level of the job, but there is value in all, from the shift boss to the CEO, visiting the operation on a regular basis. This attitude and action has coined a new management term: “management by wandering around.” Under this concept, management at all levels talks directly with employees to determine first hand what is going on from the workers’ perspective, and it gives an opportunity for management to explain the organization’s plans and intent. The objective of this practice is to develop a more committed, motivated employee.

Much of the foregoing concentrates on management’s relationship with the employee, and this is appropriate, especially if the company wants to build loyalty and improve productivity. Attention has been directed to the worker because case studies of many companies show that this approach can minimize the involvement of unions, raise morale, and yield outstanding results.

The effect of communication in the directing or implementing phase of management has been reviewed, and this area can be dramatically influenced by communication. However, planning, organizing, and controlling, as well as all management activities, are affected by communications. Strategic plans, budgets, and all reports convey information and are communication devices. These approaches are often supplemented by a formal management information system.

8.6.8.8 Management Information System

The need increases for prompt, quality information as business becomes more complex and competitive. For many years, the *management information system* (MIS) effort focused on dispersing accounting and financial information to senior management. Over time, the system expanded, and today most organizations’ management information is collected and dispersed to different levels of management. The variety and type of data as well as the recipient and user have changed dramatically. Practically all departments can be served by this group. Information dispersed might be data on absenteeism or employee turnover for the human resources department, detailed production information on stopes, levels, and shifts for the mining department, and feet (meters) of hole drilled and costs per foot (meter) for the exploration group. The primary purpose of this information is to allow control and good communications but also to provide data for decision making.

Collection and dispersion of these data can involve the use of existing reports, summarizing of existing information, or generation of new information. Initially, in the mining business, the data were collected by hand and either rearranged or used directly as base data, and then transmitted to the manager. Today computers can store, rearrange, compute additional information, and automatically provide the findings directly to the user. Computer systems allow "real time" capability which enables the user to make requests and then immediately receive a response to his/her queries. As an example, if the quality of coal coming from a given mine is not as expected, the user can ask for a readout of the ash, Btu, and quantity coming from each section. These data can give an idea of the problem area and suggest a correction.

Designing the MIS requires a determination of the data to be used, how these will be collected and stored, the method for converting raw data into useful information, and the manner in which it will be transmitted to the user. Objectives of the system include gathering and communicating useful information, but equally important is accomplishing this in a timely manner. A properly designed system can be of immeasurable value to all levels in the organization.

The MIS acts as a service department with the task of providing information of value. The information needs of the complex mining enterprise are many, and when we think of a MIS product, a detailed computer printout may come to mind. In actual practice, the product may, and probably should be, a sheet of readily readable technical or economic information, or a simple collection of articles from the daily business press that affects the company. The purpose of this output, and indeed, MIS, is to provide information that will influence management decisions and performance of the enterprise.

8.6.9 CONSULTANTS AND ADVISORS

Samuel Johnson said, "The next best thing to knowing something is knowing where to find it." Clearly, quality advice is worth seeking, and in many instances, it is a legal necessity. Specialists can include the outside auditor who checks the accounting records, consultant who verifies ore reserves, and investment bankers who provide due diligence for the new stock issue. While involvement of some of these may be mandatory, use of others is discretionary. Discretion is the byword. Best and most effective contributions are forthcoming only from the outside professional that understands the needs, philosophy, objectives, and integrity of the organization's management.

External auditors are legally required to ensure that the bookkeeping records are correct and that financial statements present a true and fair picture of the business. Comfort with company information evolves over time as the outside auditor considers data and reports prepared by internal accountants, and as the outside expert performs spot checks to assure accuracy. The auditor not only urges full disclosure but where necessary suggests policy changes. It is the responsibility of auditors to pass on their findings and recommendations to top management, board committees, and the board of directors. Additional benefits of experts include well-presented financial and accounting information that reduces criticism and shareholder lawsuits—all being undertaken to develop and protect the investor. Such specialists and consultants are plentiful and can be used frequently, and in fact public accounting services often make up a significant portion of outside charges. Financial exposure and stock exchange requirements prompt the use of the public accounting firm.

Outside legal advisors also provide critical services, especially when forming the corporation, commencing operations, obtaining environmental permits or resolving legal disputes. In-house attorneys may handle these activities, but it is often advantageous to retain experts in each specific area. This practice provides knowledgeable advice that can be called on only when required.

Outside technical advisors such as geologists, engineers, and consulting firms have their place and can pay their way, even though this may involve duplication of in-house skills. Normally, the independent professional geologist or engineer is viewed by top management and outside investors as being objective and having broader, more varied experience. Examinations and reports of these experts are usually required to satisfy financing or lending institutions, and if such advisors are to be involved, it is worthwhile having them on board early enough to influence the methods utilized. Specialists in the environmental, governmental, and political arenas are justified in the same manner.

Today's complex and confusing rules, regulations, and laws of practically every facet of business require special expertise, and this has fostered an increasing use of advisors and consultants. In the past, it was thought economical to call on these professional skills only when absolutely required, but current practice is to retain such talents so that they can be utilized on a regular basis. At one time, the use of consultants was eschewed, especially in the technical and operating area. Management was reluctant to admit that it did not have the skills to handle many of the mining, processing, and operating tasks encountered in running a mine. Thoughtful analysis should allow one to conclude that in special cases, these activities, similar to certain accounting and legal sectors, can justify outside experts. Traditionally, staff personnel claimed that taking a consultant into confidence allowed the expert to obtain company secrets. Another criticism is that the expert only obtains opinions and plans of the company, dresses these up, and resubmits them to top management. To some extent, this is a valid criticism; however, the wise manager will make sure that much more is obtained from than given to any advisor.

Favorable results are most likely if management selects a highly qualified consultant, clearly outlines the assignment, regularly monitors progress, and gives this outsider complete support. Establishing these guidelines is important, but the most critical act is the initial decision to retain an advisor or consultant. Recognizing the need and value of an outside advisor requires sensitivity and confidence, and can often be the factor that assures success.

8.6.10 MANAGEMENT PHILOSOPHY

8.6.10.1 Characteristics

Much has been written about the attributes of the successful enterprise, and as might be expected, surveys and studies frequently find a host of similar characteristics. Among these are

1. Action oriented: All levels of supervision and management are inculcated with a desire to act, change things, initiate new programs, and solve problems.

2. People dedicated: Attention is directed toward all levels of employees, regardless of the job. In order to maximize productivity, the entire work force must understand the required tasks and be dedicated to improving performance.

3. Simple in form: All activities of the company are kept as simple as possible. Headquarters staff should be lean and policies and procedures minimal. Every function needs to be scrutinized to determine whether it can be eliminated or simplified.

4. Committed to improvements and innovation: Competition is becoming more severe and global, and improvement in operations must be initiated if the business is to remain competitive and survive.

5. In contact with customer: Unfortunately, most mining companies produce a commodity product with little opportunity to differentiate. Therefore, much creativity is required to main-

tain the competitive edge. Contact with the customer offers the opportunity to hear his likes and dislikes, so that a purer or improved commodity can be produced. Typically, one obtains new business by lowering prices, but providing an available inventory, giving excellent shipping options, or extended payment terms may also be of value in retaining or acquiring customers.

6. Emphasizing the need for profitability: Accountability for profit and loss should be pushed down to the manager who has hands-on control over marketing and production. Additionally, there is merit in having a portion of the compensation of every employee tied to overall performance and profitability of the company.

7. Flexible: There is nothing permanent except change, and therefore the successful organization attempts to anticipate change and respond appropriately. Metal prices, politics, labor policies, and operating conditions change, and alert management must be willing to minimize the disadvantages and capitalize on the opportunities presented.

The foregoing attributes of the successful enterprise are most important, and few would argue that they are not worthy guidelines. Interestingly, few mining companies follow all of the suggested practices. No doubt, a portion of the inaction is caused by uncertainty of how the union will react to the changes. Also there is concern whether any compensation program based on profitability will work well when prices are low and profits unsatisfactory. Additionally, past practices, especially those involving relationships with customers, have a pattern of receiving little attention. Typically the manager has concentrated on production and allowed the commodity to sell itself. This philosophy is changing, and it is now apparent that the mining company must be more responsive to the customer if it is to retain its sales base. Last of the attributes listed previously and foremost in drawing criticism to the mining industry over the years has been its lack of flexibility. No doubt, this as well as the other reasons for not following the suggested principles have a basis, but the benefits of change are so overwhelming that any risk is miniscule compared to what may be gained by remaining flexible. The survivors of the last decade have not only exhibited flexibility but also creativity, and if these characteristics are retained mining will indeed remain a viable industry.

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Chapter 8.7
MINE CLOSURE, SEALING, AND ABANDONMENT

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8.7.1 INTRODUCTION

All mines will eventually close. When a mine permanently stops operating due to economic conditions, the depletion of reserves, or any other reason, the following activities generally occur:

1. Underground mine openings are sealed.
2. Surface facilities are removed.
3. Surface mines and the surface areas of underground mines are reclaimed.

Only after these activities are completed can the mining company abandon the site. However, even after the mine is abandoned, specific postmining liabilities rest with the mining company. This chapter presents aspects of the closure, sealing, and abandonment activities.

8.7.1.1 Definitions

To enable the reader to interpret this chapter, the following definitions are provided.

Closure: The act of closing or the condition of being closed, such as the closing of a mine. There are different degrees of closure: permanent, temporary, and semi-permanent. In this chapter, it is considered that the closing of a mine is permanent. Mines that are not open for production but that could be reopened should be considered as temporarily or semi-permanently closed.

Sealing: The securing of mine entries, drifts, adits, slopes, shafts, and boreholes with suitable materials to protect against fires, gas, and water emissions and for the safety of the public (Foreman, 1971; Thrush, 1968).

Abandonment: The act of abandoning and relinquishment of a mining claim or intention to mine; a voluntary surrender of the claim or mine to the next party. This differs from forfeiture that can be considered the involuntary surrender of a mine by neglect (Thrush, 1968).

8.7.1.2 Regulatory Requirements

In the United States, federal, state, and local governments have issued regulations that must be followed during the closure of a mine. Because regulations may change, a summary of only the existing federal regulations is presented in this chapter. Federal agencies that regulate mine closure activities are the Mine Safety and Health Administration (MSHA), Environmental Protection Agency (EPA), Office of Surface Mining Reclamation and Enforcement (OSMRE), US Geological Survey (USGS), Bureau of Land Management (BLM), US Forest Service (USFS), National Park Service (NPS), and Bureau of Indian Affairs (BIA) (Anon., 1980a). MSHA regulations are applicable for health and safety standards and apply to all coal and noncoal mines. EPA's water discharge effluent limitations apply also to all coal and noncoal mines. OSMRE reclamation regulations apply only to nonanthracite coal mines. Pennsylvania anthracite mining regulations apply to all anthracite coal mining by OSMRE reference. Since the other agencies (USGS, BLM, USFS, NPS, and BIA) have jurisdiction only in federal lands, their regulations are not presented. If applicable to a specific mine, these agencies should be contacted directly.

A brief synopsis of the applicable federal regulations appears in Tables 8.7.1, 8.7.2, and 8.7.3.

Individual states have different rules and regulations that must be followed during the closure of mines. All states must follow the federal OSMRE regulations as they apply to coal mines. OSMRE regulations allow states to request primacy in self-regulating these rules and regulations. Most states with coal mining activities have passed their own regulations that closely resemble the federal OSMRE regulations, thus regulating their coal mining industry. Individual state health and safety rules may differ from the MSHA rules and must be considered in addition to the MSHA regulations when closing a mine.

Local government may enact rules covering topics such as (1) zoning (which could affect the selected postmining land uses) and (2) land value assessment (for taxation). It is suggested that engineers involved with a mine closure contact their local and state regulatory agencies to determine if any local regulations apply.

8.7.2 MINE CLOSURE PRACTICES

8.7.2.1 Surface Mines

The closure of all new surface mines must be planned prior to the beginning of operation. A reclamation plan must be included with all new permit submittals. This planning is also important in evaluating the economic justification for the mining operation. A substantial amount of money can be saved if the most appropriate reclamation or closure procedures are thoroughly considered and carried out as planned.

Unless an alternative plan is approved at the closure of a surface mine, the mine site must be regraded to approximately the original contours. Backfilling should remain as close as possible to the active mining area, not only to reduce haulage distances but also to save on reclamation costs in the event that the surface mine must be closed and regraded due to economic conditions prior to its planned closure.

Reclamation planning and closure methods vary among types of surface mines. Some different methods are discussed briefly here.

Open Pit Mines: *Open pit mining* is defined as "the mining of ores (primarily referring to metalliferous ores) by surface mining methods where waste or overburden is first removed, and the mineral is broken and loaded" (Thrush, 1968).

At the present time, the closure of open pit mines is controversial. Currently, there are no federal regulations covering the closure of open pit mines. The operators must contact local and state agencies to determine what requirements must be followed at those governmental levels. According to Dohm (Crawford and Hustrulid, 1979), the increased environmental awareness (of open pit mining) definitely indicates that proper production planning and scheduling can minimize the costs associated with mine reclamation. The items most effectively handled by efficient planning are (1) returning ground contours to approximate premine conditions by blending in the new grades with the surrounding topography, (2) minimizing surface depressions to the greatest extent possible, and (3) revegetating all disturbed areas.

Table 8.7.1. Federal Regulations Applicable to Mine Closure, Sealing, and Abandonment, Mine Safety and Health Administration, *Code of Federal Regulations,* Chap. 30

Regulation	Paragraph	Description
Part 57—Metal and Non-Metal Underground Mines—Safety and Health Standards	57.1000	When any mine is closed, the person in charge shall notify the nearest subdistrict office and indicate whether the closure is temporary or permanent.
Part 75—Underground Coal Mines. Mandatory Safety Standards	75.1204	Within 60 days of (a mine's) permanent closure or abandonment, mine operators must file a copy of the mine map, revised and supplemented to the date of closure, with the Secretary of the Interior. The map must be certified by a registered surveyor or registered engineer.
	75.1204-1	The notice of mine closure and copies of the mine map must be filed with the Coal Mine Safety District office.
	75.1711	Mine openings declared inactive or permanently closed or abandoned for more than 90 days shall be sealed.
	75.1711-1	Shaft openings to be sealed shall be capped or filled. Filling shall be for the entire length of the shaft with the first (lower) 50 ft (15 m) being filled with incombustible material. Caps shall consist of 6-in. (152-mm) thick concrete or equivalent and equipped with a vent pipe (2 in. or 50 mm in diameter, and 15 ft or 5 m above the surface).
	75.1711-2	Slope or drift openings shall be sealed with solid, incombustible material for a distance of at least 25 ft (8 m) into the opening.
Part 77—Surface Coal Mines and Surface Work Areas of Underground Coal Mines—Mandatory Safety Standards	77.215.4	MSHA's District Manager shall be notified in writing when a refuse pile is to be abandoned. If a hazard is determined to be present, a plan for abandonment shall be submitted by the operator and approved by the District Manager. The plan shall include a schedule for its implementation and describe provisions to prevent burning and future impoundment of water and provide for major slope stability.
	77.216-5	Prior to a mine's abandonment, an abandonment plan for water, sediment, or slurry impounding structures shall be submitted by the operator and owner and approved by MSHA's District Manager. The plan shall include a schedule, a provision to preclude future impounding, and provide for slope stability.

Table 8.7.2. Federal Regulations Applicable to Mine Closure, Sealing, and Abandonment, Office of Surface Mining Reclamation and Enforcement, *Code of Federal Regulations,* Chap. 30

Regulation	Paragraph	Description
Part 715—General Performance Standards	715.13	All disturbed areas shall be restored in a timely manner to conditions that are capable of supporting the uses the lands were capable of before mining or to higher or better uses.
	715.14	Postmining graded slopes must approximate the premining natural slopes (approximate original contours). Certain exceptions, such as for mountaintop removal or leaving permanent impoundments, may be approved.
	715.15	The permittee must plan and conduct reclamation operations to minimize disturbance to the prevailing hydrologic balance in order to prevent long-term adverse changes in the hydrologic balance. Changes in water quality and quantity, both surface and groundwater, shall not affect the postmining land use. If pollution can be controlled only by treatment, the permittee shall operate and maintain the necessary water treatment facilities for as long as treatment is required. Groundwater recharge capacity shall be restored to approximate premining recharge capacity. The permittee shall be responsible for monitoring to ensure conformance.
	715.18	All dams shall be removed and the disturbed area regraded, revegetated, and stabilized unless the regulatory authority approves retention of such dams as part of the postmining land use plan.
	715.20	The permittee shall establish on all land that has been disturbed a diverse, effective, and permanent vegetative cover of species native to the area or species that will support the postmining land use.
Part 717—Underground Mining—General Performance Standards	717.14	Upon completion of the underground mining, road cuts and mine entry area cuts shall be regraded to approximate original contours.
	717.17	The permittee must plan and conduct reclamation operations to minimize disturbance to the prevailing hydrologic balance in order to prevent long-term adverse changes in the hydrologic balance. Changes in

Table 8.7.2.—cont.

Regulation	Paragraph	Description
		water quality and quantity, both surface and groundwater, shall not affect the postmining land use. If pollution can be controlled only by treatment, the permittee shall operate and maintain the necessary water treatment facilities for as long as treatment is required. Groundwater recharge capacity shall be restored to approximate premining recharge capacity. The permittee shall be responsible for monitoring to ensure conformance.
	717.18	All dams shall be removed and the disturbed area regraded, revegetated, and stabilized unless the regulatory authority approves retention of such dams as part of the postmining land use plan.
	717.20	The permittee shall establish on all land that has been disturbed by mining operations a diverse, effective, and permanent vegetative cover capable of self-regeneration and plant succession and adequate to control soil erosion. Introduced species may be substituted for native species if approved by the regulatory authority.
Part 780—Surface Mining Permit Applications—Minimum Requirement for Reclamation and Operation Plan	780.18	Each permit application shall contain a plan for reclamation of the lands within the permit area. This plan is to be followed during the closure of a mine.
	780.20	Each permit application shall include a plan that includes steps to be taken during reclamation to minimize disturbance to the hydrologic balance within the permit area and adjacent areas, to prevent material damage outside the permit area, to meet federal and state water quality laws and regulations, and to protect the rights of present water users.
	780.23	Included in the reclamation plan shall be a description of the proposed postmining land use, including a discussion of its utility and capacity to support alternative uses and the relationship of the proposed use to existing land use policies and plans.
	780.25	Each reclamation plan shall include timetables and plans to remove ponds, impoundments, banks, dams, and embankments, if appropriate.
Part 784—Underground Mining Permit Applications—Minimum Requirements for Reclamation and Operation Plan	784.13	Each permit application shall contain a plan for reclamation of the lands within the permit area. This plan is to be followed during the closure of a mine.
	784.14	Each permit application shall include a plan that includes steps to be taken during reclamation to minimize disturbance to the hydrologic balance within the permit area and adjacent areas, to prevent material damage outside the permit area, to meet federal and state water quality laws and regulations, and to protect the rights of present water users.
	784.15	Included in the reclamation plan shall be a description of the proposed postmining land use, including a discussion of its utility and capacity to support alternative uses and the relationship of the proposed use to existing land use policies and plans.
	784.16	Each reclamation plan shall include timetables and plans to remove ponds, impoundments, banks, dams, and embankments, if appropriate.
Part 800—Bond and Insurance Requirements—For Surface Coal Mining and Reclamation Operations Under Regulatory Programs	800.13	Performance bond liability shall be for the duration of the mining and reclamation operation and until successful revegetation or until achievement of reclamation.
	800.14	The amount of bond shall be sufficient to assure completion of the reclamation plan by the regulatory authority in the event of forfeiture.
	800.17	The period of bond liability shall last until all reclamation, restoration, and abatement work has been completed.
	800.40	The applicant may file an application for release of all or part of a performance bond after the applicant has completed any part of the reclamation activities. A newspaper advertisement stating that a bond release has been requested shall be placed by the operator. The advertisement shall run once a week for four weeks in a local newspaper of general circulation.
		The regulatory authority may release all or part of a bond when it is satisfied that all reclamation or a phase of reclamation is accomplished.
Part 816—Surface Mining Activities—Permanent Program Performance Standards	816.11	Signs and markers identifying the mine and permit numbers shall be retained and maintained until after the release of all bonds for the permit area.
	816.13	When no longer needed for monitoring or other purposes, each exploration hole, other drilled holes, and other exposed underground openings shall be capped, sealed, backfilled, or otherwise properly managed.

Table 8.7.2.—cont.

Regulation	Paragraph	Description
	816.49	A permanent impoundment of water may be created if it is designed for permanent use, will not result in the diminution of quality and quantity of water utilized by adjacent landowners, and will be suitable for approved postmining land use.
	816.132	Persons who cease surface mining activities shall close, backfill, or otherwise permanently reclaim all affected areas. All underground openings, equipment, structures, or other facilities not required for monitoring, unless approved as suitable for postmining land use, shall be removed and the affected area shall be reclaimed.
	816.133	All disturbed areas shall be restored in a timely manner to conditions capable of supporting the uses they were capable of supporting before mining or to higher or better uses.
	816.150	A road not to be retained for use under an approved postmining land use shall be reclaimed immediately after its use for mining and reclamation operations ceases.
Part 817—Underground Mining Activities—Permanent Program Performance Standards	817.15	When no longer needed for monitoring or other purposes, each shaft, drift, adit, tunnel, exploration hole, entry way, or other opening shall be capped, sealed, backfilled, or otherwise properly managed.
	817.48	A permanent impoundment of water may be created if it is designed for permanent use, will not result in the diminution of quality and quantity of water utilized by adjacent landowners, and will be suitable for approved postmining land use.
	817.56	Before abandoning a permit area or seeking bond release, the operator shall ensure that all temporary erosion and sedimentation control structures are removed and reclaimed and that all permanent sedimentation ponds, diversions, impoundments, and treatment facilities meet the requirements of the *Code of Federal Regulations,* Chap. 30, Pt. 817.
	817.132	Persons who cease underground mining activities shall close, backfill, or otherwise permanently reclaim all affected areas. All underground openings, equipment, structures or, other facilities not required for monitoring, unless approved as suitable for postmining land use, shall be removed and the affected area shall be reclaimed.
	817.133	All disturbed areas shall be restored in a timely manner to conditions capable of supporting the uses they were capable of supporting before mining or to higher or better uses.
	817.150	A road not to be retained for use under an approved postmining land use shall be reclaimed immediately after its use for mining and reclamation operations ceases.

Table 8.7.3. Federal Regulations Applicable to Mine Closure, Sealing, and Abandonment, Environmental Protection Agency, *Code of Federal Regulations,* Chap. 40

Regulation	Paragraph	Description
Part 401 —General Provisions	401.12	Requires achievement of effluent limitation for point sources. It is generally the view of the EPA that the effluent limitations apply until the point source is eliminated.
Part 434—Coal Mine Point Source Category	434	Provides effluent limitations for point discharges of various categories of coal mines, coal preparation plants, and associated areas.
Part 430—Mineral Mining and Processing Point Source Category	436	Provides effluent limitation for point discharges of various categories of mineral mining.

Open pit mines often may be temporarily closed since the selling price of the commodity being produced can fluctuate greatly. Past mining methods called for mine overburden to be totally removed and placed in a designed disposal site. Backfilling the pit with the previously disposed waste rock is not normally done as it would be economically disastrous to the mining company. Backfilling of the extreme low areas to eliminate any ponded water may, however, be appropriate. Grading and vegetating pit slopes is also important for ground stabilization reasons.

Quarries: *Quarries* are defined as "open or surface workings, usually for the extraction of building stone, slate, limestone, etc." (Thrush, 1968). Quarries normally sell most of the material mined and therefore leave little waste material for filling in the excavation.

The closure practices employed to achieve effective abandonment include regrading to eliminate hazardous highwalls and revegetation. Steep highwalls can sometimes be removed by "shooting down the highwall." This method (Fig. 8.7.1) can inexpensively reduce the grade and effectively remove a highwall. The regraded area could then be covered with soil and revegetated.

Novel ideas on reclamation are being developed by the the quarry industry. Because many sites are located near urban

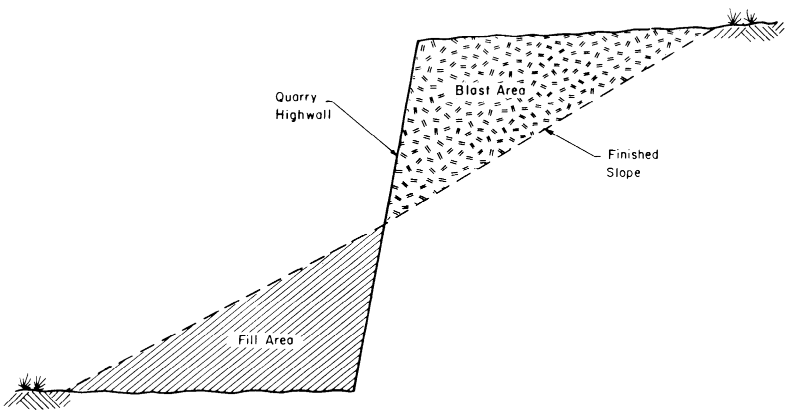


Fig. 8.7.1. Typical section—shooting down the highwall.

areas, landscape architects have become involved with mine planning to integrate the quarry site with existing communities (Culbertson et al., 1988). Three-dimensional simulation by computers is becoming increasingly popular. The view of postmining land is important in areas of high public visibility. In areas close to large population centers, some quarry sites have planned the reclamation to include a lake surrounded by a housing project in the lower parts of the quarry, thus increasing the postmining land value.

Surface Coal Mines: A reclamation plan must be approved prior to the opening of a *surface coal mine.* This plan must be followed when the mine is closed. It is standard practice to submit a reclamation schedule to regulatory agencies within a few months after a decision is made to close a mine permanently. Normally, the backfilling of cuts and the restoration of the surface is required.

Placer Mines: The term *placer* is applied to deposits of sand, gravel, and other detrital or residual material containing a valuable mineral that accumulated through weathering and mechanical concentrating processes (Wells, 1969).

Currently, there are no federal regulations that apply to the closure of placer mines. It is recommended that regulatory agencies be contacted to determine if the project site is subject to any new regulations. Most areas in the United States that are subject to placer mining are on federal or state lands. Reclamation and closure practices for each placer mine site should be discussed and agreed upon prior to leasing the property.

8.7.2.2 Mine Facility Removal

In accordance with mine reclamation regulations, all associated facilities must be removed unless they serve a useful purpose in the postmining land use. These facilities include buildings, material handling systems (conveyors, rail lines, transfer stations, storage bins, docks, etc.), electrical lines, transformers, substations, pipelines, roadways, drainage ponds, and drainage channels. Waste disposal areas must be reclaimed, and any hazardous material must be removed from the site and disposed of at a hazardous waste disposal facility. The following sections discuss certain aspects of removing these facilities.

Demolition and Salvage of Structures: A complete inventory of all available equipment, parts, and supplies should be made as soon as a decision to close a mine is made. From this list, the dispensing of each reusable or salable item can be chosen. Some items could be sent to another mine that is owned by the company. Attempts to sell the remaining items of value should be made. Items can be sold:

1. To a salvage company (usually at “scrap prices”).
2. Individually, grouped by bid, or grouped by negotiated price.
3. By public or private auction (auctioneer fees normally range from 5 to 10% of the sale)

For removal of buildings and other structures, a demolition and disposal contractor should be hired. A specialist in this type of work is likely to be less expensive than a mining company performing the removal itself. If enough materials such as copper and steel are salvageable, the contractor may do the project at no cost or even pay the mining company for the salvage rights. The mining company should develop a demolition specification to be agreed to by the contractor. A sample demolition specification, as used by the West Virginia Dept. of Energy on an abandoned mine site demolition project, follows (Anon., 1988a).

“DEMOLITION: Demolition of existing structures shall be performed using standard construction equipment wherever practical. Demolition operations shall be performed with the utmost care not to endanger life or property. The contractor shall be responsible for analyzing all of the structures to be razed so that demolition operations are performed in a manner which results in a total and safe collapse of the structures while maintaining the safety of construction laborers, equipment operators, and vehicular traffic along all public roads.

DEBRIS REMOVAL: All concrete, concrete block and timber, remnants, metal scrap, equipment, and other debris shall be removed from the project area. All foundations are to be completely removed, and the areas regraded to the final ground surfaces shown on the plans, or to the approximate original ground surface.

DEBRIS DISPOSAL: All concrete, concrete block, timber, metal scrap, structural remnants, equipment, garbage and demolition debris shall become the property of the contractor and shall be salvaged or buried within proposed embankment. Onsite burial of noncombustible materials in an approved area is permissible provided a minimum of 2 ft (0.6 m) of natural soil fill is placed over the buried material and provided that the concentration of debris to be buried in any one area is not excessive.

REGRADING: All areas where structures, foundations, equipment, etc., have been demolished and removed shall be regraded. The approximate limits of regrading are shown on the drawings. The slopes shall be regraded to form stable, uniform slopes which conform to the natural slopes in the area and promote proper drainage of surface runoff into natural drainage ways. Upon completion of regrading, the slopes shall be revegetated.”

Waste Disposal Areas: Mine wastes that are placed in dry disposal areas can be closed after the site is graded for proper

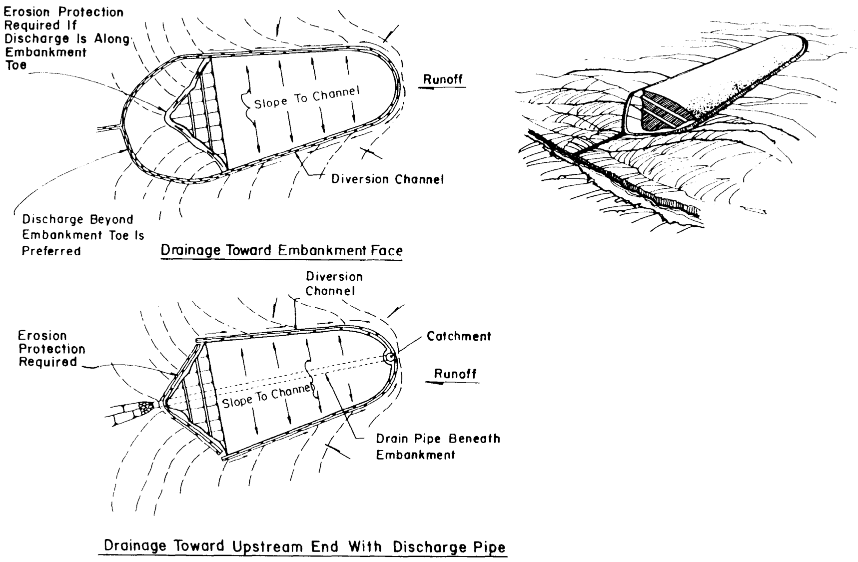


Fig. 8.7.2. Typical final drainage pattern, valley fill disposal site (Anon., 1973).

drainage, covered with soil, and revegetated. Because most waste sites consist of elevated piles, the piles must be left with stable slopes to reduce the potential for sliding. The final grading should be done in a manner that reduces water infiltration into the disposal site so that the potential for toxic leachate generation and surface erosion can be minimized. Fig. 8.7.2 shows a typical layout and grading pattern for a valley fill disposal site. The following two paragraphs are excerpted from the US Department of the Interior's *Engineering and Design Manual for Coal Refuse Disposal Facilities* (Anon., 1975a):

"The final grading of the coarse (dry) refuse would take on a convex or ridgelike configuration with some precipitation running over the first terrace of the refuse slope. However, the bulk of the surface runoff from the crest would flow into diversion ditches around the fill."

"In the instance of an elevated ridge or valley-dump type of facility, the final grading and drainage plan of the crest will have a concave configuration with subsurface drain inlets placed at low point(s). All surface runoff will thus be directed away from the edge of the slope face."

Toxic leachate from waste piles must be eliminated prior to final abandonment of a site. If this is not possible, then a means to treat the water discharge to meet EPA standards must be maintained until the standards can be met and the discharge point eliminated from the EPA's list of point discharges.

Slurry waste disposal impoundments normally are closed by breaching the embankment, letting the material solidify through drying, regrading the surface to drain, covering with soil, and revegetating. Some slurry disposal areas have recently been successfully reclaimed as wetlands (Nawrot et al., 1987). This approach provides new habitat for wildlife and, if appropriate, can reduce closure costs.

Utilities and Roadways: Unless included for a postmining use or for postmining maintenance, electrical facilities such as power lines, substations, and pipelines should be removed during the closure of a mine. Electrical power lines and substations can sometimes be sold to local utilities or other local industries but must remain in service until electrical power is no longer needed at the mine site.

Aboveground pipelines should be dismantled, removed from the site, or buried on site. Underground pipelines can sometimes be capped and abandoned in place unless local regulations specifically require their removal.

Roadways are to be removed and revegetated when they are no longer needed. This can be accomplished by scarifying the surface and regrading to blend the mine roadway into the surrounding landscape. Topsoil may be required to cover the road base material. However, most regulatory agencies will allow roadways to remain if the landowner requests their use for access and to promote the postmining utilization of the property. Roadways can also be used for access to the property during postmining maintenance activities until the reclamation bond is released.

Drainage Facilities: Sedimentation control ponds and ditches should remain active and be maintained for a few years after the mine is closed until the revegetation of the site can successfully control erosion. After their use for erosion and sedimentation control, ponds and ditches can be removed by regrading them to contours that match the surrounding landscape.

Removal of Hazardous Materials: During the operation and particularly during the closure of a mine, the mining company's liabilities with regard to hazardous substances under the Resource Conservation and Recovery Act (RCRA) and the Comprehensive Environmental Response, Compensation and Liability Act (Superfund) must be considered (see Chapter 3.4). Although there are a few exceptions, most mining wastes are not currently considered hazardous by the EPA.

Haller (1987) reports that, "the law (Superfund) provides that the owners and operators of mining sites, together with those who transport hazardous substances to such sites, can be held jointly and severally liable for the cost of cleaning up the site." If a site becomes listed by the EPA as a Superfund site, a cleanup agreement will have to be negotiated between the mining company (or whoever is deemed liable) and EPA.

To close and abandon a mine site that is not listed on the Superfund list is more common. During the closure operation, the mine operator must investigate the site to determine whether there are any hazardous substances on site. Engineers experienced with performing environmental assessments of properties should be retained to make this determination. Hazardous substances most commonly found include asbestos used as building and pipe insulation, PCBs used in electrical devices, and solvents used as cleaners. Areas that had been used as trash disposal areas during the mine operation should have the soil sampled to verify if any hazardous materials leached into the ground.

If contamination is found, a study should be performed by a qualified engineer to verify the degree of contamination. Any potentially contaminated soils must be adequately sampled to determine whether any action is required or if the soils can be left in place. If the levels of contamination are relatively high, then removal and offsite disposal or onsite treatment of the soils may be required. If hazardous substances are required to be transported offsite, a licensed hazardous waste contractor should be hired to remove the material from the site and transport it to a licensed hazardous waste disposal site. The owner should verify that the contractor and proposed disposal area are currently meeting all regulatory requirements, because if the wastes are spilled enroute to the disposal site or disposed of improperly, the owner may still be held responsible.

8.7.2.3 Underground Mines

Equipment Salvage: Prior to the closure of a mine, the operator should decide whether some equipment (parts, supplies, etc.) could be economically removed from the mine and sold or used elsewhere by the operator. An inventory of available equipment and materials should be made. To assess whether a salvage operation is economical or not, a cost estimate for the removal of each piece of equipment should be made and compared with its resale value. According to Brezovec and Heges (1986), difficult-to-move equipment, such as a coal mine longwall unit, is normally abandoned at a lower cost than the cost to recover the machinery unless, of course, a sure buyer has been found. Mobile equipment such as trucks and continuous miners can generally be sold for more than the cost of removing them. Underground cables that contain a large quantity of copper often are salvagable.

As stated in the previous section, some equipment that may contain hazardous materials must be either decontaminated or removed and disposed of at a hazardous waste disposal site. Inside mines, hazardous materials are generally limited to electrical equipment that contains PCBs.

Hydrology Analysis: Prior to designing any mine seals, the hydrology of the mine must be considered. OSMRE regulations require the completion of a hydrology analysis before permitting any new coal mines. These regulations have essentially prevented up-dip mining in acid-producing coal seams where the mine opening is at a lower elevation than the mine reserves. These regulations were adopted to minimize the impact of postmining water discharge.

A hydrology analysis is required to determine if a hydraulic head will be placed on the mine seal after closure. A crude method of calculating the hydraulic head is to subtract the mine opening elevation from the highest elevation of coal extraction. This method may be acceptable to some regulatory agencies, but it does not consider the potential for additional hydraulic head due to groundwater above the mine. Consideration of the potential hydraulic head associated with groundwater can, however, be very complex. To predict groundwater impacts accurately, hydrogeologists utilize information such as well records, geology, and surface topography to model the groundwater. Groundwater monitoring wells or piezometers may be necessary to monitor the groundwater levels in the mine's vicinity.

Ventilation Planning: During the closure of an underground mine, the shutdown of the mine's ventilation system must be well planned, particularly in gassy mines with water conditions. The ventilation shutdown plan needs to be reviewed on a mine-by-mine basis. The engineer must plan for the systematic shutdown of the system to keep all of the active work areas ventilated and safe from harmful or explosive gases. Regulatory agencies will often require this ventilation plan prior to permitting the closure activities.

For the final sealing of mine openings, again primarily for gassy mines, ventilation must also be considered. During the filling of shafts, some MSHA districts require remote testing for gas the entire length of the shaft. When a gas problem is found, all work must stop until the gas dissipates.

To seal drift, slope, or adit openings in gassy mines that are accessible, a temporary stopping can be erected beyond the work zone. Ventilation tubing can be extended to near the work area and connected to a fan to blow fresh air into the opening.

8.7.2.4 Revegetation

Revegetation of the surface areas of the mine must be accomplished in accordance with the mine's permit. There is much published information available to assist the engineer in formalizing these procedures for a particular mine. Sources of information include local universities with agricultural and/or mining departments, government agricultural agencies, and the US Bureau of Mines.

References by Williams and Schuman (1987), Lyle (1987), and Vogel (1987) are helpful manuals that can be used for mine revegetation planning.

8.7.3 SEALING OF UNDERGROUND OPENINGS

Previously, mine sealing was generally performed only as a safety precaution. Mining regulations now require sealing to be performed during closure. According to Utah's *Shaft Abandonment Guidelines* (Anon., 1987), the following parameters should be considered in sealing shafts.

1. Eliminate any danger to the health and safety of the general public.
2. Control release of hazardous, acid/toxic-forming materials or gases to the atmosphere.
3. Control the movement of underground water or hydrologic communication.

Before a sealing method is selected, the degree of mine closure must be determined. Potential future geologic or economic value, historic value, hazards, and costs must be considered. The different degrees of mine seals can be considered as follows (Anon., 1980b).

Permanent: A safeguard that would completely seal off abandoned workings and would preclude the rehabilitation and future access to the mine. This would be the case if all the ore reserve has been mined or economics dictate that future profitable operation is not deemed probable.

Temporary: Seals that prevent deliberate or accidental entry into a working while preserving the general condition of the opening for future use. If there is some potential future value that can be gained by maintaining an opening to the mine, then this type of closure method should be used. Methods employed are fencing around shafts, glory holes, adits, or drifts; locked doors for adits, drifts, or slopes; and concrete covers for shafts.

Semi-Permanent: A system of seals that completely seals or otherwise blocks an opening while maintaining the general integrity of the opening. Future access to the workings may be desirable. This method should be employed when there is future economic value, but by employing only the temporary method of closing the opening, a threat to the public may exist such as emission of radon or other gases.

As discussed earlier in this section, a hydrogeologic study must be done prior to selecting a seal type. This study will determine if a hydraulic head could build up behind the seal and estimate what it would be. The estimated hydraulic head must be used in seal design calculations.

Permanent seal types available as described in a Bureau of Mines publication (Adams and Lipscomb, 1984) are as follows.

Dry Seal: A dry seal is constructed by placing suitable material such as cement blocks in mine openings to prevent the entrance of air and water into the mine. A dry seal is suitable for openings where there is little or no flow and little danger of a hydraulic head developing.

Wet Seals: A wet seal prevents the entrance of air into a mine while allowing the mine discharge to flow through the seal. Seals of this type are constructed with a water trap similar to traps used in sinks and drains.

Hydraulic Seals: Construction of a hydraulic seal involves placing a plug in a mine entrance that is discharging water. The plug stops the discharge, and the resultant flooding excludes air from the mine and retards the oxidation of sulfide minerals.

8.7.3.1 Boreholes

Temporary seals on boreholes can consist of a locked cap over a protruding casing. When boreholes are no longer required for mine operation or monitoring purposes, a drilling contractor is usually hired to seal the opening. To permanently seal a borehole, the surface casing and protective cap should be removed to a few feet (meters) below the proposed final surface elevation. A plug must be installed in competent strata as close to the borehole and mine roof interception as possible, either remotely by the drilling contractor or from inside the mine if the location of the plug is accessible. The borehole can then be sealed by filling with a nonshrink cement grout. A pour pipe extending to the bottom of the hole should be utilized when placing the cement grout to assure uniform placement of the grout and eliminate voids. The Pennsylvania Dept. of Environmental Resources (Anon., 1988b) guidelines indicate that grout should be placed to within 2 ft (0.6 m) of the surface, with the remainder being filled with dirt to blend into the surrounding area. Other methods employed are plugging the borehole with a bentonite gel or if the borehole is known to have a grouted casing, a plug at the top and bottom and the remainder being filled with inert material may be sufficient. Fig. 8.7.3 shows two types of borehole plugs.

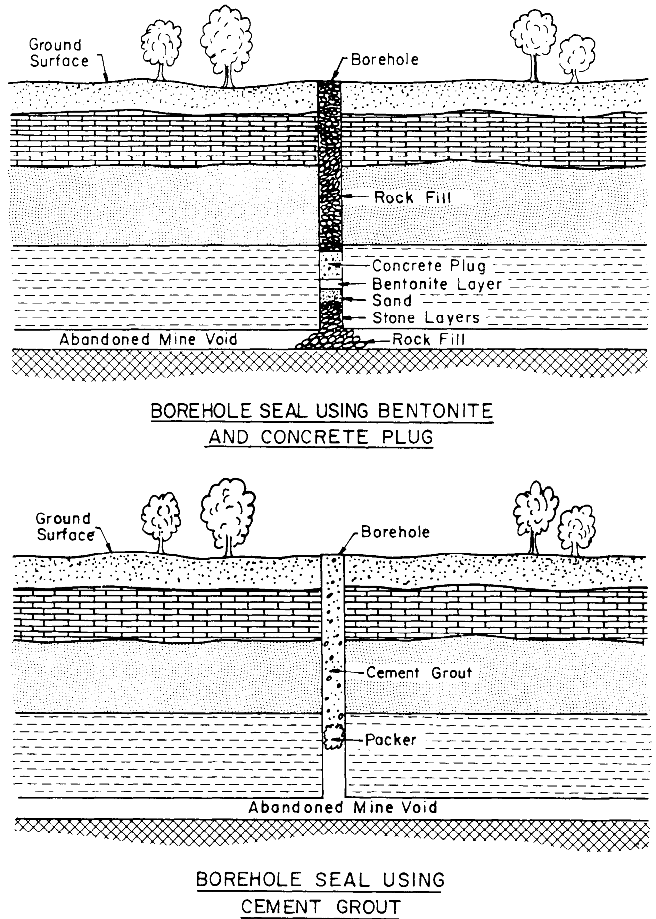


Fig. 8.7.3. Borehole seals (Anon., 1980a).

8.7.3.2 Shafts

For permanent mine closure, OSMRE requires shaft openings into coal mines to be either filled or capped. Filling shall be for the entire length of the shaft with the lower 50 ft (15 m) filled with incombustible material. Caps shall consist of 6-in. (120-mm) thick concrete or other equivalent means and be equipped with a vent pipe (2 in. or 50 mm in diameter, and 15 ft or 5 m above the surface).

Examples of shaft fill plans (where there are no expected hydrologic heads) are presented in Figs. 8.7.4 and 8.7.5. The following is a reprint of the *Shaft Abandonment Guidelines* from the Utah Dept. of Natural Resources (Anon., 1987).

"If there is no hydrologic involvement, small-sized fill material should be interspersed with large debris to allow for void filling. Any debris deposited in this manner should not create voids within the fill that could subside at a later date. Maximum compaction attained during placement is the goal of the selection of the type of backfill material."

"The backfill material selected must be free of acid/toxic-forming and combustible materials. No wood or metal debris should be considered for backfilling of shafts."

"Inorganic and organic silts and clays should be avoided as much as possible. Gravel/sand combinations are the best quality. The material shall be sized so as to minimize voids, i.e., uniformly graded or well-graded. Results of a sieve analysis and engineering soil characteristics shall be submitted to the Division with the reclamation plan about the backfill material."

"In the event of hydrologic movement, consideration must be given to the unique situation and will be handled specifically."

"Caps are recommended at the collar of shafts. A port should be included in the design to monitor the backfilled material. Maintenance of the shaft abandonment should also be included in the plan in the event more backfill may be needed."

To account for settlement in shaft backfill material, the State of Montana (Anon., 1988c) requires that, "shafts be backfilled to a finished elevation above the surrounding natural ground equal to 5% of the shaft depth or as directed by their engineer."

To ensure safety when filling in shafts, the following procedures should be utilized (Hoelle, 1988):

1. Remove surface structures surrounding the shaft.
2. Erect temporary protective fence around work area.
3. Place a sturdy barrier (wheel stop) around the shaft collar.
4. Place a steel cover over the shaft top during idle periods.
5. Monitor for hazardous gases.
6. Place a fence around the site after filling.

An example of a concrete cap used to seal an abandoned shaft is shown in Fig. 8.7.6. Inverted pyramid-shaped caps (or plugs) have also been used successfully to seal abandoned shafts. Dressel and Volosin (1985) describe this method in a US Bureau of Mines publication; it is depicted in Fig. 8.7.7.

If a hydraulic shaft seal is planned in an attempt to eliminate water discharges from a shaft, seals such as the ones shown in Figs. 8.7.8 and 8.7.9 can be designed. These hydraulic plugs must be designed using the same structural design methods as used to design a surface dam.

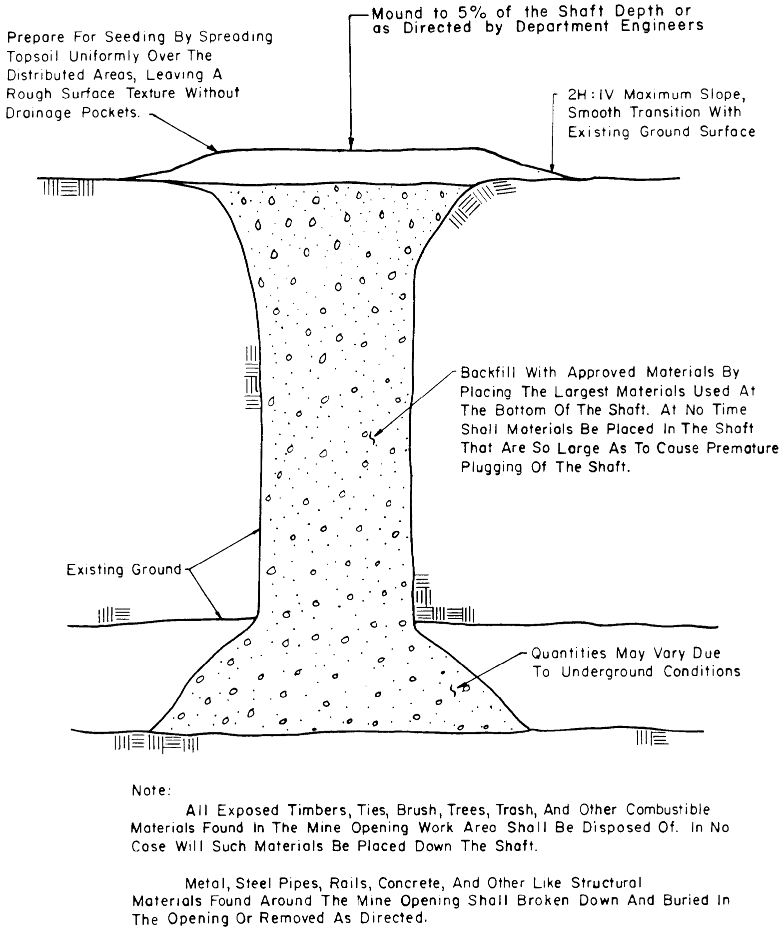


Fig. 8.7.4. Dry shaft seal (Anon., 1988c).

8.7.3.3 Slopes, Drifts, and Adits

Dry Seals: Dry seals can be used to close slopes, drifts, and adits when there is no hydraulic head anticipated at the mine opening. Several different methods can be employed. A method seldom used today is that of simply drilling and shooting down the roof and regrading the area. Fig. 8.7.10 depicts this sealing method. This cannot be done at coal mines because OSMRE requires the opening to be filled by at least 25 ft (8 m) of incombustible material. When the opening is accessible, a concrete wall is often constructed inside the 25-ft (8-m) zone and backfilled. Figs. 8.7.11 and 8.7.12 depict other typical dry seals.

Mine seals placed pneumatically like those depicted in Fig. 8.7.13 are also often used at remote locations (Roberts and Masullo, 1986). Aggregate can be pumped pneumatically through a pipeline from a location accessible to large equipment to the unaccessible mine opening that is to be sealed. If necessary, the aggregate can also be injected with a grout.

Wet Seals: Wet seals are constructed at locations where a hydraulic head is anticipated. A typical wet seal design is shown in Fig. 8.7.14. A variation of a wet seal that is meant to keep air from entering the mine is called an *air seal.* These are constructed at mines where an attempt is being made to limit the oxygen content of the mine atmosphere and thereby limit acidic water production. Fig. 8.7.15 depicts an air seal.

Hydraulic Seals: Current EPA regulations require that all mine water discharges be within acceptable limits. The limits vary according to mine type. After mine closure, mining companies have been required to continue to treat the mine discharge unless it meets the effluent discharge limitations. To eliminate treatment cost and to improve the environment, hydraulic seals have been designed and installed to act as dams and eliminate water discharge. Designing a hydraulic seal to withstand a head of water is not simple. There have been many designs used. Figs. 8.7.16 through 8.7.20 show various types. To be designed and constructed successfully, several design criteria should be met (Chekan, 1985):

1. The bulkhead should be designed to withstand the static forces of hydrostatic pressure rather than the dynamic forces of an explosion.

2. The bulkhead should be constructed from a material, such as concrete, that will resist deterioration by water.

3. The bulkhead should be constructed to be sufficiently thick and properly anchored, and the surrounding strata should be pressure grouted to minimize water seepage.

In addition to the general criteria listed, Chekan also reports that the following factors should be considered before designing and constructing a bulkhead to impound water at a coal mine:

1. The bulkhead should be located in competent ground that is not excessively fractured or broken, preferably in areas of stable ground. However, in most coal mines, ground movements such as roof convergence and floor heave are inevitable, and supplemental roof supports should be installed at the site.

2. The bulkhead, in most cases, should be designed to withstand the maximum hydrostatic pressure that can develop. Practical limits of potential inundation can be determined by plotting the expected mine pool elevations and corresponding ground

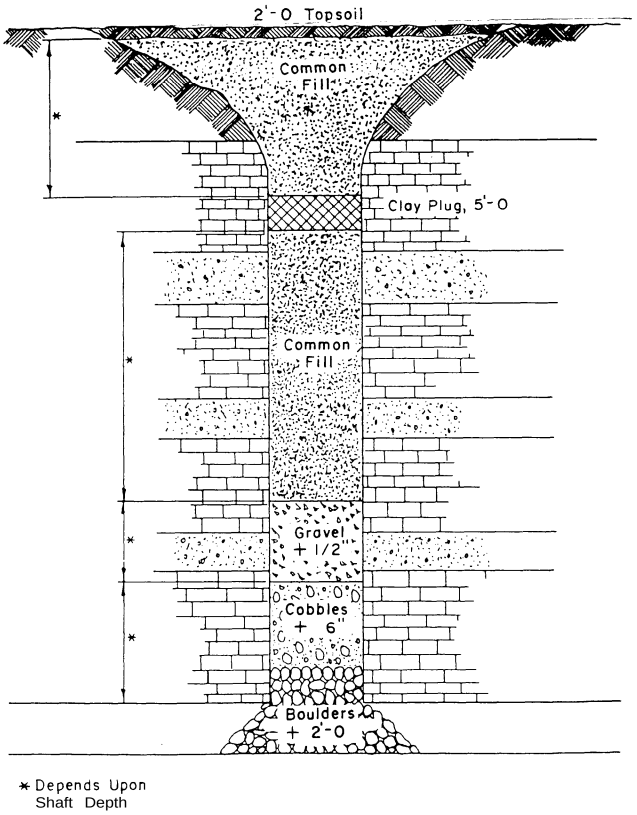


Fig. 8.7.5. Dry shaft seal (Anon., 1980b).

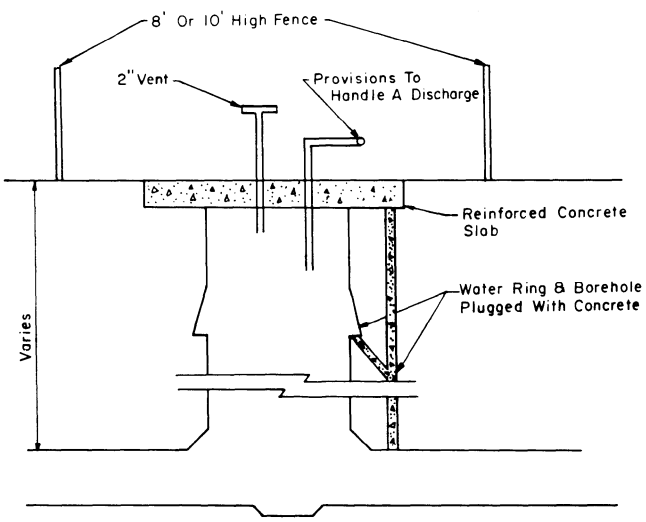


Fig. 8.7.6. Shaft seal using a concrete cap (Anon., 1985).

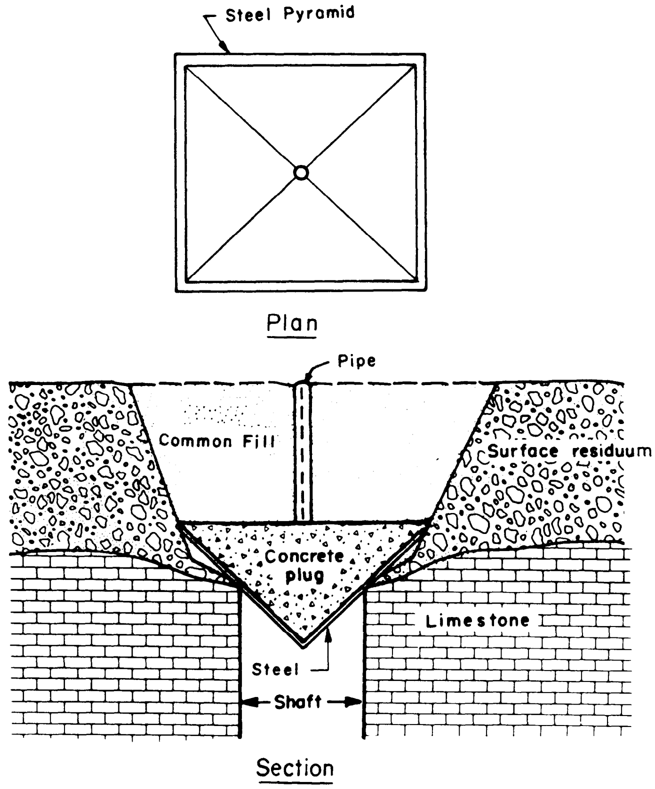


Fig. 8.7.7. Inverted pyramid shaft seal (Dressel and Volosin, 1985).

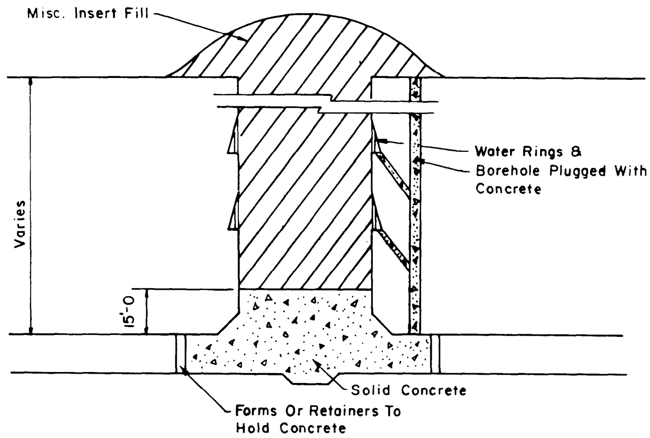


Fig. 8.7.8. Hydraulic shaft seal (Anon., 1985). Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

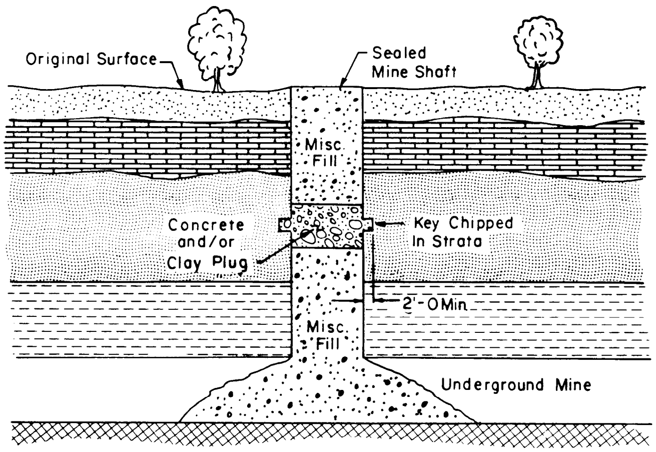
surface elevations on a coal contour map. Areas where excessive water heads may accumulate can then be projected. To convert water head H in feet to hydrostatic pressure P in psi, multiply the water head by 0.434, or $P = 0.434\ H$. In SI units, P in kPa = 9.82 H in meters.)

3. The concrete for constructing the bulkhead must be properly mixed and placed to achieve acceptable strengths upon curing.

4. Anchorage of the bulkhead to mine roof, ribs, and floor is important and depends on design as well as on strata type and condition. Some design methods rely on the strength of the concrete bearing against the irregularities in the rock surface to provide anchorage. Others require the excavation of trenches.

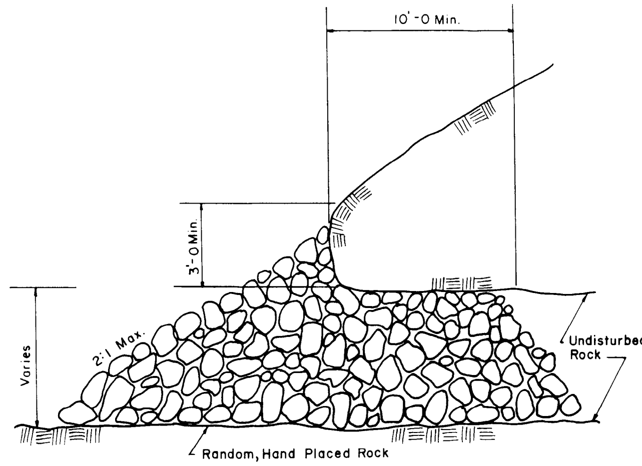
5. Adequate pressure grouting of the immediate strata surrounding the bulkhead is probably the most significant factor in the bulkhead's long-term performance. Deterioration of the anchoring strata by acid-water permeation is a major structural concern, especially if large pressures are anticipated over the life of the bulkhead.

At coal mines having drift entrances that are being sealed using hydraulic seals, it is important that the engineer consider



Conversion factor: 1 ft = 0.3048 m

Fig. 8.7.9. Hydraulic shaft seal (Anon., 1980a).



Conversion factor: 1 ft = 0.3048 m

Fig. 8.7.11. Dry seal of an adit or drift by backfilling rock (Anon., 1988c).

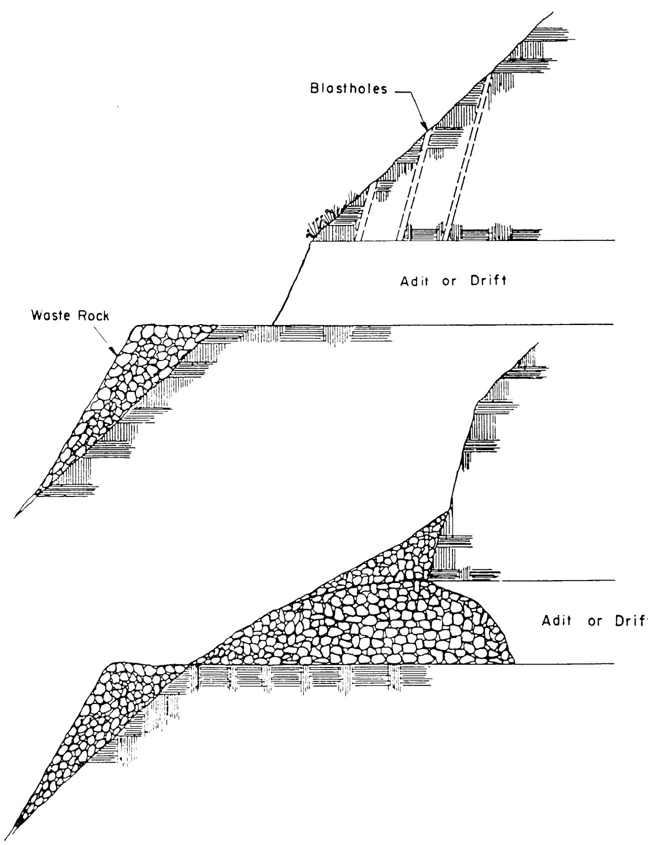
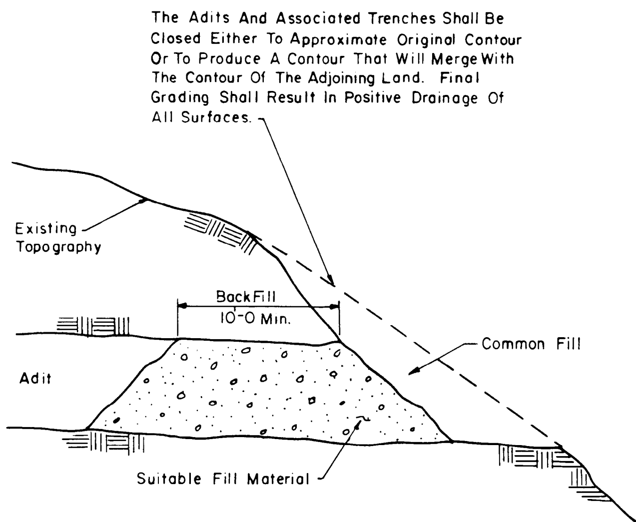


Fig. 8.7.10. Dry seal of an adit or drift by blasting (Anon., 1980a).



Conversion factor: 1 ft = 0.3048 m

Fig. 8.7.12. Dry seal of an adit or drift by backfilling (Anon., 1988c).

what maximum head the coal barriers along the outcrop can withstand without failing. The mine seal may be designed to withstand any water pressure, but the outcrop thickness or mine roof may be of such conditions to permit seepage or a "blowout." Failure could occur due to uplift of the rock strata above the coal barrier or due to lateral translation of the coal barrier.

8.7.4 ABANDONMENT

Only after the mine has been sealed and reclaimed can the mining company abandon the site. Items that must be considered prior to the mining company's legally abandoning the site include postmining land use, maintenance of the site, recovery of reclamation bond, and postmining liability.

8.7.4.1 Postmining Land Use

Permits for new mines require that the planned land use after the mine closes be identified. Normally, the land is returned to a similar premining use or an improved use. After closure of the mine and prior to completely abandoning the site, the mining company must prove to the regulatory agency's satisfaction that the land use is as it was planned to be.

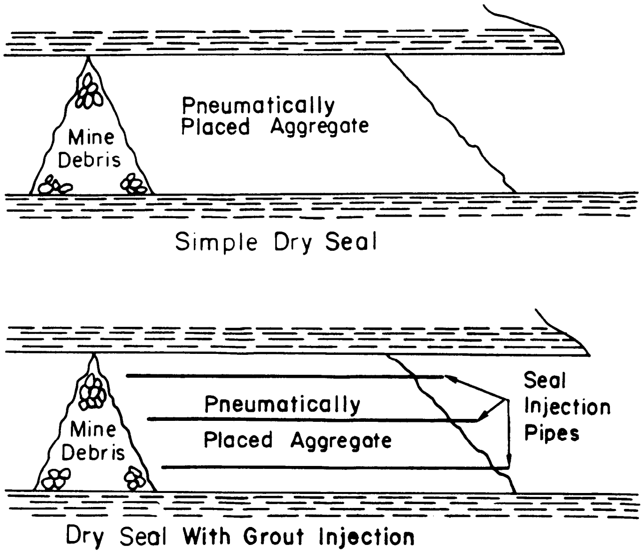


Fig. 8.7.13. Dry seal of an adit or drift by pneumatically placing aggregate (Roberts and Masullo, 1986; *permission: Coal Age*).

Alternative postmining land uses are often requested by mining companies. These alternate uses should be consistent with the land use planning of the local government. If the local land use planners support the alternative land use, no difficulty should be expected in obtaining approval from the mining regulatory agencies, as long as there are no detrimental environmental effects with an alternative use.

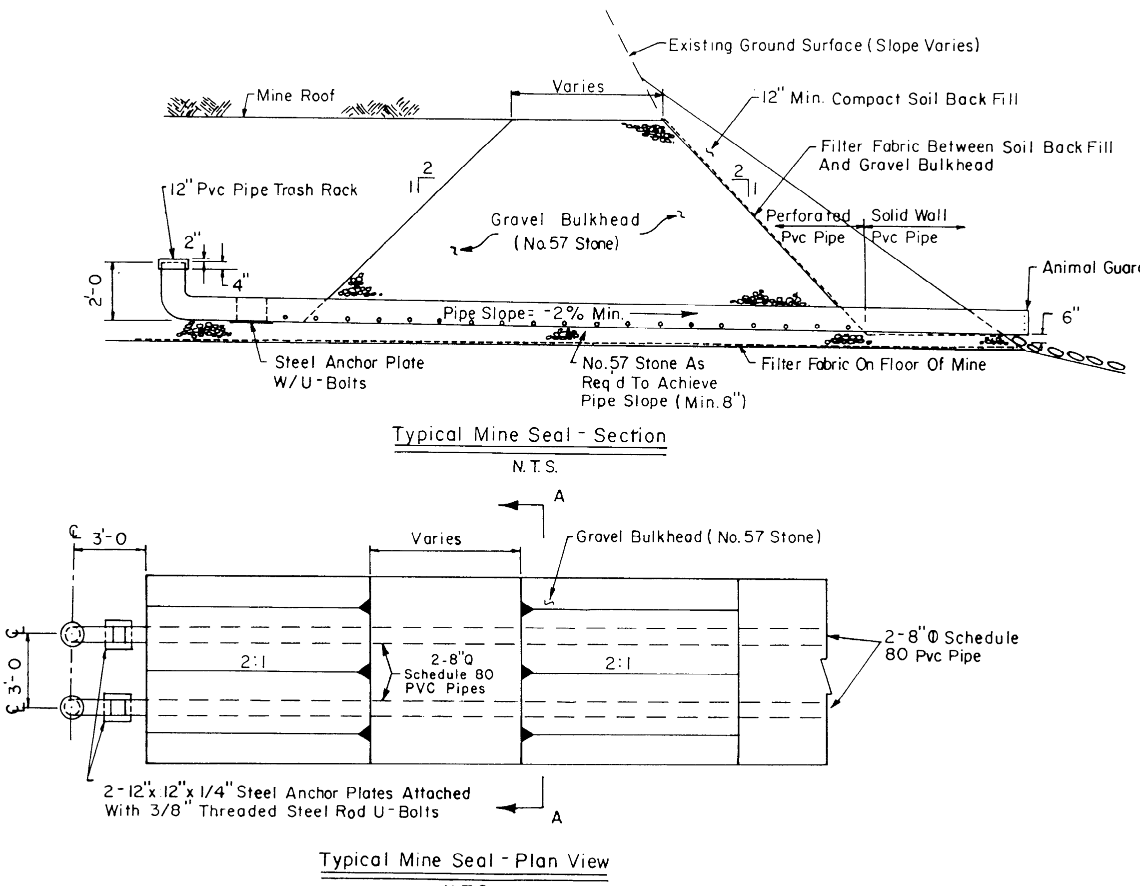
8.7.4.2 Maintenance

After reclaiming a mine site, there will probably be several years of maintenance activity required. This will include periodic inspections of the site to verify that the reclamation is effective. Inspection activities include

1. Verification that mine seals are effective.
2. Cleaning out of sediment and erosion control structures, primarily ponds and ditches.
3. Verification that water discharges are within the permitted effluent limits.
4. Regrading and reseeding of areas, as required.

8.7.4.3 Bond Release

To obtain an operating permit, most mines are required to post a reclamation performance bond that is to be used by a regulatory agency to reclaim an area in the event of forfeiture.



Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm

Fig. 8.7.14. Wet seal of an adit or drift (Anon., 1988a).

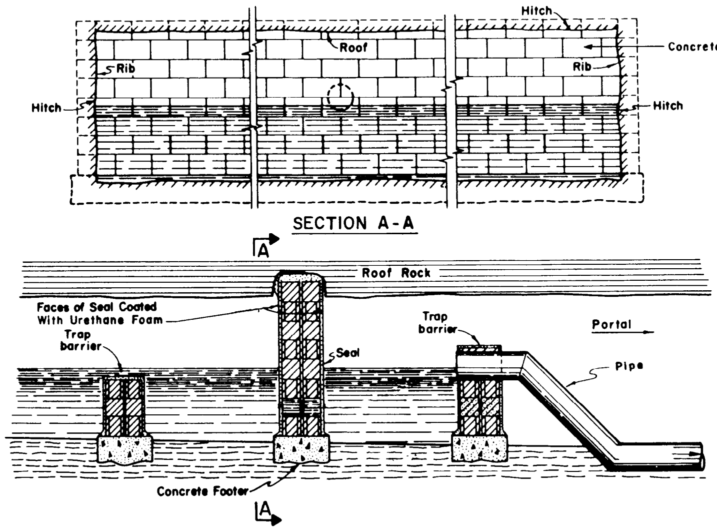


Fig. 8.7.15. Bureau of Mines air seal of an adit or drift (Moebs and Krickovic, 1970).

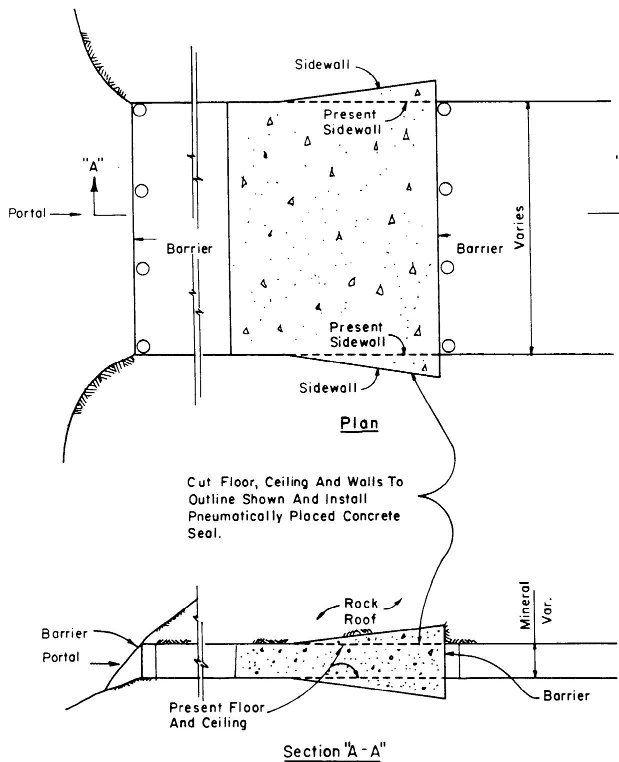
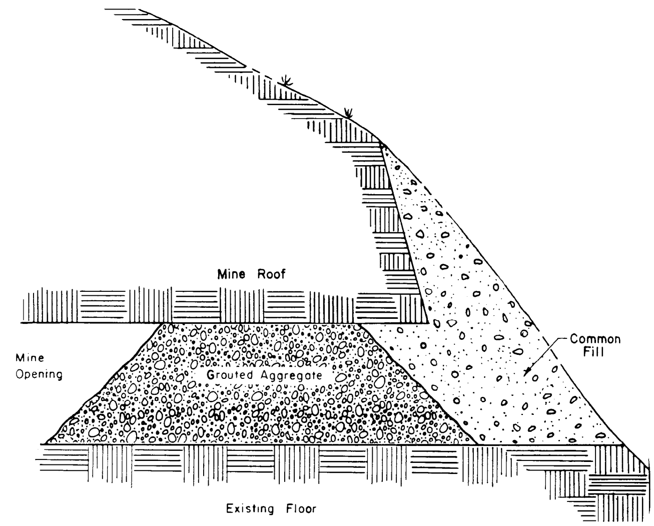


Fig. 8.7.16. Hydraulic seal of an adit or drift using gunite (Anon., 1973).



NOTE: Grout Roof Ribs and Floor if Needed

Fig. 8.7.17. Hydraulic seal of an adit or drift using grouted aggregate.

When a mining company has successfully reclaimed the closed mine (or a portion thereof), it may apply for the release of the applicable bond. OSMRE regulations for coal mines specify that the bond is not to be released until the reclamation activity has been completed and revegetation is successful. Verification that the revegetation is successful can often take several growing seasons.

Each regulatory agency may have its own procedure to obtain the release of a reclamation bond. An example of a bond release procedure (that can be considered typical) is the following procedure utilized by the Pennsylvania Dept. of Environmental Resources (Anon., 1985).

On completion of the appropriate stage of reclamation, the operator may file a completion report and request a bond release. Completion reports may be filed only when the appropriate stage of reclamation is completed. The completion report may be filed on either a designated portion or all of the permit area, as appropriate, and only at those times of the year which permit the Department to properly inspect the area.

Bond Release Procedure—Step 1. Action: The application must be examined for completeness, then logged and distributed for office and field review. (If the application is submitted during times of the year when an inspection cannot be made, the operator must be notified that the application is being held until the inspection can be made.)

A field inspection notice must be issued to the operator and the surface owner, agent, or lessee. The notice must be issued and received prior to the field inspection and within a reasonable time of the inspection so that the operator or landowner may participate in the inspection. Notice to the surface owner, or

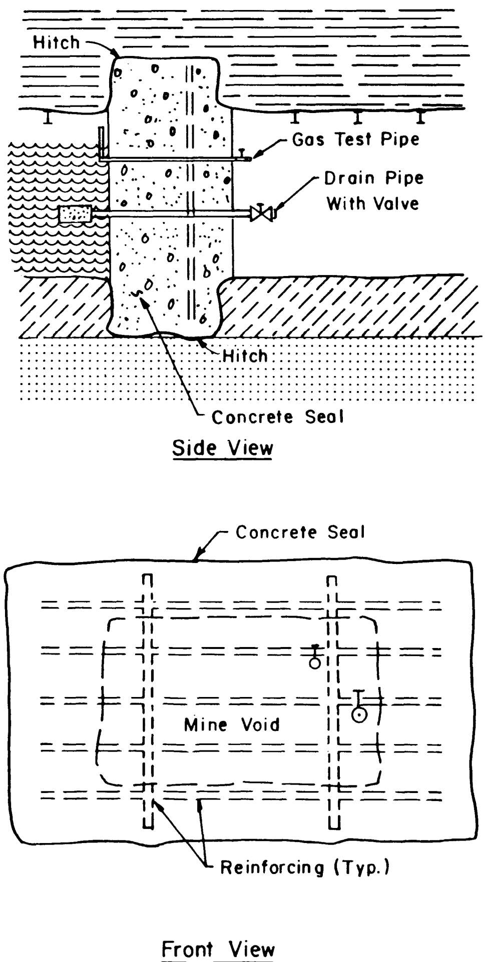


Fig. 8.7.18. Hydraulic seal of an adit or drift using a single concrete bulkhead (Garcia and Cassidy, 1938).

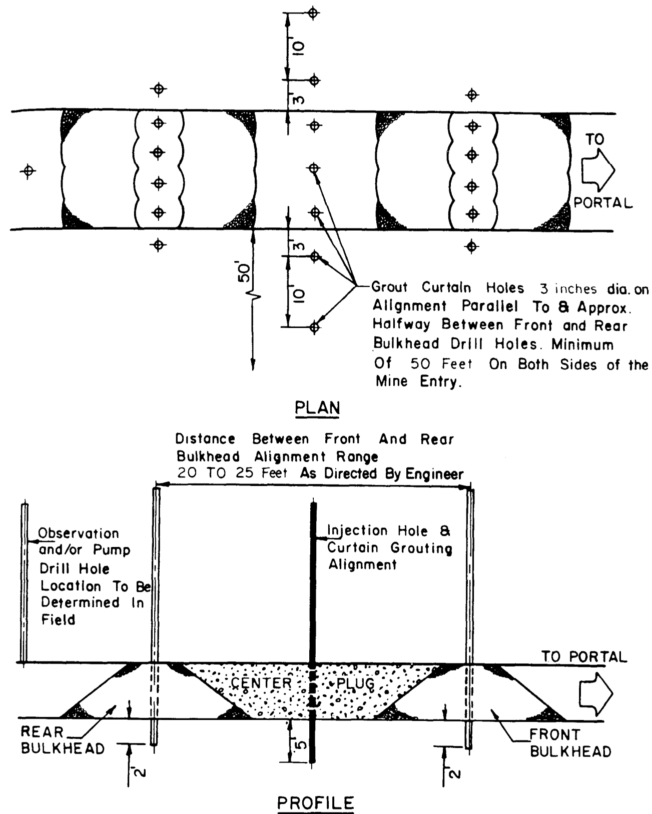


Fig. 8.7.19. Hydraulic seal of an adit or drift using a grout curtain with double bulkhead seal (Foreman and McLean, 1973). Conversion factor: 1 ft = 0.3048 m.

agent or lessee of the surface owner, will be based on the completion report.

Step 2. Action: Verify that the application, public notice advertisement and public notice letters are proper.

Step 3. Action: Field review of area. When possible, the field inspection should be scheduled at the same time as the monthly inspection and also documented as a monthly inspection.

Step 4. Action: Conduct an informal conference if requested.

Step 5. Action: Finalize review of the completion report and issue the final determination to the operator. A copy of the final determination must also be sent to the local municipality, each party that submitted written comments or objections, and each party that attended the informal conference. If a bond release is approved, the amount of bond release must be verified.

8.7.4.4 Postmining Liability

Even after the mine has closed and reclamation activities are completed, the mining company's liability may not end. Two primary potential problems are water treatment and subsidence. Each mine's development and operation plan and mining methods should be instituted so that they limit the mine's susceptibility to postclosure costs. The costs can be substantial and could affect the economic condition of the mining company.

Water Treatment: If there is water being discharged from the mine site that does not meet the effluent limitations of the current applicable regulations, treatment of the water discharge is required. This treatment may be necessary for many years or even perpetually. Because this cost can be great, future mining methods should include planning either to eliminate a postmining discharge or to assure that the discharge will meet the EPA effluent limitations. Metcalf and Eddy (Anon., 1979) and Reynolds (1982) can be used to assist engineers in the design of water treatment facilities. The US EPA (Anon., 1983) has also published a design manual on the neutralization of acid mine drainage.

Subsidence: OSMRE regulations require coal mine operators to "adopt measures consistent with known technology which prevent subsidence from causing material damage to the extent technologically and economically feasible, maximize mine stability, and maintain the value and reasonably foreseeable use of surface lands; or adopt mining technology which provides for planned subsidence in a predictable and controlled manner. Nothing in this part shall be construed to prohibit the standard method of room and pillar mining."

Although this regulation applies to coal mines only, underground mine operators of noncoal mines may want to do similar planning if they deem their specific situation warrants it. If subsidence does occur following mining and causes material damage to the surface, OSMRE regulations further require the operator to

"1. Correct any material damage resulting from subsidence caused to surface lands, to the extent technologically and economically feasible, by restoring the land to a condition capable

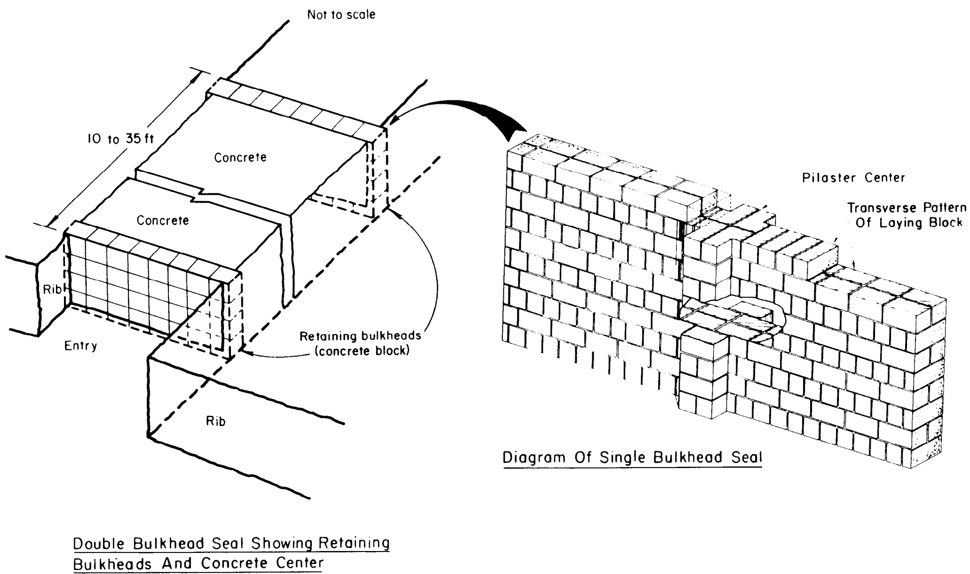


Fig. 8.7.20. Hydraulic seal of an adit or drift using a double bulkhead (Chekan, 1985). Conversion factor: 1 ft = 0.3048 m

of maintaining the value and reasonably foreseeable uses which it was capable of supporting before subsidence; and

"2. To the extent required under applicable provisions of state law, either correct material damage resulting from subsidence caused to any structures or facilities by repairing the damage or compensate the owner of such structures or facilities in the full amount of the diminution in value resulting from the subsidence. Repair of damage includes rehabilitation, restoration, or replacement of damaged structures or facilities. Compensation may be accomplished by the purchase prior to mining of a noncancellable, premium-prepaid insurance policy."

This liability to pay for subsidence-caused surface damage does not leave the mine operator after abandonment of the mine.

Subsidence-related underground coal mining is the inevitable result of high-extraction mining practices and the occasional (relatively infrequent) result of partial extraction mining practices. Mine operators can categorize various portions of their mine into the following:

Planned Subsidence—Represents lowering the ground surface in a manner predictable (within limits) as to areal extent, amount of subsidence, and amount of ground surface distortion as a result of appropriate mine design and mining procedures. Planned subsidence is the result of high-extraction technologies, such as longwall and pillar retreat in coal mines.

Unplanned Subsidence—Represents lowering of the ground surface in a manner that cannot be predicted as to areal extent, amount of subsidence, or amount of ground surface distortion, as a result of failure at mine level of the overburden support system (pillars/mine roof/mine floor) or as a result of the action of other unanticipated causes, such as the piping of unconsolidated sediments into the mine.

Planned Subsidence Prevention—Can be accomplished by utilizing a mining method that provides for permanent ground support. When the percentage extraction from a mine panel is low to moderate, the loads imposed upon pillars by the overburden are generally small in relation to the size of the pillars. In this situation, subsidence of the ground surface is virtually nil and will remain so over the long term.

In planned subsidence or planned subsidence prevention areas, the mining company should be confident that no long-term liability exists. In areas considered to have unplanned subsidence potential, subsidence may occur at an unpredictable time in the future. If any areas in this category exist in a mine being abandoned, and the company has some control over future land use, the company may wish to consider limiting land uses or requiring subsidence resistant designs for future building developments. Insurance against future claims could also be purchased.

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Part III
Unit Operations of Mining

Section 9 Production Operations

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Chapter 9.0
BASIC TASKS IN THE PRODUCTION CYCLE

LEE W. SAPERSTEIN

9.0.1.1 Application

At the heart of the mining process are a number of basic and fundamental tasks whose regular and repeated performance constitute the mining cycle. As will be discussed presently, these tasks are the basic components of the mining system. Section 9, Production Operations, provides the mine designer and engineer with an understanding of these tasks and with sufficient design principles to allow this person to create an efficient mine system. This section is meant to be an introduction and adjunct to subsequent sections that describe specific mining systems. However, sufficient design information is included here for the independent design of mine systems that may not be described elsewhere or for subsequent verification of existing designs.

For the purposes of this section, mining subsumes excavation; thus this material applies to any excavation process. Accordingly, this section is written in general enough terms to include both surface and underground mining. It may be used also for civil engineering excavations that often involve the same steps. Obviously, the actual procedures undertaken in any specific mining system will appear to be very different from those taken in any other system. This is certainly true when comparing surface with underground methods. Yet the fundamental principles are often the same. This segment is written to explicate the basics. Wherever appropriate, the section will suggest the differences between the application process to surface or to underground mining.

9.0.1.2 Unit Operations

The phrase, *unit operations,* is used often to describe the basic tasks of any systematic process. In mining, the unit operations of production are fragmentation, loading, and haulage; the auxiliary unit operations include site preparation, ground control, transportation of personnel and supplies, provision for pumping, ventilation, and power, as well as reclamation procedures not included in the other steps. Many synonyms exist for these tasks; for example, *penetration and fragmentation* are often referred to as drilling and blasting, loading as mucking, and haulage as transportation. In particular, loading and haulage are grouped in the term *materials handing.* In this section, drilling and mechanical breakage, such as ripping or continuous mining, is included in the chapter on “Rock Breakage: Mechanical;” blasting is included in the chapter on “Rock Breakage: Explosives;” and the last two production operations are included in the chapter on “Materials Handling: Loading and Haulage.” The ancillary unit operations are described in subsequent sections.

Unit operations are often recognized more easily than they are defined. Yet there are several distinctive characteristics: a unit operation defines a process and not the subject of that process; it is often an elemental step that does not divide easily into further substeps; consequently, it often is identified with a single machine or operator; finally, even though part of a cyclical process, it often can be performed continuously. This identification with process allows the same principles to apply to seemingly widely different circumstances: the calculation of tractive effort of a rubber-tired vehicle is the same for a large, diesel-electric haul truck as it is for a battery-powered scoop used in an underground coal mine; the determination of bucket size for a large stripping dragline is done with the same steps and principles as used to size an underground load-haul-dump (LHD) machine.

9.0.1.3 Mine Cycles

Should an observer stand in one place near the mine face—the point where excavation is underway—for a period of time, it will be seen that the unit operations are performed in an orderly rotation. When a period is complete, with at least one task being seen twice, then one cycle will have been performed and the face advanced by the dimension set in the fragmentation phase. Normally, the cycle is thought to begin with preparation for drilling. Over a number of cycles, the face will advance so as to define the mine's geometry.

The repetitive nature of the unit operations means that the design or sizing of the equipment can be adjusted so as to be specific to the task, and the training of the operator can also be as specific. With appropriate overall mine design, done so as to minimize moves of equipment between adjacent cycles, efficiencies of operation that are nearly as high as with a continuous operation can be obtained. In a well-designed mine, where each unit operation is sized so as to be compatible with the successive operation and equipment moves are kept to a minimum, the work effort continues as regularly as if it were a mechanical assembly.

Tables 9.0.1. and 9.0.2. are placed here to show ideal cycles for surface and underground mining. Within each table, comments are inserted to show where a particular operation may be omitted.

9.0.1.4 Task Integration

In striving for lowest cost and maximum productivity, the mine designer follows several simple maxims.

1. The productivity of each unit operation, measured in units of production per unit of time, is matched with all other operations linked by a particular cycle.

2. More than one machine is used, as in a fleet, if machines with less than the desired capacity are less expensive or more flexible than a single machine with the desired capacity.

3. Mine layout is made so as to minimize nonproductive movements of equipment.

4. Unit operations are linked wherever possible to provide for continuous flow of material, providing that the resulting operation is not more cumbersome or less reliable than the separate units. This linkage creates a fundamentally different mining system: one that is called continuous. These systems, called closed-connected when each component is physically linked to another, minimize the delays associated with changing the respective unit operations in and out of the face area. Unlike open-connected systems, however, a failure of any component will shut down the entire operation. Open-connected systems may be continuous, but do not have physical ties among the components. Descriptions of continuous systems are found in the respective chapters that describe applications.

The final chapter of this section, "Cycles and Systems," is included to help the mine designer assemble unit operations into an optimum system that has a cycle time coincident with or convenient to shift times or other working periods.

Table 9.0.1. Surface Mining Cycle

1. Install Erosion and Sediment Controls: this Step and Step 2 are performed only when mining new ground. Proceed to Step 3 if entering into a previously mined zone,
   a. Ditches, diversions, terraces, and down drains.
   b. Ponds.
2. Remove Topsoil
   a. Place in temporary, environmentally sound storage piles.
   b. Place directly onto restored mine land.
3. Prepare Drill Bench: if the material is sufficiently soft to not need blasting, then go to Step 6.
   a. Level bench with bulldozer.
   b. Inspect highwall for potential rockfalls and scale as necessary.
   c. Survey and lay out drillholes.
4. Drill Blastholes
5. Blast
   a. Load explosives into boreholes.
   b. Connect detonators so as to give desired firing order and times.
   c. Fire when safe to do so.
6. Load Fragmented Material: soft material will be fragmented by the cutting action of the loading machine.
   a. In quarry and open pit mining, loading will be into haulage conveyances.
   b. In large-scale area strip mining, the loading machine also performs the haulage (called casting in this case) function.
7. Haul Material
   a. Ore, coal, or other valuable material goes to subsequent processing.
   b. Waste goes to permanent storage.
      i. Off-site storage usually accompanies open pit mining.
      ii. Restored lands characterize strip mining.
   c. Haulage can be local, as with rubber-tired vehicles used in a gathering mode, or long distance.
   d. Haulage can be cyclical or continuous.
      i. Cyclical: trucks or load-and-carry vehicles.
      ii. Continuous: conveyor belts or pipelines.
8. Dispose of Waste or Overburden
   a. Overburden Storage.
      i. Waste piles.
      ii. Rebuild mine lands.
   b. Preparation Wastes.
      i. Waste disposal ponds.
      ii. Integrate into overburden storage.
9. Restore Topsoil
   a. Onto restored mine lands.
   b. Onto permanent waste piles.
10. Reclamation: may not be done with every cycle, but according to the season or when sufficient acreage has been restored to be efficient; in no case should restored lands be left uncovered.
    a. Revegetate.
       i. With temporary cover.
       ii. With permanent cover
    b. Maintenance until permanent succession is achieved.
    c. Remove temporary drainage controls
11. Return to Step 1.

Table 9.0.2. Underground Mining Cycle

(This table is intended for drill and blast cycles and not for continuous mining)

1. Enter Work Place After Previous Blast
   a. Check that ventilation has removed blasting fumes.
   b. Provide for dust suppression.
   c. Inspect back and ribs for loose material.
2. Install Initial Ground Support
   a. Scale, bar down, or otherwise remove loose material.
   b. Install temporary or initial ground support if it is not provided automatically by the loading machine.
3. Load Fragmented Material
   a. Onto gathering haulage.
   b. Directly onto main haulage.
4. Haul Material
   a. In gathering systems.
      i. Load-Haul-Dump
      ii. Truck
      iii. Shuttle Car
      iv. Gravity Transfer
   b. Main haulage.
      i. Truck
      ii. Train
      iii. Conveyor Belt
      iv. Pipeline
5. Install Permanent Ground Support If Needed
6. Extend Utilities If Needed
   a. Ventilation.
   b. Power (electricity or compressed air).
   c. Transportation.
7. Drill Blastholes
   a. Survey for line and grade; mark blasthole location.
   b. Drill holes.
8. Blast
   a. Load explosives into boreholes.
   b. Connect detonators so as to provide desired firing order and times.
   c. Blast when safe.
   d. Return to Step 1.

Chapter 9.1
ROCK BREAKAGE: MECHANICAL

M.C. Hood and F.F. Roxborough

9.1.1 INTRODUCTION

Almost exclusively, the methods that are used for rock excavation today employ either mechanical rock cutting tools or explosive charges. It is important, therefore, to have a good understanding of the advantages and the limitations of rock cutting with mechanical tools in order to assess both the potential for making significant technological advances using this approach and to develop a strategy for achieving these advances. Also, although many of the alternative methods for rock destruction, such as thermal or erosional, have been demonstrated as technically successful (i.e., it is possible to excavate rock using these methods), most of these techniques are impractical either because they are extremely energy intensive, or they are not cost-effective. Hence none of these alternative methods is likely to replace rock cutting with mechanical tools in the short-to-medium term (0 to 10 years). This is not to say that these other methods will not find application in rock breakage systems in this time frame. Indeed, it is most likely that the next generation of rock excavation systems will employ a hybrid cutting principle with mechanical tools assisted by one or more of these other breakage methods. Discussion of these topics occurs in the segments that follow. The main point is that future rock excavation systems are likely to continue to employ mechanical tools, and thus it is important to understand the processes of rock destruction caused by these tools.

Breaking rock by the penetration of a wedge is the basic mode of action used by virtually all rock excavation machines. There are only two essential types of mechanical rock cutting tools, *indenters* and *drag bits*. Tools in this latter category often are termed picks and these terms, drag bits and picks, are used interchangeably throughout this chapter. The distinction between the two types of tools is that an indenter breaks the rock by applying a force that is, predominantly, in a direction normal to the rock surface. In contrast, the main force applied to a sharp drag bit to effect rock breakage is in a direction approximately parallel to the rock surface. The loading geometries for both types of tools are illustrated in Fig. 9.1.1.

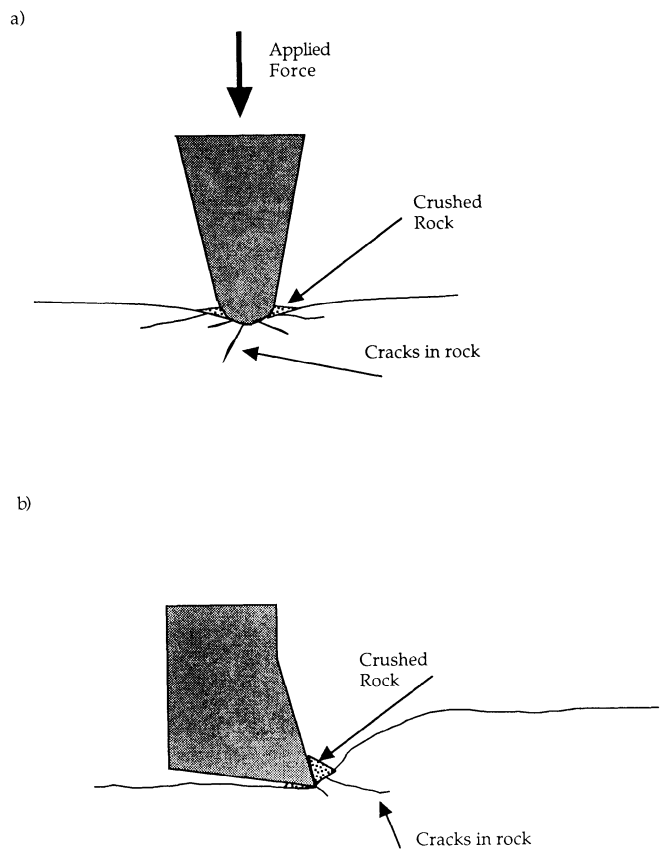


Fig. 9.1.1. Methods of rock loading by a) an indenter and b) a drag bit.

The great majority of the rock cutting tools employed today are indenters. Thus all types of roller cutters—disk cutters, rolling cone bits, etc.—break the rock in a indentation process. Similarly, all types of percussive tools, including percussion drillbits, down-hole drillbits, and high-energy impact bits, induce rock fracture by indentation. Only rotary drillbits and picks, such as those employed on coal excavation machines, break the rock by applying the main force in a direction parallel to the rock surface. Many of these various types of cutting tools are illustrated in Fig. 9.1.2.

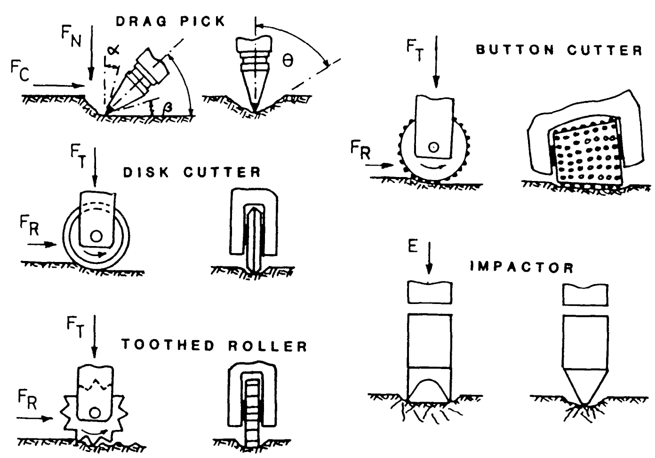


Fig. 9.1.2. Mechanical rock breaking tools.

Although indenters are more widely used than drag bits, they are less efficient; that is, an indenter requires more energy to excavate a given volume of rock than does a drag bit. This is because of the processes that lead to the formation of a major rock chip. The preponderance of the evidence points to the mode of propagation for the cracks that form the large rock chips as tensile fractures both for indenters and for drag bits. This, perhaps, is surprising because, as shown in Fig. 9.1.1, the load applied by an indenter is one of compression and that applied by a drag bit is one of shear. Details of the crack initiation and propagation processes for these two types of tools are given in the following. It is sufficient to note here that whereas a drag bit initiates a tensile fracture in the rock in a fairly direct manner, an indenter generates tensile stress in the rock by crushing a portion of the rock mass beneath the tool. This crushed material dilates and induces the tensile stresses that eventually lead to the development of a fracture in the surrounding intact rock.

Since drag bits are the more efficient type of cutting tool, it is fair to ask not only why are indenters used more widely than drag bits, but why are indenters used at all? The answer lies in the fact that the shear loading of rock by a drag bit induces bending, or tensile, stresses in the bit cutting edge. The material used for this cutting edge is brittle; generally, it is a cemented tungsten carbide. Because all brittle materials are weak in tension, this mode of loading stresses the tool in a dangerous manner that makes catastrophic failure of the bit more likely. The cutting edge of an indenter, on the other hand, can be mounted in the bit body in a manner such that this insert is loaded only, or mainly, in compression. This maximizes the strength of the tool material, which is the reason that this type of tool is widely used. Because of this limitation of tool strength, in practice today the use of drag bits is restricted to weak rock materials, such as coal or evaporites. An indentation type of tool generally is employed when drilling or cutting is carried out in any rocks stronger than these.

In fact, this discussion on the strength of the cutting tool materials really is the nub of the problem addressed in this section. Rock cutting has aptly been described as a contest between the strength of the tool and the strength of the rock. No single-strength value is appropriate for predicting the outcome of this contest. The strengths of both the rock material and the tool material depend on a number of factors, generally including:

1. Whether the material is loaded in tension or compression.
2. Whether uniaxial or multiaxial stresses are applied.
3. The sign and the magnitudes of all of the applied stresses.
4. The temperature of the material.

The reason for the higher breaking efficiency of the drag bit now is apparent. Most rocks behave as brittle materials. These materials are about an order of magnitude weaker in tension than they are in compression. Hence the energy spent by a drag bit will be low because the normal force acting on this tool is small, and thus only a small amount of work is done in rock compression by this tool. On the other hand, crushing or fine comminution of the rock requires a very large amount of energy. Since this work must be carried out in order to form rock chips with indenters, the work expended by these tools is large.

A major objective of the following discussion is to document laboratory and field experiences with various rock breaking methods in order to identify those parameters that most influence the efficiency of a breaking method.

Drills and mining machines deploy an array of one or other type of cutter on some form of powered cutting head such that a rock face is attacked by a multiplicity of cutters acting in concert. Most, but not all, cutting heads rotate about an axis disposed either parallel to or normal to the rock face. Rotary drills, continuous miners, roadheaders, shearers, bucket wheel excavators, and raise and tunnel borers all conform to this machine format. There are other machine formats such as ploughs and tractor rippers that use linear motion and external traction.

In all cases, however, rock breakage is effected by forcing a multiplicity of basically parallel and interacting wedges across a rock face to cut off a thin layer of rock. With rotary machines this process is continuous, with the cutter head moving incrementally forward as the rock is peeled off the face. It is an important feature of the machine cutting process that the wedges are made to cut, in effect, parallel to and at a relatively small depth into the exposed rock surface.

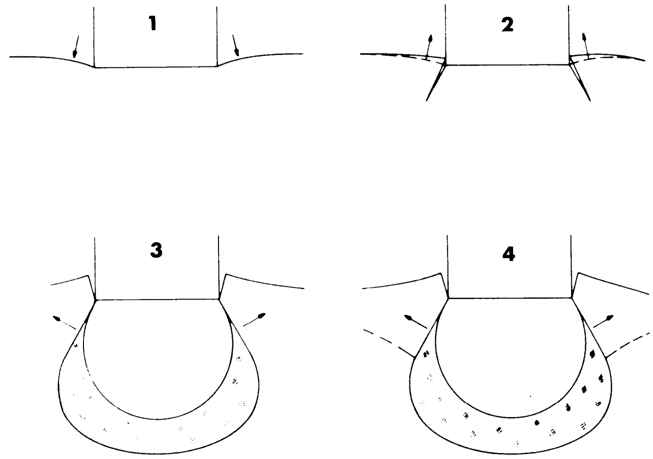


Fig. 9.1.3. Schematic representation of the failure mechanism of rock loaded by a flat-bottomed punch (after Wagner and Schumann, 1971).

These essential features have characterized machine mining from the outset of mechanization and probably will continue to dominate it for at least another generation. Research work on coal and rock cutting started with the advent of machine mining shortly after World War II and set itself the task of making the cutting process as efficient as possible. It did so by a stepwise approach to addressing the following key issues.

1. How is coal and rock broken by cutting tools operating both singly and in array?

2. What are the features of tool design and operation that characterize efficient cutting processes?

3. What are the consequences of departing from optimum design, if such exists?

4. What factors limit the application of mechanical cutting processes?

The segments that follow describe this work. For discussion of the more novel methods of rock penetration—applications as well as fundamentals—see Section 22 (e.g., Chapters 22.1, 22.3, 22.4, and 22.9).

9.1.2 ROCK:BIT INTERACTIONS

9.1.2.1 Indentation

Indentation with mechanical tools is the most widely used technique for drilling holes, ranging in diameter from a fraction of an inch (10 mm or so) to several feet (meters). With this method, the tool is placed in contact with the rock and a normal, or predominantly normal, stress is applied by the tool to the rock. The processes of fracture in brittle rock from indentation are described in this subsection.

Flat Bottomed Punches. The rock surface deforms elastically when the stress applied to a flat-bottomed indenter (punch) is at a low level (Fig. 9.1.3a). As the applied stress is increased, a Hertzian cone crack is initiated in the rock from the tool corners (Fig. 9.1.3b). As the stress continues to be increased intense comminution (crushing) of the rock takes place beneath the tool (Fig. 9.1.3c). Eventually, major fractures are initiated from the

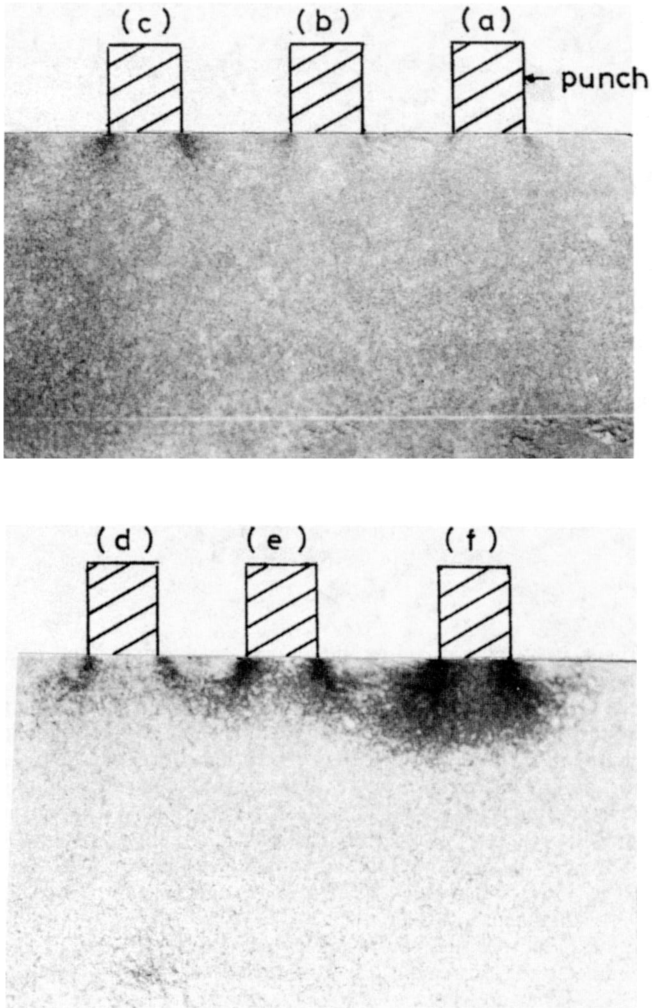


Fig. 9.1.4. Crack and microcrack development in a granite sample beneath a ½-in. (10-mm) diameter flat-bottomed punch (after Cook et al., 1984).

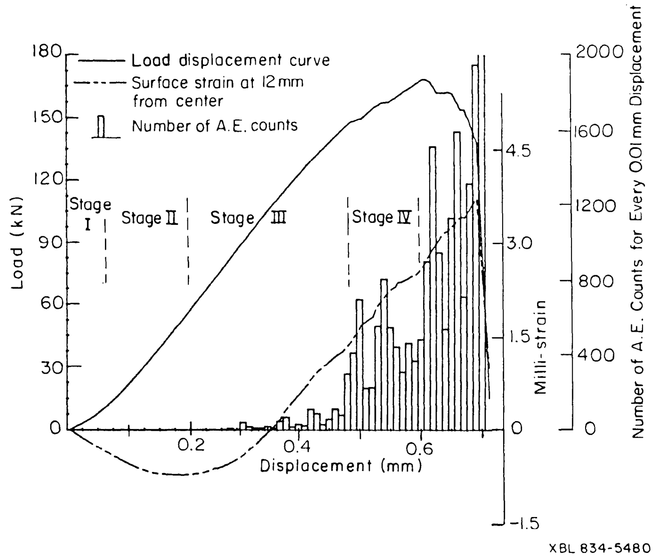


Fig. 9.1.5 Results from a typical loading cycle in granite with a ½-in. (10-mm) diameter punch, showing the punch load-displacement curve, the rock radial strain-punch displacement curve, and the number of AE events.

zone of crushed rock beneath the tool, and these propagate to the surface to form rock chips (Fig. 9.1.3d).

Cross sections through a granite sample loaded by a flat-bottomed punch are shown in Fig. 9.1.4. In this figure, the initiation of the Hertzian crack is evident in (a). As the punch load is progressively increased, the length of this crack increases in (b), (c), and (d). It is apparent from the width of the dark band indicating the crack in these photographs that this crack is surrounded by an intensely microcracked zone of rock. As the punch load is increased beyond (d), the microcracked region, indicated by the darkened region in (e) and (f), becomes much more extensive. This heavily microcracked region beneath the punch corresponds to the crushed zone in Figs. 9.1.3c and 9.1.3d.

A load-displacement curve, representative of these punch tests in granite, is given in Fig. 9.1.5. It is interesting to observe that this indentation load-displacement curve displays the same four characteristic regions observed in uniaxial or triaxial rock testing despite the obvious differences in the loading method. In Fig. 9.1.5, stage I loading corresponds to the closure of preexisting flaws in the rock. Stage II loading is the linear-elastic portion of the curve. In stage III, the rock has ceased to respond in an linear-elastic manner. In stage IV, the curve deviates substantially from linear-elastic behavior.

An understanding of the rock deformation during this loading process can be gained from inspection of the other curve and the bar chart plotted in Fig. 9.1.5. These show, respectively, values of the radial strain, measured by gages mounted on the rock surface, and the number of acoustic emission (AE) events. The energy that must be applied to extend a crack is much greater than the energy required to create new surface area on the faces of the crack. This difference is accounted for by (i) the energy consumed in the formation of the so-called process zone, this is a region where microfractures are generated in the rock ahead of the main crack, and (ii) the energy dissipated by heat and sound. The acoustic emission counts shown in Fig. 9.1.5 are measurements of the acoustic energy generated within the rock during the loading process. The number of AE counts shows the relative intensity of fracturing taking place within the rock as a function of the punch displacement.

The curve of the radial strain in Fig. 9.1.5 shows that during the initial loading, negative values of strain are observed. This indicates that the surface of the rock is in tension, corresponding to the elastic downward displacement of the rock surface shown in Fig. 9.1.3a. The maximum tensile strain takes place at the beginning of stage III on the loading curve. At this point, the Hertzian crack is initiated (Fig. 9.1.3b). As the punch load increases, this crack propagates, and the rock surface around the punch rebounds (Fig. 9.1.3b). This reduces the measured tensile radial strain. The punch continues to penetrate the rock and the radial Poisson expansion of the punch begins to exert a compressive stress in the rock at the surface. Consequently, the measured radial strain becomes positive, or compressive.

At the end of stage III loading, the rock damage has progressed to the point shown in Fig. 9.1.4e. It can be seen that the AE counts begin at some point during the stage III loading, roughly corresponding with the initiation of the Hertzian crack. The rate of fracture damage, as measured by these counts, remained at a low level until nearly the end of stage III when a sudden increase in fracturing activity can be seen. During stage

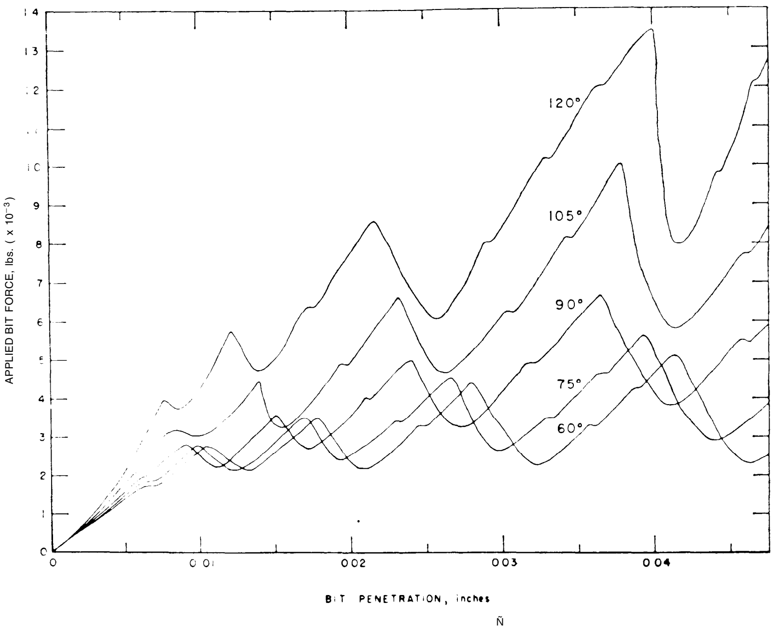


Fig. 9.1.6. Force indentation curves in Charcoal Gray Granite using a wedge indenter (after Reichmuth, 1963).

IV, an intense amount of microfracturing takes place beneath the punch (Fig. 9.1.4f), and this is reflected by the rapid increase in AE count activity. These experiments were conducted under displacement control in a stiff testing machine; consequently, it was possible to record information beyond the peak of the load-displacement curve until a rock chip formed at the surface. It can be seen that a quiescent period in AE activity took place immediately prior to the peak punch load. The AE counts were observed to increase dramatically immediately past this peak value, again indicating that major fracturing activity was taking place. Almost certainly this activity reflects the formation and propagation of the major crack (shown as a dashed line in Fig. 9.1.3d) that was to form the rock chips. The value of the measured, now compressive, strains increased in an approximately linear manner to a peak value at which point the developing crack forming the rock chip passed beneath the gage and relieved the strain.

One of the interesting findings from these indentation tests is that the stress necessary to cause the formation of rock chips decreases as the size of the punch increases (Wagner and Schumann, 1971; Cook et al., 1984). Cook and his coworkers attributed the reason for this measured change in strength with punch size to the size effect on rock strength.

Wedge Indenters: In practice, many indenters used for effecting rock breakage are wedge-shaped. Furthermore, the indenter is not pressed only once onto a pristine rock surface—usually the process is repeated many times. For example, a percussion drill bit repeatedly indents the bottom of the hole until the desired hole depth is achieved. This means that for the second loading (that is, after the formation of the first rock chips) and for all subsequent loadings, the stress applied to the intact rock to form new chips is transmitted from the indenter through a zone of crushed rock from the previous loadings. This results in an increase in the indenter (bit) force for each loading. One of the first investigators to study this phenomenon was Reichmuth (1963), and some of his results, using wedges with a variety of wedge included angles, are given in Fig. 9.1.6. Paul and Sikarskie (1965) developed a model to describe the locus of the peaks of the curves shown in this figure. This locus is the upper bound of the chipping force. The model, summarized below, sheds light on the mechanics of the rock breakage process.

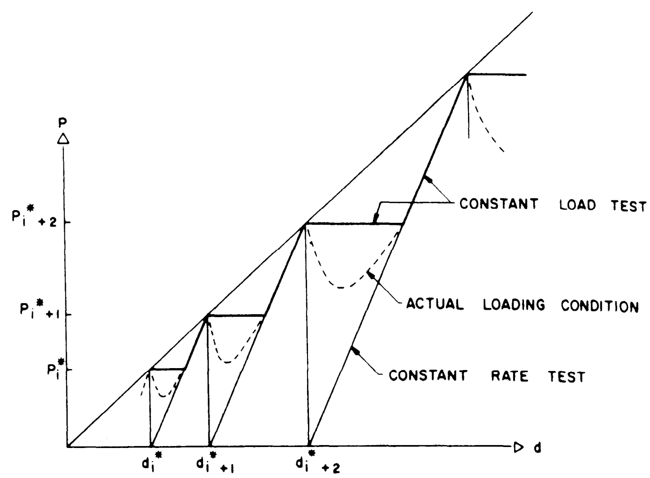


Fig. 9.1.7. Idealized loading cycles for a wedge indenter (after Paul and Sikarskie, 1965).

Paul and Sikarskie assume that the relationship between load and displacement during the loading cycle is linear and that the slope of this line, k, remains the same for the second and all successive cycles. Their model does not address the initial loading cycle. The locus describing the upper bound of the chipping force has a slope K^*. These parameters are illustrated in Fig. 9.1.7. In this figure it can be seen that the response of the loading curve between chipping cycles depends on the method of loading used. If the bit is pressed into the rock at a constant rate, then the bit force would be expected to fall to zero between cycles. On the other hand, if a constant force is applied to the bit, when a chip forms the loading system will cause the bit to advance rapidly into the rock while maintaining the load at the previous peak value. Most practical loading systems are closer to constant load than to constant displacement rate; however, these systems generally are unable to respond quickly enough to maintain the load constant immediately following the formation of a chip. Consequently the actual loading condition is more likely to be

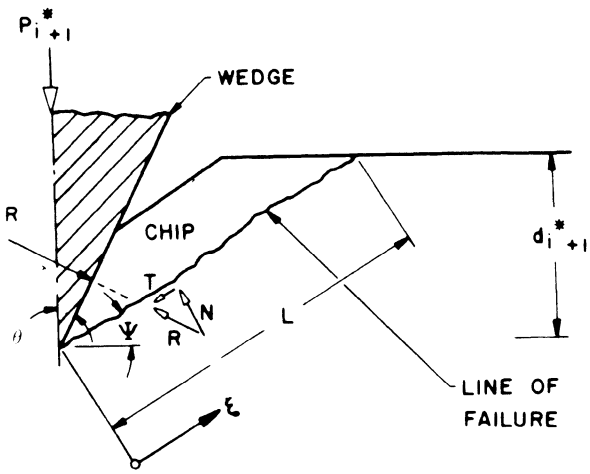


Fig. 9.1.8. Stresses acting along failure plane (after Paul and Sikarskie, 1965).

that illustrated by the dashed line in Fig. 9.1.7. It can be seen that this behavior is similar to that measured by Reichmuth (Fig. 9.1.6).

A diagram illustrating the physical representation of the breaking process is shown in Fig. 9.1.8a; the mathematical idealization is shown in Fig. 9.1.8b. The model assumes that failure along the fracture surface takes place when the Coulomb criterion is exceeded:

$$|\tau| - \mu\sigma = S_0 \tag{9.1.1}$$

where τ is the shear stress acting on the incipient fracture surface, σ is the normal stress acting on the incipient fracture surface, *m* is the coefficient of internal friction, and S_0 is the inherent shear strength of the rock.

The stresses acting along this failure plane are calculated by resolving the forces transmitted by the wedge to the rock in the manner shown in Fig. 9.1.8. When this is done and the failure criterion is applied, then the following relationship for K^* is derived:

$$K^* = \frac{P^*_{i+1}}{d^*_{i+1}} = 2C_0 \frac{\sin\theta\,(1 - \sin\phi)}{1 - \sin(\theta + \phi)} \tag{9.1.2}$$

where P^*_{i+1} is the wedge force at the formation of the 1 + *i th* chip, d^*_{i+1} is the wedge penetration at the formation of the 1 + *i th* chip, C_0 is the uniaxial compressive strength of the rock, *q* is the half-angle of the wedge, and ϕ is the angle of internal friction of the rock.

Since θ, ϕ, and C_0 are constant for a given wedge in a given rock type, this model predicts that K^* = constant; in other words, the locus of the upper bound of chipping is a straight line.

Also, substituting $P^*_{i+1} = k[d^*_{i+1} - d_i^*]$ into Eq. 9.1.2 yields

$$P^*_{i+1} = \frac{kK}{k - K} d_i^* \text{ and } d^*_{i+1} = \frac{k}{k - K} d_i^* \tag{9.1.3}$$

Hence k and d_i^* can be determined from one loading cycle. This permits the calculation of K^* as well as the loads and the displacements at all the succeeding points where chipping occurs.

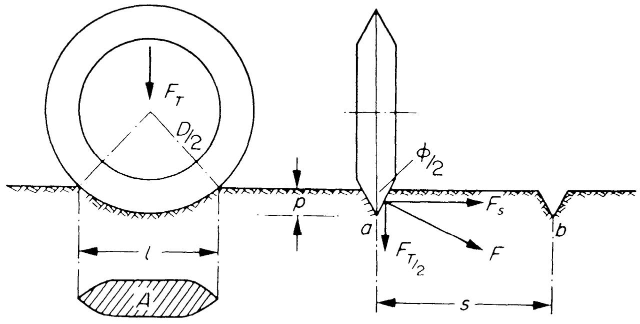


Fig. 9.1.9. Model for disk cutting (Roxborough and Phillips, 1975a).

This model overpredicts the force necessary for chipping. Recent work by Peng (1986) has shown that this problem can be overcome if an allowance is made for the effective decrease in rock strength caused by the increased area of contact between the wedge and the rock with each successive chipping cycle. Peng used a Weibull analysis to account for this effective change in rock strength. This approach yields

$$P^*_{i+1} = Kd^*_{i+1}((m - 1)/m) \tag{9.1.4}$$

where *m* is the Weibull parameter.

When m = ∞, this relationship is the same as Eq. 9.1.2. In general, however, when m < ∞, the locus of the force-displacement curve for the upper bound of chipping is not a straight line, and the model predicts that along this locus the force at any give, displacement is less than that obtained from Eq. 9.1.2. The value of m for a particular wedge in a particular rock type is obtained experimentally (Peng, 1986).

Models for Disk Cutters: Evans (1974) used essentially the same argument as Paul and Sikarskie in developing a model for rock breakage using disk cutters. Evans assumed that the cross section of a disk cutter is a wedge and derived the following relation:

$$\frac{F_T}{d} = 2\left(\frac{S_0 \cos\phi \sin(\theta + \phi_f)}{\sin^2\left(\frac{\pi}{4} - \frac{1}{2}(\theta + \phi_f + \phi)\right)}\right) \tag{9.1.5}$$

where F_T is the *thrust force on the disk* necessary to form a rock chip, *d* is the depth of cut, and ϕ_f is the friction angle between the wedge and the rock.

In a slightly rearranged form, this expression is the same as Eq. 9.1.2 with $F_T = P^*_{i+1}$ and with the term ϕ_f included to account for the friction between the wedge and the rock.

A different approach for predicting both this thrust force and the orthogonal force in the direction of cutting, the *disk rolling force,* was taken by other workers (Crow, 1975; Ozdemir et al., 1976; Roxborough and Phillips, 1975a; Roxborough and Phillips, 1975b). These researchers assumed a constant value for the indentation strength of a rock using a disk cutter. The area of the disk cutter in contact with the rock then can be calculated from considerations of the depth of rock penetration by the disk and the disk geometry. The product of this contact area at a given depth and the indentation strength then yields the thrust force acting on the disk. This approach as developed by Roxborough and Phillips (1975a, 1975b) is illustrated in Fig. 9.1.9.

The area of the disk in contact with the rock is

$$A = 2dL \tan \theta \qquad (9.1.6)$$

where L is the chord length of the disk in rock contact = $2\sqrt{Dd - d^2}$, d is the depth of disk penetration, D is the disk diameter, and q is the half-angle of disk.

Thus the thrust force F_T required to penetrate the rock to a depth d is

$$F_T = 4dC_0 \tan\theta \sqrt{Dd - d^2} \qquad (9.1.7)$$

Roxborough (1978) noted that for a free rolling disk the resultant force must pass through the center of rotation, in which case

$$\frac{F_T}{F_R} = \sqrt{\frac{D - d}{\mathrm{d}}} \qquad (9.1.8)$$

where F_R is the disk rolling force (Fig. 9.1.13)

Thus,

$$F_R = 4C_0 d^2 \tan \theta \qquad (9.1.9)$$

Eqs. 9.1.7 and 9.1.8 are plotted together with cutting force data collected by several workers in Fig. 9.1.10. It can be seen that the agreement between theory and these results is tolerable. What probably is more significant, however, is that the predicted trends in F_T and F_R with each of the disk variables are found to closely parallel measured trends. The implications are that the geometrical basis of the equations is correct but that the stress conditions applying over the disk contact area may be more complex that has been supposed.

Ozdemir et al. (1976) observed that the projected area of the disk in rock contact is given by Eq. 9.1.6 only when the disk is pressed in a quasi-static manner onto a plane surface. These authors argued that during the cutting operation, the contact area is only half of this value. In developing their model, these authors argued that the magnitude of F_T depends in part on the indentation strength of the rock and in part on that shear component of this force that causes chips to form between adjacent grooves. The following equations were derived:

$$F_T = \tan\theta \sqrt{Dd^3} \left[\frac{4C_0}{3} + 2S_0 \left(\frac{s}{d} - 2 \tan\theta\right)\right] \qquad (9.1.10)$$

$$F_R = \frac{\tan\theta \; [d^2C_0 + 4S_0\Phi(s - 2d \tan\theta)]}{D(\Phi - \sin\Phi \cos\Phi)} \qquad (9.1.11)$$

where Φ is arc $\cos\left(\frac{D - 2d}{D}\right)$ and s is the spacing between adjacent grooves.

Eqs. 9.1.10 and 9.1.11 are reported to agree reasonably well with measured data.

In contrast to this unified model of Ozdemir et al. (1976) Roxborough and Phillips (1975a) considered the interaction between disks as a separate issue to the initial disk penetration. They argued that F generated by an unrelieved disk would have the potential to shear off a rib of rock extending laterally from the disk if an adjacent groove was present. The maximum width of this rib would be determined by F_T. Accordingly, they proposed the simple model, illustrated in Fig. 9.1.9, from which they determined:

$$\frac{s}{d} = \frac{C_0}{S_0} \qquad (9.1.12)$$

Roxborough (1988) cites experimental data that accord reasonably well with Eq. 9.1.12. Other workers, notably Snowden et al. (1982), have published results that do not agree with this expression. Several workers have argued that this difficulty arises because these two models, by Ozdemir and his co-workers and Roxborough and Phillips, make the incorrect assumption that lateral breakage between grooves occurs because of shear failure. These workers argue that, on the basis of the observed failure surfaces as well as the knowledge that rock is stronger in shear than it is in tension, the chips between grooves are formed by tensile crack growth. Fenn (1985) points out that rock chips formed by multiple disks cutting adjacent grooves simultaneously are not different in appearance from chips cut by a single disk in a sequential manner. In the former case, it is apparent that a shear stress is not exerted along the fracture plane. This calls into question whether a shear failure criterion is appropriate for this fragmentation process.

Lindqvist and Ranman (1980) have developed a model for tensile failure caused by disk cutters. This model assumes that only a small fraction of the thrust force exerted by the disk is directed laterally, in other words, that most of this force is directed normally into the rock in a manner analogous to a flat-bottomed punch. The tensile stresses induce cracks that initially run parallel to the rock surface. If the spacing between grooves is sufficiently small, these tensile cracks will propagate between the grooves rather than up to the rock surface. Further, these authors argue that the elastic deformation of the surface will tend to cause the formation of chips with a convex upper surface (Fig. 9.1.11). This indeed is a feature of chip geometry that frequently is observed. Their predictive equation linking chip geometry with disk forces and rock properties is

$$F_T = \frac{2.09 \; T_0 D s^4}{16t^3(1 - \upsilon^2)} \qquad (9.1.13)$$

where T_0 is the rock tensile strength, υ is the Poisson's Ratio, and t is the chip thickness.

Results from experiments using multiple disks (i.e., more than one disk cutter on a single hub) cutting in granite are compared with theoretical values from this model in Fig. 9.1.12). The theory suggests that the spacing s is strongly influenced by the chip thickness, that is the groove depth, and this is often found to be the case in practice. Lindqvist and Ranman concede that the model relies on several tenuous assumptions, but they suggest that it constitutes a first simple expression for a tensile failure model for disk cutters.

These authors made other observations on chip formation, noting that a considerable quantity of crushed material is formed beneath the chip adjacent to the cutter. Also several fragments, 0.04 to 0.16 in. (1 to 4 mm) in size, were observed within about 0.4 in. (10 mm) of the disk. However, a clean single fracture with no debris was found over the rest of the distance between the disks.

Howarth (1980), Fenn (1985) and others report similar observations of a crushed zone and the associated crack development caused by disk cutters. For example, Fig. 9.1.13 shows carefully mapped crack distributions along with delineation of crushed zones taken from vertical rock sections by Fenn (1985).

The importance of the zone of crushed rock in indentation processes has long been recognized (Dalziel and Davies, 1964; Fairhurst and Deliac, 1986; Hartman, 1959; Maurer and Rinehart, 1960; Reichmuth, 1963) and as described before, Paul and

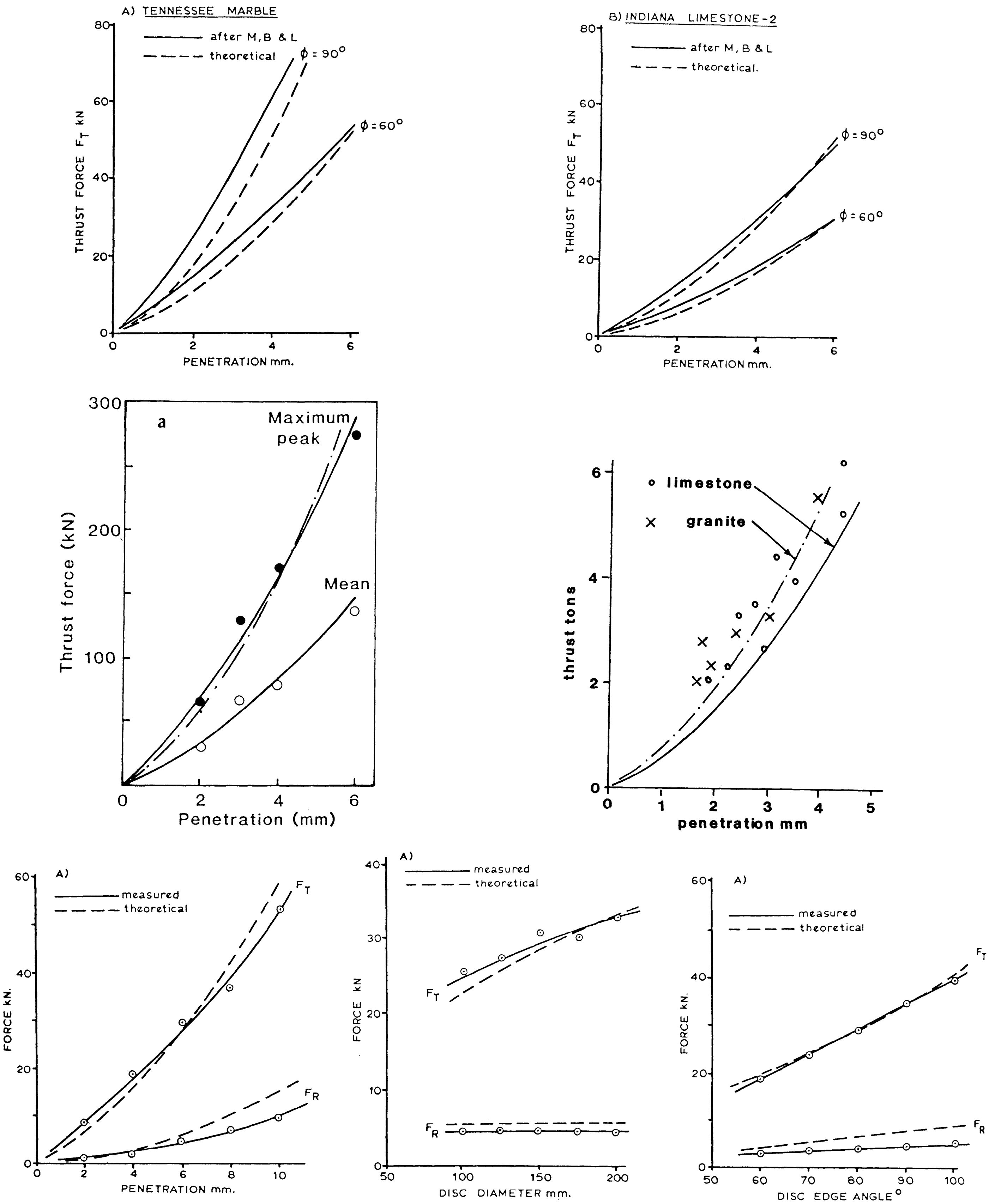


Fig. 9.1.10. Comparison of model for disk cutting developed by Roxborough and Phillips (1975a) with experimental data.

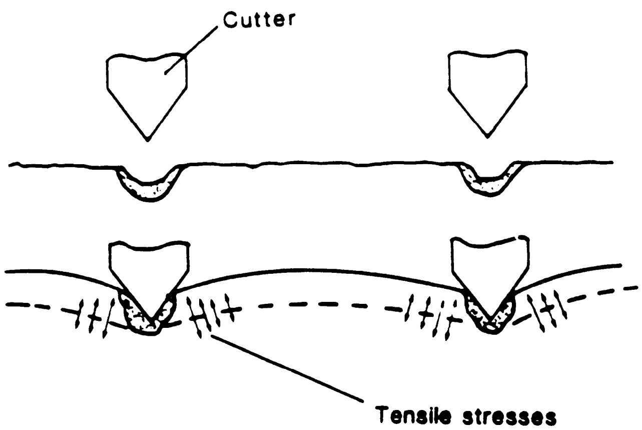


Fig. 9.1.11. Model for tensile chip formation by disk cutters (after Lindqvist and Ranman, 1980).

Sikarskie (1965). One of the first attempts to write the effects of a crushed zone into disk theory appears to come from Gaye and Stephens (198). They introduce the concept of "fragma" to describe the mass of broken material in the vicinity of the disk edge, arguing that the leading curved edge of a disk plows through the "fragma" like the bow of a boat through water. They argue further that the disk thrust force F_T required for breakage between parallel grooves arises from:

1. A component F_N corresponding to the buoyancy of the disk moving through the fragma.

$$F_N = \frac{C_0 \tan\theta}{7.5} \sqrt{Dd^3} \qquad (9.1.14)$$

2. A shear stress component $F_{N'}$ corresponding to the initial phase of the breakage process that generates the "vee" profile of the disk groove.

$$F_{N'} = \frac{sS_0 \tan\theta}{5} \sqrt{Dd} \qquad (9.1.15)$$

3. A tensile component $F_{N''}$ causing rupture of the rib of rock between the adjacent rib grooves.

$$F_{N''} = \frac{s^2 T_0 \tan\theta}{11} \sqrt{\frac{D}{d}} \qquad (9.1.16)$$

In addition, these authors calculate the rolling force acting on the cutter using the relation,

$$\frac{F_T}{F_R} = \frac{3}{4}\sqrt{\frac{D}{d}} \qquad (9.1.17)$$

The components for F_T are not additive, but they define the limiting conditions for the breakage process to proceed from one phase to the next. Gaye and Stephens draw on the observed change in performance with certain variables on a tunnel boring machine (TBM) to support the theory, and they suggest that controlled experiments should be used to test, and if necessary, to modify it.

Kutter and Sanio (1982) and Sanio (1985) have developed a model for disks taking into account the role of the crushed zone. They assume a dominantly tensile mode of failure. Their hypothesis is developed as follows (Fig. 9.1.14):

1. The high stress concentration at penetration causes a crushed zone around the edge of the disk.

2. Within the crushed zone, the stress will approximate the hydrostatic state causing tangential tensile stresses. When these stresses reach the tensile strength of the intact rock T_0, radial tensile cracks will develop.

3. One such crack will extend to the free rock surface forming a chip.

The theory is developed by assuming the crushed zone to be circular in shape. Its center is at the disk edge, and its radius is taken to be a constant fraction q of the penetration depth d.

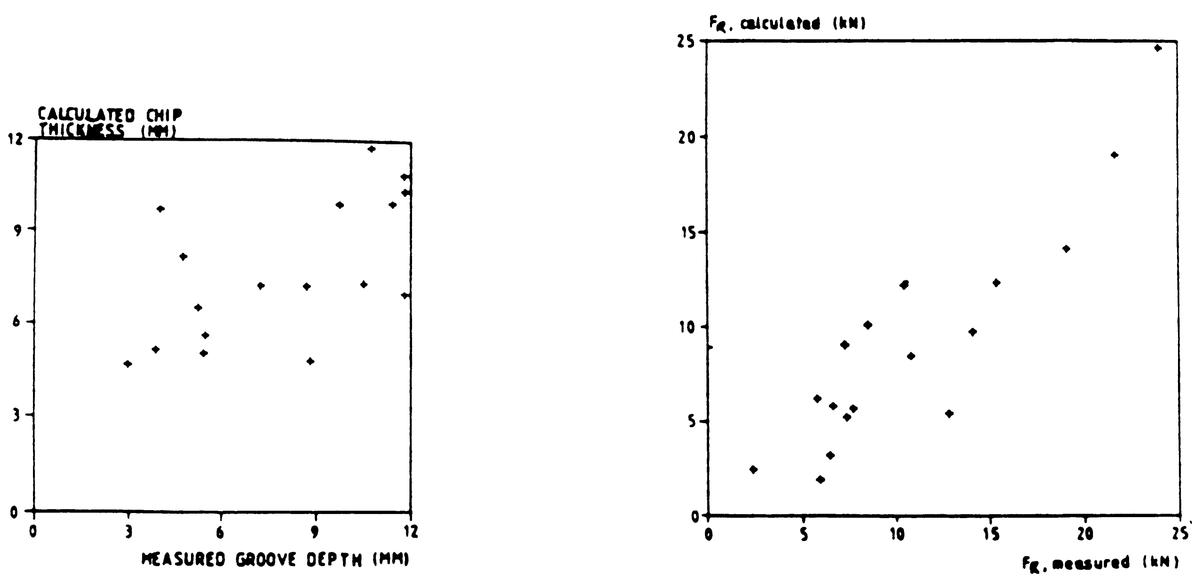


Fig. 9.1.12. Measured and predicted chip thicknesses and disk cutting forces (after Lindqvist and Ranman, 1980).

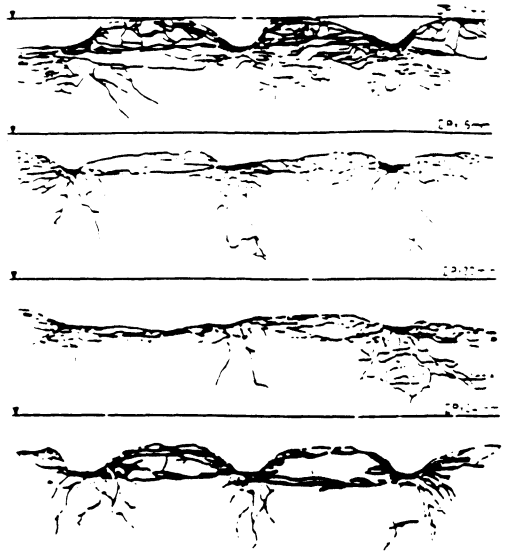


Fig. 9.1.13. Development of cracks by interacting disks (after Fenn, 1985).

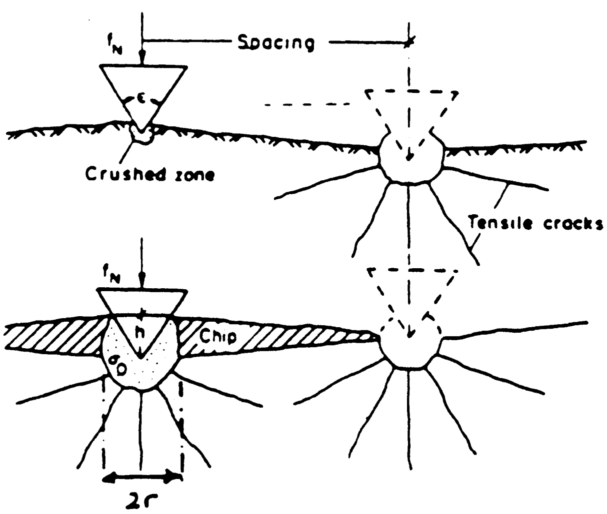


Fig. 9.1.14. Tensile breakage model for disks (Sanio, 1985).

Sanio introduces a critical stress intensity factor, or fracture toughness, K_c, adopting Ouchterlony's (1974) relation,

$$2\sigma_0 r = Kc\sqrt{a} \qquad (9.1.18)$$

where σ_0 is the hydrostatic pressure in the crushed zone, r is the radius of the crushed zone, and a is the crack length.

It is assumed that the crack length increases from zero at the leading edge of the disk to a maximum value of S, whereupon a chip is formed. This leads to the following equation:

$$F_T = \frac{2K_c \tan\theta}{3q}\sqrt{S(Dd - d^3)} \qquad (9.1.19)$$

Sanio calculates the cutter rolling force using,

$$\frac{F_T}{F_R} = \frac{5}{4}\sqrt{\frac{D}{d}} \qquad (9.1.20)$$

Sanio defines $\dfrac{K_c}{q} = S_K$ as a cutting constant that depends on the strength and defect structure of the rock as well as the number of radial cracks and the geometry of the crushed zone. This large number of variables precludes a theoretical determination of S_K. Sanio reports values for S_K in a single rock type ranging from 0.17 to 0.47, with a coefficient of variability of about 10% in each case, depending on the schistosity orientation. The Kutter-Sanio model has been tested by Sanio (1985) against experimental data using different disk cutter geometries in 14 different rock types with varying degrees of anisotropy. Sanio reports that the mean deviation between the measured and the predicted disk forces was less than 20%.

Indentation—Summary: In Paul and Sikarskie's model, the shear force acting along the fracture plane is assumed to be responsible for inducing rock failure. The magnitude of this force is determined by resolution of the force components acting on the wedge. From this argument, the force required to cause failure should be a function of the wedge included angle. Clearly from Fig. 9.1.6 this is the case. Also, when this included angle exceeds a critical angle, the ratio of the shear stress to the normal stress along the failure plane should fall below the Mohr-Coulomb criterion, and no failure should take place. A flat-bottomed punch simply is a wedge with an included angle of $\pi/2$; obviously, this exceeds the critical angle, yet failure does occur. It follows that, depending on the geometry of the indenter, more than one mechanism is causing rock failure.

Three processes have been identified during the loading operation with the flat-bottomed punch. First, Hertzian cracks are developed, then intense crushing of the rock beneath the punch takes place, and finally cracks initiating from this crushed zone develop to form rock chips. The second of these processes is associated with dilatation that causes tensile stresses to be induced in the intact rock surrounding the crushed region. Consequently the crack that forms the rock chip is initiated when this tensile stress exceeds the rock tensile strength.

It is likely that this rock failure mechanism also plays a role, perhaps even a substantial role, when wedges with smaller included angles are used to load the rock. It is fairly clear, for example, that the rock chips induced by wedge-shaped disk cutters on a boring machine are not shear cracks, because the surfaces of these cracks show none of the gouge damage usually associated with shear failure. Rather, these crack faces are clean and have the characteristics associated with tensile failure including, sometimes, a visible transition at the point where the crack length had grown sufficiently for the fracture to become unstable. Thus, while Paul and Sikarskie's model predicts that failure takes place as the result of an applied shear stress, and the Coulomb-Mohr criterion in a modified form seems adequate to predict the magnitude of the applied stress necessary to cause this failure, rock breakage takes place because a tensile fracture is initiated and propagated. This fracture is induced as a result of a combined loading: first, the tensile extension of preexisting flaws along the fracture plane and second, by tensile stresses induced by the crushed zone beneath the wedge. Korbin (1979) described this loading as a pressure bulb.

9.1.2.2 Drag Bits

The action of sharp drag bits is to cleave rock chips from the face (Fig. 9.1.15). Although a shear force is applied to the

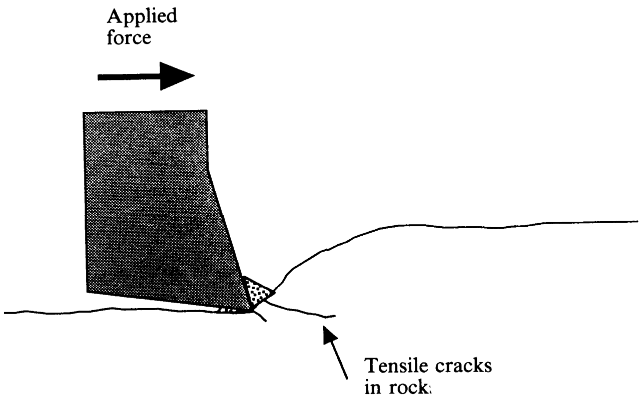


Fig. 9.1.15. Shear force applied to the rock by a drag bit inducing tensile crack growth in the rock.

rock by the bit the rock breaks as the result of tensile crack growth (Fig. 9.1.15).

Drag Bit Forces: From the outset of studies in rock cutting mechanics, attempts have been made to describe the action of cutter drag bits in mathematical terms. An early effort sought to borrow from the field of metal machining theory by adapting Merchant's (1944) continuous cutting (elastoplastic) model to the discontinuous (brittle, discrete chipping) regime of rock cutting. Thus Shuttleworth offered the equations:

$$F_C = \frac{S_0 dw \cos(\phi_f - \alpha)}{\sin\phi \cos(\phi + \phi_f - \alpha)} \qquad (9.1.21)$$

$$F_T = \frac{S_0 dw \sin(\phi_f - \alpha)}{\sin\phi \cos(\phi + \phi_f - \alpha)} \qquad (9.1.22)$$

where a is the tool rake angle, and w is the tool width.

A good qualitative and limited quantitative value was ascribed to the model in that it:

1. Indicated a linear increase in drag bit forces with depth of cut.
2. Described the monotonic decrease in forces with increasing rake angle.
3. Showed drag bit forces to increase linearly with rock strength, in this case specifically shear strength.

The model is limited, however, in the sense that it is a two-dimensional description of what in reality is a three-dimensional process. Furthermore, the observed chip failure patterns caused by wedge penetration into coal were argued to be essentially tensile, rather than shear, in character.

The first discretely rock cutting theory was proposed by Evans (1962). It arose from studies of the failure produced when a simple wedge was pushed into coal. The penetration rate was at a very slow rate, and photographs were taken at intervals during the process. Some crushing of coal was evident in the vicinity of the wedge surface, but the most interesting feature was a lengthy crack extending from the edge of the wedge that moved ahead into the unbroken coal.

The breakage pattern was even more revealing when the wedge was made to penetrate parallel to, and some distance in from, a free surface (Fig. 9.1.16). This was argued to be more representative of the practical cutting situation. Here the major, and often the only, crack occurred as a curved arc originating at the drag bit tip and emerging at the coal surface. This resulted in the formation of a large chip.

Evans argued such failure to be essentially tensile and considered the limiting equilibrium of the chip, assumed to be a circular arc, under the action of three principal forces and by invoking the minimum work hypothesis developed the equation,

$$F_C = \frac{2T_0 d \sin\theta}{1 - \sin\theta} \qquad (9.1.23)$$

where F_C is the tool cutting force, i.e., the force component in the direction of tool motion.

This model was for a symmetrical wedge that is untenable in practice and so it was adapted to the asymmetric geometry to give

$$F_C = \frac{2T_0 d \sin\frac{1}{2}\left(\frac{\pi}{2} - \alpha\right)}{1 - \sin\frac{1}{2}\left(\frac{\pi}{2} - \alpha\right)} \qquad (9.1.24)$$

It was found by experiment that this expression gave the right sort of variation of cutting force with wedge angle. Further studies of wedge penetration into different rocks showed the curved chip formation to dominate.

As an aside, the rock breakage patterns deliberately generated by stone masons from ancient times when dressing blocks of rock conform closely with those described by Evans, and the durability of the resulting rock surfaces lends support to the notion of a single dominant curved crack being the cause of breakage. Nevertheless, it was noticed that chip formation was not always curved but could be flat. Indeed, as the rake angle of a drag bit reduced, it was more likely that chips would tend to be pushed off as opposed to being prized off. Accordingly, Nishimatsu (1972) has proposed an alternative model by invoking a Coulomb-Mohr failure criterion along a plane leading to an equation for the generalized drag bit force *F:*

$$F = \frac{2S_0 dw \cos\phi}{(n + 1)\,[1 - \sin(\phi - \alpha + \phi_f)]} \qquad (9.1.25)$$

where n is the stress distribution factor.

The cutting (F_C) and normal (F_N) components are given as

$$F_C = F\cos(\phi_f - \alpha) \qquad (9.1.26)$$

$$F_N = F\sin(\phi_f - \alpha) \qquad (9.1.27)$$

As with the Shuttleworth and Evans models, this one also suggests a linear increase in drag bit force, with both cutting depth and rock strength and a monotonic reduction in force with increasing rake angle. In other words, the three models predict the same basic force trends for the principal cutting variables. Little confidence, however, can be attached to them in the quantitative sense although they may show tolerable agreement with the idealized two dimensional situations they represent. Another limitation is that the models all attempt to describe the simple wedge, and this shape is not common in practice.

In this latter context, Evans has proposed a model for pointed drag bits. For this, he postulates the rupture of rock by means of a circular hole having its axis parallel to the surface of a semi-infinite medium, which is subject to internal pressure (Fig. 9.1.17). Radial compressive stresses in the rock are accompanied by tensile hoop stresses to cause cracks to open up at the interface between the hole and the rock when the stress equals

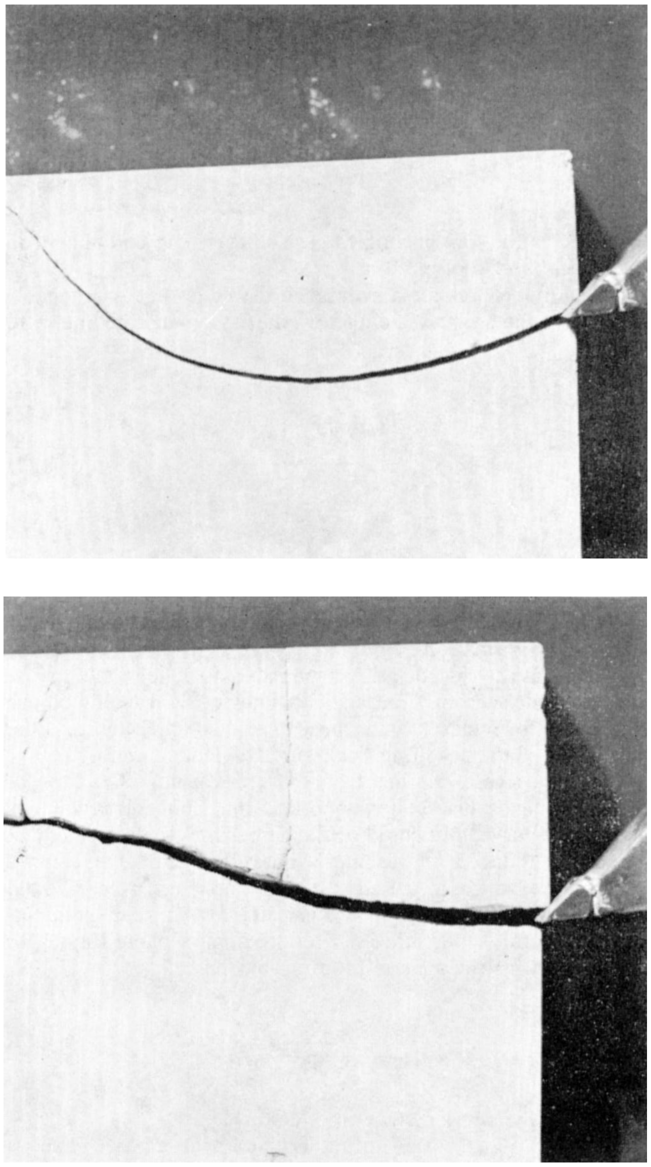


Fig. 9.1.16. Chip formation in coal by wedge penetration parallel to a free surface (after Evans, 1962).

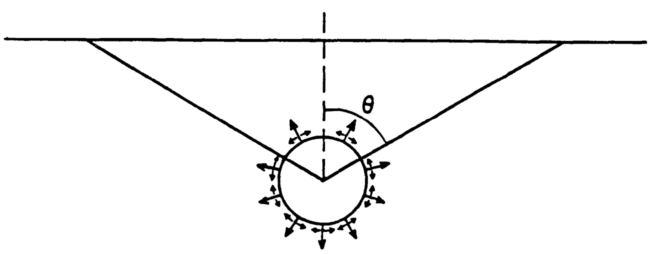


Fig. 9.1.17. Model for rock failure using pointed drag bits (after Evans and Hurt, 1979).

the tensile strength of the rock. Judging by the breakage patterns produced by pointed drag bits, a major crack will propagate to the unstressed surface to each side of the hole at an angle *q.*

The angle *q* is calculated by considering the limiting equilibrium of the half segment of the rock chip under the action of three forces. The value of *q* at which breakage takes place is argued to be that which minimizes the energy of radial displacement produced by a compressive stress *q*. This is calculated to be 60°, which is of the right order for the breakout angle found in practice.

Theoretical explanations of the effect of blunting on drag bit forces have been offered by Dalziel and Davies (1964). Employing Griffith's hypothesis, they suggested that the indentation force *F* required to penetrate a wedge presented normally into a rock surface would vary according to the radius of wedge tip r_1:

$$F\alpha\sqrt{r_1} \qquad (9.1.28)$$

Controlled coal cutting experiments with drag bits having artificially generated wear lands showed this relation to be valid, a plot of F against width of wear land yielding the exponent 0.52 ± 0.05.

Evans (1965) used this relation to adapt his tensile theory for wedge action to take account of drag bit blunting. He assumed that, for a wedge having an included angle of $2q$ with a wearflat $2b$ ground perpendicular to the wedge axis, the compressive force needed to initiate a tensile crack would be proportional to

$$F \alpha 2b^M \tag{9.1.29}$$

where M is a constant.

Evans went on to derive the equation,

$$F_C = 2T_0 d \left[\frac{\sin(\theta + \phi_f)}{2 \sin A \cos(A + \theta + \phi_f)} + 2^{(M-1)} \left(\frac{C_0}{T_0}\right)\left(\frac{b}{d}\right)^M \left(1 + \frac{\sin A \sin(\theta + \phi_f)}{\cos(A + \theta + \phi_f)}\right)\right] \tag{9.1.30}$$

Values for A in Eq. 9.1.30 are determined from

$$\frac{\cos(2A + \theta + \phi_f)}{1 - \cos 2A} = 2^{(M-1)} \left(\frac{C_0}{T_0}\right) \left(\frac{b}{d}\right)^M \cos(\theta + \phi_f) \tag{9.1.31}$$

The equation shows very good approximation to the experimental results of Dalziel and Davis when (C_0/T_0) is 6.25 and M is 0.67, whereas on theoretical grounds M should be 0.5 and (C_0/T_0) should be between 8 and 10.

The equation conforms with measured data in that progressive blunting does not have a proportionate effect on forces. It does not, however, provide any insight into the disproportionate increase in normal force over cutting force as wear proceeds.

Effect of the Principal Cutting Variables: Results from rock cutting experiments with picks have been reported from many laboratories worldwide. These cover a wide range of cutting conditions (pick shape, rock type, etc.). Inevitably, there are some conflicts of evidence, but some generalities do emerge that can be regarded as fundamental features of pick cutting.

Before describing these, some further definitions are required. Pick performances are usually defined in terms of mean and peak values of F_C and F_N. A further important performance parameter is, however, specific energy E_s, which relates cutting force to the amount of rock produced. It is defined as the work done per unit volume of rock excavated. There is also a wide variety of pick types characterized by shape (chisel, round bottom, v-front, v-bottom, pointed) and various attendant angles (rake, bottom clearance, side clearance, attack, v-front, v-bottom). Most of these have been studied, but the effects of those angles describing the more complex shapes appear to be of no more than marginal significance to the cutting process. Consequently, consideration of picks is confined to three essential shapes: the simple chisel, the round bottomed tool, and the pointed tool. These are illustrated, showing their principal geometric features, in Fig. 9.1.18.

The generalities of pick performance are now summarized with reference to Figs. 9.1.19 to 9.1.23.

1. Both F_c and F_N increase with d for all pick shapes. In most circumstances, the increase is approximately linear from or close to the origin (Fig. 9.1.19a). When using wide picks at depths much less than their width, the force increases may not be linear. Because it is largely irrelevant to the practical situation, this latter condition has not been studied in detail.

2. Specific energy varies inversely with depth of cut for all pick shapes (Fig. 9.1.19b). In other words, cutting efficiency improves with depth of cut. This arises from item 1, thus

$$E_s = \frac{F_C 1}{A * 1} = \frac{k_1 d}{k_2 d^2} = \frac{\text{constant}}{d} \tag{9.1.32}$$

3. Cutting and normal forces decrease monotonically with increasing rake angle (Fig. 9.1.19c). Most of the benefit to pick forces has been achieved at a rake angle of 20°, beyond which further marginal improvement is at an increasing penalty to pick strength and its potential to survive.

4. Increasing back clearance angle reduces pick forces up to about 5°, beyond which forces are independent of this angle (Fig. 9.1.19d).

5. Cutting speeds of up to 17 fps (5 m/s) and possibly someway beyond have no discernible affect on pick forces or specific energy.

6. Given that speed has no direct effect on pick forces and that cutting efficiency improves with cutting depth, it follows that for a given available power, greater production will be achieved by cutting slowly and deeply than at a faster and proportionately more shallow depth.

7. For a chisel-shaped pick, F_C and F_N increase linearly with pick width (Fig. 9.1.19e). The intercept on the ordinate is the force attributable to the production of breakout. In general, the breakout component appears to require about the same specific energy as the component of breakage corresponding to the area swept by the pick.

8. Pick forces increase approximately linearly with rock strength. Because of rock heterogeneity, it is difficult to say which of the strength parameters, if any, dominates the situation. Figs. 9.1.20a and 9.1.20b show the relation between, respectively, unconfined compressive and tensile strengths and pick cutting force. Both are substantially linear. These data come from controlled cutting experiments using rock samples from different geographic locations and vertical horizons in a single massive sandstone formation.

9. Further on the matter of rock strength, when a rock is saturated, it invariably leads to a reduction, often quite large, in mechanical strength. It might be expected, therefore, that saturation would correspondingly lead to a reduction in pick cutting forces. This is not necessarily found to be the case. What few data are available show pick forces to increase in saturated rock. The reasons are not fully understood but are believed to be due to pore water dissipating the high local stress concentrations normally associated with crack development.

10. For picks of different shape but having the same rake and clearance angles and operating at the same depth of cut, the pointed pick requires the least cutting and normal force. The chisel pick requires the highest forces with the round-nosed pick taking an intermediate position. However, because the chisel pick cuts a disproportionately larger volume of rock than the other two shapes, it cuts with the lowest specific energy and is thereby the most efficient shape. The pointed pick is the least efficient. Indeed, controlled laboratory tests, in whatever rock, invariably show the chisel-shaped pick to be the most efficient of all shapes. Shape features such as a ridged front, v-bottom, round bottom, side rake, and point are all less efficient than the simple chisel when operating at the same depth of cut. It is important, however, to stress the qualification "at the same depth of cut." The pointed pick because of its better penetrating capability (lower normal force) can, for a given available normal force, achieve a larger depth of cut than the chisel or other shapes. It can thereby operate more efficiently than the chisel by virtue of its ability to cut more deeply (Fig. 9.1.21).

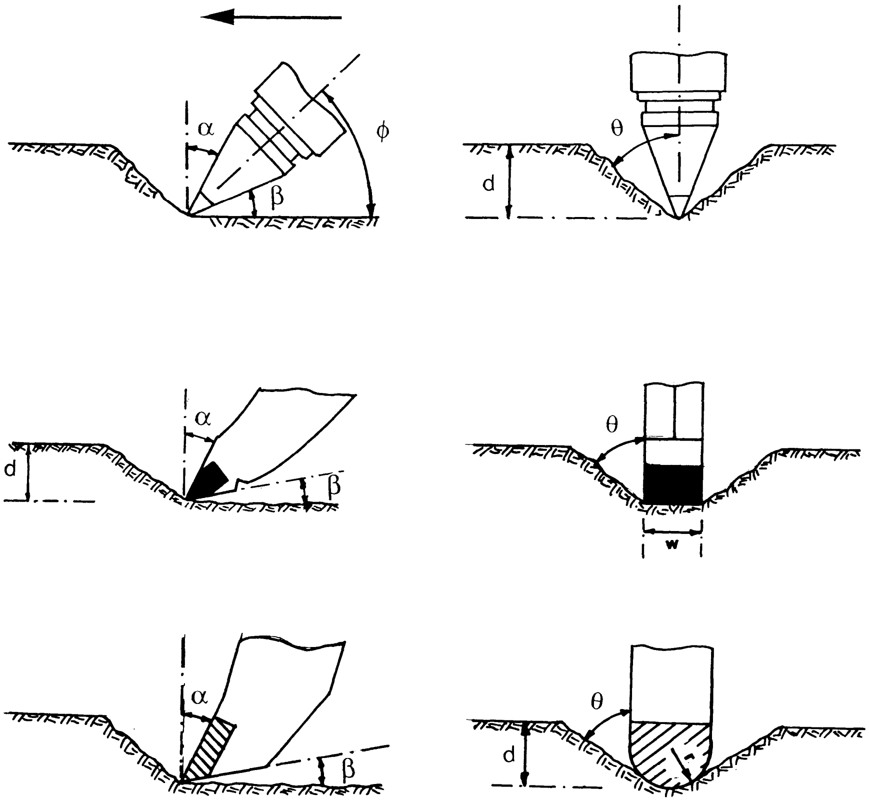


Fig. 9.1.18. Pick shapes.

11. There is strong evidence that the coarseness of the rock broken during cutting increases with cutting efficiency. The generality appears to be that, in a given rock, any combination of cutting variable that leads to an increase in cutting efficiency (reduction in specific energy) causes a corresponding increase in the coarseness of the product, and by inference (with some supporting experimental evidence) a reduction in the amount of potential airborne dust (Figs. 9.1.22 and 9.1.23).

12. Few data are available on the effects of pick rigidity. Some experiments using a stiff but compressible backing to the pick showed no discernible effect on cutting forces or specific energy. Energy stored by the compliant backing material on release produced no measurable beneficial or deleterious effect and was dissipated by propelling broken rock fragments away from the cutting edge at high velocity. An aspect of pick rigidity that appears not to have been studied to any significant extent is its effect on pick damage.

Drag Bit Interaction: Evans (1974) looked at the lateral breakage induced by both chisel and pointed drag bits as a quite separate issue to his forward breakage models. For chisel drag bits, he considered the geometry of the section of a cut made in a rock surface and assumed breakout to start from the corners of the cut and to propagate towards the surface, the lines of breakage making an angle *g* to the horizontal.

He considered unit extent of this model and the forces acting on section AOBC at breakage. This leads to an equation for breakout angle (π/2 − *g*):

$$\tan\gamma = \frac{1}{5}\left[\frac{w}{d} + \sqrt{\left(\frac{w}{d}\right)^2 + 20}\right] \qquad (9.1.33)$$

and thereafter the maximum separation between drag bits is derived from the simple geometric relation,

$$s = w + 2d\cot\gamma \qquad (9.1.34)$$

Eq. 9.1.33 yields values for *g* which are significantly larger (i.e., smaller breakout angle) than are commonly found in practice. However, the equation does indicate only a small variation in *g* over a wide but sensible practical range for both *w* and *d*. This is much in line with practical experience.

Evans and Hurt (1979) offered a solution for the spacing between pointed drag bits based on an extension of his pointed drag bit cutting theory. This considered the rupture of the rock between two parallel holes each being stressed by a drag bit (Fig. 9.1.24), which resulted in the equation

$$s = 2d\sqrt{3} \qquad (9.1.35)$$

The breakout angle caused by single pointed drag bit breakage (Fig. 9.1.17b) was determined to be a constant 60°. And two adjacent grooves at this angle that just meet at the rock surface are exactly defined by Eq. 9.1.35 (i.e., $\tan^{-1}\sqrt{3} = 60°$). This is the same result as obtained for breakout with chisel-shaped drag bits, thus suggesting that the breakage pattern for both chisel and pointed drag bits is essentially the same.

Studies on the interaction between adjacent parallel picks have been reported from several laboratories covering a range of rocks. The generalities that emerged from these studies are summarized.

1. Cutting with an array of picks involves each pick passing through the rock in sequence. Each pick thereby has the opportunity to exploit the relief provided by an adjacent groove produced by a preceding pick in the sequence.

2. There is an optimum spacing between adjacent picks in an array that is consistent with minimum specific energy. This optimum spacing increases with depth of cut, but because of

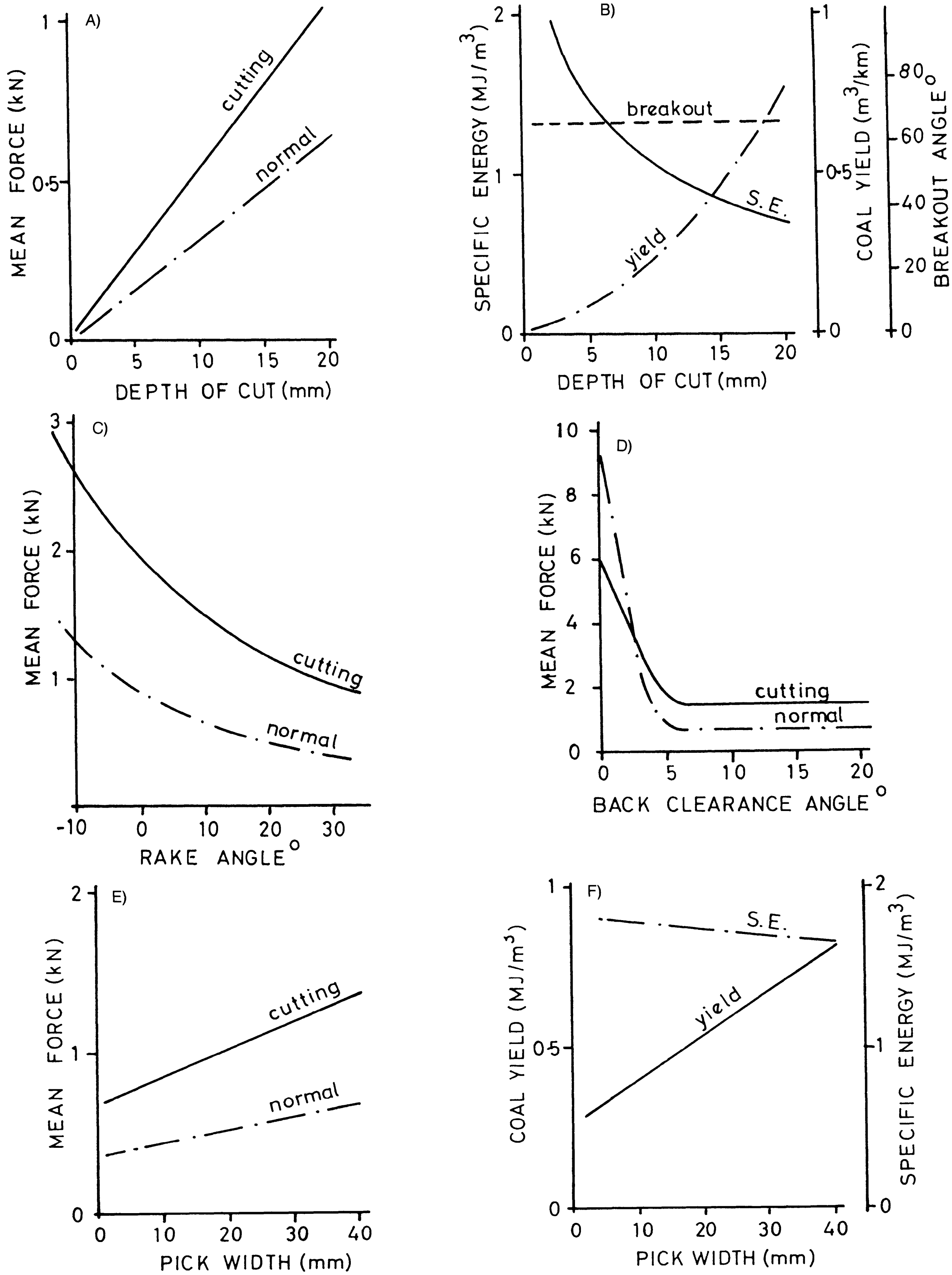


Fig. 9.1.19. Effect of principal cutting variables on pick performance.

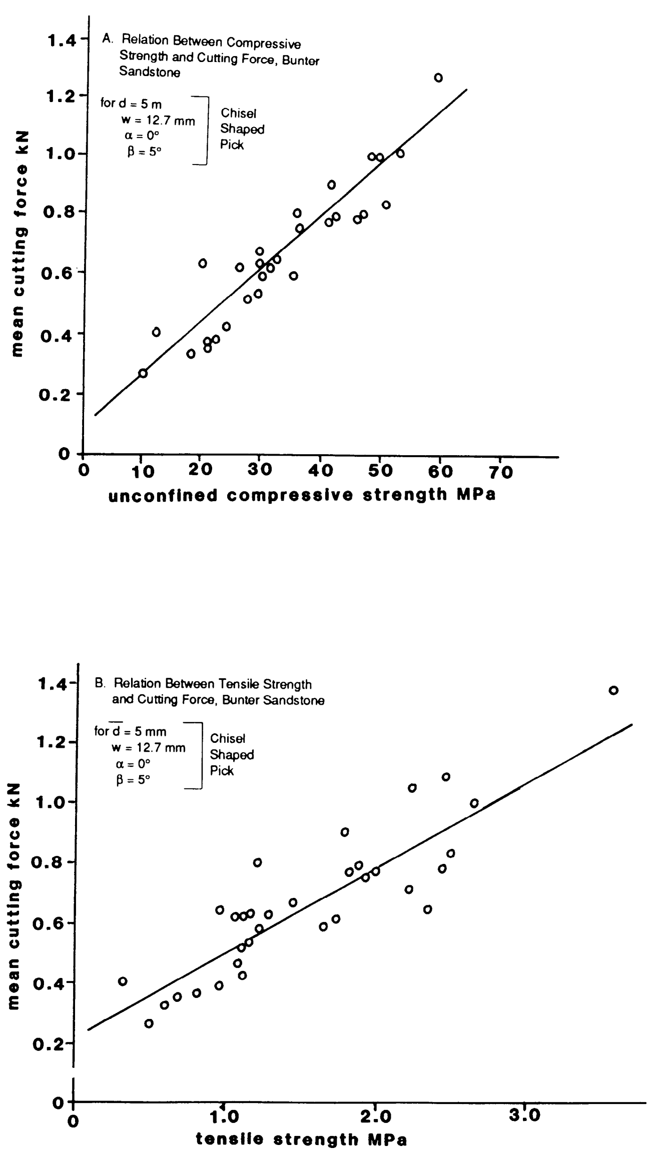


Fig. 9.1.20. Effect of rock strength on cutting force.

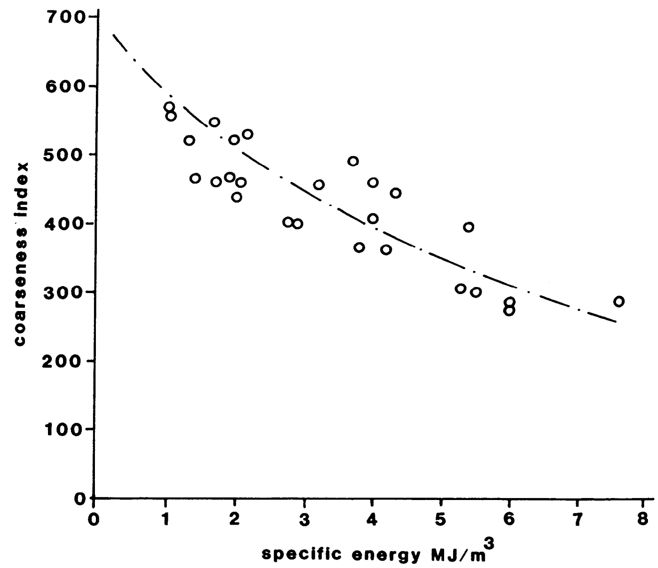


Fig. 9.1.22. Variation of product coarseness with cutting efficiency.

geometric consistency (as shown in Fig. 9.1.25), the optimum spacing can be expressed as a multiple of cutting depth. This is a singularity known as the spacing-depth ratio and its optimum value is found to be reasonably constant in a given rock.

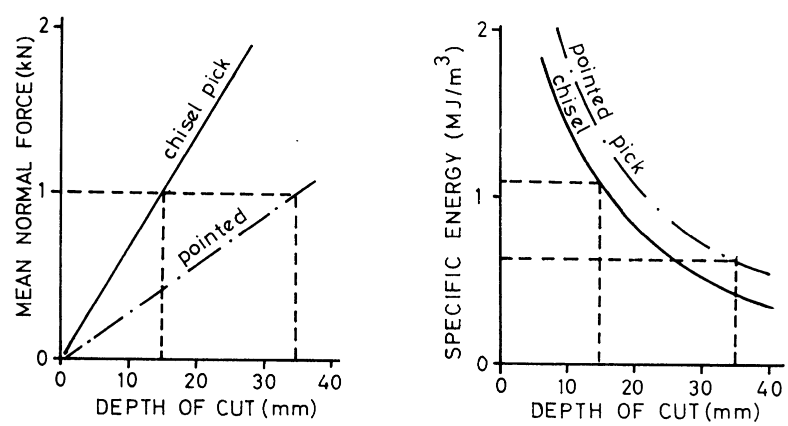
3. Values for optimum *s/d* ratio with pointed picks have been found to range from about 1.5 to 3.0, according to rock type. With chisel picks, the interaction between pick width and spacing has to be taken into account as illustrated by Fig. 9.1.26. Spacing refers essentially to the land of rock between adjacent cutters which for chisel picks is described by *s–w*. For pointed picks, where *w* is sensibly zero, *s–w* reduces to *s*.

4. It is commonly found that interaction between adjacent picks ceases when $(s-w/d) \leq 2 \tan g$. This defines any distance beyond the point where the breakout originating from adjacent picks meet at the rock surface.

5. It is also found that the optimum spacing is usually about one-half (i.e., = tan *g*) the limit of pick interaction.

6. At spacings less than the optimum, specific energy increases rapidly and reaches a maximum at zero spacing, that is, when a pick tracks in the same groove as the preceding pick. This situation is consistent with the implications of Eq. 9.1.32 as $d \rightarrow 0$.

Fig. 9.1.21. Relative performances of pointed and chisel picks.



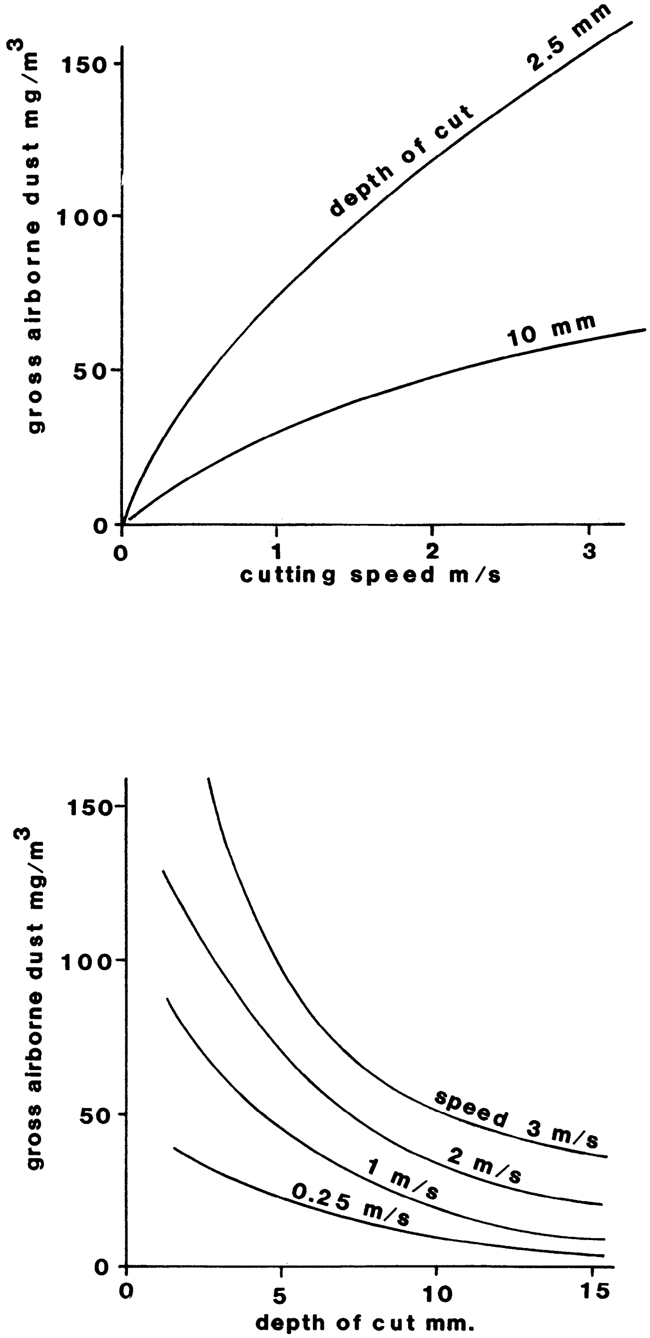


Fig. 9.1.23. Effect of depth of cut on dust generation.

7. Pick forces are at a minimum for $s = 0$ and increase approximately linearly as spacing increases, to reach a maximum equal to an unrelieved pick for $s \geq 2\ d\ \tan g$. Some typical force data expressed as a fraction of the unrelieved forces are shown in Fig. 9.1.27.

8. Practice commonly involves the following pick in an array overcutting (i.e., cutting deeper than) the preceding adjacent pick. This arises from a forward axial motion of a drum being superimposed on its rotational motion. It is thought that this and other associated factors such as variable cutting depth and cutting through generated surfaces can have little, if any, notable effect on the established principles.

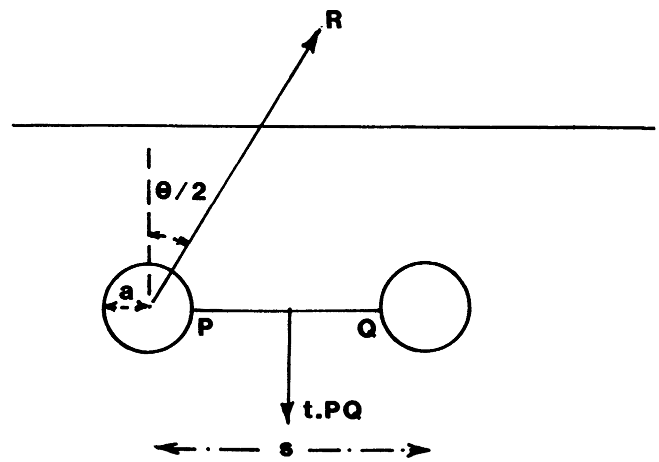


Fig. 9.1.24. Interaction of adjacent pointed drag bits.

9. Every excavation has at least one corner that defines the boundary of the cutting area, and some picks are required to cut in this region. Measurements show that the cutting force for corner cutting is typically two to three times higher than for cutting an unbounded surface. Normal force, however, appears not to increase in the same proportion and is between one and two times higher. The confinement of the corner inhibits rock breakage, and so the volume of rock cut is reduced. This leads to the specific energy for corner cutting picks to be typically three to four times higher than for face picks.

10. Situations can arise in practice where the achievable depth of cut is less than is required for effective pick interaction. This leads to a situation known as groove deepening where adjacent cutters, which fail to interact in a single pass, make repeated deepening passes in the same tracks. This does not, however, lead to breakthrough between adjacent grooves. There is a progressive loss of breakout as the grooves are deepened with no prospect for initiating interaction. Groove deepening is, in fact, a very inefficient process.

11. The cutting sequence of an array of picks should be arranged such that the first pick in the sequence is able to exploit any available free surface. Thereafter, the following picks should successively exploit the relief produced by the pick immediately preceding it. In this way, relief can be progressively carried through to the corner of the excavation. The cutting sequence should always end in the corner; it should never start there.

Comments on Drag Bit Theory: What has been described may not cover all drag bit cutting theories so far developed. They are, however, the theories that are most widely recognized and most frequently cited in the literature. They all have serious limitations. Some qualitative success can be claimed for each, but none provides an adequate quantitative description of the drag bit cutting process. The principal limitation is that all are two-dimensional and simplified descriptions of what in reality is a complex three-dimensional situation. Furthermore, each theory has its incongruities. For instance, Evans' tensile theory for wedges specifically excludes a normal force, and in his pointed drag bit theory F_C does not reduce to zero when q is 0°.

These deficiencies apart, the various theories do have some modest successes to their credit. They show the right sort of variation in drag bit forces with the main variables of rock strength, drag-bit geometry, cutting depth, drag-bit interaction and wear. Perhaps significantly, Evans' suite of theories, used collectively, shows that little if any difference can be expected between the performance of chisel and pointed drag bits that are

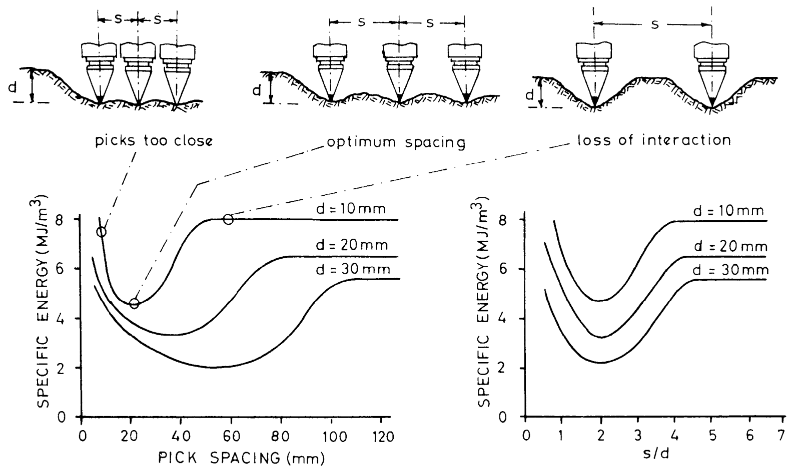


Fig. 9.1.25. Pick spacing and its effect on specific energy.

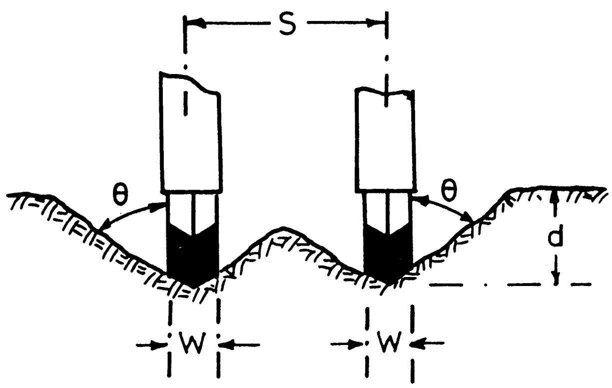


Fig. 9.1.26. interplay between pick width and spacing.

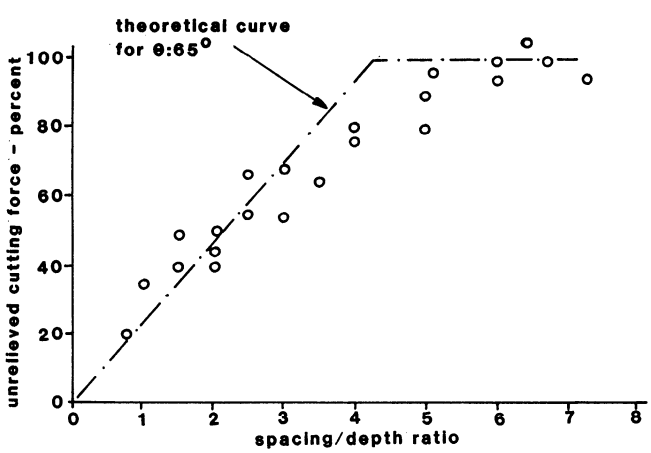


Fig. 9.1.27. Measured values for Bunter Sandstone.

optimally spaced. Such, by and large, is found to be the case in practice.

Practical Implications: Drag bits are an efficient means of breaking rock. Setting aside, for the moment, the question of their ability to survive in the hard-rock environment, they require substantially less energy than other mechanical cutters. They are also at least one order of magnitude more efficient than exotic devices such as water jets, jet piercing, and lasers. Research findings have now been largely assimilated by the industry into the design of mining machines, and significant benefits to machine performance have resulted.

There are two main problems to be solved before the present limits of drag bit application can be extended:

1. The development of new drag bit materials of substantially better wear resistance and strength than those presently available.

2. The ability of machines to provide sufficient thrust force to keep drag bits at an effective depth of cut as drag bit wear proceeds.

9.1.3 DRILLING

Rock excavation in the great majority of noncoal, underground mining operations is carried out using drill-and-blast techniques. About 85% of the holes drilled in underground mines are produced using conventional percussive drills. A small fraction of the holes are drilled with down-hole drills. The remaining, roughly 15%, of these holes are drilled with small diameter, rotary drills. Most of these small-diameter holes are drilled in coal mines for the emplacement of roof bolts. In surface mines, the vast majority of the rock is excavated by drill-and-blast methods. The machines used most commonly for excavating the blastholes in these mines are large-diameter rotary drills.

9.1.3.1 Drillbits

Percussive: Percussive drillbits break rock by indentation. The energy applied to the bit by the drilling machine causes the bit to strike the rock face at a velocity v. The peak stress σ_p induced in the rock by this impacting tool is given by

$$\sigma_p = \rho c v \tag{9.1.36}$$

where ρ is the density of the impacting body (490 lb/ft^3 or 7850 kg/m^3 for steel), and c is the compressional wave velocity of the impacting body (16,400 fps or 5000 m/s for steel).

Thus, for a steel projectile, $\sigma_p = 1{,}733v$ psi when v has units of feet per second, and $\sigma_p = 39.25v$ MPa when v has units of meters per second.

Two major types of percussive drillbit are available commercially; the brazed bit and the button bit. The brazed bit is made up from (either one or four) rectangular prisms of cemented tungsten carbide. These inserts are brazed into the end of the

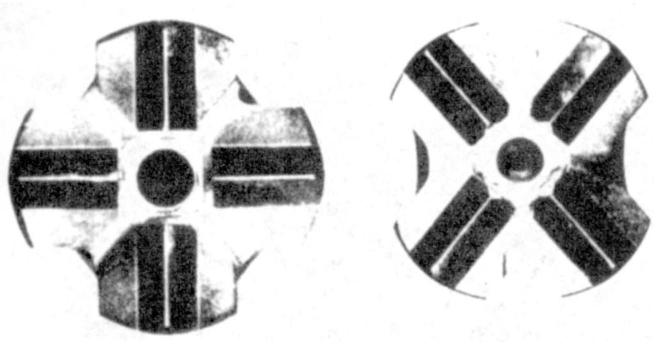


Fig. 9.1.28. Cross bit and X-bit (from Kurt, 1982, originally from Padley and Venables Ltd.).

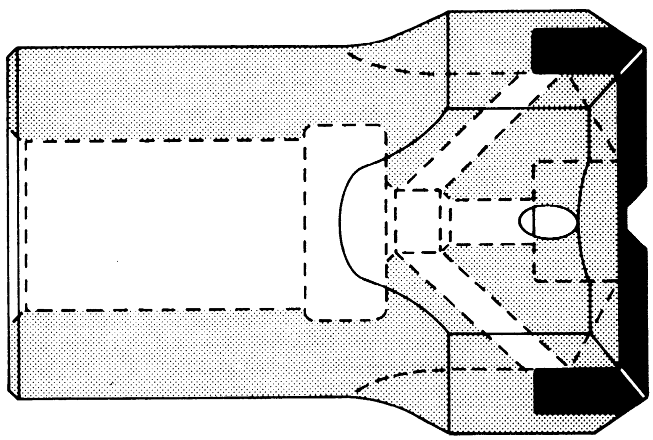


Fig. 9.1.29. Water flushing holes in bit (from Kurt, 1982).

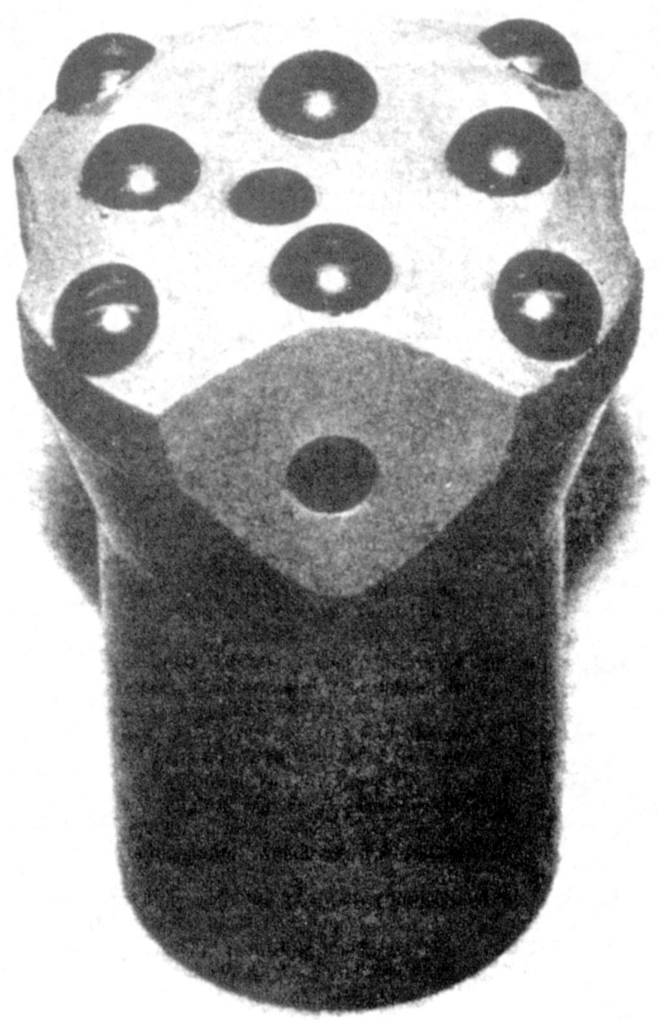


Fig. 9.1.30. Button bit (from Kurt, 1 982, originally from Ingersoll Rand Co.

steel drillbit. Two arrangements for mounting four inserts in these inserts in the bit are common; these are the crossbit, in which the inserts are mounted at 90° intervals around the bit face (Fig. 9.1.28a), and the X-bit where the angles between the inserts are 80° and 100° (Fig. 9.1.28b). The angle of the cutting edge of all these tungsten carbide inserts generally is 110°.

Drilled holes are provided in the bit body. The usual arrangement is for one center hole and four other holes located around the outside of the bit (Fig. 9.1.29). These provide the passages for conducting water to the bit face during the drilling operation. This water serves both to cool the bit inserts and to remove the broken rock debris from the region ahead of the bit. Adequate cooling is essential if the rate of bit wear is to be minimized (see 9.1.5). Continual removal of the rock debris as it is formed is essential if rapid penetration of the drillbit is to be achieved. This type of bit has the advantage that, when it becomes worn, it can be resharpened by grinding the tungsten carbide inserts.

Button bits also employ cemented tungsten carbide inserts to break the rock, but in this case the shape of these inserts is cylindrical. These cylinders are press-fitted into holes drilled into the end of the bit body, and the end of the inserts that project from the bit body is domed (Fig. 9.1.30). Because when the bit size is small, the space available for mounting the buttons is limited, button bits typically are found only in bit diameters greater than 2 in. (51 mm). The advantage that button bits have over brazed bits in the larger hole sizes is a more even distribution of cutting elements over the hole bottom. This tends to result in faster drilling rates. A disadvantage of button bits is that, generally, they cannot be resharpened.

In recent years, polycrystalline diamond compacts (PDC) have begun to find application as the cutting material in a number of different types of drillbits, including percussion button bits. Because the hardness of PDC is much greater than that of cemented tungsten carbide (WC), PDC is significantly more wear resistant than WC in rock excavation applications (see 9.1.5.2). However, PDC suffers from being substantially more susceptible to brittle fracture than WC. In some applications, where either by appropriate bit design or by limiting the classes of the rock types in which the bits are employed, PDC bits have been demonstrated to give superior performance to conventional WC bits. This improved performance generally is observed in terms of much increased rock excavation rates and dramatically improved bit life. The principal disadvantage of a PDC bit is that, at least at the present time, it is substantially more expensive (often by more than an order of magnitude) to produce than the equivalent WC bit.

In all PDC bits, a layer of polycrystalline diamond is diffusion-bonded to a cemented tungsten carbide backing material. The high-modulus (85×10^6 to 100×10^6 psi or 600 to 700

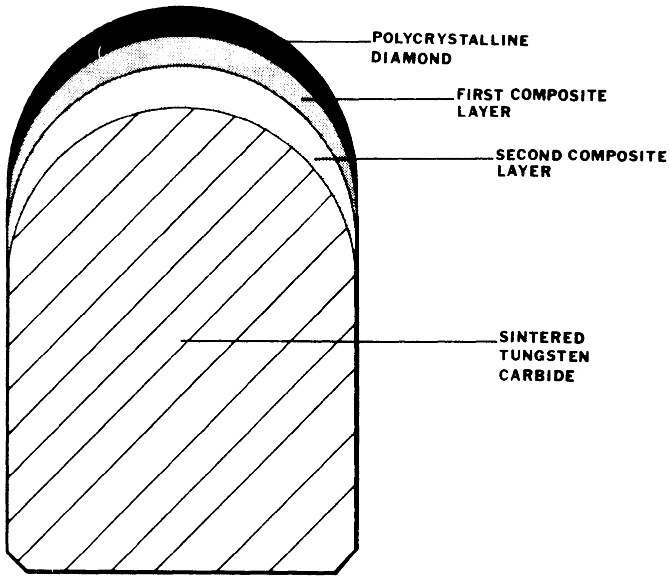


Fig. 9.1.31. Polycrystalline diamond button (from Sneddon and Hall, 1987).

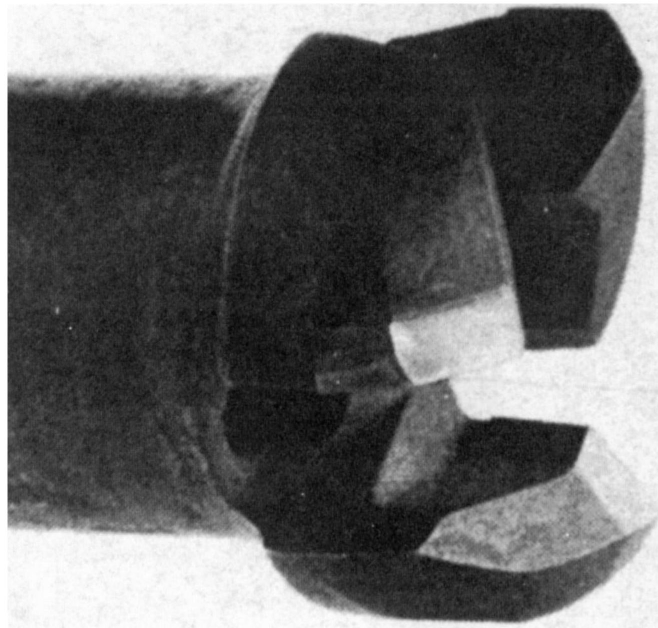


Fig. 9.1.32. Small-diameter rotary drag bit (from Rabia, 1985).

GPa) tungsten carbide provides a stiff mounting for, thereby minimizing the deflection of and thus the stresses induced in, the PDC layer. Until recently only simple shapes of PDC, typically thin cylinders about 0.04 in. (1 mm) thick and 0.5 in. (12.5 mm) in diameter, were manufactured. This material, when bonded onto the tungsten carbide substrate was used as the cutting surface of a drag bit. Today there are many sizes and variants on this shape for drag bit construction. The breakthrough in bit construction that allows PDC to be used on button bits was the development technology of fabricating PDC onto the hemispherical end of a tungsten carbide button, as opposed to the flat cylindrical shape. This breakthrough, by itself, apparently was insufficient. It was found that a single PDC layer was too brittle and resulted in premature failure of the button. The resolution to this problem was revealed in two patents (Hall, 1985; Hall et al., 1986) in which the button end was constructed from three layers of PDC and cemented carbide (Fig. 9.1.31). The layer immediately adjacent to the tungsten carbide substrate is mostly cemented carbide, the middle layer contains a greater proportion of PDC, and the outer layer has the highest volume of PDC and may be nearly identical in composition to the PDC used in drag bit cutters (Sneddon and Hall, 1987).

Field tests conducted with percussion button bits comparing standard WC bits with composite PDC buttons have shown that the latter produced an increase in bit life by about a factor of two and in drilling rate by 45% (Reinsvold et al., 1988). In these tests, the drilling costs were reduced by 33% using the PDC bits. Other advantages claimed by Sneddon and Hall (1987) for the use of these bits are the maintenance of a consistent hole diameter (because of low wear of the bit gage) and reduced wear of the other components of the drill rig (because a higher fraction of the input energy is transferred to the rock as opposed to being reflected into the rig).

Rotary: *Drag Bits*—Small diameter (2-in. or 51-mm) rotary drillbits typically are drag-type tools (Fig. 9.1.32). Again cemented tungsten carbide inserts are used for the cutting elements in these bits; these inserts are brazed into the steel bit body.

Research and development work has been conducted with rotary drills in small-diameter holes using polycrystalline diamond cutter drag bits as the cutting surfaces. The geometry of the individual cutters used on these bits is shown in Fig. 9.1.33. The main application for this type of bit is for drilling roof-bolt holes. The performance of these PDC bits has been shown frequently to be vastly superior to that of conventional WC bits. In abrasive rock formations, a PDC bit typically drills at a much faster rate, and its life often is more than an order of magnitude greater than that of an equivalent WC bit. Despite these advantages, PDC bits have not been adopted by industry for this small hole application. The reasons for this, apparently, are that the demonstrated advantages are not sufficient to overcome the cost disadvantage ($100 to $200 for a PDC bit compared with a few dollars for a WC bit) and the need for different torque and thrust requirements (a PDC bit requires high rotational speed and relatively low thrust). A PDC bit also requires more accurate control of torque and thrust in order to prevent premature bit failure.

These disadvantages have been addressed and, to a large extent, overcome for rotary drilling of large-diameter (6-in. or 150-mm) oil and gas wells using polycrystalline diamond cutter drag bits. In recent years, drillbits of the type shown in Fig. 9.1.34 fitted with a large number of PDC cutters have become increasingly popular for these applications. The rate of drilling achieved using these PDC bits often is considerably more than that obtainable with conventional roller cone bits (described in the following). Since the time spent actually drilling a well typically accounts for about a third of the total time spent by the rig on the hole, and since the cost of the rig while it is on the hole can be well in excess of $100,000/day, this improvement in drilling rate for this application can result in substantial cost savings. At the present time, the propensity of PDC cutters to fail in a brittle manner restricts the use of these bits in this oil and gas market to relatively weak, plastic rock formations such as shale. However, the range of rock types in which these bits can be applied is broadening as improvements in the quality of the cutter materials continue.

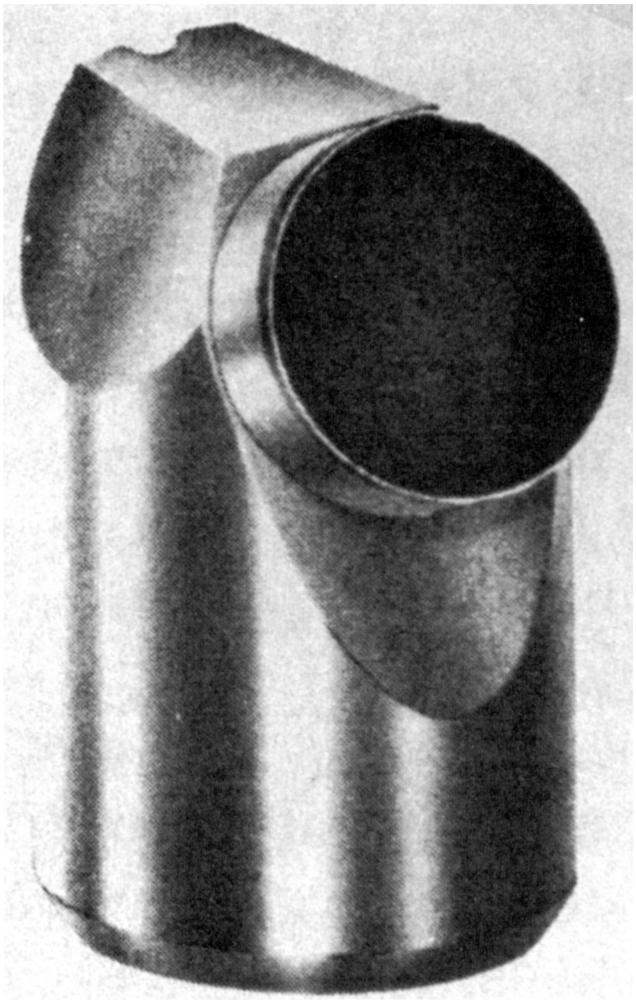


Fig. 9.1.33. Polycrystalline diamond drag bit cutter (from Bourgoyne et al., 1986, courtesy of Norton Christensen).

Fig. 9.1.34. Large diameter PDC rotary drill bit (from Bourgoyne et al., 1986, courtesy of Norton Christensen).

It is to be expected that PDC bits will find wider use in mining applications as the technology of manufacturing these tools improves, thereby reducing their tendency to fail in a brittle manner. As this market increases, the price of the tools should decrease. Lower bit prices undoubtedly would spur market interest in this new tool technology in the mining industry.

Surface-Set and Impregnated Diamond Bits—Both surface-set and impregnated diamond bits have diamonds embedded in a matrix material, such as bronze. This matrix material, in turn, is attached to the end of the drill pipe. Diagrams of this type of bit are shown in Fig. 9.1.35. The matrix of the surface set bits is selected to resist abrasion, and all of the diamonds in these bits are mounted to project from the matrix. Diamonds in the range 10 to 80/carat are used in these bits. They are arranged in the matrix in a manner that ensures overlap along the cutter path by many diamonds.

Impregnated bits, on the other hand, contain a diamond grit, typically 80 to 1000/carat, uniformly distributed throughout the matrix. In these bits, the matrix is designed to wear so that, as the diamonds on the surface become blunt, they are torn away from the matrix and new, sharp diamonds are exposed (Fig. 9.1.36).

In mining applications these bits generally are used for core drilling. The surface set bits are suitable for soft to medium-hard rock formations. The impregnated bits are used for drilling medium to very hard formations.

Roller Cone Bits—These bits usually consist of three cone elements mounted on rolling bearings. The rock is broken by indentation as the cutting elements that project from the surfaces of the cones are pressed, by a thrust force applied to the bit, into the rock surface on the hole bottom. A torque also is applied to the bit; this causes the cones to roll on their bearings and brings other cutting elements into contact with the rock face. The bits illustrated in Fig. 9.1.37 show both hardened steel teeth and cemented tungsten carbide buttons as the cutting elements. Both of these types of tool are used commonly today.

Also shown in this figure are nozzles directed at the rock face. A fluid, either compressed air or a drilling mud, is conducted down the center of the drill pipe and directed at high

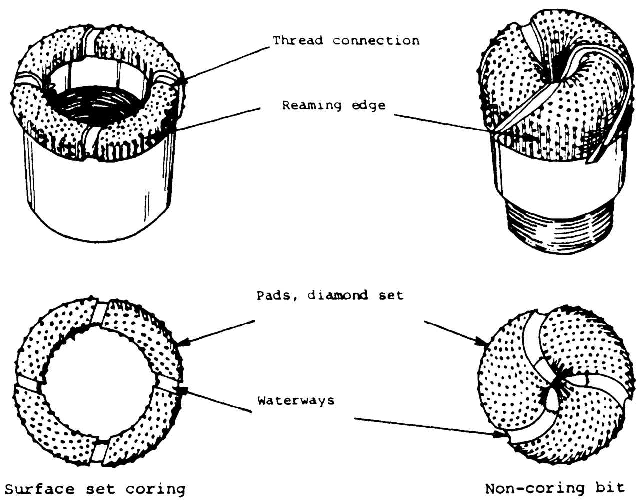


Fig. 9.1.35. Surface set and impregnated diamond rotary bits (from Sweet, 1984).

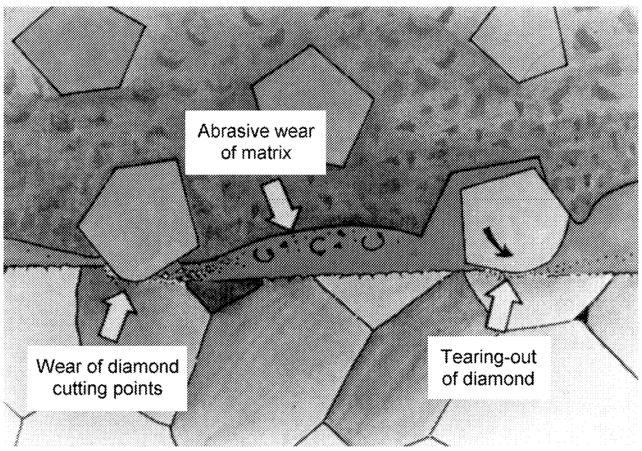


Fig. 9.1.36. Wear mechanisms of impregnated diamond bits (from Cooper, 1978).

velocity onto the face through these nozzles. This fluid serves at least three purposes. One is to cool the bearings. Another is to remove the rock fragments from the face as they are produced by the action of the cutting elements. The third is to transport this rock debris to the surface. The debris-laden fluid returns to the surface up the annulus between the drill pipe and the hole wall. In relatively shallow holes drilled in reasonably competent rock, compressed air is used as the drilling fluid. In deep wells, such as those drilled typically for oil and gas, it becomes necessary to provide a lining to the hole wall as the drilling proceeds to prevent the high in situ stresses from causing spalling of the wall rock. The drilling mud forms a cake on the hole wall and thus provides this immediate support. In most mining applications compressed air flushing is employed.

As shown in Fig. 9.1.37 each cone has two, three, or four rings of cutting elements. Considerable work has been conducted to optimize the pattern and the sequencing of the rock indentations caused by the cutting elements. This includes examining the spacing between the cutting elements in a row, the spacing between rows on an individual cone, and the interaction between

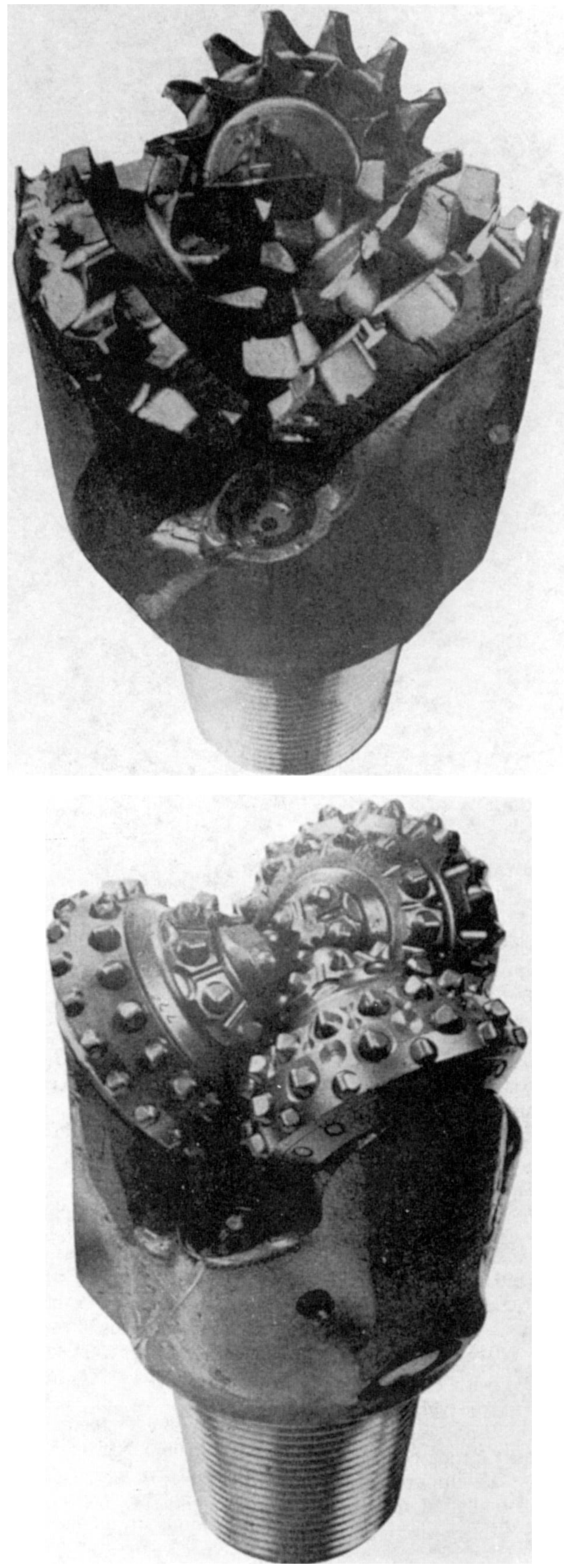


Fig. 9.1.37. Roller cone bits with a) milled steel teeth, and b) cemented tungsten carbide inserts (from Rabia, 1985, *courtesy:* Smith Tool Co.).

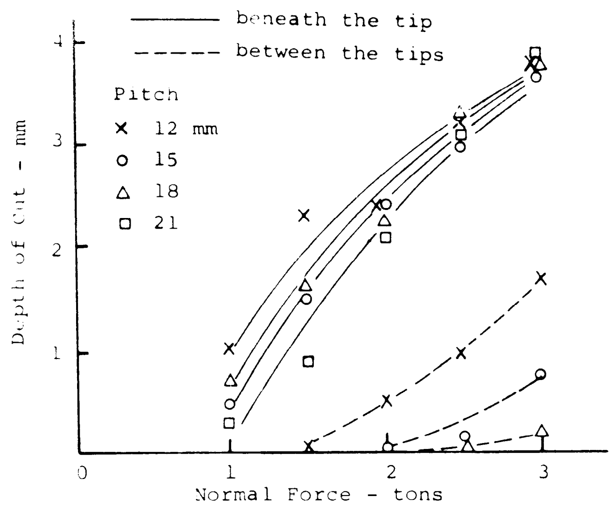


Fig. 9.1.38. Relation between thrust force and groove depth (after Takaoka et al., 1973).

cutting elements on different cones. One of the early reports in the literature on the performance of button cutters comes from Japan (Takaoka et al., 1973). These workers experimented with 7.9-in. (200-mm) diameter by 0.79-in. (20-mm) wide cutters fitted with 0.24-in. (6-mm) diameter buttons at pitches of 0.47, 0.59, 0.71, and 0.79 in. (12, 15, 18, and 20 mm). Their program involved experiments in two different type of rocks:

	Uniaxial Compressive Strength		Uniaxial Tensile Strength	
	psi	MPa	psi	MPa
Kofu Andesite	24,360	168	1,840	12.7
Sawairi Granite	19,900	137	1,320	9.1

The cutting rig applied thrust force F_T as an independent variable at levels of 1.0, 1.5, 2.0, 2.5, and 3.0 tons and measured rolling force F_R and groove geometry as dependent variables.

The observed effect of F_T on the depth of penetration is shown in Fig. 9.1.38. The authors, however, point to variations in penetration depth along the length of the groove and identify two characteristic measures of depth. As illustrated in Fig. 9.1.38a, one is a maximum, being the depth immediately beneath each button. The other is a minimum and occurs midway between adjacent buttons (i.e., at half pitch).

The results show that:

1. Maximum penetration d_1 increases with thrust force F_T at all values of pitch. The relation is generally of the form,

$$F_T \propto d_1^a \tag{9.1.37}$$

where $a > 1.0$.

2. For a given thrust force F_T, maximum penetration d_1 reduces with increasing button pitch. Takaoka, however, discounted the variation as insignificant on the grounds of the strongly opposing trend at minimum penetration d_0 and also similarly strong contrary evidence with toothed cutters.

3. Minimum penetration d_0 also increases with thrust force F_T. The relation appears to be of the form,

$$F_T \propto d_0^b \tag{9.1.38}$$

where $b < 1.0$.

4. Minimum penetration d_0 increases as button pitch s_1 decreases according to the following approximate relation:

$$F_T = A + Bd_0^c \tag{9.1.39}$$

where A and B are constants depending on button pitch and rock properties and with the exponent $c < 1.0$.

It is interesting to note the following values from Fig. 9.1.38. At pitch = 0.47 in. (12 mm), $p_0 = 0$ when d_1 = approximately 0.079 in. (2 mm); and at pitch = 0.59. in (15 mm), $d_0 = 0$ when d_1 = approximately 0.09 in. (2.3 m); pitch = 0.71 in. (18 mm), $d_o = 0$ when d_1 = approximately 0.079 in. (2 mm).

It can be deduced from these data that there is no interaction occurring between adjacent buttons when the pitch/penetration ratio exceeds about six. Takaoka made corresponding measurements of crater width (w) produced by the buttons. The results suggest that:

1. Maximum width w_1 increases with thrust force F_T in similar fashion to maximum depth d_1.
2. For a given thrust force, maximum width appears to increase only marginally with decreasing button pitch; and
3. Minimum width w_0 increases with thrust force in similar fashion to minimum depth and is clearly affected by button pitch.

Groove width was found to be linearly related to penetration depth and to be independent of button pitch. This implies the relation,

$$Q \propto d^2 \tag{9.1.40}$$

where Q is the volume of cut groove and d is the depth of cut groove

In a later study Savadis (1982) conducted similar work in other rock types and confirmed the validity of this expression.

Now, Takaoka's work implies:

$$Q \propto F_T^2 \tag{9.1.41}$$

and if Eqs. 9.1.40 and 9.1.41 are valid, it follows that

$$F_T \propto d \tag{9.1.42}$$

a relation which is clearly at variance with Fig. 9.1.38 and Eq. 9.1.37. So the probability is that 9.1.41 is, in fact,

$$Q \propto F_T^a \tag{9.1.43}$$

where $1 < a < 2$.

Takaoka and his coworkers also report values for rolling force F_R, which, as illustrated in Fig. 9.1.39, increase linearly with thrust force F_T. The peak F_T/F_R ratio ranges from about 10 to 15 according to thrust force and pitch and, thereby, according to penetration.

The method employed in practice to design these bits is to provide each cone with a row of cutting elements to cut the hole gage. These elements are known as the heel teeth, and the number of teeth per unit length of cut is greater along the gage than elsewhere across the rock face because of the difficulty, experienced in all machining operations, of removing material from a corner. The disposition of the cutting elements everywhere other than the gage row is such that the teeth intermesh. In other words, a goal of good bit design is to have the various cutting elements produce indentations across the entire rock face, while

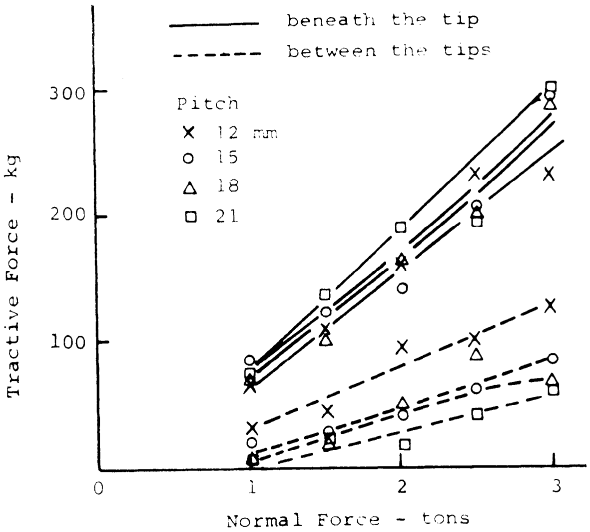


Fig. 9.1.39. Relation between normal and tractive forces (after Takaoka et al., 1973).

at the same time to ensure that no two elements track in the same indentation hole.

It is apparent that the maximum rate of advance per revolution of the drillbit is limited to the height of the tooth from the surface of the cone. The height of the steel teeth above this surface is greater than that of the buttons; consequently, steel-toothed bits would be expected to drill faster than button bits. This conclusion, however, must be qualified because in strong and/or abrasive rock types the steel teeth blunt rapidly and this reduces their effectiveness as indentation tools. Accordingly, their use is limited to relatively weak rock types and, indeed, in these formations, the rate of drilling is greater with steel-toothed bits.

The strength and abrasivity of the rock to be drilled has other influences on the design of roller cone bits. In very strong formations, it is important that the cones roll true, that is, without the cutting elements skidding across the rock face. This requirement arises because skidding in hard rocks results in rapid wear of the cutting elements. On the other hand, in rocks that are not excessively strong, the introduction of even a slight amount of skidding to the cutting elements causes significant reduction in the normal, or indentation, stress required to cause rock failure. An analysis to explain this phenomenon was provided by Hood (1977) who showed that a small shear force applied to an indenter pressing normally into an elastic surface caused considerable distortion to the stress field induced in the elastic material. This produces cracks in the elastic material ahead of the indenter at substantially lower levels of the normal stresses applied by the indenter. Two methods are used to introduce skidding of the cutting elements. One is to mount the cone with the cone apex away from the center of bit rotation. In Fig. 9.1.40a, the cone apex is at the center of rotation and thus the cone will roll true. In Fig. 9.1.40b, the cone has two angles; neither of these angles coincide with the center of bit rotation. Since the cone is constrained to rotate about the bit axis, the elements skid. This skidding action can be further increased by

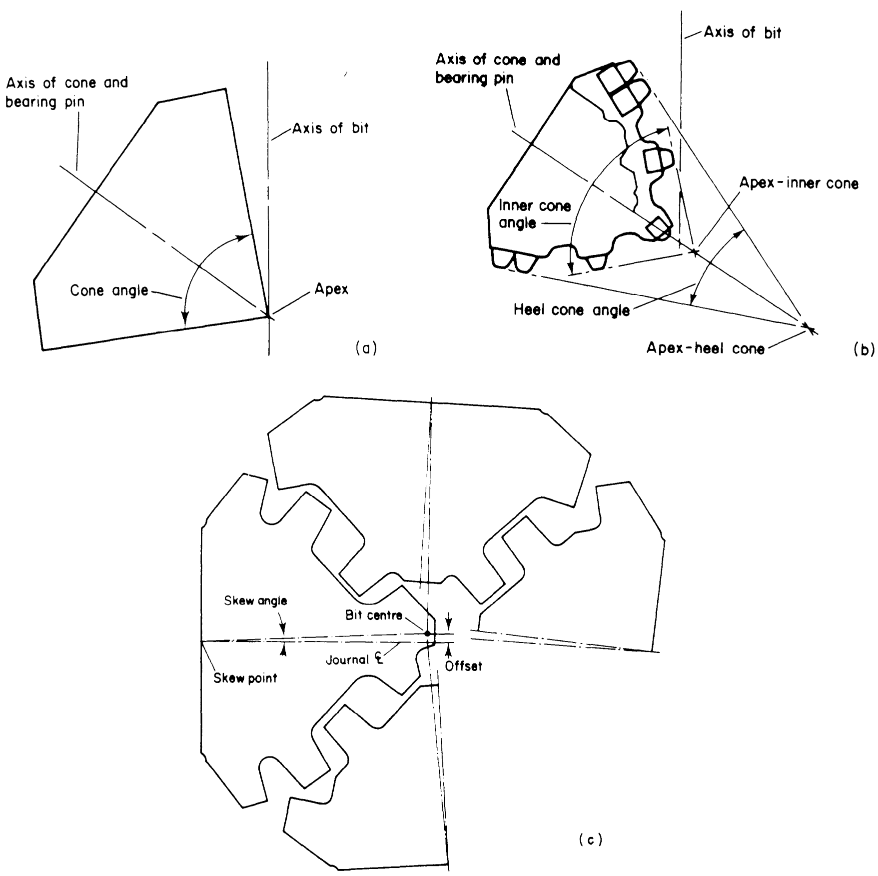


Fig. 9.1.40. Bit cones showing (a) the condition for true roll, (b) different cone apex angles, and c) cone offset angles (from Rabia, 1985).

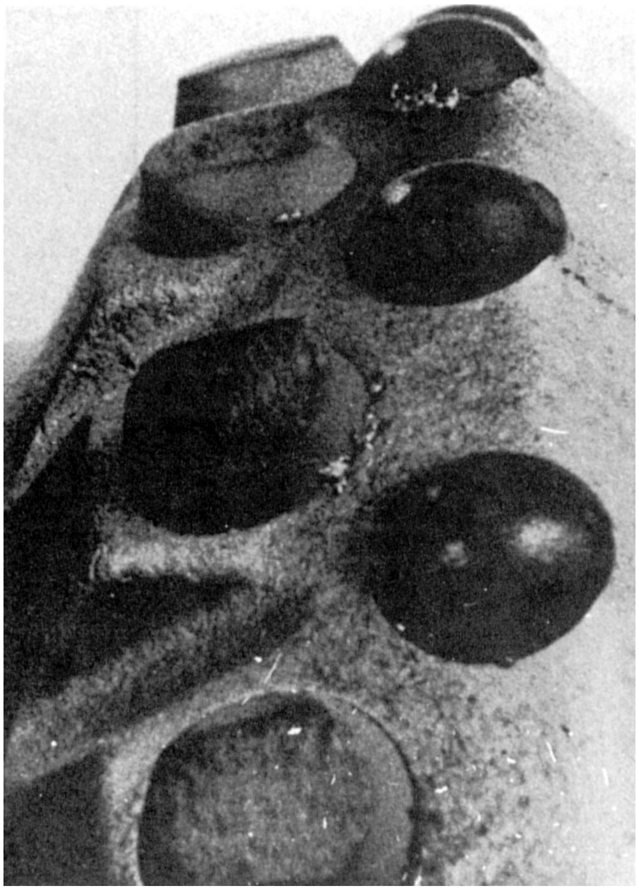


Fig. 9.1.41. Roller cone bit with PCD buttons in gage row (from Reinsvold et al., 1988).

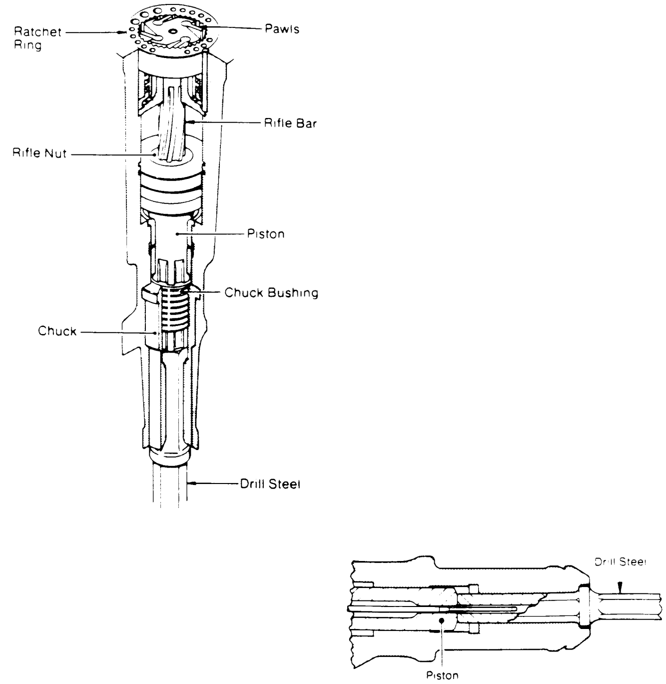


Fig. 9.1.42. Internal components of a percussive drill (from Kurt, 1982).

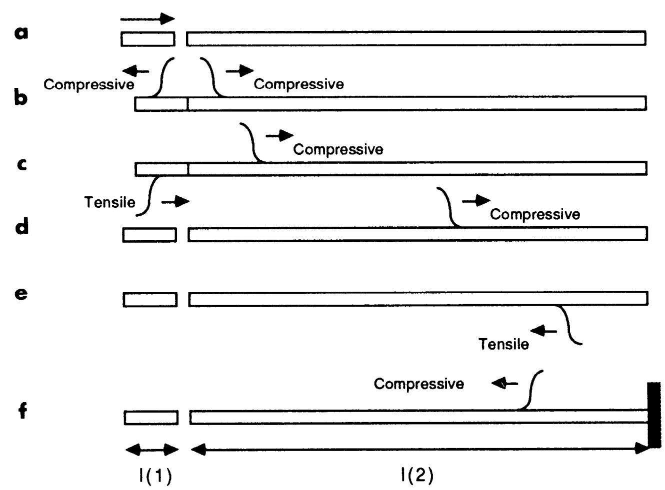


Fig. 9.1.43. Stress waves in piston and drill steel.

offsetting the centerlines of the cones from the center of bit rotation (Fig. 9.1.40c). An offset angle of 2° might be typical for bits used to drill medium-strength rock.

In strong rock, even when these inserts are mounted in cones that roll true, wear of the cemented tungsten carbide buttons remains one of the principal reasons for bit failure. The inserts that suffer the most damage are those on the heel row. Drilling experiments have shown that significant increases in bit life can be achieved when conventional WC buttons are replaced with PDC buttons in this heel row. An example of a bit with the gage row fitted with PDC buttons and with conventional WC buttons in the other rows is shown in Fig. 9.1.41. The wear of the WC inserts is apparent, whereas the PDC buttons remain, even in this heel row, in pristine condition. Again, although the technical advantages are recognized, because of the high cost of the PDC buttons, bits equipped with these buttons have not, to date, found widespread application in the mining industry.

The other principal cause of bit failure arises from deterioration of the bearings. Considerable improvements have been made in bearing technology over the past few decades. This has enabled the loads applied to the bits to be increased. Work currently is being conducted to determine the feasibility of replacing conventional rolling bearings with PDC face bearings (Sneddon and Hall, 1987). The advantages claimed for this new diamond bearing are lower friction and greater wear resistance. This could lead to the ability to apply greater loads to the bits, and thereby produce greater rates of penetration, and longer bit life.

9.1.3.2 Drilling Machines

Percussive: *Fundamental Considerations*—Fig. 9.1.42 shows the essential features of a percussion drill. A fluid, either compressed air or hydraulic fluid, causes the piston in the drill to reciprocate and to strike the drill steel in a repetitive manner (Fig. 9.1.42b). Energy is imparted to the rock through the drillbit by the compressive stress wave induced in the drill steel by this impacting action. When the piston strikes the steel (Fig. 9.1.43a), compressive stress waves are set up in both the piston and the drill steel (Fig. 9.1.43b). These waves travel, at velocity *c*, to the ends of the piston and the drill steel, respectively (Figs. 9.1.43c and d). Here they are reflected. The nature of these reflected

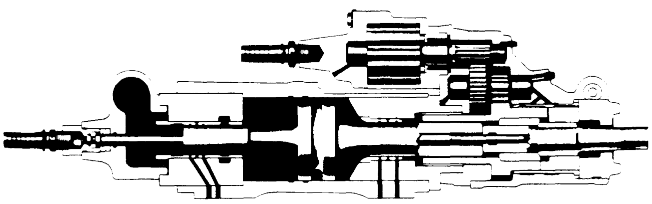


Fig. 9.1.44. Rotary-percussive drill (from Kurt, 1982).

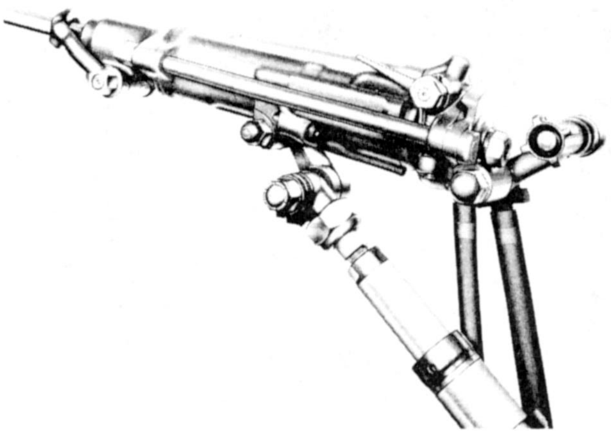


Fig. 9.1.45. Jackleg (from Kurt, 1982, *courtesy:* Ingersoll Rand Co.).

waves depends on the boundary conditions at these ends. For a bar with a free (i.e., unsupported) end, the compressive stress wave is reflected as a tensile wave (Figs. 9.1.43d and e), whereas for a bar with a fixed (i.e., rigid) end, the incident compressive wave is reflected as a compressive wave (Fig. 9.1.43f).

The end of the piston remote from the point of impact will always be a free end, and hence the wave reflected from this end will always be tensile (Fig. 9.1.43c). The end of the drill steel remote from the point of impact generally will be in contact with the rock. Because the rock deforms when the incident wave arrives at the drill steel:rock interface, this boundary is neither free nor fixed but some combination of these conditions. If all of the energy in the incident wave were employed either usefully in causing rock fracture or in frictional heating, then, obviously, there would be no reflected wave. In general, this is not the case, and some fraction of the energy is reflected. The nature of this reflected wave (tensile or compressive) depends on the extent of the contact between the drill steel and the rock and on the rock properties.

The piston remains in contact with the drill steel until a tensile wave arrives back at the piston-drill steel interface. This tensile wave causes these two bodies to separate (Figs. 9.1.43c and d). This terminates the compressive stress wave that has been set up in the other body. This enables the length and the duration of this stress pulse to be calculated. Consider the case where the piston has a length l_1, the drill steel length is l_2, and $l_1 < l_2$. Consider also the case where the diameters of the piston and the drill steel are the same; then the length of the compressive

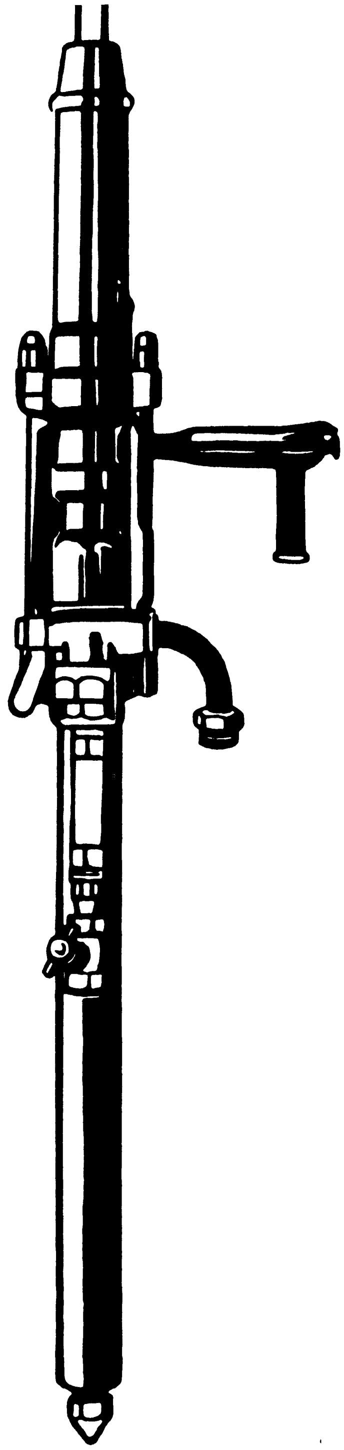


Fig. 9.1.46. Stoper (from Kurt, 1982, *courtesy:* Ingersoll Rand Co.).

wave set up in the drill steel is $2l_1$. The time t_s during which this pulse acts is given by

$$t_s = \frac{2l_1}{c} \tag{9.1.44}$$

Grantmyre and Hawkes (1975) observed that the yield strength of the steel used for the piston and the drill steel limits the stress that can be applied to the rock using the piston-type device. They noted that the maximum stress in the piston σ_h and in the drill steel σ_t are given by

$$\sigma_h = \rho c v \frac{A_t}{(A_t + A_h)} \qquad \sigma_t = \rho c v \frac{A_h}{(A_t + A_h)}$$

where A_t is the cross-sectional area of the drill steel, and A_h is the cross-sectional area of the piston.

Thus, for the case where $A_h = A_t$, $\sigma_h = \sigma_t = \rho c v/2 = 866.5v$ psi where v is given in feet per second (19.63v MPa where v is given in meters per second). In order to prevent damage to the drill steel or the piston, generally it is desirable to limit the stresses in these components to about 31,000 psi (212 MPa); thus the maximum piston velocity $v \approx 35.4$ fps (10.8 m/s). Grantmyre and Hawkes noted that, in some machines, appropriate design of the components has enabled piston velocities as high as $v \approx 50$ fps (15.25 m/s) to be used. This constraint applies only to hydraulic drills, because the piston velocities of pneumatic drills are a factor of 20 or more less than these limiting velocities.

Conventional Drills—An essential feature of any drill is that rotation of the bit takes place between successive impacts. This rotation is required because the effectiveness of the drilling process is impaired greatly by the cushioning effect of the crushed rock from the initial impact, if the bit is constrained to strike in the same part of the rock face. The rotation is achieved either using a mechanism internal to the drill, known as a rifle bar, or using an external motor. The rifle bar arrangement, shown in Fig. 9.1.42a, is an inclined spline and a ratchet. The rifle bar is held fixed by the ratchet mechanism, and the rifle nut in the head of the piston causes the piston to rotate on the return stroke. Since the piston is connected to the drill steel through the chuck nut and the chuck (Fig. 9.1.42a), this also causes the drill steel to rotate. The ratchet allows the rifle bar to rotate on the power stroke, and thus the piston and the drill steel do not rotate on this stroke.

An arrangement for independent rotation of the drill steel is shown in Fig. 9.1.44. Although independent rotation generally is regarded as advantageous because it provides control over the rotational speed and torque, it has the disadvantage of adding considerably to the weight of the drill. Consequently, this type of drill typically is used only on mechanized drill rigs.

It is important also that the drill is kept adequately lubricated during the drilling operation. This lubrication is achieved with pneumatic drills by placing an in-line oiler in the compressed air line immediately upstream of the drill. The air entrains oil as it passes through this oiler.

Drills may be either hand-held or machine-mounted. The pneumatic hand-held drills are the jackhammer (or sinker), the jackleg, and the stoper. Machine-mounted drills are known as drifters. A jackhammer is used for mine utility work, such as drilling anchor holes and holes for boulder breaking. It is also used for shaft sinking. A jackleg is a jackhammer with a pneumatic cylinder hinge-mounted to it. The cylinder is used to support the weight of the drill and to provide thrust to the drill during the drilling operation (Fig. 9.1.45). Jacklegs are used in small drifts and headings and in stopes. They are lightweight and versatile. Stopers are used for drilling upholes. They are jackhammers with a pneumatic cylinder rigidly attached to the drill (Fig. 9.1.46). The typical sizes and lengths of holes drilled by these machines are as follows:

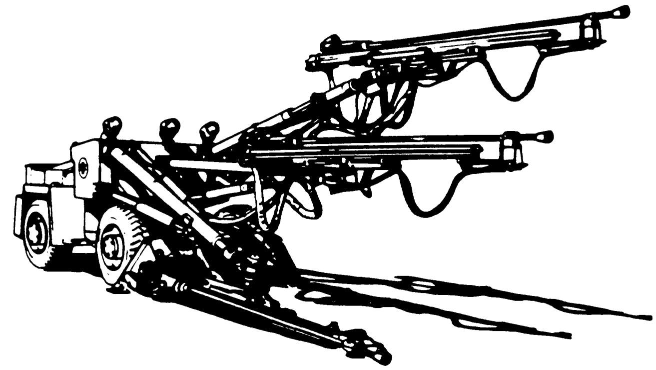


Fig. 9.1.47. Jumbo drill rig (from Roos, 1982, *courtesy:* of Gardner-Denver Co.).

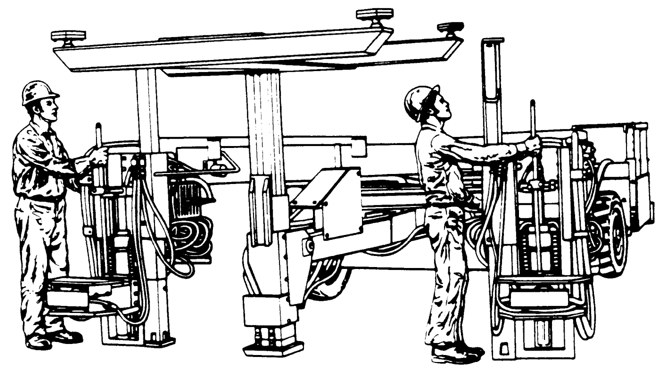


Fig. 9.1.48. Roofbolting drill rig (from Roos, 1982, *courtesy:* Gardner-Denver Co.).

	Typical Hole Sizes		Typical Hole Lengths	
	in.	mm	ft	m
Jackhammers	0.75–1.25	19–32	1–4	0.3–1.2
Jacklegs/Stopers	1.25–1.75	32–44	4–12	1.2–3.7
Drifters	1.50–2.25	38–57	4–100	1.2–30.5

Most of the drills in service today are pneumatic. These drills have the following advantages: low cost, traditional and well-established technology, simplicity of mechanical components, and, in many mines, an existing compressed air distribution system. However, these drills suffer from the disadvantages of having very low efficiency (often less than 5% of the input power to the compressor is delivered to the rock) and creating a poor environment for the machine operator (the drills are noisy and the drill exhaust generates a fog). These disadvantages are overcome to a large extent by hydraulic drills. The overall efficiency of a hydraulic drill is in the range 25 to 30%, the noise level of these machines is 8 to 10 decibels less than that of pneumatic drills, and no fog is generated. The principal disadvantages of hydraulic drills are that they are technically more sophisticated and the capital cost of the units is higher.

Hydraulic drills are finding application as hand-held units, but mostly they are found mounted on a drill jumbo. This term is applied to a self-propelled machine equipped with one or more arms fitted with drill rigs. Drill jumbos, such as that illustrated in Fig. 9.1.47, frequently are used for tunnel or developments drivage. Other configurations (Fig. 9.1.48) are used for drilling

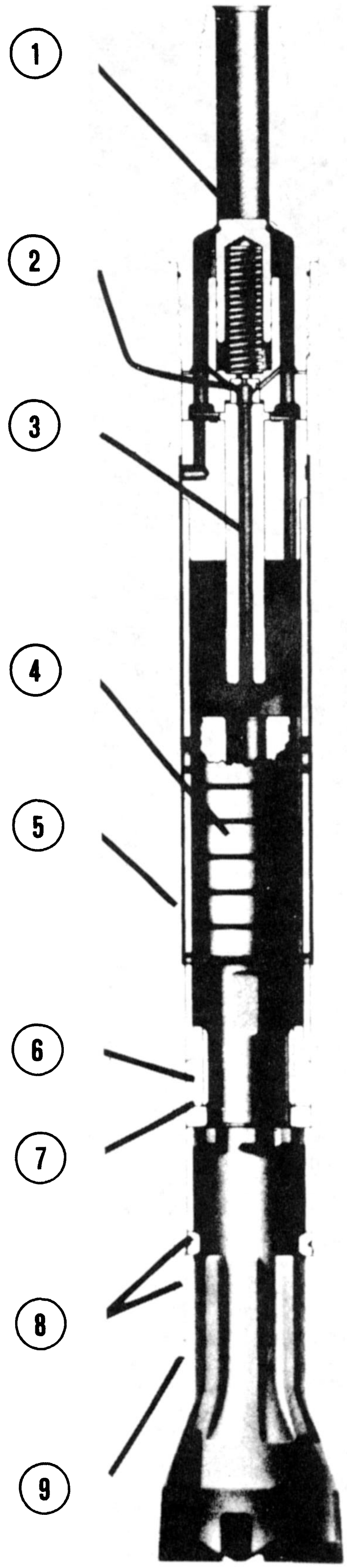


Fig. 9.1.49. Down-the-hole drill. (1) check valve; (2) air metering plug; (3) air distributor; (4) piston; (5) cylinder; (6) piston-stem bearing; (7) washer; (8) bit retaining ring; (9) splined chuck (from Anderson, 1984).

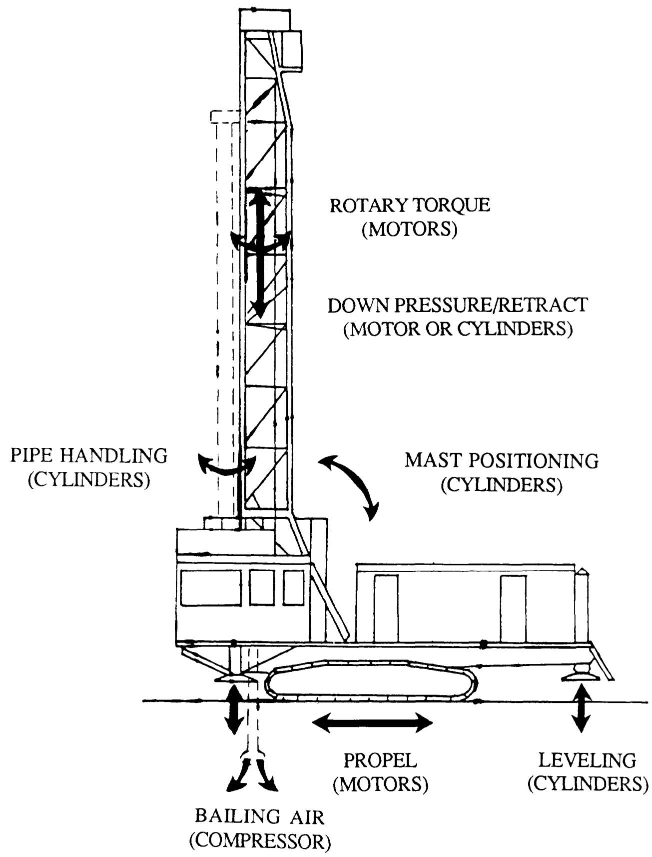


Fig. 9.1.50. Schematic of a surface rotary drill rig.

roof bolt holes, and yet other configurations are used for shaft sinking.

Down-the-Hole Drills—The previous discussion makes it clear that in a conventional percussive drill the mechanism that imparts energy to the rock remains outside the hole and this energy is transmitted in the hole through the drill rod(s) and the bit. As the name, down-hole (DTH) drill implies, with this percussive unit the mechanism for imparting energy to the rock is located in the hole. Fig. 9.1.49 shows the main features of a DTH drill. The piston in these drills is powered by compressed air. Holes 4 to 8 in. (100 to 200 mm) in diameter up to 500 ft (150 m) long can be drilled with a DTH drill (Anderson, 1982). This system eliminates energy transmission losses in the drill rods; thus a major advantage of a DTH drill is a constant penetration rate that is independent of hole depth.

Rotary Drills: Small-diameter rotary drills are used for drilling weaker rock types. These drills usually employ a rotary hydraulic motor to provide the drill torque, and a hydraulic cylinder to provide the thrust. They are mounted, in a similar manner to percussive drills, on drill jumbos for face drilling and for roof drilling.

Large-diameter holes in both underground and, more frequently, in surface mines typically are drilled with roller cone bits. The essential features of a rotary drill rig for a surface mining operation are shown in Fig. 9.1.50. These features include a motor as the rotary drive, a chain pull-down arrangement operated by hydraulic rams to provide thrust, a rod handling device, a compressor for flushing the rock debris from the hole bottom, hydraulic leveling jacks, a dust collection system, and a main motor, either diesel or electric.

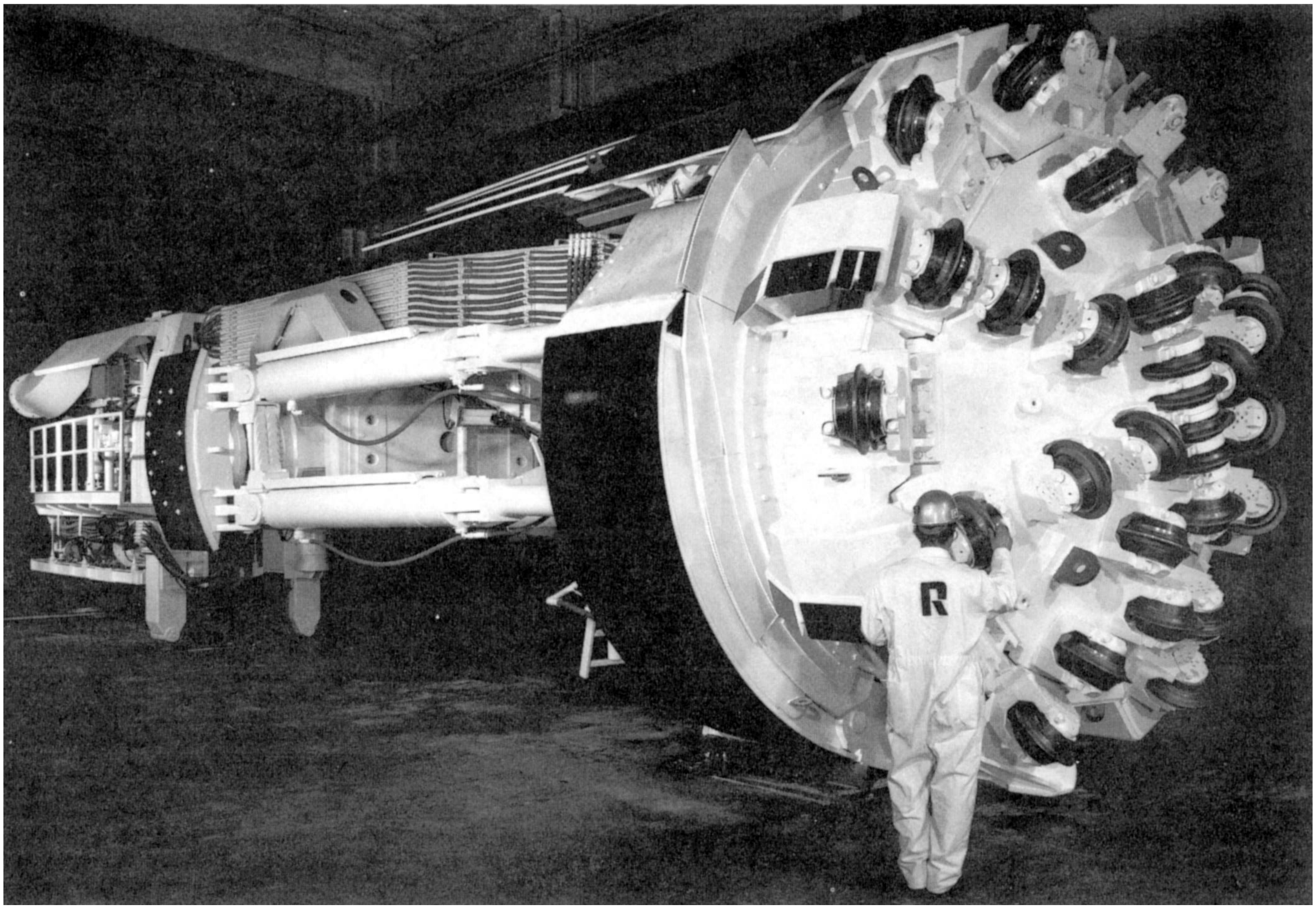


Fig. 9.1.51. A tunnel boring machine (*courtesy:* Robbins Co.)

9.1.4 EXCAVATION MACHINES

9.1.4.1 Tunneling and Boring

Coverage here is restricted to the fundamentals of rock excavation machines. For applications to rapid excavation techniques see Chapter 22.1.

Tunnel Boring Machines (TBM): This type of machine can be used to drive circular tunnels from 5.7 ft (1.75 m) to more than 36 ft (11 m) in diameter in rock types that range from weak, loosely consolidated to very strong and abrasive. In almost all cases breakage is effected by roller cutters mounted on the cutting head (Fig. 9.1.51). Because these cutters break the rock by indentation, these machines are characterized by very high thrust requirements. This thrust is provided by hydraulic rams that press the cutterhead into the rock face. The thrust reaction force is reacted through gripper pads that are pressed, again hydraulically, against the tunnel walls. The rock broken from the face by the cutters falls to the floor where it is scooped into buckets mounted around the gage of the cutterhead. This debris is lifted in the buckets to the tunnel crown, whereupon it is tipped onto a belt conveyor that runs through the center of the machine.

The most common type of cutting tool employed on these machines is the disk cutter. In some cases the cutting edge of this tool is a hardened steel surface (Fig. 9.1.52), and in other cases it is a row of cemented tungsten carbide buttons that are press-fitted into the disk rim. The most common cross section for a hardened steel disk cutter several years ago was a wedge (Fig. 9.1.53a). The disadvantage of this design is that the tool area presented to the face increases as 2 tan q (where q is the half angle of the wedge) as the cutters become worn. Thus, in order to maintain penetration rate as the cutters become blunted, it is necessary to increase both the machine thrust force and torque. The majority of disk cutters used today are "constant section" disks (Fig. 9.1.53b). Appreciable wear of these cutters can occur without causing a substantial increase in the surface area presented to the rock. Other cutter types include the kerf cutter, which is simply multiple disks mounted on the same hub (Fig. 9.1.54a), and the pineapple cutter, which is a frustrum with cemented tungsten carbide buttons press-fitted onto the surface (Fig. 9.1.54b).

A ranking of cutting efficiency of these tool types, in terms of specific energy, places the steel disk cutter as the most efficient, the disk-button cutter next, and the pineapple cutter as least efficient. However, the wear resistance, and therefore the capability of cutting strong abrasive formations, is the reverse of this efficiency ranking. Consequently, steel disks tend to be used for cutting weaker, less abrasive rocks, and pineapple cutters are used for machining the most abrasive and toughest formations.

Several other comments can be made about the cutting behavior of disk cutters. First, in contrast with drag bits, the efficiency of the rock breakage process does not decrease when disk cutters are used in a groove deepening (when multiple passes are

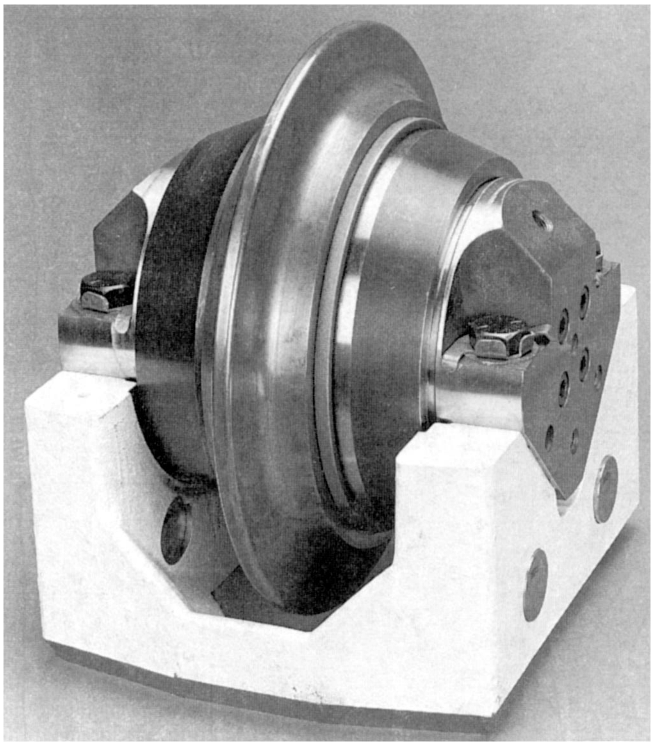


Fig. 9.1.52. Disk cutter (*courtesy:* Robbins Co.).

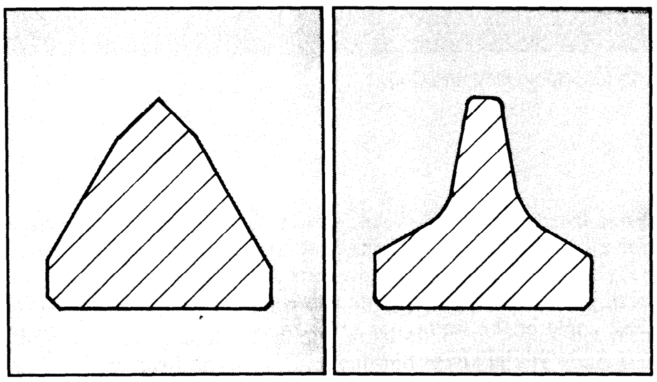


Fig. 9.1.53. Wedge and constant section disk cutters (*courtesy:* Robbins Co.).

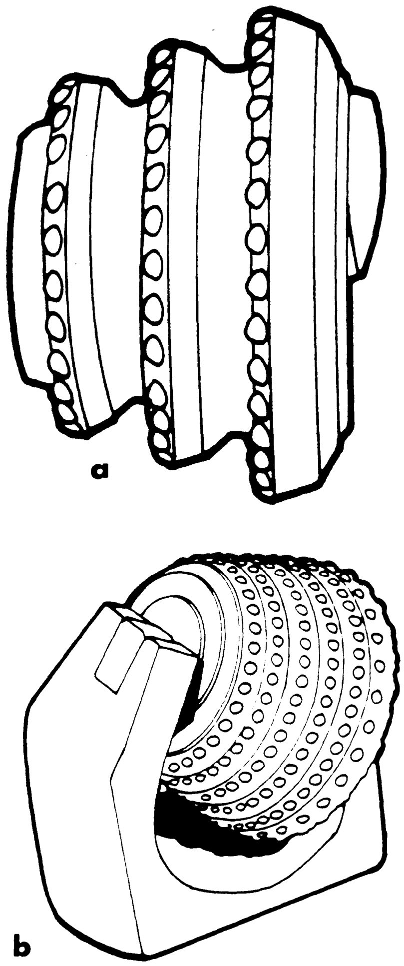


Fig. 9.1.54. (a) Kerf cutter (b) Pineapple cutter.

made by a tool taking a series of shallow cuts in a kerf before producing major rock chips) mode. This is fortunate because in practice, groove deepening is the cutting procedure most commonly employed with tunneling and with other boring machines. Second, similar to the findings for drag bits, an optimum spacing exists between an array of disk cutters working a rock face. The value of this optimum spacing depends on the depth of cut taken and on the rock type. However, whereas with drag bits an optimum *s/d* value of 2 to 3 is typical (Fig. 9.1.25), with disk cutters this value is more typically in the range 5 to 10. Third, the efficiency of the rock breakage process is independent of whether the grooves are cut simultaneously, with multiple disks on a single hub, or sequentially, with independent disks.

A factor crucial to cutter longevity, particularly in hard and abrasive formations, is whether the cutters roll true, that is without skidding, across the rock face. This factor was discussed in connection with tricone roller drillbits in the previous section. The two possible modes of disk cutter skidding are illustrated in Fig. 9.1.55a. The mode that Hood (1978) described as ploughing can apply to any type of roller cutter. In this case the cutter acts in part as roller cutter and in part as drag bit. This dragging or ploughing action can greatly exacerbate cutter wear. Problems with this type of wear occur frequently with cutters mounted near the center of the cutterhead. The spinning type of cutter skidding (Fig. 9.1.55b) occurs with kerf and pineapple cutters that are mounted on the head in a manner such that the cone apex of the cutter frustrum is not coincident with the axis of the cutter head. This type of skidding action also results in accelerated cutter wear in strong or abrasive rock types.

Over the past two decades full-face rock tunneling machines have become very widely used in civil construction applications. A principal advantage is that this type of machine represents a quasi-continuous excavation system, compared with the alternative, cyclic, drill-and-blast method. Hence the tunnel advance rates generally are much greater when a TBM is employed. For example, in a 12 ft (3.6 m) tunnel, typically the face will advance with drill-and-blast working by one blast round, say 10 ft (3 m), per shift. Thus on a 3-shift a day operation the tunnel advance is 30 ft (9 m). A TBM, in the same tunnel would be expected to

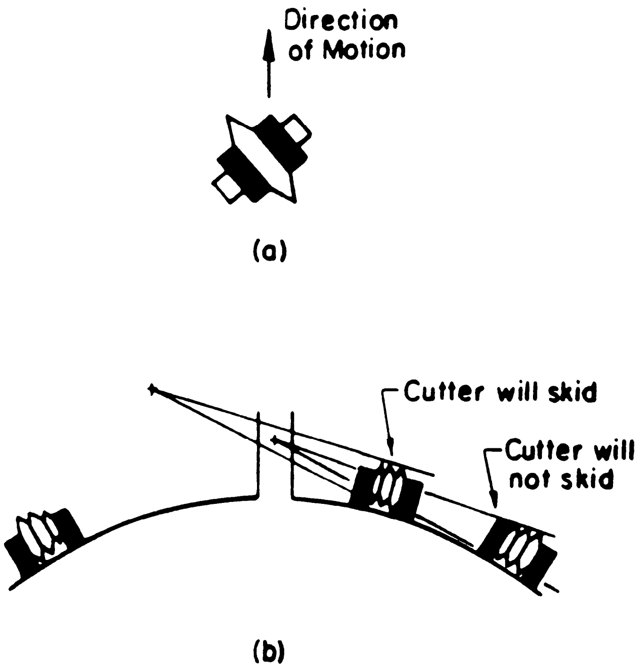


Fig. 9.1.55. (a) Ploughing mode and (b) spinning mode of cutter skidding.

advance about 33 ft (10 m) per shift, or 100 ft a day (30 m a day) on 3 shifts.

The disadvantages of this machine are:

1. High capital cost (several million dollars)
2. Tunnel cross section is necessarily circular
3. Large turning radius (typically 330 ft or 100 m)
4. Cumbersome initial machine and system installation. In practice this means that a minimum tunnel length of about 1.2 mi or 2 km is needed to justify the cost of machine setup (Handewith and Dahmen, 1982).

TBMs have not, as yet, found a wide market in the mining industry. The reasons for this probably are partly related to the initial capital cost and to the conservatism of the industry. It is more likely that the other constraints of this tunneling system—inflexibility of geometry, inability to turn the tunnel rapidly, and the requirement for long, relatively straight tunnels—are the main factors that have limited the interest of the mining community in this now well-established technological area.

Mobile Miner: The Robbins Company in Seattle has developed a new excavation machine, termed the “mobile miner” (Fig. 9.1.56) to overcome these limitations of conventional TBM systems in mining applications. The cutting head on this machine is a wheel with disk cutters mounted around the circumference. Only a portion of the rock face is contacted by the cutterwheel at any point in time; thus the number of cutters in contact with the face is small and consequently the reaction forces on the machine are less than for a TBM driving an equivalent size opening. This allows for a smaller, and therefore more maneuverable, machine than a TBM.

The cutterwheel is connected by a boom to the main body of the machine. The machine cuts by first anchoring the machine with staking jacks. The wheel then is rotated, and the cutters are sumped to the desired cut depth into the rock; this is effected with hydraulic sumping cylinders plunging the boom forward on one side of the heading. Hydraulic swing cylinders then traverse the boom across the width of the heading. Boyd (1987) observes that the cutting action of the disk cutters during this swing part of the cycle is similar to that experienced by gage cutters on a TBM. At this point the wheel is sumped again, and another cut is taken. These cycles are repeated until the sumping cylinders are fully extended (about 30 in. or 760 mm), at which point the staking jacks are retracted, the machine advances on its crawlers, and the process is repeated. The machine drives a tunnel that is rectangular with rounded corners (Fig. 9.1.57). This is advantageous both from a mining viewpoint, because it is maximizes the utility of the excavated opening and it is suitable

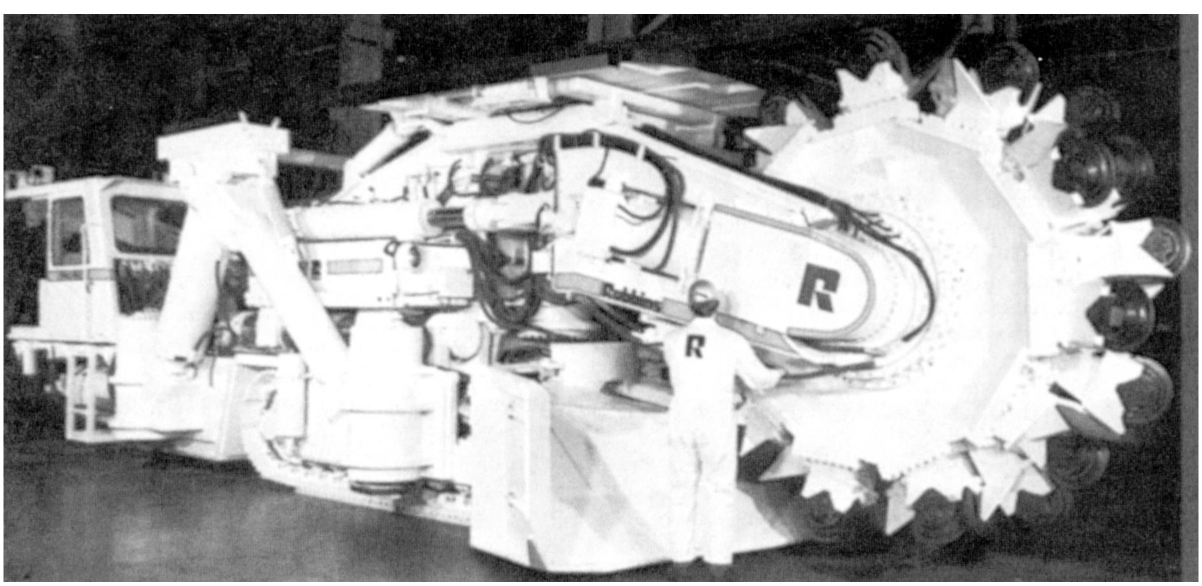


Fig. 9.1.56. Robbins Mobile Miner MM120-3001 (*courtesy:* Robbins Co.).

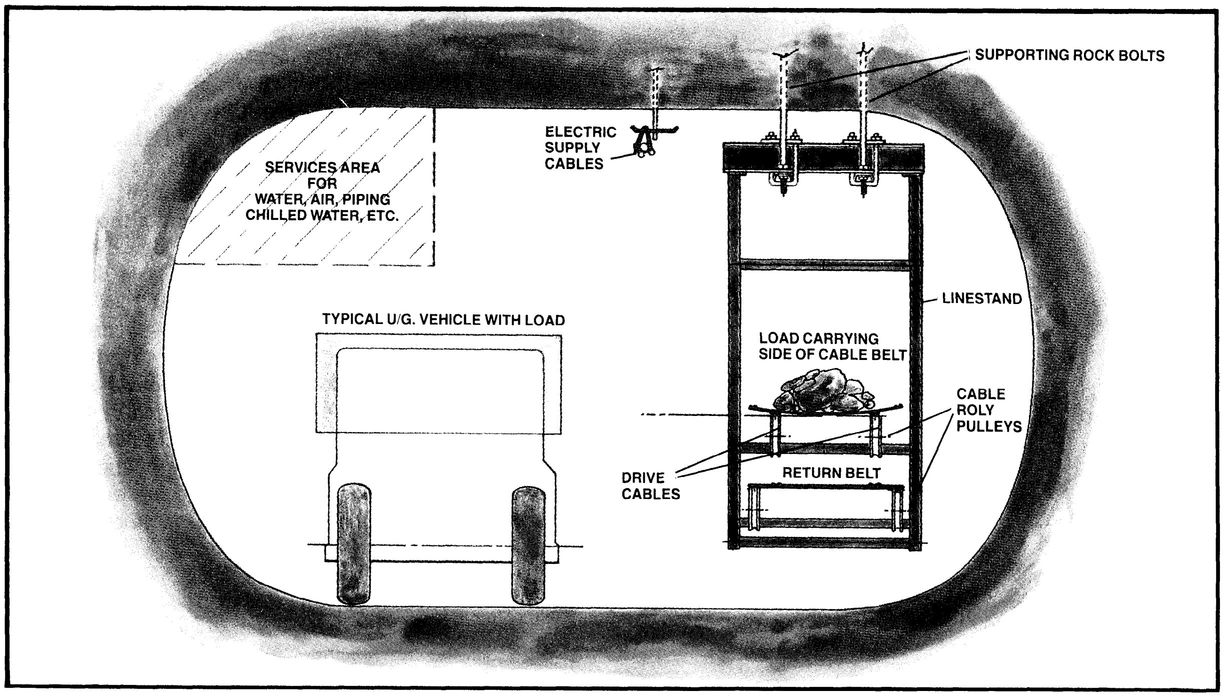


Fig. 9.1.57. Geometry of excavation cut by Mobile Miner (from Boyd, 1987).

for road transport, and from a rock mechanics viewpoint, because the rounded corners avoid the high stress concentrations usually associated with square or rectangular tunnels.

To date, only two machines of this configuration have been built. A prototype unit was tested in a quarry in Seattle. Subsequently, the first production machine was built and used to drive a decline in strong, abrasive quartzite at the Mount Isa mine in Australia. Although several problems were encountered with this first machine in the very difficult rock conditions into which it was placed, the results obtained were sufficiently encouraging that work to develop this machine concept is continuing. Another mobile miner has been ordered for use at Broken Hill Pty. mines.

Raise, Blindhole, and Shaft Borers: These machines use the same roller cutter technology described above, but they drive vertical or steeply inclined holes rather than tunnels.

Raise Borers—These machines are used to produce a circular excavation either between two existing levels in an underground mine or between the surface and an existing level in a mine. The principle of raise boring is shown in Fig. 9.1.58. First, the boring machine is set up on the upper level, and a small-diameter (of the order of 9 in. or 230 mm) pilot hole is drilled, usually with a tricone bit, down to the lower level. When this hole is completed, the drillbit is removed and replaced by a reamer head having a diameter with the same dimension as the desired excavation. Some type of roller cutters are mounted on the reamer head. This head then is rotated and pulled back up towards the machine. The rock debris falls by gravity into the lower excavation where it is removed.

These machines are very effective in driving raises, and they have become very popular, particularly in hard-rock underground mines. Frequently the direct costs of driving a raise, in terms of dollars per foot (dollars per meter), are reduced by using these machines. In addition, however, this raising system offers other significant advantages. One of these is improved safety—conventional drill-and-blast raising is notoriously dangerous. Another is improved excavation rates and improved productivity. Also the circular shape combined with the lack of blasting damage results in an excavation of greater strength and integrity than a hand-driven raise.

Blindhole Borers—A blindhole machine will produce a circular excavation without the need for a pilot hole. The machine is set up on a level in an underground mine, and a steeply inclined hole, generally several feet (meters) in diameter, is excavated upward from this level (Fig. 9.1.59). The purpose of this hole often is to make connection with a stope above the level. South Africa, in particular, makes extensive use of these machines in their gold mines.

Shaft Borers—This technique for large-hole drilling has been practiced for decades at the Nevada Test Site (NTS) and, in recent years, attention to mechanized shaft sinking for mining operations has increased. The experiences of drilling in the welded tuff found at the NTS are described in a paper by Lackey (1983), and these are summarized as follows. The uniaxial compressive strength of this tuff ranges from 4000 to 25,000 psi (28 to 170 MPa). The hole sizes drilled typically are 5.2 to 7.9 ft (1.6 to 2.4 m) in diameter and 1600 ft (490 m) deep. Hardened steel mill tooth, steel disk, kerf, or pineapple cutters are used depending on the rock strength.

Cuttings removal from the hole is achieved using either reverse circulation or dual-string circulation (Fig. 9.1.60). In both cases, air jets are injected into the upstream flow to help lift the cuttings. Efficient removal of the drill cuttings remains the single greatest limitation of blind shaft drilling (Pigott, 1985). It is apparent that the problem with cuttings removal is the large quantity of the rock debris, because of the large hole size, and that, unlike the situation for raise and blindhole boring, these cuttings must be lifted against gravity to the surface. According to Lackey (1983), the positioning of the fluid jets to clear the rock chips from the working face remains an empirically determined art.

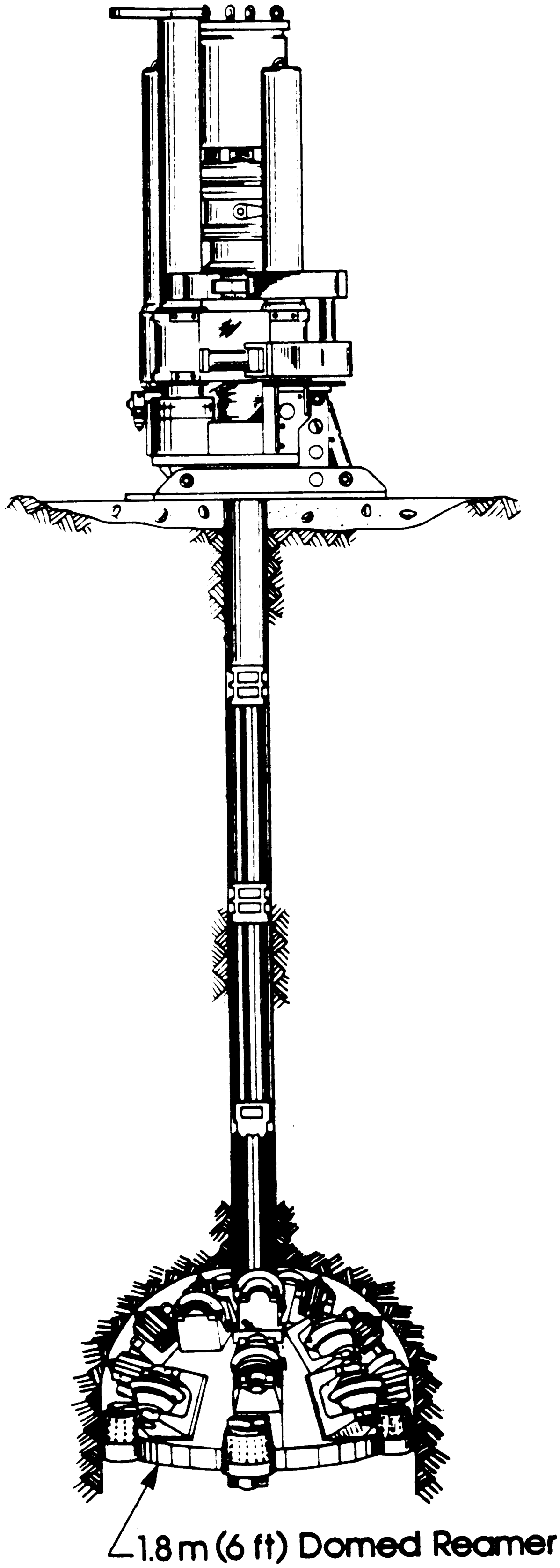


Fig. 9.1.58. Raise borer operation (from Home, 1982, *courtesy:* Robbins Co.).

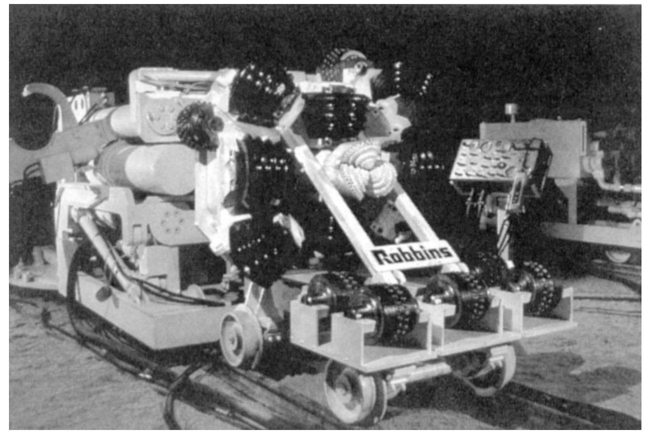


Fig. 9.1.59. Blindhole borer (*courtesy:* Robbins Co.).

The drilling machines used for these large holes are scaled up versions of standard oil field drilling systems. The right-hand diagram in Fig. 9.1.61 shows the down hole assembly, and the left-hand diagram shows the assembly, at the hole collar.

Drilling accuracy for these large holes is achieved either using the "plumb bob" effect or using a stinger in a predrilled pilot hole. The former approach uses gravity acting on the mass of the cutting head assembly to maintain the shaft vertical. Generally, less than 50% of the weight of the cutting head is applied as a thrust force to the cutters; the rest of this weight is reacted by a tensile force applied at the drilling mast and transmitted through the drill string. This large mass on a long string tends to maintain a vertical position. The latter approach employs a pilot hole, some 12 in. (300 mm) in diameter, drilled before the boring head is installed. A stinger on the boring head then is fitted into the pilot hole and this hole serves to guide the head. At the NTS, where no pilot holes are used, the accuracy of the drilled holes is reported to be somewhat more than 1 in 100. This accuracy probably is unacceptable for mine shafts where conveyances are to be employed. At an Australian gold mine, two holes were drilled successfully, one in an access shaft with an accuracy of greater than 1 in 160 over a length of 1050 ft (320 m), and the other in a ventilation shaft with an accuracy somewhat less than 1 in 130 over a length of 870 ft (264 m).

Shaft boring also was employed at a uranium mine in New Mexico. Here three large holes, one 10 ft (3 m) and the other two 6 ft (1.8 m) in diameter, were drilled to depths of 2243, 2188, and 2188 ft (684, 667, and 667 m), respectively. Subsequently, the plan called for underground connection to be made between these holes, and then the two smaller holes were to be reamed to a final diameter of 18 ft (5.5 m). The drilling rates in the 10-ft (3-m) hole were about 1 fph (0.3 m/h) and in the other two holes were 1.7 to 1.8 fph (0.52 to 0.55 m/h). The reported drilling accuracies were 1.03 ft in 2243 ft (0.314 m in 684 m); 1.33 ft in 2188 ft (0.405 m in 667 m); and 0.84 ft in 2188 ft (0.256 m in 667 m).

Roadheaders: A roadheader is a boom-type tunneling machine that uses drag bit cutting tools (Fig. 9.1.62). These machines are restricted to excavating relatively weak rock types, and their main use is for roadway drivage in coal mines. During the past three decades efforts have been devoted to increasing the cutting capability by increasing the installed power of these machines. This has resulted in increased machine size and mass (Fig. 9.1.63).

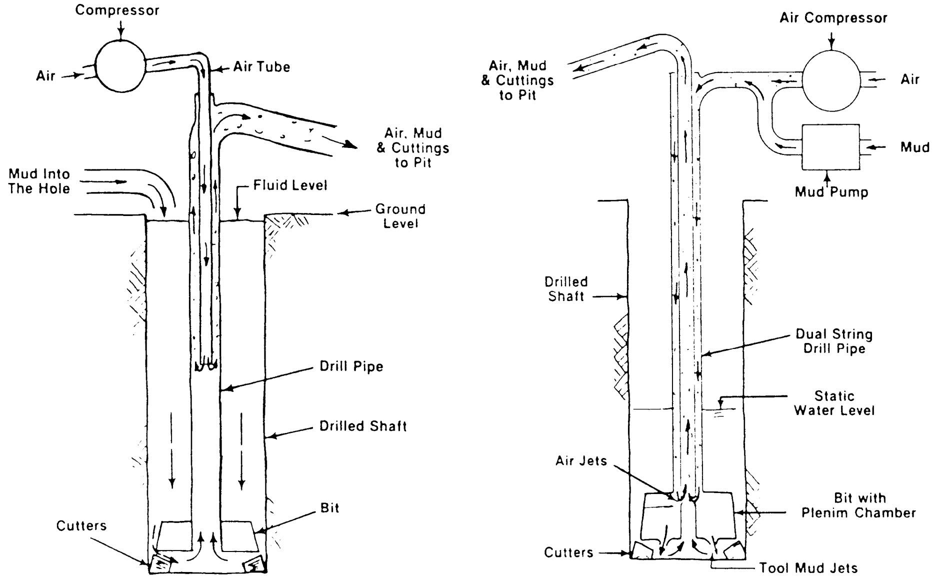


Fig. 9.1.60. Reverse circulation flushing (left) and dual string circulation (right) (from Lackey, 1983).

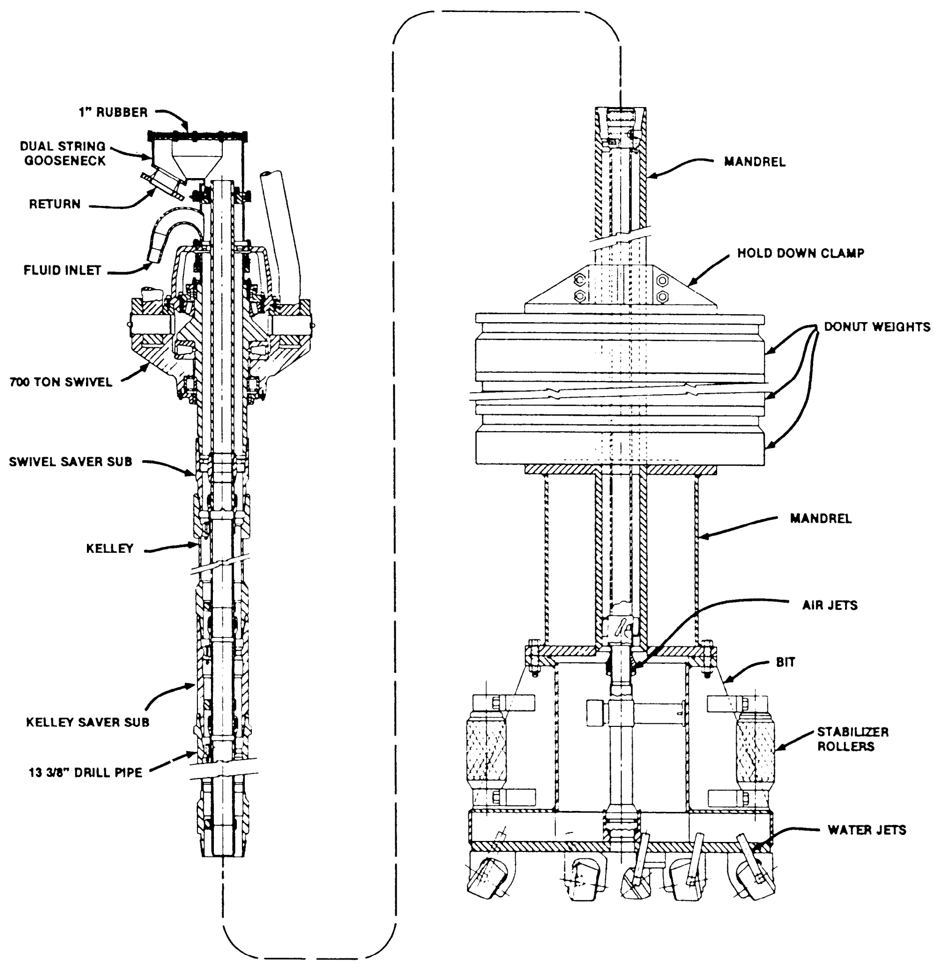


Fig. 9.1.61. Kelly and swivel together with the bottomhole assembly (from Lackey, 1983).

Roadheaders are track-mounted machines. Excavation is carried out by first sumping the cutting head into the face and then traversing the boom to cut the desired roadway profile. The broken rock deposited on the floor is loaded by a gathering arm arrangement onto an armored conveyor that runs through the center of the machine. This material then can be removed by shuttle cars.

In recent years the cutting capability of these machines has been enhanced by the addition of a water-jet-assisted cutting system. One of the direct benefits for this system, reported by Morris and Harrison (1985), is illustrated in Fig. 9.1.63. This shows that a 35-ton (32-t) machine, that normally would be restricted to cut in rocks with compressive strengths of the order of 12,000 to 14,000 psi (83 to 97 MPa), was capable of cutting in rock with compressive strengths in excess of 20,000 psi (140 MPa) when the machine was assisted with high-pressure water

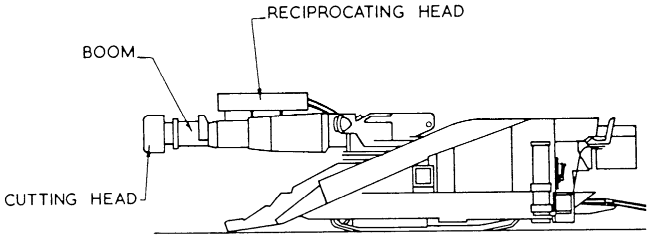


Fig. 9.1.62 The main components of a roadheader (from Morris and Harrison, 1985).

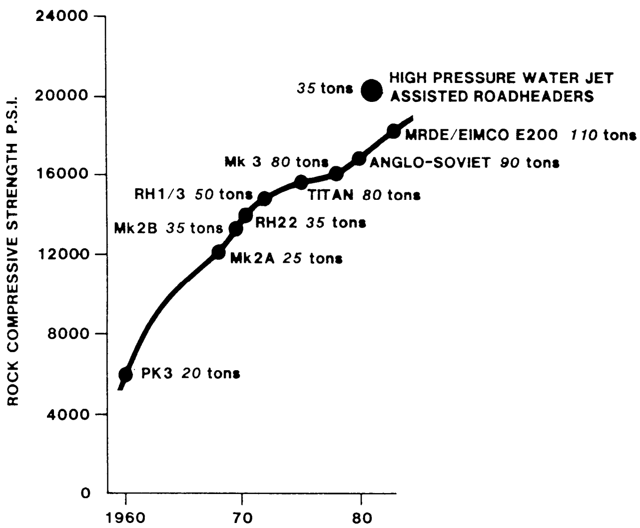


Fig. 9.1.63. Improvements in roadheader performance and the trend for increasing weight on these machines over time (from Morris and Harrison, 1985).

jets. At least two manufacturers now supply commercial water-jet-assisted roadheading machines.

9.1.4.2 Coal (and Other Weak Rock) Mining

Shearers: These machines, illustrated in Fig. (9.1.64), are used on longwall faces. Modern shearers are predominantly double-ended, ranging drum machines; that is, there is a cutting drum at both ends of the machine and these drums are mounted on hydraulically powered arms that are used to adjust the position of the drums to varying seam heights. The machine rides on skids on an armored steel face conveyor that extends the length of the face. The cutting tools are picks (drag bits). The drums are driven by an electric motor (or motors) through a gear train. The rotation of the drums serves both to cut and to load the rock onto the face conveyor. As with roadheaders the trend with these machines has been to increase the machine power and size. Today machines with an installed power of 1000 to 1340 hp (750 kW to 1.0 MW) are not uncommon.

The biggest operational problems with shearers are maintaining the cutting drums within the seam and the high dust production on the face. When the cutting drums are allowed to cut into adjacent strata, the rate of pick wear often rises dramatically, because these other rocks are generally stronger and more abrasive than the coal and evaporites normally cut by shearers. This results in increased downtime for pick changes, and often this is a substantial fraction of the overall machine downtime. Also, in coal mines particularly, cutting out of the coal seam can result in frictional sparking which can cause face ignitions of methane. Considerable work has been conducted to develop automatic steering for shearers to overcome this problem. The only commercial system for automatic steering that has been developed to date uses a signal obtained from the natural gamma radiation difference that exists between coal and many, but not all, rock types that lie adjacent to the coal seam. This signal is used to control the movement of the ranging arms and thereby to keep the cutting drums contained within the seam.

Numerous systems have been devised to control the dust levels on a longwall face. All of these remediate the dust problem, but to date, no universally accepted solution to this problem exists.

Continuous Miners: These machines (Fig. 9.1.65) also cut the rock with picks. These tools are mounted on a cutter drum some 10 to 13 ft (3 to 4 m) long. The drum, in turn, is mounted on a hydraulically powered arm that controls the vertical movement of the drum. The machine is track-mounted, and the cutting procedure is to sump by driving the machine into the face with the rotating drum raised to cut the top portion of the face. When the machine is fully sumped, generally to about one-half the diameter of the drum, the arm shears the drum down to the floor. Broken rock (coal or some other weak rock type) is dropped onto the floor during this cutting action. A gathering arm (Fig. 9.1.65), or some similar loading system, loads this broken material onto an armored conveyor that runs through the center of the machine. This product is transported from this conveyor with shuttle cars.

The problems with this machine relate not so much to the machine itself but rather to the mining system in which it works. Many of these machines are capable of mining at rates of the order of 12 tpm or 720 tph (655 t/h). However, in practice, the average production from continuous miners in US coal mines is of the order of 400 to 500 tons/shift (360 to 455 t/shift). This disparity between machine capability and actual productivity arises mainly because of the need to tram the machine frequently out from the heading in order to install roofbolts and because of the discontinuous transport system (i.e., shuttle cars). Work continues to find solutions to these problems.

9.1.5 LIMITATIONS ON THE USE OF MECHANICAL CUTTING TOOLS

9.1.5.1 Specific Energy and Specific Power

The theoretical minimum quantity of energy that must be supplied in a fragmentation process is the surface energy required for the new surface area of the fragments produced. In practice, with all materials including rocks, the quantity of energy actually necessary to effect fragmentation is much greater, often by orders of magnitude, than this theoretical minimum. The actual quantity of energy depends on the type of process employed and the nature of the rock. These factors are not independent.

A useful method for comparing the efficiencies of various rock breaking methods is to measure the quantity of energy required to excavate a unit volume of rock. This parameter, which actually is the inverse of breaking efficiency since a high energy value indicates a low efficiency, is termed the *specific energy.* As indicated previously, specific energy depends on both the breaking method employed and the rock type. This empirical parameter is the most useful measure available for predicting either the power requirements for a particular machine to exca-

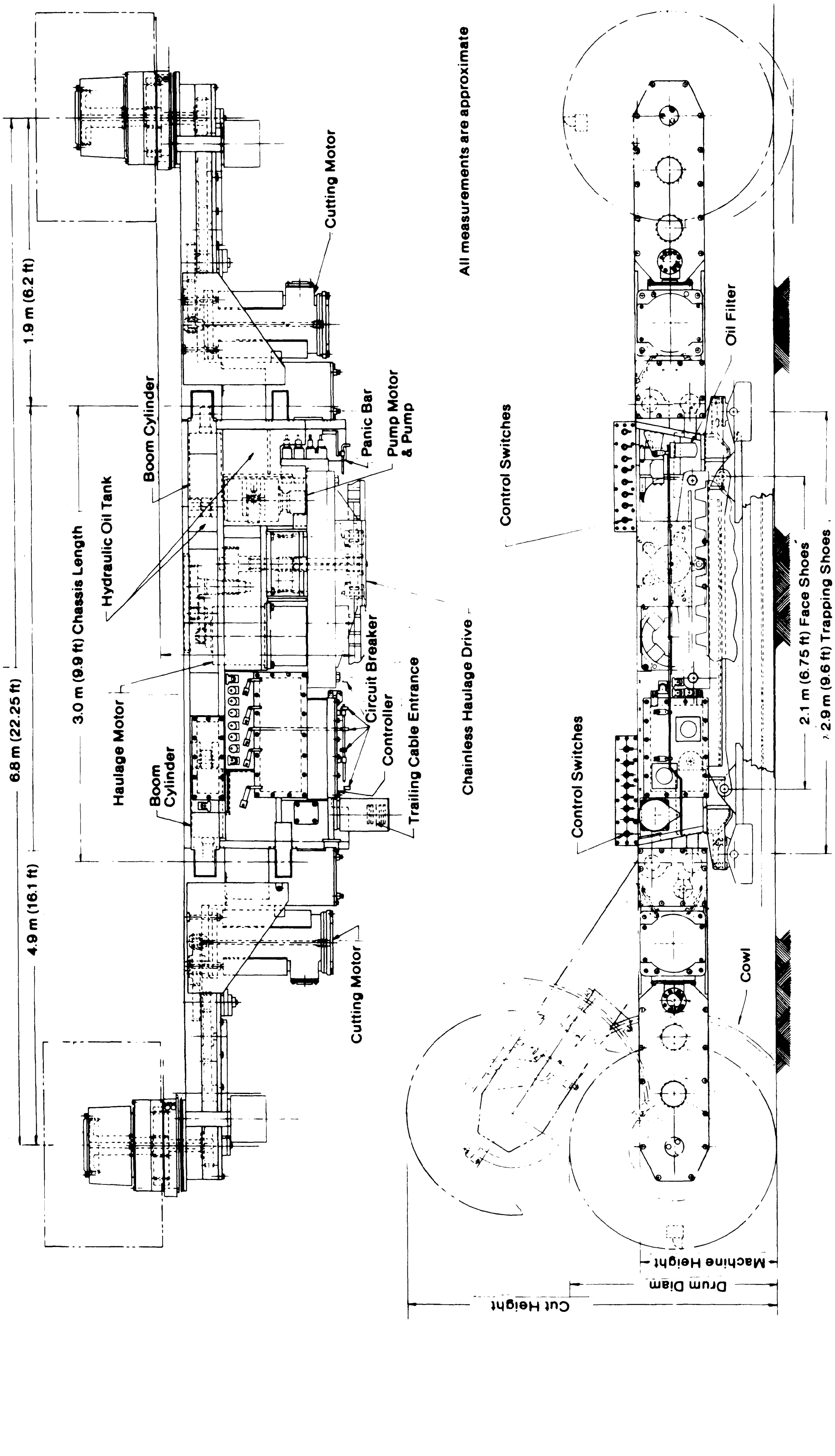


Fig. 9.1.64. Shearer (from Hislop and Erickson, 1982, *courtesy:* Joy Manufacturing Co.).

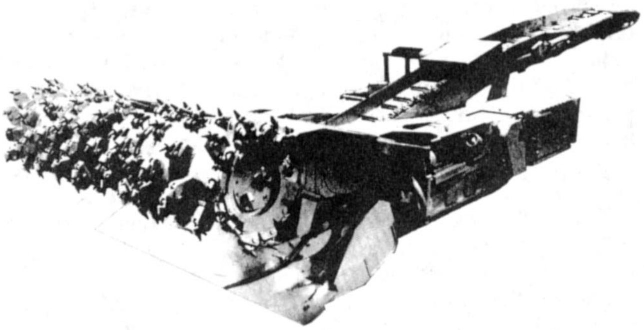


Fig. 9.1.65. Drum continuous miner (from Fitzgerald, 1982, *courtesy:* Joy Manufacturing Co.).

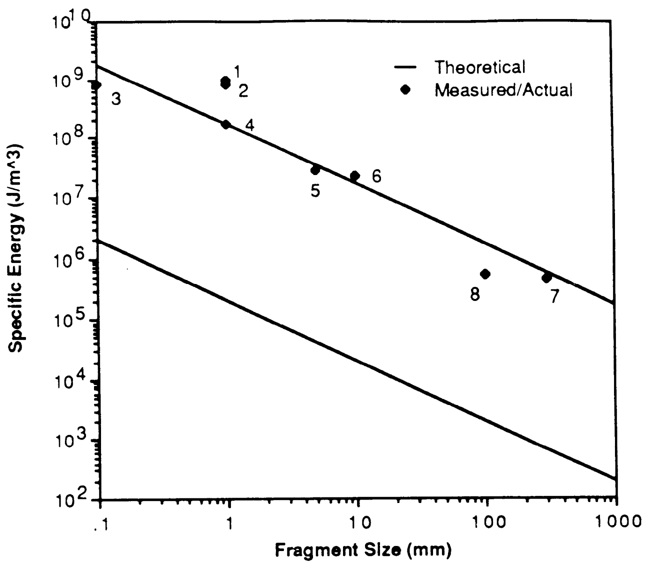


Fig. 9.1.66. Specific energy as a function of nominal particle size. The data points are for different methods of breaking strong (C_0 = 200 MPa) quartzite (after Cook and Joughin, 1970). (1) Flame-jet piercing, (2) high-pressure water jet, (3) diamond cutting or drilling, (4) percussive drilling, (5) drag bit cutting, (6) rolling cutters/boring, (7) impact-driven wedge, (8) explosive blasting.

vate at a given rate in a given rock type, or, alternatively, the rate of excavation that a particular machine might be expected to achieve in a given rock type. The following discussion, modified from Jaeger and Cook (1979), shows how this parameter can be used to make these predictions.

Cook and Joughin (1970) published data on the measured values of specific energy for a variety of breaking methods in one type of rock, a strong quartzite. They plotted these data as a function of the mean sizes of the rock fragments produced by each method. Their figure is replotted in Fig. 9.1.66. Several interesting points are illustrated in this diagram:

1. Points that lie above the line, points 1 and 2, represent methods for which the surface energy exceeds the average. Conversely, points that lie below the line, points 3 and 8, represent methods for which the surface energy is less than the average.

2. The estimated average value for surface energy of 5.78 hp-hr/ft^2 (167 kJ/m^2) is almost three orders of magnitude greater than a value for the work of fracture that might be measured in a laboratory test set up to determine this constant. This is not surprising. In a laboratory test to measure surface energy, a tensile crack is driven slowly through a rock sample. For the processes indicated in this figure, indentation loads were applied, and energy was lost through friction in the crushed material formed. More energy was lost to kinetic energy of the rock fragments. Furthermore, the values in the figure represent the energy input to the machines loading the rock. Thus the mechanical inefficiencies of the drills or cutting machines also help to account for this discrepancy between laboratory and field measurements.

3. The most efficient processes (i.e., those with the least specific energy values) are impact drive wedges and explosives, points 7 and 8. Both of these methods break the rock in tension. The next most efficient methods—percussive drilling, drag bit cutting (with blunt bits), and roller bit cutting, points 4, 5, and 6—are mechanical methods that all break the rock by indentation. The least efficient methods—flame jet piercing and high-pressure water jets, points 1 and 2—involve, respectively, thermal breaking and erosional processes.

Specific energy (hp-hr/ft^3 or MJ/m^3) is related to the rate of excavation R (ft^3/sec or m^3/s), and the power applied to the rock P (hp or MW) through:

$$E_s = \frac{3600\ P}{R} \text{ in English units, or} \tag{9.1.45}$$

$$E_s = \frac{P}{R} \text{ in SI units}$$

Considerable attention is given in the literature to specific energy. Although the quantity of energy consumed in rock breakage is considerable, most of this energy is expended in secondary breaking operations. Usually, in primary rock breaking processes, the direct energy cost is not the dominant cost factor. Thus it is pertinent to ask why much attention should be paid to this parameter. The answer is that while the energy cost often is not a limiting factor, the quantity of energy that can be (1) transferred up to the working face and (2) injected into the rock, frequently is the real limitation on the rate at which rock can be excavated.

Most primary rock breaking operations today involve the use of either mechanical tools or explosives. Even when explosives are employed, mechanical tools, in the form of drillbits, are used first to excavate the holes into which the explosive charges are placed. The popularity of mechanical tools lies in the fact that, relative to many of the alternative methods, the specific energy required for breakage with these tools is low. Consequently, for a given rate of rock excavation, the power needed at the working face also is relatively low. The disadvantage of using mechanical tools is that, in many rock types, they limit the rate of energy transfer (i.e., power) to the rock which, in turn, limits the rate of rock excavation. This limit on power arises because the rate of wear or fracture of the tool materials increases as the power transmitted through the tools to the rock is increased. In weak rocks where the specific energy requirement is low, this is not a constraint on rock excavation rates. Indeed, over the past few decades, the trend in excavation equipment has been continuously to increase the power supplied to the machine cutting head in order to enhance the cutting rate. This might be termed the "brute force" approach. However, in medium-strength and strong rocks, this option of increasing the cutting rate by increasing power is not available because this additional power results in unacceptably short tool life.

As noted above, only a small fraction of the energy applied to a mechanical tool is usefully expended in the creation of new fracture surfaces. Most of this energy is dissipated as heat. Therefore, one approach for increasing excavation rates or for extending the range of rock types in which mechanical tools might be employed would be to increase the proportion of the tool energy used for creating rock fractures. This could be termed the "smart" approach. A basic understanding of the basic principles of rock fragmentation induced by mechanical tools, given in the preceding sections, is necessary in order to develop a system utilizing the available power in a more productive manner.

An alternative approach would be to develop stronger tool materials, and work that is being carried out in this area shows some promise.

Many of the nonmechanical methods for rock breaking are associated with very high specific energies. In some cases, these energy requirements are severe enough that the energy cost for the rock excavation process would be prohibitive. Even when this is not the case, problems would likely be experienced either in transmitting sufficient power to the rock face, to maintain a reasonable rate of excavation, or in removal of the heat produced, from energy not expended in creating new rock surfaces.

One solution to these problems is to use a combination of methods to effect rock breakage. For example, a nonmechanical method can be used to preweaken the rock by cutting a series of kerfs in the face. A mechanical tool then might be employed to break out the majority of the rock. In this case, because only a small volume of the total rock excavated is removed by the nonmechanical method, Maurer (1980) suggested that a new term be introduced to define the energy requirements for breaking. This term he called the *specific kerfing energy* E_{ke}, which he defined as:

$$E_{ke} = \frac{\text{energy}}{\text{kerf area}} = \frac{\text{power}}{\text{kerf depth} \times \text{traverse speed}} \qquad (9.1.46)$$

where kerf area is the surface area on one side of the kerf.

Another technique, known as jet assisted cutting, uses water jets to assist the action of mechanical tools during the rock cutting operation. In this case the jets are not used to kerf the rock, but rather they act to continuously remove the crushed rock from beneath the tool. This substantially reduces the tool forces necessary to cut the rock because the cushioning effect of the crushed rock is eliminated allowing the tool to press more directly on the intact rock to form chips. Hybrid techniques using these approaches have been developed and, in fact, some of these methods show promise for achieving a substantial breakthrough in rock excavation technology. Some of these methods are reviewed elsewhere in this *Handbook*.

9.1.5.2 Factors that Affect Tool Life

Although some disk cutters and some roller cone bits continue to be manufactured from hardened steel, cemented tungsten carbide is by far the most widely used material for rock cutting tools today. A relatively new material, polycrystalline diamond compact, is finding increasing application for drag bits, particularly in drillbits (see 9.1.3.1). This material has the promise, as yet unfulfilled, of producing a major breakthrough in cutting technology.

The properties of tungsten carbide that make it the material of choice are high hardness combined with high (relative to other hard cutting materials) toughness. Hardness provides wear resistance. Toughness is a measure of resistance to brittle fracture. High toughness, therefore, minimizes the probability of inducing fractures that could lead to tool failure. Kenny (1971) notes that this definition of toughness is not synonymous with an absence of brittleness. Rubber and cemented tungsten carbides are examples of two materials that are both tough (i.e., resistant to fracture) and brittle when loaded in tension. In these cases, there is no distinction between strength and toughness. Polycrystalline diamond is much harder, and thus more wear resistant, but it is more susceptible to brittle fracture (i.e., it is less tough), than cemented tungsten carbide.

Wear Mechanisms: The four most significant wear processes for materials in general are identified by Rabinowicz (1965) as adhesive wear, abrasive wear, corrosive wear, and surface fatigue wear. These processes are described briefly in the following.

Adhesive Wear—When two surfaces are brought together, an attraction force is set up between the atoms on the surfaces of the two bodies. If the bodies then are separated, this attractive force attempts to pull material from one surface onto the other. Any material removed in this way is termed an adhesive wear fragment. Although this is the most common of all wear mechanisms, it is not the most important mechanism for rock cutting tools.

Abrasive Wear—This form of wear occurs either when a hard, rough surface slides against and plows grooves in a softer surface, or when abrasive particles are introduced between two surfaces and these particles abrade material from one or both surfaces. This is the most common and the most important wear mechanism for rock cutting tools. In this case, the rock debris formed by the tool constitutes the wear particles that abrade the tool.

Fortuitously, these two important wear processes, adhesive and abrasive wear, obey the same Holm-Archard relationship (Holm, 1946; Archard, 1953):

$$V = \frac{k_a P\, l}{H} \qquad (9.1.47)$$

where k_a is a wear coefficient, a constant dependent on whether the wear process is adhesive or abrasive; P is the applied load; l is the distance of sliding; and H is the hardness.

This expression shows the importance of hardness of the tool material to the wear process.

Corrosive Wear—This form of wear occurs when sliding takes place in a corrosive environment. This mechanism is not of great concern for rock cutting tools.

Surface Fatigue Wear—Repeated mechanical or thermal loading cycles can lead to the formation of fatigue cracks in the surface or subsurface of a material. These cracks eventually cause breakup of the surface with the formation of large fragments.

Cemented Tungsten Carbides: This tool material is manufactured by sintering powders of tungsten carbide and cobalt. The cobalt is a binder that is distributed throughout the body of the material in thin films between the tungsten carbide grains. It constitutes 5 to 15% by weight of the mix (for mining tools) and gives increased toughness to the material. The properties of the tool material are affected both by the cobalt content and by the size of the tungsten carbide and cobalt grains.

Cemented tungsten carbide is manufactured in a wide variety of sizes and shapes, depending on the rock cutting application. The larger the size of the carbide insert the lower its strength. The carbide is mounted into a (usually hardened) steel bit body. This mounting is achieved either with a braze joint or by press-fitting the carbide insert into a machined hole in the bit body. The hardness of the cemented tungsten carbides produced for

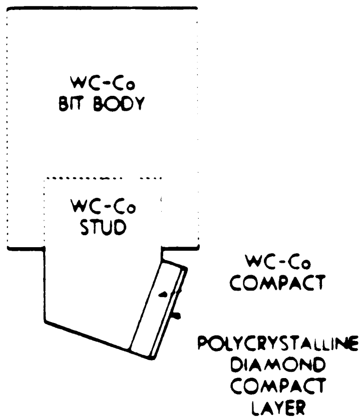


Fig. 9.1.67. Mounting of PDC onto a cemented tungsten carbide stud.

mining tools generally is in the range 1.45×10^6 to 2.175×10^6 psi or 10 to 15 GPa (Perrott, 1980).

Polycrystalline Diamond Compacts (PDC): These compacts are manufactured by sintering synthetic diamond crystals with a (mostly cobalt) binder. The binder acts to promote diamond bonding and, in contrast to the cemented carbides, it only plays a role in the mechanical behavior of the composite at high temperatures (Hibbs and Lee, 1978). After the sintering process, the binder forms roughly spherical inclusions rather than films between the grains, which means that hard bridges exist between the diamond grains. These boundaries are, however, very strong, and when fracture occurs it is as likely to propagate through the grain as along a grain boundary (loc. cit.).

The room-temperature hardness of polycrystalline diamond compacts is approximately 9.86×10^6 psi or 68 GPa (Lee and Hibbs, 1979). The importance of this additional hardness now is apparent. Provided that the dominant wear mechanisms for the cutting tools are adhesive, abrasive, or both, it can be seen from Eq. 9.1.47 that for a given loading situation, the wear rate (that is the volume of tool material removed per unit length of cut) of a PDC bit will be less than that of a cemented tungsten carbide bit by the ratio of the hardness of these two materials, or a factor of about five.

PDC, however, is more susceptible to brittle failure than cemented tungsten carbide (Glowka and Stone, 1984). Consequently, to give the required strength to the tool material, a PDC cutting insert is constructed by sintering a thin layer of polycrystalline diamonds onto one end of a short cylinder of cemented tungsten carbide. This carbide provides support to the diamond wafer. This cylinder, in turn, is attached, usually with a braze joint onto a (generally) cemented tungsten carbide stud (Fig. 9.1.67). These studs are then mounted, often by press-fitting, into the bit body (Fig. 9.1.34).

Tool Fracture and Wear: *Fracture*—Because cemented carbides and PDCs are brittle materials, they are approximately an order of magnitude stronger when loaded in compression than when loaded in tension. Furthermore, their strength, as for any brittle material, is enhanced dramatically when (even relatively small) compressive loads are applied to the body in directions orthogonal to the line of action of the main force. This is termed triaxial loading.

Consequently, bit designers aim to ensure that the bit geometry and the mode of loading will not cause tensile stresses to be applied to the bit. Also, where possible, they will mount cemented tungsten carbide inserts in a bit body in a manner such that triaxial compressive stresses are applied to the inserts. For example, when tungsten carbide button inserts are press-fitted into the bit body the hardened steel body applies a radial compressive stress to the inserts.

Despite these precautions, in many applications, one of the main reasons for replacing bits (both cemented tungsten carbide and PDC) in service is because the inserts fail in a brittle manner.

One of the main factors responsible for this type of failure is impact loading of the bits. An impact blow induces compressive stress waves in the bit insert. These are reflected back from free surfaces as tensile stress waves, which, because the material is very susceptible to tensile failure, can induce cracking in the insert. The magnitude of this tensile stress may be sufficient to initiate and propagate a crack to induce failure with a single blow. More typically, impact fatigue will take place. In this case the crack is extended incrementally by each loading cycle until eventually failure takes place. To prevent or to minimize this type of failure it is important to minimize impact loading to the cutting tools. This is most easily achieved through the use of a stiff drive to the cutting head.

Wear—The two processes considered to be of most importance for rock cutting tools are abrasive wear and surface fatigue wear. Abrasive wear can take place only if the hardness of the abrasive particles in the system is greater than, or at least of the same order as, the hardness of the cutting tool. In other words, if these particles are much softer than the cutting tool, abrasive wear will not take place. In rock cutting, the rock debris constitute the abrasive particles. One of the most common hard constituents of rock is quartz. The room temperature hardness of this mineral is only 1.595×10^6 psi (11 ± 0.3 GPa) (Perrott, 1980), generally less than that of cemented tungsten carbide and considerably less than that of PDCs.

It is recognized that the mechanisms of wear are different depending on the relative hardness of the abrasive particles and the material experiencing wear. When the hardnesses of these two materials are comparable, the particles are termed a "soft" abrasive (Richardson, 1968). On the other hand, when the hardness of the particles far exceeds that of the material of interest, the particles are termed a "hard" abrasive. According to Larsen-Basse (1980), quartz acts as a hard abrasive when the hardness of cemented tungsten carbide is less than about 1.566×10^6 psi (10.8 GPa); otherwise it acts as a soft abrasive. The rate of wear is found to be proportional to the applied load for both hard and soft abrasives. In other words Eq. 9.1.47 is correct, in terms of the applied load, for both types of abrasive. This is not the case for hardness. The wear rate is much greater (often an order of magnitude greater) when hard abrasives are used (Larsen-Basse, 1981). Thus Eq. 9.1.47 is correct, in terms of hardness, only for hard abrasives. An abrasive is considered hard, in general, when the ratio of the abrasive hardness to the materials' hardness is 1.2 or greater.

Hard abrasives with coarse (1.57-in. or 40-mm) particles act as micromechanical cutting tools, cratering and cutting grooves in the surface of a cemented carbide. These craters and grooves are much larger than the carbide grains, and the material from these damaged regions is easily removed (loc. cit.).

Soft abrasives cannot indent the surface; instead, damage to the tungsten carbide grains occurs through surface fatigue. Also these abrasives cause erosion and extrusion of the cobalt binder from the region around the grains close to the surface. Work by Brainard and Buckley (1979) showed that the removal of cobalt, even to shallow depths below the surface, greatly increases the wear rate of the carbide by allowing the tungsten carbide grains to be more easily plucked from the surface.

Although thermal fatigue has not been thoroughly investigated for rock cutting tools, it has been shown to be an important failure process in metal cutting (Bhatia et al., 1978), and probably it also is significant in rock cutting (Perrott, 1980).

From this discussion, it is apparent that rock particles would be expected to act only as a soft abrasive on cemented tungsten carbides. In consequence, it would be expected that abrasive wear of these rock cutting tools would not pose substantial difficulties, although thermal fatigue still might produce significant tool wear. In fact, however, for reasons discussed below, abrasive wear of cemented tungsten carbide rock cutting tools is a major problem.

Rock debris, with a relative hardness some six times less than that of diamond, would not be expected to cause abrasive wear of PDCs, and this indeed appears to be the case. In this material, each diamond grain acts like a single-crystal cutter. Failure of the material while cutting rock occurs through one of following processes (Hibbs et al., 1978):

1. The edges of individual crystals are crushed, and the broken material is removed in small pieces; alternatively, a large portion of an individual crystal is broken in a simple brittle fracture process.

2. By gross fracturing, either as a result of impact loading or crack formation from bending stresses caused by the rock pressing on the front face of the bit; in both cases, the fracture is transgranular indicating the high strength of the grain boundaries.

3. By the growth of pits that are formed initially by the removal of surface metallic inclusions during the tool shaping process.

4. From internal flaws, such as poorly bonded grains.

The factor that has been overlooked in the discussion thus far is the influence of temperature on the hardness of the tool materials. It is known that the temperature of a tool rises, usually to several hundred degrees Celsius or more, during rock cutting operations (see, for example, Hood, 1978; Whitbread, 1960). The hardness of both cemented tungsten carbide and diamond decreases dramatically with increasing temperature (Figs. 9.1.68 and 9.1.69). From Fig. 9.1.68, it is apparent that discrepancies exist in the literature regarding the hardness values of cemented tungsten carbide with temperature. Osburn (1969) attempted to explain these discrepancies in terms of different methods of sample preparation by the various investigators.

The hardness of quartz as a function of temperature also is given in Fig. 9.1.68. It can be seen that the hardness of this mineral decreases rapidly as the temperature increases from room temperature to 1063°F (573°C), at which point quartz experiences a phase change. The hardness then increases with temperature over approximately a 200°F (100°C) range before decreasing rapidly again.

The hardness of a diamond compact as measured using a Vickers diamond indenter (Lee and Hibbs, 1979) is given by the lower curve in Fig. 9.1.69. The upper curve in this figure, reproduced by Lee and Hibbs (1979), is taken from Loladze and Bockuchava (1972). The type of diamond tested for this upper curve is unknown. Lee and Hibbs (1979) observed that polycrystalline diamond compacts suffer gross thermal damage when they are heated above 1382°F (750°C). However, the main point of note is that at 1292°F (700°C), they measured the hardness of a PDC as 6.754×10^6 psi or 4750 kg/mm^2, or 47 GPa (since 1000 kg/mm^2 = 9.807 GPa).

In terms of damage caused to tools by thermal heating during rock cutting operations, drag bits present a much greater problem than roller cutters. This is because an individual element on a roller cutter heated by the rock breaking process, has the opportunity to cool as the cutter rotates to bring other cutting elements into contact with the rock. Typically, any one element will be heated by frictional contact with the rock for less than 10% of the time that the cutting machine is operating. A drag bit, on the other hand, frequently is in continuous contact with the rock while the machine is operating. Even when these bits are mounted on a rotating drum, such as a continuous miner, generally they are in contact with the rock for about half of the time that the machine is cutting. Hence they have far less opportunity to cool. For this reason, research efforts to model the temperatures generated by tools during the cutting operation have concentrated on drag bits. Analytical models for these drag bit temperatures have been proposed by Cook (1982) and Glowka and Stone (1983).

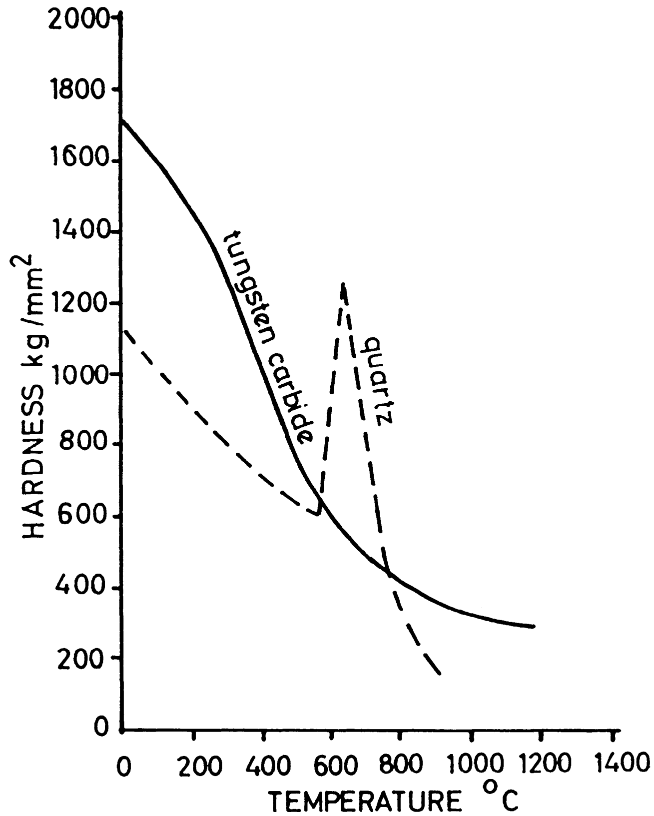


Fig. 9.1.68. Reported effects of temperature on the hardness of cemented tungsten carbide and quartz (after Osburn, 1969).

The latter authors used a laboratory study cutting in sandstone (Lee and Hibbs, 1979) in which it was shown that PDC wear rates accelerate when bit temperatures exceed 662°F (350°C). Glowka and Stone used their model to predict the maximum bit cutting force, μF_{cr}, that could be exerted on the bit without causing it to exceed this threshold temperature. In this expression, μ is taken as constant and is termed the coefficient of cutting friction, and F_{cr} is the critical weight-on-bit, (equivalent to the bit normal force). F_{cr} controls the depth of cut taken by the bit. Glowka and Stone correlated their calculated bit temperatures with measured bit wear rates and found the consistent pattern of constant bit wear rates (about 10^{-10}cm^3 of wear/cm of cut length) when bit temperatures were maintained below 350°C. Above this wearflat temperature, the bit wear rate accelerated rapidly, in some cases by as much as two orders of magnitude. Their results also showed that liquid cooling of the bits was an important factor in reducing bit temperatures. Glowka and Stone interpreted these results as support for the case that bit wear is correlated directly with reduced bit hardness at elevated

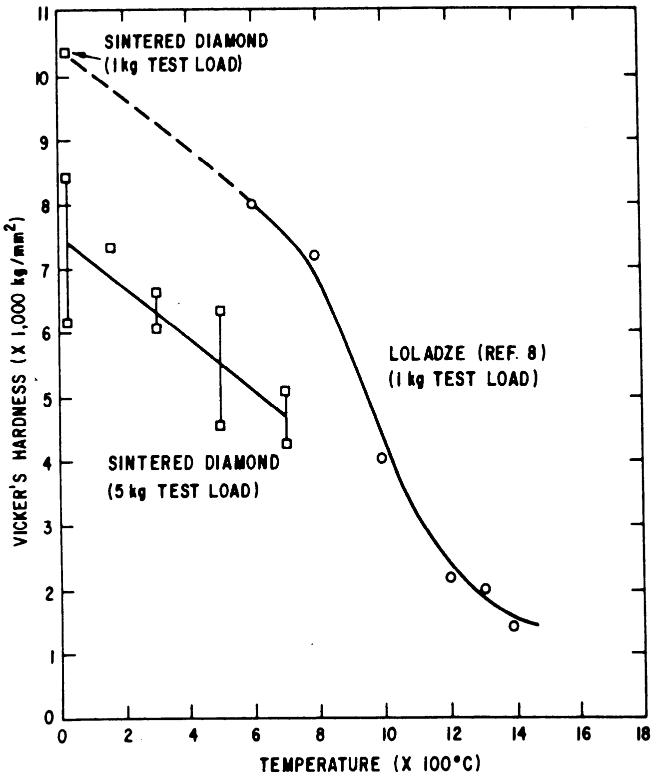


Fig. 9.1.69. The effect of temperature on the hardness of diamond (after Lee and Hibbs, 1979).

temperatures, and they recommended that the operating conditions for PDC bits should be such that bit wearflat temperatures are always maintained below 350°C. They used their model to determine the maximum forces that should be transmitted to a bit to ensure that this criterion is satisfied.

Perrott (1980) suggested that a lower bound for cutting tool wear could, in principle, be determined by calculating or measuring the temperature at the rock/tool interface during a cutting operation and by making the assumption that the abrasive rock particles and the tool are in thermal equilibrium. The hardnesses of the particles and the tool at this temperature then could be determined, and this would establish whether the particles were acting as hard or as soft abrasives. Similarly, an upper bound for wear could be determined by assuming that the abrasive particles remain at room temperature while the tool is heated by the cutting process. The author notes that the latter assumption is likely to be the more realistic one because the tool is continuously sliding over new abrasive particles.

Making this latter assumption Perrott proceeded to back calculate temperatures at the rock/tool interface during a cutting operation, using wear data published by other workers. The assumption was made that the tool wear rate accelerates when the ratio of the (room temperature) hardness of the abrasive to the (hot) hardness of the tool is 1.25 or greater. In other words, wherever these other investigators reported a transition in wear behavior from slight or moderate to severe, Perrott assumed that this change was caused by the abrasive particles starting to act as a hard abrasive. Using this approach, he calculated temperatures at the interface of 662 to 752°F (350 to 400°C) for results published by Latin (1961) and 392 to 482°F (200 to 250°C) for results published by Larsen-Basse and Koyangi (1978). Perrott noted that temperature measurements of rock bits during cutting operations confirm that temperatures of this magnitude commonly are generated at the rock/tool interface. In fact, work by Hood (1978) demonstrated that blunt drag bits cutting in strong abrasive rock can generate temperatures of 752°F (400°C) several millimeters back from this interface in the braze joint between the cemented tungsten carbide insert and the steel bit body. Calculations by Whillier (1976) demonstrated that with these temperatures in the braze joint the interface temperature was likely in excess of 1832°F (1000°C). These calculations were verified by Hood (1978) who found streaks of the WC-Co composite in the rock debris at the bottom of the groove machined by the bit. Since the cobalt binder melts at a temperature of 2723°F or 1495°C (Cook, 1981), this finding of thermal breakdown of the bit insert showed that the temperature generated at the bit/rock interface must have exceeded this value during the cutting operation.

In summary, temperature is the main factor that controls the wear rates of both cemented tungsten carbide and PDC bits. Above some critical temperature (this temperature value is not yet well established but probably is of the order of 392 to 662°F (200 to 350°C)), a WC-Co composite begins to wear rapidly because it sees quartz particles as hard abrasives. Although the wear mechanism of PDC bits is different, mainly microchipping rather than abrasive wear, this process also is temperature sensitive. The critical temperature below which it appears that these bits should be operated is 662°F (350°C). At temperatures in excess of 1382°F (750°C), a PDC tool fails catastrophically.

The temperatures generated at the bit wearflat is a function of the power applied to the tool. Thus one way to control these temperatures, and thus to control bit life, is to restrict this applied power. Unfortunately, this is not a satisfactory solution; the power to the tool can be reduced either by decreasing the cutting velocity of the tool or by reducing the depth of cut taken by the tool. Obviously, either of these steps causes a reduction in the rate of rock excavation, hardly a desirable option. Another approach is suggested by Glowka and Stone's finding that liquid cooling of the bit results in substantial reductions in bit temperatures.

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Chapter 9.2
ROCK BREAKAGE: EXPLOSIVES

CHARLES H. DOWDING AND C.T. AIMONE

9.2.1 BLAST DESIGN

C.T. AIMONE

The application of explosives to the fragmentation of rock for mining or extractive purposes is often referred to as a science as well as an art. In early years, explosives development became a science as chemists strove to perfect stable yet powerful explosive mixtures for a variety of purposes. In recent years, the needs to minimize the costs of blasting and to control the environmental effects of noise and ground motions have brought about carefully engineered blast designs, aided by research. The research includes computer modeling, explosives formulation, high-speed photography, and development of new initiators. In addition, improved quality control in the manufacturing of explosive products and devices has evolved. Scientists and engineers, however, along with practicing blasters, continue to agonize over the ideal approach to optimizing the fracturing process. Systematic research in rock blasting has provided a good understanding of the many factors that influence fragmentation. Many of these factors involve site geology, explosives selection, timing for individual detonations, drilling pattern, and explosives loading characteristics. Parameter studies in blasting research are limited to two to three factors. The cumulative effects of these factors can only be obtained through costly trial-and-error blasts. As a result, obtaining precise control of these factors is a constant challenge. The art in blasting evolves from the skillful way in which the blaster uses an understanding of the influencing factors to maximize blasting efficiency.

Chapter 9.2 provides a basic understanding of the principles involved in the selection and application of commercial explosives to rock blasting in the mining industries and of the monitoring and control of blast effects. In 9.2.1, entitled blast design, the calculations and examples presented are to be used as guidelines and are representative of the many different methods of calculation and design. This chapter begins with a discussion of the chemical and physical properties of explosive mixtures. The following description of explosive products and initiating systems includes a discussion on comparative properties of explosives. An overview of important explosive thermodynamic properties is given with example calculations. Rock blasting practices are presented with fracture theory and the effects of geology. Aspects of safety in the transportation, storage, and handling of explosives are discussed. Blast-design procedures and examples applied to surface and underground mining are presented. Estimating drilling and blasting costs and an overview of recent advances in rock blasting conclude this portion of the chapter.

Throughout this segment, units of cubic centimeters (cm^3) are used for numerical examples that involve explosive quantities on a bulk or volumetric basis. Notwithstanding usage norms involving SI units, cm^3 is used extensively in the explosives industry to describe explosive properties. Density values appear in this text without units; however, for calculation purposes, the units are assumed to be g/cm^3.

Segment 9.2.2, monitoring and control of blast effects, summarizes vibration measurement and structural response associated with rock blasting. Experimental observations are included to demonstrate the effects of blasting on ground motion parameters. Blast design procedures to set safe criteria for the control of blast effects are exemplified. Blasting regulations and computerized monitoring instrumentation are also discussed.

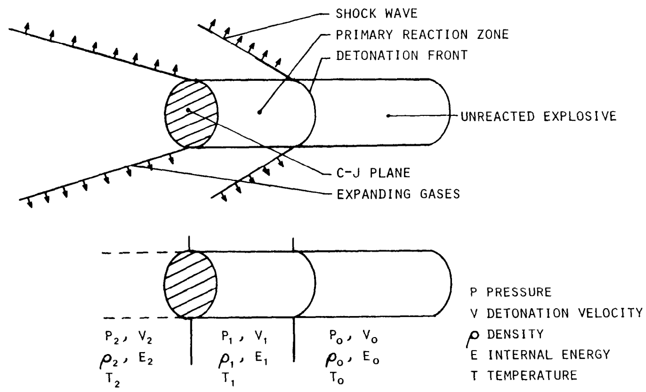

Fig. 9.2.1.1. The detonation process for cylindrical explosives.

9.2.1.1 Classification of Commercial Explosives

Detonation Theory: An *explosive,* or blasting agent, is a compound or a mixture of compounds, which, when initiated by heat, impact, friction, or shock, is capable of undergoing a rapid decomposition, releasing tremendous amounts of heat and gas. The decomposition is a self-propagating, exothermic reaction called an *explosion.* The stable end products are gases that are compressed, under elevated temperature, to very high pressures. It is the sudden rise in temperature and pressure from ambient conditions that results in a shock wave, or a *detonation*, traveling through the unreacted explosive. The velocity of detonations lies in the approximate range of 5000 to 30,000 fps (1500 to 9000 m/s), well above the speed of sound in the explosive material. *Deflagration* is the chemical burning of explosive ingredients at a rate well below the sonic velocity. It is associated with heat only and carries no shock. Deflagration occurs when less than ideal hole-loading conditions or explosive formulation are involved.

All commercial explosives are mixtures of carbon, hydrogen, oxygen, and nitrogen. The maximum energy release upon detonation occurs when the explosive mix is formulated for oxygen balance. An *oxygen-balanced mixture* is one in which there is no excess or deficiency in oxygen, such that the gaseous products formed are chiefly H_2O (water vapor), CO_2 (carbon dioxide), and N_2 (nitrogen). In actual blasting practice, small amounts of noxious gases such as NO (nitric oxide), CO (carbon monoxide), NH_4 (ammonia), CH_4 (methane), and solid carbon, are formed resulting in nonideal detonations and somewhat less than ideal pressures and energies. Commercial explosive formulation attempts to achieve an oxygen-balanced mixture.

The work done by chemical explosives in the fragmentation and displacement of rock depends on the shock energy as well as the energy of the expanding gases. Fig. 9.2.1.1 shows an

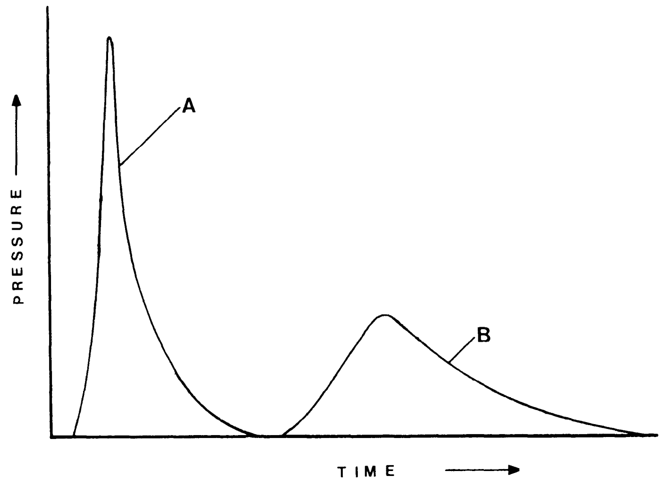

Fig. 9.2.1.2. Pressure pulse shapes for a high explosive (A) and a commercial explosive containing high gas volume (B).

idealized detonation wave traveling through a cylindrical explosive shape, producing an increase in pressure. The steady-state chemical reaction takes place behind the shock front within the reaction zone. At the end of this zone, a nonsteady-state region exists. It is created by a flow of expanding gases in a direction opposite to that of the traveling wave front. The boundary between the steady and nonsteady states is referred to as the *Chapman-Jouget (C-J) plane.* At this plane, the chemical reaction is complete, assuming an ideal detonation. It is at the C-J plane that all the thermodynamic properties of pressure P, velocity V, temperature T, internal energy E or heat of formation Q, and density r, are calculated. Fig. 9.2.1.2 shows generalized pressure waves produced from two detonations. The maximum pressure and duration of a wave pulse is directly proportional to the shock energy and gas pressure of the explosive, respectively. High explosives such as military explosives or highly sensitive commercial explosives are characterized by an intense shattering effect upon detonation (known as *brisance*). They liberate gaseous products very quickly. The distance between the shock front and the C-J plane is very short and results in a pressure pulse of high amplitude and short duration. The pressure pulse for less-sensitive commercial explosives shows a decreased pressure amplitude and a longer pulse length. In this case, the reaction is slower and the gas volume is greater.

Comparative Properties of Explosives: Explosives and blasting agents are characterized by various properties that indicate how they will perform under field conditions. These properties include fume class, density, water resistance, temperature effects, detonation velocity, detonation pressure, borehole pressure, sensitivity, and strength.

Fume Class—Fumes are noxious gases that are produced from the detonation of explosives. The production of these gases is most critical in underground and other confined workings. Many factors affect the volume of poisonous gas produced including oxygen balance and adverse loading of explosives. The *fume class* is a measure of the toxic gases in cubic feet per 0.44 lb (200 g) of unreacted explosive. Table 9.2.1.1 lists the current fume classifications as adopted by the Institute of Makers of Explosives (IME). The US Bureau of Mines (USBM) limits the volume of poisonous gases produced by permissible explosives (those used in underground coal and other gaseous mines) to 2.5 lb (1.14 kg).

Table 9.2.1.1. Standards for Fume Class

Class	Volume poisonous gas per 0.44 lb explosive, ft^3
1	0.16
2	0.16–0.33
3	0.33–0.67

Source: Dick et al., 1983.

Density—The *density* of an explosive is defined as the weight per unit volume or the specific gravity. Commercial explosives range in density from 0.5 to 1.7. Explosives with a density less than 1 will float in water. Therefore, in water-filled holes, an explosive with a density greater than 1 is required. For certain granular explosives such as dynamite, density correlates to the energy released in a given borehole volume. However, for water-based explosives, this is not the case, and often the reverse is true. Density is most useful in determining the *loading density LD* or the weight of explosives one can load per unit length of borehole (in pound per foot or kilogram per meter), and is calculated in English units as

$$LD = 0.3405\ \rho\ D^2 \tag{9.2.1.1}$$

where r is density, and D is explosive column diameter in in. Knowledge of loading density is required for blast-design calculations.

Water Resistance—The ability of an explosive to withstand exposure to water for long periods of time without loss of strength or ability to detonate defines the *water resistance.* A numerical rating is used based on the results of tests performed on the explosive. However, explosive manufacturers individually rate products based on a relative basis as good, fair, or poor rating. The presence of moisture in amounts greater than 5% dissolves chemical components in dry blasting agents and alters the composition of gases produced, contributing to the formation of noxious fumes and lower energy output. Gelled granular products have good water resistance, and certain water-based mixtures have an excellent rating.

Temperature Effects—Extreme low temperatures affect the stability as well as the performance of explosives. The sensitivity and detonation velocity are hampered for certain water-based explosives at low temperatures while dynamites can become dangerously unstable below freezing temperatures. Explosives manufacturers recommend the appropriate range of temperature for storage and use.

Detonation Velocity—The *detonation velocity* is the speed at which the detonation front moves through a column of explosives. For high explosives such as dynamite, the strength of an explosive increases with detonation rate. For dry blasting agents and water-based explosives, field loading conditions greatly affect detonation velocity. Such conditions include borehole diameter, density, confinement within the borehole, the presence of water, and other factors. The speed of detonation is important when blasting in hard, competent rock where a brisance effect is desired for good fragmentation.

For most explosives, there is a minimum diameter D_{min} below which detonation velocity increases nonlinearly with increasing borehole diameter (Fig. 9.2.1.3). Above D_{min}, the explosive has reached its steady-state velocity. At this point, all thermodynamic properties are at a maximum as the reaction front approaches a plane shock front. At diameters less than D_{min}, com-

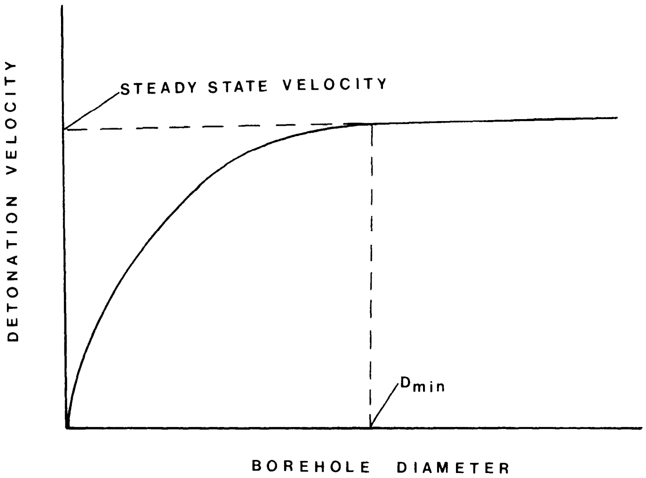

Fig. 9.2.1.3. Generalized relationship between detonation velocity and borehole diameter.

plete reactions do not take place, and less than ideal energy and pressure evolve from the slower detonation rates. This represents a loss in terms of dollars spent on explosive energy.

Detonation Pressure—The *detonation pressure* is the maximum theoretical pressure achieved within the reaction zone and measured at the C-J plane in a column of explosives. The actual pressure achieved is somewhat less than this maximum due to nonideal loading conditions always present in practice and due to certain explosive formulation. Most commercial explosives achieve pressures in the range of 0.29 to 3.48 × 10^6 psi (2 to 24 GPa). Although detonation pressure is related to the temperature of the reaction, a number of simplifying formulas are available for estimating detonation pressure for granular explosives based on detonation velocity and density, such as (in English units):

$$P = 3.37\,(10)^{-3}\,\rho\,V^2 \qquad (9.2.1.2)$$

where P is detonation pressure in psi, ρ is density, and V is detonation velocity in fps (Anon., 1987a).

Borehole Pressure—*Borehole pressure* is the maximum pressure exerted within the borehole upon completion of the explosive reaction measured behind the C-J plane. Such measurements cannot be made directly and are done during underwater tests performed for energy and strength determinations. With the use of hydrodynamic computer codes, theoretical calculations of borehole pressures are made. There is little agreement in the literature regarding specific estimates of actual borehole pressures. In general, pressures after detonation within the borehole are estimated to be less than 30% of the theoretical detonation pressure.

Sensitivity— The definition of explosive *sensitivity* is two-fold. It includes sensitivity against accidental detonations in addition to the ease by which explosives can be intentionally detonated. From the standpoint of safety and accidental detonations, the sensitivity of an explosive to shock, impact, friction, and heat determine its storage and handling characteristics. Standardized tests for high explosives have been adapted for commercial explosives that include the friction (pendulum), impact (fallhammer), and projectile tests, among others. These tests are explained by Meyer (1981).

The term properly used to define the propagating ability of an explosive is *sensitiveness.* In this respect, tests such as the No. 8 strength blasting cap test, air-gap test, and the minimum critical diameter test are used. The cap sensitivity test measures the minimum energy required for initiation and is used to classify explosives (e.g., cap sensitive vs. noncap sensitive products) or the ability to initiate an explosive directly with a standard cap. The No. 8 cap is an industry standard cap of specific dimensions and charge characteristics. The air-gap test measures the distance between the ends of adjacent cartridge explosives for which reliable initiation can be propagated from one cartridge to another. The *critical diameter* of an explosive is the smallest diameter at which an explosive will maintain a steady-state detonation. Below this critical diameter, explosives may deflagrate or "dead press." Dead pressing occurs when an explosive is densified to a point that no free oxygen is available to ensure the start or progression of detonation. Establishing the critical diameter of all explosives is an important explosive selection criteria.

Strength—The strength of an explosive is a measure of its ability to break rock. The terms "weight strength" and "bulk strength" were useful many years ago when explosives were primarily comprised of nitroglycerin cartridges, packaged in 50-lb (23-kg) boxes. In recent years, with the development of bulk blasting agents and less sensitive ingredients, new testing methods have been established to determine relative energies for all commercial products regardless of ingredients or packaging. The performance potential of an explosive is a function of the detonation velocity and density, as well as the volume of liberated gases and the heat of the reaction. A number of methods are used to establish this energy including the use of theoretical computer models and tests such as crater, ballistic mortar, and underwater tests. Of these methods, underwater tests give the best correlation to rock-breakage performance.

Underwater tests were developed to measure both the shock energy and the gas (bubble) energy released during the detonation of standard test samples. Cole (1948) describes the calculations involved while the test is detailed by Anon. (1987a) and Anon.(1977). These energy values have been useful in predicting the rock-breaking capabilities of explosives for comparative purposes.

Other terminology widely used by manufacturers is based on the theoretical heat of reaction determined by explosive formulation. *Absolute bulk strength* (ABS) in calories per cubic centimeter and *absolute weight strength* (AWS) in calories per gram are computed from the heat liberated during the detonation and formation of gaseous end products. Note ABS and AWS can be computed from one another if density is known, and it is the volumetric basis of reaction heat which correlates with energy. Most manufacturers of explosives will include either value with technical product literature.

A mixture of ammonium nitrate and fuel oil (ANFO) is by far the most widely used commercial blasting product. Depending on the proportions of the mix, the heat of reaction is approximately 850 cal/g. As a dry, free-running blasting agent, ANFO is capable of being loaded or packaged at varying densities. For a typical density of 0.85 and an AWS of 850 cal/g, the ABS = (850 cal/g) (0.85) = 723 cal/cm^3.

Other common strength terms are the *relative weight strength* (RWS) and *relative bulk strength* (RBS) in which the relative measure of energy available per unit weight or volume of an explosive is compared to an equal weight or volume of the standard commercial explosive ANFO. The RWS and RBS are computed as a percentage of that available from ANFO. In Ex. 9.2.1.1, explosives A and B have ABS values of 645 and 980 cal/cm^3 and densities of 0.8 and 1.25, respectively, and the RBS and RWS are computed.

Example 9.2.1.1. Determine the relative strengths of explosives A and B.

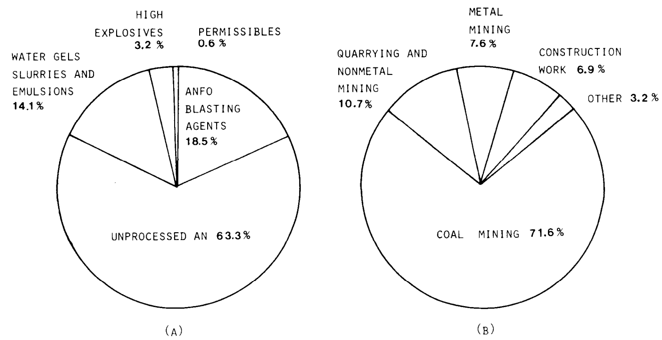

Fig. 9.2.1.4. Industrial explosives and blasting agents sold in the US for consumption in the US (A) by classification for 1988 and (B) by use for 1987 (Anon., 1989).

Solution. Relative bulk strength:

Explosive A

$[(645 \text{ cal/cm}^3)/(723 \text{ cal/cm}^3)]$ 100% = 89.2%

Explosive B

$[(980 \text{ cal/cm}^3)/(723 \text{ cal/cm}^3)]$ 100% = 135.5%

Relative weight strength:

Explosive A

$[(645 \text{ cal/cm}^3)/(0.8)\ (850 \text{ cal/cm}^3)]$ 100% = 94.9%

Explosive B

$[(980 \text{ cal/cm}^3)/(1.25)\ (850 \text{ cal/cm}^3)]$ 100% = 92.2%

The value of RBS and RWS of ANFO is 100%. It should be noted that based on a volumetric basis, explosive A is less powerful than ANFO while explosive B is more powerful than ANFO.

Commercial Explosive Products: There are a number of ways in which commercial explosives are classified. Often *military explosives,* referred to as high explosives (HE), are classified separately from industrial explosives. Examples of military explosives include TNT, PETN, RDX (cyclonite), Tetryl, and compositions such as A3, B, B4, C4, which are mixtures of RDX, TNT, and additives to provide a moldable consistency. These explosives are chiefly used in the weapons industry. However, small amounts of HE are added to commercial explosives to increase strength and sensitiveness. Many commercial or industrial explosives are classified as HE because they contain critical amounts of military explosives or nitroglycerin, and usually they are cap sensitive. Others, such as dry blasting agents, are not classified as HE, and require boosters or primers of HE for initiation.

Industrial explosives are classified as one of the following: nitroglycerin-based, dry blasting agents, water gels, emulsions, permissibles, primers, and boosters. Two-component explosives, a common category, actually contain mixtures or characteristics that fall in other classifications. Often the difference among these products is formulation; however, product packaging and consistency can also change a classification. The USBM (Anon., 1989) reported the relative consumption of these products as a percentage of total industrial explosive use and consumption by industry. These consumptions are shown in Fig. 9.2.1.4.

Explosive components are referred to as oxidizers, fuels, absorbants, thickeners, and stabilizers. *Oxidizers* contribute oxygen for oxygen balance, and include nitrated salts such as ammonium nitrate (AN), sodium nitrate (SN), and calcium nitrate (CN). *Fuels* include fuel oil, carbon, granular aluminum, TNT, black powder, or any carbonaceous material that produces heat. Many of these components are also referred to as sensitizers and can also act as absorbants. *Absorbants* are products, such as wood pulp, sawdust, cotton, and cellulose, that incorporate liquid explosive components such as nitroglycerin. *Stabilizers* include flame retardants, gelatins, densifiers, water, gum, emulsifying agents, and thickeners.

Nitroglycerin-based Explosives—Dynamite is a trade name introduced by Alfred Nobel. It is comprised primarily of a stable yet powerful mix of nitroglycerin (nitrostarch). Since its invention, a number of nitroglycerin (NG)-based products have been developed and are outlined in Table 9.2.1.2. The three basic types include granular, gelatin, and semigelatin, which are all considered HE. Gelatins and semigelatins contain nitrocotton that combines with NG to form a gel structure whose consistency is controlled by the percentage of nitrocellulose. Dynamites are packaged in cylindrical cartridges from $\frac{7}{8}$ in. (22 mm) in diameter and 8 to 24 in. (203 to 610 mm) in length. The quality of the waxed paper wrapping is important to water resistance, fume production, and the ease and safety of loading.

Straight dynamite derives its energy source from NG, SN, and AN, including absorbants such as wood pulp and flour that also act as combustibles. These dynamites come in varying weight strengths (40, 50, and 60% being the most common) which correspond to the weight percentage of NG contained in the formulation. Ditching dynamite (50% NG) is a commonly used product. These dynamites are characterized by high velocity and brisance, low flame temperature, and good water resistance; however, these dynamites are sensitive to shock and produce poor fume quality.

Ammonia dynamite (or "extra" dynamite) is a granular mix that contains a smaller quantity of NG mixed with AN and SN. Two varieties are available. The high-density, high-velocity product is available in weight strengths of 20 to 60%. They have poor water resistance and low explosion temperature. The low-density variety comes in either a high- or low-velocity series and a constant 65% weight strength.

Gelatin dynamites are either straight gelatin or ammonium (extra) gelatin. Each has similar mixtures as straight and ammonia (extra) dynamites with the addition of nitrocellulose for a gel consistency. Ammonia gelatins are commonly used as boosters and as primers to initiate cap insensitive explosives.

Blasting gelatin is one of the strongest commercial products, with 92 to 94% NG gelatinized with 6 to 8% nitrocellulose. As the most energetic product available, it primarily is used in the seismic industry, although some commercial products are used as primers and boosters.

Semigelatins are ammonia gelatin with a small amount of nitrocellulose and a 65% weight strength. They are also used as primers and boosters.

Dry Blasting Agents—Dry blasting agents are one form of a general category of blasting agents. A *blasting agent* is, by definition, a mixture of fuel and oxidizer. It is not classified as an explosive, and cannot be detonated with a No. 8 blasting cap. A *dry agent* is a granular, free-running mix of a solid oxidizer (usually AN), prilled into porous pellets onto which a liquid fuel oil or propellant is absorbed. ANFO is the most widely used blasting product, with approximately 94.5% industrial-grade ammonia nitrate and 5.5% No. 2 grade diesel fuel oil for a nearly oxygen-balanced mix. It is available in bulk form for onsite mixing of the AN and fuel or in 50-lb (23-kg) premixed bags as pourable forms. Waxed card board and woven polypropylene tubes of diameters required for various borehole sizes are available for wet hole conditions. Typical values of specific gravity range from 0.75 to 0.95.

The properties of dry blasting agents vary significantly with borehole diameter, density, confinement, particle size, water conditions, and size of primer used for initiation. Fig. 9.2.1.5 illustrates the effect of charge diameter on detonation velocity for ANFO and other selected explosives. The steady state detonation

Table 9.2.1.2. Basic Properties of Nitroglycerin-based Products

Explosive	Specific Gravity	Detonation Velocity, fps	Water Resistance	Fume Quality
Straight dynamite	1.3–1.4	9,000 to 19,000	Poor to Good	Poor
Extra dynamite	0.8–1.3	6,500 to 12,500	Poor to Fair	Fair to Good
Blasting gelatin	1.3	25,000	Excellent	Poor
Straight gelatin	1.3–1.7	11,000 to 25,000	Excellent	Poor to Good
Extra gelatin	1.3–1.5	16,000 to 20,000	Very good	Good to Very good
Semi-gelatin	0.9–1.3	10,500 to 12,000	Fair to Very good	Very good

Source: Dick et al., 1983. Conversion factor: 1 fps = 0.3048 m/s.

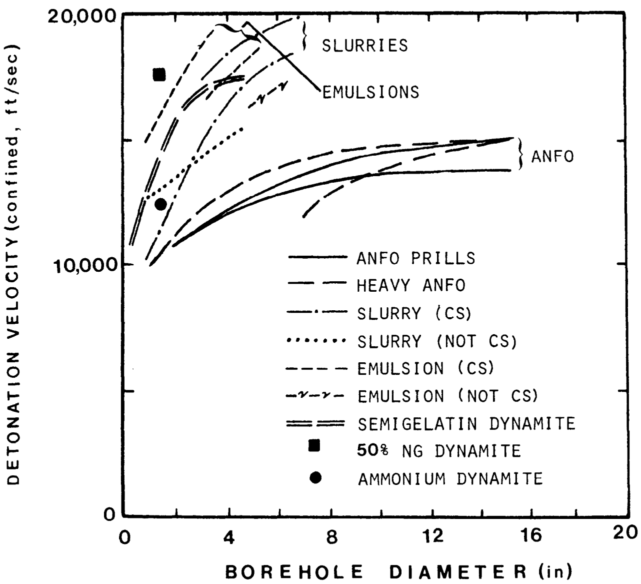

Fig. 9.2.1.5. Variation in detonation velocity with borehole diameter for selected commercial explosives. Conversion factors: 1 in. = 25.4 mm, 1 fps = 0.3048 m/s.

velocity of ANFO is over 15,000 fps (4500 m/s) and is achieved in borehole diameters greater than 15 in. (381 mm). The critical diameter of ANFO is between 2 and 4 in. (51 and 102 mm) and is a topic of controversy among blasters and, in particular, those who blast underground using small-diameter holes. The exact values of the critical diameter depends on the loading conditions; however, ANFO does not detonate reliably within the range cited above. Only explosive manufacturers can verify the minimum reliable diameter for initiation. The cost of ANFO ranges from as low as \$0.10/lb (\$0.22/kg) for bulk form to \$0.15 to \$0.18/lb (\$0.33 to \$0.40/kg) for package forms.

One form of dry blasting agent that is used in a variety of applications is referred to as a high-density (HD) ammonium nitrate agent. AN prills, crushed to a flakelike consistency, are mixed just prior to use with approximately 11% propellant (rocket fuel) such as nitromethane or nitroethane. The resulting product has a density higher than ANFO and generates more gas pressures based on the fuel used. The product is packaged in two components, often mixed in small batches for specialty uses such as secondary boulder breakage or removing tree stumps. Two-component blasting agents and explosive, however, can comprise a variety of mixtures for specialty blasting purposes including seismic exploration and are generally not used in large quantities.

Table 9.2.1.3. Typical Compositions of Some Slurries and Emulsion

Aluminum Sensitized Slurry	Water Gel Slurry
10% Aluminum	13% Amine nitrate
15% Water	15% Water
5% Ethylene glycol	5% Sodium nitrate
44% Ammonium nitrate	3% Ammonium perchlorate
25% Calcium nitrate	63% Ammonium nitrate
1% Guar gum	1% Guar gum
Explosives Sensitized Slurry	**Emulsion**
25% TNT, smokeless powder or nitrostarch	6% Wax/oil
	2% Emulsifier
25% Water	14% Water
15% Sodium nitrate	76% Ammonium nitrate
44% Ammonium nitrate	2% Hollow microballoons
1% Guar gum	

Aluminum in granular form can be added to ANFO to increase the heat or energy output. For increasing percentages of aluminum by weight up to 6%, a measurable increase in fragmentation energy is noted (Thornley and Funk, 1981). The cost of additional aluminum beyond 6% does not result in proportionally increased work output and, therefore, is not cost effective.

Wet Blasting Agents—Blasting agents that contain more than 5% water by weight are referred to as *wet blasting agents.* Within this category are water gels or slurries, emulsions, and heavy ANFO. The latter is a combination of prilled ANFO and emulsion. The development of wet blasting agents, led by slurries in the 1950s, came about in response to the disadvantages of ANFO in certain applications. These were the lack of water resistance and the low bulk strength due to low density. Typical compositions of water gels and emulsions are shown in Table 9.2.1.3. The critical diameter of wet blasting agents is often less than 1 in. (25 mm).

Three varieties of wet blasting agents are in common use in the mining industry.

1. *Slurries* (water gels) are a colloidal suspension of solid AN particles suspended in a liquid AN solution that is gelled, using cross-linking agents. The gels (guar gum) effectively surround the solid AN, rendering the oxidizer water resistant while thickening the explosive mix. Fuels and sensitizers such as TNT, nitrostarch, Composition B, ethyl alcohol, fuel oil, and glass bubbles (microspheres) are dissolved or added to the liquid phase. Granular aluminum, added as a sensitizer, increases weight and bulk strength. Up to 18% aluminum by weight has been found to provide increased energy output. In general, 20% water is used. Certain mixtures, containing high-explosive sensi-

tizers, are cap sensitive and hence should not be classified as a blasting agent, but rather as a slurry explosive.

Slurries are characterized by excellent water resistance, high density and bulk strength, good oxygen balance, confinement, and coupling within the borehole. The density of slurries ranges from 1.1 to 1.3, while detonation velocities vary from 13,500 to 20,000 fps (4115 to 6096 m/s), with borehole diameter as shown in Fig. 9.2.1.5. Detonation pressures range from 0.73 to 1.5 × 10^6 psi (5 to 10 GPa). Experiments performed by Bauer et al. (1984) show that decreasing explosive temperature decreases detonation velocity, while certain explosive diameters failed to detonate below 0°F (–17.8°C).

Products can be delivered for onsite mixing or premixed (only the gelling agent is added during hole loading) for bulk storage and loading. Plant-mixed slurries are packaged in polyethylene cylindrical bags in a variety of diameters and lengths. Onsite mixing has the advantage of varying explosive density and strength for hole loading conditions or for geological requirements. Prices range from $0.18lb ($0.40/kg) for bulk products to as high as $0.35/lb ($0.77/kg) for packaged slurries and those containing aluminum or other sensitizers.

2. *Emulsions* are a two-liquid phase containing microscopic droplets of aqueous nitrates of salts (chiefly AN) dispersed in fuel oil, wax, or paraffin using an emulsifying agent. The water-in-oil structure depends on entrapped air or microspheres for sensitivity, thereby eliminating the need for expensive explosive compounds. Microspheres, microscopic glass, or plastic air-filled bubbles and the AN droplets form the oxidizer, while the fuel oil exists as the oil phase. Emulsions provide high detonation pressures of 1.45 to 1.74 × 10^6 psi (10 to 12 GPa) with detonation velocities between 14,500 to 18,500 fps (4420 to 5640 m/s), as shown in Fig. 9.2.1.5. Densities range from 1.15 to 1.45. Emulsions have excellent water-resistant properties regardless of packaging, and they remain stable at low temperatures. The cost of emulsion products is within the range for slurries.

The excellent properties of emulsions are due to the intimate contact between the microscopic droplets and the continuous oil phase containing the fuel. Over time, the oil phase migrates, and droplets combine to form larger particles sizes whose bulk surface area is reduced. Less fuel is in contact with the oxidizer, and less than ideal explosive properties are achieved. Current research on emulsions will lead to formulations with longer shelf life.

Emulsions can be mixed onsite and pumped from bulk trucks. Premixed emulsions are available in plastic tubes in a variety of diameters and lengths. Depending on product diameter and sensitizers used, emulsions can be cap sensitive; however, the manufacturer must be consulted for primer requirements.

Certain site-mixed emulsions require heating the mixture prior to loading depending on the hydrocarbon used as the fuel. Waxes and paraffin require heating to fluidize the product for ease of loading while fuel oil can be pumped cold.

3. *Heavy ANFO*. Recent developments in explosives have produced a heavy ANFO product that is comprised of up to 45 to 50% ammonium nitrate emulsion mixed with prilled ANFO in an attempt to increase the bulk density of ANFO. The only fuel component is in the ANFO (or a liquid fuel), while the emulsion contains no solid fuel, making the mixture a "repumpable" consistency. The final product has improved strength and provides good water resistance in comparison to ANFO, with a price range between that of ANFO and emulsions. Research has shown, however, that the ability of emulsion to prevent ANFO from being dissolved in the presence of water, thereby reducing blasting efficiency, is questionable.

Bulk trucks for the loading of heavy ANFOs are designed to blend components prior to loading and provide loading of emulsion or ANFO alone, or a combination of the two, in varying proportions. Heavy ANFO products are also available in a variety of packages and sensitivities.

Permissibles—Permissible explosives are those made of special mixtures, which produce short-lived detonation flames, and do not ignite methane or fine coal dust within the air of underground coal mines. Salt (NaCl) is used in the mix to produce a quenching reaction. The USBM (Anon., 1961) developed criteria for permissible explosives and maintains a list of manufacturers supplying permissible products.

Primers and Boosters—A *primer* charge is an explosive ignited by an initiator, which, in turn, initiates a noncap-sensitive explosive or blasting agent. A primer contains cap-sensitive high explosive ingredients. Often cartridges of dynamites, highly sensitized slurries, or emulsions are used with blasting caps or detonating cord. Other primers are cast into specific shapes and weights, using TNT and PETN, designed with wells for initiator acceptance. Most primers are used in weights ranging from 12 oz (340 g) to over 5 lb (2.27 kg). An overview of explosive priming techniques is given by Dick et al. (1983).

Boosters are highly sensitized explosives or blasting agents, used either in bulk form or in packages of weights greater than those used for primers. Boosters are placed within the explosive column where additional breaking energy is required. Oftentimes, cartridge or plastic-bagged dynamites or sensitized wet blasting agents are used as primers as well as boosters. Boosters are often used near the bottom of the blasthole at the toe level as an additional charge for excessive toe burden distances. They are also placed within the explosive column adjacent to geological zones that are difficult to break or intermittently within the main explosive charge to ensure continuous detonation.

Initiators and Initiation Systems: *Initiators* are devices containing high explosives that, upon receiving an appropriate mechanical or electrical impulse, produce a detonation or burning action. Initiators are used as components within a system of explosives and other devices to start the detonation of all other components. Initiation systems are either electric or nonelectric, and include blasting caps, safety fuse, detonating cord, or nonelectric shock tubes. Certain primers, which are shape-cast or contained in plastic shells, are provided with wells for cap or cord insertion.

Electric blasting caps are the most commonly used method of initiation. Electrical energy (ac or dc) is sent through copper or iron legwires to heat an internal-connecting bridgewire. This heat, in turn, starts a chain reaction of explosives burning within the metal cap shell, through a powder delay train. This process detonates a high-explosive base charge, igniting a cap-sensitive explosive. They are manufactured with an instantaneous (no delay train) time of initiation, or time delays (in milliseconds) used in delayed blasting practices. Time delays with intervals of 25, 50, 100, 500, and 1000 ms are available for short- (ms) or long-period (LP) delays. Table 9.2.1.4 summarizes typical delay time intervals available for electric and nonelectric initiating systems. Short delays are used in surface blasting operations, while longer delays are used underground where blasting conditions are more confined. The use of time delays in blasting enhances fragmentation and the control of ground vibrations (see 9.2.2). In recent years, improvements have been made in the manufacturing of blasting caps that increase the accuracy in detonation time. The next generation of high-precision detonators will contain an electronic circuit instead of pyrotechnical delay elements. The integrated circuits will permit microsecond rather than millisecond timing accuracy and allow programmability for onsite selection of each cap detonation timing.

Ac power lines and capacitor-discharge dc power sources approved for blasting are used to energize caps. Precise calcula-

Table 9.2.1.4. Typical Delay Times for Electric and Nonelectric Delays

Short-Period Delays				Long-Period Delays	
Electric		Nonelectric			
Period	Delay time, ms	Period	Delay time, ms	Period	Delay time, sec
0	0	0	0	0	0
1	25	1	25	½	0.1
2	50	2	50	1	0.2
3	75	3	75	1½	0.3
4	100	4	100	2	0.4
5	150	5	125	2½	0.5
6	200	6	150	3	0.6
7	250	7	175	4	1.0
8	300	8	200	5	1.4
9	350	9	250	6	1.8
10	400	10	300	7	2.4
11	450	11	350	8	3.0
12	500	12	400	9	3.8
13	550	13	450	10	4.6
14	600	14	500	11	5.5
15	650	15	600	12	6.4
16	700			13	7.4
17	800			14	8.5
18	900			15	9.6
19	1000				

tions are needed to determine the entire blasting circuit resistance, including all accessory connecting wires. This is to ensure that the power source supplies the correct current to each cap in the circuit. Excellent source on blasting circuit calculations are provided in the *Blasters Handbook* (Anon., 1977) and *Handbook of Electric Blasting* (Anon., 1985a). Safe blasting practices dictate that precautions are used to avoid blasting in the vicinity of extraneous electricity such as stray current, static electricity, electrical storms, and radio frequency energy when using electric caps (Anon., 1974).

Nonelectric initiation systems include a cap similar to that of an electric cap, but they are connected to plastic tubing or a transmission line that carries an initiation (shock and heat) to initiate the cap. The energy source in the tubing is either a gas mixture or an internal coating of special explosive. Nonelectric tubing is not used in underground coal or gassy mines as it carries an open flame. The plastic tube itself does not detonate; therefore, the only noise source is the cap itself. Caps and tubes of varying lengths are connected with special connectors between holes to configure unique blast pattern arrays. Surface delay elements, when used in conjunction with in-hole delays, provide nearly infinite numbers of delays in blasting patterns. Delays are available in short and long periods as well as in-hole and surface delays. The advantage of nonelectric systems over electric systems is the ability to design blasts with a greater number of holes than traditional electric blasting. In addition, concerns about the effects of accidental detonations of electric caps due to stray currents are eliminated with the use of nonelectric systems.

Blasting Safety: Due to the fact that explosive materials are deemed hazardous, the sales, transport, storage, and use of explosives are regulated by a multitude of federal, state, and local agencies. In addition, safe blasting practices dictate that companies or agencies engaged in the use of explosives provide standard operating procedures (SOPs) for personnel that outline the acceptable safe and legal handling of hazardous materials. Depending on the situation under which explosives are being used, the blaster may be required to comply with a number of different codes as well as be tested for certification or license. The Bureau of Alcohol, Tobacco and Firearms of the US Department of Treasury (Anon., 1988c) regulates commerce in explosives importation, manufacturing, distribution, and storage. The US Department of Transportation (DOT) regulates the transport of explosives on highways, railroad, water, and in the air (Anon., 1988b). Of importance is the classification of explosives outlined in these regulations and the requirement of all shippers and manufacturers of packaged products for shipment to identify all explosives and their classification with standard labels. The following list outlines the classification defined by DOT and used in the commercial blasting industry:

Class A Explosives: Explosives that contain detonating materials or materials of maximum hazard: in general, these explosives, are cap-sensitive.

Class B Explosives: Explosives that carry a flammable hazard.

Class C Explosives: Explosives that contain components classified as either Class A or B but in limited quantities.

Blasting Agents: A mixture of fuel and oxidizer that is not defined as an explosive and cannot be detonated with a No. 8 test blasting cap.

Under the Department of Labor, the Mine Safety and Health Administration (MSHA) (Anon., 1988a), and the Occupational Safety and Health Administration (OSHA) (Anon., 1988) regulate the storage and use of explosives in certain mining activities and in general industry use, respectively. MSHA outlines safe blasting practices for surface and underground metal, nonmetal, sand and gravel, and underground coal mines. OSHA similarly outlines procedures for good practices to handle and load explosives in other industries. The Office of Surface Mining, Department of Interior (Anon., 1983), is concerned with the environmental effects and safety of explosives used in surface coal mines. These regulations restrict the quantities of explosive loading in the vicinity of dwellings to control noise, ground vibrations, and flyrock.

An excellent source of explosive safety information is the Institute of Makers of Explosives. The IME is an association of commercial explosive manufacturers concerned with safe and

Table 9.2.1.5. Thermodynamic Data for Selected Explosive Components and Gases

Explosive	Formula	Moles/ kg C	H	O	N	Enthalpy, kcal/ kg
Nitroglycerin	$C_3H_5O_9N_3$	13.21	22.02	39.62	13.21	– 392.0
Ammonium nitrate	$C_4O_3N_2$	—	49.97	34.48	24.99	–1091.0
TNT	$C_7H_5O_6N_3$	30.82	22.01	26.40	13.20	– 62.5
Diesel fuel	CH_2					+ 898.0

Gases	Formula	Heat Capacity Constants, cal/mole-°K a	b, 10^{-3}	c, 10^{-6}	Enthalpy, kcal/mole
Carbon dioxide	CO_2	6.85	8.53	–2.48	–94.05
Water (vapor)	H_2O	6.89	3.28	–0.34	–57.80
Nitrogen	N_2	6.30	1.82	–0.35	0
Oxygen	O_2	6.13	2.99	–9.81	0

Source: Meyer, 1981.

proper practices in transport, storage, and use of their products. Its members are involved with legal affairs and regulatory monitoring of legislation affecting industry operations. A number of valuable publications are available from the IME (Anon., 1985 through 1987).

9.2.1.2 Explosive Thermochemistry

All explosives are mixtures of carbon, hydrogen, nitrogen, and oxygen, plus other components used for special purposes. In order to achieve maximum rock-breaking efficiency from an explosive, an oxygen-balanced mixture is formulated to ensure the formation of nonnoxious gases upon detonation. With an oxygen-balanced formula, it is assumed that optimum values of thermodynamic properties such as energy, temperature, and pressure are achieved, and that these values are not affected by changes in the reactants (explosive mixture). This is the case for ideal reactions. Unfortunately, commercial explosives are nonideal materials. Changes in the physical nature rather than the chemistry of the explosive mixture, such as particle size and borehole diameter, vary the rate of detonation and hence affect thermodynamic variables. Furthermore, many granular explosives, such as ANFO and nitroglycerin, do not completely react, whereas water-based blasting agents react more efficiently, releasing optimum energy as predicted by formulation.

It is possible, however, to estimate the thermodynamic properties of an explosive reaction, assuming an oxygen balance. The following calculations are made to illustrate the methods used to estimate explosive properties of interest. Although there are a number of methods available (Cook, 1958, 1974; Manon, 1976–1977; Meyer, 1981), the procedures selected herein are the least difficult to apply. Thermodynamic data used for these calculations are found in Table 9.2.1.5 and given by Meyer (1981).

Oxygen Balance: As explained in 9.2.1.1, an oxygen-balanced explosive formula is one in which the amount of oxygen O_2 is sufficient to form the desired detonation gases CO_2, H_2O, and N_2, but not noxious fumes such as CO, NO, and CH_4 (if the mix is oxygen deficient), or free oxygen (if the mix has an excess of oxygen). Oxygen balance for an explosive component or a mixture of components is usually reported in percentages or as moles of monatomic oxygen. It is computed using the mass balance relationship for a reaction of 1 kg of explosive, assuming the formation of ideal gaseous products. The computations are illustrated using nitroglycerin, TNT, and ANFO mixture.

Example 9.2.1.2. Find the oxygen balance for nitroglycerin, $C_3H_5O_9N_3$.

Solution. The mass-balance equation is

$$\begin{aligned} C_{13.21}H_{22.02}O_{39.62}N_{13.21} &\rightarrow 13.21\ CO_2 \\ &+ 22.02/2\ H_2O + 13.21/2\ N_2 \\ &+ (39.62/2 - 13.21 - 11.01/2)\ O_2 \end{aligned} \tag{9.2.1.2}$$

where subscripts denote mole numbers of each element in 1 kg. Oxygen in excess

$$\begin{aligned} &= +1.095 \text{ moles } O_2 \\ &= +2.19 \text{ moles O} \end{aligned}$$

Oxygen balance (OB)

$$\begin{aligned} &= \{[\text{O (excess)}]/\text{O (available)}\}\ (100\%) \\ &= 5.53\% \end{aligned}$$

With the exception of ammonium nitrate and nitroglycerin, most explosive components are oxygen deficient. Oxygen balance for TNT is given next in Ex. 9.2.1.3.

Example 9.2.1.3. Find the oxygen balance for TNT, $C_7H_5O_6N_3$.

Solution.

$$\begin{aligned} C_{30.82}H_{22.01}O_{26.4}N_{13.2} &\rightarrow 30.82\ CO_2 \\ &+ 22.02/2\ H_2O + 13.2/2\ N_2 + (26.4/2 \\ &- 30.82 - 22.01/4)\ O_2 = \\ &-42.25 \text{ moles O} \end{aligned} \tag{9.2.1.3}$$

Oxygen balance for an oxygen-deficient component is

$$\begin{aligned} &= \{\text{O (deficient)}/[\text{O (deficient)} - \text{O (required)}]\}\ (100\%) \\ &= -63.7\% \end{aligned}$$

Example 9.2.1.4. Find the oxygen balance for the mixture, ANFO.

Solution. For the mix of 94.5% AN, NH_4NO_3,

$$5.5\% \text{ FO, } CH_2 \text{ (approximate)}$$

For 1 kg of each component, the subscripted mole numbers are

$$\text{AN} \quad H_{49.97}O_{37.48}\,N_{24.99}$$
$$\text{FO} \quad C_{69.9}\,H_{139.86}$$

Mass balance for the mixture

Percentage of component in mixture	Mole contribution to each element			
	C	H	O	H
94.5% AN	—	47.22	35.42	23.62
5.5% FO	2.05	7.85	—	—
Total moles in 1 kg mix	2.05	55.07	35.42	23.62

$$C_{2.05}H_{54.91}O_{35.42}N_{23.62} \rightarrow 2.05\ CO_2$$
$$+\ 55.07/2\ H_2O + 23.62/2\ NO_2$$
$$+\ 1.89\ O_2 = +3.78 \text{ moles O}$$
$$\text{OB} = 10.7\% \tag{9.2.1.4}$$

Explosive Energy: Energy is expressed in terms of heat of formation (or enthalpy) for a reaction involving constant volume, given as kilocalories (kcal). It is computed as the sum of the heats of formation for the products minus the sum for the reactants. The reaction is based on 1 kg of explosive component or mixture (reactants). The enthalpies are given for the reactants in terms of kilocalorie per kilogram (calorie per gram or weight strength), and for the gaseous products, in terms of kcal/mole. Using the single component NG and the number of moles formed of the various gas products as shown in Eq. 9.2.1.3, Ex. 9.2.1.5 calculates energy.

Example 9.2.1.5. Energy (heat of formation) for nitroglycerin

Solution.

$$Q = [13.21 \text{ moles } (-94.05 \text{ kcal/mole}) + 11.01 \text{ moles}$$
$$(-57.8 \text{ kcal/mole})] - (-392.0 \text{ kcal})] =$$
$$-\,1487 \text{ kcal/kg of NG}$$

For mixtures, the enthalpy contribution of each component must be determined.

Example 9.2.1.6. Find the energy for ANFO.

Solution.

Net enthalpy

= [(– 1091 kcal/kg)(0.945 kg AN) + (898 kcal/kg)(0.055 kg FO)]

= –981.61 kcal

Energy, Q

= [2.05 mole (–94.05 kcal/mole) + 27.54 mole (–57.8 kcal/mole) – (–981.61 kcal)]

= –803.64 kcal/kg (9.2.1.5)

Explosion Temperature: The theoretical temperature can be calculated assuming that the detonation reaction takes place under an adiabatic condition, and that the reaction takes place as a constant pressure process from ambient temperature T_1. In this case, the temperature of the explosion T_2 is calculated by expressing the enthalpy of the products as a function of their temperature. The enthalpy Q is expressed in terms of T_1 and T_2 for n moles of product as

$$Q = n \int_{T_1}^{T_2} (C_p)\, dT \tag{9.2.1.6}$$

Expressing C_P, molal heat capacity, as a quadratic function of temperature, $C_P = a + bT + cT^2$, where a, b, and c are constants experimentally derived and available in standard chemistry hand books. Integrating Eq. 9.2.1.6, the heat of formation is

$$Q = n\,[a\,(T_2 - T_1) + b/2\,(T_2^2 - T_1^2) + c/3\,(T_2^3 - T_1^3)] \tag{9.2.1.7}$$

For a reaction involving more than one product, the enthalpy of each product is obtained in the form of Eq. 9.2.1.7. The sum of all products is set equal to the heat of formation of the reaction (based on 1 mole of explosive), and explosion temperature T_2 is determined.

The following example involving ANFO illustrates the calculations involved to compute explosion temperature. The constants for heat capacity of the products are given in Table 9.2.1.5.

Example 9.2.1.7. Find the temperature of ANFO detonation.

Solution.

Heat of formation based on cal/mole:

Component	Percentage	Atomic Mass, g/mole	Weighted Atomic Mass, g/mole
AN	94.5%	80	75.60
FO	5.5%	14	0.77
Total			76.37

mole/kg = (1000g/kg)/(76.37 g/mole) = 13.09 mole/kg

Enthalpy

= (–803.64 kcal/kg)/(13.09 mole/kg)

= –61,393 cal/mole (9.2.1.8)

From Eq. 9.2.1.5, for 13.09 moles of ANFO explosive, 1 mole of ANFO produces

$$0.157 \text{ mole } CO_2 + 2.104 \text{ moles } H_2O + 0.9 \text{ mole } N_2$$
$$+\ 0.144\ O_2 = 3.31 \text{ moles of gas products}$$

Substituting T_1 = 298°K in Eq. 9.2.1.7 for each gas product gives the following results:

$$CO_2 = .157\,[6.85\,(T_2 - 298) + 8.533/2\,(10^{-3})(T_2^2 - 298^2) - 2.48/3\,(10^{-6})(T_2^3 - 298^3)]$$

$$H_2O = 2.104\,[6.89\,(T_2 - 298) + 3.28/2\,(10^{-3})(T_2^2 - 298^2) - 0.34/3\,(10^{-6})(T_2^3 - 298^3)]$$

$$N_2 = 0.90\,[6.30\,(T_2 - 298) + 1.82/2\,(10^{-3})(T_2^2 - 298^2) - 0.35/3\,(10^{-6})(T_2^3 - 298^3)]$$

O_2 = 0.144 [6.13 (T_2 – 298) + 2.99/2 (10^{-3})(T_2^2 – 298^2) – 0.81/3 (10^{-6})(T_2^3 – 298^3)]

The sum is equal to –61,393 cal/mole, and the cubic form (Eq. 9.2.1.7) becomes

$$0 = 22.137\ T_2 + 0.00516\ T_2^2 - 5.13\ (10^{-7})\ T^3 - 68365$$

The solution to the cubic equation has three roots (determined graphically or solved numerically), one of which lies between reasonable values of 2000 to 5000°K. The graphical solution gives the explosion temperature for ANFO as 2204°K.

Detonation Pressure: The equation of state for explosive gases produced by detonations must define the temperature-pressure-volume relationships at high temperatures and pressures. Many equations of state for nonideal gas pressure calculations are proposed (Cook, 1958; Fickett and Davis, 1979; Mader, 1979; Johansson and Persson, 1970). These solutions require the use of large hydrodynamic computer codes and the knowledge of empirically derived constants from high-pressure experiments. A simple expression used to estimate pressure is the covolume equation of state:

$$P\,(V_e - \alpha) = n\,R\,T \qquad (9.2.1.9)$$

where V_e is specific volume of the explosive (inverse of r explosive density), T is explosion temperature in °K, n is the number of gas moles, and R is the gas constant, 82.06 cm^3-atm/mole-°K. Covolume a is a measure of the actual volume of gas molecules. Pressure is thus related to the inverse of P – a, or free volume. Experimental values of a are given by Cook (1958) as a function of r and approximated by,

$$\alpha = e^{-0.4\rho} \times 10^3\ cm^3/kg \qquad (9.2.1.10)$$

Example 9.2.1.8. Calculate the detonation pressure for ANFO.

Solution.

Given 1 kg of ANFO mixture,

n = 43.285 mole

T = 2204 °K

r = 0.85

a = 711.8 cm^3/kg (from Eq. 9.2.1.9)

V_e = (1000 g/kg)/(r) = 1176.5 cm^3/kg

$$\text{Pressure, } P = \frac{(43.285\text{ moles})(82.06\text{ cm}^3\text{-atm/mole-°K})(2204\text{°K})(14.7\text{ psi-atm})}{(1176.5\text{ cm}^3/\text{kg} - 711.8\text{ cm}^3/\text{kg})}$$

$$= 0.247 \times 10^6\ \text{psi (1.7 GPa)} \qquad (9.2.1.11)$$

For explosives that are not oxygen balanced such as TNT, experimental data for n are required and vary widely among experimentalists. Cook (1958) among others gives experimental data for TNT.

Example 9.2.1.9. Calculate the detonation pressure for TNT.

Solution.

Given 1 kg of TNT,

n = 23 moles

T = 4100°K

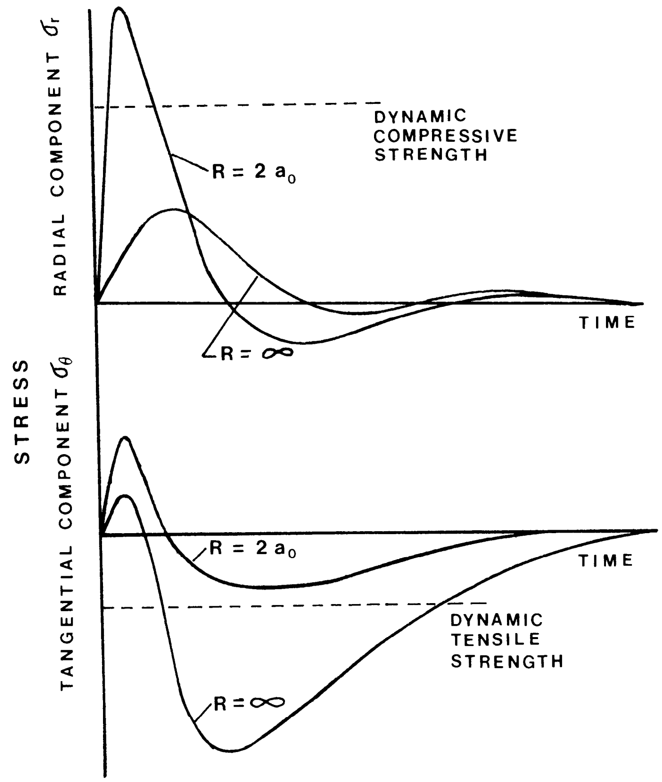

Fig. 9.2.1.6. Generalized stress vs. time for radial and tangential components of stress at two distances R from the borehole center (a_o = original borehole radius).

r = 1.59

P = 1.1 × 10^6 psi (7.57 GPa)

which agrees with experimental results.

9.2.1.3 Blasting Practices

The use of explosives to break rock requires the proper selection of explosives and blasting devices, the careful design of borehole patterns, loading characteristics, and delay blasting sequence, and the control of ground vibration, airblast, and flyrock. Efficient blast designs produce the desired particle size distributions and placement of muckpiles for ease of rock removal and handling.

Rock Breakage Using Explosives: There are a number of theories used to describe rock fragmentation by blasting (Winzer and Ritter, 1980; Anon., 1987a). Two broad areas of breakage mechanisms include (1) the role of stress waves generated from the explosive detonation (shock) force, and (2) the role of borehole pressures created by the detonation gas products.

Effect of Stress Waves—The theoretical treatment of explosively generated stress waves is given by Kutter and Fairhurst (1971), Rinehart (1975), and Mohanty (1985). Upon detonation within a borehole, a shock wave is generated and travels into the rock, quickly decaying in peak pressure amplitude and dispersing in shape as the wave travels away from the borehole. The cylindrically divergent wave carries both a radial and tangential stress component whose stress time histories are shown, idealized, in Fig. 9.2.1.6. The response of the rock adjacent to the borehole

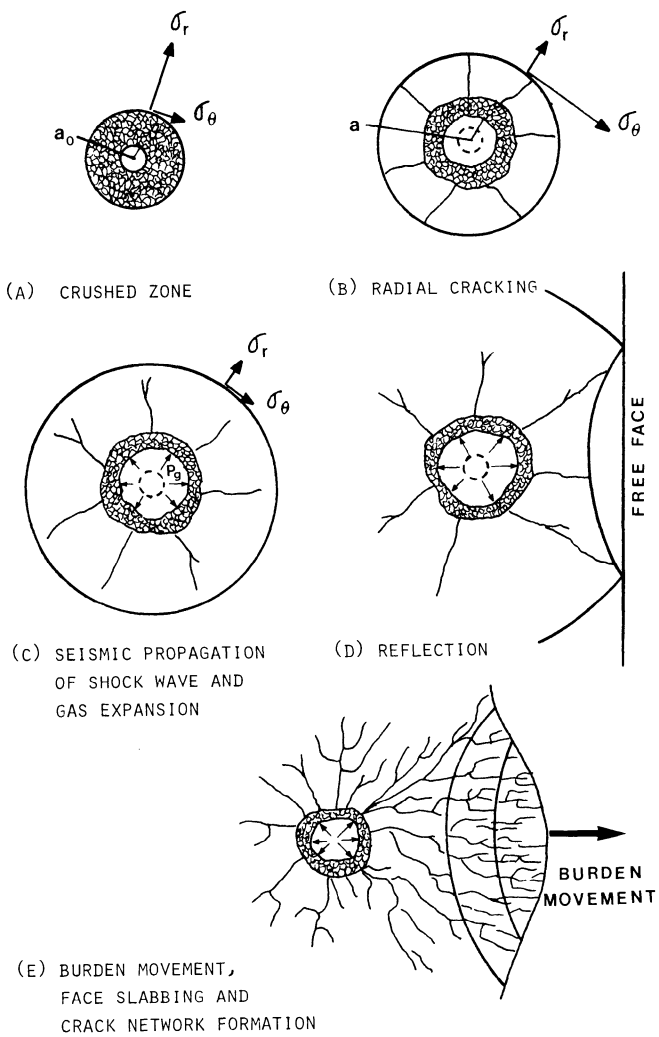

Fig. 9.2.1.7. Generalized plan view through detonating borehole showing sequence of events occurring in the rock mass, where a_0 and *a* are charge radius and expanded borehole radius, respectively, P_g is borehole pressure, s_q and s_r are tangential and radial stress components, respectively.

is shown in Fig. 9.2.1.7. Close in to the borehole, the radial component carries a high compressive stress exceeding the dynamic compressive strength of the rock. The rock experiences intense shearing in this zone. At greater distance from the borehole, this component attenuates below the rock strength, defining the end to the crushing zone. The tangential component carries a tensile phase that rapidly increases to a level greater than the dynamic tensile rock strength. The rock surrounding the borehole develops radial fractures as the rock fails in tension. Once the tensile tail of this component attenuates below the tensile strength, the fractures cease, and the wave travels radially away as an elastic sound wave.

Depending on the explosive and the rock strengths, characteristic degrees of rock damage occur at varying distances from the borehole. Experimental data for limits of the crushed and fractured zone distances are given for a variety of rock and explosive strengths (Siskind and Fumanti, 1974; Anon., 1987a). The crushed zone ranges from 2 to 4 borehole radii away from the center of the borehole, while the fracture zone averages 20 borehole radii away and extends to 50 radii away.

In the vicinity of free surfaces, such as the ground surface or a mining bench face, an enhancement of fracture patterns occurs. This is caused by the transition of the radial component from a compressive to a tensile pulse at the free face as the wave is reflected. The interaction of all waveforms produces intense fracturing within the rock mass between the borehole and free surfaces (e.g., burdens). Recording of crack development in polymeric materials with time-elapsed photoelastic techniques has been used to characterize fracture patterns and fragmentation for stress waves (Kutter and Fairhurst, 1971; Barker et al., 1978, 1978a). Multiple reflections of outgoing and reflected waves (primary and shear) occur while fracturing takes place, dictating flaw initiation sites. The resulting fragmentation, however, is highly dependent on inherent geological discontinuities that are absent in polymeric materials. In studying free-surface fracturing in limestone, Winzer and Ritter (1980) noted surface crack formation formed independently of radial crack development. Timed photography showed the effects of bedding and joint (inherent flaw) influence on face cracking during burden acceleration. Trapped waves reflecting in detached rock particles contributed to continued particle size reduction.

In competent dense rock, excellent fragmentation occurs when using explosives that produce high shock forces. NG-based explosives and highly sensitized blasting agents work well in these rocks.

Effects of Gas Pressures—Subsequent to detonation, gas products are formed, internally pressurizing the borehole and adjacent cracks. Some researchers believe these pressures, often referred to as borehole pressures, are sufficiently high to extend radial cracks (Porter and Fairhurst, 1970; Langefors and Kihlstrom, 1978) while others believe these pressures promote only heaving. Heaving assists in muckpile placement by opening internal surfaces preconditioned by stress waves (Kutter and Fairhurst, 1971). High-speed photography of blasting has shown additional fragmentation occurring during gas-driven heaving of rock thrown from the bench (Hagen, 1977; Chiappetta et al., 1983), while Winzer and Ritter (1980) believe stress waves trapped in detached blocks promote increased fragmentation. In general, explosives formulated with nitrates of ammonium or sodium produce large volumes of gas contributing to good heaving and throwing. Their use in well-bedded and jointed material such as sedimentary rock or highly altered ore, promote good fragmentation.

Effects of Geology on Blasting—Rock properties that influence blasting results are strength, density, sonic velocity, and structure. Rock structures are defined using the frequency and orientation of bedding planes, joints, and other naturally occurring fractures. Characteristic impedance, which is the product of density and velocity, is useful for matching an explosive to the intact rock specimen properties. Knowing the density and wave velocity of the rock, one selects an explosive with a similar impedance value. Often such decisions must also consider the effects of fracture frequency defining the competency of a rock.

The selection of explosive loading quantities and explosive type can be guided by rock-mass fracture data (Hoek and Bray, 1977). The decision to use a ripper, for example, rather than explosives for construction applications can be made with the help of empirical data provided by the equipment manufacturer using rock seismic velocity charts or rock fracture frequency and compressive rock data (Anon., 1987b). If explosives are selected, the orientation of borehole pattern and hole-loading characteristics are important to the blasting results. As shown in Fig. 9.2.1.8, hard rock layers require additional energy while soft clay seam intervals may not require loading. The placement of explosives within the holes can dictate muckpile movement and influence overall muckpile shape. The orientation of boreholes,

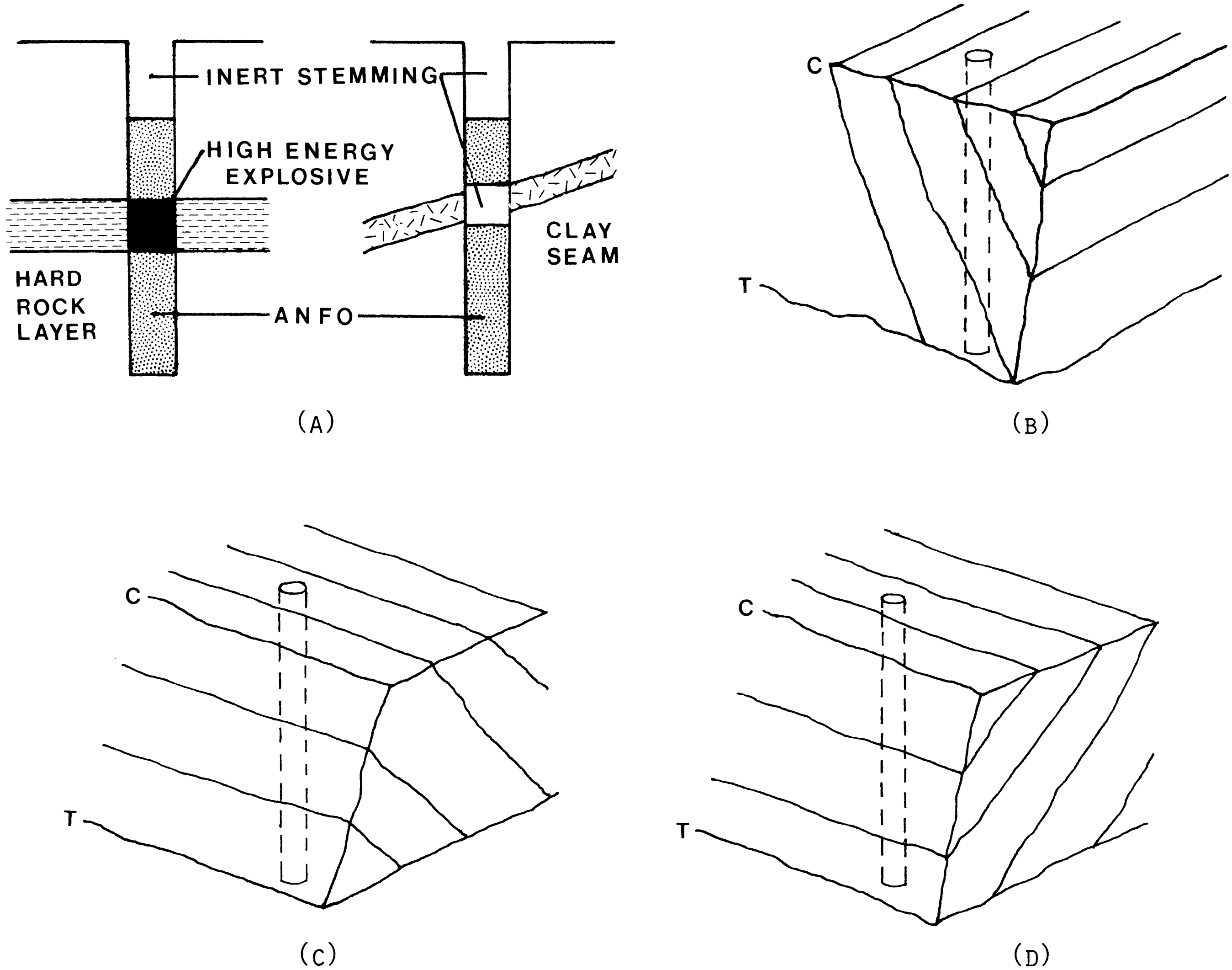

Fig. 9.2.1.8. Geological considerations in blasting: (A) explosive loading for geological variations; (B) blasting against the strike; (C) and (D) blasting with the strike, with the dip and against the dip, respectively (*C* = bench crest; *T* = bench toe).

pattern spacing, and orientation of free faces will determine the efficiency of open pit blasts (Ashby, 1980). Borehole spacing aligned with the joint strike can be widened for optimum fracture development, while shooting with the dip, rather than against, results in safer highwalls and requires the use of less explosive energy.

Surface Blast Design: Surface blast designs require the selection of hole spacing *S*, burden *B*, charge weight *W* or powder factor *PF*, top-hole stemming length *T*, and subgrade drilling depth *J*. Design parameters are shown in Fig. 9.2.1.9. Borehole patterns are drilled square ($S/B = 1$) or rectangular ($S/B \geq 1$) on center or offset (staggered). The sequence of hole initiation timing, *S/B* ratio, actual timing between charge detonations, and number of blasthole rows determine the shape of the broken rock pile as well as the degree of rock fragmentation.

An empirical approach is taken in blast design as blasting is a never-ending process of fine-tuning and modifications. This approach is necessary due to the many factors that cannot be controlled, such as geology and explosive loading conditions. Empirical relationships used in the design of blasting have been proposed by Ash (1963), Pugliese (1973), Van Ormer (1973), Hagen (1981), Dick et al. (1983), and many others. Borehole diameter and burden are perhaps the most important factors used in design. Burden values should be selected based on geology and explosive energy output. Usually hole diameter is set by the drill rig capacity, which is matched to the range of hole depths anticipated for the job. It is desirable to select a size that will provide an adequate powder factor (the ratio of explosive quantity used to the yield of rock breakage) for breakage while distributing the explosive evenly throughout the hole depth. Fragmentation and particle size distribution are a function of hole diameter and burden. The capacity of the excavation equipment dictates the required fragmentation. The charge length to charge diameter ratio for a cylindrical charge should be five or greater.

Ash (1963) has provided simple empirical formulas to compute burden, spacing, subgrade, and stem lengths using "K factors," shown in Table 9.2.1.6. Other rules of thumb are in agreement with the range of acceptable multiplying factors given by Ash. However, many relationships exist for stem length *T*. This is because the determination of *T* is essentially site specific. If too short a value of *T* is selected, air pressure can evolve

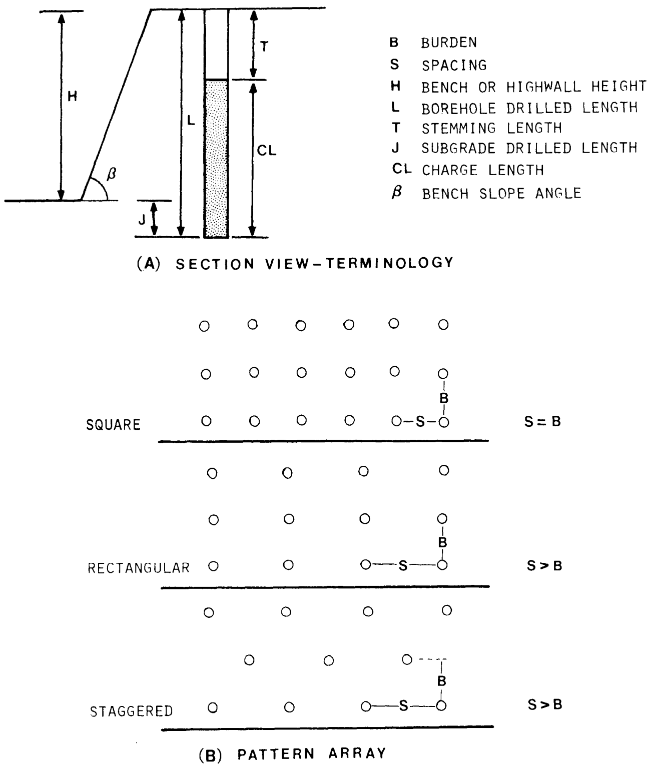

Fig. 9.2.1.9. Blasthole section view (A) showing terminology used in design and (B) pattern array for layout of holes.

Table 9.2.1.6. Selected Factors for First-Approximation Surface Blast Designs

		Using ANFO:
Burden	$B = K_B$	K_B = 22 for rock density < 2.7 g/cm^3
		= 30 for rock density > 2.7 g/cm^3
		Using slurry, dynamite or other high explosive:
		= 27 for rock density < 2.7 g/cm^3
		= 35 for rock density > 2.7 g/cm^3
Spacing	$S = K_S B$	K_S = 1 to 2, depending on initiation
Subgrade	$J = K_J B$	K_J = 0.2 to 0.5 (average 0.3)
Stemming	$T = K_T B$	K_T = 0.5 to 1.3 (average 0.7)

Source: Ash, 1963.

through the stem, disturbing nearby residents. If T is too long, rock breakage near the hole collar is poor. Recommendations for T (in feet) range from 1.2D to 2D (based on charge diameter in inches), or 0.5B to 1.3B. The lower range of these two relationships should be used with caution if airblast or flyrock is a problem.

Once the design parameters are established, the powder factor PF is computed as the quantity of explosives used divided by the volume yield of material blasted, or,

$$PF = \frac{(W \text{ lb})\ (27 \text{ ft}^3/\text{yd}^3)}{[(B)\ (S)\ (H)]\ \text{ft}^3} \qquad (9.2.1.12)$$

Powder factors range 0.25 to 2.5 lb/yd^3 (0.15 to 1.5 kg/m^3) for surface blasting but average 0.5 to 1 lb/yd^3 (0.3 to 0.6 kg/m^3). Higher powder factors result in fine fragmentation and are

Table 9.2.1.7. Typical Powder Factors Used in Rock Blasting

Excavation Method		Range in Powder Factors, lb/yd^3
Surface metal mining		0.6–1.0
Surface coal mining	60 yd^3 dragline	0.5–0.7
	30 yd^3 shovel	0.6–1.1
	17 yd^3 front-end loader	0.6–1.6
Coal mining blast casting		0.9–1.5
Quarrying		0.6–1.5
Construction	open excavations	0.25–0.8
	trenching	2.0–3.0

Conversion factor: 1 yd^3 = 0.7646 m^3, 1 lb/yd^3 = 0.5933 kg/m^3.

required for small capacity removal equipment such as front-end loaders. Higher powder factors result in coarser fragmentation and are typically used for rock removal using draglines and large shovels. Table 9.2.1.7 gives typical powder factor values for various surface blasting situations. Powder factor is often reported as the rock yield in tons per pound of explosive used.

Example 9.2.1.10. Determine the blast design for a copper porphyry open pit mine.

Given blasthole diameter D = 9.25 in., bench height H = 50 ft, rock density r = 2.55 (quartz monzonite), and explosive density r = 0.85.

Subdrilling is required and the nearest dwelling is 7 mi away.

Solution. The following values are obtained as a first approximation:

burden	B = 30 (9.25/12)	subdrilling	J = 0.3 B
	= 23 ft		= 7 ft
spacing S	= 1.3	collar stemming	T = 1.2 D
	= 30 ft		= 12 ft

Thus drilled length L = 57 ft and loaded hole length is 45 ft. The ratio of H/B is 2.17, and the ratio of charge length to diameter is 5.2. The maximum charge that can be loaded is

$$LD = 0.3405\ (0.85)\ (9.25 \text{ in.})^2 = 24.8 \text{ lb/ft}$$

$$W = 45 \text{ ft } (24.8 \text{ lb/ft}) = 1116 \text{ lb loaded/hole}$$

Powder factor and rock breakage yield per hole are

$$PF = \frac{(1116 \text{ lb})\ (27 \text{ ft}^3/\text{yd}^3)}{(32 \text{ ft})(40 \text{ ft})(50 \text{ ft})}$$

$$= 0.87 \text{ lb/yd}^3$$

$$\text{Volume} = 1280 \text{ yd}^3 \text{ of usable ore}$$

The yardage is computed assuming the subgrade material is shot but not removed until the lower bench level is blasted.

Many blast designs use decked charges formed by dividing the explosive column into two or more individual charges, initiated on the same or different time delays, separated by inert stemming material. Decking is employed to (1) conserve explosive use adjacent to weak rock zones, faults, or clay seams; (2) reduce the charge quantity detonated on one time delay, lowering ground vibrations; or (3) bring the powder column up higher in the hole to assure good breakage near the collar. Decked charges should be separated by stemming materials at a length beyond which two adjacent decks do not affect one another. If interdeck stemming is too small, the deck designed to initiate on the earlier time delay may prematurely initiate the second deck. This situation is referred to as *sympathetic detonation* and may lead to excessively high ground vibrations or flyrock. A rule of thumb for the design of interdeck stem length is to employ the hole radius dimension in feet.

The following example gives the design procedures for a blast design in which the explosive charge is limited to control ground vibrations.

Example 9.2.1.11. Using Ex. 9.2.1.10 and limiting W to 300 lb/delay, a modification to the previous design is required such that the powder factor is kept at 0.87 lb/yd^3, using 300-lb decks. Collar stem and subgrade drilling remain unchanged.

Solution. Charge length per deck, (300 lb)/(24.8 lb/ft) = 12 ft/deck.

If three decks are used, 36 ft of hole length is required, leaving 9 ft of interdeck stem length remaining between the decks, or 4.5 ft between two decks. Using the expression for powder factor, a new burden and spacing are computed:

$$PF = \frac{(3 \text{ decks})\ (300 \text{ lb/deck})\ (27 \text{ ft}^3/\text{yd}^3)}{(B)\ (1.3\ B)\ (50 \text{ ft})} = 0.87 \text{ lb/yd}^3$$

$$B^2 = 430 \text{ ft}^2$$

$$B = 21 \text{ ft}$$

$$S = 1.3\ (21 \text{ ft}) = 27 \text{ ft}$$

Thus total hole charge weight W=900 lb. Yield per hole is 1050 yd^3.

Hole Loading Practices—Safe loading practices are given by Dick et al. (1983) and Osen (1985). Recommended procedures include taking adequate precautions and using proper accessories during the hole-loading process. Keeping careful records summarizing blast design dimensions and loading quantities is essential to safety and economics.

Once drilled, all holes should be plugged or covered to prevent rocks and drill cuttings from filling the hole. Prior to loading, the driller's log should be checked for hole depths, subdrilling, and indications of hard or soft seams, voids, and the presence of water. The blaster must check each hole for any change in conditions prior to final design. This is to ensure that all safety precautions are considered. Abandoned holes must be filled with cuttings to prevent flyrock.

Transporting the blasting crew and explosives to the blasting site is done once the site has been identified with markers and the site cleared of all personnel not involved in hole loading. Initiators, cap-sensitive explosives, and noncap-sensitive agents must be transported in accordance with regulations in approved vehicles, carrying classification signs. Proper tools for loading include a cloth measuring tape, a wooden tamping pole, a pair of wire cutters and strippers, a mirror, grappling hooks, and a powder punch. If electric caps are to be used, an approved blasting multimeter or galvonometer must be used to check circuit resistance.

If the blastholes are filled with water, and a water-resistant explosive of a density greater than one is not used, the holes must be pumped or blown free of excess water. Plastic sleeves or hole liners, available to protect from water seeping back into the hole over a limited time period, minimize the desensitization of granular explosives and blasting agents.

In holes that are partially filled with water, a wet blasting agent recommended in wet-hole loading is used below the water level, while granular agents may be placed above. In this case, plastic tubes are slit along the axis for good coupling.

Once the delay sequence is designed, in-hole or surface delays are placed adjacent to respective holes. Each hole is then loaded after it is measured for correct depth. If primers are used for noncap-sensitive explosives, they are made up at the time of loading. Caps or detonating cords are inserted and wrapped around the primer in accordance with manufacturer's recommendations. A small amount of main charge is added to the hole bottom, then the primer carefully lowered. The main charge is poured (for free-running explosives) or carefully lowered (if packaged), while measuring the loading depth to ensure that each deck or column rises to the designed length (based on known loading density). Care must be taken not to overload holes. Any loading of charge that does not result in a measurable powder column rise must be stopped immediately. This situation may indicate a loss of explosive into cavities or open fractures. It may be necessary to plug the open hole section and stem a safe distance, continuing the loading with a separate deck. If this is not possible, a second primer and initiator may be placed at the charge top and the hole should be stemmed to the top.

The location of the primer for a single explosive column is a matter of choice. Primers can be placed at the toe, the middle, or near the top of a column of deck charge. In certain bedded rock, toe priming provides the best results, while some blasters feel middle priming works well. High-energy primers, if used too close to the collar stem base, can displace the stemming material, sending wasted energy into the air. Top priming should be used with care, particularly when collar stem lengths are short or the stemming is damp, to prevent flyrock and excessive air concussion. Multiple priming is often employed as an added safety measure in the event of explosive column cutoffs. Detonating cord downline cutoffs can occur in horizontally bedded rock formations as bedding units shift during the detonation of adjacent holes (Fig. 9.2.1.10). A second downline and primer near the explosive column top may prevent a misfire.

Initiating systems are also a matter of choice. Detonating cord is easy to use; however, cutoffs can occur if flyrock severs surface lines. High-grain core load detonating cord tends to promote the "rifling" of explosive energy through an open channel within the collar stem as the stem material compresses from the initiation shock force, as shown in Fig. 9.2.1.10. Angles between a branch and the line carrying the incoming detonation wave should be 90° or more, and connections should be made with slip-proof knots.

When using electric blasting caps, precautions must be made to monitor extraneous currents at the blast site. Always check the resistance of each cap before stemming the hole to ensure continuity. If the detonator is damaged, a second initiator can be placed in the hole. Once all cap, connecting, and firing line wires are joined, the resistance must be checked and compared to the calculated resistance for agreement. Nonelectric initiating systems are recommended when blasting in the vicinity of stray currents or to control air concussion. As nonelectric systems comprise surface and in-hole delays as well as tubing, the blaster must follow the manufacturer's recommended procedures for layout and connection.

Detonating cord downlines must be cut from the supply spool and held securely during hole loading. Should the cord slip into the hole during loading, a second primer and initiator must be immediately placed in the hole before loading is continued. Blasting cap legwires must also be held securely during loading, and care must be taken not to damage legwires.

Stemming material should provide a high degree of frictional resistance against the explosive detonation force moving upward from the charge base. In dry holes, drill cuttings work well;

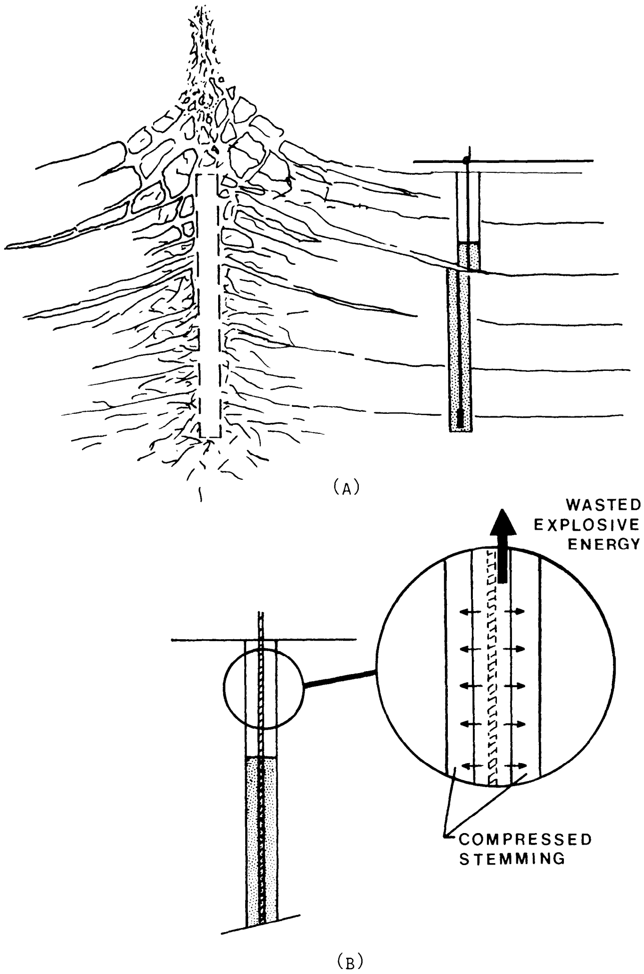

Fig. 9.2.1.10. (A) Detonating cord downline cutoff caused by shifting of horizontally bedded rock, and (B) stemming compression leading to a loss of explosive energy

however, when cuttings are wet, they provide little resistance. In this case, $^3/_8$-in. (9.53-mm) crushed stone is recommended.

Prior to shot initiation, the blast site and surrounding area must be cleared and all access to the blast site must be guarded. A sequence of audible sirens of a set code precedes the blast, with an "all clear" siren following the blast. During shot firing, all personnel must be adequately covered and fully protected from flyrock. A certain time period must pass before entering the blast site as both flyrock and noxious fumes may be present for an extended period of time after detonation.

If a misfire occurs, the charge should be flushed from the hole with water or air. If this is not possible, the cord, legwires, or tubing may be reconnected, if available, once continuity is established. If continuity is not established, the hole should be reprimed after the stemming is flushed out. In this case, care must be taken during initiation as excessive flyrock may result. In the case that it is not possible to use these approaches, qualified professional assistance must be sought. With the assistance of experts trained to handle misfires, the charge may be carefully dug out or a hole may be drilled adjacent to the unshot hole. The adjacent hole is then lightly loaded, and detonated, to lift the rock surrounding the unshot charge for removal. Under no circumstances should these attempts be made by unexperienced blasting personnel without consultation with their supervisors and experts in this field.

Surface Delay Blasting—Delay blasting techniques are employed to improve fragmentation, control of rock movement, overbreak, and to reduce ground vibrations. Delays are incorporated into the blast design using electric or nonelectric caps or delay connectors with detonating cord. The delay patterns used in design will determine the sequence of hole or deck initiations, thereby, dictate the overall direction of blasted rock movement and resulting fragmentation. Depending on the *S/B* ratio, the actual timing (in milliseconds) between detonating charges will determine muck pile displacement height and distance from the bench. Fig. 9.2.1.11 shows variations of timing patterns used for surface blasting. Depending on initiation sequence, an effective burden B_e and effective spacing S_e result as shown in Fig. 9.2.1.11. The effective spacing is the distance between holes in a row defined by adjacent time delays (e.g., delays by rows). Effective burden is the distance in the direction of resultant rock mass movement. The V and echelon (diagonal) patterns are used when rock placement is restricted. Designs using two free faces usually provide improved fragmentation and throw control over those using a single face.

The design of initiation timing for multiple-hole blasting is critical to the blasting effectiveness. If the interhole delay is too short, the movement of row burdens is restricted and fragmentation is poor. High ground vibrations result, and backbreak along the new highwall may persist, jeopardizing the stability of the slope. If interhole delays are too long, cutoffs of surface delays may occur. The minimum time for design is controlled by the stress wave travel distance (= 2 B_e) in order for radial cracking to begin to develop, contributing to the detachment of the rock mass in the vicinity of the hole. This detachment forms an internal free face (or relief) to which successive detonations will interact with the reflection of stress waves. The minimum timing is, therefore,

$$t = 2\ B_e\ /\ C_p \times 10^3 \tag{9.2.1.13}$$

where t is stress wave travel time in ms, B_e is effective burden or distance from the hole to the free face in feet, and C_p is velocity of sound for the rock in fps. The maximum timing is that at which the burden is fully detached and accelerating as gas pressures build. Research by Barker and Fourney (1978, 1978a), Winzer and Ritter (1980), and others, has shown that stress wave travel time is a fraction of the time required to develop radial cracks. Furthermore, studies using high-speed photography indicate that the burden moves within a timeframe which is between 2 to 10 times the wave travel time to the face. Hagen (1977) noted the time to burden movement ranges from 5 to 50 ms, and suggests an optimum range of timing for design between 1.5 to 2.5 ms/ft of B_e.

Timing studies have been performed to investigate resulting fragmentation and muck pile shapes. Reduced-scale research using a variation in delay ratios suggests improved fragmentation for timing between 11 to 17 ms/ft of B_e (Stagg and Nutting, 1987), while Bergmann et al. (1974) demonstrated improved fragmentation for *S/B* ratios of two at timing ratios of 1 ms/ft of B_e or greater.

Production-scale, multiple-row blasting has resulted in recommended timing to improve fragmentation. Andrews (1981) suggests delays of 1 to 5 ms/ft within rows and 2 to 15 ms/ft of B_e between rows (or on the echelon). Anderson et al. (1982) measured flyrock velocity, or gas venting, through the collar stemming to establish a 3.4 ms/ft of hole spacing and 8.4 ms/ft of B_e recommendation for optimum breakage and forward movement. Similar work in which muck pile profiles were

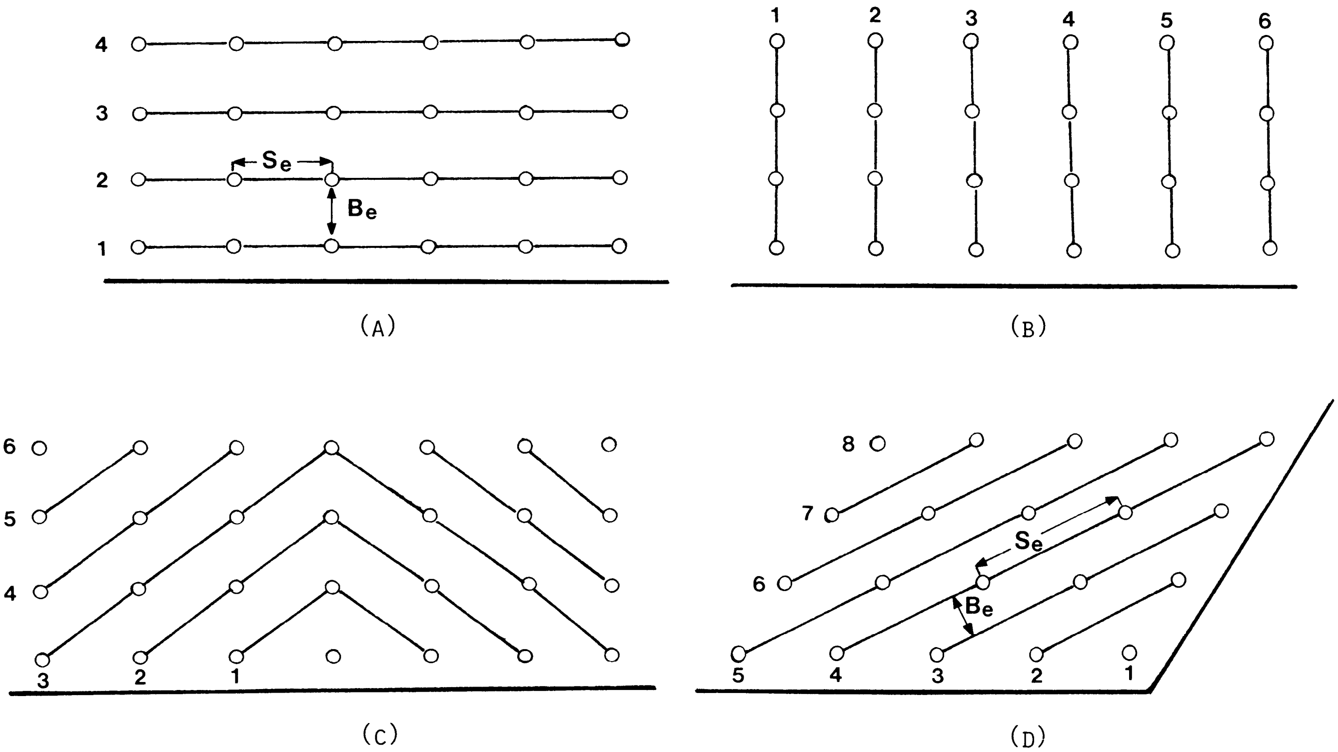

Fig. 9.2.1.11. Typical initiation patterns for surface blasting showing initiation by rows (A) parallel and (B) perpendicular to a single free face, (C) in a "V" configuration and, (D) echelon pattern using two free faces; B_e and S_e are effective burden and spacing, respectively.

mapped indicates that optimum forward throw and muck pile height reduction occur for delay ratios of 4.2 ms/ft of S_e and 10 ms/ft of B_e, while forward throw is minimized, resulting in high muck piles, with ratios of 1.5 to 2 ms/ft of S_e and 5 to 6 ms/ft of B_e (Winzer et al., 1981). Hagen (1977) has shown for single-row production shooting and *S/B* of 1.2 to 1.6 that timing ratios greater than 1.2 ms/ft of B_e are ideal. Hagen recommended 1.2 ms/ft of B_e for multiple-row production blasting in hard rock, while using high powder factors and short stem lengths. A 2.4 ms/ft of B_e was recommended for soft rock with long stem lengths and low powder factors. To control ground vibrations, Kopp (1987) recommended that a 1.3 ms/ft of *S* and 1.2 to 4.3 ms/ft of B_e be used.

The timing ratios cited are found to vary over a wide range. A great deal of research on the effects of initiation timing cannot be compared due to the lack of similar variables such as geology, scale, and explosive type. Winzer et al. (1983) recognized the need to qualify delay ratios, in a general way, based on existing fracture density. Competent dense rock requires lower delay ratios to achieve fine fragmentation, while weak fractured rock fragments best with higher delay ratios.

Underground Blast Design: Blasting rounds are used in the development of tunnels, shafts, raises, stopes, caving, and other underground openings. Powder factors range from 1.5 to over 10 lb/yd^3 (0.9 to 6 kg/m^3). The lower values are used in large open rooms in soft weak rock while the higher values are used in confined raises and shafts for hard competent rock.

Underground explosives should be selected for ease of handling and loading. Explosives with a fume class of 1 are required, and permissibles must be used in gassy mines. Wet and dry blasting agents as well as dynamites are used in a variety of blasting situations. The storage of explosives underground should be made for limited periods of time in the case of atmospheric conditions which present high humidity and temperatures. Cartons must be kept sealed until used, and a rigid system of stock rotation employed. Many loading procedures outlined in the segment on surface blast design also apply to underground loading. During blasted rock removal, a constant watch for unshot explosives must be made. Prior to drilling the subsequent round, the face must be closely inspected for evidence of unshot explosives from the previous round. The holes are then blown with compressed air to clear blockages and remove water from the holes.

For loading short holes using cartridge explosives, the charge is tamped with nonsparking poles. The cartridge containing the initiator should never be tamped, but rather pushed gently into the holes. In recent years, advances in hole loading have been made with the use of mechanized pneumatic and pumping machines. Pneumatic loading uses pressurized airstream flow to inject conventional cartridge as well as dry bulk explosives (Ljung, 1978; Smith, 1982; Russell, 1984; Day and Joyce, 1988). In loading dry bulk ANFO, care must be taken to adjust the pressure regulator such that the tank or line pressure remains below the level recommended by the manufacturer. Static electricity buildup is a problem with this method, and nonelectric or anti-static blasting caps must be used with pneumatically loaded ANFO. The success of loading bulk ANFO and wet blasting agents in upholes, using pressurized air and pumps, has allowed the application of large-diameter holes (6.5 in. or 165 mm) to stoping techniques and the development of the vertical crater retreat (VCR) method of stoping (Bauer, 1978; Lang, 1978).

Priming methods used in underground holes are a matter of choice. Large-diameter holes are primed at the base of the hole.

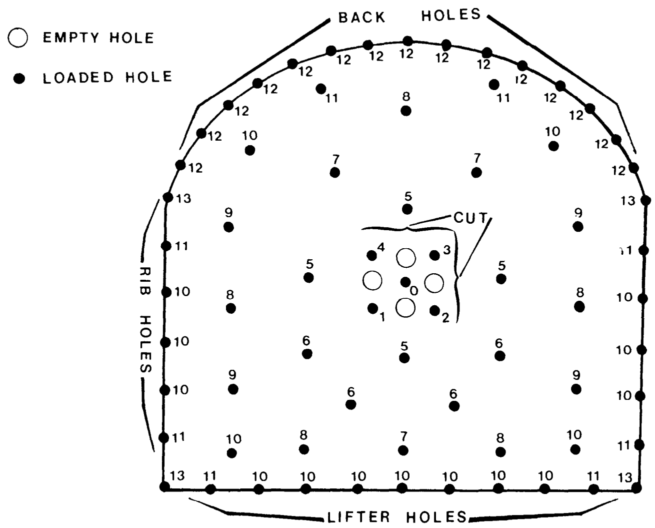

Fig. 9.2.1.12. Typical tunnel round showing drillhole terminology and an initiation sequence for delay timing; all holes between the cut and perimeter holes are referred to as relief holes.

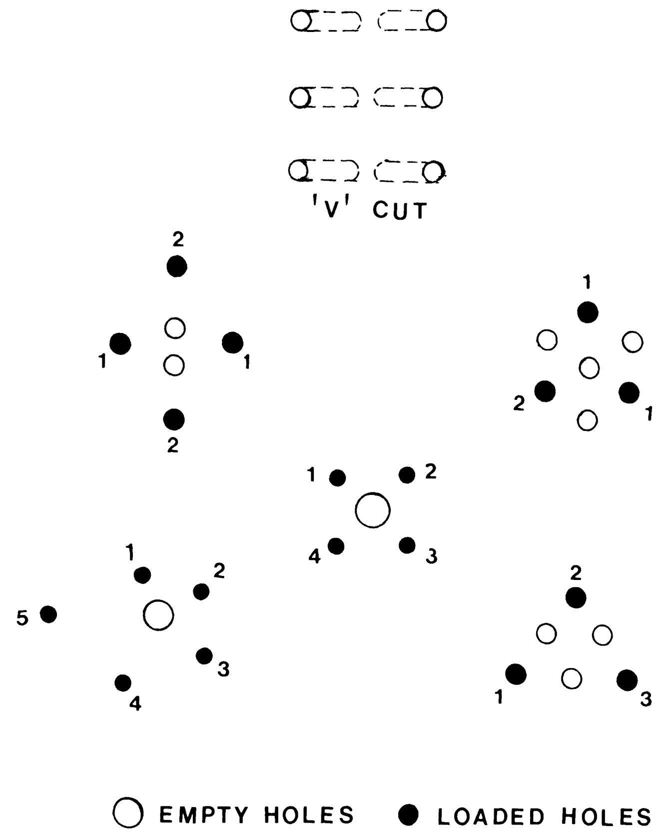

Fig. 9.2.1.13. Example of burn and V cuts used in underground blast designs.

Blasting caps are inserted with the exploding tip facing the line of charge. Small-diameter horizontal holes should be primed at the hole bottom to ensure maximum confinement. If the primer containing the initiator is placed at the hole collar, it could be expelled upon detonation if the stem length is too short. A misfire could result with unshot explosives remaining in the hole. If a misfire occurs, it should be handled as described in the segment on surface blasthole loading.

Post-shot procedures must include a safe waiting time for fumes to disperse from the working area. This time varies based on the explosive and rock type, as well as the ventilation system. Measurements should be taken to establish this safe time.

The design of underground blasting rounds is covered in detail by Langefors and Kihlstrom (1978), Gustafsson (1981), and Dick et al. (1983). Two types of blasting rounds are those with one free face and those with more than one free face. Single-face rounds are used in development openings (tunnels, shafts, raises) as well as in room and pillar, longwall, and shrinkage stoping methods of mining. All mining methods require single-face blast designs for development work. Multiple-face rounds are used in open stopes, sublevel caving, and large-diameter tunnels using benching methods. In many cases, multiple-face rounds are designed similar to surface blasting.

Single-Face Rounds—Names given to the various blastholes drilled within a round are shown in Fig. 9.2.1.12. *Cuts* refer to a group of holes, centrally located at the face and detonated on the first few delay intervals. The purpose of the cut is to provide initial relief to which the remaining holes break. Cuts comprise parallel holes, referred to as the burn cut, or angled holes, defined as V cuts. Various types of burn and V cuts are shown in Fig. 9.2.1.13. In the case of single free faces, the *burden* is defined as the distance, on the face, between each hole and adjacent relief. This relief can be provided by the empty holes in the cut, by the blasted and ejected cut itself, or by holes surrounding the cut as the delayed sequence of holes are initiated.

Langefors and Kihlstrom (1978) provide design criteria for the distance between loaded and empty holes to maximize breakage and advance of the face for the burn cut round. As a general rule, the burden between central empty holes and the nearest small-diameter loaded hole should be 0.7 times the area of the empty holes.

The location of the cut on the face will dictate the direction and distance of throw. In soft rock, such as coal or potash, a saw or cutter chain is used to produce a kerf cut near the floor, roof, or along the midsection of a face to serve as a plane of relief in addition to protecting the wall rock from backbreak.

Borehole patterns are selected based on rock type and size of the face. Drillhole alignment is critical for blasting results. If holes are incorrectly spaced, wallrock damage or poor fragmentation can occur. Bottom-hole deviation is defined as the offset distance measured at the hole bottom divided by the planned hole length and is given in percentage. Hole alignment is a function of hole size, total hole length, direction drilled and geology. Longhole deviations can be approximated by 0.015 times hole length in feet (0.05 times hole length in meters). Vertical, large-diameter down-holes can be drilled with little deviation. Small-diameter holes drilled within a fan or ring result in large deviations. When loading holes, deviations must be considered such that adequate hole bottom powder factors are used. If powder factors are too low, the advance per round, or pull, is poor. If holes are too close together, resulting in high powder factors, dead pressing of explosives may occur leading to bootlegs, or misfires. Sympathetic detonation could also occur, prematurely initiating adjacent holes simultaneously, and results with in-place fragmented rock and no displacement. This rock is referred to as "frozen" in place, and generally leads to no advancement of the face.

Empirical relationships available for spacing of holes and powder factors are given by Langefors and Kihlstrom (1978), Gustafsson (1981), and Holmberg (1982). Approximate relationships between total heading area and total hole area, powder factors, and approximate number of holes required based on geology and hole size are given. Such relationships are based on hard-rock blasting techniques used in Sweden and Austria. They are based on empirical data and cannot be expected to apply to each underground blasting situation in other types of rock.

Wilbur (1982) gives drillhole requirements for US tunnel projects and a generalized range of geology. The relationship for N, the number of blastholes per round based on tunnel area A in square feet, is approximated for soft or highly fractured rock (9.2.1.14) and for hard or massive rock (9.2.1.15) by

$$N = 0.124\ A + 10 \tag{9.2.1.14}$$

$$N = 0.158\ A + 28 \tag{9.2.1.15}$$

Gustafsson provides numerous design examples for relationships developed in Austrian tunnels. Design formulas are based on holes loaded with a nonuniform distribution of explosive charge or loading density. A heavy charge, loaded at the bottom one-third of the hole, and a column charge equal to one-half the bottom charge is used in tunnel blastholes. The burden in feet is equal to (hole depth − 1.3/2), and spacing in feet is 1.1 times the burden. Stemming length is one-half the burden depth.

Such relationships only serve as examples of the procedures to be taken in the design process. For each design situation, a trial-and-error approach is usually taken by experienced and qualified blasters.

Delays used are either short- (ms) or long- (sec) period electric or nonelectric delays. In general, long periods provide coarse fragmentation, and muck pile placement is high and close to the blasted face. These delays are necessary in tight headings such as raises where more time is necessary to displace rock for individual delays. Short-period delays generate finer fragmentation and a long, low muck pile profile.

Delay patterns and approximate hole spacings used for single-face blasting underground are shown in Fig. 9.2.1.14. Blast designs used for sinking shafts are similar to those used for tunnels. For full-face rounds, a burn cut or V cut is used. A bench round or sump cut, shown in Fig. 9.2.1.14, allows for dewatering during development. Powder factors range from 2 to 7 lb/yd^3 (1.20 to 4.2 kg/m^3). Hole diameters average 2 in. (50.8 mm) in tunnels, and diameters of 4 to 6 in. (101.6 to 152.4 mm) are used in stoping techniques.

Raises using short holes less than 120 ft (36.6 m) or long holes greater than 120 ft (36.6 m) are drilled 2 to 6.5 in. (50.8 to 165 mm) in diameter and loaded up or down from a horizontal drift. Drillhole deviations limit raise drilling to less than 148 ft (45 m). Drilling and loading upholes from below can be dangerous and time consuming. Down-hole drilling and loading is safer and provides a high productivity. This is usually done with a central large relief hole or using the vertical crater retreat (VCR) method. The VCR method has been adapted to drilling large-diameter down-holes, providing a safer and more efficient means of advancing raises from the bottom taking advantage of gravity, while equipment and men remain at the top sill.

The shrinkage stope method used for production can be adapted to VCR methods in large diameter (6.5 in. or 165 mm) holes. In this method, down-holes are drilled from an upper drift, plugged near the hole bottom, loaded, and shot using delays. The sequence is repeated several times, advancing the back of the undercut with horizontal slices. Each round is loaded with a charge length to diameter ratio of 6 or less with parallel hole spacing designed 20 times the hole diameter dimension. The advantage of this method used in shrinkage stoping is higher productivity, improved safety, good fragmentation, and cost per ton lower than traditional overhand shrinkage methods.

Multiple-Face Rounds— Underground blasting techniques using multiple faces are shown in Fig. 9.2.1.15. Benching, similar to surface methods, is employed for room and pillar and open stoping methods. Medium- to large-diameter holes, drilled either vertically using subdrilling or horizontally, are used.

Shrinkage stopes are blasted using either the overhand or blasthole shrinkage method. Hand-held jacklegs or stopers are used to drill small-diameter upholes or holes in an inclined back, as well as horizontal holes, using overhand or breasting methods, while working support is provided by the previously blasted muck pile.

Cut and fill mining methods excavate horizontal slices of ore working from the stope bottom upward. In this method, the ore is blasted and removed after each slice, then replaced with waste material usually comprising cemented mill tailings. Hydraulic filling provides a competent, even floor for the drilling of horizontal or vertical holes along the back using mechanized drilling equipment.

Sublevel or blasthole stoping methods include small-diameter hole ring and fan drilling techniques or large-diameter parallel holes drilled the entire stope length. Sublevel stopes are developed with small-diameter holes approximately 2 in. (51 mm) in diameter. Drilling efficiency is limited to 60 to 80 ft (18 to 25 m) for rings. Fan upholes, angled 45° to 88°, are drilled 45 to 65 ft (12 to 20 m) in length. Stope widths range 20 to 150 ft (6 to 46 m), and heights up to 250 ft (76 m) are common. An end pillar raise or a central slot raise is initially blasted from wall to wall to which successive slab rounds break. Fan or ring spacings along the drill drifts, or the burden distance between holes, vary 5 to 10 ft (1.5 to 3 m), and hole bottom, or toe, spacings between holes in a fan or ring range from 10 to 20 ft (3 to 6 m). Hagen (1988) recommends the use of staggered drilling patterns between fans or rings with a spacing 3.5 to 4 times the burden distance. The use of this ratio should minimize sympathetic detonation, improve fragmentation, lower ground vibrations, and provide a good distribution of explosive energy. ANFO or water gels are commonly used with powder factors ranging from 2 to 6 lb/yd^3 (1.2 to 3.6 kg/m^3). Short period electric or nonelectric blasting caps with delay intervals of 25 ms are generally used.

Large-diameter parallel holes 4 to 7 in. (102 to 178 mm) in diameter are often used to blast entire stopes without sublevels to heights of 250 ft (76 m). The vertical limits of the stope are defined by an upper drill drift or top sill and an undercut drift. The upper sill is developed to the width of the ore body. Down-the-hole (DTH) drilling equipment is used to drill down-holes 150 to 200 ft (45 to 65 m) in length. Two methods of production blasting include vertical slabbing to an open stope and crater blasting or VCR. Panel blasting requires a raise and slot to be developed in addition to an undercut fan beneath each slab round. A 0.32-lb/ton (0.15-kg/t) powder factor is typically used. The application of crater blasting in stope mining requires a minimum of development within the ore pillar. Its use depends on the size of the ore body and the stability of wallrock. The VCR method has been shown to reduce pillar damage and overbreak, resulting in less ore dilution. Drilling and blasting cycles were described previously under single-face shrinkage stopes. Spacings and burdens range 8 to 10 ft (2.4 to 3 m) square or staggered on a 7 × 9-ft (2.1 × 2.7-m) pattern for a typical drillhole diameter of 6.5 in. (165 mm). Charge weights per hole using ANFO or water gels vary from 10 to 20 lb (4.5 to 9 kg), and a vertical advance of 10 to 15 ft (3.0 to 4.6 m) is typical. A major drawback with the use of this method is the difficulty in keeping

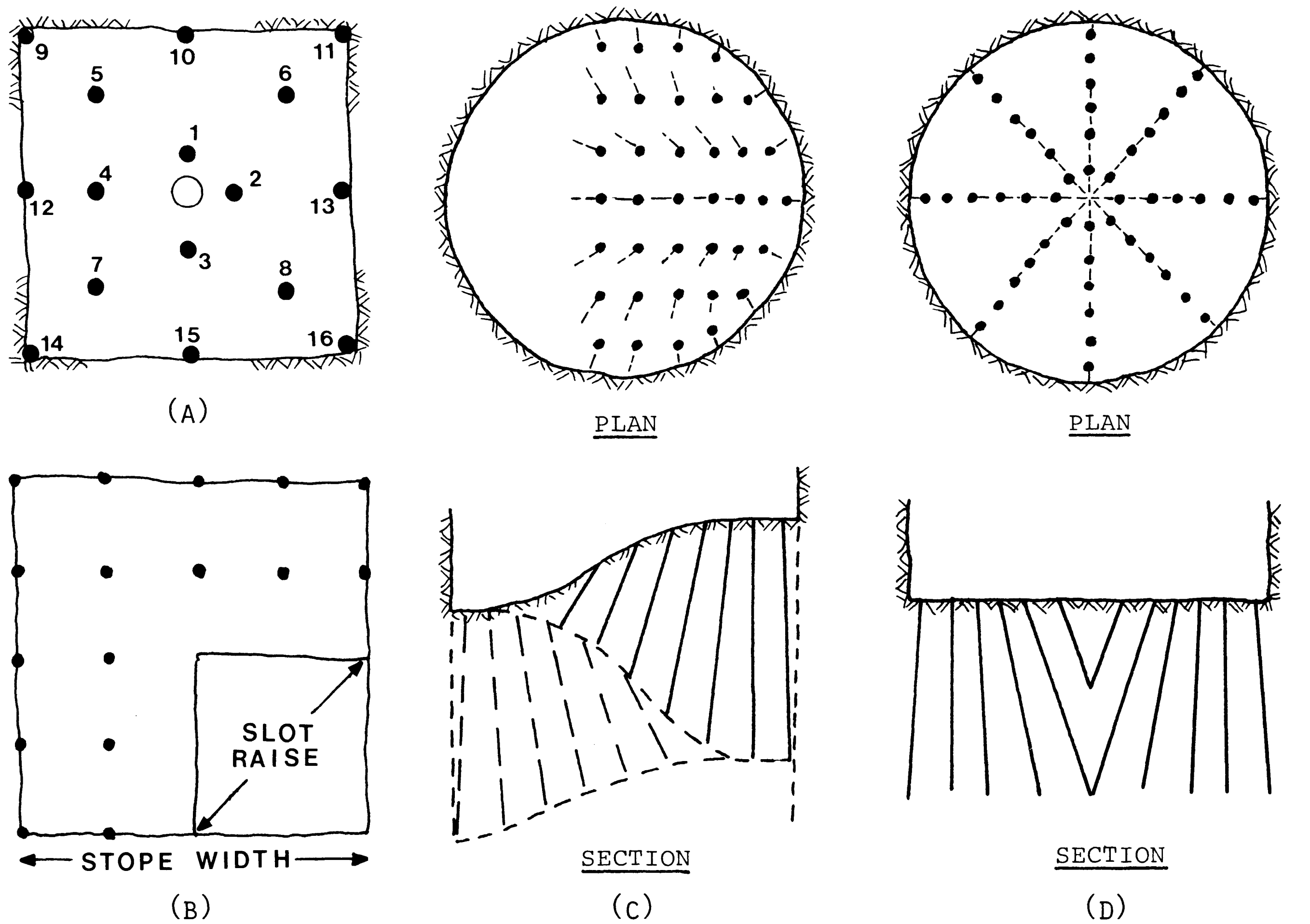

Fig. 9.2.1.14. Single face drilling patterns for (A) a long-hole raise, (B) drop raise for blasthole stopes (note: slot raise pattern similar to long-hole raise design), (C) shaft bench round, and (D) full face V shaft round.

holes open during blasting cycles in fractured and faulted ground.

Block and sublevel caving techniques are used at depths below which stoping methods become unstable or used for low-grade, fractured ore bodies. The sublevel caving method employs the use of fan drilling using long, small-diameter holes, between sublevels or drill drifts to undercut and blast the ore zone. An initial slot is developed at the wallrock, and vertical uphole fans are drilled in a diamond pattern from sublevels in sequence. Generally, eight holes, inclined 85° toward the slot, are drilled. Hole diameters average 2 in. (51 mm). Burdens and spacings vary from 4 to 6 ft (1.2 to 1.8 m) and 5 to 6 ft (1.5 to 1.8 m), respectively. Blasting is performed against broken waste rock as the wall rock caves. Powder factors range from 0.6 to 0.9 lb/ton (0.3 to 0.4 kg/t).

Block caving techniques require an initial development blast above the undercut level to start caving. Jumbos with 2- to 3-in. (51- to 76-mm) diameter holes drill fan rounds, oriented 45° to vertical and 15 to 38 ft (4.6 to 11.6 m) in length. Fan spacing is generally 5 ft (1.5 m). Secondary blasting is often required to dislodge oversize material or broken muck that has hung up within the raises. Bagged water gels are used in raises at a powder factor of 0.2 lb/ton (0.09 kg/t).

Controlled Blasting Techniques: *Controlled blasting* employs the use of reduced explosive quantities loaded into holes that are generally smaller in diameter and spaced closer than the main blast. The holes are often placed along the periphery of an excavation or round. This method is used to control overbreak, reduce fractures within remaining rock walls, and reduce ground vibrations. Such methods are variations of line drilling in which closely spaced, unloaded holes form a natural excavation line beyond which no rock is to be blasted. In general, hole diameters range from 2 to 3 in. (51 to 76 mm), and spacings range from 0.3 to 1 ft (0.1 to 0.3 m).

Two versions of line drilling used in surface blasting are presplitting and cushion blasting. *Presplitting* (or *preshearing)* uses a line of closely spaced loaded holes in the range of 2 to 4 in. (51 to 101.6 mm) in diameter and 2 to 4 ft (0.61 to 1.22 m) is spacing. They are drilled along the periphery of an excavation and initiated before the main blast is detonated. In this respect, an internal free face is formed, containing stress waves from successively detonated holes within the boundaries. Explosive charges range from 0.1 to 0.7 lb/ft (0.06 to 0.42 kg/m). Presplitting is often employed with trench blasting techniques.

Cushion blasting (also referred to as *smooth-wall* or *trim blasting)* is performed using hole sizes of 2 to 6 in. (51 to 152 mm) and loading from 0.1 to 1.5 lb/ft (0.06 to 0.89 kg/m). Hole spacings are generally set between 3 to 9 ft (0.9 to 2.74 m). In this method, explosive charges in cartridge form are decoupled

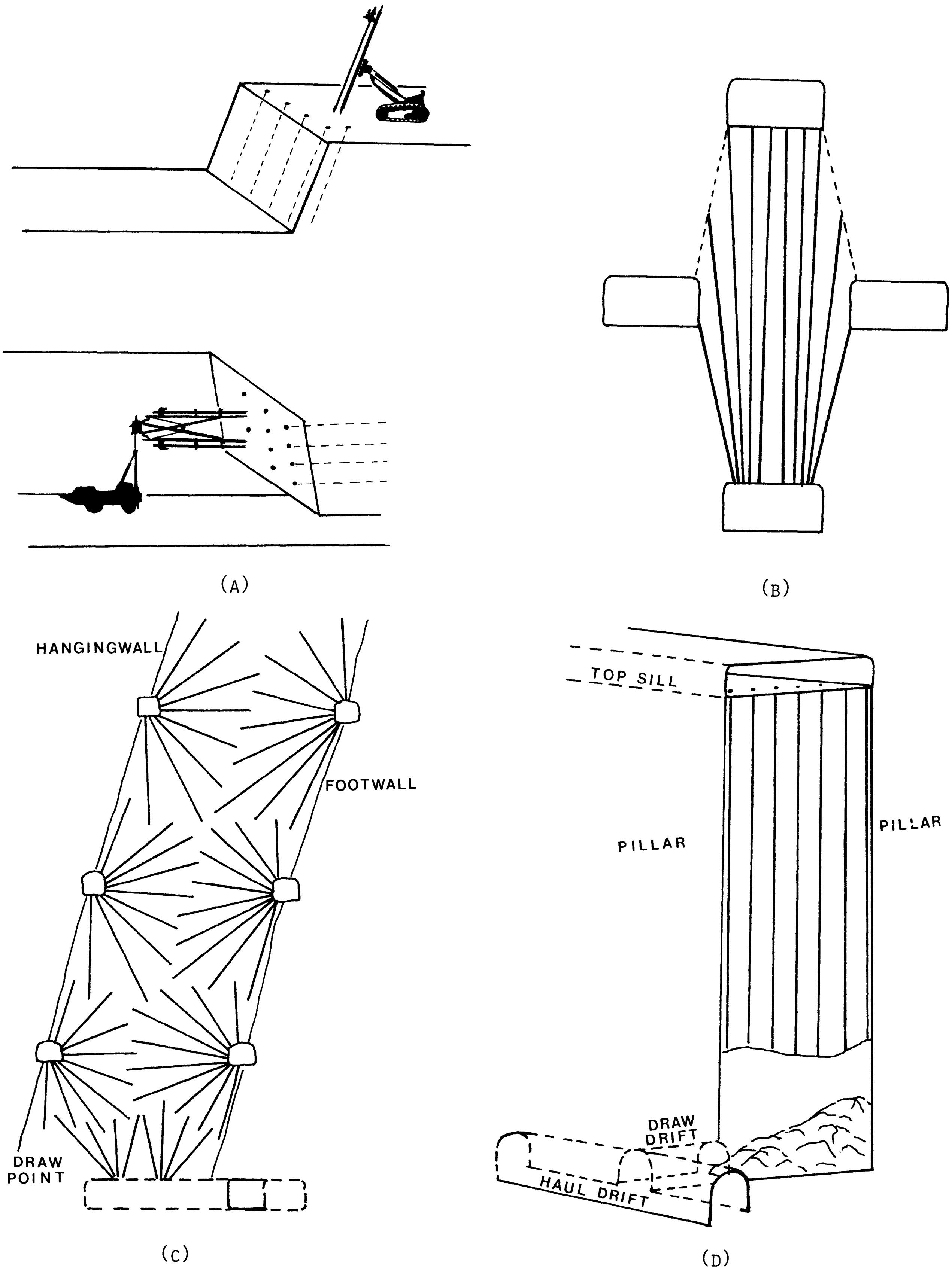

Fig. 9.2.1.15. Multiple face drilling patterns for (A) room and pillar, cut and fill, and shrinkage stoping methods; (B) sublevel caving; (C) sublevel stoping; and (D) long-hole stoping or vertical crater retreat methods.

from the borehole wall, intermittently spaced along the borehole length.

Smooth or trim blasting is the underground application of cushion blasting. For large-diameter blasting underground, an effective method of protecting pillars and stope walls is the decoupling of charges with air. Often a mixture of 50% ANFO and 50% polystyrene beads is used to lower the explosive energy in prefractured rock and for pillar protection. The use of spacers or plugs is used to distribute the charges, lowering the powder factor and eliminating the chance for charges to prematurely initiate adjacent holes designed for later initiation timing. Often the charge is placed in cardboard liners one-half the diameter of the borehole, filling the annular area with drill cuttings. Trim blastholes are shot after the main blast round is detonated.

9.2.1.4 Estimating Drilling and Blasting Costs

Drilling and blasting are unit operations required for development and production. The components of costs for drilling and blasting include labor, direct costs of operating equipment, and supplies. In surface mines, the basis of cost is computed per ton (tonne) of ore produced or per cubic yard (cubic meter) of material broken for removal. The costs are directly related to powder factor and depend on geology, type of explosives, and the size of the blastholes and excavating equipment. Underground costs are a function of mining method, level of mechanization, productivity, and operational cycle time. Development costs are given per foot (meter) of advance for tunnels and raises and are expensed over the ore block associated with development. Production drilling and blasting is computed per foot (meter) of hole drilled or per ton (tonne) of ore produced.

For equipment, such as a drill rig, the costs can be summarized with the following relationship:

$$\text{Drill cost, \$/ft} = \frac{(C_{own} + C_{op})\,BL\,(\text{hr}) + BC}{BL\,(\text{ft})} \qquad (9.2.1.16)$$

where C_{own} is cost per hour to own the rig, C_{op} is cost per hour to operate the rig, BL is bit life in hours or in feet, and BC is bit cost. The cost to own includes taxes, interest, insurance, amortization, and depreciation. The cost to operate includes labor, fuel, and parts and supplies, such as tires and drill steel. Labor costs, also applied to blasting costs, include base salary plus benefits. Benefits, which range from 30 to 40% of base salary, include insurance, health care, pension, and vacation. Incentive pay, as a percentage of base salary, is often provided when productivity increases over a predetermined average. Productivity is measured as feet (meters) drilled for the drill crew or loaded and shot per employee-shift for the blasting crew. Incentive pay is also provided for accurate underground drillhole alignment and depth.

Blasting costs comprise explosives, boosters and primers, initiation systems, and other expendables. Labor costs include the hours spent by the blasting crew to handle and transport explosives, load holes, detonate the shot, and take inventory and prepare paperwork. The cost of bulk loading and storage equipment is also included.

A comparison of drilling and blasting costs for various mining methods is shown in Table 9.2.1.8. Blasting costs are directly related to powder factor and the cost per pound of the main explosive charge. Labor costs can represent 5 to 40% of the total blasting costs, while the cost of expendable blasting accessories such as primers and initiators is generally less than 20% of total costs. Example cost calculations are given for the surface blast designs in Exs. 9.2.1.10 and 9.2.1.11.

Table 9.2.1.8. Comparison of Drilling and Blasting Costs for Various Mining Methods

Extraction Methods	Cost, $/yd^3
Metal mines and quarries	0.08–2.00
Construction	0.03–0.35
Tunneling	0.80–2.00
Longhole stoping	0.50–1.20
Vertical crater retreat mining	0.45–0.90
Cut and fill stoping	0.60–1.00
Shrinkage stoping	0.50–0.75

Conversion factor: $1 /yd^3 = $1.31 /m^3.
Source: Aimone, 1979.

Example 9.2.12. Using single-column loading from Ex. 9.2.1.10, estimate the drilling and blasting costs. The following cost data are used:

blasting:	ANFO	$ 0.12/lb	drilling:	$1.30/ft
	primer	1.50/ea		
	caps	1.30/ea		
	labor	16.00/hr		

A three-person blasting crew is assumed, and hourly wages include benefits. The unit drilling cost includes all costs to own and operate the rig, computed using Eq. 9.2.1.16.

Production data:

weekly production	150,000 tons
tonnage factor	2.4 tons/yd^3
yield/hole	1280 yd^3
holes/week required	50
yield/week	64,000 yd^3 or 153,600 tons

Solution.

1. Blasting cost

Explosives per hole:

ANFO 1114 lb @ $0.12/lb	$133.68	
Primer and cap	2.80	
Total per hole	$136.48	
Total per week, 50 × $136.48		$6824
Labor, 8 hr, 3 miners @ $ 16/hr		384
Total blasting per week		$7208
Total per yd^3		$0.113

2. Drilling cost

50 holes, 57 ft per hole @ $1.30/ft	$3705
Total per yd^3	$0.058

Example 9.2.1.13. Using three-deck loading from Ex. 9.2.1.11, estimate the drilling and blasting costs.

Production data:

yield per hole	1050 yd^3
holes per week	60
yield per week	63,000 yd^3 or 151,200 tons

Solution.

1. Blasting Cost

Explosives per hole:

ANFO, 900 lb @ $ 0.12/lb	$108.00	
3 primers and caps	8.40	
Total per hole	$116.40	
Total per week, 60 × $116.40		$6984

Labor, 9 hr, 3 miner, @ $ 16/hr	432
Total blasting per week	$7416
Total per yd^3	$0.118
2. Drilling cost	
60 holes, 57 ft per hole @ $1.30/ft	$4446
Total per yd^3	$0.071

Cost Comparison for Increase in Explosive Energy: The following example gives a cost comparison for two explosives of different energy levels. A design for explosive A, with a density of 0.85 and per-pound cost of $0.12, is compared with a design for explosive B, whose density is 1.3 and cost $0.20/lb. The design involves 4-in. (101.6-mm) diameter holes for a 150,000-yd^3 (114,690-m^3) excavation using 28-ft (8.53-m) long holes, 8-ft (2.44-m) collar stem, and 4-ft (1.22-m) subdrilling. The excavation subgrade is 24 ft (7.3 m) below current surface. Primer and cap costs are $2.80, while it is assumed that blasting labor is $4.00/hole, and drilling costs are $ 0.90/ft ($2.95/m).

Example 9.2.14. Cost comparison.

	Explosive A	**Explosive B**
explosive density	0.85	1.30
loading density	4.63 lb/ft	7.08 lb/ft
charge weight/hole	93 lb	142 lb
total blasting cost/hole	$17.96	$35.20
pattern ($B \times S$)	10 × 13 ft	12 × 17 ft
yield per hole	116 yd^3	181 yd^3
number of holes required	1293	829
powder factor	0.80 lb/yd^3	0.78 lb/yd^3
blasting cost	$0.155/yd^3	$0.194/yd^3
drilling cost	$0.217/yd^3	$0.139/yd^3
total costs	$0.372/yd^3	$0.334/yd^3

The higher cost of blasting with the higher-density explosive is offset with a lower drilling cost. With a higher-energy explosive, an expanded pattern is drilled with fewer holes. The overall cost savings is $5830.

Cost Comparison of Two Stoping Blasting Methods: Example 9.2.1.15 provides a cost comparison between conventional sublevel stope blasting (longhole stoping) and cratering techniques. Production costs are considered and development costs are not included.

Example 9.2.15. Cost comparison.

Production ore stopes are 250 ft (76 m) high, 420 ft (128 m) in length, and 35 ft (11 m) in width. Production is 1000 tpd (900 t/day), and stope production is 326,700 tons (296.4 kt). In longhole stoping, 2-in. (51-mm) drillholes are used to drill rings from sublevels with approximately 60 ft (18 m) of vertical separation. Eight holes are drilled per ring with an average hole depth of 35 ft (11 m). The crater method (VCR) uses 6.5-in. (165-mm) holes drilled vertically from the upper drill drift. A total labor cost of $18.75/hr or $150/employee-shift, is used. Drilling and blasting activities take place over 6.5 hr of the 8-hr shift.

Longhole stoping costs:

Drillhole size	2 in.
Explosive density	1.03
Explosive loading density	1.4 lb/ft
Powder factor	0.7 lb/ton
Explosive cost	$0.12/lb

Solution.

Blasting yield	
(1.4 lb/ft)/(0.7 lb/ton)	= 2 tons ore/ft of drillhole
Total stope drilling required	
326,700 tons/(2 tons/ft)	= 163,350 ft
Productivity and costs	
Drilling labor	
(35 ft/hr drill rate)(6.5 hr)	= 228 ft/employee-shift
($150/employee-shift)(228 ft/employee-shift)	= $0.66/ft
Drilling supplies and nonlabor costs	= $0.10/ft
Total drilling costs	= $0.76/ft = $0.38/ton
Blasting	
Explosives, (0.7 lb/ton)($0.12/lb)	= $0.084/ton
Primer and initiator	= $0.035/ton
Total blasting supplies	= $0.119/ton = $0.238/ft
Labor productivity	
(35 ft/hole)/(0.25 hr/hole loading time)	= 140 ft/employee-hr
(140 ft/employee-hr)(4 hr loading/shift)	= 560 ft/employee-shift
Labor cost	
($150/employee-shift)/(560 ft/employee-shift)	= 0.27/ft = $0.14/ton
Total blasting costs	= $0.51/ft = $0.25/ton
Total drilling and blasting costs	= $0.634/ton

VCR stoping costs:

Blasthole pattern	9 × 9 ft
Total VCR holes/stope	182
VCR drillhole length	220 ft
Total crater hole drilling	40,040 ft
VCR production	287,500 tons
Undercut development	39,200 tons
Explosive density	1.35
Explosive loading density	19.42 lb/ft
Explosive cost	$0.20/lb
Loading length-to-diameter	6
Charge weight per slab round	60 lb

Based on preliminary tests, a powder factor of 0.83 lb/ton (0.5 g/kg) is adequate to break 72 tons (65 t) for a 10-ft (3-m) retreat along the back. Total slab tonnage is 13,104 (11.9 kt). Rotary drilling cost, including labor, is $0.60/ft.

VCR blasting per 10-ft advance		
explosives		
(182 holes)(60 lb/hole)($0.20/lb)	$2184	
primer and initiator		
(182 holes)($2.50/hole)	455	
labor		
(54.6hr)(2 miners)($18.75/hr)	2048	
Total blasting costs	$2.57/ft	= $0.36/ton
VCR drilling per 10-ft advance		
($0.60/ft)(10 ft/hole)(182 holes)	$1092	= $0.083/ton
Total VCR drilling and blasting costs		= $0.443/ton

The costs to drill and blast the 39,200-ton (35.6 kt) undercut must be included. This is assumed to be a total cost of $0.634/ton as calculated for the smaller diameter hole. Thus the entire stope drilling and blasting cost is $0.466/ton. The application of VCR to stoping techniques for this example results in a 26% reduction in drilling and blasting costs over conventional long-hole stoping.

9.2.1.5 Research in Explosives Applications

Within the past two decades, advances have been made to improve the efficiency of explosives and rock blasting. These advances have been, in part, largely due to analytical and computational techniques of high-speed photography and computer modeling. These methods have brought about a greater understanding of the distribution of explosive energy during the rock fracturing process. Other areas of advancement include initiators and explosives formulation.

Much attention has been given to modeling the explosive fracture and fragmentation processes (Cunningham, 1987; Danell and Leung, 1987; Kuszmaul, 1987; Kirby et al., 1987; Paine et al., 1987; Exadaktylos et al., 1987; Crum and Stagg, 1989) and to modeling explosive detonations (Leiper and Plessis, 1987). With fracture and fragmentation modeling, the effects of variations in design parameters on blasting results are readily obtained without costly field trial and error attempts. The output from many codes show particle size distributions, radial fracture formation, muck pile profiles, and stress profiles or damage functions, plotted along a borehole axis for a hypothetical blast. The blast may include charges placed within a two- or three-dimensional rock mass. Thermodynamic characteristics of the explosives and time delays are also modeled. The usefulness of these models are, however, limited as they cannot completely take into account the variability of geology or hole-loading conditions in practice. Output from computer models must be field tested for validation.

Explosive performance, particularly in the areas of wet blasting agents and heavy ANFOs, has been extensively researched (Bauer et al., 1984; Lee, 1987; Van Ommeren, 1989). Explosive manufacturers are investigating ANFO/emulsion blends to increase shelf life and the number of times the mix can be repumped. One area of ongoing research is the formulation of a water-based replacement for dynamite or a non-nitroglycerin explosives with the energy of dynamite. Improvements in the performance of AN prills, with changes in particle sizes and the selective penetration of liquids, is being investigated.

High-speed photography (Anon., 1983a; Chiappetta et al., 1983), used to evaluate blast designs, initiator timing, and explosive performance, has led to new product development and a better understanding of blasting theory. Cinematography has been used to assist in the selection of optimum hole loading, stemming, spacings, and burdens for control of throw and muck pile placement. Optimum delay sequencing can be identified from photographic work, both underground and on the surface. Surface-cast blasting, a method used primarily in surface-coal mining, was perfected using high-speed photography (Burleson, 1988; Guiliani and Otuonye, 1989). Cast blasting, using angled or vertical holes in a limited number of rows, is performed with high-gas producing explosives, to "cast" the burden rock. In the case of blasting overburden wasterock, the material is thrown to the final spoil area to minimize equipment handling. In theory, the cost tradeoff between increased blasting costs and decreased material handling costs lowers overall operating costs.

The concept of air decking applied to surface blasting techniques has been made with the use of high-speed photography (Bussey and Borg, 1988). Air decking, developed 50 years ago, involves the use of a small, concentrated charge at the hole bottom and a rigid plug near the hole collar to hold a length of stemming. Between the plug and charge is a length of unloaded hole, or air deck. Its application to blast design greatly enhances fragmentation and controlled fracturing near the collar and throughout the hole length. Upon detonation, the shock wave generated at the hole bottom travels through the air deck, reflecting at the stem base, intensifying the shock energy within the rock at the hole collar with a minimum (limited) explosive charge. The air deck is effective in modifying the pressure pulse shape within this region by lowering the stress wave amplitude; however, the pulse is increased in duration. The stress wave amplitude remains at the level that promotes tensile fracturing. It is the increase in pulse duration or the time application of this tensile pulse that enhances fragmentation. Currently, the method is employed successfully for surface presplitting.

Two major advances have occurred in the area of initiators. These are the developments of precise pryotechnical delays for electric blasting caps and electronic integrated circuit delays. Timing errors are inherent to the design and manufacture of all standard electric blasting caps (Winzer, 1978). Such errors become critical when accurate timing is required for fragmentation and the control of ground vibrations. Precise pyrotechnical delay caps are now manufactured to provide an increase in accuracy on the order of milliseconds; however, they cost slightly more than standard caps.

Programmable blasting caps are being developed with millisecond accuracy (Worsey and Tyler, 1983; Wilson et al., 1987; Larsson et al., 1988). The caps comprise an integrated circuit, using a microprocessor to receive and distribute programmed information on detonation time, current required for detonation, and other coding to prevent access by unauthorized personnel. All forms of electrical hazards are eliminated with these caps, as a unique rate of current must be supplied to the caps before detonation can occur.

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9.2.2 MONITORING AND CONTROL OF BLAST EFFECTS

CHARLES H. DOWDING

9.2.2.1 Introduction

Monitoring and control of blast effects near critical rock masses or constructed facilities depend upon two main considerations. First, shot designs must reduce the amount of explosives detonated at any instant and adjust the initiation sequence to reduce resulting ground and airborne disturbances. Second, the amount of explosives detonated per volume of rock and the shot pattern must be adjusted to ensure adequate fragmentation. Therefore, at the same time, the initiation sequence must be separated in time but not in space.

There is an optimum design which achieves both objectives of control of disturbances and production of adequate fragmentation. This optimum can be reached only through an understanding of the physics of rock mass and structural response to blast disturbance and the interaction between rock fragmentation and shot design. This segment summarizes the state of the art in vibration measurement and structural response to facilitate such an optimum blast design.

Furthermore, it is an effort to transfer advances in earthquake engineering and nuclear blast protective design to blast vibration monitoring and control while summarizing the most recent experimental observations of mining-induced ground motions and structural response. It is hoped that such a transfer and summary of the state of the art will mitigate several recent trends.

For one, there has been a general downward trend in regulatory limits on allowable blast-induced vibrations. In addition to new observations, this drift, in part, can be attributed to the tendency to take the limit of the last study and divide it in half "to be safe." Unfortunately, too many studies whose limits were divided were themselves only summaries of past work that had also divided past limits. The discussion here presents the background for original experiments conducted to determine safe blasting controls and therefore will allow the reader to set appropriate limits based upon the original past work within the framework of existing regulations.

Another trend is the misapplication of peak particle velocity limits that were determined for cosmetic cracking of residential structures. These limits have been applied to tunnel liners, radio towers, slabs-on-grade, and curing concrete. This segment draws attention to studies made to determine limits specifically for these and other cases. Where no studies exist, it presents methods based upon response spectra or ground strains that allow setting of appropriate criteria or limits.

Frequency of vibration and ground strain form the foundation for the presentation. The importance of frequency cannot be over estimated, as it is as critical as peak particle velocity in determining the response of aboveground structures. For below ground structures, frequency, in combination with propagation velocity, controls response. In both cases, cracking results from induced strains, where particle velocity is employed as an index of the strain level.

In addition, computerized monitoring instrumentation is described. Such computerization simultaneously increases monitoring efficiency as well as decreases costs, both original capital costs as well as those associated with record keeping. The latter labor-saving efficiency associated with automated record keeping continues to be undervalued by many mining operations.

9.2.2.2 Range of Blast Effects

Blast effects on surrounding earth materials and structures can be divided into the permanent and transient displacements. While the focus of this chapter is the transient displacements, effects of permanent displacements are presented as they are associated with significant transient effects at relatively small distances.

Permanent Degradation and Displacement of Adjacent Rock: Permanent effects, with the exception of fly rock, are only described within a few hundreds of feet (meters), and can be divided into degradation and displacement.

Degradation —Degradation is normally described by cracking intensity. Such blast-induced cracking has been observed experimentally to vary with hole diameter and rock type (Siskind and Fumanti, 1974; Holmberg and Persson, 1978). Small-hole-diameter construction blasting has induced cracking at distances of 3½ to 7 ft (1 to 2 m), and larger-hole-diameter mining blasts are capable of producing cracks at distances of 35 to 70 ft (10 to 20 m). Careful blast design can dramatically reduce these maximum distances.

Displacement— Displacement can be produced by either delayed gas pressures (those that accumulate during detonation) or by vibration-induced shaking. Delayed gas pressure have dislocated blocks as large as 1300 yd^3 (1000 m^3) during construction blasting (Dowding, 1985). Such movement is unusual, but is associated with isolated blocks, leakage of gas pressures along open joints, and poor shot design with large burdens. Vibratory or shaking-induced displacement is normally associated with unstable blocks in rock slopes and can occur wherever static factors of safety are low and ground motions produce permanent displacements that are greater than the first-order asperity wave length (Dowding and Gilbert, 1988). Gas pressure related displacement can occur out to several hundred feet (meters).

Fly Rock—Fly rock is a special case of permanent displacement of rock by explosive expulsion from the top of the blasthole. Rock has been observed to have been propelled as far as 330 to 3300 ft (100 to 1000 m) (Roth, 1979). Statistical studies have shown that the probability of these extreme events is quite low under normal circumstances, 1 in 10,000,000 at 2000 ft (600 m) (Lundborg, 1981). Since the probability increases with decreasing distance, blasting mats are required in any construction blasting in an urban environment to prevent all fly rock.

Soil Densification or Compaction—Another special case of permanent displacement is the vibratory densification of a nearby mass of loose, clean sand. The propensity for such densification is a function of the soil's density, mineralogy, and grain size distribution. Soils that are densifiable are loose sands, with less than 5% silt-size particles. These clean sands were densified out to distances of 70 ft (20 m) (Ivanov, 1967) after detonation of single, 11-lb (5-kg) charges within the loose sand mass itself. Soils that are either slightly cemented or contain more than 5% fines are a great deal less subject to vibratory densification from typical ground motions.

Transient Structural Response: Transient effects result from the vibratory nature of the ground and airborne disturbances that propagate outward from a blast. In this discussion, it is assumed that no permanent displacements are produced. Thus the only effects are those associated with the vibratory response of facilities in or on the rock or soil mass surrounding the blast. Transient means that the peak displacement is only temporary, lasts less than one-hundredth of a second, and the structure returns to its original position afterwards.

Transient structural effects can be arranged to reflect the expected distance from a blast. Beginning with the closest, transient effects are structural distortion, faulted or displaced cracks, falling objects, cosmetic cracking of wall coverings, excessive instrument and machinery response, human response, and micro disturbance.

The first four effects, those that relate to structural response, are normally grouped together for experimental observation as structural response, and do not normally occur when vibration levels are regulated to prevent cosmetic cracking.

Excessive structural response has been separated into three categories arranged below in the order of declining severity and increasing distance of occurrence (Northwood et al., 1963; Siskind et al., 1980b). Beginning with effects that occur closest to the blast, the categories are listed here.

1. MAJOR (Permanent Distortion). Resulting in serious weakening of the structure (e.g., large cracks or shifting of foundations or bearing walls, major settlement resulting in distortion or weakening of the superstructure, walls out of plumb).
2. MINOR (Displaced Cracks). Surficial, not affecting the strength of the structures (e.g., broken windows, loosened or fallen plaster), hairline cracks in masonry.
3. THRESHOLD (Cosmetic Cracking). Opening of old cracks and formation of new plaster cracks, dislodging of loose objects (e.g., loose bricks in chimneys).

These specific definitions of response should not be described collectively as "damage." To do so blurs the distinction between threshold or cosmetic cracking and major response or structural distress.

Regulation to Prevent Cosmetic Cracking of Residential Structures—Regulatory controls in North America are based on the occurrence of threshold cracking of plaster and gypsum wall board in residential structures (Siskind et al., 1980; Dowding, 1985). Observed cracking is cosmetic in nature and does not affect structural stability. These cosmetic cracks are hair-sized and are similar to cracks that occur during the natural aging of structures. In fact they are indistinguishable from those that result from natural aging. Control limits are based upon direct observations of test homes immediately before and immediately after blast events, to avoid confusion with the similar cracks that might occur from natural processes. These controls do not apply to engineered structures that are constructed of steel and concrete, buried structures, or adjacent rock.

Distinction of Blast-induced Cracking from Natural Cracking: Control of blast-induced transient effects to prevent threshold or cosmetic cracking reduces blast-induced displacement or strains in structures to or below that caused by every day human activities and changes in the weather (Stagg et al., 1984; Dowding, 1988). These cosmetic cracks in many cases are smaller than cracks caused by other natural or occupant initiated processes that are active in all constructed facilities. Thus blast-induced threshold cracks can be scientifically observed only with visual inspection immediately before and after each blast. Observations made under less stringently controlled conditions have little scientific merit because of the high probability of environmentally produced cracks occurring between or before visual inspections.

Multiple Origins of Cracks—Several institutional references (Anon., 1977; Anon., 1956; Thoenen and Windes, 1942) present excellent summaries of the multiple origins of cracks. Basically, cracks are found to be caused by the following:

1. Differential thermal expansion.
2. Structural overloading.
3. Chemical changes in mortar, bricks, plaster, and stucco.
4. Shrinkage and swelling of wood.
5. Fatigue and aging of wall coverings.

Table 9.2.2.1. Comparison of Strain Levels Induced by Household Activities, Daily Environmental Changes, and Blasting

Loading Phenomena	Site[a]	Microstrain Induced by Phenomena, μin/in.	Corresponding Blast Level[b]	
			in./sec	mm/s
Daily environmental changes	K_1	149	1.2	30.0
	K_2	385	3.0	76.0
Household activities				
Walking	S_2	9.1	0.03	0.8
Heel drops	S_2	16.0	0.03	0.8
Jumping	S_2	37.3	0.28	7.1
Door slams	S_1	48.8	0.50	12.7
Pounding nails	$S_{1,2}$	88.7	0.88	22.4

Source: Stagg et al., 1984

[a]K_1 and K_2 were placed across a tape joint between two sheets of gypsum wallboard.

[b]Blast equivalent based on envelope line of strain vs. ground vibration.

6. Differential foundation settlement.

Overtime, all of the causes listed are likely to crack walls, whether or not blasting occurs.

There are three important implications associated with the list above. Structures expand and contract preferentially along existing weaknesses (cracks). Seasonal expansion and contraction along these cracks will return patching and repainting to the original cracked state within several years. This persistent cracking is annoying to those owners who are unaware of the difficulty of patching existing cracks of any kind. Second, the distortion that caused the cracking also creates stress concentrations which may lower a wall-covering's resistance to vibration cracking; however, current regulatory limits already implicitly include these distortion effects as explained in 9.2.2.6. Third, these natural cracks continue to occur over time. Therefore, any postblast inspection at low vibration levels is likely to find new cracks from natural aging unless preblast inspection is conducted immediately before the blast.

Response of Structures to Everyday Activities: A comparison of strains produced by blast vibrations and everyday events with those needed to fail wall-covering materials gives perspective to the observation of cracking at low particle velocities. Table 9.2.2.1 compares strains from daily environmental changes (temperature and humidity) and household activities measured in the US Bureau of Mines test house (Stagg et al., 1984). The door was slammed adjacent to the wall on which the strains were measured.

It appears that in the course of daily life, an active family will produce strains in walls similar to those produced by blasting vibrations of 0.1 to 0.5 ips (2.5 to 12 mm/s). Most astonishing are the measurements in a wood-framed home of relatively enormous strains from daily changes in temperature and humidity. These alone are large enough to crack plaster.

Blast-induced Air Overpressures: Blast-induced air overpressures are the air pressure waves generated by explosions. The higher-frequency portion of the pressure wave is audible and is the sound that accompanies a blast; the lower-frequency portion is not audible, but excites structures and in turn causes a secondary and audible rattle within a structure.

Overpressure waves are of interest for three reasons. First, the audible portion produces direct noise. Second, the inaudible portion by itself or in combination with ground motion can produce structural motions that in turn produce noise. Third, they may crack windows; however, air-blast pressure alone would have to be unusually high for such cracking. Previous researchers (Kamperman and Nicholson, 1970; Borsky, 1965) have found that response noise within a structure (from blasting and sonic booms respectively) is the source of many complaints. It appears that structure and wall motions, which are induced by airblasts and sonic booms, rattle loose objects within the structure, which then startle the occupants.

Human Response: Humans are quite sensitive to motion and noise that accompany blast-induced ground and airborne disturbances. Therefore, human response is significant in the reporting of blast-induced cracking. Motion and noise from blasting can be startling and lead to a search for some physical manifestation of the startling phenomena. Many times, a previously unnoticed crack provides such confirmation of the event. Furthermore, if a person is worried and observes a crack that was not noticed before, the crack's perceived significance increases over one noticed in the absence of any startling activity. These concerns are real and in the mind of the observer are sincere.

In typical mining situations, significant blast-induced inaudible air overpressure and audible noise immediately follows the ground motion and intensifies human response. Both the ground and airborne disturbances excite walls, rattle dishes, and together tend to produce more noise inside a structure than outside. Thus both the audible noise as well as the wall rattle produced by inaudible pressures contribute to human response. To complicate matters even more, inaudible air overpressures can vibrate walls to produce audible noise at large distances, which are inaccurately reported by occupants as ground motions.

9.2.2.3 Character of Blast Excitation and Structural Response

As shown in Fig. 9.2.2.1, both the ground and airborne disturbances (upper-four time histories) produce structure response (lower-four time histories). Because of the importance of frequency, the full wave form or time history should be recorded. When a critical location is known, blast response is best described by the strain at that location. Alternatively, particle velocity (that shown in Fig. 9.2.2.1) can be measured outside the structure of concern, as many recent cracking studies have correlated cracking with excitation particle velocity measured in the ground.

Ground Motion: *Ground motion* can be described by three mutually perpendicular components labeled L (longitudinal), T (transverse), and V (vertical) (Fig. 9.2.2.1). The L and T directions are oriented in the horizontal plane with L directed along the line between the blast and recording transducer. When a study focuses upon structural response, axes can be labeled H1, H2, and V, with H1 and H2 oriented parallel to the structure's principal axis.

Variation of peak motions in each component (L, V, and T in Fig. 9.2.2.1) has led to difficulty in determining which is more important. Horizontal motions seem to control the horizontal response of walls, and superstructures and vertical motions seem to control the vertical response of floors. In an absolute sense, the peak ground motion is actually the maximum vector sum of the three components, which usually occurs at the largest peak of the three components, the dashed line in Fig. 9.2.2.1. This true maximum vector sum is not the FALSE maximum vector sum calculated with the maxima for each component (dots in Fig. 9.2.2.1) no matter their time of occurrence. The FALSE maximum vector sum may be as much as 40% greater than the

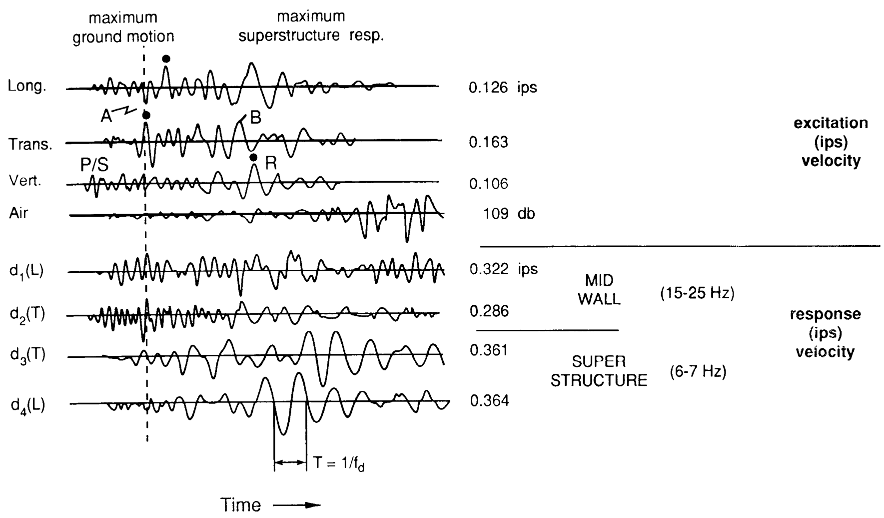


Fig. 9.2.2.1. Comparison of blast excitation by ground and airborne disturbances and residential structure response of walls and superstructure. Measurements were made some 2000 ft (600 m) from a typical surface coal mining blast. (After Dowding, 1988). Conversion factor: 1 ips = 25.4 mm/s.

TRUE maximum vector sum, which is normally 5 to 10% greater than the maximum, single component peak.

In general, experimental observations of threshold or cosmetic cracking, which form the basis of blasting controls in North America, have been correlated with the maximum single component regardless of direction. Therefore, use of the FALSE maximum vector sum, for control, provides a large, unaccounted for, factor of safety.

Two wave types are produced by blasting, *body* and *surface*, and are illustrated by the ground motion in Fig. 9.2.2.1 measured some 2000 ft (600 m) from a typical surface coal mining blast. Body waves travel through earth materials, whereas surface waves travel along surfaces and interfaces of earth materials. The most important surface wave is the Rayleigh, denoted R on the vertical trace in Fig. 9.2.2.1. Body waves can be further subdivided into compressive (compression/tension) or soundlike waves, and distortional or shear waves, denoted as P/S on the vertical trace in Fig. 9.2.2.1. Explosions produce predominantly body waves at small distances. These body waves propagate outward in a spherical manner until they intersect a boundary such as another rock layer, soil, or the ground surface. At this intersection, shear and surface waves are produced. Rayleigh surface waves become important at larger transmission distances as illustrated in the vertical trace by the relatively larger "R" amplitude compared to the "P/S" amplitude.

Sinusoidal Approximation—Typical blast vibrations, no matter the wave type, can be approximated as sinusoidally varying in either time or distance along the radial or longitudinal line as shown by the time variations in Figs. 9.2.2.2a and b. This approximation is useful because it makes calculations for strain and acceleration from particle velocity much simpler than that for an irregular pulse. Ground motion from a blast is similar to the motion of cork caused by a passing water wave. Displacement of the cork from its at rest position is similar to the displacement u of a particle in the ground from its at rest position. Similarly, the cork's velocity $\dot{u}$ as it bobs up and down is analogous to that of a particle in the ground, hence the term *particle velocity.*

The water wave that excites the cork can be described by its wave length g, the distance between wave crests; the wave speed or propagation velocity c at which it travels outward from the stone's impact; and the frequency f or the number of times the cork bobs up and down in one second. Frequency f is equal to $1/T$ or the reciprocal of the period or time it takes the cork to complete one cycle of motion. Frequency is measured in cycles per second or hertz, H_3. Propagation velocity c should not be confused with particle velocity $\dot{u}$, as c is the speed with which the water wave passes by the cork, and $\dot{u}$ is the speed at which the cork moves up and down while the wave passes. Blast vibration waves also can be described by their wavelength, propagation velocity, and frequency in the same fashion as the water wave.

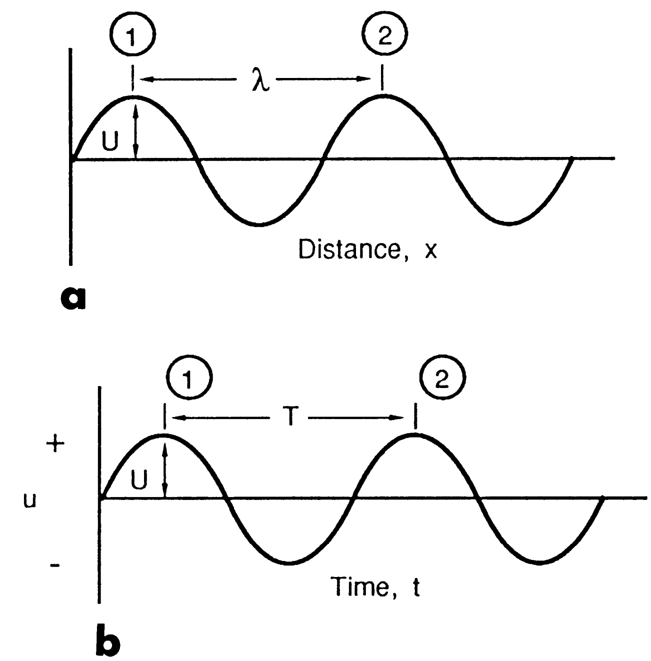


Fig. 9.2.2.2. Sinusoidal approximations: (a) sinusoidal displacement at a fixed point (x = constant); (b) sinusoidal displacement at one instant (t = constant). (Dowding, 1985).

Kinemetric Relationships of Ground Motion—The general form for the sinusoidal approximation is best understood by beginning with the equation for sinusoidal displacement u:

$$u = U \sin(2\pi f t) \qquad (9.2.2.1)$$

where U is maximum displacement, f is frequency, and t is time.

The relationship between the maximum particle displacement u_{max}, particle velocity $\dot{u}_{max}$, and acceleration $\ddot{u}_{max}$, is also greatly simplified by the sinusoidal approximation and is found through differentiation with respect to time of Eq. 9.2.2.1, as shown, whenever the sin/cos function maximizes at 1:

$$\begin{aligned} u_{max} &= U \\ \dot{u}_{max} &= U\, 2\pi f = 2\pi f u_{max} \\ \ddot{u}_{max} &= U\, 4\pi^2 f^2 = 2\ \pi f \dot{u}_{max} \end{aligned} \tag{9.2.2.2}$$

Usually, acceleration is normalized (divided) by gravitational acceleration, 386.4 in./sec^2 (9814 mm/s^2). Therefore, an acceleration of 79 in./sec^2 (2000 mm/s^2) is

$$\frac{2000}{9814} = \frac{79}{386} = 0.2\ g$$

or two-tenths that of gravity.

Kinematic relations between particle displacement, velocity, and acceleration for complex wave forms are exactly related through integration or differentiation of any of the wave forms. For instance, an acceleration time history can be integrated once for a velocity time history, which in turn can be integrated for a displacement time history. Even though a particle velocity record can be differentiated to find acceleration, it is not recommended, as the procedure is sensitive to small changes in the slope of the velocity time history. Further discussion of the inaccuracies of differentiation and integration can be found in Dowding (1985) and in texts devoted to interpretation of time histories (e.g., Hudson, 1979).

Transient Nature of Blast Motions—Great care should be taken not to confuse the effects of steady-state, single-frequency motions with those of transient, blast motions. Most vibration studies conducted by personnel trained in mechanical and electrical engineering and geophysics implicitly assume that the motions are continuous (last many cycles) and steady state (have constant frequency and amplitude). As can be seen in Fig. 9.2.2.1, blast-induced motions last only one or two cycles at a relatively constant amplitude and frequency. Such conditions are not similar enough to steady state motions to allow specific application of steady state approximations such as resonance.

Estimation of Dominant Frequency—Adoption of frequency-based vibration criteria has made the estimation and calculation of the dominant frequency an important concern. Dominant frequency can be estimated through visual inspection of the time history or calculated with Fourier frequency spectra or, alternatively, response spectra.

The accuracy or difficulty of visually estimating the dominant frequency depends upon the complexity of the time history. The easiest type of time history record for frequency estimation is one with a single dominant pulse like that shown in the inset in Fig. 9.2.2.3. This dominant frequency can be determined through the hand measurement of the time of the two zero crossings on either side of the peak. The difference between these times is one-half of the period, which is the inverse of twice the frequency of the dominant peak as shown in the figure.

As shown in Fig. 9.2.2.3, the relatively large explosions produced by surface coal mining, when measured at typically distant structures, tend to produce vibrations with lower principal frequencies than those of construction blasts. Construction blasts involve smaller explosions, but the typically small distances between a structure and a blast as well as rock-to-rock transmission paths tend to produce the highest frequencies. Such high-frequency motions associated with construction blasts have less potential for cracking adjacent structures (Dowding, 1985).

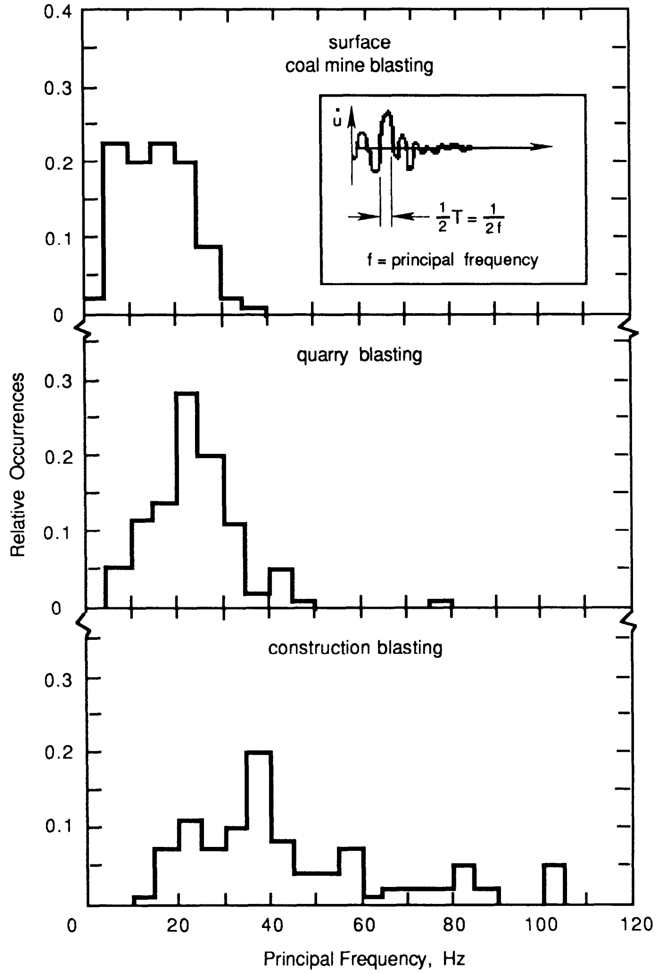


Fig. 9.2.2.3. Dominant frequency histograms at nearest structures categorized by industry. Dominant frequency is defined in the inset. (After Siskind et al., 1980b.)

The most difficult type of record to interpret is that which contains nearly equal peaks at two dominant frequencies such as that in Fig. 9.2.2.1. The two dominant frequencies are the initial 15- to 20-Hz portion (peak A) and the later 5- to 10-Hz portion (peak B). As can be seen in the figure, the initial portion produces the highest wall response while the second produces the greatest superstructure response. For the best frequency correlation of both types of response, both frequencies should be calculated.

The best computational approach to determining the dominant frequency involves the response spectrum. The response spectrum is preferred over the Fourier frequency spectrum because it can be related to structural strains (Dowding, 1985). A compromise approach is to calculate the dominant frequency associated with each peak by the zero crossing approach described above.

Since many time histories do not contain as broad a range of dominant frequencies as that in Fig. 9.2.2.1, most approaches require only the calculation of the frequency associated with the maximum particle velocity for blasts that produce small particle velocities. The more complex frequency analyses are employed only when peak particle velocities approach control limits.

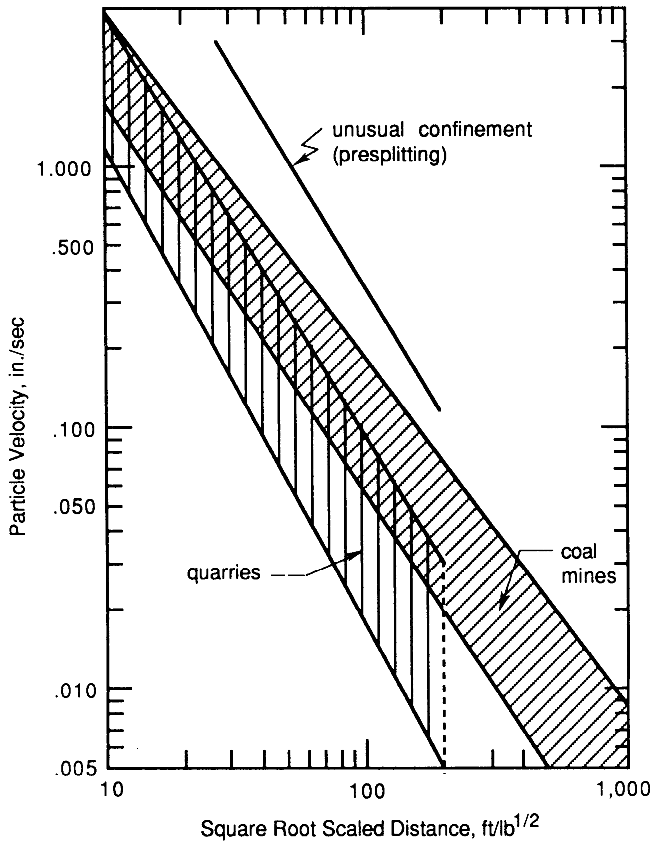


Fig. 9.2.2.4. Attenuation relationships showing scatter from geological and blast design effects as well as high expected velocities from confined shots, such as presplitting. (After Siskind et al., 1980b). Conversion factors: 1 ips = 25.4 mm/s, 1 ft/lb$^{1/2}$ = 0.8197 m/kg$^{1/2}$.

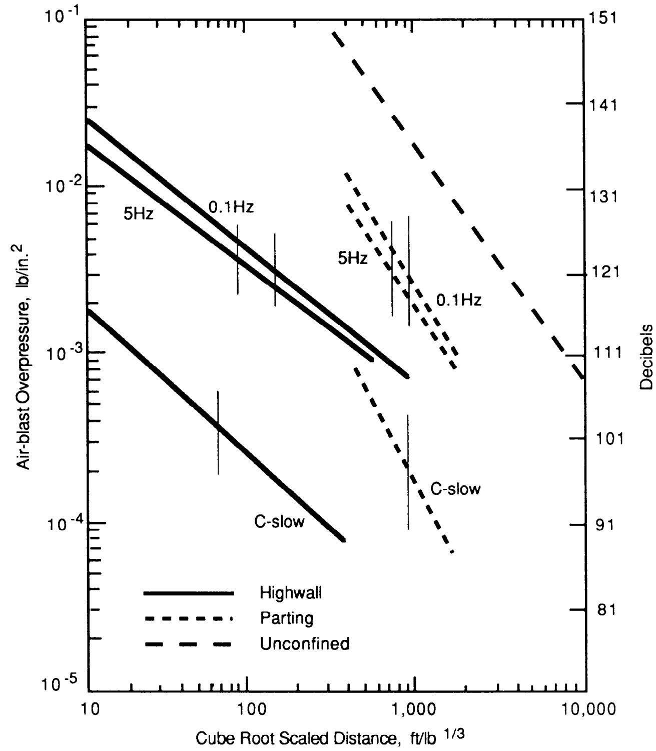


Fig. 9.2.2.5. Attenuation relationships for air overpressures produced by confined (highwall) and partially confined (parting) surface coal mining blasts, as well as unconfined blasts. (Siskind et al., 1980a). Conversion factors: 1 lb/in.^{2} = 6.894 kPa, 1 ft/lb$^{1/3}$ = 0.8759 m/kg$^{1/3}$.

Propagation Effects—Ground motions always decreased with increasing distance. Effects of constructive and destructive interference and geology are included within the scatter of data about the mean trend of the decay in amplitude with distance. While this scatter is large, the associated decay with distance is observed in all blast-vibration studies. Typical examples of this decay are shown in Fig. 9.2.2.4 where maximum particle velocity is plotted as a function of square-root scaled distance from the blast.

Square-root scaling, or plotting peak particle velocity as a function of the distance R divided by the *square root* of the charge weight, $R/W^{1/2}$, is more traditional than the cube-root scaling, which incorporates energy considerations (Hendron, 1977). Both square- or cube-root scaling can be employed to compare field data and to predict the attenuation or decay of peak particle velocity; however, square-root scaling is more popular.

Several square-root attenuation relationships employed in the United States are shown in Fig. 9.2.2.4. They are banded to reflect scatter, which is typical of blasting operations. Curve P should be used for presplitting, cratering, and beginning new bench levels. It is also the basis for Office of Surface Mining (OSM) regulations for conservative shot design when monitoring instruments are not employed.

Dominant frequencies also tend to decline with increasing distance and with increasing importance of surface waves. At larger distances typical for mining, higher frequency body waves begin to have relatively lower peak amplitudes than the lower frequency surface waves, as shown in Fig. 9.2.2.1. Since lower frequencies can elicit greater structural response (Medearis, 1976) as shown in 9.2.2.5, OSM scaled-distance limits decline with increasing absolute distance.

Blast-Induced Air Overpressures: Just as with ground motions, blast-induced air overpressure waves can be described with time histories as shown in Fig. 9.2.2.1. The higher frequency portion of the pressure wave is audible sound. While the lower frequency portion is not audible, it excites structures, which in turn causes a secondary and audible rattle within the structure. The air-blast excitation of the walls is shown by comparing the last one-quarter of the time histories of air blast and wall response in Fig. 9.2.2.1. Unlike ground motions, air overpressures can be described completely with only one transducer, since at any one point air pressure is equal in all three orthogonal directions.

Propagation Effects—Propagation of blast-induced air overpressures has been studied by numerous investigators and is generally reported with cube-root rather than square-root scaled distances. Peak pressures are reported in terms of decibels (dB), which are defined as

$$\mathrm{dB} = 20 \log_{10}\left(\frac{P}{P_o}\right) \qquad (9.3.2.3)$$

where P is the measured peak sound pressure and P_o is a reference pressure of 2.9×10^{-9} psi ($20 \times 10^{-6}(P_a)$).

Fig. 9.2.2.5 summarizes the effect of two important instrumentation and shot variables. First, the effect of the weighting scales is dramatically evident. C weighting greatly reduces the recorded peak pressure at any scaled distance. This does not mean the peak is reduced by changing instruments, but rather that the C weighting system does not respond to the low-frequency pressure pulses. These low-frequency pressure peaks excite structures and occupants whether or not they are sensed by the measurement instruments. The other (5- and 0.1-Hz) labels denote the lower-frequency bounds of the recording capabilities of the "linear" systems.

Second, the effect of venting caused by inadequate stemming can be observed in Fig. 9.2.2.5 from the higher average pressures produced by the parting shots at any scaled distance. Parting shots are detonated in thin rock layers between coal strata in surface mines. Consequently, there is less hole height available for stemming, and these shots many times eject the stemming and thereby produce abnormally high air overpressures. The unconfined relationship should be used for demolition of structures after modification for effects of weather and ground reflection.

Various effects of the wind have been reported and should be added to the average relations presented in Fig. 9.2.2.5. Wiss and Linehan's (1978) study of air overpressures produced by surface coal mining showed that in moderate winds the typical 7.7-dB reduction for each doubling of distance is reduced by

$$7.7 - 1.6\ V_{\mathrm{mph}} \cos\theta \text{ dB} \qquad (9.2.2.4)$$

where V_{mph} is wind velocity in miles per hour and q is the angle between the line connecting the blast and transducer and the wind direction.

An air-temperature inversion causes the sound pressure wave to be refracted back to the ground and at times to be amplified at small, 16-acre (65 km^2) sized locations. Such an inversion occurs when the normal decrease in temperature with altitude is reversed because of the presence of a warmer upper layer. Schomer et al. (1976) has shown that for propagation distances of 2 to 40 miles (3 to 60 km), inversions produce zones of intensification of up to three times the average, attenuated or low air overpressures *at those distances*, with an average increase of 1.8 times (5.1 dB). At distances less than 2 miles (3 km), where high air overpressures are likely to occur, his measurements show no inversion effects.

Structural Strains vs. Particle Velocity: While particle velocity is the traditional measurement of choice, structural strains control cracking. They should be measured directly from relative displacements on structures or within rock masses when critical locations are known, and can be obtained with a variety of strain and relative displacement gages (Stagg et al., 1984). Unfortunately, these critical locations may be either unknown or too many in number to economically measure. Therefore, some means of estimation is necessary.

Ground motion and air overpressure time histories can be employed to calculate the relative displacement of structural components with a knowledge of the responding structure's dynamic response characteristics (Dowding, 1985). These relative displacements can in turn be employed to calculate strains. The accuracy of these estimates is limited by the degree to which the structure behaves as a single degree of freedom system and the accuracy of the estimate of the dynamic response characteristics.

Appropriate Measurement of Particle Velocity: While any of the three kinemetric descriptors (displacement, velocity, or acceleration) could be employed to describe ground motion, particle velocity is the most preferable. It has the best correlation with scientific observation of blast-induced cracking, which forms the basis of vibration control. Furthermore, it can be integrated to calculate displacement. If acceleration is desired, it should be measured directly to avoid differentiation of the particle velocity time history.

The location for measurement varies throughout the world. In North America, the excitation or ground motion is measured on the ground adjacent to the structure of interest. In Europe, the excitation motion is measured on the structure's foundation. The difference stems from historical precedent and location of transducers during scientific observation of cracking rather than difference in philosophy. In North America, many times it is impossible to place transducers on adjacent property owned by a party not involved in the project. Furthermore, if it is desired to describe the excitation motions, then they should be measured outside of and not on the structure. If it is desired to measure structural response motions then they should be measured on the most responsive structural members, which are not the basement or foundation walls because of the restraint provided by the ground.

Time histories of the three components of motion should be measured because of the importance of excitation frequency. Recording of peak motions will not yield information about the dominant frequency and time history details that control structural response. Peak motions and dominant frequency can be employed to describe low-level, non-critical motions. Therefore, machines employed to monitor critical motions (type I in 9.2.2.4) should be capable of recording time histories of selected critical motions. Machines that record only peak motions (type II in 9.2.2.4) can be employed with those that record time histories to provide redundant measurement where frequency content does not vary widely.

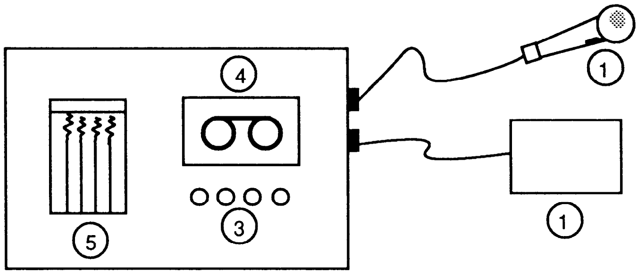
9.2.2.4 Measurement Instruments and Their Deployment

This segment describes characteristics of instruments that measure the ground motions (acceleration, velocity, displacement) and air blast (air overpressure). Since there are many excellent sources for information on instruments, the principal characteristics of available systems will be summarized rather than exhaustively reviewed. The most complete single reference for detailed instrumentation information that is updated periodically is the *Shock and Vibration Handbook* (Harris and Crede, 1976). Specific information on blast vibration monitors is contained in specific publications by the US Bureau of Mines and the Office of Surface Mining (i.e., Rosenthal and Morlock, 1987).

An idealized, field-portable blast monitoring system operating on a 12-V battery is illustrated in Fig. 9.2.2.6. It consists of transducers (1) that convert physical motion or pressure to an electrical current, which is transmitted through cables (2) to an amplifying system (3); and a magnetic tape, paper or computer digital, recorder (4) that preserves the relative time variation of the original signal for eventual permanent, hard-copy reproduction by a pen recorder or light-beam galvanometric recorder or dot matrix printer (5). As one can imagine, there is an almost endless variety of configurations of these five basic components. However, the best involve micro processors (computers) for data acquisition, storage and reproduction.

Transducers: Transducers are one of the weaker links in the measurement system because they must translate kinemetric motions or pressures to electrical signals. The remaining components transform electrical signals or light beams and are not restricted by mechanical displacement. The main characteristics of transducers that affect their performance are sensitivity and frequency response.

Sensitivity of an instrument is the ratio of its electrical output to its kinemetric displacement, velocity, and acceleration or overpressure for energy-converting transducers (i.e., do not require



1. Velocity (3 orthogonal) and sound pressure transducers
2. Cables
3. Amplifier
4. Recorder (tape, disk, or memory)
5. Light beam oscilloscope or dot matrix printer

Fig. 9.2.2.6. Idealized, field portable blast monitoring system that shows the schematic relationship of the five principal components. (Dowding, 1985).

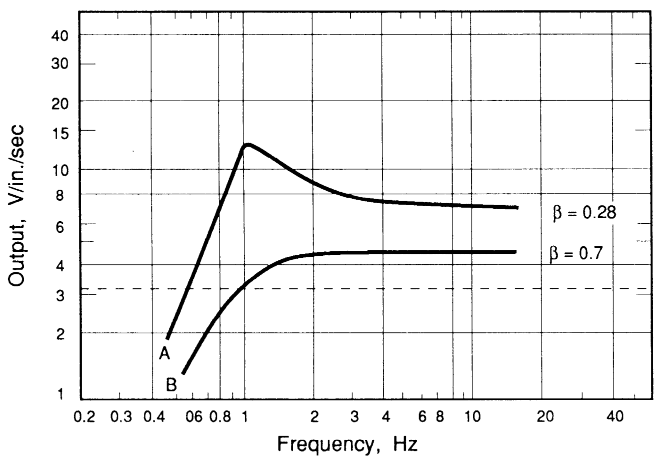


Fig. 9.2.2.7. Example response spectra of a velocity transducer with differing percentages of damping. With 70% of critical damping, this system is ± 3 dB (± 30%) down 1 Hz. (Dowding, 1985.) Conversion factor: 1 ips = 25.4 mm/s.

an energy source). Since allowable limits are specified in terms of ground particle velocity, all blast monitors come equipped with velocity gages.

Frequency response is the frequency range over which the electrical output is constant with a constant mechanical motion. This constancy is normally expressed in terms of decibels. For instance, linear within 3 dB between 5 and 200 Hz means that the transducer produces a voltage output that is constant within 30% between 5 and 200 Hz. Generally, it is better to look at the transducer's response spectra (such as those shown in Fig. 9.2.2.7) to determine the frequencies where this difference occurs. For example, the difference occurs at low frequencies for the velocity transducers in the figure. The importance of the frequency response of air overpressure transducers was discussed in 9.2.2.3.

Transducer Attachment—One of the most critical aspects of vibration monitoring is the mounting of the transducers in the field. The importance of mounting is a function of the particle acceleration of the wave train being monitored. The type of mounting on a horizontal surface is the least critical when the vertical maximum particle accelerations are less than 0.3 g. In this range, the possibilities of rocking the transducer or the transducer package are small, and the transducer may be placed upon a horizontal measurement surface without a device to supply a holding force. When the maximum particle accelerations fall between 0.3 and 1.0 g, the transducer or transducer package should be buried completely when the measurement surface consists of soil (Johnson, 1962). When the measurement surface consists of rock, asphalt, or concrete, the transducers should be fastened to the measurement surface with either double-sided tape, epoxy, or quick-setting cement (Hydrocal or other gypsum-based cements set within 15 to 30 min). If these methods are unsatisfactory, or accelerations exceed 1.0 g, only cement or bolts are sufficient to hold the transducer to a hard surface.

All transducers mounted on vertical surfaces should be bolted in place. Air overpressure transducers should be placed at least 3 ft (1 m) aboveground, pointed downward (to prevent rain damage) and fitted with a windscreen to reduce wind excitation-induced false events.

Digital, Tape, and Hard-Copy Recorders: Microprocessor (computer) or digital recording systems now dominate technical recording because of the ease of computer linkage. The signal is sampled at a certain rate, say, 1000 times/sec, and each sample is converted to a single magnitude. This magnitude and its associated time are then stored in computer memory. Digital recording has several advantages. It is very accurate, as variation in the speed of the tape, if it is used, has no effect, and records can be directly accessed by a computer. Details of the digitization process are discussed elsewhere (Dowding, 1985).

Of those blast-monitoring systems with tape recorders, most employ compact FM cassettes. Many of the systems involve separate record and reproduction modules to reduce the complexity of recording. Care should be exercised to determine the exact details of the system before purchasing, as tape recorder performance varies at low temperature.

A permanent record or "hard copy" of the vibration time history is usually made on photographic film, floppy disk or battery-powered memory chips or paper. Almost all present film-based recorders employ special field-developable, ultraviolet light sensitive paper in combination with light-beam galvanometers to record high frequency motions. The newest generation recorder employs dot matrix printers and/or floppy disks with microcomputers. Unfortunately, those monitors that print after a vibration event may not be recording another event while printing. If multiple shots are likely, this reset time should be determined. Furthermore, printer behavior in cold weather is variable and should also be investigated.

Most recorders can be bought as either single- or multichannel units. A four-channel unit is necessary in blast monitoring to record simultaneously the three components of the ground motion (L, V, and T) and the air blast. The present trend in vibration equipment is to include a signal-conditioning amplifier in the recorder to allow flexible amplification of the signals.

Frequency analysis of records requires a time history and thus some form of permanent record. Instruments recording only peak particle velocities will not allow a frequency analysis. Sending records through the mail for interpretation, results in a delay of five days, and sometimes up to a month. Systems with light sensitive paper or dot matrix printers allow immediate interpretation of frequency without additional costly equipment.

Calibration: It is obvious that the entire vibration measurement system should be calibrated, as it is futile to record data if they cannot be exploited because of a lack of reference. Manufacturers supply calibration curves with their instruments that are similar to the response spectra for transducers shown in Fig. 9.2.2.7. Recalibration or checking requires special vibrating platforms where frequency and displacement are controlled, and, in the field, a calibrating circuit to pulse the magnetic core of the geophone (Stagg and Engler, 1980).

Number of Instruments: While the obvious irreducible number of instruments for each blast is one, two would provide a more thorough documentation of the spatial distribution of effects. If only one instrument is employed, then it should be located at the nearest or most critical receiver. This single type I instrument should record time histories of the three axes of particle velocity as well as air over-pressure. Since it must monitor continuously, it must trigger (begin recording) automatically, and be capable of monitoring even while printing or communicating results. When blasting will occur at more than one general location (i.e., involve nearest structures separated by hundreds of feet or meters), then two and four are the irreducible and optimum number of instruments, respectively. The third should be a spare to insure continuous coverage in case of failure.

The second and fourth instruments in the situations described here may provide a smaller level of service and will be termed type II. They must at least continuously record the peak particle velocity in one axis and may or may not measure air overpressure. The best axis is the vertical, since no horizontal direction decision is required and surface waves usually involve a significant vertical component regardless of the direction of the maximum horizontal component, These instruments should be located at a greater distance than the nearest structure to monitor a large area.

The third or spare instrument can be either type I or II. Where air overpressures will be problematic or frequencies critical, the spare should be type I. This spare instrument can also be employed to monitor sites where complaints develop. This public relations work is essential in North America where lawsuits arise even when all blast effects comply with regulatory guidelines.

This approach describes the least number of instruments. Applicable regulations and mining schedules may require a larger number.

Instrument Deployment During Test Blasts: When blasting projects begin, when geological conditions change radially, or when new initiation systems are introduced, test blasts should be conducted to minimize the number of instruments necessary to monitor production blasts. These tests are conducted to produce project-specific attenuation relations for both air overpressures and ground motion. Such relations vary from project to project because of changes in geology and blasting practices. Additionally, the test blasts allow the determination of the frequency content of motions at different scaled and absolute distances. Frequency is important in estimating structural response through response spectrum analyses.

The attenuation relation is not solely a site property. Although it is dependent upon geology, it is also heavily dependent upon the blast geometry and timing. For instance, with the same charge per delay, a blast with a larger burden will produce an attenuation relation with a similar slope or decay with distance, but with a larger intercept. Furthermore, differing initiation timing will produce changes in the time history, both length and frequency content.

During test blasts, a minimum of four instruments should be deployed to measure peak particle velocity at widely differing scaled distances for the same blast. Therefore, for any one blast design, parameters and initiation sequence are constant, and the resulting attenuation relationship shows only the effect of distance, direction, and/or geology. Seismographs and/or transducers should be placed along a single line with constant geology to determine best the attenuation relationship, or at all critical structures to determine the effects of direction and variable geology. Ideally, the linear orientation should be along a path with constant thickness of soil and not cross any large geologic discontinuities such as faults. If geology changes radically, then two such attenuation lines are necessary, but not necessarily with each blast.

A number of approaches to blast design for vibration control are now available that employ a single-delay, single-hole test blast and a number of instruments to record the attenuation and frequency change around the site. These single-time histories are then synthesized to reproduce the additive time history effects of multiple delay, multiple hole blasts at the differing instrument sites. Such synthesis of time histories to guide blast design has met with variable success, but does not replace monitoring of blast effects at critical structures during production blasting.

9.2.2.5 Evaluation of Measurements

Documentation of blast effects involves two radically differing endeavors: measurement of ground and air disturbances as well as observation of cosmetic cracking. Measurement can now be accomplished remotely with computers to eliminate completely human interaction, whereas scientific observation must involve meticulous human inspection immediately before and after a blast. While the focus of this section is instrumental monitoring, the alleged appearance of cracking by neighboring property owners is nonetheless a very serious consideration.

Principal problems in the evaluation of measured effects involve (1) accounting for geologic and weather effects on the overall attenuation with a small number of instruments and (2) incorporating structural response and frequency effects. Principal problems with the observation of blast-induced cracking involve (1) separating blast-induced from environmentally and human-induced cracking and (2) reducing the enormous amounts of time necessary for direct observation. Observational problem are normally overcome by employing instrumentationally measurable blasting controls at low enough levels to prevent the threshold of cosmetic cracking to even old, degraded structures and eliminating observation altogether. Otherwise blast-induced cracking can be observed only with immediate before and after blast inspection. The remainder of this discussion concentrates on the instrumentation approach and calculation of structural response.

Structural Response and Frequency Effects: Structures respond to both ground and airborne disturbance, as shown by the bottom four time histories in Fig. 9.2.2.1. Walls respond more to the higher frequency (15 to 20 Hz) waves in the early portion of the ground motion, while the superstructure or overall skeleton of the structure responds more to the last or lower frequency (5 to 10 Hz) portion. Walls are again excited by the arrival of the air pressure wave. Structural response can be calculated from the ground motions if the natural frequency and damping of structural components are known or estimated.

Langan (1980) has shown that measured structural response has a higher correlation coefficient with calculated single degree of freedom (SDF) response than with peak ground motion. Therefore, structural motions can be estimated more accurately by assuming that they are proportional to response spectrum values at the particular structure's natural frequency than by assuming that they are proportional to the peak ground motion. This improved correlation is largely a result of the consideration of frequency in the response spectrum, which is calculated from SDF response.

Origin of the SDF Model—One of the critical structural response factors is the amount of differential displacement (δ in Fig. 9.2.2.8) that occurs between or along structural members because it is proportional to strain, which, in turn, causes crack-

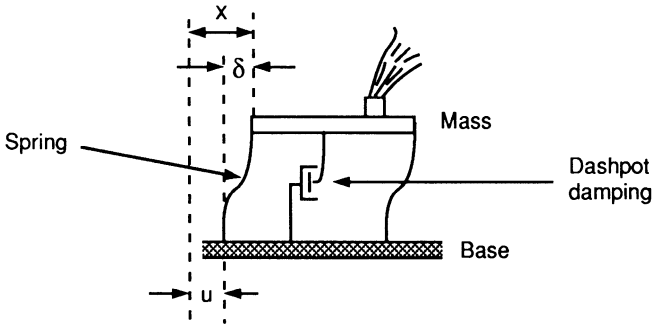


Fig. 9.2.2.8. Single degree of freedom model of house that shows relative displacements of the walls δ, and the analogy between model mass and roof mass, model stiffness, and wall stiffness. (Dowding, 1985.)

ing. Such displacements can be computed with a mathematical idealization of the SDF model shown in Fig. 9.2.2.8.

It is necessary to simplify a structure so that computations are practical. The fundamental characteristics of a structure that govern its behavior under vibratory or dynamic loading are (1) the masses of the main components (analogous to floor and roof masses), (2) the spring stiffnesses of the main components (analogous to wall stiffness), and (3) the amount of damping or energy dissipation (analogous to differential movement in cracks, joints, and connections). Behavior of one- or two-story buildings is directly analogous to the behavior of an SDF system when movement in only one direction is considered. When multistoried structures are considered, it is necessary to model the structure as multidegree-of-freedom systems. However, even such a system may be idealized as a single-degree-of-freedom system to calculate the fundamental mode of response.

If a structure's damped natural frequency f_d and its fraction of critical damping b are known, values of dynamic properties, f_d and b, can be accurately measured from a free vibration time history of the building response. These measured parameters automatically account for the factors that are difficult to quantify, such as the degree of fixity of the columns and the damping coefficient. As shown in Fig. 9.2.2.1, these parameters can be measured from the structure's free response. Time between peaks is the period, $T = 1/f_d$, and the decay of free oscillation is proportional to the damping, b.

Estimation of Dynamic Response Properties—The fundamental natural frequency f_d of the superstructure of any tall building can be estimated from compilations of work in earthquake engineering (Newmark and Hall, 1982):

$$f_d = \frac{1}{0.1N} \tag{9.3.2.5}$$

where N is the number of stories. Substitution of 1 and 2 for residential structures for N yields f values of 10 and 5 Hz for one- and two-story structures, which compares favorably with the results of actual measurements.

Damping b is a function of building construction and to some extent the intensity of vibration. Thus it cannot be simplified as easily as the natural frequency. Measurement reveals a wide range of damping for residential structures with an average of 5% (Dowding et al., 1981). This value is also appropriate for initial estimates involving taller engineered structures. Further details for engineered structures can be found in Newmark and Hall (1982).

Table 9.2.2.2. Natural Frequencies for Unusual Structures

Type	Height, m	*f*, Hz
Radio tower[a]	30	3.8
Petroleum distillation tower[a]	21	1.2
Coal silo	60	0.6
Bryce Canyon rock pinnacle[b]	27	3

Sources: [a]Medearis (1975b).
[b]Dowding and Kendorski (1983).
Conversion factor: 1 ft = 0.3048 m.

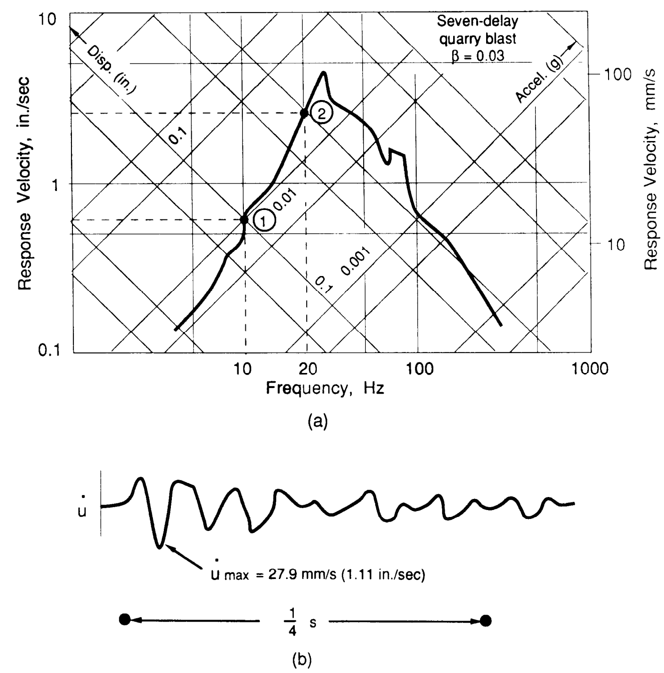


Fig. 9.2.2.9. Construction of pseudovelocity response spectrum: (a) response spectra; (b) associated excitation time history. The time history is operated upon by the single degree of freedom (SDF) equation to produce a computed relative displacement (δ), which is then multiplied by the circular natural frequency ($2\pi f$; or $2\pi 10$ for point (1)) to produce the pseudo response velocity. (Dowding, 1985).

Walls and floors vibrate independently of the superstructure and have their own, but similar, fundamental frequencies of vibration that range between 12 and 20 Hz with an average value of 15 Hz (Dowding et al., 1981). Floors tend to have lower natural frequencies in office buildings with large floor spans, but are similar to wall natural frequencies in residential structures.

Dynamic response properties of some tall, unique structures cannot be estimated with the l/0.1 N equation. Field-measured natural frequencies for these types of structures are given in Table 9.2.2.2.

Response Spectrum—The pseudovelocity response spectrum of a single ground motion, such as that of a seven-delay quarry blast in Fig. 9.2.2.9, is generated from the relative displacement δ_{max} values of a number of different SDF systems when excited by that motion. Consider two different components of the same structure, the 10-Hz superstructure and the 20-Hz wall. If the ground motions, $\dot{u}(t)$ in Fig. 9.2.2.9b, of the seven-delay quarry blast are processed twice by the SDF response equation with

f = 10 and 20 Hz and b held constant at 3%, two δ_{max} values will result.

The first computation is made with the 10-Hz system, which has a circular natural frequency of

$$p = 2\pi(f) = 2\pi(10) \quad (9.2.2.6)$$

and results in an SDF equation computed

$$\delta_{max} = 0.25 \text{ mm } (0.01 \text{ in.}) \quad (9.2.2.7)$$

This δ_{max} is then converted to pseudovelocity, PV as

$$PV_{10} = p\delta_{max} = 2\pi(10)(0.25)$$
$$= 15.7 \text{ mm/s } (0.62 \text{ ips}) \quad (9.2.2.8)$$

and is plotted as point 1 in Fig. 9.2.2.9a. The same computation is then repeated for the 20-Hz system.

$$p = 2\pi(20)$$
$$\delta_{max} = 0.5 \text{ mm } (0.02 \text{ in.})$$
$$PV_{20} = 2\pi(20)(0.5) = 63.5 \text{ mm/s } (2.5 \text{ ips})$$

and PV_{20} is plotted as point 2 in Fig. 9.2.2.9a. If the same ground motion time history in Fig. 9.2.2.9b is processed a number of times for a variety offs with b constant, the resulting pseudovelocities will form the solid line in Fig. 9.2.2.9a.

Fourier Spectra DO NOT Directly Predict Response—With increasing use of computers, calculation of various spectra from time histories have become commonplace. The two most common are the Fourier frequency and pseudovelocity response spectra. Although they are essentially different in meaning and typical use, they are similar for undamped response where the maximum motion occurs near the end of the time history (Dowding, 1985; Hudson, 1979). Since response spectra are calculated for damped response and peaks normally occur in the middle as well as the beginning of the time history, the two spectra are not usually the same.

Only the pseudovelocity response spectrum can be employed to calculate directly structural response. Because of the similarity of Fourier and response spectra, either can be employed to determine the dominant frequency in the ground motion.

Case Histories Demonstrate Importance of Response Spectrum Analysis: Fig. 9.2.2.10 compares time histories and response spectra from the longitudinal components of an urban construction blast and a surface coal mine blast. Although the peak particle velocities are similar: 0.15 ips (3.8 mm/s) for the construction blast A; and 0.13 ips (3.3 mm/s) for the surface mining blast B; the response spectra differ radically. This difference is greatest in the range of natural frequencies of residential structures and their components, 5 to 20 Hz. In this range the surface mining motions produce response velocities that are 10 times greater than the construction blast.

Surface Mine Blast—Surface mining induced ground motion was produced by a multiple row blast. Some sixty 83-ft (25-m) deep, 15-in. (380-mm) diameter, holes were arranged in a four-row pattern. The burden between rows was 20 ft (6.1 m) and the hole spacing was 25 ft (7.6 m). Each hole contained four decks (or charges that are detonated at intervals separated by at least 17 ms). The ammonium nitrate-fuel oil (ANFO) charge weight per deck ranged from 100 to 130 lb (45 to 60 kg). Therefore, the largest charge per delay was 130 lb (60 kg), and the total charge was 27,700 lb (12,600 kg).

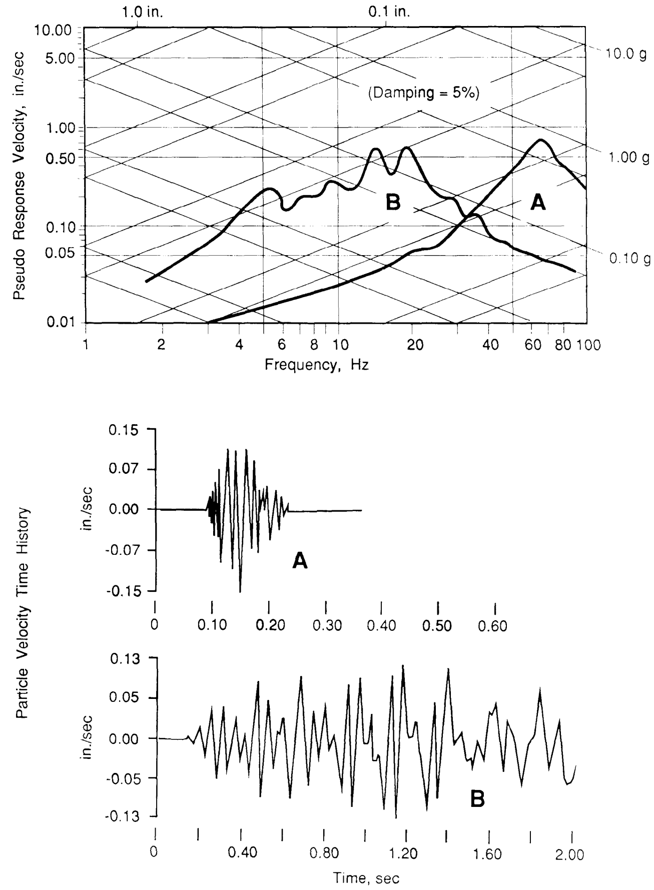


Fig. 9.2.2.10. Comparison of time histories spectra from construction and surface mining blasts respectively lasting 0.15 and 2.0 sec. Even though the particle velocities are approximately equal, responses in the 5 to 20 Hz frequency range differ greatly. Conversion factor: 1 ips = 25.4 mm/s.

Geology between the blast and the transducer, located at the nearest residence, consisted of sedimentary rock with 10 to 30 ft (3 to 10 m) of overlying silty glacial till. A small, 33-ft (10-m) deep, gully was located some 1300 ft (400 m) north of and between the blast and transducer. Soil depth at the transducer was 10 to 15 ft (3 to 4 m). Some 2750 ft (825 m) separated the shot from the transducer, where the longitudinal velocity time history in Fig. 9.2.2.10 with a peak of 0.13 ips (3.3 mm/s) was recorded. The accompanying transverse and vertical peak particle velocities were 0.18 and 0.09 ips (4.6 and 2.3 mm/s) with dominant frequencies between 11 and 13 Hz. Other similarly designed shots with distances between 1900 and 2700 ft (580 and 825 m) produced peak particle velocities between 0.17 and 0.23 ips (4.3 and 5.8 mm/s) with dominant frequencies between 13 and 17 Hz.

These particle velocities are high at 2700 ft (825 m) according to scaled distance relationships. These greater than normal particle velocities may be a result of unusual confinement (too large a burden) or delay overlap. For instance, if two delays had overlapped to add, then the maximum charge per delay would have doubled and the square-root scaled distance would have declined by 30%. Particle velocities measured at smaller scaled distances were closer to expected levels.

Urban Construction Blast—Construction blast-induced motion was produced by a much smaller shot than the surface mining example. Some five 12-ft (3.6-m) deep, 1.5-in. (38-mm) diameter holes were arranged in a single row. Each hole was charged with a stick gelatin dynamite and initiated separately with a constant 25 ms between each delay. The burdens and hole spacings were small, approximately 2 to 3 ft (0.6 to 0.9 m). The total charge was 20 lb (9 kg), and the maximum charge per delay was 5 lb (2.3 kg).

The structure of concern, a historic theater, and the recording transducers were located some 50 ft (15 m) away. Rock being fragmented consisted of granitized biotite schist. This shot produced peak particle velocity in the L, T, and V axes of 0.15, 0.16, and 0.28 ips (3.8, 4.1, and 7.1 mm/s), respectively, with dominant frequencies between 75 and 125 Hz.

Because these dominant frequencies were so high compared to the natural frequencies of the theater's structural components, their response was less than the peak excitation particle velocity. The transducer recording the excitation motions was located in the basement because there was no stable location outside as rock was being removed immediately adjacent to the theater's wall. Another vertical axis transducer was placed at the mid span of the theater's balcony, whose left most support was immediately above the transducer measuring the excitation motions. The peak vertical particle velocity of the theater's balcony was only 0.14 ips (3.6 mm/sec), approximately half of the peak excitation motion in that axis.

Restrained Structures and Rock Masses: Capacity for free response allows aboveground structures such as homes and rock pinnacles to amplify selectively incoming ground motions. On the other hand, buried or restrained structures such as pipelines and rock masses cannot respond freely. Regardless whether response is restrained or free, cracks are initiated by strains in either case. Whereas strains in a freely responding structure are proportional to the relative displacement between the ground and the superstructure as shown in Fig. 9.2.2.11, strains in a restrained structure such as pipelines will usually be those of the surrounding ground and can be approximated as those produced by plane wave propagation and are

$$\epsilon = \frac{\dot{u}}{c_c} \text{ and } \gamma = \frac{\dot{u}}{c_s} \qquad (9.2.2.9)$$

where ϵ and γ are axial and shear strains, c_c and c_s are compressive and shear wave propagation velocities, and $\dot{u}$ are maximum compressive and shear wave particle velocities, respectively (Dowding, 1985). For cases involving one critical location along a pipeline, the pipe strains should be measured directly on the metal (Dowding et al., 1990). For cases involving tunnel and/or cavern liners critical strains can be estimated through calculation of the relative flexibility of the rock and liner (Hendron and Fernandez, 1983).

9.2.2.6 Controlling Blast Effects

Direct regulation or specification of effects, rather than design, is the most effective control from a regulatory viewpoint because effects are so dependent upon details of the shot geometry and initiation sequence. Such dependency renders control impossible by simple regulatory specification of two- or three-shot design parameters. For instance, consider control by specification of the maximum charge weight detonated per instant at given distances from the nearest structure. Even with such detailed specification, intended vibration limits at the structure may be exceeded because of poor choice in the location of holes and/or their relative time of initiation. Discussion of shot design is presented earlier in this chapter (segment 9.2.1).

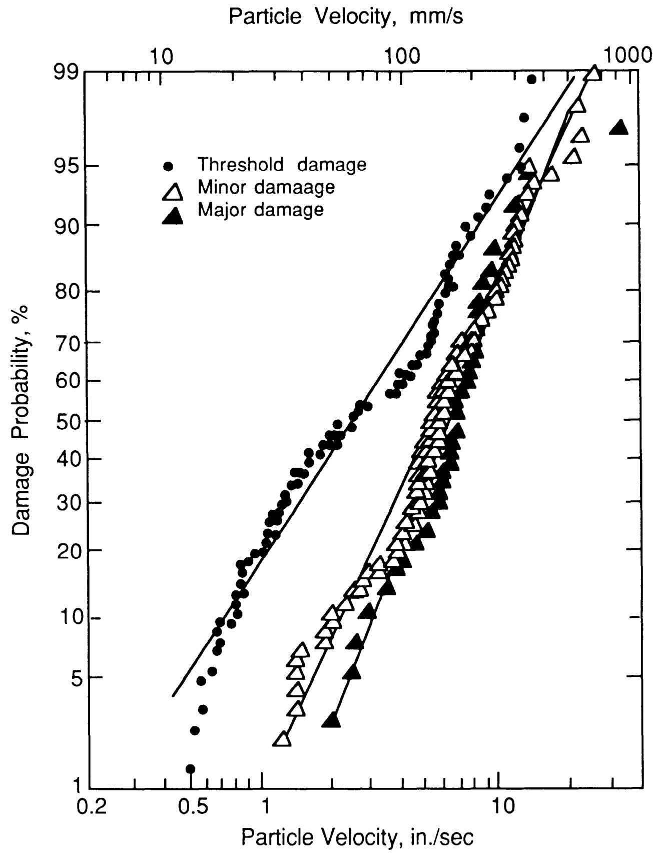


Fig. 9.2.2.11. Probability analysis of worldwide blast cracking data. (Siskind et al., 1980b.) Threshold damage is the occurrence of hair-sized, cosmetic cracks similar to those caused by natural, environmentally induced expansion and contraction.

Present regulatory control limits in many countries are below those levels at which cosmetic cracking may appear. There are two principal reasons for such tight restrictions. First, regulatory limits are influenced heavily by human response to blast-induced vibration and noise. Since humans are approximately 10 times more sensitive than structures to vibration, low regulatory limits are understandable. Second, many regulations appear to have been adopted without the documented, scientific experimentation necessary to determine the vibration levels that cause cracking.

Statistical Analysis of Data with Pre- and Post-Blast Inspection: Unmeasurables in observation can be taken into account indirectly by considering the appearance of cosmetic cracks as a probabilistic event. In order to investigate the effects of certain data sets on the overall conclusions, the probability computations of cracking at given particle velocity levels have been made several times (Siskind et al., 1980b, 1981). All of these studies involve both immediate pre- and post-blast inspection of walls in residential structures, many of which were old, distorted, and whose walls were covered with plaster. Definitions of the observed cracking in each study are described in 9.2.2.2.

Data from various sets of observations were analyzed with cracking points and the assumption that every cracking point excludes the possibility of noncracking at a higher particle velocity (Siskind et al., 1980b, p.55). If the probability of cracking is calculated as the percentage of points at lower levels of velocity, the result is the log-normal scaled plot of the probability of

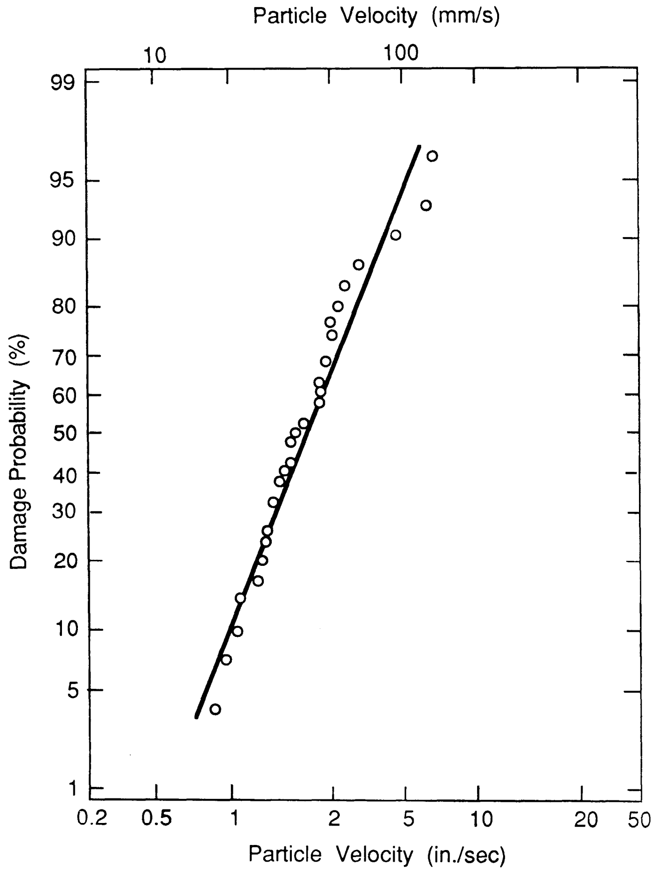


Fig. 9.2.2.12. Probability analysis of blast-induced threshold cracks observed by US Bureau of Mines. (Siskind, 1981.)

cracking vs. particle velocity in Fig. 9.2.2.11. This approach seems conservative as low particle velocity observations do not count noncracking at higher levels.

According to Fig. 9.2.2.11, there appears to be a lower limit of particle velocity of 0.5 ips (12 mm/s) below which no cosmetic or threshold cracking (extension of hairline cracks) has been observed from blasting anywhere in the world. This observation includes the data with unusually low frequencies that were collected by Dvorak (1962). His data are those that tend to populate the lower region of Fig. 9.2.2.11. High-frequency data (> 40 Hz) show that a 5% probability of minor cracking does not occur until particle velocities reach 3 ips (75 mm/s) (Siskind et al., 1980b).

Admissibility of Dvorak's data has been questioned because of the absence of time histories; some of the other studies, such as that by Langefors et al. (1958), are also plagued by the same lack of time histories. To resolve this difficulty, only the new US Bureau of Mines observations have been included in a recomputation of probabilities in Fig. 9.2.2.12. The observations include low frequency motions associated with surface mining. Again there is a particle velocity, 0.79 ips (20 mm/s), below which no blast-induced cracking was observed.

Comparison of Blast and Environmental Effects: Crack width changes from ground motions less than 1 ips (25 mm/s) are less than those caused by the passage of weekly weather fronts (Dowding, 1988). This conclusion was reached after measuring the displacement response of a poorly built, nonengineered house to surface coal mining vibrations for some 8 months. Displacements were measured at ten different wall positions that included cracked and untracked wall covering. Weather and blast-induced crack displacements across the most dynamically responsive wall covering crack are compared in Fig. 9.2.2.13. The continuous and highly cyclical curve is that of displacements produced by environmental change. The small circles are the maximum, zero-to-peak, dynamic displacements recorded by the same gage. Even though the maximum recorded particle velocity was as high as 0.95 ips (24 mm/s), the maximum weather induced displacements were three times that produced by blasting. On other gages, weather changes produced displacements that were 10 times greater than those produced by blasting.

Special Considerations: The statistically determined control limits are too low for basement walls and engineered structures. They were based on response of residential structures and the lower limit cases involving cracking of above ground plaster or gypsum wall board wall coverings in older, distorted structures.

Engineered Structures—Concrete is a good deal stronger than plaster. Therefore, engineered structures constructed of concrete can withstand maximum particle velocities of at least 4 ips (100 mm/s) without cracking (Crawford and Ward, 1965). Furthermore, buried structures such as pipelines and tunnel linings are not free to respond, as were the above ground residential structures whose response provides the data from which most limits are chosen. Therefore underground structures are able to withstand even greater excitation motions (Dowding, 1985).

Specific engineered structures should be analyzed in terms of the strain that can be withstood by critical elements and the strain should be measured. This approach is particularly appropriate for singular structures with isolated portions nearer to the blast source.

Fatigue or Repeated Events—Since current regulatory limits are so low as to restrain blast-induced displacements below those caused by the passage of weekly weather fronts, the question of repeated events becomes moot. Weather by itself over the years produces greater repeated event effects than does blasting.

A repeated event experiment conducted at the US Bureau of Mines test house (Stagg et al., 1984), confirms the low level of current regulatory controls with respect to fatigue cracking. The test house was framed in wood with paper-backed gypsum board interior walls. When continuously vibrated at an equivalent ground particle velocity of 0.5 ips (12 mm/s), no response was observed until 52,000 cycles, when a taped joint between sheets of gypsum board cracked. These taped joints are the weakest and most compliant zones in a house with paper-backed gypsum board walls.

Rock Mass Displacement and Cracking—Cracking in rock immediately adjacent to a blast can be controlled by limiting the particle velocities to 27 ips (700 mm/s) in the volume of rock to be protected (Holmberg and Persson, 1978).

Rock displacement by forces produced by delayed gas pressure cannot be controlled by specifying an allowable particle velocity. Fortunately, these displacements occur only very close to a blast, within 100 to 165 ft (30 to 50 m), and are associated with blocks that are unconstrained by other surrounding rock.

Sliding instability of individual rock blocks must be evaluated on a case by case basis. Each block must have an adequate factor of safety to prevent static failure (Dowding and Gilbert, 1988).

Frequency Based Control with Dominant Frequency: Fig. 9.2.2.14 shows the limit adopted by the US Office of Surface Mining that is based on a suggested, but not rigorously validated, proposal by the US Bureau of Mines (Siskind et al., 1980a). Corner 1 represents the lowest particle velocity at which USBM

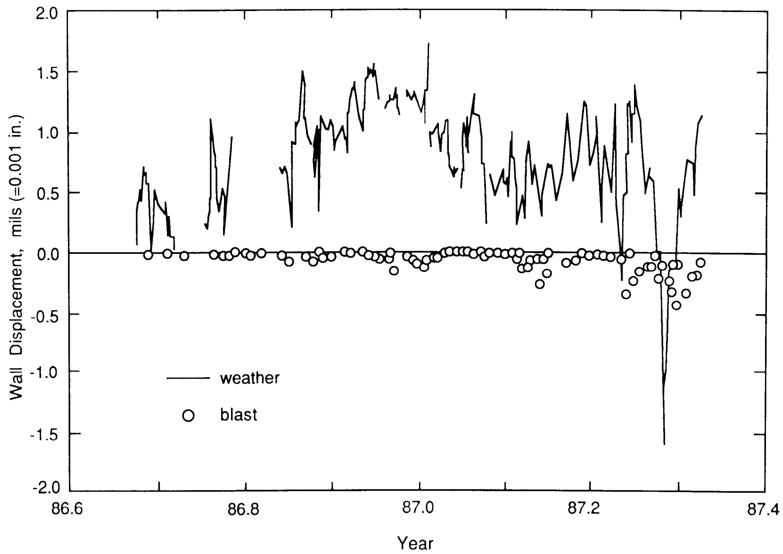


Fig. 9.2.2.13. Comparison of displacements produced by weather-induced changes in humidity and temperature (continuous line) with those produced by surface coal mine induced ground motions (O's). Conversion factor: 1 mil = 0.0254 mm.

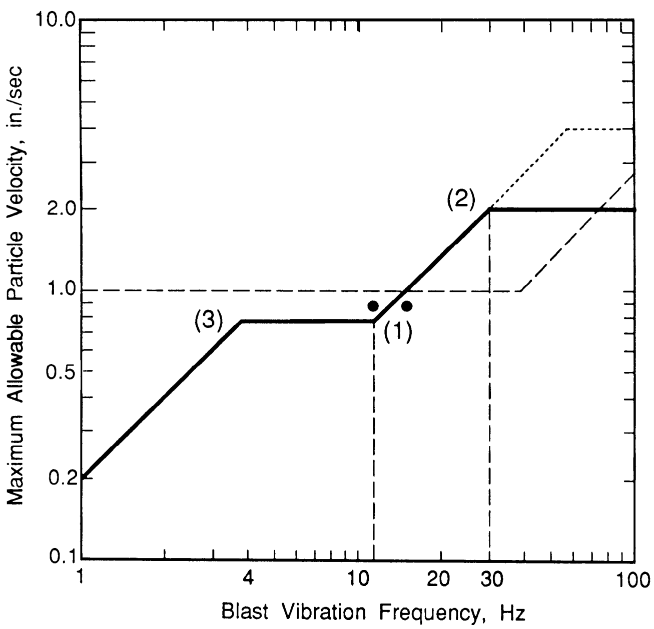


Fig. 9.2.2.14. Frequency-based blast vibration control limit to protect residential structures modified by US Office of Surface Mining from US Bureau of Mines suggestion (Federal Register, Vol. 48, No. 46, 1983). Corner 1 is verified. Dotted line has been employed safely for close construction blasting near engineered structures. Dashed line has been employed safely for construction blasting in urban areas near older homes and historic buildings. Conversion factor: 1 ips = 25.4 mm/s.

personnel have observed cosmetic cracking. Neither of the corners 2 and 3 have been confirmed.

Dominant frequency that is consistent with Fig. 9.2.2.14 is that associated with the peaks in the time history with amplitudes greater than 50% of the peak or maximum particle velocity. The frequency of these peaks were calculated from the zero crossing method as shown in the inset for Fig. 9.2.2.3. Determining frequency from that associated with the peak particle velocity is a good first approximation and eliminates the need for sophisticated Fourier or, alternatively, response spectra analysis. Response spectrum analyses are the most precise approach to account for the frequency effects of structural response and should be employed in singular cases where an exacting analysis is required.

Table 9.2.2.3. Air Overpressure Control Limits as a Function of Instrument Frequency Weighting

Lower frequency limit of measuring system, in Hz (–3 dB)	Maximum level, in dB
0.1 Hz or lower—flat response[a]	134 peak
2 Hz or lower—flat response	133 peak
6 Hz or lower—flat response	129 peak
C-weighted—slow response[a]	105 peak dBC

[a]Only when approved by the regulatory authority.

Regulatory Compliance for Air Overpressure: Although broken glass is normally associated with excessive air blast overpressures, limits in the United States are based upon wall response necessary to produce wall strains equivalent to those produced by surface coal mining-induced ground motions with peak particle velocity of 0.75 ips (19 mm/s). These limits are presented in Table 9.2.2.3. If a wall strain level equivalent to that produced by 1.0 ips (25 mm/s) particle velocity (measured in the ground) were chosen, the allowable overpressure would increase by 3 dB. Most cases of broken glass are reported to have been observed at air overpressures of 136 to 140 dB (as measured with a linear transducer).

Because of the different sound weighting scales that might be employed by monitoring instruments, the recommended levels in Table 9.2.2.3 differ by instrument system. Since structures are most sensitive to low frequency motions and the greatest air pressures occur at these inaudible frequencies, A-weighted scales cannot be employed at all. Since C-weighted scales are the least sensitive at low frequencies, their use requires the most restrictive limits.

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Chapter 9.3
MATERIALS HANDLING: LOADING AND HAULAGE

RICHARD J. SWEIGARD

9.3.1 INTRODUCTION

The combined tasks of loading and haulage are the foundation of the mining industry. In the simplest scenario, a loading device is used to load fragmented ore into a unit conveyance, which carries the ore to a facility where it will be beneficiated. In the past, the loader has been as primitive as a laborer equipped with a shovel and the conveyance, a mule-driven cart. Although loading and haulage as performed today encompass much more, the basic operations are still the same. In addition to the loading and haulage of the mineral commodity to be exploited, many operations require loading and haulage of large volumes of waste material to expose the economic deposit. Some loading and haulage operations, such as disposal of mill tailings or replacement of topsoil at strip mines, are ancillary to the actual production cycle; however, the same principles are applicable.

The loader and conveyance must be selected to suit the needs of the operation. The mining industry is accustomed to dealing with large volumes of material. Some of the largest earthmoving machines in existence are employed in surface coal mines, where a single machine may excavate in excess of 25 million yd^3 (19 Mm^3) of material per year. Although small volumes are the exception, more selective operations result in a reduction of the total volume of material to be moved and employ much smaller loading machines, since the point of attack is very specific.

In general terms, the material most commonly handled by the mining industry is rock that has been fragmented either by mechanical or explosive methods. This rock can be an ore containing an economic mineral; it can be the salable mineral product itself in a fairly undiluted form, such as coal, phosphate, and industrial minerals; or it can be waste rock that must be excavated to expose the economic mineral deposit. Some of the physical properties that affect the handling of materials are abrasion, adhesion, cohesion, angle of repose, compressibility, bulk density, particle density, friability, moisture content, hygroscopicity, fragment size, fragment shape, and void ratio. Fragment size is determined primarily by the method of fragmentation. Although any material will have some in situ moisture content, this property may be altered during the materials handling process. The remainder of the properties listed are inherent to specific materials and must be considered in the design or selection of a materials handling system.

In selecting equipment for loading and haulage, it is necessary initially to determine the production requirements of the system. Generally, these requirements are given in units of weight or volume per unit of time. Typical units vary for different operations. For example, the production of coal loading machines may be given in terms of the amount produced per minute, hour, or shift while the production of large overburden stripping machines may be given in cubic yards (cubic meters)/hour, but the overall production requirement is reported usually in millions of cubic yards (cubic meters)/year.

Once the production requirements are known, various combinations of equipment and sequences of operations can be developed that are designed to produce the required volume of material. This step assumes a knowledge of the production capacity for each specific component of the proposed loading and haulage system. The production capacity of any loading or haulage component can be described, in general terms, by the volume of material it can handle at any instant (capacity) and the time required to complete one cycle of operation, or in other words, the time it takes for that volume of material to move completely through that component of the system (production rate). Of course, this simple volume and cycle time formulation must be modified to account for varying job conditions and the knowledge that mechanical systems do not always perform at peak efficiency.

The simplest materials handling systems involve a discrete volume of material produced by a single machine with a well-defined cycle time. Systems become more complex when multiple machines are needed to satisfy production requirements, continuous flows of material are produced by loading or haulage equipment, or it is necessary to provide storage within the system to accommodate surges by various components.

Engineering judgment is a necessity in developing initial loading and haulage plans, since it is impossible to evaluate every conceivable system that would potentially satisfy the production requirements. The selection of a loading and haulage system, then, is based upon safety considerations and an economic evaluation aimed at meeting the production requirements at a minimum cost per unit of production.

A straightforward or deterministic approach is taken to production calculations in this chapter of the *Handbook.* In the next segment (9.3.2), equipment used for loading and haulage are classified according to the type of function they perform and the manner in which they operate. In the following segment (9.3.3), a generalized equipment selection process is described, and in the last (9.3.4), production calculation steps are described for each class or subclass of equipment. Although the descriptions are fairly specific for particular types of equipment, the same formulation may apply to several varieties of equipment as noted at the beginning of each segment. A caution is in order to alert the reader that the deterministic approach to loading and haulage production calculations may be misleading if various uncertainties are misunderstood or ignored. The reader is, therefore, also referred to Chapter 9.4, Cycles and Systems, where these uncertainties are given due consideration.

9.3.2 EQUIPMENT CLASSIFICATION

To discuss loading and haulage it is necessary to address specific types of equipment or groups of equipment with common characteristics. This requires some system for classifying or grouping the various types of equipment. A few general classifications have been proposed in materials handling texts (Carson, 1961; Apple, 1963). Adler and Lineberry (1986) presented a comprehensive classification system for all mining and reclamation equipment. Atkinson (1971) presented an equipment application scheme for surface mining that classified equipment for each unit operation as either continuous or noncontinuous.

A slightly different classification system is presented here to satisfy the particular objectives of this portion of the *Handbook.* Those objectives are to provide the basic selection principles while maintaining some measure of generality. This should facilitate the preliminary design of loading and haulage systems, al-

Table 9.3.1. Classification of Loading and Haulage Equipment

	Load		Haul		Combined	
	No tramming	Minimal tramming	Nonfixed path	Fixed path	Mobile	Fixed-base
Discrete unit	Loading shovel Backhoe Hydraulic front shovel	Front-end loader (FEL)	Truck Shuttle car*	Rail Skip hoist Slusher Aerial tramway	Tractor scraper Dozer Load-haul-dump (LHD)	Dragline Stripping shovel
Continuous flow	Bucket wheel excavator (BWE) Bucket chain excavator Dredge Continuous miner		Bulk solids	Fluid transport		
			Belt conveyor Screw conveyor Chain conveyor	Slurry pipeline Pneumatic transport	This category is not characterized by single machines. Rather, continuous loaders and haulers are placed in series to create a continuous flow of material, e.g., longwall shearer and chain conveyor, BWE and belt conveyor, auger head and screw conveyor. Therefore, production capacity of the combined system is the smaller production capacity of the two components.	

*Electrically powered shuttle cars do not follow a fixed path, but are constrained by the trailing cable and cable reel location (except with battery-powered cars).

though more detailed equipment information will be needed for final selection. The classification system used here is not intended to be a comprehensive classification of all mining equipment; rather, it addresses only loading and haulage equipment.

The classification system used in this section is outlined in Table 9.3.1. The two primary distinctions in this system are the type of function that the machine performs (i.e., loading, haulage, or a combination of loading and haulage) and the form that the material has in the process (i.e., either discrete units or a continuous flow of material).

The distinction between discrete units of material and a continuous material flow is an important one from the standpoint of production capacity calculations. All the production calculations for loading and haulage equipment that treat the material as discrete units can be characterized by a cycle. These cycles are sequences of constituent operations that the machine performs in loading or hauling one complete unit of material. Estimating the cycle time, then, is the key to determining the production capacity of this equipment type. The output of the continuous flow operation is characterized typically by the cross-sectional area of the material flow multiplied by the speed at which the material is traveling.

Presented in matrix format (see Table 9.3.1), the two primary means of classification result in six separate categories of equipment. As noted in the table, the category of combined loading and haulage equipment utilizing continuous flow is not characterized readily by any one piece of machinery, but requires separate continuous flow loading and haulage units to be operated in series. The other categories are subdivided, as necessary, to describe major distinctions in the manner that the primary function is performed. For example, as stated previously, all loading and haulage equipment that deal with discrete units of material can be described in terms of a cycle time. However, these categories can be subdivided into one group that does not require the machine to travel during its cycle and another group that requires translocation as part of its cycle. Travel time, then, can become a major component of the overall cycle time for this class of equipment. Likewise, haulage machinery can be subdivided into one group that always travels a fixed path, such as a rail car, and another group that is not confined to a fixed path, such as a truck. Even in this example, however, the path is not totally unrestricted since trucks must typically follow prepared haul roads. Finally, in the case of continuous flow haulage systems, a distinction is made between bulk solids conveyors and fluid transport due to the large differences in the production formulations.

9.3.3 EQUIPMENT SELECTION PROCESS

Before a generalized selection process is described, it is necessary to define several fundamental terms that are independent of specific equipment types. These terms are as follows.

1. *Production:* Total volume or weight of material to be handled in a specific operation. It can refer to either the economic mineral to be produced or to the waste material. Mineral production is given most frequently in units of weight while waste rock is often expressed in units of volume. It is common, for example, to refer to total annual production.

2. *Production rate:* The theoretical production volume or weight of a machine per unit of time. It is usually expressed on an hourly basis but can be given for other units of time such as a shift or a day.

3. *Productivity:* The actual production per unit of time when all efficiency and other management factors are considered. It can also be referred to as a net production rate, or production per unit of labor and time (e.g., tons/employee-shift).

4. *Efficiency:* The percentage of the estimated production rate that is actually handled by a machine. Reductions in production rate can be related to the machine itself, personnel, or job conditions. The efficiency factor can be expressed as the average number of minutes worked at full production in one hour divided by 60 minutes (Anon., 1975).

5. *Availability:* That portion of the scheduled operating time that a machine is mechanically ready to work (Anon., 1976).

6. *Utility:* That portion of the available time that a machine is actually working (Anon., 1976).

7. *Capacity:* Refers to the volume of material that a loading or haulage unit can hold at any point in time (e.g., volume of a loading machine bucket or a truck bed). Capacity can be classified according to the following two types:

a. Struck capacity: The volume of material in a loading or haulage unit when it is filled to the top, but with no material above the sides or carried on any external attachments such as bucket teeth (Peurifoy, 1956).

b. Heaped capacity: The maximum volume of material that a loading or haulage unit can handle when the material is heaped above the sides. While the struck capacity is a constant for any unit, the heaped capacity is a function of the material properties and the shape of the unit (Peurifoy, 1956).

8. *Rated capacity:* The load that a machine can carry in terms of weight. Most machines are designed to carry a particular weight rather than volume. Therefore, the volume of material handled will be dependent upon the density of the material and will vary with density for a given machine while the maximum weight is a constant and is a function of the strength of the machine components.

9. *Swell factor:* The fractional increase in material volume that occurs when it is fragmented and removed from its natural state (bank volume) and deposited in a loosened state (loose volume). It can be expressed as either a decimal fraction or as a percentage.

10. *Bucket fill factor:* An adjustment to the capacity of a loading machine bucket. It is expressed generally as a decimal and corrects the capacity of the bucket to the actual volume of material it moves by taking into consideration the material heaping characteristics, the angle of repose, and the skill of the operator in filling the bucket (Anon., 1976).

11. *Cycle:* As mining is described generally as a *cycle of unit operations* (Chapter 9.4), so each unit operation is generally cyclical in nature. The unit operations of loading and haulage can be divided into an orderly rotation of steps or suboperations. For example, the most common components of a discrete unit haulage cycle are load, haul, dump, and return. From an equipment selection or production planning standpoint, the duration of each component is of primary importance. The sum of the time durations for one complete cycle is called the *cycle time.*

The objective of the *equipment selection process* for loading and haulage is quite simple. It is to select a single machine or combination of machines that is capable of moving a specified amount of material over a known distance within a given period of time. While minimization of cost may be stated as a goal, most designs are based upon a minimum acceptable rate of return so that estimated costs must not exceed those that would result in a rate of return less than the established minimum. Following is a generalized step-by-step selection process. This is presented as a preface to the more machine-specific production calculations given in the next subsection. The basic steps are as follows.

1. *Determine required production:* The total production requirement can be affected by a number of factors that are external to a particular mine. These factors include sales projections, contracts, amount of available reserves, and other operations of the company in question. Based upon consideration of these factors, management must make a decision concerning the total amount of mineral to be produced. Production requirements are stated generally for an entire year.

The total annual production requirement for the mine must then be converted into daily or hourly production rates for each operation. Production rates for specific operations within the mining cycle will be affected by such factors as percentage recovery, ore grade, and stripping ratio in the case of surface operations. For example, as the stripping ratio increases in a surface coal mine, the overburden removal rate must increase proportionally to maintain the same coal production rate. Therefore, loading and haulage production rates must be determined for the mineral commodity and for waste rock when its removal is necessary for mineral extraction.

2. *Determine reach or haul path:* Fixed-base equipment load at one location and then swing in an arc to dump at a second location. The maximum horizontal distance over which a machine can either load or dump is referred to as its *reach.* The geometry of the deposit to be excavated is the primary factor in determining the required reach of a machine.

Haul path refers to the distance and grade over which mobile equipment must travel. For both mobile haulers and combined mobile loader-hauler, units there is some distance to be traversed between the loading point and the dumping point. However, this distance is not necessarily represented by a straight line. In underground operations, the haul path is constrained mainly by the geometry of the mine openings, but can also be affected by such factors as ventilation and power availability. Topography is the primary consideration in determining surface haul paths. However, property boundaries and legal right-of-ways may also be involved.

3. *Calculate cycle time:* The cycle time for a unit operation can be divided into two primary components. The first component consists of those tasks that have a relatively constant duration from one application to the next. This includes such tasks as turning, spotting, dumping, and loading. Estimates of the time required for each of these fixed components of the cycle are generally available from equipment manufacturers. These estimates are based upon experience and are given for specific equipment models operating over a range of job conditions. The variable component of the cycle time is associated with the travel time for mobile equipment and the swing time for fixed-base equipment. Swing time is controlled primarily by the swing angle. Travel time for mobile equipment is much more variable. It is dependent not only upon the haul distance but also the geometry of the haul path and the speed of the vehicle, which is itself a function of available power, total resistance, and load.

4. *Calculate capacity:* The general relationship between production rate, cycle time, and capacity is quite simple and can be stated as

$$\text{production rate} = \text{capacity} \times (\text{no. of cycles/unit time}) \tag{9.3.1}$$

When all efficiency factors are considered,

$$\text{productivity} = \text{production rate} \times \text{efficiency factors} \tag{9.3.2}$$

The calculation of required capacity is rather straightforward once production requirements have been established and estimates have been made for cycle times and all efficiency factors. It is important to remember, however, that machines must be designed to handle a specified weight rather than a particular volume. Therefore, it is necessary to consider the rated capacity of a machine along with the density of the material to be certain that the required production can be achieved with the equipment selected.

5. *Iterate to improve productivity:* In step 3 above, certain assumptions had to be made about the class of machinery to be used so that a cycle time could be estimated based upon the determined haul path. The capacity calculated in step 4 may not be achievable with the class of machinery assumed in step 3. This would then require an adjustment in the cycle time, which would in turn require a further refinement in the design capacity. Several iterations may be necessary before a satisfactory solution is achieved. There can be, however, more than one equipment solution to the loading and haulage problem. Knowing that

production rate is directly proportional to capacity and inversely proportional to cycle time, the engineer may perform several iterations of this process in order to produce a number of loading and haulage alternatives for cost comparison.

6. *Calculate fleet size:* Up to this point, the discussion of equipment selection has assumed either a single loader-hauler unit or a single loader paired with a single haulage unit. However, it may not be possible or desirable to supply the entire required capacity in this manner. Although there is an economy of scale (i.e., the unit cost of material handled tends to decrease as the capacity of the machine increases), this gain must be weighed against the uncertainty associated with the availability of a single machine. Whereas a fleet can continue to produce when one machine is unavailable, a single-machine operation is idled when the machine is mechanically unavailable. Various algorithms exist for calculating the probability of any number of machines being available at a point in time for a given fleet size (Anon., 1979). The total number of machines needed to satisfy the production requirement can then be determined using a probabilistic cumulative availability.

7. *Iterate to reduce owning and operating costs:* The technical phase of the selection process will most likely identify a number of feasible loading and haulage system alternatives. These alternatives may include a single machine, a pair of machines working in tandem, or a fleet of machines. A cost comparison should be performed to evaluate the total cost per unit of production considering differences in capital costs, operating costs, and the estimated life of the equipment. An additional step beyond traditional engineering economic analysis involves computer simulation of the various systems under consideration. Simulation attempts to deal with many of the assumptions that are inherent in the deterministic equipment selection algorithms and to provide production and cost estimates that are less approximate. These techniques not only facilitate a straightforward comparison of alternatives but also assist in identifying potential changes in the system design that can result in better alternatives. This subject is the primary focus of Chapter 9.4.

9.3.4 LOADING AND HAULAGE PRODUCTION CALCULATIONS

The subject of production calculations has been addressed extensively by various authors and equipment manufacturers. Several of these earlier works are referenced extensively in the following segment. Also, rather than repeating similar steps in each group of calculations, the reader is referenced to the preceding or following discussion to a type of calculation that is similar.

For additional applications coverage, refer to Chapters 13.3 and 17.2.

9.3.4.1 Production Capacity of Discrete Unit Loaders

Discrete unit loaders can be divided into two classes: those that require no tramming and those that require minimal tramming. Discrete unit loaders that require no tramming include backhoes, hydraulic excavators, mining (or loading) shovels, and small draglines that are used to load-haulage vehicles. Those that require minimal tramming include front-end loaders (FELs) and load-haul-dumps (LHDs). The segment dealing with loaders that require no tramming is extracted from the *Caterpillar Performance Handbook* (Anon., 1990), and the subsection on loaders requiring minimal tramming is from Haley (1973).

Loaders That Require No Tramming. The hydraulic excavator (Fig. 9.3.1) is typical of discrete unit loaders that do not require tramming. As with any other piece of materials handling equipment, hydraulic excavator earthmoving production is dependent on average bucket payload, average cycle time, and job efficiency. If an estimator can accurately predict excavator cycle time and bucket payload, a machine's earthmoving production can be derived from the following formulas:

$$\text{yd}^3\,(\text{m}^3)/\text{60-min hr} = \text{cycles}/(\text{60-min hr} \times \text{avg. bucket payload in yd}^3\,(\text{m}^3) \tag{9.3.3}$$

$$\text{yd}^3\,(\text{m}^3)/\text{60-min hr} = \frac{\text{60-min/hr}}{\text{cycle time} - \text{min}} \times \text{avg. bucket payload in yd}^3\,(\text{m}^3) \tag{9.3.4}$$

$$\text{avg. bucket payload} = \text{heaped bucket capacity} \times \text{bucket fill factor} \tag{9.3.5}$$

$$\text{actual yd}^3\,(\text{m}^3)/\text{hr} = \text{yd}^3(\text{m}^3)/\text{60-min hr} \times \text{job efficiency factor} \tag{9.3.6}$$

Bucket Payload—An excavator's bucket payload (actual amount of material in the bucket on each digging cycle) is dependent on bucket size, shape, curl force, and certain soil characteristics (e.g., the fill factor for that soil). Fill factors for several types of material are listed in Table 9.3.2.

$$\text{average bucket payload} = \text{heaped bucket capacity} \times \text{bucket fill factor} \tag{9.3.7}$$

Estimating Cycle Time—The digging cycle of the excavator is composed of four segments:

1. load bucket
2. swing loaded
3. dump bucket
4. swing empty

Total excavator cycle time is dependent on machine size (small machines can cycle faster than large machines) and job conditions. With excellent job conditions, the excavator can cycle quickly. As job conditions become more severe (tougher digging, deeper trench, more obstacles, etc.), the excavator slows down accordingly. As the soil gets harder to dig, it takes longer to fill the bucket. As the trench gets deeper and spoil pile larger, the bucket has to travel farther and the upper structure has to swing farther on each digging cycle, and so on.

Spoil pile or truck location also affects cycle time. If a truck is located on the floor of the excavation beside material being moved, 10- to 17-sec cycles are practical. The other extreme would be a truck or spoil pile located above the excavator 180° from the excavation.

The cycle time estimating chart (Fig. 9.3.2) outlines the range of total cycle time that can be expected as job conditions range from excellent to severe. Many variables affect how fast the excavator is able to work. The chart defines the range of cycle times frequently experienced with a machine and provides a guide to what is an "easy" or a "hard" job. The estimator can then evaluate the conditions of the job and use the cycle time estimating chart to select the appropriate range. A practical method of further calibrating the chart is to observe excavators working in the field and correlate measured cycle time to job conditions, operator ability, etc.

Fig. 9.3.1. Hydraulic excavator front shovel (*courtesy:* Caterpillar Inc.).

Table 9.3.2. Bucket Fill Factors

Material	Fill Factor Range (Percent of heaped bucket capacity), %
Moist loam or sandy clay	100–100
Sand and gravel	95–110
Hard, tough clay	80–90
Rock—well blasted	60–75
Rock—poorly blasted	40–50

Source: Anon., 1988.

The same factors must be considered in production calculations for loading shovels, backhoes, and small draglines used for loading. Example 9.3.1 illustrates the selection of loading shovels for an open-pit mine (Anon., 1979).

Example 9.3.1. Select the bucket size for a fleet on mining shovels at an iron ore operation given the following assumptions about the operation.

Daily required capacity/machine (3 machines)	32,700 tpd
Estimated daily operating time	17.02 hours
Diggability rating	very hard digging
Estimated work cycle	37 seconds
Material bulk weight	6000 lb/yd^3
Swell factor	0.60
Dipper fill factor	0.80

Solution. Conventional double backup loading planned; therefore, a standard boom length is satisfactory.

$$\text{Shovel cycles per day} = \frac{\text{operating hr} \times 3600}{\text{work cycle in sec}} = \frac{17.02 \times 3600}{37} = 1656 \text{ cycles/day}$$

Required material.

$$\text{Tonnage per cycle} = \frac{\text{daily tonnage}}{\text{no. of cycles}} = \frac{32{,}700}{1656} = 19.8 \text{ tons or } 39{,}600 \text{ lb}$$

$$\text{Dipper size} = \frac{\text{tonnage per cycle} \times 2000}{\text{bank density in lb/yd}^3 \times \text{swell factor} \times \text{dipper fill factor}} = \frac{39{,}600}{6000 \times 0.6 \times 0.8} = 13.8, \text{ say } 14 \text{ yd}^3$$

Loaders That Require Minimal Tramming: The output of a front-end loader (Fig. 9.3.3) or similar discrete unit loader requiring tramming is the product of the mass of material delivered per cycle × cycles per hour. Cycle times vary in accordance with job conditions and loader stability.

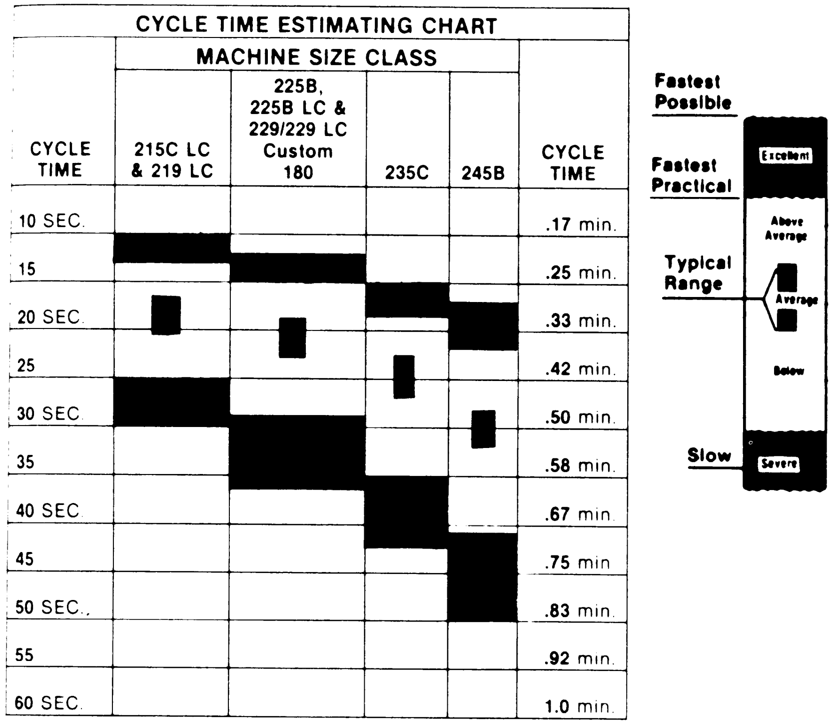
Material delivered per cycle reflects bucket size, carry characteristics (bucket fill factor), and weight of the material. A common error is assuming that a full-rated load will be delivered each cycle. In all cases, the density of the material in the loose state should be determined in lb/ft^3 (kg/m^3). Weighing a known sample volume by some means is recommended.

Rated (maximum) hourly production (lb or kg) = rated bucket size (yd^3 or m^3) × carry factor × material wt (lb/ft^3 or kg/m^3) × cycles per hr (9.3.8).

For longer periods, an overall efficiency factor also should be applied.

Bucket Size—Bucket capacities are determined in a manner similar to their determination in the previous section for nontramming loaders.

Loader Cycle Times—Loader cycle time is made up of load, haul, dump, and return. In any stockpile or bank work, these maneuvers require four reversals to direction. In loading trucks



CYCLE TIME -vs- JOB CONDITION DESCRIPTION

– Easy digging (unpacked earth, sand, gravel, ditch cleaning, etc.). Digging to less than 40% of machine's maximum depth capability. Swing angle less than 30°. Dump onto spoil pile or truck in excavation. No obstructions. Good operator.

– Medium digging (packed earth, tough dry clay, soil with less than 25% rock content). Depth to 50% of machine's maximum capability. Swing angle to 60°. Large dump target. Few obstructions.

– Medium to hard digging (hard packed soil with up to 50% rock content). Depth to 70% of machine's maximum capability. Swing angle to 90°. Loading trucks with truck spotted close to excavator.

– Hard digging (shot rock or tough soil with up to 75% rock content). Depth to 90% of machine's maximum capability. Swing angle to 120°. Shored trench. Small dump target. Working over pipe crew.

– Toughest digging (sandstone, caliche, shale, certain limestones, hard frost). Over 90% of machine's maximum depth capability. Swing over 120° Loading bucket in man box. Dump into small target requiring maximum excavator reach. People and obstructions in the work area.

Fig. 9.3.2. Cycle-time estimating chart for Caterpillar hydraulic excavators based upon machine size class and job conditions (*courtesy;* Caterpillar Inc.).

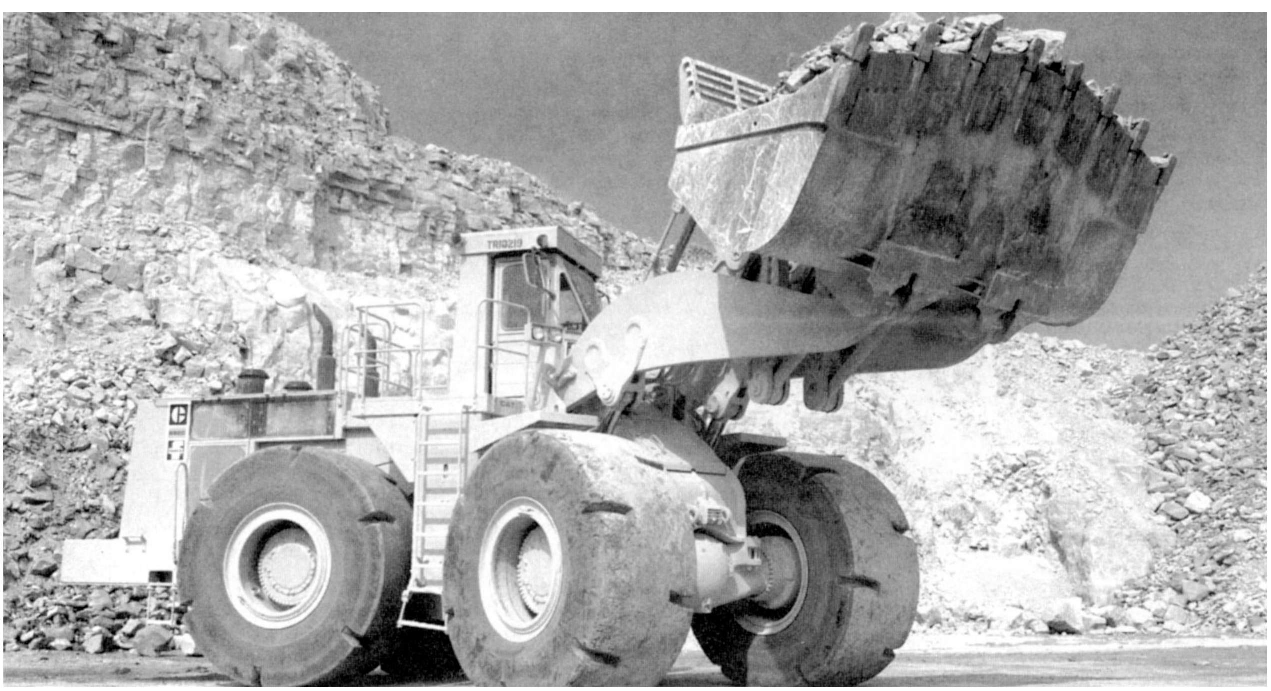


Fig. 9.3.3. Front-end wheel loader (*courtesy:* Caterpillar, Inc.).

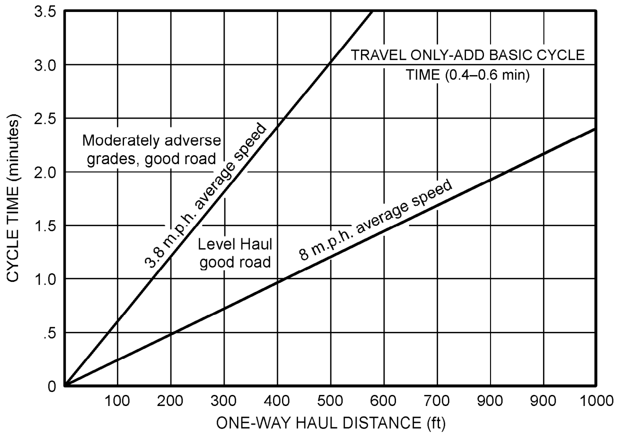


Fig. 9.3.4. Example of wheel loader haul and return estimating chart (Haley, 1973). Conversion factors: 1 ft = 0.3048 in., 1 mph = 1.609 km/h.

from a stockpile, the distance traveled is held to a minimum. Depending on the operator, material, truck size, and underfoot conditions, cycle time segments will vary upward from the following:

	Cycle time (min)
Load	0.06
Haul	0.15
Dump	0.05
Return	0.14
Total	0.40

On other than truck-loading applications, extended hauls usually are involved. Regardless of haul length, the turns and reversals of truck loading or basic maneuvers must be completed. Fig. 9.3.4 is an example of a guide for estimating load-and-carry cycle times. (Caution: the basic cycle time, usually 0.4 to 0.6 min, must be added.)

9.3.4.2 Production Capacity of Continuous Flow Loaders

Bucket wheel excavators (BWEs) are probably the predominant type of equipment in this class, but it also includes bucket chain excavators and bucket dredges.

The theoretical output of a BWE, as reported by Ramani et al. (1980), is based on the bucket size and the number of bucket discharges per minute. If I is nominal bucket capacity in yd^3 (m^3), Z is number of buckets in the wheel, V_1 is cutting speed of the wheel in ft/sec (m/s) and D is diameter of the wheel in ft (m), then

$$S_s = V_1 \times Z/\pi D \qquad (9.3.9)$$

where S_s = number of bucket discharges per second

$$Q_t = I \times S_s \times 3600 \qquad (9.3.10)$$

and Q_t = theoretical capacity of the excavator in yd^3/hr (m^3/h).

As can be seen from these equations, the number of bucket dischargers is dependent on the peripheral speed. The peripheral speed of a bucket wheel is limited by the ability of the wheel to discharge its bucket content on the chute against the counteracting centrifugal force. In theory, the maximum peripheral speed must be such that the bucket discharge will be ensured.

Mathematically,

$$M \times g = M \times V_1^2/R \qquad (9.3.11)$$

where M is mass of material in the bucket, lb (kg), R is in radius of the wheel in ft (m), and g is acceleration due to gravity, ft/sec^2(m/s^2), which yields the following expression:

$$V_1 = \sqrt{g \times R} = V_{\max} \qquad (9.3.12)$$

Practical values of speed lie between 0.4 to 0.6 $V_{\max}$. To keep the wear on the bucket's cutting knives or teeth at a minimum, speeds do not exceed 17 fps (5 m/s). The peripheral speed selected will also depend greatly on the nature of material to be excavated. In principle, however, a higher peripheral speed will be decided upon if hard material is to be cut, in which case maximum output may not be attained. Based on a constant output, the doubling of the peripheral speed will halve the amount of material excavated by each bucket; thus cutting performance will be reduced.

Yet another factor that affects the output of a BWE is the bucket filling capacity. It has been found that, in hard ground, bucket filling is around 30 to 40% of the nominal capacity. The relationship between digging resistance and the hourly capacity of the BWE is

$$Q_1/Q_2 = k_2^2/k_1^2 \qquad (9.3.13)$$

where Q_1 is BWE hourly capacity in yd^3 (m^3) in soil with specific cutting resistance k_1, and Q_2 is BWE hourly capacity in yd^3 (m^3) in soil with specific cutting resistance k_2. Thus, the actual capacity of the BWE in any soil is given by

$$Q_a = I \times B_f \times S_s \times 3600 \qquad (9.3.14)$$

where B_f is bucket filling capacity in the soil expressed as a fraction of the nominal bucket capacity, S_s is number of bucket discharges per second, and Q_a is actual capacity of the BWE in yd^3/hr (m^3/h). In soils with high cutting resistance, higher cutting speeds with lower bucket filling will result in a very small Q_a as compared to Q_t. The ratio may be as small as 0.2. One can visualize that in hard soils the excavating operation of the BWE has changed to a milling operation.

Example 9.3.2. A bucket wheel excavator has eight buckets with a nominal capacity of 2 m^3/bucket. The wheel has a diameter of 15 m and operates at a speed of 0.4 m/s. In a material with a digging resistance of 25 kg/cm, the BWE is producing 400 m^3/h. If the speed of the bucket does not change, what would be the bucket-fill factor (i.e., fraction of nominal bucket capacity) for the buckets when cutting a material with a digging resistance of 45 kg/cm?

Solution. The production rate in the new material can be determined as follows (Eq. 9.3.13):

$$Q_2 = Q\,\frac{k_1^2}{k_2^2} = 400\,\frac{25^2}{45^2} = 123.5 \text{ m}^3/\text{h}$$

The production rate can also be expressed as

$$Q = I \times B_f \times S_s \times 3600$$

Table 9.3.3. Bench Height/Wheel Diameter Factor for BWE Power Requirement Calculation

Height/diameter ratio	0.1	0.2	0.3	0.4	0.5	0.6	0.67	0.7	
Value of C									
English Units ($\times 10^4$)		3.06	2.57	2.30	2.11	1.96	1.85	1.78	1.74
SI Units		295	248	222	203	189	178	171	168

Source: Ramani et al., 1980.

Solving for B_f,

$$B_f = \frac{Q}{I \times S_s \times 3600}$$

but, $S_s = \dfrac{VZ}{\pi D}$, therefore,

$$B_f = \frac{Q}{I \times \dfrac{V \times Z}{\pi D} \times 3600} = \frac{123.5}{2 \times \dfrac{0.4 \times 8}{\pi \times 15} \times 3600}$$

$$= 0.252 \text{ or } 25.2\%$$

The required cutting power depends on the type of ground, the cross-sectional area and the shape of the slice cut, the configuration and sharpness of the cutter blades, the shape of the teeth, and the cutting speed. Hard ground requires a high specific cutting force, high cutting speeds, and additional cutting blades between the buckets. Reducing the output and the rate of swing decreases the slice cross section and increases the specific cutting force for a given wheel drive rating. This will be clear from the following expression (Rasper, 1964):

$$N_G = \frac{K}{\eta \times C}\sqrt{Q \times S \times R} \tag{9.3.15}$$

where N_G is power required for cutting in hp(kW), Q is actual digging capacity in yd^3/hr (m^3/h), S is number of bucket discharges per min, R is radius of the cutting wheel in ft (m), C is a constant depending on the bench height/wheel diameter ratio (for the cutting height = 2/3 wheel diameter, its value is 171), η is efficiency of the wheel drive, and k is specific cutting force in lb/ft (kg/cm).

The change in the value of C with a change in the height/diameter ratio can be observed in Table 9.3.3 (Rasper, 1964).

9.3.4.3. Carrying Capacity of Discrete Unit Haulers

Discrete unit haulers fall into two categories: those that follow a fixed path and those that are free to move about in any direction. The nonfixed-path haulers are characterized primarily by trucks. Electric shuttle cars (nonbattery) can also be included in this category, even though their travel is restricted by a trailing cable. The cycle time calculations presented in this segment emphasize gradeability and engine performance, which are issues relating particularly to truck haulage. Three references are excerpted extensively in this segment: Peurifoy (1956), the Bucyrus-Erie Open Pit Training Program (Anon., 1979), and Bishop (1972). The fixed-path production calculations address primarily rail haulage, Bise (1986) is referenced in this segment.

Fig. 9.3.5. Typical rear-dump mine haul truck (*courtesy:* Caterpillar Inc.).

Mobile Haulers (Nonfixed Path): The productive capacity of a truck (Fig. 9.3.5) or wagon, as discussed by Peurifoy (1956), depends on the size of its load and the number of trips it can make in an hour. The size of the load can be determined from the specifications furnished by the manufacturer. The number of trips per hour will depend on the weight of the vehicle, the horsepower of the engine, the haul distance, and the condition of the haul road.

Bucyrus-Erie Co. (Anon., 1979) describes three different ways in which this productivity can be calculated with each method having a distinct meaning or use:

1. *Theoretical Productivity:* The tons (kilograms) or cubic yards (cubic meters) per hour produced by an operating unit if no delays were encountered. This indicates 100% potential, which is rarely, if ever achieved.

$$\text{tons (kg) per hr, tph (kph)} = \frac{60}{\text{(cycle time)}} \times \text{(truck rating)} \tag{9.3.16}$$

$$\text{bank yd}^3\ (\text{m}^3) \text{ per hr, bcy/hr (bcm/hr)} = \frac{60 \times \text{(truck rating)}}{\text{(cycle time)} \times \text{(swell factor)} \times \text{(density)}} \tag{9.3.17}$$

2. *Average Productivity:* The tons (kilograms) or cubic yards (cubic meters) per hour produced by an operating unit, taking into account fixed and variable delays. This rate should be applied to the desired period of time (shift, day, etc.) to estimate total production:

Table 9.3.4. Turning-Spotting-Dumping Time (min)

Operating conditions	Bottom-dump tractor-trailer	Rear-dump	Side-dump semitrailer
Favorable	0.3	1.0	0.7
Average	0.6	1.3	1.0
Unfavorable	1.5	1.5–2.0	1.5

Source: Bishop, 1972.

$$\text{tph (kph)} = \frac{(U - D) \times 60 \times E \times TR}{U \times C} \quad (9.3.18)$$

$$\text{bcy/hr (bcm/h)} = \frac{(U - D) \times 60 \times E \times TR}{U \times C \times SF \times M} \quad (9.3.19)$$

where U is unit of time (say, 8 hr), D fixed delays (hr), E is job efficiency (adjusts for variable delays), TR is truck rating (tons or kg), C is cycle time (min), M is material density (tons/bank yd^3 or kg/bank m^3), and SF is swell factor.

3. *Peak Productivity per Hour:* The tons (kilograms) or cubic yards (cubic meters) per hour produced by an operating unit taking into account variable delays only. This rate would be used to determine the number of haulage units to be assigned to a shovel to achieve a given required production.

$$\text{tph (kph)} = \frac{60 \times \text{(job efficiency)} \times \text{(truck rating)}}{\text{(cycle time)}} \quad (9.3.20)$$

$$\text{bcy/hr (bcm/hr)} = \frac{60 \times \text{(job efficiency)} \times \text{(truck rating)}}{\text{(cycle time)} \times \text{(swell factor)} \times \text{(density)}} \quad (9.3.21)$$

Haulage Cycle Time—The information in this segment, taken from Bishop (1972), is intended as a guide in the determination of the time required by the vehicle for its complete haulage cycle, including loading, hauling, dumping, returning, and spotting times.

1. **Loading Time:** There are several methods for determining the number of shovel, dragline, or loader passes per load and the resulting load time. One of the more simple and reasonably accurate methods is as follows:

Weight per yd^3 (m^3) of material (loose) = weight in bank × swell factor (9.3.22)

No. tons (kg) per pass = bucket capacity (yd^3 or m^3) × fill factor × loose weight per yd^3 (t) or m^3 (kg) (9.3.23)

No. passes per load = rate capacity of haulage unit (tons or kg) /ton (kg) per pass (9.3.24)

Load time (min) = no. passes × excavator cycle time (min) (9.3.25)

The answer to the number of passes is rounded off to the next largest whole number, and generally the potential excess load is ignored in the production study. Three passes are generally considered to be the minimum, and five passes are optimal for most types of operations. Care should be taken to insure that the same units are used for both vehicle rating and bucket loading purposes in the above computation.

2. **Turn, Spot, and Dump Time:** Turning, spotting, and dumping time depends upon the type of unit and the specific operating conditions. As a guide, average values for the different types of haulage units under the indicated conditions are shown in Table 9.3.4.

Table 9.3.5. Spot at Loading Machine (min)

Operating conditions	Bottom-dump tractor-trailer	Rear-dump	Side-dump semitrailer
Favorable	0.15	0.15	0.15
Average	0.50	0.30	0.50
Unfavorable	1.00	0.50	1.00

Source: Bishop, 1972.

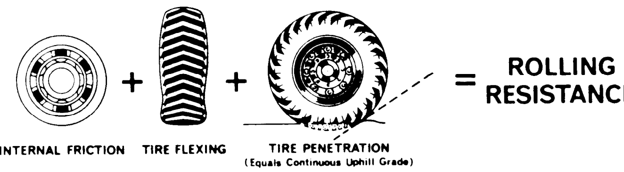


Fig. 9.3.6. Rolling resistance created by tire penetration, tire flexing, and internal resistance in final drive set. Indicated as pounds pull per ton of vehicle weight (Bishop, 1972). Conversion factor: 1 lb/ton = 0.5 g/kg.

3. **Spot Time at Loading Position:** The spotting time at the loading position also depends upon the type of haulage unit and the operating conditions. The average values are given in Table 9.3.5.

Haphazard spotting at the shovel is one of the most common and wasteful malpractices of the shovel cycle. Cutting a 30° arc from the swing out and return (only $^1/_{12}$ of the full 360° circle) will increase shovel production by approximately 15%. If the haulage unit is spotted for loading on a 180° swing when the job could be planned for 90°, nearly one-third of the possible production is lost.

In addition, accurate spotting of trucks at a correct angle and in the same relative position considerably improves the operator's timing and speeds up shovel operation. Trucks always should spot on the same radius to save racking the dipper in and out. This should line up as closely as possible with the radius of dipper as it leaves the bank.

4. **Haul Time:** *Gradeability* can be defined as the ability of a vehicle to negotiate a given grade, taking into account both grade and rolling resistance. The sum of these two values is expressed as "total resistance in percent of vehicle weight."

Grade resistance is defined as the rimpull or tractive effort required to overcome gravity in propelling a vehicle up an incline. It amounts to 20 lb/ton (10 g/kg) or 1% of unit weight for each percent of grade. For example, a 5% grade would offer a resistance of 100 lb (50 g) for each ton (kg) of vehicle weight.

Rolling resistance is the amount of rimpull or tractive effort required to overcome the retarding effect between the tires and the ground. It includes the resistance caused by the tire penetration into the ground, by the flexing of the tires under the load, and (to a degree) by the friction in the wheel bearings (Fig. 9.3.6). It is normally expressed as pounds pull per ton of vehicle weight, or as a percentage of vehicle weight. For example, a common value used to cover a well-maintained, smooth, hard, dry dirt and gravel road is 40 lb/ton (20 g/kg), or 2%.

Rimpull is discussed by Peurifoy (1956). It is a term used to designate the tractive force between the rubber tires of driving wheels and the surface on which they travel. If the coefficient of traction is high enough to eliminate tire slippage, the maximum rimpull is a function of the power of the engine and the gear ratios between the engine and the driving wheels. If the driving

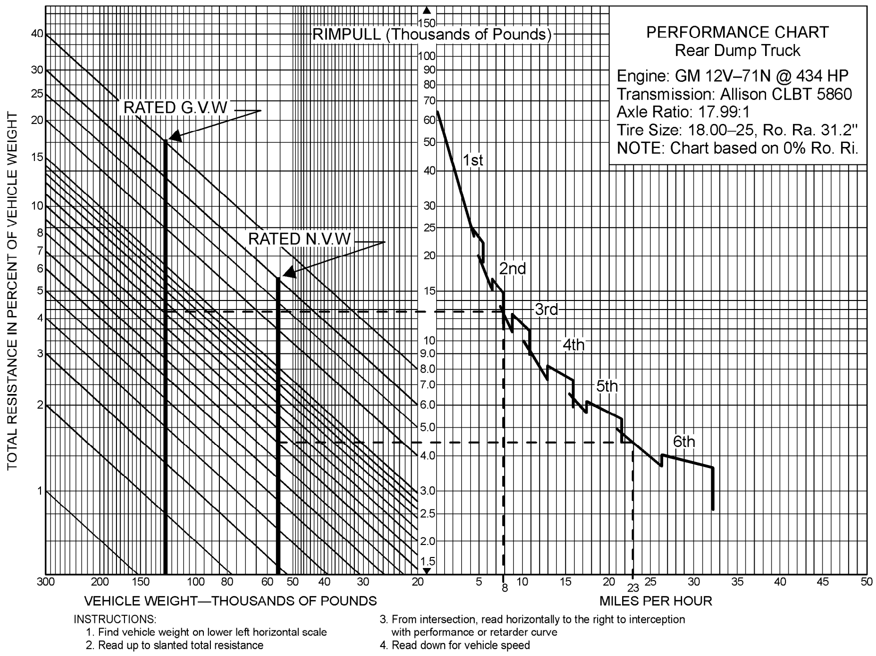


Fig. 9.3.7. Typical performance chart covering a 35-ton capacity rear-dump truck equipped with 434-hp engine, 6-speed powershift transmission, and 18.00-25 (32) tires (Bishop, 1972). Conversion factors: 1 lb = 0.4536 kg, 1 mph = 1.609 km/h.

wheels slip on the haul surface, the maximum effective rimpull will be equal to the total pressure between the tires and the surface multiplied by the coefficient of traction. Rimpull is expressed in pounds (kilograms).

If the rimpull of a vehicle is not known, it may be determined from the formula

$$\text{rimpull (lb)} = \frac{375 \times \text{hp} \times \text{efficiency}}{\text{speed (mph)}} \qquad (9.3.26)$$

or

$$\text{rimpull (kg)} = \frac{383 \times \text{power (kW)} \times \text{efficiency}}{\text{speed(km/h)}} \qquad (9.3.26a)$$

In computing the pull that a tractor can exert on a towed load, it is necessary to deduct from the rimpull of the tractor the tractive force required to overcome the rolling resistance plus any grade resistance for the tractor.

Bishop (1972), however, states that the use of this alone will not measure the performance or gradeability of the vehicle. To achieve this, one must have engine performance, gear ratios, tire data, weights, etc. This information is not normally available, so the engineer, in order to determine the speed a vehicle will negotiate a particular grade having a certain rolling resistance (total resistance), will have to refer to the vehicle manufacturer's performance chart. Fig. 9.3.7, showing a representative performance chart, has been marked to show the possible speed of the loaded vehicle operating on an adverse 8% grade having a 2% rolling resistance (10% total resistance). It will be noted when following a path of dotted (---) lines that the loaded unit can climb the 8% grade at a speed of 8 mph (12.9 km/h) in 2nd gear lock-up. To illustrate possible empty vehicle performance, the chart has been marked to show operation on a 6% grade with a 2% rolling resistance (8% total resistance). It will be noted that the empty unit can negotiate this grade at a speed of 23.5 mph (37.8 km/h) in 6th gear converter.

Most haulage units today are fitted with an auxiliary braking arrangement usually referred to as a "retarder." This is a device designed to serve as a brake working through the drive line to hold back the vehicle on downgrade haul roads. The use of the retarder saves the regular service brake for normal or emergency stops and reduces brake maintenance costs. Fig. 9.3.8, showing a representative retarder chart, has been marked to show the downgrade retarding ability of the loaded vehicle on an 8% grade, having a 2% rolling resistance. In this case, however, the rolling resistance serves to hold the vehicle back and thus, the total grade resistance would be 8% – 2% = 6%.

Following the dotted (---) lines in a similar manner to that of the performance chart, it is noted that the retarder has the capability of holding the loaded vehicle to a maximum speed of about 24 mph (38.6 km/hr). This, undoubtedly, is too fast for a normal 8% haul, but by shifting back to 4th gear, a slower speed of about 18 mph (29 km/h) can be achieved. As a further example, the chart has been marked to show that the retarder has the capability of controlling the empty vehicle on a 15% grade at speeds up to a maximum of 24 mph (38.6 km/h).

Having selected a transmission gear or range from the performance charts, it is necessary to modify the indicated road speeds through the use of a speed factor to reflect an average rather than a maximum speed. Table 9.3.6 shows the factors required to establish these average travel speeds over various lengths of haul.

A unit having a "stick shift" transmission and clutch cannot accelerate as rapidly as one with the torque convertor and full powershift transmission. As a result, the lower speed factors, at least up to 1500 ft (450 m) distances, should be considered.

To determine the average travel speed, multiply the maximum attainable speed as indicated in the performance chart by the factor shown, unless safety or other considerations impose lower limits.

Knowing the average travel speed from the above, the travel time can be calculated by means of the following formula:

$$\text{Travel time in minutes} = \frac{\text{distance in feet}}{\text{speed in mph} \times 88} \qquad (9.3.27)$$

or

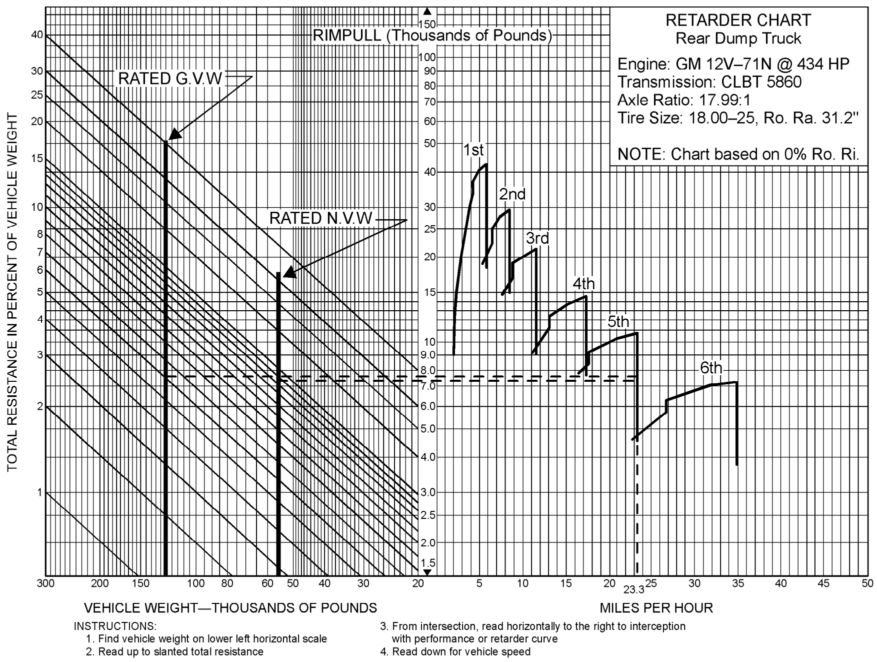


Fig. 9.3.8. Typical retarder chart covering a 35-ton capacity rear-dump truck equipped with 434-hp engine, 6-speed power-shift transmission, and 18.00-25 (32) tires (Bishop, 1972). Conversion factors: 1 lb = 0.4536 kg, 1 mph = 1.609 km/h.

Table 9.3.6. Speed Factors

Length of haul road section: ft	Length of haul road section: m	Short, level hauls 500–1000 ft (150–300 m) total length	Unit starting from stop	Unit in motion when entering haul road section
0–350	0–110	0.20	0.25–0.50	0.50–2.00
350–750	107–229	0.30	0.35–0.60	0.60–0.75
750–1500	229–457	0.40	0.50–0.65	0.70–0.80
1500–2500	457–762		0.60–0.70	0.75–0.80
2500–3500	762–1067		0.65–0.75	0.80–0.85
Over 3500	over 1067		0.70–0.85	0.80–0.90

Source: Bishop, 1972.

$$\text{Travel time in minutes} = \frac{\text{distance in meters}}{\text{speed in km/h} \times 16.7} \qquad (9.3.27a)$$

5. **Return Time:** Return time of the hauling unit is often governed by job conditions and safety precautions, rather than by the performance ability of the unit. If no grade conditions or operating hazards are present, these factors apply to the top speed empty:

	Under 500 ft (150 m)	Over 500 ft (150 m)
Favorable	0.65	0.85
Average	0.60	0.80
Unfavorable	0.55	0.75

If complicated grade conditions are present, use the high factors in Table 9.3.6.

Example 9.3.3. (Anon., 1975). An operator is determining which of two possible haul roads should be used:

1. The first alternative is a level roadway, 6000 ft (1830 m) long with a rolling resistance factor of 120 lb/ton and a coefficient of traction of 0.45.

2. The second alternative is a 3000-ft (914-m) roadway with a 5% adverse grade on the haul, and a rolling resistance factor of 200 lb/ton and coefficient of traction of 0.40.

The hauling unit is an off-highway truck with a gross vehicle weight of 130,000 lb; with 81,000 lb on the rear wheels when loaded to rated capacity.

Solution. Fig. 9.3.9 gives the appropriate rimpull curve for an off-highway truck and may be used where needed. Fig. 9.3.10 is the travel time.

Alternative 1:

Rolling resistance factor	= 120 lb/ton
	= 6% (20 lb/ton= 1%)
Grade resistance on haul	= 0%
Grade resistance on return	= 0%
Effective grade on haul	= 6% + 0%
	= 6%
Effective grade on return	= 6% = –0%
	= 6%

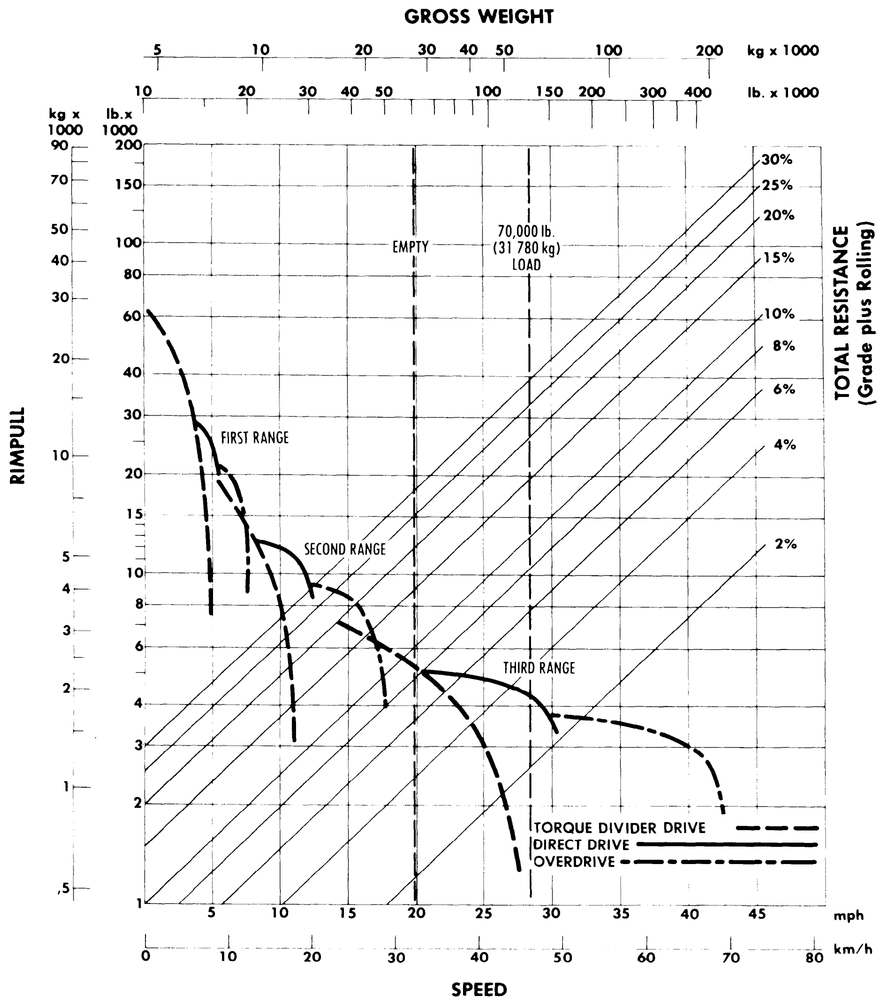


Fig. 9.3.9. Rimpull curve for off-highway truck used in Ex. 9.3.3. (*courtesy:* Caterpillar Inc.).

Gross vehicle weight = 130,000 lb
Power required = 130,000 lb × 6% = 7800 lb
Power available = 8000 lb at 16 mph (Fig. 9.3.9)
Power usable = 81,000 (weight on drive wheels) × 0.45 (coefficient of traction) = 36,450 lb
Vehicle will be able to haul under the given conditions

Haul time	4.70 min (Fig. 9.3.10)
Return time +	2.40 min (Fig. 9.3.10)
	7.10 min

Alternative 2:

Rolling resistance factor	= 200 lb/ton
	= 10% (20 lb/ton = 1%)
Grade resistance on haul	= 5%
Grade resistance on return	= −5%
Effective grade on haul	= 10% + 5% = 15%
Effective grade on return	= 10% − 5% = 5%
a) Gross vehicle weight	= 130,000 lb
Power required on the haul	= 130,000 lb × 15% = 19,500 lb
Power required on the return	= 130,000 lb × 5% = 6500 lb
Power available	= 20,000 lb at 7 mph (Fig. 9.3.9)

Power usable = 81,000 lb (weight on drive wheels) × .40 (coefficient of traction)= 32,400 lb
Vehicle will be able to haul under the given conditions

b) Haul time	5.50 min (Fig. 9.3.10)
Return time +	1.30 min (Fig. 9.3.10)
	6.80 min

Regardless of which path is used, the load, dump, and manuever times will remain the same. Therefore, the total of haul and return times will determine the path taken. It takes 0.3 minutes/cycle longer on the first alternative than on the second, which means that the contractor should choose the second to maximize production.

Discrete Unit Haulers With Fixed Paths: Productive capacity has the same form as mobile haulers with nonfixed paths, i.e.,

$$\text{Productivity} = \frac{(\text{rated capacity of hauler})\,(\text{efficiency factors})}{(\text{cycle time})} \tag{9.3.28}$$

The determination of cycle times involves quite specific calculations that are unique for each type of hauler. Rail haulage was addressed by Bise (1986), and portions of that discussion are included here to illustrate the selection of discrete unit haulers with fixed paths. Skip hoists are another type of discrete unit hauler that travel a fixed path (see Chapter 17.5).

The selection of mine locomotives for rail haulage centers on the weight and horsepower of the prime mover. Fig. 9.3.11

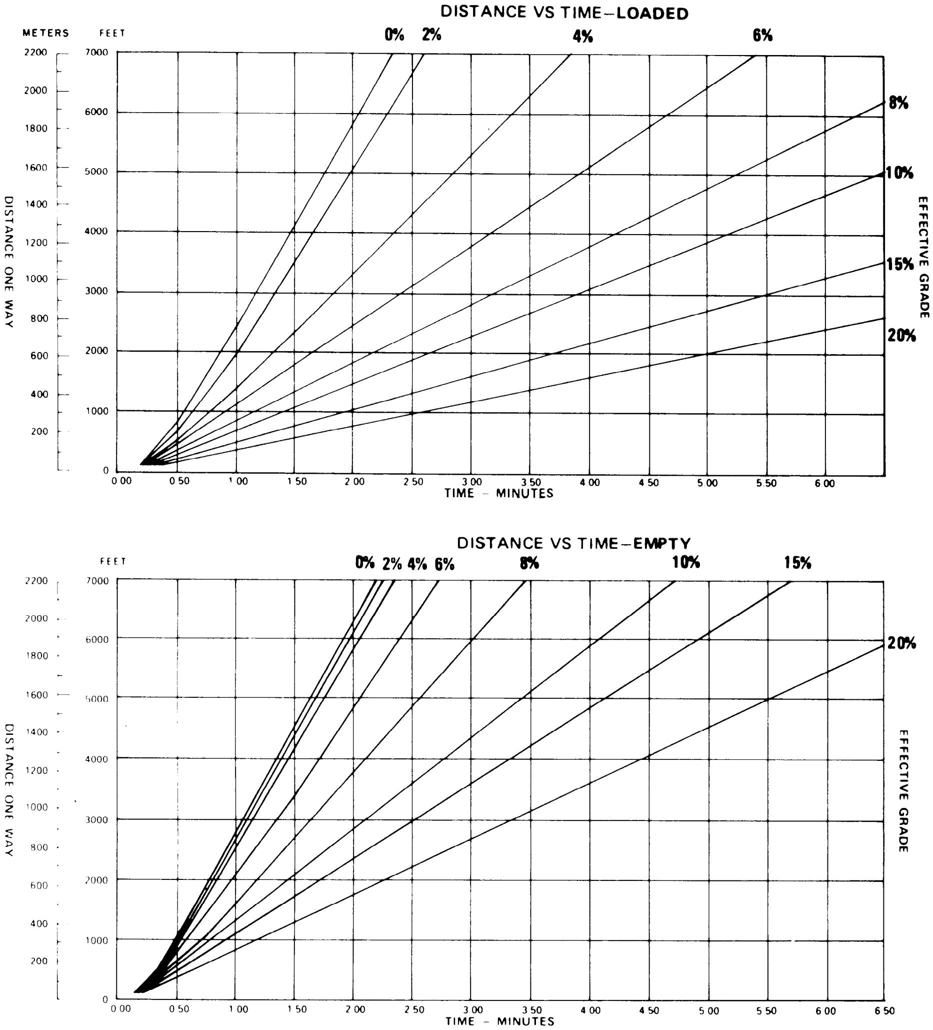


Fig. 9.3.10. Travel time curves for off-highway truck used in Ex. 9.3.3. (*courtesy:* Caterpillar, Inc.).

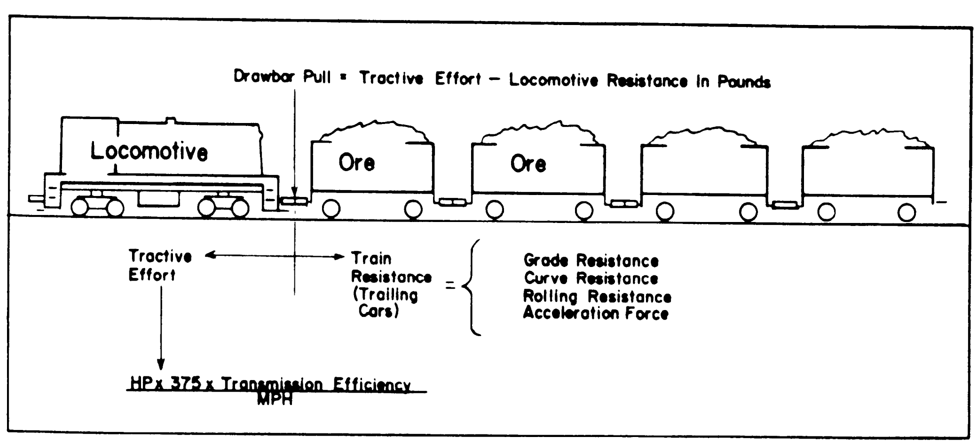


Fig. 9.3.11. Accelerating and resisting forces in railroad design (Brauns, 1973).

shows the relationship between a mine locomotive and its trailing loads. The *tractive effort,* exerted by the locomotive at the rim of its driving wheels, is a function of its weight and its ability to adhere to the track. *Adhesive values* for locomotives represent the portions of the prime mover's weight available to haul the trip. *Drawbar pull* is that portion of the locomotive's tractive effort that remains after the locomotive's resistance has been subtracted.

The locomotive's horsepower can be determined from the following relationship:

$$\text{power (hp)} = \frac{(TE)\,(V)}{(375)\,(E)} \tag{9.3.29}$$

$$\text{power (kw)} = \frac{(TE)\,(V)}{(383)\,(E)} \tag{9.3.29a}$$

where TE is the available tractive effort in lb (kg), V is the velocity in mph (km/h), and E is the transmission efficiency.

To move a trip, a mine locomotive must be capable of overcoming the following resistance for both itself and the mine cars: (1) rolling resistance, (2) curve resistance, (3) grade resistance, and (4) acceleration or deceleration.

The *rolling resistance* of the entire train is equal to the weight of the locomotive and mine cars (including the payload, if any) multiplied by a frictional coefficient. When roller bearings are used, the coefficient is 20 lb/ton (10 g/kg) of weight; the coefficient for plain bearings is 30 lb/ton (15 g/kg) of weight.

Curve resistance is a function of the radius of curvature and gage of the track, wheel base and diameter, speed, and load. It is often ignored because, in a properly installed system, it accounts for less than 1 lb/ton (0.5 g/kg) per degree of curve for that part of the train on the curve.

On a 1% grade, a ton (kilogram) must be raised 1 ft (m) for every 100 ft (m) that the train advances, making the grade resistance 20 lb/ton (10 g/kg) for each 1% of grade.

Tractive effort is also required to accelerate or decelerate. In mine locomotive calculations, it is assumed that it takes 100 lb (50 g) to accelerate each ton (kilogram) of the train's weight to achieve an acceleration rate of 1 mph/sec (1.6 km/h/s). Normally, mine locomotives accelerate between 0.1 (0.16) and 0.2 (0.32) mph/sec (km/h/s); therefore, the acceleration resistance is usually between 10 and 20 lb/ton (g/kg). Naturally, when the trip is decelerating, it can be analyzed in a similar manner.

9.3.4.4 Carrying Capacity of Continuous Flow Bulk Solids Conveyors

Two major types of continuous-flow bulk solids conveyors are addressed below: belt conveyors and screw conveyors. The carrying capacity of chain conveyors can be analyzed in much the same manner as belt conveyors. The calculations outlined in this segment are extracted from Brook (1971).

Belt Conveyers: The carrying capacity of the belt depends on how the material can be piled up on the belt width. As the belt is continuously passing over sets of supporting idlers, the material is slightly disturbed all the time and tends to spread out on the belt. The carrying capacity of the conveyor is given by the equation,

$$T = a\,b\,v \tag{9.3.30}$$

where T is carrying capacity, a is average cross-sectional area of material, b is bulk density, and v is speed of the conveyor belt.

For a belt of width W, the value of the area a varies approximately between $W^2/10$ and $W^2/12$, depending on the nature of the material. The value of cross-sectional area given by $W^2/10$ represents a high loading, but in some cases this should be considered, especially if it is possible to load the belt at this intensity by flood feeding, when finding the greatest force acting on the belting.

Example 9.3.4 (Brook, 1971). A belt 0.9 m wide conveys material of bulk density 1.35 t/m^3 at a speed of 1.75 m/s. What is the carrying capacity?

Solution. Referring to Eq. 9.3.30 and assuming $a = W^2/11$, then

$$T = \frac{(0.9)^2}{11}\ \text{m}^2 \times 1.35\ \text{t/m}^3 \times 1.75\ \text{m}$$

$$= \frac{0.81}{11} \times 1.35 \times 1.75\ \text{m}^2\ \frac{\text{t}}{\text{m}^3}\ \frac{\text{m}}{\text{s}}$$

$$= 0.1735\ \text{t/s}$$

$$= 0.1735\ \text{t/s} \times \frac{3600\ \text{s}}{1\ \text{hr}} = 625\ \text{t/h}$$

The speed of the belt is limited mainly by the accuracy of alignment possible. At fast speeds, any slight wandering of the belt is difficult to correct by the slight forward angling of the outer wing idler rollers, which depicts the angling greatly exaggerated, or by the angling of complete sets of idlers to try to counteract misalignments.

The belt strength affects the maximum force which can be taken by the belt, and the value of the maximum force depends on the power required and the drivehead frictional grip. The power required by a belt conveyor can be divided into three components:

1. Power for the empty belt, W_e.
2. Power to convey the material, W_m.
3. Power to raise the material, W_r.

The total power required, at the belt, is then $W_T = W_e + W_m \pm W_r$.

The value of W_r is written "plus or minus" as if the material is being lowered. This helps to run the belt, and the power requirement for this is thus negative. The power as calculated here is the power required at the driving drum of the conveyor, and so the motor power required will be greater because of power losses in the gearing at the drivehead.

Assuming an efficiency of 90% for this gearing, the motor power is then given by

$$W = W_T/0.9 \tag{9.3.31}$$

The power required to drive the empty belt depends on the total force required to move the empty belt and on the belt speed.

The force required

$$N_e = \text{total weight on idlers} \times \text{friction coefficient} = M_i g \mu_e \tag{9.3.32}$$

The friction coefficient of ball bearing idlers is very low, and a value of 0.03 is generally used. The total weight on the idlers depends on the mass of two lengths of belt and the mass of the idlers themselves. This is conveniently expressed as mass per unit length, and the value is the total for both strands of the conveyor and the associated idlers. The weight on the idlers for a conveyor of length l is then given by $m_i lg$ where m_i is the mass per unit length. The actual value of the mass per unit length can be obtained from manufacturer's data and varies with the design of conveyor. Belts that are to carry dense materials require close spacing of idlers, and this increases the weight on the idler bearings. The value of l, the length of the conveyor, is increased by l_x = 148 ft (45 m) to allow for end pulley friction, so that

$$N_e = m_i\,(l + l_x) g \mu_e \tag{9.3.33}$$

and the power required is

$$W_e = N_e v \tag{9.3.34}$$

or

$$W_e = m_i(l + l_x)g\mu_e v \tag{9.3.35}$$

The power required to convey the material can be calculated in a similar fashion:

$$W_m = m_m l\, g\mu_m v \tag{9.3.36}$$

The value of m_m, the mass of material per unit length, is obtained from

$$m_m = \frac{T}{v} \tag{9.3.37}$$

Substituting this in the formula for W_m gives

$$W_m = \frac{T}{v}\, l\, g\mu_m v \tag{9.3.38}$$

or

$$W_m = T\, l\, g\mu_m \tag{9.3.39}$$

The value of the friction coefficient μ_m is again 0.03 for well-maintained conveyors, but this is sometimes raised to 0.04 if the conditions are unfavorable. The simple length l is used and not $l + l_x$.

The power required to raise the material at the rate T through height h is obtained directly as

$$W_r = T\, g\, h \tag{9.3.40}$$

The effective belt tension, or the tension difference at the drivehead, can then be found from the total belt power W_t:

$$W_t = W_e + W_m \pm W_r \tag{9.3.41}$$

Effective tension $P_e = W_T/v$

The maximum tension P_1 is obtained from the formula $P_e = P_1 - P_2$. The value of P_2 is assumed to be just sufficient to prevent slip, and for no slip to occur:

$$\frac{P_1}{P_2} = e^{\mu\theta} = n \tag{9.3.42}$$

Slip is avoided in practice by using a value of n less than that possible by assuming a low value for m, the value of q being fixed by the arrangement of the drum or drums at the drivehead. The angle q is about 250° for a single-drum drive and 400° to 450° for a two-drum drive. The value of μ is assumed to be 0.25 for a rubber-covered conveyor belting on a steel drum and 0.35 for rubber on a lagged drum, the corresponding values for PVC-covered belting being 0.2 and 0.28, respectively.

For no slip, $P_1/P_2 = n$ at the limiting conditions, and therefore

$$P_2 = P_1/n, \text{ and } P_1 - P_2 = P_1/n = P_1(n - 1)/n$$

so that

$$P_1 = \frac{n}{n - 1} P_e \tag{9.3.43}$$

Screw Conveyors: The capacity of such a conveyor is

$$T = a\, b\, v \tag{9.3.44}$$

where a is the average cross section of material passing, and is given approximately by

$$a = \frac{k\pi d^2}{4} \tag{9.3.45}$$

where d is the diameter of trough and k is the loading factor.

The average speed of the material is v and is equal to about one pitch of the screw per revolution of the helix, despite the erratic mode of movement. For most types of conveyor, the pitch p is equal to the diameter d, so that if the rotary speed of the screw is n, usually measured in rev/sec or rev/min, the velocity v is given by $v = n\, p$, or approximately, $n\, d$.

The size of screw conveyors ranges from about 6 in. (150 mm) diameter to 30 in. (750 mm) diameter, and the value the loading factor k varies from 15 to 45% depending on the type of material to be conveyed. The rotary speed n varies between 50 and 100 rev/min.

Although a combination of smooth helix and a rough trough is the best for effective axial movement, the actual friction coefficients will be near to each other for material against the helix, and the material on the floor of the trough, where often a bed of small particles of the material builds up. To estimate the power required, the power to move the material can be calculated, and then multiplied by three to allow for paddle friction, bearing friction, etc., and the efficiency of the driving gear allowed for by dividing by 75%:

$$\text{Force required to move material} = \frac{T\, g\, l\, \mu}{v} \tag{9.3.46}$$

where T is the capacity, v is the speed of material, l is the conveyor length, and m is the friction coefficient. The motor power required is then

$$W = \frac{3\,(T\, g\, l\, \mu\, v)}{v\,(0.75)} = \frac{3\, T\, g\, l\, \mu}{0.75} \tag{9.3.47}$$

9.3.4.5 Carrying Capacity of Continuous Flow Fluid Transport

The carrying capacity of any hydraulic transport system is addressed by Brook (1971) and can be expressed as

$$T = a\, s\, v \tag{9.3.48}$$

where a is average cross-sectional area of solids, s is solids density, and v is fluid velocity.

If the volumetric concentration of solids is c, and the full area of the pipe of flume is A, then $c = a/A$. Strictly speaking c is the actual concentration, and not the delivered concentration, but unless the speed is very low and near to the speed that may cause blockage of the system, these two concentrations can be considered equal.

The velocity required for fluid transport is rather greater than the velocity required to cause the solids to "float" in an upward stream of fluid, and a useful practical guide is

$$v = K\sqrt{d\left(\frac{s - r}{r}\right)} \qquad (9.3.49)$$

where K is a constant dependent upon the particle size, d is the pipe diameter, s is the solids density, and r is the fluid density.

For pipe sizes above about 20 in. (0.5 m), the value of d should be kept as 20 in. (0.5 m).

For both suction and pressure systems, a pressure difference of $p = p_i - p_o$ must be overcome, where p_i is inlet pressure and p_o outlet pressure. The pressure difference may be due to fluid friction on the conduit walls, sliding friction of the solids is on the conduit walls, increase in potential energy of both fluid and solids, and may be affected by the conditions at inlet or outlet.

The power required is given by

$$W = p\,A\,v = pQ \qquad (9.3.50)$$

where $Q = Av$, which is the total volumetric flow.

This is the power required in the fluid, and if the pressure is generated by a pump or a fan (compressor), the power of the driving motor will also depend on the pump or fan efficiency, which is often about 75%, so that a typical value of overall efficiency of pump and motor is about 66.7%.

The four main components of the pressure differences are

1. Pressure due to fluid friction on walls.
2. Pressure due to sliding friction of solids.
3. Pressure required to increase potential energy of fluid and solids.
4. Pressure required to increase kinetic energy of fluid and solids.

Some of these components may be neglected in particular circumstances, but they may be important in other cases. Energy is also involved when temperature changes are considered, particularly with pneumatic transport, but in most cases temperature can be ignored.

The pressure difference to be overcome by the fluid is thus

$$p = p_f + p_s + p_p + p_k \qquad (9.3.51)$$

Comprised of these components:

1. pressure due to fluid friction:

$$p_f = \frac{fl}{m} \cdot \frac{rv^2}{2} \qquad (9.3.52)$$

where f is friction coefficient, l is conduit equivalent length, and m is hydraulic mean radius.

2. pressure due to sliding friction:

$$p_s = \mu\,k\,c\,l_h\,g(s - r) \qquad (9.4.53)$$

where m is coefficient of friction of the solids on the conduit, k is portion of weight actually touching the walls ($0.1 \le k \le 1.0$), and $l_h = l \cos q$ (q is pipe slope angle).

3. pressure to increase potential energy:

$$p_p = g\,h\,[r + c(s - r)] \qquad (9.3.54)$$

where h is increase in elevation over l.

4. pressure to increase kinetic energy:

$$p_k = \frac{v^2}{2}\,[r + c(s - r)] \qquad (9.3.55)$$

5. overall pressure difference:

$$p = p_i - p_o$$

where p_i is intake pressure and p_o is discharge pressure.

9.3.4.6 Production Capacity of Discrete Unit Combined Loader-haulers

Discrete unit combined loader-haulers can be further divided into mobile equipment and fixed-based equipment. Mobile equipment includes wheel tractor scrapers (Fig. 9.3.12) and both rubber-tired and track dozers. Fixed-base equipment includes draglines and stripping shovels that excavate material and place it in a final destination without moving the base of the machine.

Mobile Equipment: The hourly production of mobile equipment has been addressed by Caterpillar Inc. (Anon., 1975) and can be estimated using the following relation:

$$\begin{aligned} \text{production/hr} &= \frac{\text{loads}}{\text{cycle}} + \frac{\text{cycles}}{\text{hr}} \\ &\times \text{(job efficiency factor)} \\ &\times \text{(correction factors)} \end{aligned} \qquad (9.3.56)$$

Although the machine will change from one example to another, the following steps for estimating production will be the same.

1. *Machine Capacity.* The first step is to determine the machine capacity. This is expressed as "THE LOAD" per cycle. How much will the unit carry on each cycle? This will depend on the size of the bowl of the scraper. For this step, only the rated capacity of the scraper is needed. This can be found on the specification sheets.

2. *Cycle Time.* Next, calculate the cycle time of the machine. All cycle times have four parts: load, haul, dump, and return. By finding the cycle time, the number of cycles completed per hour can be determined.

a. *Load Time:* For wheel tractor scrapers, it normally ranges from 0.6 to 1.0 min depending upon bowl capacity, size of pusher, job conditions, single or tandem power, etc.

b. *Dump Time:* Maneuvering and dumping averages from 0.6 to 0.8 min.

c. *Haul Time:* Haul time will depend on weight carried, power available, tractive effort, effective grade, conditions of haul road, and distance moved.

Fig. 9.3.13 is a graph similar to those in the *Caterpillar Performance Handbook* (Anon., 1990). It is used to figure the travel time one-way with a loaded wheel tractor scraper of a particular size.

If the effective grade is known, Fig. 9.3.13 can be used.

effective grade = rolling resistance (%) + grade resistance (%). (9.3.57)

d. *Return Time:* When the unit returns, two things will be different from the haul. First, it will be empty, or weigh less. Second, it will have an opposite grade affect.

Fig. 9.3.12. Typical wheel tractor scraper (*courtesy:* Caterpillar, Inc.).

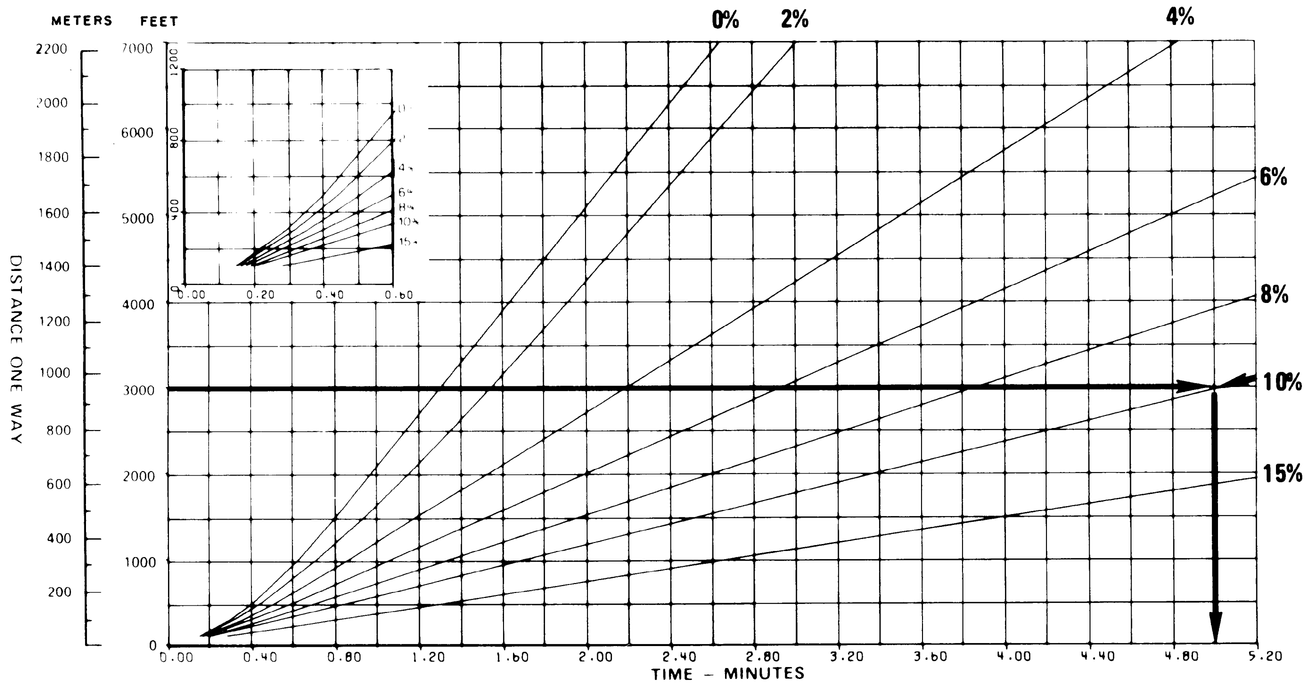


Fig. 9.3.13. Example of one-way travel time curve for a loaded wheel tractor scraper as a function of distance and effective grade (*courtesy:* Caterpillar, Inc.).

Retarder and Brake Performance—When operating on steep downgrades, wheel tractor scrapers may have to limit their operating speed for safety reasons. Operating speeds need to be limited in order that the braking or retarding capabilities of the machine are not exceeded. This will affect travel time. The maximum safe sustainable speed can be obtained by using performance curves like the one in Fig. 9.3.14.

To determine maximum safe speed: Read from gross vehicle weight down to the percentage effective grade. From this weight-effective grade point, read horizontally to the curve with the highest obtainable speed range, then down to maximum descent speed brakes can safely handle without exceeding cooling capacity.

Once the maximum safe speed is found, the travel time for the segment can be calculated using the formula,

$$\text{time (min)} = \frac{\text{distance (ft)}}{\text{speed (mph)} \times 88} \qquad (9.3.58)$$

or

$$\text{time (min)} = \frac{\text{distance (m)}}{\text{speed (km/h)} \times 16.7} \qquad (9.3.58a)$$

3. *Job Factors.* The last step is to consider the job efficiency

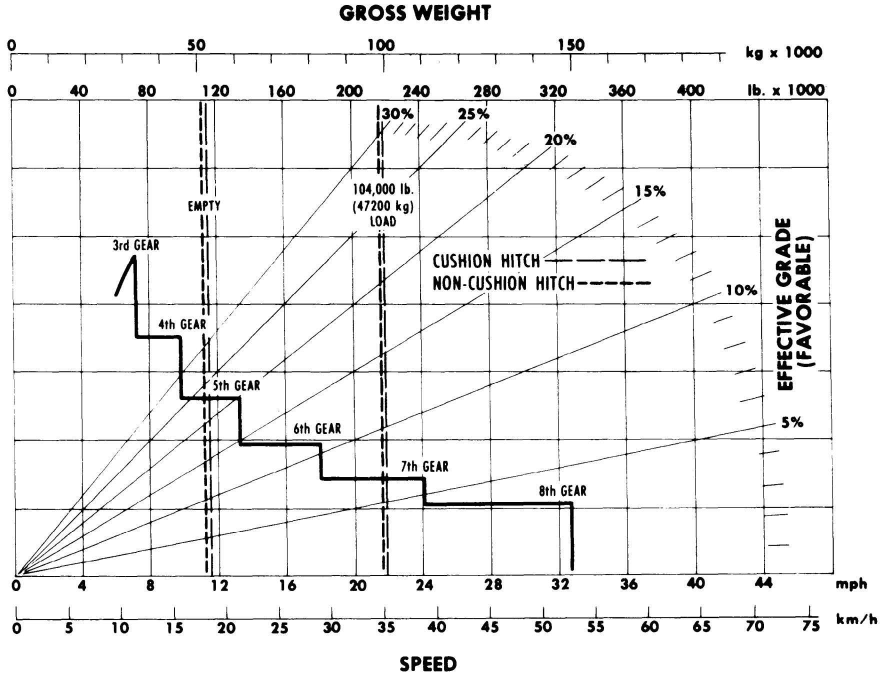


Fig. 9.3.14. Sample performance curve for determining maximum safe sustainable speed for wheel tractor-scraper when operating on a downgrade (*courtesy:* Caterpillar, Inc.).

and any correction factors. These factors could be based on operator proficiency, production methods, weather, traffic, acts of God, etc. A contractor's ability to determine and apply these correction factors under his particular conditions will, to a large extent, determine his degree of success in the earthmoving business.

To find the final hourly production, the estimated amount would be multiplied by any correction factor relating to the job. Correction factors for bulldozer production can be determined from Table 9.3.7 and Fig. 9.3.15. These correction factors can be applied to production estimates, such as those found in Fig. 9.3.16, for different sizes of dozers and types of blades.

Example 9.3.5. Estimate the hourly production of a track-type tractor fitted with a universal (8U) blade to doze anthracite coal from a stockpile 200 ft down a 5% grade to a conveyor hopper. The operator is average and operates an average of 50 min/hr.

Solution. Use Fig. 9.3.16 for estimating.

1 yd^3/hr (m^3/h) = 525 (Fig. 9.3.16) correction factors:

0.75 average operation (Table 9.3.7)
1.20 loose stockpile (Table 9.3.7)
0.83 job efficiency (Table 9.3.7)
1.20 light material U-blade (Table 9.3.7)
1.10 grade correction (Fig. 9.3.15)

estimated hourly production
= (525 lyd^3/hr) x (0.75) x (1.20) × (0.83) x (1.20) × (1.10) = 517.5 lyd^3/hr

Assuming anthracite weighs 2000 lb/lyd^3 estimated hourly production will equal 517.5 tph.

Fixed-Base Equipment: Capacities of stripping shovels and draglines (Fig. 9.3.17) are calculated generally on the basis of annual overburden yardage, as outlined in the Bucyrus-Erie Surface Mining Supervisory Training Program (Anon., 1976):

$$Y = \frac{BC \times Q \times A \times U \times B}{C \times (1 + S)} \tag{9.3.59}$$

Table 9.3.7. Correction Factors for Bulldozer Production

Job Condition Corrections:	Track-type tractor	Wheel-type tractor
Operator—Excellent	1.00	1.00
Average	0.75	0.60
Poor	0–0.60	0–0.50
Material—		
Type— Loose stockpile	1.20	1.20
Hard to cut; frozen—		
with tilt cylinder	0.80	0.75
without tilt cylinder	0.70	—
cable controlled blade	0.60	—
Hard to drift: "dead" (dry, non-cohesive material or very sticky material)	0.80	0.80
Rock, ripped or blasted	0.60–0.80	—
Slot Dozing	1.20	1.20
Side by side dozing	1.15–1.25	1.15–1.25
Visibility—Dust, rain, snow, fog or darkness	0.80	0.70
Job efficiency—50 min/hr	0.84	0.84
40 min/hr	0.67	0.67
Direct drive transmission (0.1 min. fixed time)	0.80	—
**Bulldozer*—Angling (A) blade	0.50–0.75	—
Cushioned (C) blade	0.50–0.75	0.50–0.75
DS narrow gage	0.90	—
Light material U-blade (coal)	1.20	1.20
Blade bowl (stockpiles)	1.30	1.30
Grades—See following graph.		

*Note: Angling blades and cushion blades are not considered production dozing tools. Depending on job conditions, the A-blade and C-blade will average 50–75% of straight blade production.

Source: Anon., 1975.

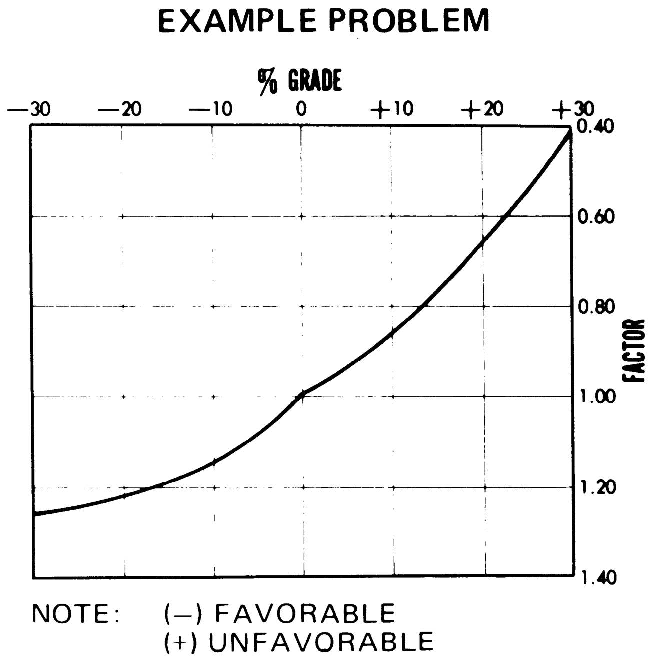


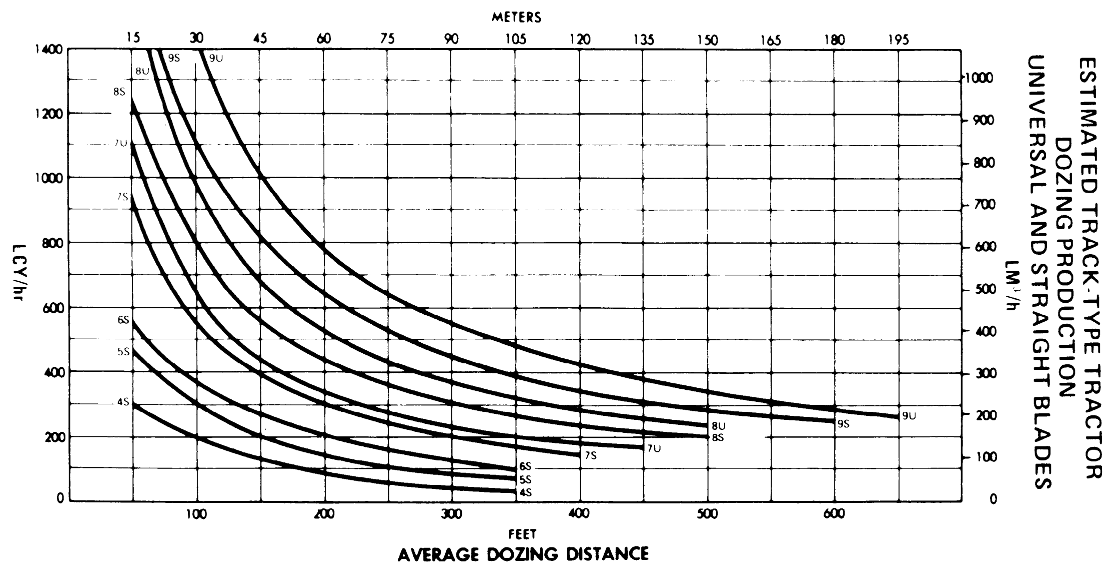
Fig. 9.3.15. Correction curve for adjusting bulldozer production when operating on a grade (*courtesy:* Caterpillar, Inc.).

where Y is annual overburden stripping requirement in bank yd^3 (m^3), BC is bucket capacity in yd^3 (m^3), C is cycle time in sec, S is swell factor expressed as percentage, O is dragline scheduled yearly operating time sec, A is dragline availability as a percentage, U is dragline utility expressed as a percentage, and B is bucket fill factor expressed as a percentage.

The cycle *time* refers to the length of time that is taken between commencing to dump one bucket load to commencing to dump the subsequent load. In other words, it is the time for one complete cyclic operation. For this calculation, however, this should not be a design figure or theoretical maximum; rather, it should be the overall average cycle time that can be expected over a period of years. The time should also include all the unrecorded delays inherent in the operation including all inefficiencies in motors, operators, occasional poor digging, etc. Typical values of dragline cycle time range between 58 to 68 sec.

When the material has been loaded into the bucket, a similar increase in material volume is experienced. This is comparable to the swell in in situ overburden volume that occurs in the spoil pile itself. Thus the rated bucket capacity must be adjusted by the swell to correctly express the equivalent in situ bank volume. Swell factors in the 20 to 35% range are common.

The *bucket fill factor* is another adjustment to the rated bucket capacity. This adjustment corrects the rated cubic yards (cubic meters) to the actual volume, which is dependent on the material heaping qualities, the angle of repose, and the consistency with which the operator completely fills the bucket. Although it is theoretically possible to have a bucket fill factor greater than 100%, usual actual values for draglines are in the 80 to 95% range.



*Note: This graph gives the estimated track-type tractor dozer production using universal (U) and straight (S) blades. It is based on the following conditions:

a. 100% efficiency (60-minute hour).
b. Power shift machines with 0.05 fixed times.
c. Machine cuts for 50 ft, then drifts blade load to dump over a high wall.
d. Soil density (weight) of 2300 lb/lyd^3 (3000 lb/byd^3), material swell 30%, load factor 0.76.
e. Coefficient of traction 0.5 or better.
f. Hydraulic controlled blades used.

Fig. 9.3.16. Production curve for estimating track-type tractor dozer production based upon dozing distance, machine class, and type of blade (*courtesy:* Caterpillar, Inc.).

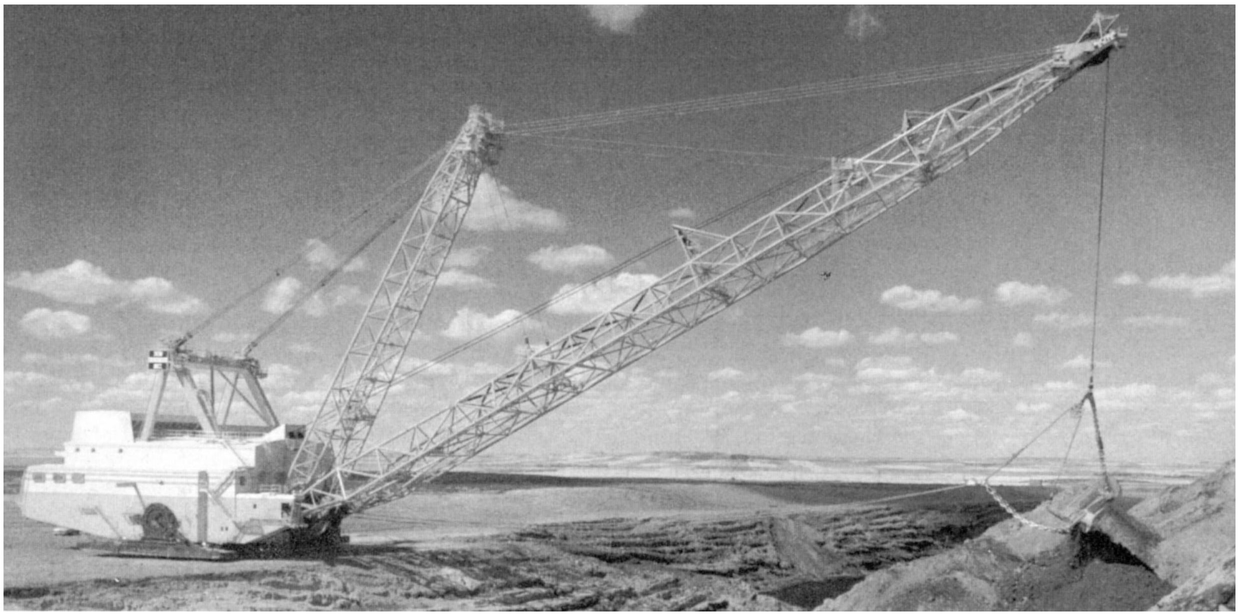


Fig. 9.3.17. Typical large walking dragline (*courtesy:* Bucyrus-Erie Co.).

The *scheduled operating time* makes allowance for periods of time when the dragline will not be operated. For some operations, this will be 100%, that is, the dragline is always scheduled to work, even on holidays such as Christmas Day, etc. These operations are probably in the minority today although, in general, operators tend to work their stripping equipment more continuously than, say, the loading equipment. A scheduled operating time for draglines of 90% or better is usually attained at most properties.

The *availability* is the part of the scheduled hours, expressed as a percentage, that the machine is electrically and mechanically ready to work. Thus this is the total time the operator has available in which to strip overburden. A 90% machine availability can be attained, although a more usual figure would be 80%.

Finally, the *utility* is the part of the machine available time that the machine is actually stripping. The types of delay the utility allows for are those other than mechanical and electrical, such as lunch, loss of electric power, waiting for blasting, etc. This factor varies from mine to mine, but values in the low 90s can regularly be attained at well-organized properties.

The primary dimensions required for dragline selection are

1. Dragline reach.
2. Cut depth capability.
3. Stacking height.
4. Bucket capacity.

The first three factors are dependent on the stripping method with associated overburden cut depth, coal thickness, proposed pit width, etc. The fourth factor is controlled by the proposed production rate, which then dictates the required stripping rate.

An additional requirement is introduced when multiple draglines are proposed. This requirement is the balancing of production between the machines to ensure a continuous and orderly operation.

There are two main methods to obtain the required dragline dimensions, these being by using graphs or by computation. Both use the "cut diagram" or simple range diagram for either actual measurement or derivation.

Fig. 9.3.18 illustrates a typical cross section through a pit. The overburden is dug from the area cut A and is deposited in

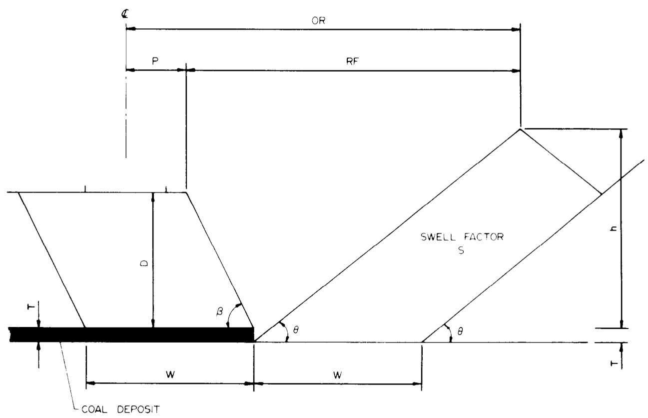


Fig. 9.3.18. Simple side-casting dragline range diagram showing relevant geometrical parameters (*courtesy:* Bucyrus-Erie Co.).

the area spoil A. As before, this diagram is a view perpendicular to the direction that the dragline is working and is thus a simple range diagram. By considering it to be a slice of uniform thickness, areas on the diagram become directly proportional to the relative volumes. Thus for a swell factor of 25%, the area marked spoil A should be 25% larger than the area marked cut A. Using this technique, the actual conditions in the field can be graphically or mathematically represented.

The graphical method utilizes a precisely drawn scaled diagram similar to Fig. 9.3.18. Then by direct measurement, the required dragline dimensions can be found. This method is long and laborious, especially when a number of different combinations of bank heights, coal thicknesses, pit widths, etc., are to be analyzed. However, it does find excellent application for the more complex mining methods.

The computational method is performed by deriving the mathematical relationship that describes the particular stripping method. In this relationship, factors such as bank height, coal thickness, spoil pile angle of repose, etc., can be considered as variables. Thus the effect of changing one or more of these factors can be readily calculated.

Sometimes, for the most common stripping methods, these relationships are computed over a wide range of values and are then presented in graphical or tabular form. These graphs or tables then provide good quick first-order results for the given situation.

Fig. 9.3.18 illustrates a pit cross section with the relevant geometrical parameters labeled. It should be noted that this particular cross section illustrates an operation that spoils just up to the coal. The spoil pile toe does not ride up the coal side. Other cross sections can be treated in a similar manner.

The parameters used in the analysis are

b = Highwall slope with the horizontal in degrees
q = Spoil pile slope with the horizontal in degrees
D = Overburden depth, *OR* = Dragline operating radius
P = Dragline positioning, *RF* = Dragline reach factor
S = Spoil pile swell factor, expressed as a decimal fraction
W = Pit width
h = Height of the spoil pile peak above the top surface of the coal
T = Coal thickness

Basically, this method equates the area of the cut slice with the area of the spoil slice, adjusted for swell, which is equivalent to equating the appropriate yardage because of the assumed unit cross-sectional thickness.

The reach factor is given by the following equation:

$$RF = \frac{D(1 + S)}{\tan\theta} + \frac{W}{4} + \frac{D}{tan\beta} - \frac{t}{\tan\theta} \qquad (9.3.60)$$

where t is the height the spoil toe is allowed to ride up the coal face.

The dragline reach factor is the horizontal distance between the bench crest and the spoil pile peak. The dragline operating radius can be found from the relation,

$$\text{operating radius} = \text{position} + \text{reach factor}. \qquad (9.3.61)$$

The required dragline digging depth is the dimension $\boldsymbol{D}$ or the total depth of overburden between the bench floor and the coal surface.

Finally, the required dragline stacking height is given by

$$\text{stacking height} = h - D \qquad (9.3.62)$$

which is the height the spoil pile exceeds the height of the cut above the coal surface, where

$$h = \frac{RF - D\cot\beta}{\cot\theta} - T \qquad (9.3.63)$$

Example 9.3.6. Consider an operation with the following parameters:

b = 1:3 (highwall slope)
q = 1.25:1 (spoil pile slope)
D = 90 ft (27 m) (overburden depth)
S = 25% (swell factor)
W = 120 ft (37 m) (pit width)
T = 5 ft (2 m) (coal thickness)
t = 0 ft (0 m) (spoil pile toe just touches the coal)

Solution. The first calculation is to determine the dragline reach factor. Using Eq. 9.3.60,

$$RF = \frac{90\ (1 + 0.25)}{4/5} + \frac{120}{4} + \frac{90}{3/1} - \frac{0}{4/5}$$
$$= 140.63 + 30.00 + 30.00 - 0$$
$$RF = 200.63, \text{ say } 200 \text{ ft}$$

The second calculation is to determine the stacking height.

$$\text{stacking height} = h - D$$

where h is the height of the spoil pile peak above the top surface of the coal.

In order to make this calculation, h must first be determined using Eq. 9.3.63:

$$h = \frac{200.63 - 90\ (l/3)}{5/4} - 5$$
$$= 136.50 - 5$$
$$= 131.50 \text{ ft}$$

Therefore the stacking height is given by

$$131.50 - 90.0 = 41.5 \text{ ft}$$

This is seen to be a very moderate requirement compared with the reach factor of 200 ft (61 m). This is usually the case for the standard stripping methods, and only in the more complex systems does the stacking height directly affect the dragline selection.

Before a machine can be selected, certain adjustments must be made to the parameters to make them readily comparable to the machine design criteria.

The actual capacity of the bucket is of secondary importance when considering forces and loads on the boom, etc. The result is the designer uses a term *maximum suspended load* (MSL) when considering the business end of a dragline, which specifies the maximum allowable weight of the loaded bucket plus rigging that the machine is designed to support, and is calculated as follows:

$$MSL = \text{rated bucket capacity (bucket weight/yd}^3 \text{ or m}^3 + \text{material weight/yd}^3 \text{ or m}^3) \qquad (9.3.64)$$

This means the actual rated capacity of the bucket is not significant compared with its overall loaded weight. Thus not only the bucket's own weight is involved, but also the weight of the material being stripped, which varies with overburden type.

The weight of the bucket itself is usually expressed in terms of pounds (kilograms) per cubic yard (cubic meter) of rated capacity. This figure can vary from 1500 to 2100 lb/yd^3 (890 to 1250 kg/m^3) depending on the bucket size and class of service. Larger buckets tend to weigh more per unit of volume than smaller ones because of a nonlinear increase in the component's mass. Similarly a heavyduty bucket will weigh more than a medium- or light-duty at the same rated capacity. An average figure often taken for "ball-park" estimate is 2000 lb/yd^3 (1187 kg/m^3) of rated capacity.

A convenient method of machine selection is given in Fig. 9.3.19. This is a chart of several machines plotted on a graph of reach factor against maximum suspended load.

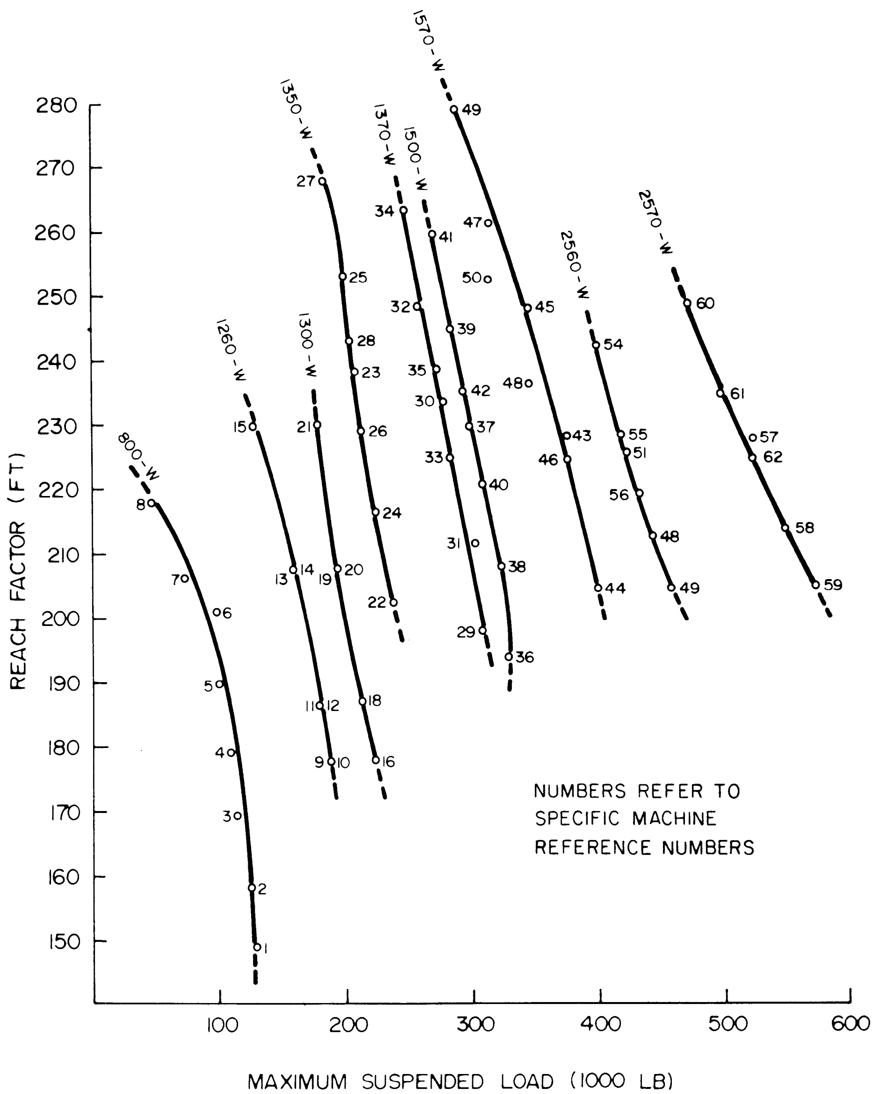


Fig. 9.3.19. Dragline standard machine selection chart (*courtesy:* Bucyrus-Erie Co.). Conversion factors: 1 ft = 0.3048 m, 1 lb = 0.4536 kg.

It is necessary to choose a dragline that meets both the reach factor and maximum suspended load criteria.

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Chapter 9.4
CYCLES AND SYSTEMS

JON C. YINGLING

9.4.1 INTRODUCTION

Though mine production unit operations must be matched to the characteristics of the site at hand–sufficient power must be available at the excavator's cutterhead for effective fragmentation, haulage vehicles must be matched to the physical characteristics of the roadways, bucket design must be matched to the handling characteristics of the material being excavated, etc.—the process of design is not finished when equipment is identified that satisfies these requirements. How the unit operations will work together must be considered.

The integration of the unit operations discussed in previous chapters of this section into a system that can execute the production cycle efficiently requires great attention, both in initial design of the system and in managing its day-to-day operations. Some of the important decision areas are as follows:

1. *Scheduling and sequencing unit operations executed in parallel* (i.e., simultaneously) so that any one operation seldom results in excessive delay in execution of another.

2. *Balancing production capacity of unit operations* (e.g., sizing a truck fleet to match a loading shovel or a conveyor network to match the output of multiple production sections). A good solution to this problem is often much more involved than simply matching either the peak or the average material production capacities of the different unit operations.

3. *Assessment of systems level design effects on availability.* Overall, mine production systems tend to be highly interdependent and serial in nature. Failure of one unit operation may cause production in a number of other dependent operations to cease. Buffer storage capacity in transportation systems and use of fleet operations to perform a task (a number of smaller machines instead of one large one) are sometimes appropriate approaches to improving overall availability and, accordingly, the long-term rate of production output from these systems.

4. *Assessment of working-section layout effects on the performance of the production system.* For instance, entry layout and cut sequence in room and pillar mines influence the total nonproductive time spent moving equipment between the working places, the lengths of haulage runs, and tradeoffs between haulage distances and downtime due to extension of the section belts.

5. *Real-time operations control strategies,* such as the rules used to dispatch pooled truck fleets to service multiple loading operations at a surface mine.

Mine production systems engineering aims at evaluating the many alternative designs and operational strategies that can be developed for a given application. It serves as a vehicle for identifying good, perhaps optimum, choices. It views the unit operations collectively as integrated systems rather than independent operations. The interfacing and interaction of the unit operations is explicitly considered. It evaluates performance in terms with overall relevance to mine management rather than in terms only locally relevant to the unit.

Though some general rules of "good practice" can be stated (see 9.0.1.4) and should always be considered, these rules are not highly prescriptive of the detailed design and control of these systems. Production systems engineering is primarily a collection of techniques, that, if properly applied, can give good answers to often rather complicated questions. The tremendous incentives for efficiency of mine production systems are powerful motivations for routine application of these techniques. At the many operations where profit margins are low, these incentives are often demands.

Consistent with the scope of Section 9, this chapter focuses on topics of mine production systems engineering relevant to systems-level aspects of equipment selection and utilization. Specifically, the following three technologies are treated in some detail:

1. *Simulation of mine production systems.* This technology has taken a major role in detailed analysis of the performance of these systems. It uniquely can capture the stochastic (i.e., random) and dynamic character of these systems in models with little abstraction. Simulation models can be constructed to address any of the issues listed above.

2. *Fleet dynamics and dispatch strategies.* Addressed here are approaches for real time control of mine production systems involving fleet operations. These approaches are based on heuristic strategies, although the heuristics themselves sometimes employ solutions to rather sophisticated mathematical models. (The adjective "heuristic" is used in this chapter to refer to decision rules that arise from one's intuition regarding approaches that *appear* appropriate for design or control of a system. The general performance of these rules cannot be characterized by deductive mathematical reasoning, and they lack complete theoretical foundation. However, they often provide a very useful and practical basis for decision making.) In the case of truck dispatch, this technology has been proven to utilize equipment more effectively.

3. *Stochastic process models of mine production systems.* Though simulation techniques can be used to model the performance of mining systems involving stochastic elements (e.g., truck queues at a loader, material flows on a conveyor network), a more detailed mathematical analysis using the theory of stochastic processes can sometimes lead to a concise analytical model. Relative to simulation models of the same process, these models can greatly simplify comparisons of system design and control alternatives and can thereby lead to stronger conclusions. They can be very useful in cases where the level of abstraction required to establish the model is not excessive.

Other aspects of production systems engineering that are broader in scope and longer term in nature, such as mine development planning and production scheduling, are discussed from the perspective of mine exploitation in Chapters 8.3 and 8.4. Moreover, consistent with the theme of Section 9, the perspective of this chapter is general. Numerous other discussions of mine production systems engineering can be found throughout this *Handbook,* addressing specific application areas, including the examples listed.

9.4.2 GENERAL CONCEPTS AND TERMINOLOGY

A vocabulary that helps one to characterize these systems and to understand the tools that may be used in predicting and analyzing their performance follows.

A *system* is a collection of components on which *processes* operate, causing the components to interact for some objective purpose. Typically, the degree to which intended purposes of the system being met are assessed in terms of one or more *performance measures,* e.g., "production rate," "tons per employee-hour," or "cost per cubic meter of material moved." The options that management have in design and control of the system are expressed in terms of a set of *decision variables.* For example, in designing a materials handling system, management may be able to specify the number and size of trucks in its fleet or the width and speed of its belt conveyors. The natural objective of management is to specify the decision variables so that system performance in terms of the appropriate performance measures is good, preferably "optimum."

Production systems engineering relies extensively on the use of mathematical/logical *system models.* System models are established using knowledge of the performance of individual unit operations and the rules for interaction among these operations. The model should attempt to capture the essence of system performance and should adequately portray changes in performance as values of the decision variables are changed.

An appropriate balance must always be sought with respect to the level of detail incorporated in the model. Too much detail unnecessarily consumes the analyst's time. It may also hamper the tractability of obtaining a solution to the model or realizing extended analytical objectives such as mathematical optimization. Conversely, too little detail may result in a model that is an abstraction of little relevance to the problem at hand.

The immediate utility of the model is that it enables examination and evaluation of alternative values of the decision variables. One can readily discern good options from bad ones. This typically is done at a fraction of the cost of experimenting with the real world system. The model provides considerable opportunity for both thoroughness and creativity on the part of the designer or analyst.

It is sometimes possible to explore by mathematical techniques *all* possible specifications of the decision variables and to select the one that results in the best values for the performance measure(s). If one can accomplish this, *optimization* has been achieved. However, one often must be content at considering a limited number of alternatives and, therefore, has little assurance that the optimum solution has been found. The term optimization is frequently misused, referring to studies where the analyst simply selected the best from a few alternative designs that were examined rather than attempting comprehensive exploration of the range of the decision variables.

The *scope* of the models, that is, the extent of the real world system that one attempts to represent explicitly in the system models, is an issue of considerable importance. One should attempt to keep the scope as small as possible consistent with the objectives in studying the system. For example, face operations might be the focus without special concern for outby transportation or vice versa. One stands a much better chance in arriving at a good, useful solution with small, focused problems.

Scope is delineated by *system bounds,* partitioning the system under study from its environment. The internal workings of the system, within the system bounds, are said to be *endogenous* (endogenous variables, endogenous events), whereas external impacts on the system are termed *exogenous.* Typically, many transactions occur across the system bounds, both outputs from the system to the environment and exogenous inputs from the environment to the system. It is always assumed that the system outputs in no way determine the exogenous inputs.

To clarify the issue of scope and system bounds, consider a belt network in an underground coal mine where shuttle cars are used for face transportation. Assume that spillage problems have been occurring at transfer points because of insufficient belt capacity for handling situations where peak loads from several

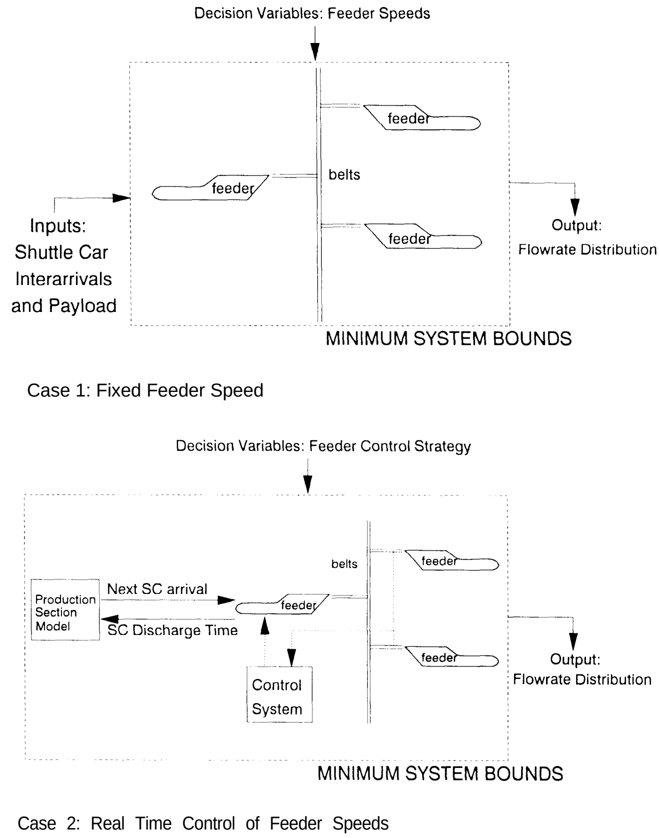

Fig. 9.4.1. Contrast of minimum system bounds for two approaches to control belt spillage

production sections are superimposed. One wishes to adjust speeds of the section feeders to reduce the incidence and extent of such problems. One approach might be just to reduce speeds permanently somewhat from current levels, spreading the discrete loads of material from the shuttle car into longer, thinner layers on the belt. Another might be to install belt load sensors, a data transmission network, and some type of intelligent control system to reduce speeds only on an "as needed" basis to prevent spills. Minimal system bounds (i.e., system bounds that keep the overall scope of the model as small as possible) will be contrasted for rigorous system models that one might construct to study these two alternative approaches to control of the system.

In the first case, one could set the system bounds at the section feeder, including the conveyor network in the model but excluding face operations. One would need to input shuttle car interarrival times and loads exogenously to the model, but these inputs would in no way be influenced by outputs of the system model for a given, fixed feeder speed.

For a rigorous model of the second case, however, one would have to broaden the model's boundaries relative to the first case, explicitly incorporating the face operations in some fashion. The reason for this is that feeder speed would be varied with time by the control system in response to circumstances on the belt network. When feeder speed changes, the discharge time of any shuttle car currently unloading is affected. This, in turn, influences subsequent interarrival times. The face operations and the belt network are intimately coupled here and, if rigor is desired, should not be modeled independently (Fig. 9.4.1).

The *state* of a system is defined in the model by the value of a collection of variables sufficient to characterize system operation and performance. *Static models* are solved for a single value of the state variables. *Dynamic models* are solved to obtain a trace of how the state variables change over time or, perhaps, over space or through a series of stages.

The first test for sufficiency of state variables is that one should be able to ascertain performance measures of interest with respect to making comparisons among alternatives. For example, consider a surface mine haulage system where trucks wait in line to be loaded by a shovel, travel to the dump site, dump (without having to wait in line), and return to the load site. One might be interested in analyzing the rate of production of this system for a given size of truck fleet. For this purpose, state might be characterized at any point in time by the number of trucks waiting to be loaded, the status of the loader (idle/busy), and the number of trucks in the process of dumping (ignoring equipment breakdowns). The stated performance variable as well as others that might also be of interest, such as average haul cycle times, average truck delay times, etc., can obviously be inferred from knowledge of how these variables change through time.

For dynamic models, such as the example just noted, a second test of sufficiency is also required. One must be capable of generating the subsequent state of the system given the current value of the state variables, exogenous inputs, and the known functional relationships among the system components. If one can do this, such a system is referred to as possessing the *Markovian property*. Specifically, this property states that the subsequent state of the system depends on the prior history of the process only through the present state. The next state may be generated from the model input using known relations describing single-step transitions. This criterion, in conjunction with the ability to infer performance variables of interest, provides a test to determine whether a given definition of state variables for the system is insufficient, sufficient, or superfluous.

With *discrete-event* dynamic models, the state variables only change at a countable number of points in time. A plot of the state variables vs. time appears as a step function. Using the state variables mentioned previously, the truck haulage system might be modeled as a discrete event system. *Events* are those occurrences at discrete points of time that result in the state changes. For example, arrival of a truck to the loading site changes the number of trucks waiting to be loaded and may change the status of the loader unit from idle to busy. Fig. 9.4.2 shows a plot of the state variables “queue length” and “loader status” vs. time for this system.

With *continuous* dynamic models, the state variables change continuously with respect to time. An example where a continuous model might be used would be in sizing the capacity of a mine drainage sump so that pumping can be shifted to off-peak periods to reduce power costs. The level of the sump, a state variable, would change continuously with respect to time as a function of inflow and outflow rates. As will be noted in the following with respect to a discussion of conveyor network modeling, sometimes one can convert from a continuous to a discrete model by making an appropriate selection of state variables. In general, discrete-event systems are easier to model than continuous systems. Some real world systems may require that a mixed discrete/continuous representation be used.

Stochastic models are used where the exogenous input to the model is random in nature. Note that this input is characterized in the form of some appropriate type of probability distribution (e.g., a distribution fitted to dump cycle times obtained in a time study of haulage operations). The input distributions are known, both their form (e.g., normal, exponential, gamma) and appropriate parameters. The modeling effort is to see, for a given specification of the decision variables, how the system responds

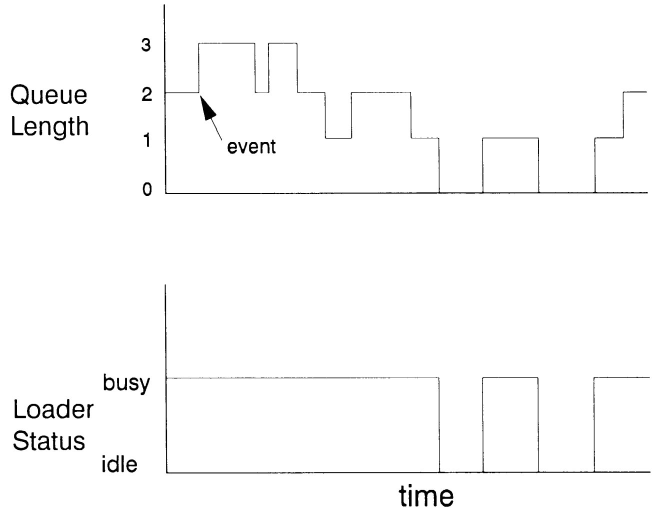

Fig. 9.4.2. Plot of state variables vs. time for a discrete-event dynamic model.

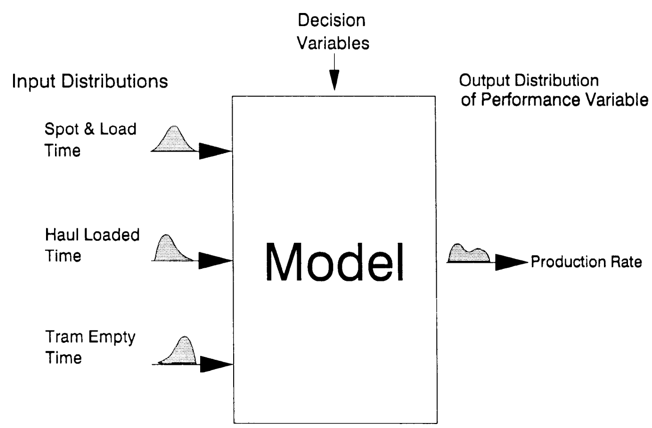

Fig. 9.4.3. Output from a stochastic model of production system performance.

to this input. Such response will also be stochastic in nature (Fig. 9.4.3). By modeling, one attempts to infer or deduce the distribution of the response, or important properties of this distribution, such as mean levels of the performance variables.

If, on the other hand, the input is not stochastic, the model is *deterministic*. In general, deterministic situations are easier to model and analyze.

Many times, mean levels of input variables are inserted into a mathematical expression that represents performance of the system such as the equations given for cycle calculations for haulage systems in Chapter 9.3. For example, one might insert mean trip times, mean spotting times, mean loading cycle times, mean dipper fill ratio, truck volume, and long-term average unit availability into a mathematical equation representing the production rate of a truck/loader system. Note, however, that probability theory provides no guarantee that the resulting answer would represent the mean level of the performance variable. Such approaches, when randomness is significant, only provide approximations. *It is an expressed role of the techniques discussed in this chapter to eliminate the errors inherent in such approaches*

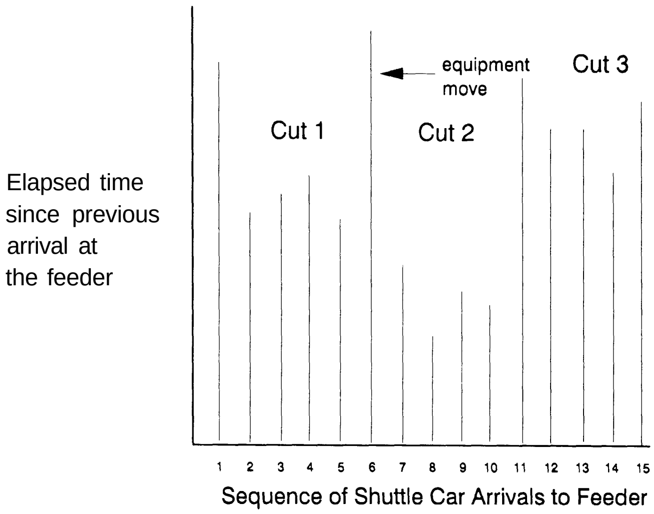

Fig. 9.4.4. Nonstationary input distribution required to describe shuttle car arrivals to the feeder.

and establish more accurate and meaningful performance predictions.

A model where the exogenous input and internal relationships do not vary systematically over time may yield what is called a *steady-state* solution. All static models are steady state. Such a solution represents an equilibrium behavior reflecting long-term performance of the system. A requirement for a stochastic model to provide a steady-state solution is that the parameters of the input distributions are held constant.

Note that if the the foregoing condition is not true, the input is said to be *nonstationary* and the *transient* behavior of the system must be studied. Although one might think that study of transient behavior is more difficult than steady state, often the reverse is true. In simulation of production systems, particular short periods of interest in analysis of transient behavior can often be defined. For instance, performance of a rail transport system might be studied only during the hour around shift changes when demands on the system are at their peak (assuming both men and ore are being transported simultaneously). The model might appropriately focus on transient behavior for this interval of time.

Input distributions to many mining models are nonstationary. For instance, the input to the belt network model discussed previously would systematically change as the mining machine proceeds through its cut sequence. The time between consecutive shuttle car arrivals should reflect this underlying cycle. When cuts are close to the feeder, interarrival times will be small. When cuts are far from the feeder, they will be large. If rigor is desired, interarrivals should not be taken as independent draws from a probability distribution since the influence of cut sequence on the sequence of interarrivals will be lost (Fig. 9.4.4).

The concepts previously discussed are useful in characterizing a system, selecting an appropriate modeling technique, determining the proper structure of the model, and in assessing the output or results produced by the model. The chapter continues with an overview of techniques that may be employed for detailed performance analysis of integrated systems of mine unit operations.

9.4.3 SIMULATION OF MINE PRODUCTION SYSTEMS

Discrete-event and *continuous simulation* are the modeling techniques that have been most widely applied in mine production systems engineering for the detailed analysis of equipment interaction. There are several good reasons for this. Most importantly, these approaches can readily accommodate the strong dynamic and stochastic character of these systems. Further, great levels of detail can readily be incorporated in simulation models. If properly carried out, such detail can insure valid representations of the real system without undesirable abstractions.

The technique is also one of the easiest to learn. The main prerequisites are a modest degree of computer literacy and a reasonably strong background in probability and statistics. Constructing the models is usually a conceptually easy process (though perhaps time consuming in some cases). Analysis of the output in proper scientific fashion is the more technically difficult issue.

9.4.3.1 Nature of Simulation Models

Simulation models are distinguished from other systems models in that the basis for model construction and solution is the *run*. A run is used to generate an artificial history of the process, specifying how state variables change either through time or through a sequence of stages. These histories, in whole or in part, provide information useful to the analysis of system performance.

To understand what is meant by a run and why the idea is so useful in constructing many types of process models, the following example will be considered. A number of boxes of different sizes, shapes, and weights are to be loaded on a cargo plane. The center of gravity of the boxes is of concern and some loading pattern that locates the center of gravity in a position acceptable for stable flying needs to be identified.

Generating a candidate loading pattern in a single step is a mentally forbidding task. However, if the task is broken into a sequence of stages, it becomes much less intimidating. For instance, a candidate loading pattern might be generated step by step using a three-dimensional graphics package, putting one box alongside or on top of others as one would if the boxes were being physically stacked. This is an example of using a run to construct a candidate solution to the model. In this example, only the terminal state of the run is of interest; but in many cases the entire history of the process evolution is of value.

The advantage of this approach to systems modeling is that it is often quite easy to specify the mechanics of the run. One can readily express rules for stacking boxes one atop the others. Given these rules, one can arrive at a candidate loading pattern by building up the pattern sequentially. However, given a collection of boxes of assorted weights, sizes, and shapes, a feasible loading pattern could not be deduced in a single step. In short, the mechanism of a run can generate information useful to the solution of a problem when attempts to summarize system behavior, via deductive mathematical analysis, cannot be made fruitful. However, as will be seen later in this chapter, the information provided by a run is often of weaker nature than that which might be obtained if analytical techniques can be applied to the problem.

Note that many models called simulations are misnamed. For example, one often says that a chemical process, a coal preparation flowsheet, or a mine ventilation network has been simulated. However, if steady state flows through the system are

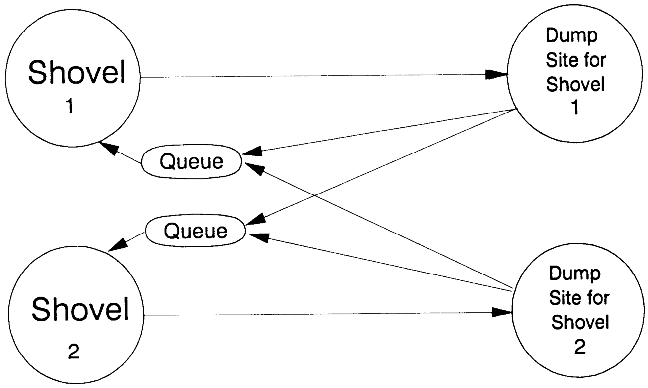

Fig. 9.4.5. System structure for truck haulage example.

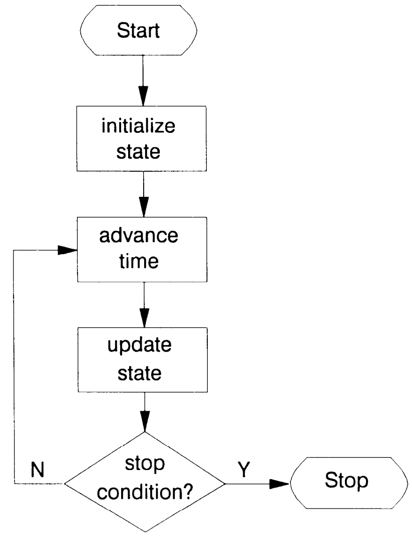

Fig. 9.4.6. Mechanism of a simulation model.

being computed, one is really solving a model expressed as a system of equations in each of these cases. A run has not been executed, generating an artificial history of system behavior. This misnomer arises from the fact that many of the equation-solving techniques used for these models are similar to the run mechanism.

9.4.3.2 How Simulation Models Are Structured

Discrete-event Models: In the next few paragraphs a simple example is introduced to help the reader gain some appreciation for discrete-event simulation. Consider a situation where two loading shovels are serviced by a pooled fleet of four identical trucks (Fig. 9.4.5). Trucks wait in line at a shovel, first-come, first-served, if they arrive and find the shovel busy loading another truck. The material from each loading shovel is dumped at a distinct site, but trucks need not wait for trucks ahead of them to dump.

A dispatcher continuously keeps track of the trucks, and when a truck has finished dumping, it is assigned to return to one of the two shovels depending on the current situation. Specifically, assume the dispatcher keeps track of the number of trucks waiting in line at each shovel and the number in transit to that shovel from the dump site (previously dispatched). He calls this number "assigned transport capacity." He sends any truck that has just completed dumping to the shovel with the lower value of assigned transport capacity.

In evaluating system performance, assume one is interested in the production capacity of the system, that is, the amount of material dumped per unit time at each of the two dump sites, the utilization of each of the two shovels, and the amount of time the trucks spend waiting at the shovels.

In constructing a simulation model for this system, the following input data for the model might be established:

1. Combined time to spot and load a truck at the shovel.
2. Time to transport the loaded truck between each shovel and its respective dump site.
3. Time to transport the empty truck from each dump site to either loading shovel.

These data might come from time studies or from other sources (e.g., determination of haul cycle times using rimpull curves and data descriptive of haul road characteristics). Any or all of the input variables might be described as distributions rather than constant values. Depending on the detailed objectives of the study and needs for accuracy, it might be more appropriate to break the aggregate truck spotting/loading time into spotting and a sequence of swing cycles. But things will be kept simple here.

A set of state variables for the system might include:

1. Each shovel's operating status (idle or busy).
2. Assigned transport capacity (ATC) for each shovel.
3. The number of trucks waiting in line at each shovel.

The state variables for a given simulation are not unique; alternatives often exist. Moreover, the state variables employed depend on the objectives of the study. Progressing through the section, it should become clear to the reader that the shovel operating status is necessary as a state variable here only because of the interest in utilization of the shovels.

In constructing the model, the objective is to generate a run, that is, an artificial history of the behavior of the loading/haulage system as it progresses forward in time.

In general terms, the overall operation of a simulation model is described in Fig. 9.4.6. The *initial conditions* provide the starting point. They include, among other items, a specification of the initial state of the system. In the example, one might specify that one truck is waiting in line at each loader, loading by the shovel has just started for the second truck, ATC is two, and both shovels are busy. Time is subsequently advanced and state is updated as appropriate. Next, a check is made to see if it is time to stop the run. This might be ascertained by checking the current state with some terminating condition or by checking to see if the simulated time exceeds some predetermined stopping time specified by the analyst. If not, time is advanced and state is updated again as shown in the figure.

The key issue is time advance. Changes in the values of the state variables for the example only occur at discrete points in time. In such a situation, it is clear that one knows the behavior of the system completely if a sequence of "snapshots" of the system at those points in time where state changes can be generated. For generating a run efficiently, time should advance from the time of occurrence of one event to the time of occurrence of the next event.

Note, however, that some simulation models for mining systems advance time in fixed increments of short duration. Events and state changes may or may not occur at the end of these increments. Though appropriate for systems where continuous state variables are required for the analysis, for most discrete-event simulations, this technique loses not only in speed of execution but also in accuracy since state is not permitted to change, as it would in the real world, at arbitrary points in time.

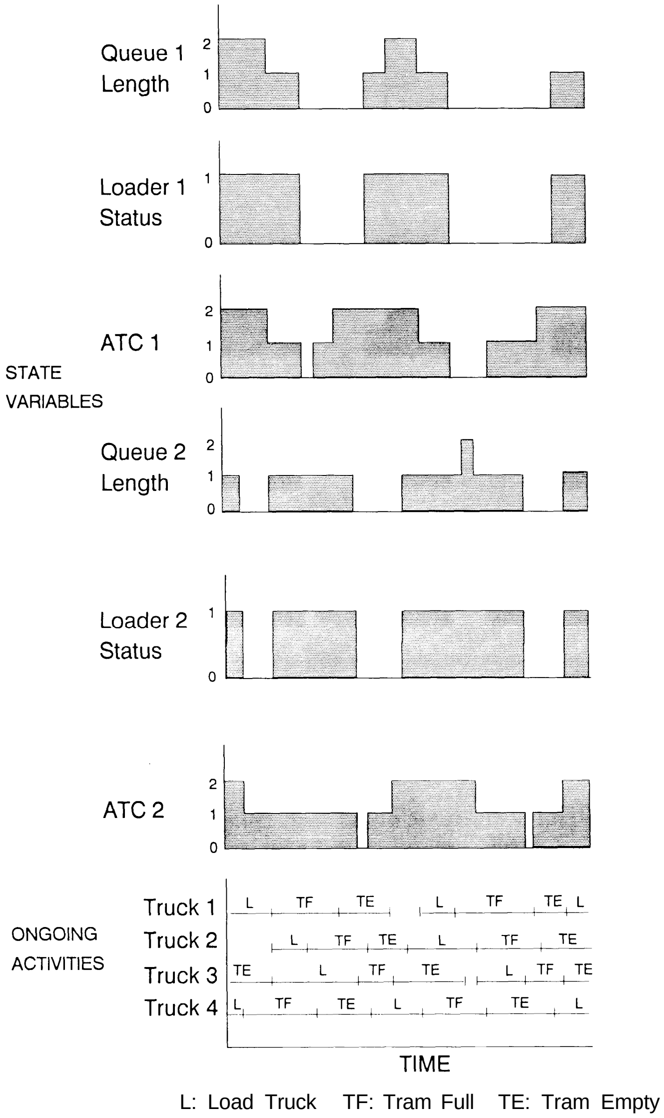

Fig. 9.4.7. Possible run for truck/shovel example.

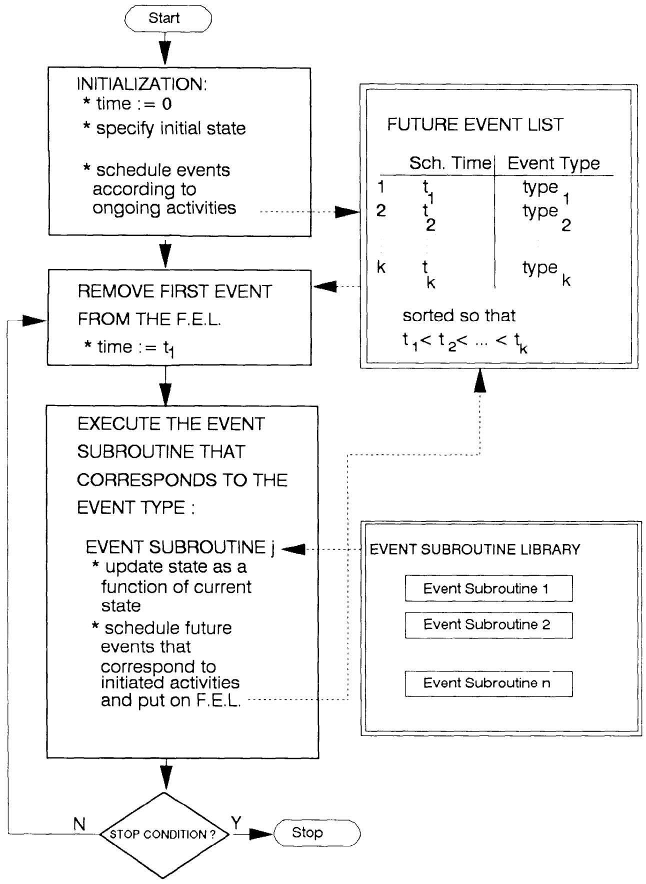

Fig. 9.4.8. Flowchart for the event scheduling approach to discrete-event simulation and corresponding program structure.

In discrete-event simulation, those input variables that specify a time duration are called *activities.* For the example, all three inputs—spotting/loading time, transport empty time, transport full time—are activities. Fig. 9.4.7 shows how the state variables might change with time for an example run of the truck haulage system. Referring to this figure, one sees that:

1. All events (an *event* corresponds to any change in the value of any of the state variables) occur upon the termination of activities. For instance, whenever a "transport empty" activity is terminated, an event occurs where the queue length increases by one and, if not already in the busy mode, shovel status changes to busy.

2. Upon termination of an activity, one might, using the external input to the model, be able to forecast the termination of some future activity. If this can be done, a point in time when some future event will occur has been identified. For instance, when the truck loading activity is finished, one can forecast, using the model input, when that particular truck will finish dumping. If "transport loaded" times are random, a typical time that follows the input distribution of these times would have to be "drawn" from this distribution. Nonetheless, the forecast can be made just using the *input* to the model.

Note that a future event cannot always be predicted upon termination of an activity. For instance, if a truck arrives and finds the shovel busy loading another truck, no activities are initiated, and no future events can be forecast. If the truck arrives and finds the shovel idle, a truck loading activity begins and the model input can be used to forecast when it will end. This illustrates that whether or not some future event can be forecast when the current event occurs is a function of the current state of the system.

The names for events are usually derived from the activities used to predict their occurrence. In the example, there are three events: (1) truck arrival at a shovel, (2) completion of truck loading at a shovel, and (3) completion of truck dumping at a dump site.

Now a position has been reached where it is possible to see how a computer might be programmed to execute a simulation run via what is known as the *event scheduling approach.* This is one of two major general purpose approaches for constructing discrete-event simulation models; the other is discussed later. Fig. 9.4.8 shows an overall flowchart for this approach. As noted previously, the initial conditions define the initial state of the system. In addition, they also give a few events for *priming* the model. For the example, if the initial state described above were to be used, a "completion of truck loading" event would have to

Table 9.4.1. Event Logic Tables for the Shovel Truck Problem

Definitions

State Variables:
- QL_i: queue length, shovel i.
- LS_i: status of shovel i, idle or busy.
- ATC_i: assigned transport capacity, shovel i.
- $TNOW$: current time

Activities:
- LT_{ik}: time to load truck k at shovel i.
- $TRAMF_{ik}$: time for truck k to tram full from shovel site i to dump site i.
- $TRAME_{ijk}$: time for truck k to tram empty from dump site i to shovel site j.

Event Logic Tables

Event 1. Truck k arrives at loader site i:

Condition	*Action*
Always	$QL_i \leftarrow QL_i + 1$
If LS_i = idle	$LS_i \leftarrow$ Busy Schedule a load completion event (type 2) for truck k at $TNOW + LT_{ik}$
If LS_i = busy	Put truck k at the end of the shovel i queue

Event 2. Truck k completes loading at shovel i

Condition	*Action*
Always	$QL_i \leftarrow QL_i - 1$ $ATC_i \leftarrow ATC_i - 1$ Schedule a dump completion event (type 3) for truck k at $TNOW + TRAMF_{ik}$
If $QL_i > 0$	Schedule a load completion event (type 2) at $TNOW + TRAMF_{ik'}$, where k' is the index of the truck at the front of the shovel queue
If $QL_i = 0$	$LS_i \leftarrow$ idle

Event 3. Truck k completes dumping at the dump site for shovel i

Condition	*Action*
If $ATC_1 \leq ATC_2$	$ATC_1 \leftarrow ATC_1 + 1$ Schedule a truck arrival event (type 1) for truck k at loader site 1 at $TNOW + TRAM_{i1k}$
IF $ATC_2 < ATC_1$	$ATC_2 \leftarrow ATC_2 + 1$ Schedule a truck arrival event (type 1) for truck k at loader site 2 at $TNOW + TRAM_{i2k}$

be specified for each shovel. In general, one priming event should be specified for each ongoing activity as the simulation run begins.

The priming events, both their identity and time of occurrence, are stored on the *future event list* (FEL). The events on this list are kept in order from earliest to latest. The computer then removes the most imminent event from the list, advances simulated time to the time of occurrence of that event and executes an *event subroutine* (there is one for each type of event). The event subroutine does two things:

1. It updates state.
2. It forecasts any new events due to activities initiated at the time of the current event and puts these new events on the FEL.

The results of both of these operations are determined strictly by the event type and the current value of the state variables using known functional relationships regarding the behavior of the system. No other information is required. Table 9.4.1 gives *event logic tables* for the three events of the example. These tables effectively describe the action of the event subroutines that one would incorporate with the event scheduling algorithm shown previously in Fig. 9.4.8 when coding the model.

Note that at any point in time, the content of the FEL is determined by the ongoing activities. A new event placed on the FEL would not necessarily go at the bottom of the list. For example, when truck 3 finished loading at shovel 1, its dump completion event would be placed on the FEL. While it is traveling loaded to the dump site, truck 4 may have begun loading at shovel 2. Its load completion event would then be placed on the FEL, and it may be placed ahead of the dump completion event for truck 3. Further, in general, events may be removed from the FEL before they occur. This would happen if activities exist that preempt ongoing activities.

Two new concepts, *entities* and *attributes*, useful for coding simulation models, are now introduced. Entities correspond to those dynamic objects in the physical system that require explicit representation in the model. In the example, entities might be used to represent the trucks and the shovels. Attributes are properties of a given entity, for example the operational status of the shovel, the size of the material load in a truck.

In the computer program, entities are often represented as records and order is maintained using files. For example, a sorted first-in, first-out (FIFO) file of records representing truck entities might be used for the waiting lines at each of the shovels. When an end of loading event occurs, the top record of this list is removed to identify the next truck to be loaded by the shovel. Discrete-event simulation programs have much in common with standard data processing routines. Good codes (i.e., codes that are efficient in their use of computer memory and in speed of execution) written from scratch in a general purpose language such as FORTRAN or Pascal require the use of list processing techniques and search algorithms to handle properly dynamic representations of the system using entities and attributes.

The event scheduling approach, discussed previously, is primarily an approach for organizing the simulation program. A quite different organization of simulation programs called the *process interaction approach* will now be discussed. Here coding is based on what are called *process routines.* One process routine is written for each type of entity in the system. In the example, two process routines would exist, one for truck entities and one for shovel entities.

The process routine describes everything that can happen to the entity as it passes sequentially through the system. Fig. 9.4.9 gives a process routine for the truck entities in the example. To understand how a process interaction simulation program works, it is useful to imagine that each entity is given, as it enters the system, a copy of the process routine that corresponds to the type of entity it happens to be. The entity then executes the routine sequentially in a fashion described below until it exits the system or the run otherwise is terminated.

The entity's process routine is activated. It starts at some point in the routine and executes as many steps in the routine as possible with zero time advance. These steps might change the value of global state variables, change the value of attributes of the entity, or schedule future events and put them on the FEL. Note that the FEL is common to both the event scheduling and process interaction approaches. Both are variable time increment techniques that advance the clock from the time of one event to the next.

Execution of the routine is temporarily suspended once time advance is required for the entity to take an additional step. These stop points correspond to the blocks before the dashed connection lines in the flowchart of Fig. 9.4.9. Note that some of the suspensions correspond to the beginning of activities. Here a future event will be placed on the FEL. Initiation of truck loading, block 6, is an example of this. Other stops correspond to indefinite delays, for example, wait in the queue at the shovel until removed, block 4.

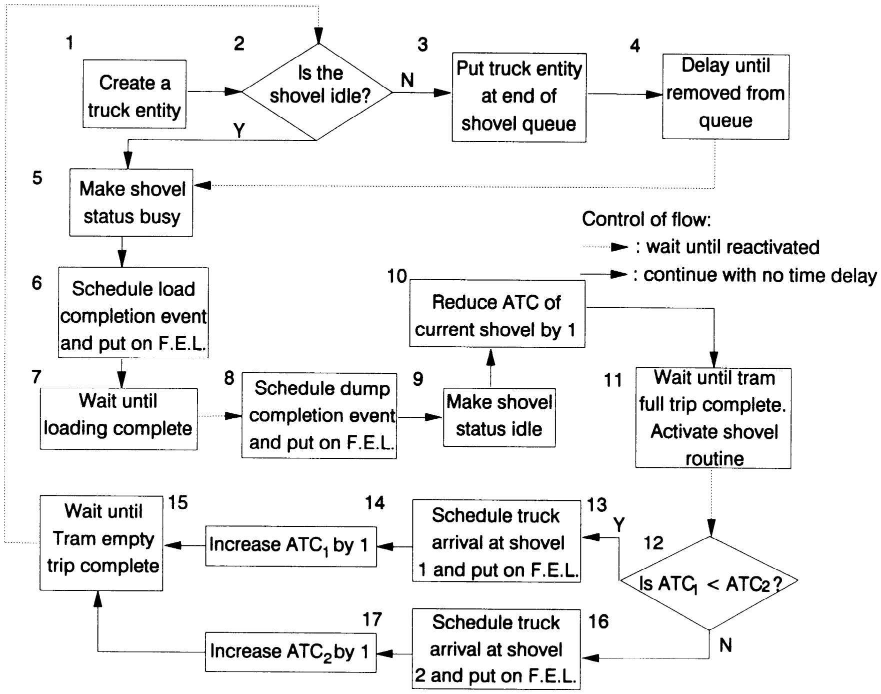

Fig. 9.4.9. Process routine for truck entity.

Once the process interaction routine stops, the process routines for other entities may be activated to see if they can also be advanced with zero time delay. In the example, a shovel entity's process routine would at some point in time remove a particular truck entity from the queue and subsequently stop execution. Before time advances, execution of that truck's process routine would resume from block 5 in Fig. 9.4.9.

Once all of the process routines have been advanced as far as possible with zero time advance, the most imminent event is removed from the FEL and time is advanced to the time of occurrence of that event. This corresponds to conclusion of an activity, and it reactivates the process routine of the entity that scheduled the event. The process routine of that entity is then executed sequentially starting from the block following the previous stop point.

If one writes a simulation model in a general programming language such as FORTRAN, the event scheduling approach should be used rather than process interaction. If the program writer must take care of all details, process interaction routines become very complex and the organization of the code is not natural. However, most of the special purpose *simulation languages,* including all of the more recent languages, are based on the process interaction approach.

When writing a program in these languages, one effectively defines a process interaction routine for the system entities. However, the statements available are at a macro level; a single statement may take care of many detailed operations automatically. These statements are geared to common situations that arise again and again in simulation models, and they are very powerful. Also some complete process interaction routines for common situations are built into these languages. In the example problem, one probably would not have to write any code for the shovel's process routine, just one for the trucks.

With these macro statements, writing a process interaction routine is often quite natural and is typically much less involved than writing an event routine for the event scheduling approach. Table 9.4.2 gives a process interaction routine for the example problem written in the SIMAN simulation language. Note that only 18 lines of code were required for this model, and one need not explicitly write such a routine for the shovel. The use of simulation languages in mine systems simulation is discussed in more detail in a later section.

Continuous Simulation Models: Continuous simulation differs from discrete-event simulation in that the state variables do not change strictly at discrete points in time. Rather, they change continuously with respect to time. Simulation is not required if a closed-form relation that gives the value of the state variable for any time and any specification of initial state can be established. However, often one can only specify the rate of change of the state variable with respect to time as a function of the current state.

For instance, from basic mechanics, one can specify a differential equation relating the state variable x, position of the vehicle, to equipment performance data that describe acceleration a_v of the vehicle:

$$\frac{d^2x}{dt^2} = a_v \qquad (9.4.1)$$

If a_v were a constant, it would be simple to establish a closed-form expression of x as a function of time and initial position. However, a_v is determined by the the rimpull characteristic (a measure of available force for acceleration) and the motion resistance of the vehicle, both of which, for a given vehicle and operating conditions, change as a function of the vehicle's velocity (see 9.4.2.4).

If the rate of change of the state variable is described as a first-order ordinary differential equation (i.e., the highest-order derivative is one, and all derivatives are taken with respect to the same variable, e.g., time), then a first-order Taylor's series expansion of x about t yields:

$$x(t + \Delta t) = x(t) + dx/dt(\Delta t) + O(\Delta t^2) \qquad (9.4.2)$$

Ignoring higher order terms, this expansion suggests a simple

Table 9.4.2. SIMAN Code for Truck/Shovel Simulation

```
;Most of the input data are supplied to SIMAN by another routine called
;the "experimental frame." For instance, it gives spatial layout of the
;system, the velocity that the trucks can travel, the queue disciplines
;at the loader sites, and parameters for the various random variables.
;
           CREATE, 4;                     one per truck (4 total)
;Truck movement entities reside in this queue until the transporter is
;assigned to them.
START      STATION, 1;
           QUEUE, 1, 4;
           REQUEST, 1:                    !truck identity is
             TRUCK(2);                    attribute 2.
           ASSIGN:X(1)=X(1)+1;
;It is assumed that the first two trucks go to shovel 1 and the
;second two go to shovel 2. The global variable x(1) is used to
;distinguish between the truck pairs. It is incremented for each truck
;passing through the block.
;
           BRANCH,1:
             IF,X(1) .LE. 2,TOFIRST:
             ELSE,TOSECOND;
;
;Prior to transporting to the shovel we increment the corresponding
;ATC which is maintained in global variable X(i+1) for shovel i,
;i = 1,2
;
TOFIRST    ASSIGN:X(2)=X(2)+1;            X(2) is ATC for shovel 1
           TRANSPORT:TRUCK(A(2)),2;       station 2=shovel 1
TOSECOND   ASSIGN:X(3)=X(3)+1;            X(3) is ATC for shovel 2
           TRANSPORT:TRUCK(A(2)),3:       station 3=shovel 2
;
;Stations 2 and 3 correspond to the shovel queues. Since the process
;is similar for both shovels we use what is called a SIMAN macro. it
;allows to write one piece of code for the operations at each shovel.
;The attribute M is the current station number of the truck movement
;entity.
;
;SHOVELS STATION,2-3;
;The truck waits in this queue, first come first served, until the
;shovel is free. It requests the shovel by the "SEIZE" block.
;
           QUEUE,M;                       The shovel queues
           SEIZE:SHOVEL(M - 1),1;
;
;Delay the truck by spot and load time. "RN" says this activity has a
;random normal distribution.
;
           DELAY:RN(1,1):
           ASSIGN:X(M)=X(M)-1             spot and load time
           TRANSPORT:TRUCK(A(2)),M+2;     to dump site
;
;Stations 4 and 5 correspond to the dump sites. For this program we
;have broken the "transport and dump" activity into its two components.
;
           STATION,4-5;                   the dump sites
           DELAY:RN(2,1);                 dump time
;
;Return the truck to the appropriate shovel based on the current
;value of the ATC.
;
WHERETO:   BRANCH,1:
             IF,X(2) .LE. X(3),TOFIRST:
             ELSE,TOSECOND;
END;
```

numerical approach to computing $x(t)$. Start with $x(t_0)$, the initial state at time t_0, then, ignoring the higher-order terms, use the expression and the expression for dx/dt as a function of the state variables to estimate $x(t_0 + \Delta t)$ for some small increment Δt. Repeat this procedure to estimate $x(t_0 + 2\Delta t)$, $x(t_0 + 3\Delta t)$, etc. The magnitude of Δt is called the step size. The error in computing x for a single step is given by the unknown term $O(\Delta t^2)$, which can be made small by selecting a small step size.

This approach is seldom used directly because, relative to other available approaches, errors are large and computational efficiency is poor. However, this illustrates the basic idea behind a class of algorithms called Runge-Kutta methods that are routinely applied for this type of application. With minor extensions, these techniques can be applied to higher-order ordinary differential equations like the second-order one given previously to describe vehicle motion.

9.4.3.3 Coding Options for Mine Simulation Models

One has three options for establishing a model for use in mine production systems analysis: (1) write the code from scratch in a general purpose programming language, (2) select an appropriate and existing, often commercial, mining simulation software package, or (3) write the code from scratch in a simulation language. The issues one should consider in selecting from among these alternatives are now briefly discussed.

The first option is probably one that should only be considered by research institutions or engineering firms. The justification for considering this option is that a general purpose language offers complete flexibility in constructing the model. In contrast, the other two options might be constrained in the type of process they can model and/or in the efficiency with which they can execute a particular type of simulation run. These constraints do not exist for a general purpose language.

However, this option is often a time-consuming one. These models typically involve thousands of line of code and have, in the past, taken months, even years, to code. One should become very familiar with list-processing techniques and data structures prior to attempting such programming. Note that execution speed for models written in a general purpose language should, if proper data processing routines are employed, at least equal that achieved by the simulation languages. Often these speeds can be exceeded because the situation being modeled may allow use of more specialized routines than those implemented by the simulation language.

If one elects to code in FORTRAN, one might consider use of the GASP language. Though often referred to as a simulation language, GASP is a collection of FORTRAN subroutines that can be used to organize filing structure for the program, handle filing operations, handle input and output, and provide a clock-advance mechanism. The program would be coded using event scheduling. Several other simulation language packages also offer such utility routines for user-written FORTRAN codes.

However, writing simulation models in general languages other than FORTRAN that have better data structures and dynamic storage allocation (e.g., Pascal or C) is often more natural. FORTRAN is only prominent in simulation because of tradition, not because of any special suitability of the language itself.

Numerous simulation software packages exist for mine production systems, such as conveyor network models, truck/shovel models, surface mine simulators, continuous mining section simulators, longwall simulators, etc. Some of the more prominent of these packages are discussed in the next segment of this chapter.

These packages provide the core routines necessary for simulating these types of operations, and they accept alternative user specifications on system structure. The main advantage of such packages is that the user need not generate the code. Further, some of these packages have an impressive range of capability. For example, the General Purpose Surface Mining Simulator (GPSMS) package incorporates detailed ore body and spatial description of the mine along with capability to simulate a wide range of production operations and their interaction on this property (Albert, 1989).

There are some limitations of these packages, however, that should also be kept in mind. Input requirements can be extensive and learning these requirements can be quite time consuming. The better packages have given considerable attention to simplifying the user interface. For packages with broad scope, one may not be interested in using all this capability, but the package may, regardless, require extensive input description. Many of the existing packages use discrete increment time advance. Though necessary for simulating systems with continuous state variables, this approach leads to very slow execution times and inaccuracy for discrete-event simulations, and is inappropriate for some of the systems modeled.

Perhaps the most serious limitation of this class of software is that it is nearly impossible for the package developer to foresee all the systems alternatives that users might like to model. The scope of available options may not meet user needs. Moreover, since the software is written in general purpose languages with thousands of lines of code, major modifications of the package to accommodate user needs, other than by the original developer, is often a forbidding task.

The third option is the use of simulation languages. Here the user must write his own code. However, because of the powerful nature of the statements of these languages, the amount of code required to write the model is typically dramatically reduced relative to that required when writing the model in a general purpose language. This is especially true when the process interaction approach is employed.

For instance, a simulation model was developed by the author of a surface mine where a truck fleet serves as an intermediate transport link between a pair of loading shovels and two skip hoists. Features addressed in the model included a detailed description of the truck loading cycle, haul road segmentation with different operating conditions, pooled use of the truck fleet with special dispatch rules, modeling of truck failures, use of standby trucks that are activated when breakdowns occur, interfacing with a fixed capacity bunker at the hoist stations with special control logic for skip loading, and shift start-up and shutdown routines. Using a simulation language, a model for this system was written with fewer than 130 lines of code. This is substantially less than 5% of the amount of code that would be required to construct the same model using a general purpose programming language.

The proficient programmer in these languages can write reasonably complex models in a matter of hours or days, not months. In the course of developing the code, the programmer becomes intimately familiar with how the process is being modeled. The level of model validity can be judged better relative to judgments made when using a package developed by others. Most importantly, the programmer has considerable flexibility in customizing the model and the output it provides to suit needs and analytical objectives.

Simulation languages possess what is called a *world view*. This reflects the scope of the macro-statements used in the model and the type of real-world processes to which they are oriented. This is an important factor to consider since it influences the ease with which one might model a given situation or the ability to model it at all using the language. Many simulation languages are oriented toward modeling queueing systems. The SIMAN language, developed for manufacturing applications, has a number of features useful for mine systems modeling. These include material handling modules for discrete vehicle and conveyor transport and special features for accommodating spatial aspects of the system. Other languages have recently started to introduce such features.

It is not likely that one will find the world view of most simulation languages too restrictive or awkward for mining applications; most situations can be modeled using these languages. General constructs in the language can usually be combined to model specialized processes when the specialized constructs are absent. Differences may exist in the facility with which various situations can be modeled. Moreover, to overcome restrictions in scope, most of the modern simulation languages allow the user to interface routines written in a general purpose language with the routines written in the simulation language. One can thereby accomplish operations that the language cannot perform itself directly. At least in principle, one is not constrained in the scope of processes that may be modeled if user-defined routines can be integrated with the code.

If possible, it is suggested that someone interested in modeling a particular process contact users of the various simulation languages to see if example codes they have developed for various mining applications are available. Because of their brevity, these codes can be readily "digested" by individuals familiar with the simulation language. These examples can aid the learning process and greatly speed development of a custom model for a particular application.

The major simulation languages currently available include GPSS (GPSS/PC, GPSS/H), SLAM, SIMAN, and SIMSCRIPT II. Since these languages and their supporting software packages have been undergoing extensive updating and improvements in recent years and these trends are likely to continue for the foreseeable future, detailed comparisons of features will not be undertaken here. The following issues, however, should be considered when purchasing a license for a simulation language.

1. *Availability of the package in a PC version.* The computing power of today's personal computers is adequate for many (but certainly not all) applications. OS/2 packages that are now becoming available will increase the size of model that can be investigated using a microcomputer.

2. *Available modeling orientations supported by the language.* Preferably the language should support process interaction, event scheduling, and continuous simulation. Process interaction is typically the main approach employed for discrete-event simulation using the language. User-written event routines may often be interfaced with the process routines to increase flexibility and scope of the language; the package will typically have utility subroutines available to the user for creating these programs. If modeling of systems with continuous state variables is supported, mixed continuous/discrete systems can be represented. Detailed examination of the approach used for interfacing user-written routines and for mixing discrete/continuous representations should be undertaken if one believes there will be need to use these capabilities.

3. *Special application orientations for the language.* Some languages are directed especially toward queuing systems simulations, which is a major aspect of many real-world simulations, including mining systems (see 9.4.5). Some packages have built in features to make the simulation of materials handling systems easy.

4. *Output analysis software.* Custom reporting capability should be provided, and it should be easy to gather a wide range of system performance statistics. Many packages enable one to create files with detailed performance data generated during a run and include rather sophisticated statistical software for analyses using these data.

5. *Capability of animation of the simulation run.* This includes a graphics interface to the simulation package that provides a dynamic display of the system state using symbolic representations that the user designs. Effectively, it provides an opportunity to obtain a dynamic, global view of system operation at a faster or slower pace than could be observed in the real world. The importance of this capability depends on user needs. It may provide a means for gaining insights into system operation and can be an effective communication tool when presenting results to management or clients. At present, costs for animation capability can be high.

6. *Availability of good documentation with example programs to illustrate modeling approaches.* A well-written manual with illustrative examples can greatly speed the process of learning the code.

7. *Quality of debugging facilities.* The language package should have convenient facilities for debugging with abilities to generate program traces, set break points, etc. Debugging effort for programs in a simulation language is, relative to typical computer programming applications, slightly disproportionate to the size of the programs because of the powerful nature of the statements. Animation capability can also aid the debugging process.

9.4.3.4 Using the Models in Systems Analysis

The capability of simulation models to accommodate inputs that are described as random variables is most useful. Mining engineers are well aware of the ramifications such randomness has on the performance of real-world mine production systems. One might attempt to answer questions such as

1. How do truck breakdown rates and repair team performance influence the preferred size of a truck fleet?

2. How do different belt sizes and speeds influence the amount of spilled material at a transfer point, given the stochastic patterns of loaded material on the intersecting belts?

3. How much increased production can be expected if a bunker of a particular size is installed in the belt network to provide a storage buffer when outby belts fail?

4. What are the implications of the randomness in truck loading and haulage cycle times on the way a dispatcher should control the truck fleet so that production rate is maximized and blend qualities are maintained?

In answering questions such as these, there is a key issue to recognize (an issue that, unfortunately, is often ignored by practitioners). If the input variables are random variables, the performance variable(s) also is a random variable. The proper goal, therefore, is to know the probability distribution for the performance variable as a function of the input variables and specification of the decision variables.

However, the simulation model, because it is based on a run mechanism, does not give us this distribution directly. Rather, a single run gives one possible *realization* of this process. For example, it describes the way production might go for a particular shift. One must use multiple and/or extended runs and statistical inference to infer properties of the distribution of the performance variable (e.g., the mean level of the variable). Drawing conclusions on performance of a system from a single, short simulation run can be as misleading as management using, say, one shift's production figures for evaluating a field-implemented system.

The first issue of output analysis is to establish a clear definition of the performance variable(s) of interest. There is usually opportunity to collect observations on a large number of variables of interest during the simulation run. In the truck haulage model of 9.4.3.2, one might be interested in utilization of each loader, the average waiting time of the truck in the loader queue, the average production rate from each loader, the average total cycle time for the trucks, and perhaps others. Although any or all of these data are readily obtained from a simulation run of this system model, one is strongly discouraged from considering more than one or two variables in formal comparisons between alternative specifications of the decision variables.

There are two reasons for this statement. First, as will become more clear in the discussion that follows, there is uncertainty involved in any estimate of performance from a stochastic simulation run. If one wants to ascertain the value of several performance variables with high accuracy, often an extreme amount of computer time will be required.

Second, one is faced with difficulties in comparing alternative systems when multiple performance criteria are involved. In the example, if system A has higher loader utilization but higher average total truck cycle time than system B, how does one trade off a unit of loader utilization vs. a unit of total truck cycle time to decide which system is preferred? Multi-criteria decision making, that is, ranking alternatives while considering multiple performance variables of distinctly different natures, is an area of active research interest. However, most results from this field are not yet considered very practical.

It should be noted that one can often aggregate many performance variables in terms of their contribution to cost or profit from the operation. In the truck haulage example, a revenue might be associated with each ton of ore produced, a cost might be imposed for each hour the vehicle is operated and for each failure incident. In turn, a single criterion such as net profit becomes the performance variable of interest in comparing alternative system design or control options.

Other criteria might become performance constraints that all system designs must satisfy if they are to be considered true candidates for implementation. For instance, one might wish to select the production system alternative that produces at the lowest cost per ton subject to the constraint that the alternative produces at an minimum average rate of X tons per month. Here the performance variable "production rate" is not used directly in comparing alternatives. One just ensures that all candidates produce at the minimum acceptable level.

Though it is strongly encouraged that the analyst establish a single performance criterion for formally comparing system alternatives, this is not to say that one should ignore the other criteria. Ad hoc, intuitive experiments with a simulation model can often provide considerable insight into system operation and often suggests useful design and control options. Within reason, such experimentation is strongly encouraged. Indeed, it is here where the animation of the simulation run can often serve a useful role in the analysis. Nonetheless, the study should conclude with formal, scientific comparisons.

A final point must be emphasized regarding selection of performance criteria. Most of the literature discusses basing system comparisons on the value of the mean or average level of the performance variable (e.g., mean rate of production, mean availability). The mean is often, but not always, an appropriate basis for comparisons. The mean is so useful because it "boils down" the entire distribution to a single value. This value can be directly used in making comparisons in cases where total rewards or costs are proportional to the accumulated value of the performance variable over repeated or extended runs of the system. For example, total production revenues are likely proportional to the sum of individual tonnages produced during

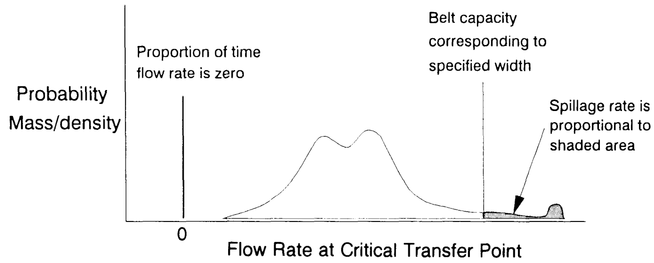

Fig. 9.4.10. Performance variable of interest for conveyor spillage example.

future shifts. Here the mean is a reasonable basis for comparing two or more system designs.

However, often other properties of the distribution of the performance variable, such as its variance, are also of interest. Consider the following example.

In simulating material flows in a conveyor network, flow rate (cubic yards or cubic meters per hour) at each of the transfer points might be included among the system state variables. Belt width effects on flow rate at some critical intersection in the model might not be explicitly incorporated in the model (a fixed conveyor width would simply truncate the flow rate when it exceeds some fixed level). Rather, one might attempt to size the belt, ex post facto, once the model output has been obtained. Here one is using what some might call a *functional model*—a model where unit sizing or capacity effects on its operation are ignored. Such models are often appropriate when designing a system since unit size is typically a decision variable.

What kind of performance variable would be of interest here? One that comes to mind is that time-persistent statistics might be gathered on the flow rate variable for the critical intersection. These data could be used, for instance, to generate a histogram (Fig. 9.4.10), where the magnitude of the vertical bars is equal to the proportion of time the flow rate is at any particular observed value. This histogram, assuming it accurately portrays long-run performance, could be used to provide information on expected spillage for any particular belt width. One would simply truncate the distribution at some fixed capacity level corresponding to a particular belt size; spillage rate would be proportional to the area above the point of truncation.

Here it is clear that one would like to know the *entire distribution* of the performance variable. The information provided by the mean flow rate is insufficient for and, in fact, has little relevance to the needs of this analysis.

As noted previously, simulation output does not give the distribution of the performance variable directly. Rather, this distribution, or properties of the distribution, must be inferred from data generated during a run using statistical techniques.

There are some interesting complications that arise when applying statistical methods to simulation output. Consider again the truck haulage example. Assume that truck delay time in the loader queue is the primary performance variable of interest. It is quite clear that the delay times of individual trucks will be related to one another. If the delay time of one truck is long, it is very likely that the delay of the next truck will also be long. Similarly, if the delay for one truck is short, the delay of the next truck is also likely to be short. This illustrates a key feature of simulation output. Individual observations of output variables are likely to be correlated (positive correlation in this case). Classical methods of statistical inference that are based on assumptions that successive observations are independent and follow the same distribution, that is, they are independent and identically distributed random variables (IIDRV), do not apply to simulation output.

Two distinct cases can be identified for analysis of output: *terminating simulations* and *steady-state simulations.* Such distinction arises not from the nature of the simulated process itself but from the nature of the performance variable of interest to the analyst when evaluating that process. With terminating simulations, the desired measure of performance is defined relative to a specific interval of simulated time. With steady state simulation, the desired measure of performance is defined relative to the limiting distribution of the variable as simulated time goes to infinity.

As an example of a terminating simulation, one might be interested in the volume of material moved *for a production shift* in the truck haulage example. A terminating simulation must have well-defined start-up conditions and terminating conditions. In the example, one might start as previously described in 9.4.3.2. One might terminate using a rule such that if a truck dumps its material within 10 min of quitting time, it returns to the service area.

Often one might be interested in system performance only during periods of "peak demands" on the system. For example, for some performance variable, one might simulate a track haulage system only during the period around shift changes, say, from 3 pm to 5 pm. Such simulations are also terminating; however, definition of initial conditions may be more difficult than in the previous example since activities of the system that are in progress when the simulation run begins may not be known and might be random.

As an example of a steady-state simulation, one might study, using a simulation model that accommodates unit failures, the availability of the furthest inby face conveyors as influenced by the size and location of surge storage bins in the belt network. If contemplating incorporating such buffer capacity in the network, one would certainly want to know implications on mine production over an extended time horizon. However, there are no natural starting or terminating conditions for considering performance of this system. Steady-state simulations are frequently employed where performance measures are long-term in nature and where there are no natural events to terminate the run.

An inherent characteristic of steady-state simulation is that the initial state of the system influences the early observations of the performance variable. For instance, in the truck haulage example, assume that for some reason one is interested in the steady state value of the mean delay of the trucks at the loader queue. The simulation might be started with all trucks queued up at the two loaders. Under such circumstances, the early delays will be longer than they typically would be. If data from the early portion of the run are used in estimating the mean steady-state delay, the estimate will be biased high because of these long delays. Any other starting condition is also arbitrary and will introduce some type of bias. This is called *initialization bias,* and it should be dealt with carefully.

The distributions for the exogenous input variables for steady-state simulations must remain constant throughout the simulation run. In contrast, terminating simulations are by nature transient. These distributions may change over simulated time. Moreover, no effort is made to eliminate the effect of initial state on the performance observations. In fact, one wants to explicitly capture the effects of initial state on performance. It is considered a fundamental aspect of the process under study.

Terminating simulations are much easier to analyze than steady-state simulations. Assume interest is focused on the mean value of some performance variable. The analytical procedure is to make multiple runs of the simulation model, each spanning the performance period of interest. Each of these runs, or replications, would use a different sequence of random numbers. Then the mean value of performance variable, $\overline{X}_i$, observed for each individual replication $i^i = 1,2, \ldots, r$, where r is the total number of replications, is calculated. Though the individual observations of the performance variable *within* a replication are not statistically independent, the $\overline{X}_i$, $i = 1,2, \ldots, r$, since they are generated using independent random number streams, clearly may be treated as IIDRV, and classical statistical techniques may be applied. Since the $\overline{X}_i$ are averages, they will tend to be normally distributed in many applications. (There are central limit theorems for correlated random variables.) The data can be used to compute a confidence interval for the performance interval as follows. Let

$$\overline{X} = \sum_{i=1}^{r} \overline{X}_i \tag{9.4.3}$$

and

$$S_{\overline{X}} = \left[\frac{\sum_{i=1}^{n} (\overline{X}_i - \overline{\mathrm{X}})^2}{r(r-1)}\right]^{1/2} \tag{9.4.4}$$

Then compute a $(100 - a)\%$ confidence interval for the mean value of the performance variable $\mu_{\overline{x}}$ as

$$\overline{X} \pm t_{a/2,r-1}\, S_{\overline{X}} \tag{9.4.5}$$

where $t_{\beta,k}$ is the value such that the $Pr\{U < t_{\beta,k}\} = \beta$ if U has a t distribution with k degrees of freedom.

What has been discussed is a procedure that has high probability, specifically $100 - a$, of constructing an interval that contains the unknown value of $\mu_{\overline{x}}$. Preferably, the interval will be small in width, but this depends on the innate variability of X. One can overcome wide intervals by increasing the r, as is evident from the previous formulas. In using this approach to compute a confidence interval, one does not say that $\mu_{\overline{x}}$ is contained in the interval with probability $1 - a$. The probability statement is about the procedure. A probability statement about $\mu_{\overline{x}}$ is inappropriate since it is a constant (though unknown), not a random variable.

Fig. 9.4.11 illustrates two major approaches for characterizing the mean value of a performance variable in a steady-state simulation. The first illustrated approach is similar to that used in terminating simulations. One makes several replications of the model using different random number sequences. Like the terminating case, the mean level for each replication will be IIDRV. However, since interest is in steady-state performance, observations from the early part of each run are eliminated to prevent these observations from biasing the estimator.

The second approach is called the batch approach. Rather than making several replications, a single long run is made. This long run is divided into a number of batches. For example, if 4000 truck delays are observed in the simulation run, the run might be divided into 40 batches of 100 observations each. (There is little incentive to ever create more than 40 batches.) Although consecutive individual observations of the performance variables are correlated, intuition suggests that if the batches are big enough, the means for each batch will tend to be uncorrelated. This is frequently observed to be true. These means will also tend to be normally distributed. The batch observations might thereby be treated as IIDRV.

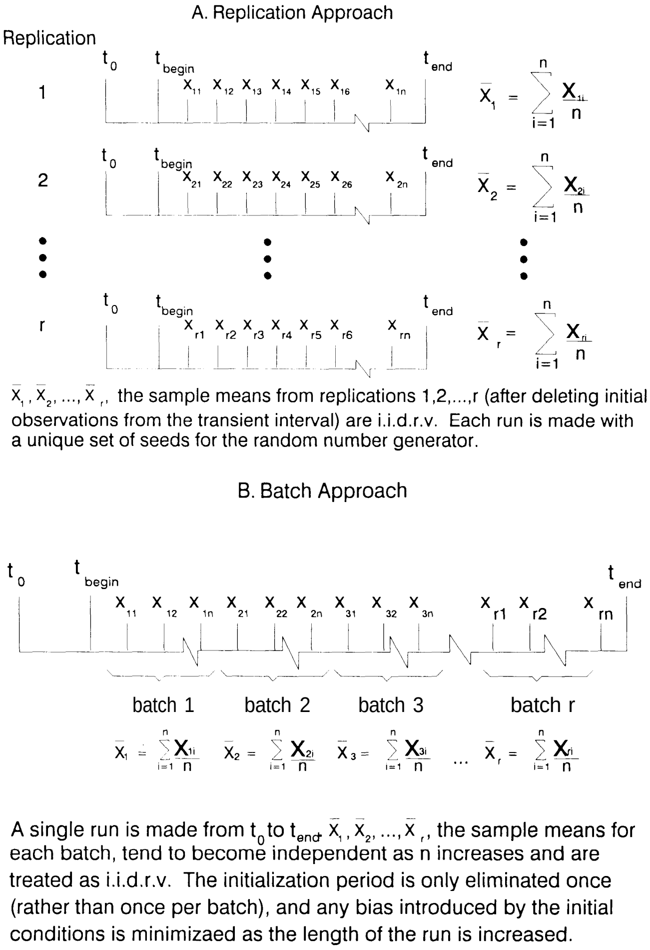

Fig. 9.4.11. Two approaches to estimating a performance variable for a steady-state simulation.

The batch approach has the advantage over the replication approach in more efficient use of data since one only throws data away for the initial transient period once, not *r* times. Moreover, it is difficult to ascertain how many data should be eliminated to eliminate initialization bias. This differs from one system to the next. In the batch approach, initialization bias becomes less and less significant as more data are collected by extending the length of the run. However, with the replication approach, one might collect more data by adding replications. If the length of the initialization period has been underestimated, a narrow confidence interval might be constructed around a biased estimator of performance and such an interval can be very misleading. Both the batch and the replication approach would use the same formulas given previously for terminating simulations in the numerical computation of the confidence interval.

Room exists here only to highlight some of the major points in analysis of output from simulation models. Hopefully, it is clear to the reader that simulation models with stochastic input variables do not directly describe performance of the system. Rather, they give data from which performance can be inferred with appropriate use of statistical techniques. Failure to consider formally these techniques is a weak point of many applications of simulation to mining problems. The reader is referred to the texts Law and Kelton (1982), Banks and Carson (1984), and Fishman (1978) for more detailed discussions of output data analysis. These texts address additional topics as follows:

1. How to determine how many replications of a terminating simulation are necessary to characterize a performance variable with a level of precision specified by the analyst.

2. Procedures to determine how long the initialization period lasts for a steady-state simulation. However, these procedures can only be viewed as giving approximate answers to this difficult question.

3. Procedures for determining the batch size and run length necessary for determining the mean level of a steady-state performance variable with a certain level of precision using the batch approach.

4. Procedures for comparing system alternatives. These include techniques for computing confidence intervals on the difference between the mean levels of performance in two systems, for selecting the best of a number of distinct system alternatives, and the use of formal experimental design in characterizing system response to the adjustments of the decision variables.

5. Procedures for reducing the variance of the response variables by manipulations of the input data to the model. With these procedures, response variables can often be estimated with greater precision, and comparisons between alternatives can often be more easily made.

9.4.3.5 Discussion of Specific Applications

The objective of this segment is to identify typical applications of simulation modeling in mine production systems engineering, to discuss some of the key technical issues in structuring these models, and to identify software packages that might be applied to these problems. Major differences in structure exist between models involving continuous material handling systems and discrete vehicle transportation systems, and this forms the basis for organizing this section.

Continuous Material Handling Systems: The major continuous material handling application involves conveyor network modeling. Some of the problems that might be addressed using simulation models of these systems are as follows.

1. *Design of systems for throughput capacity.* When a belt handles the output from several production sections, designing capacity to meet the combined peak loads from all of the sections can lead to unneedful expense in terms of capital and operating costs. Simulators can describe the distribution of material flow rates into any belt transfer point. This in turn can be used to evaluate a particular system configuration—specified in terms of network structure, belt speeds, use of surge bins to trim peak flow rates, use of limit switches and other operating controls, etc.—for its ability to handle material flows without excessive spillage or curtailments of availability of the network to the production sections.

2. *Assessment of network configuration impacts on availability of the transportation system to the production systems.* In many applications the production section and belt system form an extensive serial chain of operations. Belt system failures frequently provide a major constraint on production output. The simulators provide a means for testing alternative configuration options to assess their impact on network availability. This includes assessing the impact of buffer storage capacity (e.g., bunkers), appropriately sized and inserted at strategic locations, which are a means to temporarily breaking the serial dependence of one portion of the network on other portions.

3. *Testing control strategies for belt monitoring and control systems.* The potential for computer-based monitoring and control in underground mines is now just becoming evident, with the belt system a major focus of attention. A variety of means for real-time control of the dynamic material flows potentially exist, such as variable speed belts and feeders and feedback to

the production operations. Simulation will no doubt play an important role in development of these strategies.

Turning the discussion to highlight some of the key technical aspects of modeling these systems, there are two major approaches used to represent material flows. In the first approach, a conveyor is viewed as a collection of small adjacent cells, which is represented in the computer as an array. Attributes associated with each cell can denote properties of the material, for example, thickness on the belt or type of material. Time is advanced in small discrete time increments and, effectively, the attributes are shifted ahead a single register, representing material flow in the outby direction. At junctions or transfer points, cell attributes are combined in an appropriate fashion to represent superposition of material flows. For example, if the outby belt travels at the same speed as the inby belts, thickness attributes are simply added. This is, effectively, a simple form of continuous simulation. These models execute slowly relative to the approach discussed in the following, especially if one physically shifts records in the array to model material movement. It is much more efficient to shift a pointer to the array location that corresponds to the cell at the head of the belt, as done in some packages, than it is to move all of the records for each time advance.

The second approach employs discrete-event simulation. For many analytical purposes, one obtains adequate information if the *flow rates* are known at each transfer point, into and out of each bin or feeder, and out of each discharge point. For example, spillage and throughflows can be ascertained directly using these rates. Though knowledge of position of material along the conveyor at any point in time requires continuous representation as discussed previously, flow rates may be modeled as variables that only undergo changes at discrete points in time. If a change in flow rate occurs at one transfer point, one simply uses belt speed to forecast the time when a resulting change in flow rate will occur at the next outby transfer point. The magnitude of this future rate change is determined by multiplying the ratio of the speed of the incoming belt to the outgoing belt times the current rate change. A change of flow rate into a feeder/bin can be used to schedule a feeder bin overflow event if the new input flow rate exceeds the maximum discharge rate of the feeder, an event that may be preempted if rates decrease in the future before the scheduled event occurs. It may also schedule an increase in discharge rate if the current outflow rate is not currently at maximum. Further, such a change in flow rate can be used to schedule a future decrease in the outflow rate of the feeder if the input rate is less than the current outflow rate. Models using this representation may be coded using the event-scheduling algorithm mentioned previously. Since a variable time increment approach is used, they operate much faster than the other class of models. The BELTSIM program uses an approach similar to this (Bucklen et al., 1969).

Perhaps the most critical aspect of conveyor network modeling is obtaining a realistic representation of material arrival to the system. Exceeding the throughput capacity of units in a reasonably designed conveyor network will typically be a "rare event." Predicting accurately, with a reasonable amount of run time, the incidence of rare events via stochastic simulation can be difficult. For this reason, it is important that the description of material arrivals accurately portray the real-world situation.

For example, one might represent material arrivals from a continuous miner section in a coal mine using a sequence of discrete arrivals to the system. Shuttle car interarrivals would be random, but the parameters of the interarrival distribution would not be held constant through the simulation run. Rather, these parameters would be changed in a cyclic fashion to reflect the underlying cut sequence. A random place-change delay would also be allowed between each cut. The entire sequence of interarrivals may be preempted from time to time with delays representing major equipment failures. If additional realism is desired, interarrival times might alternate between high and low values for the two cars for those cuts where the tram paths differ substantially in length. One might also provide for shuttle car failure, changing the interarrival pattern when this event occurs. For a longwall unit, material arrivals would be continuous, but rates would vary for different portions of the cut. The overall arrival pattern would be cyclic and interruptions might be allowed to represent equipment failures. Available conveyor simulation packages vary in their ability to accommodate such detail. Some are quite flexible.

A number of packages for simulation of conveyor networks exist. These include BELTSIM (Bucklen et al., 1969), Continuous Material Handling Simulator (CMHS) (Tan and Ramani, 1988), Coal Mine Belt Capacity Simulator (CMBCS) (Thompson and Adler, 1988), BETHBELT- (Newhart, 1977), and UnderGround Materials Handling Simulator (UGMHS), (Manula, 1974). The recent paper by Sturgul (1989), discusses the use of a simulation language to construct a belt simulator.

Discrete Vehicle Transportation Systems: Simulation also provides a basis for analysis of many detailed aspects of mining systems employing discrete vehicle haulage in combination with excavation/loading equipment. Some of the issues one might attempt to address follow.

1. *Design of haul road profiles.* For example, should required climbs be achieved through use of short steep road segments or extended segments of lesser grade?

2. *Fleet makeup of transporters.* This addresses the determination of the best structure of a fleet of haulage vehicles at a mine, including the number of vehicles, their size, the number of units kept on standby in the event of breakdowns, and static allocation of the transport units among multiple loaders in order to maintain desired production ratios among the loaders.

3. *Analysis of real-time fleet control strategies.* Modern computer-based dispatching and fleet management systems often employ sophisticated computational schemes for improving the effectiveness of equipment utilization and maintaining control of quality of the mined product. However, the schemes are based on heuristic approaches (see 9.4.4). Simulation provides a technical basis for testing the quality of these heuristics and comparing alternative tactics.

4. *Detailed analysis of equipment interactions on overall system performance.* For example, one might consider modifications of the loading site to change maneuvering required for truck spotting or to allow double-sided loading at a shovel.

5. *Analysis of working section and pit layout options.* Examples of such analyses include where should an in-pit crusher be located to best serve multiple load sites and what are the ramifications of alternative cut sequences for a room and pillar section?

The detailed description of vehicle motion may require use of continuous simulation. A basic differential equation (9.4.1) can be used to describe the spatial position of a vehicle. When the vehicle is increasing in speed, a_v is typically determined from the expression,

$$a_v = \frac{F_r - F_m}{M_c} \tag{9.4.6}$$

In this relation, F_r is the rimpull force at the current velocity, which may be taken directly from the rimpull chart for the vehicle. F_m is the resistance force offered by the vehicle, which depends on rolling resistance, air resistance, and grade resistance. Corrections for rolling and air resistance are usually incorporated as a function of current velocity of the vehicle when making such simulations. M_c is the corrected mass that accounts not only for the actual mass of the vehicle, but incorporates a corrective term, which is also a function of current velocity, to account for losses in the drive system of the vehicle. It is seen that a_v can be known given a collection of state variables that includes the current velocity of the vehicle, grade, and rolling resistance of the haul road. Given a_v, Eq. 9.4.6 can be integrated to forecast the velocity and position of the vehicle at the end of a subsequent time increment.

Other factors taken into account in such simulations are speed limits for the road segment, which restrict maximum attainable speed, braking/retardation, and losses for the "dead time" when shifting gears (depending on the nature of the drive system). The VEHSIM package (Anon., 1984) is widely used for such simulations. It incorporates performance parameters of many Caterpillar vehicles and allows users to input parameters for other vehicles.

Recall that the selection of state variables, in part, depends on the the objectives of the simulation study and is not directly defined by the physical nature of the system. In simulations of material excavation and handling systems, performance is typically addressed in terms such as material production or throughput rates, equipment utilization, average cycle or delay times, etc. Performance measures such as these may be reliably obtained from simulation runs that do not provide knowledge of vehicle position at each instant of time. If one is in a situation where travel times are not available for various hauls (e.g., from time study data), it is often a good practice to use a vehicle motion simulator to provide data descriptive of haulage times for various segments of the haulage cycle. Subsequently, one uses these data as a basis for the input to a discrete event simulation model of the mining operation. The discrete-event model will operate much more efficiently, will not be repetitively computing deterministic haulage times, will readily accommodate stochastic aspects of the process, and can flexibly handle detailed aspects of the process relevant to the performance variables of interest.

A number of software packages have been developed for modeling these systems for both surface and underground mining. They include:

1. GPSMS, General Purpose Surface Mine Simulator (Albert, 1989), couples ore body modeling with operations simulation. Block models may be constructed to describe local geology and mine property boundaries. Operations descriptions are input to the package via a two-level structure. The first, "Equipment Face Activity Description," details excavation operations. The second, "Equipment Deployment Description" describes equipment operations and activities spatially and sequentially (i.e., the overall mining plan) and ties extraction with materials handling. Support modules are incorporated for simulating dragline operation, truck haulage (continuous simulation using rimpull curves), and conveyor haulage. This package extends the capability of its predecessors, the Total System Surface Mine Simulator (Albert, 1979) and Open Pit Materials Handling Simulator (O'Neil, 1966), although these packages have unique modules that may be of value and are not supported in GPSMS.

2. FACESIM (Prelaz, et al., 1968; Suboleski and Lucas, 1969) and UnderGround Material Handling Simulator (UGMHS) (Manula, 1979) can be used to simulate room and pillar mining operations, including ancillary operations. The input structure for UGMHS is similar to that for GPSMS. Motion of haulage vehicles may be established using continuous simulation in UGMHS, whereas distributions or constant values are input using FACESIM. The paper by Bise and Albert (1984) compares these packages and the use of simpler deterministic models of the mining cycle.

3. LHDSIM (Beckett et al., 1979) is a package for simulating a load-haul-dump system for room and pillar mining operations.

4. SCSMLT (Peng et al., 1988), simulates haulage in open pit mines and has features for interfacing discrete vehicle haulage systems to a crusher/continuous haulage system.

Examples of applications where the modeling was executed using a simulation language include Sturgul (1987), where alternative locations for an in-pit crusher were investigated; Mutmansky and Mwasinga (1988), who examined the general applicability of SIMAN to modeling mine production systems and applied the language to model a truck/loader system; Weyher (1976) who modeled room and pillar operations using GPSS; and Harrison and Sturgul (1988), who examined the main haulage system for a large underground mine and modeled several alternative transportation systems mixing both truck and train transport.

9.4.3.6 Simulation Case Study

A case study will illustrate the process of conducting a simulation study. The study has been "manufactured" by the author, but it is intended to portray major aspects of the execution of a simulation study in a realistic fashion.

Statement of Problem: A multi-mine coal operator has recently experienced some difficulty in meeting sulfur specifications for one of its major clients. Although construction work will begin shortly for new operations in a low-sulfur reserve owned by the company, in the interim, a greater quantity of low-sulfur coal is being obtained through the addition of two continuous miner sections at one of its existing operations working a low-sulfur seam. The addition of the two units brings the total number of working sections at the mine to four. A problem has been experienced because of the inadequate size of mainline belts to handle the peak production capacity of the four units. Several overload-related spills have occurred in the past few weeks since the new units have come on-line, and action needs to be taken so that this situation does not continue.

Decision Alternatives: The mine is nearing the end of its life, and since the belt line carrying the combined flow from the four units is quite long, the capital expenditure for modifying the existing conveyors so that they have higher capacity cannot be justified.

However, one option that is easy to implement and is believed to have potential for alleviating this problem is fine tuning the feeder discharge rates at the production sections. By slowing the feeders from their current levels, the discharge of an individual shuttle car is spread on longer, thinner layers on the conveyor belt. This, in turn, directly reduces the magnitude of the load observed at the transfer point where flows from the four units converge. Such a modification is expected to reduce the incidence and severity of spills. Note that it may not be desirable, from the perspective of overall production output from the mine, to set all of the feeders at the same speed. Hence the decision alternatives considered in this case study may be expressed in terms of four setpoint values, one for each of the four section feeders.

Of course, there are other control options that probably should be considered. With minor modifications, the model presented in the following is flexible enough to consider many other decision alternatives that one might conceive such as, for example, staggering start-up times of the production units so that the duration of the periods when all four units are active is minimized. However, for the purpose of this study, only feeder discharge rates will be considered.

Structure of the Simulation Model: The simulation model will incorporate (1) face operations for the four production units, (2) the feeders, and (3) the belt network to the transfer point where the material flow from the four units converges. The third component of the model is actually quite trivial for this study.

A fairly detailed model of face operations is necessary for two reasons. First, for the units where feeder speed will be reduced from current values, there is no direct way of predicting what the shuttle car interarrival times will be after the change is implemented. Certainly, there is a chance that on some cycles, the car dumping time will be increased from its current value if the material from the previous car has not cleared the feeder, but there is no direct way of knowing how often this will occur and what the net effect will be on total shuttle car cycle time.

Second, for this particular problem, there is a coupling between the belt network and face operations that should be considered. Changing feeder speeds has potential for both positive and negative impact on overall economics at the mine. By increasing shuttle car dump times on occasion, the short-term rate of production output from the mine will be negatively affected. On the other hand, if a major spill is avoided, one prevents shutting down the face units while the spill is cleaned up and production is increased through the increase in available operating time. As discussed in more detail in the following, the average rate of production output is employed to compare decision alternatives. To accurately reflect the coal output, the serial dependency of the face units on the operation of the belt should be incorporated in the model. It is interesting to note that the coupling exists in this case due to the nature of the performance criterion that is used in making the comparisons.

An issue that is believed to be important for the study is the effect of cut sequence. The variable distance between the cut and the feeder might have a pronounced influence on the effect of changing feeder speeds. If the cuts are close to the feeder, shuttle car interarrivals will be short and the feeder is likely to have more surge when the arrival occurs. Short hauls also have more dead time, and a delay in dumping is not as likely to turn into an overall delay in cycle time. For this reason, the model was constructed to incorporate the cut sequence followed by each section.

The model was coded in the SIMAN simulation language. Process interaction routines (in SIMAN, these are called "block models") were written to describe operation of the face units and the belts. Coal flow through the feeders and belts were modeled using flow rates as state variables since this approach has considerable advantages when modeling bulk material flows on continuous haulage systems (see 9.4.3.4). The SIMAN language does not readily accommodate such a representation for a surge storage device like a feeder, and for this reason the following event routines were written: (1) shuttle car arrival at feeder event, (2) feeder full event, (3) shuttle car empty event, and (4) feeder surge depletion event. Table 9.4.3 gives, as an example, the event logic table for event 1.

There are several interesting features of the block model. First, as one might expect, a separate process routine is not required for each face unit. The sequence of operations undertaken by a continuous miner, a shuttle car, and a roof bolter is very similar among the four units. The SIMAN macro concept (similar mechanisms are available in other simulation languages) allows entities from each of the four units to share the same process routines that describe the overall sequence of operations. Moreover, details necessary for describing spatial aspects of the process are readily decoupled from the process routine. In the model that has been developed, the shuttle car routine is roughly defined as follows:

1. Wait at the miner change point if the other shuttle car has not cleared the change point.
2. Once access to the miner is possible, seize a resource to block entry of the other shuttle car and tram to miner.

Table 9.4.3. Event Logic Table for Event – Shuttle Car Arrival at Feeder

First compute the current surge level in the feeder, E, as

$$E = \begin{cases} (T_e - t)\,D & D > 0 \\ 0 & D = 0 \end{cases}$$

where D is the current discharge rate of the feeder, t is the current time, and T_e is the time that the next surge depletion event (event 4) has been scheduled. (If $D > 0$ such an event is always scheduled.)

Condition	*Action*
Always	Set feeder input rate I to the shuttle car discharge rate R.
	Schedule a shuttle car empty event (event 3) at time
	$t + L/I$
	where L is the payload of the shuttle car.
If $D < I$ (feeder filling)	Schedule a feeder full event (event 2) at time
	$t + (K - E)/(I - D)$
	where K is feeder surge capacity. Also, delete any previously scheduled surge depletion event (event 4).
If $D > I$ (emptying) and $E > 0$	Schedule a surge depletion event (event 4) at time
	$t + E/(D - I)$

3. Wait while loaded.
4. Tram to miner change point and release the resource that allows the other shuttle car to enter.
5. Tram along a sequence of stations until arriving at the feeder change point.
6. Seize a feeder access resource and tram to feeder.
7. Activate a shuttle car arrival event (user-written event routine 1) and wait.
8. Release the shuttle car from the wait state when shuttle car empty event (user-written event routine 3) has occurred.
9. Return to the feeder change point and release the feeder access resource.
10. Tram along a sequence of stations until returning to the current miner change point.

Externally defined input data to the model define stations that correspond to physical locations of the crosscuts and entries. In addition, sequences of these stations are defined that control vehicle movements. Equipment moves along the cut sequence are handled in similar fashion. Only a few statements were required to model the four continuous miner units and movement of material along the belts.

Input data to the program, in addition to the station layouts and tram sequences noted above, are as follows:

1. Feeder discharge rates and surge capacity.
2. Belt transit times from the section feeder to the point of convergence.
3. Tonnage per cut.
4. Distribution of shuttle car, miner, and bolter tram rates.
5. Distribution of shuttle car payload size.
6. Distribution of time to load a shuttle car.
7. Distribution of time to bolt a cut.
8. Distribution of downtime when an overload spill occurs.
9. Distribution of time between unscheduled delays in the face production cycle and duration of such delays.

As can be seen, the amount of field work to obtain the various input distributions required by the model is not excessive. Simulation can usually proceed with a rather small investment in time studies. Moreover, these studies typically should focus on the various elemental times of the production operation, spending perhaps a few hours observing each of the major parts of the operation. Simulation is a means of synthesizing time study data and drawing meaningful conclusions from this data that otherwise cannot be obtained. In this case, perhaps the most difficult data to obtain would be the distributions involving unscheduled delays in the face production cycles. For the purpose of relative comparisons among the decision alternatives that have been studied, it might be reasonable to ignore such delays. Ignoring such delays would, however, be inappropriate if one hoped to obtain accurate cost figures from the simulation study.

Application of the Model: From examination of the capacity of the belts handling the combined flow from the four units, it is clear that the overload incidents occur at this mine only when peak loads from all four units meet simultaneously. In such a situation, reducing the frequency of overloading incidents can only be accomplished by reducing the sum of the discharge rates of the feeders to the minimum capacity of the belts handling the combined flow of the four units. Moreover, there is clearly no incentive to reduce the discharge rates below this level since this will only reduce the rate of production from the working faces. These questions remain:

1. Will the reduction in feeder discharge rates cause an excessive decrease in the rate of production that cannot be offset by the gains in operating time?

2. How should one best go about allocating the discharge rate reductions among the four units?

To help answer these questions, four alternative strategies have been investigated. Alternative 1 represents the status quo. Alternatives 2, 3, and 4 all specify the same combined feeder discharge capacities equal to the capacity of the belts handling the combined flows. With alternative 2, the rate reduction is split equally among the four units; with 3, it is split equally among three units; and with 4, it is split among just two units. Allocating all of the reduction to one unit was not considered since this would clearly be an excessive choke on output from that unit.

Though there is some cost associated with cleaning up a spill, the costs are an order of magnitude less than the lost production costs associated with any belt shutdowns that might occur. These shutdowns are reflected directly in the simulation model. Ignoring cleanup costs, the alternatives have been compared in terms of the time average flow rate on the mainline belts, a measure of total production output from the mine.

Simulation runs were made for a single production shift since this represents a natural termination point for the run, and the output is analyzed as realizations from a terminating simulation. To deal with the potentially significant influence of cut locations and the cut sequence, the starting cuts were randomized for each run. (One could perhaps find better ways to handle the influence of the cut sequence, which introduces a long cycle into the operations, than through randomization; this appears to be a rather complex issue from a technical perspective.) A total of 20 runs was made for each alternative and the results are given in Table 9.4.4. As the histogram for the performance variable for alternative 1 in Fig. 9.4.12 shows, there is considerable variability of the output response. The sample mean production rate for each alternative is shown in the table along with a 95% confidence interval for the mean.

The results appear to imply that a reduction in feeder speed is desirable and that it is best to allocate the reduction equally among the four units (alternative 2). An equal allocation in this case is not surprising, but such a result probably would not hold if there were significant differences in the production cycle among the four units. Though a comparison of sample means is the device for selecting the best alternative, such a selection cannot be made with certainty. The sample means only represent estimates of the true, but unknown, mean. The fact that the confidence intervals of several alternatives overlap provides an indication that one might expect some difficulty when coming to a decision about the best alternative.

Table 9.4.4. Output for the Four Alternatives Considered in the Case Study

	Alternative			
Run	1	2	3	4
1	4.15	4.46	4.44	4.74
2	3.32	4.09	4.10	4.18
3	3.64	5.12	4.13	4.42
4	3.42	4.80	4.81	4.15
5	4.21	4.41	4.12	4.06
6	3.50	4.56	4.12	4.55
7	3.63	4.33	4.55	3.85
8	4.20	3.72	4.48	3.95
9	3.30	4.11	4.51	3.96
10	4.09	4.71	4.22	4.73
11	4.21	4.35	4.17	4.52
12	3.83	4.66	4.38	4.12
13	4.07	4.38	4.02	4.75
14	4.48	5.01	4.44	4.06
15	3.67	4.47	4.54	4.10
16	3.97	4.86	4.24	4.38
17	4.08	4.18	4.54	4.43
18	3.89	4.38	3.89	3.60
19	4.13	4.43	4.32	4.25
20	3.10	4.56	4.24	4.37
mean	3.84	4.48	4.31	4.26
std. dev.	0.376	0.330	0.225	0.313
95% conf. int.				
lower bound	3.66	4.32	4.20	4.11
upper bound	4.02	4.64	4.42	4.41

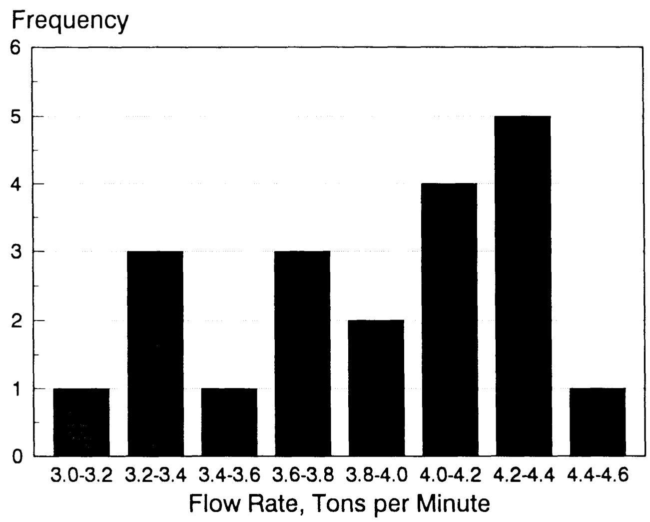

Fig. 9.4.12. Production rate distribution for alternative 1, current feeder setpoints.

In this problem, one might apply a procedure where the results of the first 20 runs are used as a basis for determining how many more runs to make before making a final decision on which alternative is best. These initial runs give considerable information in the inherent variability of system performance for each alternative. Using these measures of variability, the number of additional runs might be established on the basis of (1) the level of probability of a wrong decision that one is willing to accept, and (2) the minimum difference in performance among alternatives that one considers to be of significance. Once these additional runs are made, a final decision could be made by comparing the means, and one would have assurance that the two stated conditions on uncertainty are satisfied when making the decision. Further discussion of the nature of such procedures are beyond the scope of this presentation; interested readers are referred to the texts cited at the end of 9.4.3.4.

9.4.4 FLEET DYNAMICS AND DISPATCH

The stochastic aspects of mine operations—an equipment failure incident, variation in tram times or loading rates, etc.,—preclude development and field implementation of a fully prescriptive plan for operations. Rather than attempt to follow a rigid sequence of operations, often much can be gained from ongoing, real-time analysis of the current status and recent history of the operations. This analysis might then be used as a basis for taking control actions to improve subsequent performance of the system. This is perhaps most evident, and the technology is most fully developed and accepted, for real-time management (dispatch) of pooled truck fleets serving multiple loading units. Ideas similar to these might be applied to any type of mine production system employing discrete vehicles, although a large size of operations and the ability to allocate flexibly the transport vehicles among multiple excavators/loaders enhances the potential utility of such approaches.

The field is one with little theory; the basis for control actions involves application of heuristic approaches. Though innately heuristic, some procedures prescribe control actions on the basis of solutions to quite sophisticated mathematical models. The proof of the pudding is in the eating, and though heuristic, these techniques have been reported to be quite effective in increasing the rate of production from these mines for a given level of loading and transport equipment (Clevenger, 1983). Some approaches have also helped in simultaneously meeting a number of additional operational objectives such as better control of ore-grade targets.

An opportunity one has in certain discrete vehicle material transport systems involves the fact that the vehicle may be flexibly routed. Rather than running the vehicle in a locked cycle between a particular loader and dump site, one may use the vehicle to service multiple loaders and dump sites. Moreover, rather than following a predetermined, rigid sequence of movements, routing can be dynamically controlled, switching the sequence that loader and dump sites are visited by the truck from one cycle to the next.

The underlying reasons why advantages in system performance can be gained by such dynamic control include:

1. One can respond to information on the current state of the system that reflects the realization of particular random variables (e.g., the observed cycle time to load a particular truck) that could not have been predicted exactly ahead of time;

2. One can appropriately respond to deviations from the expected levels of performance that were used to develop the operational plan (e.g., one shovel is having an easy time digging and is completing its load cycle more quickly than expected and this provides opportunity for increased production over what was expected if levels of truck capacity allocated to the shovel are increased).

In examining truck dispatch strategies that have been employed, distinctions among them can be made with respect to two major factors as follows.

1. *The extent a forecast of future states of the system is employed to make the current dispatch decision.* Some approaches consider only the current truck (e.g., immediately upon completion of dumping) when making the dispatch decision prescribing where the truck should next be routed. Others attempt to forecast the state of the system for a number of future dispatches and make the current dispatch while cognizant of this forecast. For instance, one might send the current truck to its "best" shovel (e.g., the one that will be ready to load the truck at the earliest future point in time), but there is a good second choice for this truck. The next truck that will complete dumping also has the same "best" shovel but no second choice that is very good. Considering performance of the two trucks together, it would be better to send the current truck to its second-best choice.

2. *The scope of the operational objectives that drive the selection of dispatch assignment.* Some strategies base dispatch decisions on narrow criteria (e.g., send the truck to the shovel where it is expected to be loaded first). Other strategies base dispatch decisions in accordance with how well the decision contributes to conformance with a production plan. The plan is developed to enhance specific performance criteria such as overall mine production rate. It may also recognize various operational constraints such as achievement of grade targets with the ore blend from the multiple loading sites and desirable ore/gangue production ratios. These production plans are typically short term in nature, are routinely updated, and are frequently based on solution to math programming models for short-term production scheduling and equipment allocations.

Two examples of some simple "one truck at a time" dispatch heuristics are as follows:

1. As mentioned before, "send the current truck to be dispatched to the loader where it is expected to be loaded first." One would use figures on average load times and tram rates and current positions of previously dispatched trucks headed to the loader in question to forecast when the loader will have serviced all of these previously assigned trucks. Then travel time to this loader for the truck to be dispatched is compared to the forecast time when the loader will be free to estimate the future point in time that truck loading for the dispatch vehicle would begin. This is done for all loaders and the truck is sent on the shortest path to the one that is expected to load it at the earliest point in time.

2. "Send the current truck to be dispatched to the loader that is next expected to become idle (after servicing all previous trucks assigned to that loader)." One would forecast the time each loader is expected to become idle as discussed in the first example, and without consideration of the distance to that loader, the current truck is sent to the loader expected to become idle at the earliest point in time.

These rules are very simple and have some intuitive appeal. Note that one may use recent history on load time and tram rates to obtain better forecasts as operating conditions change at the mine. The first rule is noted to cause loader utilizations to become unbalanced (Lizotte and Bonate, 1987), since trucks tend frequently to be sent to the closest shovels, ignoring the remote ones. In the second case, production ratios will be more controlled but are primarily influenced by allocation of the shovel to the available loading sites. Both rules may result in shortsighted decisions because the current decision may be unfavorable when considered in conjunction with dispatches that might be made in the near future. Further, the objective criteria for both rules are very narrow and may not conform to overall operating objectives for the operations.

To see how one might approach overcoming the shortcomings of these simple rules, the operation of one of the more sophisticated dispatching systems, DISPATCH, which has been developed recently and is described by White and Olson (1986), is now discussed.

The DISPATCH system first solves a sequence of two linear programming problems to establish a short-term production plan. This plan is updated routinely as will be discussed in the following. When the plan is established, it is based on the current status of mine operations, reflecting siting and current availability of the loading shovels, estimates of quality characteristics of ore at working faces and stockpiles, and operational status of the process plant.

The first linear programming (LP) model defines "optimum" production rates for the various shovels, including those operating from stockpiles. The objective function of this optimization accounts for penalties associated with failure to meet process-plant input demands and violations of blend-quality constraints, and weights established by management reflecting the current relative desirability that plant feed originate from stockpiles or from active faces.

Given these overall flows from the material sources (stockpiles and working faces), a second LP is solved to determine the allocation of truck resources to move material flows efficiently along available haulageways. The model prescribes how many cubic yards (cubic meters) of haulage capacity should be allocated to all "paths" in the system. The second model is set up to maximize production rate from the system. It is loosely coupled to the first model in that the first model prescribes minimum production from working faces and stockpiles for the second model. Any excess production from working faces that can be achieved would be stockpiled.

The use of the output of the second model is to provide a criterion for truck assignments in real-time dispatch. Roughly speaking, the objective of the dispatch will be to implement and maintain the "optimum" haulage allocation determined by the second model, which is given in terms of the volume of haulage capacity that should be allocated to each path. Contrast this to the objectives of dispatch decisions for the two simple heuristics cited.

As mentioned previously, the two LP models form a short-term production plan. Whenever a major upset such as a shovel breakdown or process plant outage occurs, the models are solved again to establish a new short-term plan. Moreover, blend constraints in the first-level model are structured to allow some slack in conformance to rigid bounds on ore grade. An estimate of the moving average quality of the ore feed to the process plant is continuously maintained. There is a bound imposed on the maximum period of time that can pass without replanning so that excessive and sustained deviations from target grades do not occur. The more slack permitted in complying with grade requirements when solving the first-level model, the more frequently operating plans must be updated.

The actual dispatching procedure uses the deviation between actual current assignment of trucks to a path and the optimum assignment levels determined by the second stage LP as a basis for evaluating dispatch decision alternatives. All trucks that are in transit from a loader to a dumpsite are considered collectively with the current truck to be dispatched when making assignments. At the time of a dispatch, the time each loader will "need" a truck assignment (typically in the future) is forecast. "Need" is determined by the LP haulage capacity prescription. The loader with the earliest need is considered first. The truck closest to this shovel is nominally assigned to it. Eliminating this loader and truck from consideration, the shovel with the second earliest need is considered in the same fashion, etc., until all loaders have been considered. This will typically result in an assignment for the current truck to be dispatched.

The point of this discussion is to illustrate a way that dynamic dispatch systems can be used to (1) help achieve a goal-oriented production plan that, because of its short-term nature, is itself responsive to changing conditions at the mine, and (2) make decisions using updated forecasts of expected future states of the system rather than making each truck assignment independent of such forecasts.

The overall logical structure of most of the more recently developed dispatch systems will resemble the one discussed here. However, they will vary considerably in details. Although the mathematical models are described rather than explicitly reported, the system at the Mount Wright mine (Soumis et al., 1989) apparently differs from DISPATCH in a number of respects. Some of these include (1) consideration of alternative loader site locations when making preliminary equipment assignments, (2) use of a nonlinear objective function in determining optimum haulage allocation, which results in a more balanced distribution of truck capacity assignments, (3) use of a simple process-plant model for estimating the actual costs of deviations from blend targets (in the previous procedure these were, more or less, guesses), and (4) solution of a classical assignment model for dispatch decisions where the objective function minimizes the squared difference between current forecast waiting times for the shovels and trucks and average waiting times expected for implementing the operational plan.

This segment is concluded with two additional notes. First, the dispatch algorithm (but not the planning models) for these systems must be able to execute in a very short period of time (a matter a few seconds). With careful consideration of the mathematical techniques employed, one can use rather extensive models, as evidenced by the Mount Wright system, for making dispatch decisions. Second is a reminder that the procedures are heuristics, and many modifications are conceivable. Simulation, as discussed in the previous section, can serve to evaluate alternative dispatch heuristics and tactics for a given mining operation.

9.4.5 STOCHASTIC PROCESS MODELS OF MINE PRODUCTION SYSTEMS

The weakness of simulation as a tool for systems analysis should be evident from the discussion of 9.4.3. One wants to know the distribution of performance variables, perhaps, even how these distributions change as a function of time, but simulation only gives data from which one may infer some of the major properties of these distributions. It does not give the distributions directly. Moreover, it is typically quite cumbersome to make generalizations regarding system performance and its response to the decision variables on the basis of simulation output.

On the other hand, the theory of *stochastic processes* provides a basis for analytical modeling of these processes. The field is comprised of a number of basic topics including Poisson processes, Markov chains, renewal theory, continuous-time Markov processes, semi-Markov processes, Brownian motion, etc. These, in turn, service many applications including results in queuing theory, inventory theory, reliability theory, sampling plans for quality control, and inference-based measurements using signals gathered by sensing instruments, among many others.

The analytical results may, in some cases, give the distributions of the performance variables directly. They may also facilitate comparisons among alternative values of the decision variables, perhaps permitting mathematical optimization. Often they give valuable insights in to the nature of system operation that would not have otherwise been obtained. An example of this for a mining application is given in the following.

Typically, more assumptions are required in order to progress with this approach to systems analysis than are required when using simulation. If one is not careful, the result of the modeling effort may be an abstraction of little relevance to the real-world system. One is cautioned against "pulling a formula from a book" without understanding the assumptions and nature of the result. Nonetheless, the results that can be obtained by using these approaches often justify the efforts.

Subsequently, some examples of the more recent work in mining-related applications of stochastic process models that have resulted in—or with more work may lead to—practical achievements are discussed. These applications are in the area of modeling of continuous material handling systems and queueing theory applications. Space exists here only to present a descriptive discussion. Readers interested in learning about fundamental aspects of these techniques are referred to the well-known texts by Ross (1983), Cinlar (1975), and Karlin and Taylor (1975).

9.4.5.1 Stochastic Process Models of Continuous Material Handling Systems

As examples of the type of issues that can be addressed in applying this technology to analysis of continuous material handling systems, the work of Baral et al. (1987) and Sevim (1987) are discussed.

Baral et al. Model: This work addresses the implications of installing a bunker as a storage buffer in a serial arrangement of conveyors to increase the availability of that system to the production section it services. The role of such bunkers is to hold production for intervals of time when outby conveyors have failed, temporarily isolating the production section from its dependence on the outby transport network. They must be larger than bins used to reduce surge loading on an outby belt to control spillage. The service of the system depends on the availability of the conveyors between the face and the bunker and between the bunker and the end of the conveyor line. For the bunker to increase production, the inby belts must be available to haul material to the bunker when the outby belts fail. Moreover, the outby belts must be capable of clearing a substantial portion of the stored material between successive failures of the outby belts, else the effective capacity of the bunker might be reduced to the point where it provides little isolation. The availabilities of the bunker inflow and outflow belts are a function of where the bunker is located in the sequence of belts. Thus both the size of the bin and its location in the network are decision variables with important ramifications on performance of such a system.

It was assumed by the authors that the time between failures and time to repair the belts are exponentially distributed random variables. This permits the system to be modeled as a continuous-time Markov process. For a simple two-conveyor system with a bunker between the first and second conveyor, state is described by the triplet (up/down status of the inby conveyor, up/down status of the outby conveyor, and proportion of bin volume currently filled). It is clear that one can assess the availability of the system to the production section given the value of these variables. For larger networks, it was shown that the belts inby the bunker and the belts outby the bunker may be considered in aggregate analogous to the two-belt system to reduce the state space and simplify computations. Expressions were derived for steady-state availability of the conveyor/bunker system to the production system as a function of the bunker volume, and the time to failure and the time to repair distributions for the individual conveyors.

Analysis of these expressions suggested the following strategy for optimal specification of bunker storage. The bunker should be located at the location where the differential between the availability of the inby belts and the outby belts is the smallest, that is, at the location where these availabilities are closest to being equal. (Simple expressions can be used to compute the availability for an arbitrary serial chain of belts.) It is shown that the rate of increase of availability decreases and reaches an asymptote as bunker volume increases. Volume should be set prior to reaching the asymptote. Plots can be constructed to aid this decision.

Perhaps the most serious criticism of this work might be the assumption of constant flow on the belt when the system is up. The ramifications of this assumption are not known. It appears that additional work might also be done on how one combines multiple production sections serviced by the same network, although this was addressed by the authors to a limited extent.

Sevim Model: The work of Sevim (1987) focused on a different aspect of performance: the use of a bin to smooth outflow from multiple production sections that operate in a discontinuous fashion. Here, in contrast to the work just discussed, greater attention was given to modeling the production sections. Output from the face was modeled as a semi-Markov process, which allows for an arbitrary form of distribution of the times between up and down states as well as different classes of up and down states. It was assumed that the rate of production was constant during the up periods. Transient distributions of the probability of coal flow from a section were established for the first half-shift (from arrival at the section until the lunch break). It was also shown how these could be combined for multiple sections serviced by the same belt network. One thereby has knowledge of the probabilities (as a function of time of day) associated with the various aggregate levels of material flow on a belt. These distributions were then used to describe the inflow into a bin installed for the purpose of smoothing flow. A heuristic approach was established for sizing this bin. Both constant and controlled outflow rates from the bin were considered. Similar work applied to control of flow in slurry transport systems is presented in Sevim and Yegulalp (1984).

9.4.5.2 Queuing Models of Mine Production Systems

The basic structure of queuing systems is as follows. A *calling population* provides a source of *customers* who request service at some *service station.* One should take a broad interpretation of the term customers; in a mining context it might refer to shuttle cars waiting to be filled, belts needing repair, or even, as will be seen in an example that follows, working faces requesting an operation in the cut, drill, blast, load, roof-bolt sequence of coal mining operations. The calling population may either be finite or infinite. The latter is most common in queuing models and is simpler analytically. It typically refers to a large source of customers external to the system such as the body of potential customers for a fast food restaurant. In each of the three mining cases discussed before, and probably in most queuing systems of interest in mining, the calling population is finite and fixed.

The service station referred to previously contains one or more parallel *service channels;* each such channel is called a *server.* If the number of channels is finite, there is a limit in the number of customers that can be serviced simultaneously. Arriving customers finding all channels filled must wait for service in a *queue.* The queue may or may not correspond to a waiting line in the real-world system being modeled. For instance, broken machines (customers) may have to wait for the repair crew (servers) to finish a number of jobs before being serviced. But the machines are never "lined up" physically.

The queue itself provides an opportunity for controlling the system. It may be desirable to service some waiting customers ahead of others independent of their order of arrival to the queue. For example, one might load 80-ton (72-t) trucks ahead of 50-ton (45-t) trucks regardless of which truck arrived first. These controls are specified in terms of a *queue discipline,* which gives rules for determining the priority for removing customers from the queue when a service channel becomes available.

The queue may have limited capacity, and customers arriving when the queue is full may be balked from the system or sent to another service station. Each server is typically assumed able to service only one customer at a time. This customer is retained and occupies the server for a period of time called the *service time.* For example, an empty, spotted truck ties up the loading shovel until it is filled. Once service is completed, the server is now available to service another customer and the customer is released.

The system may consist of multiple service stations. A single customer may progress from one service station to the next in a fixed or a flexible sequence. This arrangement is frequently called a *queuing network.* In many mining systems, the customer is cycled among a number of service stations. For example, the truck goes from a loader to a dump site and back again. When multiple services stations are involved, state is often described in terms of the number of customers at each station.

Overall, the structure of queuing models is quite broad and flexible. Hopefully this brief discussion of structure has left the reader with the impression that a number of mining processes might potentially be modeled in this framework.

Two assumptions are frequently made to make queuing models more tractable analytically. First is that one is interested in equilibrium probabilities at steady state as opposed to transient performance of the system. As explained in 9.4.3.4, the acceptability of this assumption depends on the objectives of the study, not the system itself. Second is that the temporal variables of the system—customer interarrival times, customer service times, interstation transfer times—are exponentially distributed. This allows the theory of continuous-time Markov processes to be applied to compute the steady-state probabilities.

The latter assumption is a restrictive one. Though the exponential distribution is an excellent model for many situations such as time between failures of certain types of components, time between arrivals to a system from a large external calling population where customers act independently, it is not a good model for service times (e.g., time to load a truck) and interstation transit times (e.g., time to travel from the loader site to the dump-site) in many mining applications.

An approach that may be used to overcome this limitation is to employ what are called *generalized Erlangian distributions* or *phase-type distributions.* The general idea is to view a temporal variable with an arbitrary distribution, such as a service time, as being executed in a sequence of phases. The order of the phases is defined by either series or parallel structures or a combination of both. A serial structure implies that one phase is executed sequentially after another until the required number of phases in the series is completed. A parallel structure implies that multiple serial structures are available and one of these is selected with a specified probability then executed as just described. Time for each phase in the sequence is exponentially distributed. The phase structure and parameters of these exponential distributions may be determined so that as customers pass through the sequence of phases, the distribution of total elapsed time closely matches the arbitrary distribution that the modeler wants. Since all times are now exponentially distributed, the result is that a continuous-time Markov process model can be employed, with accompanying advantages in tractability. This, however, comes at the expense of an expansion of the state space relative to the initial problem. The theory of stochastic networks (Kelly, 1979) provides means for computing the steady state probabilities for the various states of the system.

Recently, this approach has been applied to mining systems in the work by Kappas and Yegulalp (1987). In their model of a room and pillar coal mining operation, working faces are viewed as customers that transit through a fixed series of service stations corresponding to the undercutting, drilling, blasting, loading, and roof-bolting operations of the conventional mining cycle. Distributions of general form were fitted to field data on the time required to complete each of these operations. A phase structure was fitted for each service station that resulted in a distribution of total elapsed time that matched the first two moments of the general distribution exactly (mean and variance) and was close to the third moment (skewness). They used specialized versions of computational algorithms for computing steady-state probabilities of the various states, which were given in terms of the number of faces undergoing each operation of the mining cycle. From these probabilities, various aspects of performance, such as steady-state production rate, could be obtained. They compared the effect of various numbers of working faces on production rates. Though a number of shortcomings were noted by the authors, including the uncertain utility of steady-state measures and inadequacies in accommodating spatial aspects of entry layouts when comparing system alternatives, the work certainly shows promise in bridging the major gaps in applying queuing theory to mining systems over previous applications where exponential times for activities were assumed.

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Section 10 Geomechanics

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Chapter 10.0
INTRODUCTION

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Geomechanics signifies the science and engineering of soil and rock materials and of rock masses. It is a field of professional practice and research that draws heavily on the elements of soil and rock mechanics.

Geomechanics contributes to a number of disciplines: mining engineering, civil engineering, geological engineering, petroleum and natural gas engineering, and engineering geology (including the elements of geohydrology and geophysics). These disciplines are involved in the design and construction of such projects as mines, tunnels, foundations for structures, excavated rock slopes, dams, shafts, boreholes, oil reservoir wells, underground storage facilities for oil and gas, and hazardous waste repositories. Geomechanics allows us to improve both our environment and our quality of life.

In the context of this *Handbook*, the scope of geomechanics is discussed mainly from the point of view of applications to mining engineering. However, the coverage of the subject material is so comprehensive that it is relevant to other disciplines as well.

Section 10 covers the field of geomechanics in a structured fashion, beginning with the fundamental principles and leading into practical applications in mining engineering. Throughout the treatment of the subject, knowledge of geologic conditions is assumed since this is discussed in other sections of the *Handbook*.

An important feature of this section is a series of practical examples illustrating the various aspects of geomechanics.

Since geomechanics is concerned with both soil and rock mechanics, Chapter 10.1 introduces the fundamentals of soil mechanics while Chapter 10.2 does the same for rock mechanics. In these two chapters, the relevant principles of mechanics are given, and the properties of soil and rock are discussed together with the methods of their determination. Chapter 10.1 discusses such practical aspects as the bearing capacity and slope stability of soil and weak rock, while Chapter 10.2 emphasizes approaches for solution of problems in rock mechanics.

Chapter 10.3 deals with geomechanics instrumentation, which is of vital importance for assessing the stability of surface and underground excavations as well as in determining rock mass properties. For this purpose, the principles of rock instrumentation are treated at some length, but comprehensive information is also provided on the multitude of instruments currently available to the mining engineer.

Slope stability is the next applied topic, covered in Chapter 10.4, which is most relevant to surface mining. It includes the discussion of economic optimization of slopes and treats stability criteria as well as various modes of slope failure. Next, stresses in a slope and geologic data collection are discussed, and the reader is led through the design methods for slopes and the management of slope operations.

Chapter 10.5 deals with ground control mainly in underground mining. It presents methods for geologic site characterization and engineering rock mass classifications, together with determination of rock mass properties and in situ stresses. This is followed by practical design considerations of rock bolt reinforcement, design of pillars, and assessment of mine floor stability. The chapter concludes with a detailed treatment of ground control in longwall mining.

This last subject leads logically to a discussion of surface subsidence, which is presented in Chapter 10.6. Principles of subsidence as well as control and prevention of damage are dealt with in detail. Issues pertinent to abandoned mined lands and the legal aspects of mining subsidence are also included.

To ensure that all major applications of geomechanics are covered in this section, Chapter 10.7 is a potpourri of geomechanics applications not covered in other chapters of the section, such as nuclear waste disposal in mined underground repositories. The chapter also focuses on preserving rock mass integrity and deals with special topics that may face the mining engineer.

Overall, this section on geomechanics should enable the reader, already armed with geologic information, to determine soil and rock mass properties, establish in situ stresses, and design safe and economic excavations for surface and underground mining operations.

Chapter 10.1
SOIL MECHANICS

E.J. CORDING AND A.F. CEPEDA-DÍAZ

10.1.1. DEFINITIONS

Soil is a three-phase (solid, liquid, and gaseous) medium consisting of weakly bonded or unbonded mineral grains and voids filled with water and air. Mineral grains are more strongly bonded in rock than in soil, although in reality, there is no sharp boundary between soil and rock. In particular, properties of weak rock bear many similarities to those of soil and are included in the discussion in this chapter. *Soil mechanics* refers to the fundamental principles that govern the behavior of soils as related to the design, construction, and performance of engineering works. The most significant engineering properties of soil and rock are permeability, compressibility, strength, and durability.

Principles of soil mechanics are applied toward the solution of a variety of problems including seepage or water flow through soils; settlement of foundations and fills on compressible soils; swelling and slaking of clay-shales in foundations, slopes, and underground openings; earth pressures against retaining walls; stability of slopes and excavations; design of earth dams and retention ponds; bearing capacity of foundations and mine pillars on weak soils; and tunneling in soils, weak rock, or fault zones.

10.1.2. SOIL AND ROCK PROPERTIES

10.1.2.1 Characteristics of Soil

Soil and weak rock possess several characteristics that differentiate them from other engineering materials.

Fine-grained soils interact with water in such a way that there is no resemblance between their dry and wet behavior, and their properties are largely influenced by the physicochemistry of the soil-water system. Fine-grained saturated soils are generally cohesive as a result of the net balance between attractive and repulsive inter-particle forces.

Soft or weak sedimentary rocks, such as shale, and other *rock-soil transitional materials,* such as weathered rock or residual soils, not only have strengths that may approach those of highly overconsolidated soils but also have behavior similar to soils, inasmuch as it is influenced by the physicochemistry of the solid-water system. Thus the principles of soil mechanics can be also used to explain the behavior of so-called *weak rock* whose pore water response, volume change behavior (e.g., swelling), and shear strength characteristics are similar to those of soil.

Stiff soils and weak rocks commonly contain fissures and joints that affect permeability, compressibility, strength, and durability of the soil/rock mass. Thus the principles used in rock mechanics for evaluating a discontinuous medium are also applicable to these materials.

The *concept of effective stress* is one of the most important principles of soil mechanics. Both compressibility and strength of soil are controlled by effective stress. The concept establishes that for a saturated soil-water system, the effective stress σ' applied to the soil skeleton is equal to the total stress σ externally applied minus the existing pore water pressure u:

$$\sigma' = \sigma - u \tag{10.1.1}$$

The maximum shearing resistance (shear strength) that a soil can develop is a function of the effective stress acting on the soil. The pore water pressure is positive when u is in compression (greater than atmospheric pressure). An increase in the pore water pressure for a given total stress will reduce the effective stress and, consequently, the shear strength of the soil. The pore water pressure is negative, on the other hand, when u is in tension (less than atmospheric pressure). In the latter case, which occurs for saturated and partially saturated soils above the groundwater table, $\sigma' \geq \sigma$. The concept of effective stress applies not only to soils but to rock and other materials with continuous voids formed by pore spaces or joints.

Saturated materials that are soft enough to undergo significant volume change upon application or removal of pressure (soil and many weak rocks) exhibit time-dependent consolidation or swelling, which is a manifestation of the principle of effective stress. Upon application of an increment of pressure $\Delta\sigma$ to a saturated soil, its volume will not decrease significantly until water is expelled from the pore spaces. Because the soil skeleton is more compressible than water, the pressure increment is initially applied to the water, thus $\Delta u \approx \Delta\sigma$, the excess pore-water pressure. With time, as the water is squeezed from the pore spaces, the volume decreases, and the water pressure reduces to its original value ($\Delta u = 0$). The pressure increment is progressively transferred from the water to the soil skeleton, until finally the pressure increment becomes fully an effective pressure, $\Delta\sigma' = \Delta\sigma$. This time-dependent process is termed *consolidation* for the volume decrease resulting from an increase in effective stress, and *swelling* for the volume increase resulting from a decrease in effective stress. Excess pore-water pressure dissipates rapidly for permeable materials such as sands, and slowly for low permeability soils such as clay.

For a large mass of low permeability soil, the process is so slow that significant changes in volume or water content may not occur over the period of loading, so that the material remains undrained, and its strength and stiffness can be assumed to be unchanged from the initial condition, before the load increment was applied. Upon application of load and over a long period of time, drainage occurs and the soil consolidates, with corresponding increase in strength and stiffness. Upon unloading, however, the soil swells with time, and strength and stiffness decrease.

In contrast, most sandy soils subjected to static loads are assumed to be loaded slowly enough that significant excess pore water pressure Δu does not build up; therefore the soil is drained, and the pressure increment is applied directly to the soil skeleton throughout the loading increment ($\Delta\sigma' = \Delta\sigma$; $\Delta u = 0$).

10.1.2.2 Index Properties of Soil and Weak Rock

The spatial distribution and properties of soil, rock, fluids, and gas at a site are investigated utilizing a knowledge of the local geology and the information gathered from strategically located boreholes, test pits, and test shafts. These borings and excavations are used to log the distribution and changes in material properties, to index the soil properties by means of in situ tests such as standard penetration resistance and water pressure tests, and to collect either disturbed (or remolded) samples for laboratory index testing, such as water content and soil plasticity, or undisturbed samples for unconfined compressive strength and more elaborate laboratory tests.

Soil and weak rock are classified for engineering purposes in terms of *index properties,* which are by definition significant (i.e., engineering behavior can be inferred from them), simple and quick to measure, reproducible, and easy to express (e.g., with a numerical value).

Properties of the Disturbed Material: A soil is considered *fine grained* (clay and silt) or *coarse grained* (sand and gravel) if its predominant grain size is smaller or greater, respectively, than the smallest particle diameter that can be identified with the naked eye (i.e., 74 *m*m or US No. 200 Standard sieve).

With little to no clay or silt, coarse-grained soils are generally cohesionless, and their properties remain practically unchanged whether the soil is unsaturated or fully saturated with water. Coarse-grained soils are indexed in terms of their grain size and grain size distribution.

The Atterberg limits serve as an index test for measuring the cohesive characteristics of fine-grained soils, referred to as *soil plasticity.* The Atterberg limits (Anon., 1985a) define the range of water contents (percentage ratio between the weight of water and the weight of solids) in which the disaggregated finer soil fraction can deform without running like a liquid (liquid limit) and without rupturing (plastic limit) under the standard test conditions. The *shrinkage limit* is defined as the water content at which the fully saturated soil cake stops decreasing in volume. Any further moisture loss simply starts emptying the soil pores. High plasticity soils, having a large plastic range, are subject to high swelling and shrinkage.

Both grain size and plasticity form the basis for the categories in the Unified Soil Classification System shown in Table 10.1.1 (Anon., 1953). These index tests are performed on disturbed samples, which do not retain the structure and density of the soil or rock in situ.

Undisturbed Properties of the Material: In order to evaluate in situ permeability, compressibility, or strength, it is necessary to test the undisturbed material, either with field tests performed on the in-place material or with laboratory tests performed on relatively undisturbed samples collected from the field.

The structure and density of a soil, and therefore its strength and compressibility, are related to its past stress history as well as to the present stresses existing in the ground. The past stress history is largely summarized in terms of the preconsolidation pressures σ_p, which in some cases is equal to the maximum previous effective overburden pressure that acted on the soil but is also influenced by other diagenetic processes (such as bonding and cementation) to which the soil has been subjected in its geologic past. These processes create an overconsolidated material, that has a greater density, higher strength, and lower compressibility than would be obtained for a normally consolidated material, which has never been subjected to a pressure greater than the existing effective overburden pressure.

Standard penetration resistance in granular cohesionless soils and shear vane resistance in cohesive soils are in situ-measured index properties typically included in subsurface exploration programs to identify and characterize the strength and compressibility of various soil sublayers. The laboratory unconfined compressive strength also serves as an index of the strength and compressibility of cohesive soils. The natural water content relative to the plastic and liquid limits is also used to infer the strength and stiffness of cohesive soils.

Standard penetration resistance (Anon., 1985b) is defined as the number of blows N required to drive a standard 2-in. (50.8-mm) OD split spoon sampler 12 in. (305 mm) long under a weight of 140 lb (63.6 kg) falling 30 in. (762 mm). The N-blow count is used as an index of the density of granular soils and is correlated with relative density in Table 10.1.2. Relative density is defined as

$$D_r = 100\% \times (e_{\max} - e)/(e_{\max} - e_{\min}) \qquad (10.1.2)$$

where $e_{\min}$ is void ratio of soil in its densest condition, $e_{\max}$ is void ratio of soil in its loosest condition, and e is in-place void ratio. There is also a rough correlation between the standard penetration resistance and the unconfined compressive strength of clay, as shown in Table 10.1.2.

High blow counts (N > 100/ft) are usually indicative of boulders or weak rock. It is possible to drive the split spoon sampler into weak rock, such as a poorly cemented sandstone, or a weathered rock. The material obtained in the sampler is usually broken and even completely disaggregated in the process of sampling, so that the original structure of the rock is lost. In such a case, it is desirable to obtain additional samples using a double- or triple-tube rock core bit, perhaps one that has a thin-walled inner liner to protect the sample, in order to recover a sample that allows the structure of the rock to be retained. It is often difficult to obtain representative samples from small-diameter boreholes and samples (2 in. or 50.8 mm) in such materials. The use of large-diameter core samples (Adachi, 1989) or drilled shafts large enough for access for inspection and mapping has proven quite valuable in such conditions (Cording, et al., 1989).

10.1.2.3 Permeability

Permeability k is expressed in units of length/time and indicates the discharge velocity with which water under a unit hydraulic gradient flows through soil and rock. Procedures are available for measuring permeability in the field and in the laboratory. Results will depend on the direction of the water flow during the test because soil and rock generally have a direction-dependent permeability or anisotropy. In a bedded deposit, horizontal permeability k_h usually exceeds vertical permeability k_v. For example, for homogeneous soft clay, typical ranges for horizontal permeability are k_h = 1.0 to 1.5 k_v, whereas for varved clay typical ranges are k_h = 1.5 to 5.0 k_v. Much greater ratios of k_h/k_v will apply to bedded deposits having alternating layers of clay and sand.

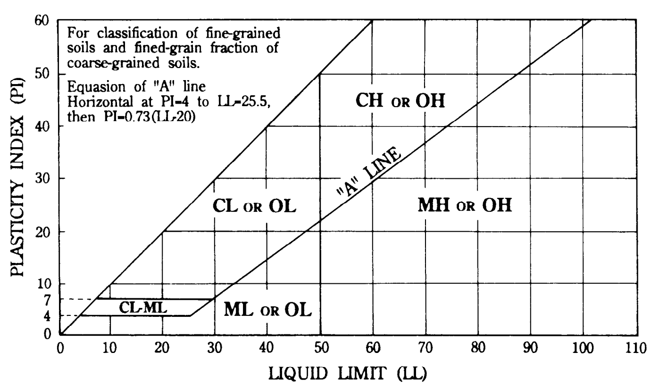
The coefficient of permeability k is a function of the size, shape (*tortuosity*), and number of the pores or joints (channels) in the soil/rock mass. Table 10.1.3 presents typical values for k for various soil and weak-rock materials. For a given soil or weak rock, k decreases with decreasing values of void ratio e (volume of voids /volume of solids) and therefore with increasing effective stress. The slope of the e-log k relationship is generally referred to as $C_k = \Delta e/\Delta \log k$. For soft clays C_k = 0.5 to 2.0, whereas for shales C_k < 0.5.

Permeability is particularly sensitive to heterogeneities such as stratification, fissures, and joints, and thus their presence determines either the minimum specimen size required in the laboratory to have a representative sample or the need for in situ testing to obtain a valid measurement. Disturbance of the soil during boring or sampling can also have a strong effect on the measurement of permeability. In many cases, laboratory determination of permeability does not give results comparable to the in situ permeability. In the field, the local permeability surrounding a borehole can be assessed by a water pressure test in which a quantity of water is pumped into or extracted from a section of a borehole under a given pressure. More comprehensive results, indicative of the performance of a larger portion of an aquifer, are obtained by pump tests in which piezometric

Table 10.1.1. Soil Classification Chart

Criteria for Assigning Group Symbols and Names Using Laboratory Tests[a]			Soil Classification Group Symbol, Name[b]	
Coarse-grained soils: > 50% retained on #200 sieve:				
Gravels: > 50% of coarse fraction retained on #200 sieve	Clean gravels > 5% fines[c]	Cu ≥ 4 and 1 ≤ Cc ≤ 3[e]	GW	Well-graded gravel[f]
		Cu < 4 and/or 1 > Cc > 3[e]	GW	Poorly graded gravel[f]
	Gravels with Fines > 12% fines[c]	Fines classed as ML or MH	GM	Silty gravel[f,g,h]
		Fines classed as CL or CH	GC	Clayey gravel[f,g,h]
Sands: > 50% of coarse fraction passes #200 sieve	Clean Sands <5% fines[d]	$C_u \geq 6$ and $1 \leq C_c \leq 3$[e]	SW	Well-graded sand[i]
		$C_u < 6$ &/or $1 > C_c > 3$[e]	SP	Poorly graded sand[i]
	Sands with fines > 12% fines[d]	Fines classed as ML or MH	SM	Silty sand[g,h,i]
		Fines classed as CL or CH	SC	Clayey sand[g,h,i]
Fine-grained soils: <50% retained on #200 sieve:				
Silts & clays: liquid limit, $L_w < 50$	Inorganic	PI > 7 & above "A" line	CL	Lean clay[k,l,m]
		PI < 4 or below "A" line	ML	Silt[k,l,m]
	Organic	$\dfrac{L_w\text{, oven dried}}{L_w\text{, not dried}} < 0.75$	OL	Organic clay[k,l,m,n] or silt[k,l,m,o]
Silts & clays: $L_w < 50$	Inorganic	PI on or above "A" line	CH	Fat clay[k,l,n]
		PI below "A" line	MH	Elastic silt[k,l,m]
	Organic	$\dfrac{L_w\text{, oven dried}}{L_w\text{, not dried}} < 0.75$	OH	Organic clay[k,l,m,p] or silt[k,l,m,q]
Highly organic soils		Dark organic matter, odor	PT	Peat

[a]Based on the material passing the 3-in. (75-mm) sieve.
[b]If field sample contained cobbles or boulders, or both, add "with cobbles or boulders, or both" to group name.
[c,d]Gravels or sands with 5 to 12% fines require dual symbols, such as: GW-GM well-graded gravel with silt, SP-SC poorly graded sand with clay.
[e]$C_u = D_{60}/D_{10}$ $C_c = \dfrac{(D_{30})^2}{D_{10} \times D_{60}}$
[f]If soil contains ≥ 15% sand, add "with sand" to group name.
[g]If fines classify as CL-ML, use dual symbol GC-GM, or SC-SM.
[h]If fines are organic, add "with organic fines" to group name.
[i]If soil contains ≥ 15% gravel, add "with gravel" to group name.
[j]If Atterberg limits plot in hatched area, soil is a CL-ML, silty clay.
[k]If soil contains 15 to 29% plus #200, add "with sand" or "with gravel."
[l]If soil contains ≥ 30% plus #200, predominantly sand, add "sandy."
[m]If soil contains ≥ 30% plus #200, predominantly gravel, add "gravelly."
[n]PI ≥ 4 and on or above "A" line.
[o]PI < 4 or plots below "A" line.
[p]PI on or above "A" line.
[q]PI plots below "A" line.



PLASTICITY CHART

Table 10.1.2. Characterization of Relative Density and Strength

Relative Density of Sand			Strength of Clay		
N^a	Relative Density %	Descriptive Term	N^a	Unconfined Compressive, tons/ft^2	Consistency
0–4	0–15	Very loose	<2	<0.25	Very soft
4–10	15–35	Loose	2–4	0.25–0.50	Soft
10–30	35–65	Medium	4–8	0.50–1.00	Medium
30–50	65–85	Dense	8–15	1.00–2.00	Stiff
>50	85–100	Very dense	15–30	2.00–	Very stiff
			>30	>4.00	Hard

[a]Standard Penetration Resistance, blows/ft
Conversion factor: 1 ton/ft^2 = 1.05 MPa.

Table 10.1.3. Coefficient of Permeability

Material Description	k, cm/s
Clean gravel	$1–10^2$
Coarse to medium sand	$10^{-2}–1$
Fine sand	$10^{-4}–10^{-2}$
Silt	$10^{-6}–10^{-4}$
Natural clays	$10^{-8}–10^{-6}$
Highly overconsolidated clays and shales	$10^{-12}–10^{-8}$

Conversion factor: 1 in. = 2.54 cm

levels are monitored in several borings, with time, as a nearby well is pumped.

Darcy's empirical equation (c. 1856), initially established for the flow of water through coarse-grained soils, has also proven to be valid for water flow through fine-grained soils including heavily overconsolidated clays and shales (Mesri and Cepeda-Díaz, 1987). *Darcy's law* states that the discharge velocity v_s in a direction s is equal to the coefficient of permeability in the s direction k_s times the hydraulic gradient i, or

$$v_s = k_s\, i \qquad (10.1.3)$$

where the dimensionless parameter i is equal to the change of total hydraulic head dh in the distance interval ds along the macroscopic or apparent flow path $s(i = dh/ds)$. Therefore, the time-rate of flow q (volume flow rate) through an area A perpendicular to the direction of the flow is given by

$$q = k_s i A \qquad (10.1.4)$$

which is used to directly evaluate one-dimensional *steady-state flow*.

A modified version of the previous equation is used to calculate the quantity of flow for two-dimensional steady-state flow through soil. The solution to the mathematical problem can be obtained graphically in the form of a scaled flow net (Cedergren, 1967) that, in the case of a homogeneous and isotropic medium ($k_h = k_v$), consists of two mutually orthogonal families of curves (flow lines and equipotential lines) forming curvilinear "square" cells within the boundaries of the problem and thus also forming both a number of flow channels n_f and a number of equipotential drops n_d, for a given head loss h_ℓ (Fig. 10.1.1). Since each flow channel carries the same flow rate as the others and continuity of flow (conservation of mass) must be satisfied, the fluid motion can be described at the individual square cell level where the hydraulic gradient i times the discharge area of the channel is equal to h_ℓ/n_d per unit of width and thus iA (per unit of width for the entire flow net) $= (h_\ell \cdot n_f)/n_d$. For two-dimensional steady-state flow, therefore, the time-rate of flow ($q = k_s iA$) is given by

$$q = k_s(n_f/n_d)h_\ell \qquad (10.1.5)$$

per unit of width perpendicular to the plane of the flow net. A flow net solution enables the hydraulic gradient and the pore water pressure to be calculated at any location within the hydraulic boundaries considered, and thus the pore water pressures as a result of water seepage can be determined along potential surfaces of sliding in order to evaluate the stability of a soil mass.

In many materials, such as jointed rock or bedded soils, permeabilities are not uniform. As a result, the flow net will be distorted, with flow lines and equipotential lines not forming squares.

10.1.2.4 Compressibility and Volume Change Characteristics

Components of Compressibility: The compressibility or expansibility of a soil or weak rock can be separated into several components. They are (1) initial "elastic" deformation, (2) time-dependent hydrodynamic volume change (primary consolidation or swelling), and (3) time-dependent volume change (secondary consolidation or swelling) and/or time-dependent deformation (creep).

Initial "Elastic" Deformation: Immediately after the application or removal of load, a fine-grained saturated soil will deform without any change in volume or water content, referred to as the "undrained condition." In an unsaturated soil or in a coarse-grained soil, the initial elastic deformations may include some volume changes as well as a distortional component.

Assuming that the material has not yet failed, the amount of strain for a given pressure increment can be estimated using a modulus of deformation, that is equivalent to a modulus of elasticity E but represents a tangent or secant to a stress-strain curve that is usually nonlinear. The deformation modulus E_u of a sample is determined in the laboratory from unconfined or triaxial compression tests, in which the specimen is subjected to an increasing axial stress σ_1, while the all-round confining pressure σ_3 is maintained constant. The slope of the curve relating the stress-difference, $\sigma_1 - \sigma_3$, vs. axial strain ϵ represents the modulus of the sample, and $(\sigma_1 - \sigma_3)_{\max}$ is the compressive strength. The unconfined compressive strength is generally referred to as q_u. For most soils and weak rocks, E_u = (50 to 300) q_u.

The in situ modulus of deformation is expected to differ from that determined on a laboratory specimen. For most soils, sample disturbance causes a significant reduction in soil stiffness, and

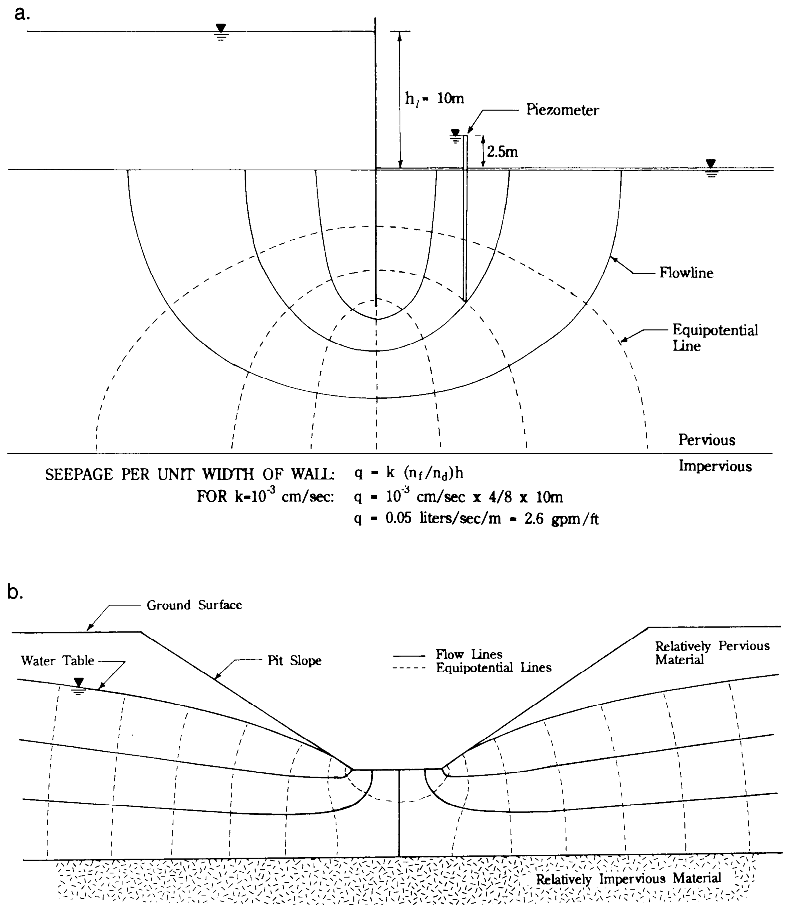


Fig. 10.1.1. Flow nets for medium soil with homogeneous, isotropic permeability. Conversion factors: 1 in. = 2.54 cm, 1 ft = 0.3048 m.

thus the laboratory test generally underestimates the in situ modulus of deformation. On the other hand, soil and rock masses that contain fissures or joints usually have a lower modulus than that of the intact laboratory sample, which does not contain a representative amount of the fissures or joints that are present in situ. In this case, the laboratory modulus would overestimate the in situ modulus of deformation.

The greater the stiffness of the soil or rock, the more significant is the initial deformation with respect to the overall deformation experienced with time by the soil or rock mass.

Time-dependent Volume Change: In a saturated fine-grained soil, the initial elastic deformations described previously occur with very little volume change in the soil matrix. With time, as the load remains in place, water is squeezed out of the soil matrix and the volume of the soil decreases. This process is referred to as *primary consolidation.* In soft clays, primary consolidation is the cause of most of the settlement that the clay will undergo upon application of a load. Compression that occurs with time, after the effective stress σ' has reached a final value under either isotropic loading or one-dimensional deformation conditions, is called *secondary consolidation,* and is a creep phenomenon.

Upon unloading, as water is sucked into the soil, the volume increases and the effective stress decreases. This time-dependent process is called *primary swelling.* Swelling that occurs with time after σ' has reached a final value (under either isotropic unloading or one-dimensional deformation conditions) is called *secondary swelling.*

For fine-grained soils, the magnitude of settlement or heave occurring during primary consolidation or swelling is determined in the laboratory by means of the one-dimensional consolidation test in which the specimen is restrained so that lateral strains are zero as vertical load is applied to the sample. As each load decrement or increment is applied, water is allowed to flow into or out of the soil through porous stones at the top and bottom of the sample, and the vertical deformation of the specimen is measured with time. The change in void ratio of the sample is proportional to the vertical displacement. The end of the primary consolidation or swelling, at time = t_p, occurs when the excess pore pressure, generated upon application of each load increment, has completely dissipated. This can be determined graphically.

For a homogeneous soil layer of thickness H and initial void ratio e_o, settlement or heave ΔH is given by

$$\Delta H = \{(e - e_o)/(1 + e_o)\} \times H \qquad (10.1.6)$$

where e is the void ratio at any particular vertical stress σ'_v and/or time t. The *magnitude of settlement* or heave at the end of primary consolidation or swelling for fine-grained soils is generally obtained from a plot of void ratio e vs. the logarithm of the effective stress σ'_v applied to the sample. Fig. 10.1.2 shows the typical e-log σ'_v relationship consisting of three parts: recompression, compression, and rebound for a heavily overconsolidated clay (Patapsco shale) with a preconsolidation pressure σ'_p of 38 tons/ft^2 or 40 MPa. The slope of the curve, $\Delta e/\Delta \log \sigma'_v$, for each distinct portion is identified as the recompression index C_r, compression index C_c, and swelling index C_s. Note that in the recompression range, the corresponding settlement is significantly smaller than that resulting when σ'_{vf} (final σ'_v) is in the compression range. Loading of soft clays, whose preconsolida-

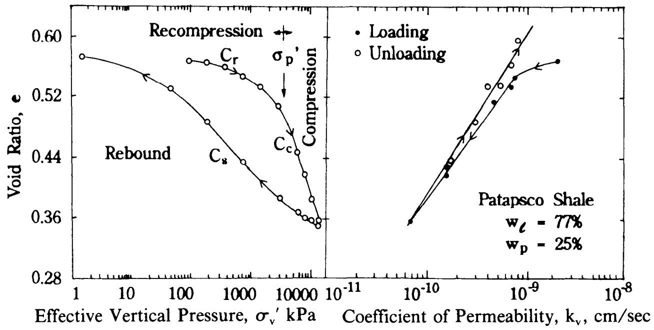


Fig. 10.1.2. Typical stress-volume change and permeability characteristics of soil and weak rock (after Cepeda-Díaz, 1987). Conversion factors: 1 in. = 2.54 cm, 1 psi^2 = 6.895 kPa.

tion pressure σ'_p is relatively small and the compression index C_c relatively large, can result in large settlements when the pressures applied cause the preconsolidation pressure to be exceeded. Magnitude of heave can be estimated using the rebound curve whose slope is C_s. Fig. 10.1.2 also presents direct measured permeability data at various σ'_v and corresponding e in the form of an e-log k_v plot, for parallel examination with the e-log σ'_v relationship.

The *time-rate of settlement* (*or heave*), which is controlled by the rate at which pore water can be expelled (or taken in), is a function of the permeability k and the compressibility (or swellability) of the soil, $a_v = \Delta e/\Delta\sigma'_v$, as expressed by the coefficient of consolidation (or swelling), $c_v = k(1+e)/a_v\gamma_w$, according to the Terzaghi theory of consolidation (Terzaghi and Peck, 1967). The mathematical solution of the latter theory provides the relationship between the average degree of consolidation (or swelling), U in % $= \Delta H/\Delta H_p$, and a dimensionless time factor T whose numerical value is equal to $c_v t/H^2$, where t is real time and H is the maximum drainage distance. Therefore, once c_v is evaluated (usually, by fitting Terzaghi's theory to the observed deformation-time data in the one-dimensional consolidation test), the settlement ΔH at any time $t < t_p$ during the consolidation process can be predicted. For example, according to the Terzaghi theory of consolidation (or swelling), the time factor for U = 95% *is* T_{95} = 1.12. Therefore, the (real) time for 95% average degree of consolidation (or swelling) to occur is given by $t_{95} = 1.12\ H^2/c_{cv}$. For a maximum drainage distance H = 20 ft (6.1 m) and a coefficient of consolidation (or swelling) of 10 ft^2/yr (0.93 m^2/s), the time for end-of-primary to occur can be approximated as $t_p \approx t_{95}$ = 1.12 (20 ft)^2/(10 ft^2/yr) = 45 yr. In soils containing lenses of sand with a clay or silt, the effective drainage distance is difficult to determine, and hence the time to achieve consolidation or swelling is difficult to estimate. One of the most effective means is to monitor the settlement of test fills.

Secondary Compression and Swelling: The slope of the observed e-log t curve for $t > t_p$ is called the secondary compression index C_α in the case of consolidation and the secondary swelling index $C_{\alpha s}$ in the case of swelling. Since C_α/C_c is a constant for any one soil, including shales, its value can be used in combination with the e-log σ'_v relationship to compute the secondary settlement at any $t > t_p$ (Mesri and Godlewski, 1977). The $C_{\alpha s}$-*OCR* (final overconsolidation ratio after swelling $\sigma'_{max}/\sigma'_{vf}$) relationship can be also obtained and used to compute the secondary swelling at any time $t > t_p$.

10.1.2.5 Shear Strength

The *shear strength s* or shearing resistance of a soil or rock is a function of the normal effective stress on the shear plane at failure σ'_{nf}. Thus s is controlled by the existing normal effective stress before shear ($\sigma'_n = \sigma_n - u$) and the excess pore water pressure induced during shearing up to failure u_{sf}, $\sigma'_{nf} = \sigma'_n - u_{sf}$. For $u_f = u + u_{sf}$, σ'_{nf} can be expressed as

$$\sigma'_{nf} = \sigma - u_f \tag{10.1.7}$$

During shearing, loose coarse-grained cohesionless soils and normally consolidated clays tend to compress, whereas dense cohesionless soils and overconsolidated clays and shales tend to increase in volume (dilate), as a result of particle rearrangement. Excess pore water pressures develop when water cannot move in or out of the pores fast enough to accommodate the volume changes induced during the shearing process. In this case, the soil is sheared under so-called undrained conditions.

Effective stress analyses can be used to evaluate the strength of a soil for either drained or undrained conditions. However, to use an effective stress analysis in the undrained case, the pore-water pressures generated at failure must be known. This information is difficult to obtain or predict. Therefore, for soft clays subjected to short term loadings, an undrained analysis is used, in which the undrained shear strength s_u is determined from unconfined compression tests, undrained triaxial tests, or in situ vane shear tests.

Sand: With the exception of dynamic loading, clean coarse-grained cohesionless soils can be treated as shearing under drained conditions as a result of their high permeability. Their shear strength can be expressed as

$$s = \sigma'_{nf} \tan \phi' \tag{10.1.8}$$

where ϕ' is drained friction angle (or friction angle in terms of σ' at failure). For granular soils, ϕ' is primarily a function of the relative density of the soil, and thus $\phi' \approx 28°$ for loose sand and $\phi' \approx 44°$ for very dense sand.

Loose saturated sands subject to dynamic loads, such as earthquakes, are susceptible to development of positive pore-water pressures that reduce the effective stresses to zero and liquify the sand.

Fine-grained Soils: In the case of fine-grained soils, their relatively low permeability is such that significant build-up of excess pore water pressure is expected to occur during the loading and unloading operations associated with construction. In a short period of time, the excess pore water pressures that are generated will not have dissipated, and the clay will not have undergone changes in volume or water content. Thus soft clays can be treated as being undrained, with an undrained shear strength s_u that has not changed from its original value. The condition is described in following paragraphs under the heading of undrained shear strength.

Use of the undrained shear strength is simple, and it has application to several significant engineering problems. However, many failures have resulted from using the undrained shear strength when it was not appropriate, particularly in situations in which negative pore-water pressures were generated initially, but then dissipated to cause a reduction in the strength of the soil during the period the undrained shear strength was assumed to be applicable. This is most likely to occur for stiff, overconsolidated clays. In addition, the presence of sand seams or fissures in a clay can cause a more rapid dissipation of negative pore-water pressures than would be otherwise anticipated, leading to an earlier failure during the construction period than expected. Thus, when using the undrained shear strength, it is important to recognize those conditions for which it is unconservative.

For these reasons, it is always a good practice to investigate the stability of fine-grained soils for both the short-term period (during and after the end of construction) when undrained conditions prevail and the long-term period when the material is drained.

Peak, Fully Softened, and Residual Shear Strength: The drained shear strength that a given clay or shale can actually mobilize (peak, fully softened, or residual strength) depends on whether the in situ material is intact, fissured, or contains discontinuities (e.g., bedding planes, shear planes) along which significant movements have occurred or will occur under the displacements imposed on the soil mass.

Peak Shear Strength—If the in situ fine-grained material is relatively intact, the peak shear strength is expected to be mobilized, and thus the failure envelope for the effective stress range considered can be characterized in terms of the "effective" strength parameters c' and ϕ',

$$s = c' + \sigma'_{nf} \tan \phi' \qquad (10.1.9)$$

where c' is the "cohesion" intercept. The greater the magnitude of the preconsolidation pressure σ'_p, the greater is the cohesion intercept c'. For normally consolidated clays, $c' = 0$. Values of ϕ' are typically in the range of 22° ± 10° and reflect the ability of a random arrangement of clay particles (of particular size and shape) to interact and interlock with each other.

In the case of overconsolidated clays, the typical shear stress-strain and strength curves presented in Fig. 10.1.3a show that the peak shearing resistance is mobilized with no major structural rearrangement. Near the maximum shearing resistance, the changes in volumetric strain v are small and mainly in compression.

Fully Softened Shear Strength—Additional shear displacement beyond the *peak (intact) shear strength* is accompanied by softening (volume increase as the disaggregated clay takes up water) and corresponding decrease in shearing resistance as c' decreases. By the time the softening process has been completed, only the *fully softened shear strength* is mobilized ($c' \approx 0$), and the material behaves as a normally consolidated clay (Eq. 10.1.8). Long-term stability of cuts in overconsolidated, fissured clays has been determined to be related to the fully softened shear strength.

The typical shear stress-strain and strength curves for a given sand in both dense and loose states, presented in Fig. 10.1.3b, show to what extent the behavior of dense and loose granular soils is similar to that of overconsolidated clays and normally consolidated clays, respectively. For the nonplaty granular soils, there is no possibility of particle alignment during shear, and thus the *ultimate shear strength* for the dense sand (which loosens after reaching its peak strength) corresponds to the maximum shear strength of the loose sand (which densifies during shear).

Residual Shear Strength—Additional shear displacement, if concentrated along a narrow shear zone, causes further decrease in shearing resistance, below that of the fully softened case, as the plate-shaped particles in the relatively narrow shear zone become increasingly aligned in the direction of shear. In this configuration, the soil or rock is only able to mobilize the *residual shear strength* or minimum s it can ever have, or

$$s = \sigma'_{nf} \tan \phi'_r \qquad (10.1.10)$$

The residual friction angle ϕ'_r is an index property of the natural soil composition, and it reflects the size and shape of the soil particles (Mesri and Cepeda-Díaz, 1986). As shown in Fig. 10.1.4a, for low plasticity clays and shales liquid limit w_ℓ < 50%), ϕ'_r can be as high as 30°, whereas for clays and shales with w_ℓ > 100%, ϕ'_r can be as low as 4 to 5°.

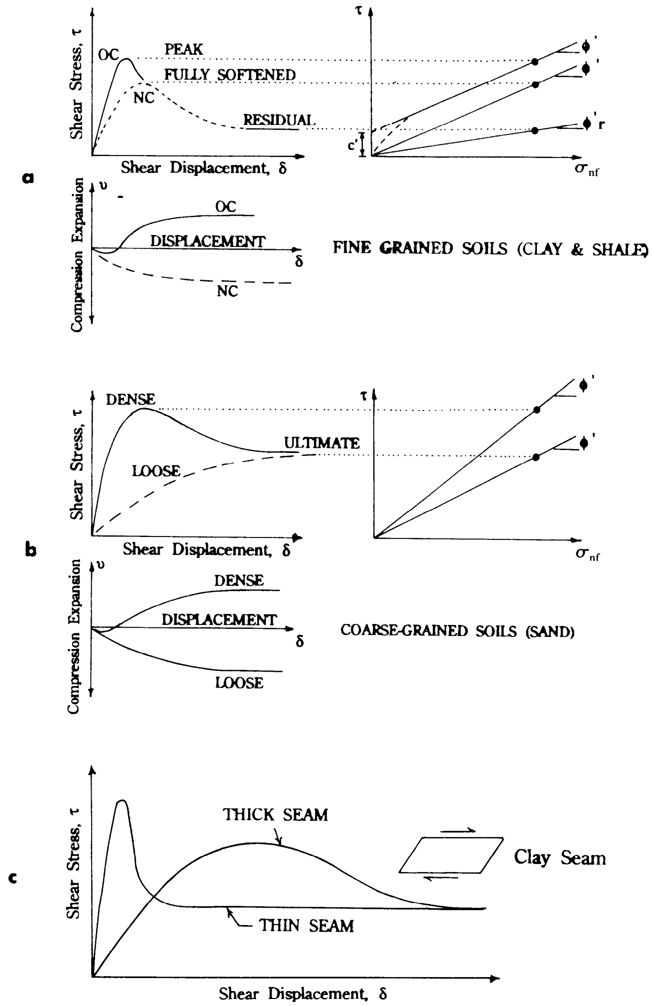


Fig. 10.1.3. Typical stress-strain and strength characteristics of soils and weak rock.

A correlation between the peak friction angle ϕ' and clay mineralogy also exists, and thus there is a correlation between ϕ' and ϕ'_r as shown in Fig. 10.1.4b.

In materials that contain thin seams of soft material or preexisting planes of weakness, such as bedded shales or shear zones, residual shear strengths are reached with much smaller displacements than the displacements required to reach residual strength for a thicker material. Fig. 10.1.3c illustrates the difference in shear displacements observed in direct shear tests on thick and thin seams of clay (Nieto, 1974). In practice, residual strengths are more likely to be representative of thin-bedded materials, such as a bedded shale, in contrast to a massive clay with indistinct bedding. Small displacements along the bedding planes can cause a rapid reduction of the peak strength to the residual. Additionally, any excess pore-water pressures generated in the bedding seam during shear are rapidly dissipated, so that drained conditions may prevail. If the mass of soil or rock presents evidence of discontinuities along which significant movements have already occurred, such as bedding or shear planes, then only the residual shear strength is expected to be mobilized and the failure envelope can be characterized in terms of the residual friction angle ϕ'_r.

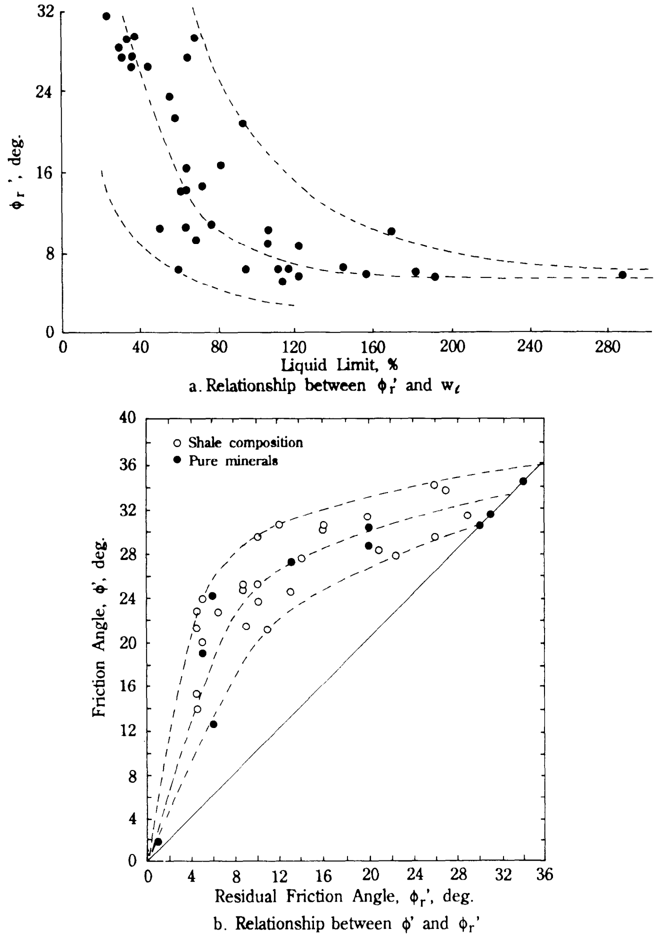


Fig. 10.1.4. Relationship between ϕ_r', ϕ', and *Lw* (after Cepeda-Díaz, 1987).

Undrained Shear Strength: The undrained shear strength of a clay can be obtained in terms of c' and ϕ' as long as u_f can be determined; however, this is often difficult in the field and sometimes infeasible in the laboratory. Therefore, the undrained shear strength s_u is usually determined or approximated from direct and simpler measurements, such as field vane tests, laboratory unconfined compression tests, or unconsolidated undrained triaxial tests on undisturbed samples. The *unconfined compressive strength* ($q_u \approx 2s_u$) is a measure of the consistency of fine-grained cohesive soils, and as such it has been used for classification purposes (Table 10.1.2).

Back-calculation of the mobilized shear strength $(s_u)_{\text{mob}}$ in a number of slope and foundation failures in various soft clay deposits indicates that for the cases investigated, $(s_u)_{\text{mob}} \approx 0.22\ \sigma_p'$ (Mesri and Godlewski, 1977). Typically, s_u determined from laboratory unconfined compression tests is less than $(s_u)_{\text{mob}}$ largely because of sample disturbance. Corrections are also applied to the measured laboratory undrained shear strength to account for field conditions that are not exactly duplicated in the tests, such as strength anisotropy, influence of shearing rate, and progressive failure.

The relationship between the undrained shear strength and the effective stress before shear can also be investigated using triaxial compression tests on undisturbed samples that are first consolidated to the various effective stresses selected and then sheared under undrained conditions. Such tests are usually referred to as consolidated-undrained triaxial tests, or briefly, CU tests.

Drainage Conditions and Shear Strength Parameters in the Field: The undrained shearing of highly overconsolidated clays and shales leads to a decrease in *s* with time as the induced excess negative pore-water pressures gradually dissipate, and thus for this type of material the long-term drained condition is generally the least favorable condition. For normally consolidated clays, on the other hand, the *undrained shear strength* (corresponding to a given σ_n' before shear) may be considerably lower than the corresponding drained shear strength as a result of the positive excess pore water pressures induced during undrained shearing. During foundation loading, for example, the least favorable condition for normally consolidated clays and slightly overconsolidated clays ($OCR < 2$) is generally the short-term undrained condition, and thus using the undrained shear strength for this latter type of materials is usually conservative.

In the case of unloading of slope cuts and other excavations, excess pore water pressures resulting from unloading are superimposed on the excess pore water pressure generated by undrained shearing. This can result in a net negative excess pore water pressure, even for normally or slightly overconsolidated clays. With time, as the excess negative pore water pressure is dissipated, the shear strength will decrease. Thus the undrained shear strength will be unconservative if used to estimate long term strength of excavations, particularly in the case of highly overconsolidated clays or shales.

In the case of bedded shales or weak rock with joints, the strength along the discontinuity, rather than the unconfined compressive strength of the material, will control many of the failures. In some cases, the strength along the bedding surface or other discontinuity can be estimated using a peak friction angle, which may include a dilatant component to account for irregularities on the surface. However, the residual friction angle should be used if irregularities have already been sheared off, or if they will be lost with small displacements. In the case of failure surfaces that cut at an angle to the discontinuity, the peak strength of the intact material may be applicable. However, combined failures, through both intact material and along discontinuities, will often occur in shale and other weak rock, and the interaction of the discontinuity with the intact portion of the weak rock will cause strengths to be less than would be predicted for failure of the intact material alone.

10.1.2.6 Creep

Creep refers to the time-dependent deformation of soil or rock resulting from internal rearrangement of particles in response to the application of a sustained stress difference = $(\sigma_1 - \sigma_3)$, generally smaller than the stress difference of the soil at failure = $(\sigma_1 - \sigma_3)_f$, where σ_1 and σ_3 are the major and minor compressive principal stresses, respectively. Deformation during undrained creep results from shape distortion as the soil mobilizes a constant shearing resistance in response to the shear stresses applied upon loading or unloading.

Creep models have been applied toward the solution of a variety of engineering problems, such as the closure of and loads on tunnels, chambers, and pillars in creep-sensitive materials, such as salt, shale, and fault zones. Highly stressed and creep-sensitive ground encountered in underground excavations is described in tunnelman's terminology as squeezing ground, inasmuch as it leads to a gradual closing of the opening under practically undrained conditions.

Undrained creep behavior is closely related to the *drained creep* phenomenon associated with secondary consolidation and swelling inasmuch as the mechanisms that cause volume change in the drained case (at a constant effective stress) correspondingly lead to the development of excess pore-water pressures for undrained (constant volume) loadings.

A fundamental approach to creep behavior involving the response of soil to isotropic and deviatoric stresses is quite complex and still in the development stage. For this reason, creep predictions have been traditionally based on semi-empirical creep models obtained through attempts to best-fit the observed laboratory creep data of a variety of soils. In this way, Singh and Mitchell (1968, 1969) developed the interrelationship among creep strain rate $\dot{\epsilon} = d\epsilon/dt$, stress level with respect to a reference strength $D = (\sigma_1 - \sigma_3)/(\sigma_1 - \sigma_3)$ and time t, which is widely applicable to both drained and undrained creep, and expressed by

$$\dot{\epsilon} = \frac{d\epsilon}{dt} = A_1 \, e^{\alpha D} \left(\frac{t_1}{t}\right)^m \tag{10.1.11}$$

where t_1 is the reference time. The m parameter is the slope of the linear $\log \dot{\epsilon}$-$\log t$ relationship existing for any intermediate stress level D ($\approx$ 30 to 90%) for which m is also a constant. In other words, m establishes the rate of decrease of the strain rate with time. The a parameter gives the dependency of strain rate on stress level in the range of $D \approx$ 30 to 90% for a given time ($t_1 < t_2 < \ldots t_n$) for which a is also a constant. The parameter A_1 is the extrapolated value of strain rate at the reference time t_1 for $D = 0$. The creep potential of a clay is closely related to the value of m, which is in the range of 0.5 to 1.3 for a variety of soils. As m decreases, creep deformations become more significant, and the soil is more susceptible to creep rupture under sustained loads. For any one soil, there seems to be a maximum shear stress the soil can sustain without creep rupture, called the upper yield strength.

Expressing the strain-stress model as a hyperbola provides a better physical meaning for the creep parameters. The parameters in the hyperbolic model are the ratio of undrained modulus to undrained shear strength E_u/s_u and the axial strain at failure ϵ_f (Mesri et al, 1981).

$$\epsilon = \frac{2}{(E_u/s_u)_1} \frac{D_1}{1 - (R_f)_1 \, D_1} \left(\frac{t}{t_1}\right)^{\lambda} \tag{10.1.12}$$

where $R_f = 1 - \dfrac{2}{(E_u/s_u)\epsilon_f}$ and all the pertinent parameters are evaluated at an initial time t_1.

The creep parameter λ that controls time or strain rate effects also correlates very roughly with E_u/s_u: $\lambda \approx$ (l/6000) E_u/s_u. Typical values of E_u/s_u are in the range of 100 to 600 for clays and shales, and thus the values of λ are typically in the range of 0.01 to 0.10. (Note that $\lambda = 1 - m$).

Laboratory creep tests on samples of clay gouge can be used to determine the creep rates and pressures anticipated around a tunnel in squeezing ground, as discussed in 10.1.6.

10.1.2.7 Deterioration

Deterioration of overconsolidated clays and weak argillaceous rocks in response to weathering, swelling, and associated softening is perhaps one of their most important engineering characteristics. Unstable shales, for example, have plagued the petroleum industry for many years; the washout experienced by shale formations in response to borehole drilling can be tolerable in some wells but uncontrollable in others. Deterioration of shale mine roofs have been a major cause of the time-dependent roof collapses that have troubled the coal mining industry.

Table 10.1.4. Shale Slake-Durability Classification

Slake-Durability Index	Descriptive Term
98	Very high
95-98	High
85-95	Medium high
60-85	Medium
30-60	Low
30	Very low

Disintegration of shale into relatively small pieces upon exposure to different humidity-moisture environments, including submersion, is usually referred to as *slaking.* For tunnels and other underground excavations, the slaking characteristics of shale affect stand-up time, overbreak, support loads, and construction and support procedures required to minimize deterioration. Successive disintegration and breaking away of loose material from the tunnel crown section results in a fresh rock surface which is exposed each time to a further cycle of weathering. Slaking also progresses along surfaces of fractures behind rock slabs, contributing to time-dependent instability of large blocks.

Wetting and drying are also known to significantly increase the expansion of exposed clay and shale strata (and thus the heaving of overlying structures) as compared to the magnitude of swell and associated softening caused only by unloading with access to water. The additional heave and expansion occurs in response to the further breakdown of diagenetic bonds and alteration of the shale structure. The rate of disintegration of the upper shale layers also controls the rate at which shale slopes are "worked down." The colluvium of disintegrated shale (which could be tens of feet or meters thick) often forms an unstable mass that slides along its contact with the underlying intact shale. Thus the overall strength and stability of slopes is frequently controlled by the strength of the softened and deteriorated material.

During drying, shale is often subjected to inhomogeneous and anisotropic shrinkage that can cause cracks or microfissures to open, providing the conduits for future moisture redistribution. Upon access to water, shear stresses resulting from differential swelling may also cause local failures and shale disintegration. Moreover, pore air compression resulting from immersion of unsaturated shale in water can apply internal pressures that may eventually exceed the local tensile strength of the material and cause additional shale slaking.

The resistance of shale to deterioration upon exposure to drying and wetting cycles is frequently estimated by subjecting representative specimens to the standard *slake durability test* (Anon., 1978). In this test, 10 oven-dried lumps of shale 40 to 60 g each are rotated (20 rpm) for 10 min in a partly immersed drum of 2.0 mm mesh. The slake durability index is the oven-dried weight percentage retained in the drum after two (oven drying-wetting) standard cycles. Shale slake durability is characterized as described in Table 10.1.4.

The deterioration response of shales encompasses a broad range because of the broad range of shale compositions and the wide variety of field humidity and moisture environments to which they are subjected. As a result, index tests and classifications based on limited, often severe testing environments do not permit a full evaluation of the shale performance under field conditions, particularly in underground excavations in which moisture changes may be caused by relatively small humidity changes rather than alternate soaking and drying.

A study of the deterioration of two dozen natural shales exposed to controlled temperature and relative humidity conditions revealed the following (Cepeda-Díaz, 1987). In general, shales with natural water contents w_o smaller than the shrinkage limit w_s are fissile, whereas shales with natural water contents higher than w_s are massive. *Fissile* shales (group F) deteriorate by opening along weak planes of stratification, followed by separation of thin slabs perpendicular to the stratification. On the other hand, *massive* shales (group M) lack a preferred orientation of discontinuities and often break down to chunks and chips along random cracks, fractures, and slickensided surfaces. Fissile shales generally have natural water contents smaller than 15% and liquid limits w_ℓ not greater than 100% (using ball-mill disaggregated samples passing the US No. 200 standard sieve). Beyond this range of values for w_o and w_ℓ, shales are generally massive.

Fissile and massive shales can be further subdivided into three groups (1, 2, and 3) depending on the moisture change response and deterioration behavior upon exposure and thus related to their equilibrium relative humidity. *Equilibrium relative humidity ERH* is the *relative humidity RH* (at a given temperature) in which freshly exposed specimens of a given shale remain at their natural water content. At $RH > ERH$, intact shale gains moisture from the air environment, whereas at $RH < ERH$, intact shale loses water to the environment. Shales in group 1 deteriorate if subjected to relative humidities greater than *ERH* and include shales with $w_\ell > 40\%$ and $w_o < 0.2\,w_\ell - 8\%$ but with natural water contents not greater than 15%. On the other extreme, shales in group 3 (which have *ERH* greater than 97.5%) deteriorate when subjected to environments drier than *ERH* and include shales with $w_\ell > 10\%$ and $w_o > 0.2\,w_\ell - 2\%$. Shales in group 2 deteriorate in environments different than *ERH* and include shales with $w_\ell > 10\%$ and natural water contents other than those characteristics of groups 1 or 3.

Shales that deteriorate if subjected to *RH* greater than *ERH* (groups 1 and 2) generally have natural water contents lower than that corresponding to $w_o = 0.2w_\ell$. On the other hand, shales that deteriorate only if subjected to environments drier than *ERH* (group 3) generally have natural water contents higher than $w_o = 0.2\,w_\ell$. Thus it is consistent that shales with natural water contents equal to both $w_o = 0.2w_\ell = w_s$ remain intact upon exposure to any *RH* environment. At such moisture and structural state, the total volume change associated with moisture gain or loss upon environmental exposure may be relatively small, and thus the shale does not experience a significant amount of anisotropy and inhomogeneous deformation, that is, no deterioration.

For underground excavations such as tunnels and mines where field environmental control can be considered, the following guidelines are suggested for preventing the deterioration of various types of exposed shale strata (Cepeda-Díaz, 1987). Shales in group 1 should always be maintained at environments drier than *ERH* and preferably not wetter than 75% *RH*. Dry environments not only avoid the damaging effect of humidity fluctuations at the high *RH* range but also discourage the chemical alteration of iron sulfides, if present in the shale. Iron sulfides are known to oxidize in the presence of water and oxygen to form limonite and sulfuric acid. The sulfuric acid reacts with any calcite available in the rock to produce carbonic acid and gypsum with resulting large increase in volume and shale disintegration. Shales in group 2 should always be maintained at relative humidities near *ERH* and preferably not exceeding *ERH* ± 2%. For group 2 shales, *ERH* values are expected to be in the range of 75 to 97.5%. Shales in group 3 ($ERH > 97.5\%$) should always be maintained at environments wetter than *ERH*, and thus very often close to 100% *RH*.

Because shales in group 3 rarely deteriorate upon immersion in water at their natural water content, the use of sprayers to keep the exposed shale strata from drying may be a suitable alternative to the high humidity control of these shales. For shales in groups 1 and 2, any direct contact with water should be suppressed to avoid severe deterioration. It should be pointed out that, independent of the initial type of shale deterioration upon initial environmental exposure, wetting as a result of water inflow, immersion, or spraying (rain in the case of open cuts) is likely to induce severe deterioration if the exposed shale strata has been allowed to dry before wetting.

10.1.3. MECHANICS OF BEHAVIOR

In many problems, in both soil and rock mechanics, the stresses or the interrelation of stresses and deformations in the soil or rock mass must be determined. In these problems, the boundary conditions—the geometry of the mass and the distribution of loads or displacements applied to the mass—must be considered in obtaining a solution.

The problems have traditionally been divided into two groups (e.g., see Terzaghi, 1943): (1) those in which the stresses in a region or a surface in the mass has reached a limit or failure condition, and (2) those in which the stress levels are below the limit condition, and changes in stress within the medium are a function of the deformations.

1. *Limiting stress conditions.* The loads or the state of stress can be determined from the equilibrium of a portion of the mass, assuming that the stresses along a surface or region within the medium have reached their limit—the strength of the material. Slope stability analysis, earth pressure theory, and bearing capacity problems are all concerned with the evaluation of the loadings upon the soil or rock mass when certain surfaces within the mass are at a limiting level of stress.

2. *Stress-deformation conditions.* In many problems it is assumed that changes in stress are a function of deformations. The simplest assumption is that stresses are proportional to strain, in other words, the material is elastic. Closed elastic solutions are available for relatively simple boundary conditions, and often can be used to obtain a first approximation to the mechanics of the problem. More sophisticated stress-deformation models may assume that the material has a nonlinear stress-strain relationship, that may even approach a limit at some strain level, and that energy losses occur as loads are cycled. Solutions for these models, as well as for elastic models utilizing more complex boundary conditions, require the use of numerical methods, such as finite element or discrete element models, and relatively large computer capacity (usually a workstation computer rather than the current personal computers).

Stress-deformation relationships are used to evaluate not only foundation settlements but also the interaction between the soil medium and a structure—such as an anchored bulkhead, braced excavation, or tunnel lining—whose stresses and deformations are a function of its relative stiffness with respect to the soil mass.

Another group of stress-deformation problems are those in which settlements or heave occur with time as water flows into or out of the soil matrix in response to changes in the state of stress—the process of swelling or consolidation, respectively, discussed in 10.1.2.4. In soft soils, volume changes occurring as water flows into or out of the soil are the major cause of settlement, whereas deformation of the soil at constant water content is usually a minor portion of the total displacement.

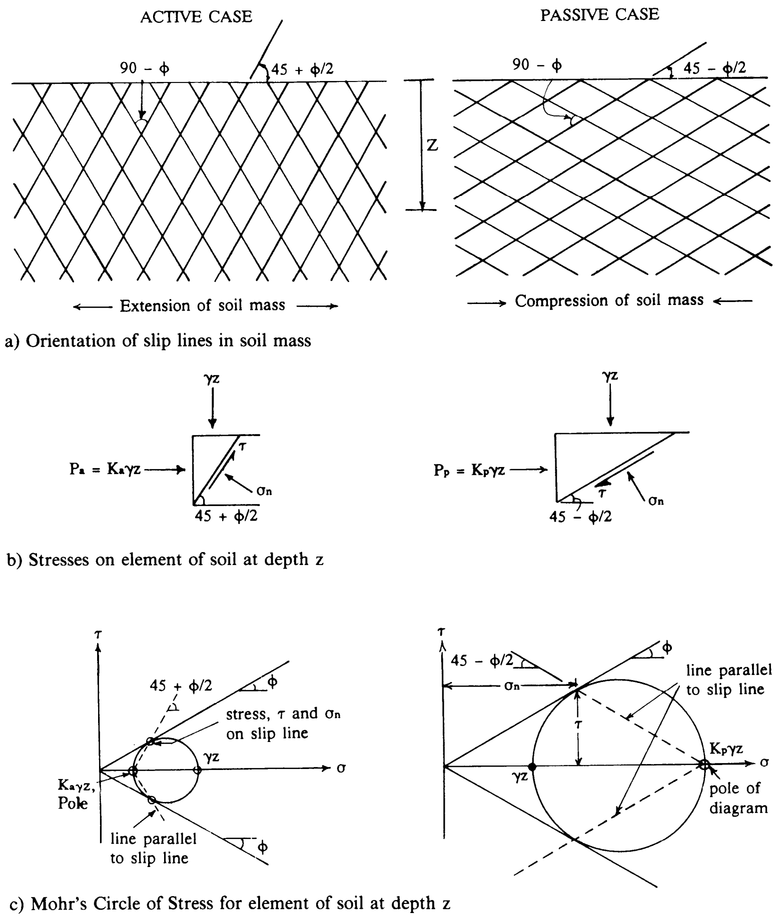


Fig. 10.1.5. Rankine active and passive earth pressures.

10.1.4. LIMITING EQUILIBRIUM

10.1.4.1 Active and Passive Earth Pressures

Lateral extension or compression of a soil mass will produce decreases or increases, respectively, in the lateral soil pressures until a limit is reached. The limits are termed the *active and passive earth pressures* and represent, respectively, the minimum and maximum possible natural lateral stresses that can exist in the ground. The active pressure P_a is used to represent the earth pressure acting behind retaining walls, while the passive pressure P_p is the maximum pressure that can act at the toe of a wall embedded in the soil.

Fig. 10.1.5 shows the active and passive pressures resulting from the extension and compression of an infinite soil mass with a horizontal ground surface.

The magnitude of the active and passive stresses acting on a vertical plane is obtained from Mohr's circle of stress, recognizing that the vertical stress is equal to the overburden pressure z, and the soil element is at its limit, and therefore the circle of stress is tangent to the strength envelope, which is assumed to be a straight line, $s = c + \sigma \tan \phi$. Rankine (1857) provided the first solution to this problem.

The principal stresses at the limit condition are related as follows:

$$\sigma_1 = \sigma_3 \tan^2 (45^\circ + \phi/2) + 2\, c \tan (45^\circ + \phi/2) \quad (10.1.13)$$

For the case of lateral extension of the soil mass, the lateral stresses are reduced to their minimum value, and s_3 will equal the lateral stress, which is the active earth pressure. The coefficient of active pressure K_a is equal to $1/\tan^2 (45 + \phi/2)$ for the case of a horizontal ground surface. For the case of lateral compression of the soil, the lateral stresses are increased to their maximum value, and σ_1 will equal the lateral stress, which is the passive earth pressure. The coefficient of passive pressure K_P is equal to $\tan^2 (45 + \phi/2)$ for the case of a horizontal ground surface.

The more general Rankine solution is for the active and passive pressures that develop in an infinite slope (Fig. 10.1.6). Assuming that the slope is infinite, the vertical force on a plane parallel to the ground surface is equal to the weight of the overburden. The active and passive earth pressures are the stresses acting on a vertical plane. They are oriented parallel to the ground surface and thus have both a normal and shear component. Fig. 10.1.6 illustrates the condition for the active earth pressure. The magnitude and direction of the active earth pressure and the orientation of the shear planes can be obtained from Mohr's circle.

If the entire soil mass is in a limiting state of equilibrium, as is assumed in the Rankine case, then the active and passive stresses acting on the vertical plane for a frictional material will increase linearly with depth, and the total active or passive force over a given height z will act at the lower third point and have a magnitude equal to $z/2$ times the active or passive pressure at the depth z. This condition is obtained for a retaining wall when

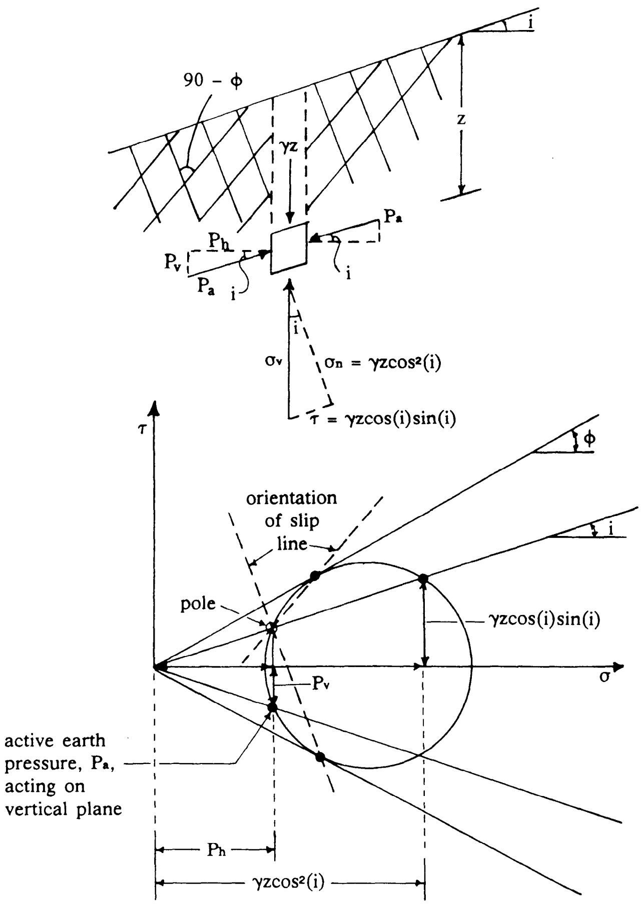


Fig. 10.1.6. Rankine active earth pressure for inclined surface.

outward deflections at the top of the wall are in the order of 0.1 to 0.5% of its height. Thus the validity of designs based on an assumed active state of stress behind the wall depends on the outward deflections that the retaining structure actually undergo.

For walls in which the upper portion of the wall is not allowed to deflect outward, the distribution of stresses on the wall will not increase linearly with depth, even though the movements of the lower portions of the wall may be sufficient to reduce the total forces on the wall to the active forces. This is the case with braced walls in which the upper portion of the wall is restrained early in the excavation process, usually in order to limit ground movements that could adversely affect structures and utilities adjacent to the wall. As a result, the center of pressure for the forces acting on the wall is at a higher elevation than the lower third point, although the total magnitude of the forces can be estimated using Rankine or Coulomb assumptions. Field measurement of strut loads for braced excavations are used to develop apparent earth pressure envelopes for design. The envelopes, which encompass the variations that can be expected in strut loads due to construction procedures, provide an almost rectangular (constant) pressure distribution with depth and a magnitude of the total force under the envelope that is approximately 1.3 to 1.75 times the active Rankine or Coulomb pressure (Peck et al., 1974).

The Rankine case, in which it is assumed that the ground surface is of infinite extent, either horizontal or sloping, can be applied to real problems in which the ground surface is not of infinite extent, as long as the stresses on the vertical plane can be assumed to act parallel to the ground surface. It usually produces a conservative estimate of either the active or passive pressures on a wall because it usually does not fully account for the shear stresses that act on the wall.

The magnitude and location of the total active or passive forces acting on the back of a retaining structure can also be determined using the assumptions involved in Coulomb wedge analysis, as described in the next section. The Coulomb analysis is more general than the Rankine method in that any orientation of the forces on the back of the retaining wall can be assumed, and surcharges and irregular backfill slopes can be handled.

10.1.4.2 Coulomb Wedge Analysis

A more general solution for limiting conditions in a soil mass can be obtained from the Coulomb (1776) analysis, which evaluates the forces acting on a wedge of soil. In this analysis, it is assumed that a planar failure surface has developed in the soil mass and that the soil wedge slides along a planar surface forming the back of the wall. The analysis is commonly used to evaluate active earth pressures behind retaining walls.

Fig. 10.1.7 illustrates the procedure for a frictional material. As in most limit equilibrium problems, the analysis is handled in a series of steps:

1. A free-body diagram of the wedge of soil is drawn, showing all forces acting on the boundaries of the wedge, as well as the body forces (weight W_s and acceleration forces) acting in the free body. A failure plane forms one of the wedge boundaries. A trial failure plane is selected, oriented at angle a to the horizontal.

2. A force polygon is constructed to include all the forces acting on the free-body diagram. The summation of forces can be obtained either graphically or by trigonometry, considering the geometry of the force polygon. Water pressures U_1 and U_2 acting on the boundaries of the wedge are included as external forces. Although the summation of forces equals zero, the assumption of a plane failure surface may result in the summation of moments not equaling zero.

3. Because the orientation of the failure surface is usually not known, several trial wedges are analyzed, each with a planar failure surface at a different orientation a. The largest value of the active force or the smallest value of the passive force represents the critical condition, in which the first planar failure surface would develop.

In the case of soil or rock containing preexisting planes of weakness of known orientation, the failure plane is selected parallel to the plane of weakness, for cases in which its orientation is close to the orientation of the critical failure plane. In this case, it is not necessary to search for the orientation of the critical failure plane by analyzing multiple trial wedges.

The Coulomb method described in the foregoing can be used for general retaining wall problems, but there are some limitations. For the passive pressure case, when there is a significant component of wall friction, a planar failure surface does not closely approximate the actual failure surface that occurs in the field, and thus the Coulomb method is unconservative (overestimation of the passive pressure). In such cases, the critical plane will be curved and the passive pressure can be analyzed by methods described in the following.

10.1.4.3 Application of Wedge Analysis to Slope Stability Problems

Fig. 10.1.8 illustrates the procedure for determining the stability of a slope by analyzing multiple wedges or slices of the

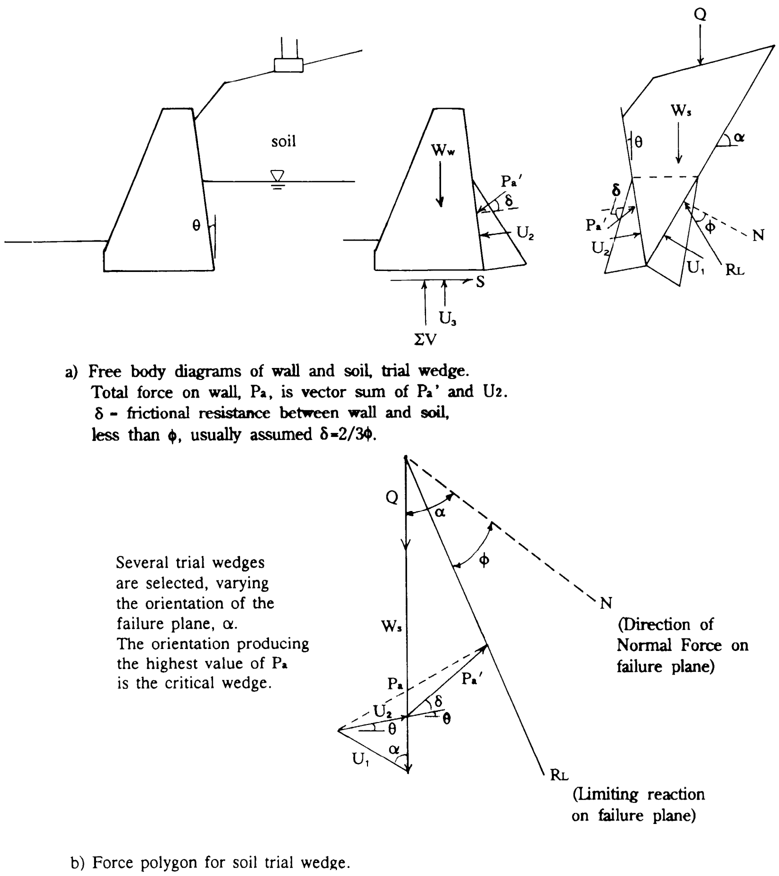


Fig. 10.1.7. Coulomb active earth pressure.

potential sliding mass. Multiple slices are treated by summing forces in the same manner as the Coulomb wedge in 10.1.4.2. A slice is related to the adjacent slice by equating the force acting between the two slices. There are several different techniques for evaluating the forces between slices. In general, forces acting between vertical slices are assumed to act parallel to the slope, as would be the case for an infinite slope. Where one slice or wedge has a significant vertical displacement with respect to an adjacent wedge or slice, the forces acting between the two may approach their maximum obliquity, at ϕ degrees to the normal. The mobilized or developed friction angle ϕ_D (required to produce a factor of safety of one) is determined from trials using the graphical construction for the composite force polygon of Fig. 10.1.8c. The factor of safety *FS* is the ratio of tan ϕ to tan ϕ_D.

In most slope stability analyses, the slope is assumed to be two-dimensional. In actual cases in which the slope does not extend a large distance in the third dimension, the actual factor of safety will be higher than would be computed for a two-dimensional slope having the same profile. The three-dimensional analysis of rock wedges (Chapter 10.4) shows a similar trend. Although the two-dimensional assumption would provide a conservative estimate of the factor of safety of a slope, making the assumption that a slope is two-dimensional in the back analysis of shear strength parameters would overestimate the shear strength if the slope is actually three-dimensional.

10.1.4.4 Circular Failure Surface for Stability of Cohesive Material

The maximum height of a slope or a cut in a cohesive material, in which the undrained shearing resistance s_u is assumed to apply, can be determined from the stability chart of Fig. 10.1.9 (Peck et al., 1974). The factor of safety *FS* of the slope is found from the expression,

$$FS = N_\sigma s_u/\gamma H \tag{10.1.14}$$

Use of the undrained shear strength s_u will be unconservative if the slope is excavated in overconsolidated or fissured clays, or clays containing sand and silt seams. In such cases, effective shear strength parameters should be used, which assume the material has frictional resistance and, in some cases, a small effective cohesion.

10.1.4.5 Curved Failure Surfaces for Passive Earth Pressures

The Coulomb analysis produces unconservative results for the passive case when the passive force on the wall acts at a significant angle δ to the normal. In this case, a curved failure surface should be assumed in order to determine the passive earth pressure. The failure zone will consist of a zone of radial shear and a passive Rankine zone (Fig. 10.1.10). For a frictional material, the shape of the failure surface in the zone of radial shear approximates a logarithmic spiral. Additionally, the log spiral simplifies the calculations because the limiting reactions acting on the log spiral failure surface will meet at a point and can be ignored in the analysis by summing moments about that point. Fig. 10.1.10 illustrates the pattern of slip lines for the case of the angle δ on the wall at its maximum value, equal to the angle of friction. In this case, the plane of the wall becomes a slip line for the soil mass.

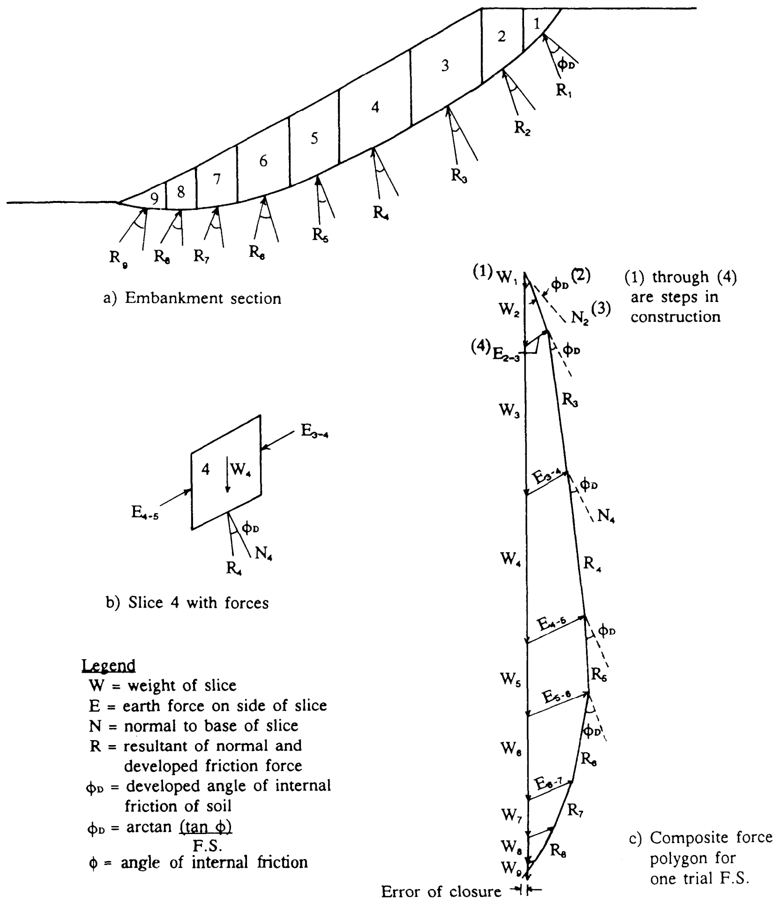


Fig. 10.1.8. Method of slices for analyzing slope stability.

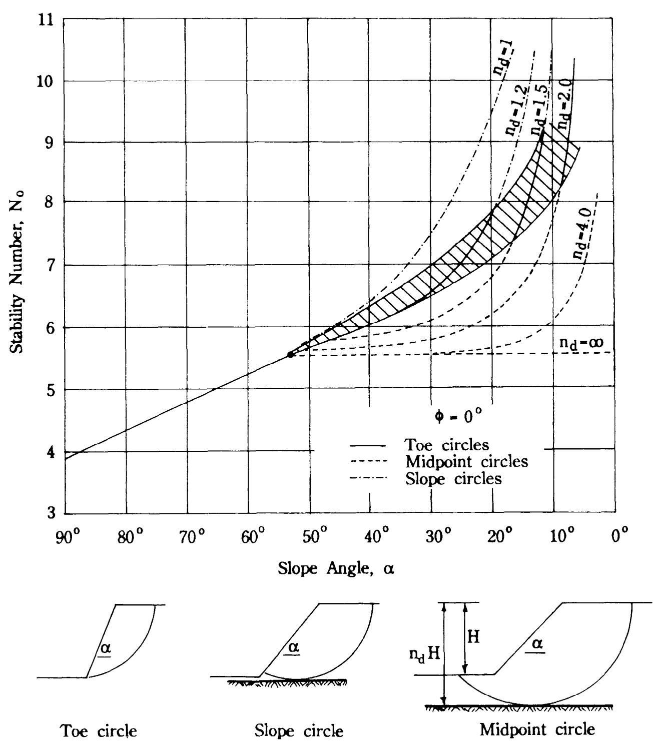


Fig. 10.1.9. Stability chart for slopes in cohesive soil (after Taylor, 1937).

10.1.4.6 General Bearing-capacity Relationship

Relationships for the bearing capacity of soils loaded by foundations have been developed from the theory of plasticity. The solutions of Prandtl (1921) and Reisner (1924) were obtained for a rough, rigid strip footing on the surface of a weightless, perfectly plastic material (Fig. 10.1.11). Note that the pattern of the failure surfaces is similar to the combined active and passive zones illustrated in Fig. 10.1.10. The ultimate bearing capacity q_{ult} is related to the undrained shear strength of the soil c and the surcharge pressure q as follows:

$$q_{ult} = cN_c + qN_q \tag{10.1.15}$$

where N_c and N_q are dimensionless bearing-capacity factors that are a function of the friction angle ϕ:

$$N_q = \exp(\pi \tan \phi) \text{ and } \tan^2(45° + \phi/2) \tag{10.1.16}$$

$$N_c = (N_q - 1) \cot \phi \tag{10.1.17}$$

The general bearing capacity relationship (Terzaghi, 1943) is an approximate solution that was obtained by superposing on Eq. 10.1.15 an additional relationship for a frictional soil with weight, but without cohesion intercept (c = 0) and without a surcharge (q = 0). The superposition produces an error on the safe side not exceeding 20% for friction angles of 30 to 40°, and, of course, no error for a friction angle of zero (Vesic, 1975). The additional term is proportional to the shear capacity developed from the normal stresses in the weighted soil mass below the footing, and is therefore proportional to the size of the mass B and is also a function of the friction angle ϕ.

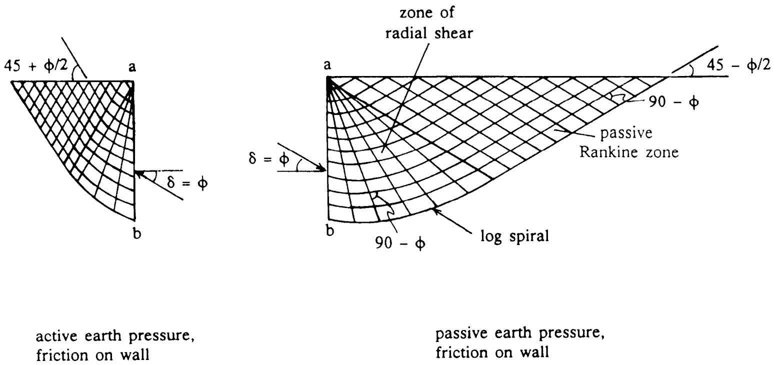


Fig.10.1.10. Active and passive earth pressures, friction on wall.

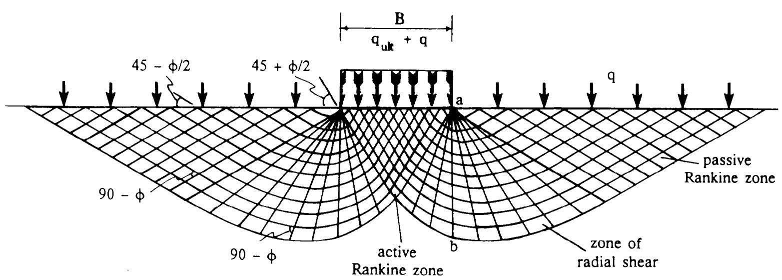


Fig. 10.1.11. Bearing capacity of a shallow footing; no friction on footing base.

The resulting general bearing-capacity relationship for a long strip footing of width B is

$$q_{\text{ult}} = cN_c + qN_q + 1/2B\gamma N_\gamma \qquad (10.1.18)$$

where N_g a dimensionless bearing capacity factor that is a function of ϕ and must be evaluated numerically. It can be approximated by the analytical expression,

$$N_\gamma = 2(N_q + 1)\tan\phi \qquad (10.1.19)$$

with an error on the safe side of 5 to 10% (Vesic, 1975).

Vesic (1975) notes, contrary to some of the earlier discussions in the literature, that the shape of the failure zone and the magnitude of the ultimate bearing capacity are not strongly affected by the presence or lack of friction at the base of the footing. He further notes that the ultimate bearing capacity is not strongly influenced by the stress distribution at the base of the footing, which could range from a uniform pressure for a flexible foundation to the variable stress distribution created by a rigid footing (or punch) displacing a constant amount into the soil mass. However, if the base of the footing rests on a thin soft layer above a stronger layer (such as the case of the bearing capacity of underclays beneath a mine pillar), significant reductions in bearing capacity can result by changing from a rough to smooth footing or by changing from a rigid punch to a uniform pressure acting at the base of the footing.

For foundations greater than approximately 4 ft (1.2 m) wide in sands and other frictional materials, the bearing capacity factors are quite high so that bearing capacity failures develop only for very high foundation loads. In these cases, the capacity of the foundations are controlled by allowable settlements.

Bearing Capacity of Cohesive Soils: Adjustments to the general bearing-capacity relationship have been made for other footing shapes and for the extension of the failure planes into the soil surcharge above the footing base. A linear approximation of the bearing capacity relationship for cohesive, frictionless materials (Skempton, 1951) is

$$q_{\text{ult}} = (1 + 0.2D_f/B)\,(1 + 0.2B/L)s_u N_c \qquad (10.1.20)$$

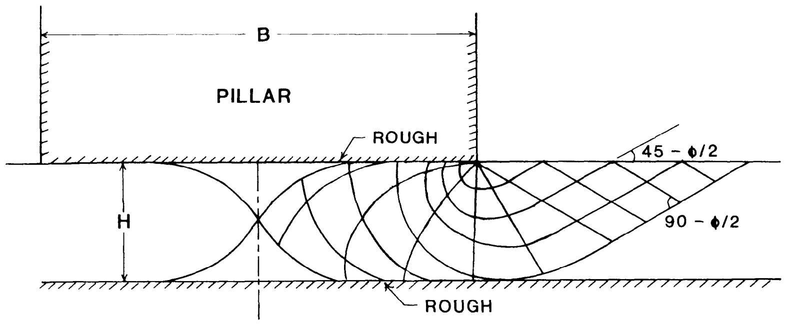
where N_c = 5.14; B is the footing width; D_f is the depth of the surcharge above the footing base, and is set equal to $2B$ for values of D_f greater than $2B$; and B/L is the ratio of width to length of the footing. The ultimate bearing capacity q_{ult}, is the pressure in excess of the pressure applied by the surrounding soil surcharge $g\ D_f$.

The foregoing relationship is used to estimate the allowable bearing capacity q_a for foundations on clay. The allowable bearing capacity is usually selected with a factor of safety of three against the ultimate bearing capacity q_{ult}. In addition, the foundation can accept a pressure equal to the surrounding soil surcharge pressure $g\ D_f$:

$$q_a = q_{\text{ult}}/3 + \gamma D_f \qquad (10.1.21)$$

10.1.4.7 Bearing Capacity of Mine Pillars on Underclays

In many coal mines, underclays are present beneath the coal pillar and the floor of the mine. Stresses in the pillars are often high enough to cause a bearing-capacity failure. Several types of bearing failure can occur. The pillar can punch into the underclay, or a general bearing-capacity failure can develop, with large heave occurring in the floor adjacent to the pillar. Typically, the underclays are in layers having a small thickness with respect to the pillar width. In such a case, the slip surfaces illustrated for the general bearing capacity relationship cannot form and the underclay will have to be squeezed out between the base of the pillar and underlying stronger rock layers, as illustrated in Fig. 10.1.12.



For φ = 0, Both Surfaces Rough:

B/H	0	1.4	5	10	15	20
Nc	5.14	5.14	6.4	9.2	11.8	13.4

$q_{ult} = S_u N_c$

Fig. 10.1.12. Squeeze of clay from thin zone beneath pillar (after Mandel and Salençon, 1972).

As a result, the magnitude of the ultimate bearing capacity is greater than would be obtained for a deep deposit of the same material. Vesic (1975) and Mandel and Salençon (1972) show that, as the ratio of the width of the pillar to the thickness of the soft layer increases, the ultimate bearing capacity for a cohesive, frictionless soil increases from 1 to approximately 2.5 times the ultimate bearing capacity that would be obtained for a deep deposit of the same soil.

Experience with coal pillars overlying underclays shows an interesting interaction between the pillar and the underclay. As the underclay is squeezed laterally outward from beneath the pillar, the pillar is subjected to tensile stresses, causing vertical fractures to develop in the pillar. Not only does closure of the opening result from the squeezing of the underclays, but the process can lead to collapse of the pillar and further closure of the mine opening. The formation of vertical tensile fractures in the pillar is a demonstration of the inability of the pillar to sustain the shear stresses that tend to develop between the pillar and the underclay as the clay squeezes outward. Mandel and Salençon show that the increase in bearing capacity factor N_c is much less significant if there is no shear resistance on one of the layers bounding the thin failing layer.

The shear stresses that develop at the base of the pillar have a much greater effect on the bearing capacity for a thin soil layer than they do for a deep layer. The failure of a footing above a deep layer is by a general downward movement of the soil beneath the footing whereas in the failure of a pillar overlying a thin weak layer, the soil is squeezed out laterally, between the pillar above and the stronger layer below. The presence of vertical fractures in a pillar effectively breaks the pillar into a group of narrow pillars, causing the effective ratio of pillar width to layer depth to decrease, and can lead to a reduction in the shear stresses along the pillar-underclay contact and a decrease in the bearing capacity.

Time effects also occur in the failure of coal pillars overlying underclays. Squeezing of the underclays will occur with time as a result of creep of the clay under high stress and the loss of strength of the underclay beneath the floor due to swelling in response to excavation unloading. Pillar settlement and floor heave may begin during active mining, but often the time-dependent effects cause continued failures, floor heave, and large settlements long after excavation has been completed. Large and relatively abrupt subsidence has taken place over abandoned room and pillar workings many years after the facilities were abandoned.

10.1.5. SETTLEMENT OF FOUNDATIONS

10.1.5.1 Soft Clays

Upon application of a load from a footing or fill placed upon a layer of soft clay, most of the settlement will take place with time, as water is squeezed from the soil pores and the volume of the voids decreases. The settlement of a foundation on a soft clay can be estimated by determining the change in effective stress on the clay layer as well as the effective stress existing before loading, and then determining the change in void ratio by using an appropriate slope on a plot of void ratio vs. change in effective pressure (e vs. log σ'_v). The slope to be used can be estimated from laboratory tests or other field tests. For a material loaded above its maximum previous consolidation pressure or preconsolidation pressure σ'_p, steeper slopes corresponding to C_c will apply. For stress changes below σ'_p, flatter slopes corresponding to C_r will apply (see 10.1.2.4).

The stresses resulting from a footing load will tend to spread out with depth below the footing, and thus the magnitude of the imposed stress will be reduced. In order to estimate the stresses at depth, an elastic distribution of stresses is usually assumed. Newmark charts, available in most soil mechanics texts, provide a means of graphically determining the stresses at any depth below the footing. An approximate estimate can be also obtained by assuming that the stresses spread on a $2V{:}1H$ slope from the edge of the footing.

10.1.5.2 Sands

The *standard penetration test* and its corresponding N-values are generally used to predict the allowable contact pressure to which a granular soil can be subjected by a foundation in order not to exceed tolerable settlements. Compression of saturated coarse-grained soils occurs rapidly as a result of their high permeability.

For footings with widths B greater than about 4 ft (1.2 m), the allowable pressure (q_a) expected to produce a total settlement smaller than 1 in. (25.4 mm) and corresponding differential settlements not exceeding 1/2 in. (12.7 mm) can be approximated by

$$q_a = 0.11\ N \qquad (10.1.22)$$

where q_a is in tons/ft^2, and N is the smallest average value of the standard penetration resistance. The value of N is located within a depth interval equal to B below the foundation level after corrections are made, if necessary, to take into account the effect of the overburden pressure and the position of the water table (Peck et al., 1974).

The ratio between the differential settlement and the total settlement is significantly less for very wide footings or raft foundations than for regular size footings, and thus the allowable pressure for rafts is that corresponding to a total settlement of 2 in. (50.8 mm), which can be approximated by $q_a = 0.22\ N$. For very narrow footings ($B < 4$ ft or 1.2 m), a small increase in pressure may lead to a significant enough settlement to consider it a bearing capacity failure. Therefore, the margin against bearing-capacity failure for these footings should also be investigated as explained in 10.1.4.6.

10.1.5.3 Weak Rock and Stiff Soils

The compressibility of foundations in weak rock and stiff soils can be evaluated using the theory of elasticity, assuming an appropriate deformation modulus for the material. The Boussinesq equation gives the displacements and stresses in an elastic mass due to a point load applied at the ground surface. The equation is integrated to determine the displacements and stresses due to loads distributed over some area at the ground surface. Newmark developed a graphical method for determining stresses or displacements for any assumed distribution of pressures at the ground surface. For the simple case of a uniform pressure p acting over a circular area of radius a, on the surface of an elastic medium, the surface displacement d at the center of the circle can be estimated as

$$d = 2pa(1 - \mu^2)/E \qquad (10.1.23)$$

where E and m are, respectively, in situ deformation modulus and Poissons ratio.

For rock containing natural fractures, the stiffness obtained in a laboratory test on an intact sample should be reduced in order to obtain the in situ deformation modulus. Parameters that index the degree of fracturing of the rock mass, such as the rock quality designation (RQD) or the ratio of field seismic velocity to laboratory sonic velocity, can be used to estimate the amount that the field deformation modulus should be reduced from the laboratory modulus. Plate bearing tests (typically with plate diameters of 3 ft or 1 m) can be used to determine the in situ modulus at selected locations in the field.

10.1.6. TUNNELING

10.1.6.1 Tunneling Conditions

The following should be considered in constructing a tunnel in soil or weak rock: (1) ability to safely excavate and support the tunnel; (2) ability to limit damage to third parties, in particular, to control ground movements that could affect nearby buildings and utilities; and (3) ability to perform its intended function over the life of the project, in particular, an adequate lining (Peck, 1969).

Commonly, long circular tunnels in soil are constructed using circular shields that are shoved through the ground, excavating the soil at the face of the shield, and providing initial support to the perimeter of the tunnel heading and permitting the initial tunnel lining to be erected in the tail section of the shield without exposing the soil surface.

Stability of the tunnel heading and prevention of damaging ground settlements are major concerns in soft ground tunneling. Design of the tunneling process is more difficult in cases in which the ground conditions vary across the tunnel face or vary along the length of the tunnel alignment. A single shield may not be able to accommodate all the conditions efficiently, and it may be necessary to change tunneling procedures or to use ground modification techniques, such as dewatering or grouting, to advance the tunnel.

Mixed-face conditions, in which rock and soil are encountered in a single tunnel face, create difficulties in excavating the rock while supporting the soils, which are often unstable. Bedding and layering of soils create variations in permeability that allow water to perch on top of low permeability layers and to flow into the tunnel. To minimize such flows requires placement of dewatering wells on very close spacings adjacent to the tunnel.

Soil tunneling procedures must also be employed when tunneling in rock, when a mixed face of soil and rock is encountered at the contact between rock and the overlying soil overburden, or when fault zones are encountered in the rock. Commonly, water in the soil and permeable rock fractures contributes to the instability of these zones. Although rapid and efficient tunneling may not be possible in the soil, it is desirable that the methods used to get through the soil zones allow safe, steady progress. This is more likely if it is anticipated in the design of the tunneling system that such conditions may be encountered.

The initial tunnel lining must be designed to accommodate the loads applied during construction, as well as the stresses produced by ground loads. In shielded tunnels, initial support commonly consists of steel ribs and lagging, steel liner plate segments, or concrete segments. Significant loads can develop during erection and subsequent grouting or expansion of the segments against the ground. If the ground has been excessively loosened or if it forms an irregular perimeter in poor contact with the lining, then nonuniform loadings and bending are produced that may result in significant damage and even buckling of the lining. As the shield advances, ground loads are transferred to the installed lining. Elastic analyses and experience show that there is significant interaction between the ground and the lining. If the lining is in full contact with the ground around its perimeter, then active ground loads applied to one portion of the lining perimeter will cause a small outward deflection and the build-up of a passive ground reaction on other portions of the perimeter that will support the lining and permit it to act as a stable arch. If the lining is relatively flexible with respect to the soil mass, then the bending moments that develop in the lining will be quite small.

10.1.6.2 Tunnelman's Ground Classification

Because the tunneling process may temporarily expose the ground over head or on vertical surfaces, soil characteristics that are of great concern to the tunneler are the ability of the soil to stand until the support can be placed. One of the major factors controlling the stand-up time of the soil is its short-term cohesion or cementing, as well as the amount of soil surface that is exposed ahead of the supports at one time. It is often difficult to determine the anticipated tunnel behavior from the standard index properties obtained in the soil exploration program. Often, in sampling, cohesive granular soils or partially cemented soils will be disturbed so that their in situ cohesive characteristics are lost. Another factor that strongly influences stand-up time is the groundwater flow. Flow occurring into the tunnel creates high seepage-pressure gradients near the tunnel face that can result in instability of the soil, causing it to flow into the tunnel.

The Tunnelman's Ground Classification is a qualitative description of the types of soil behavior encountered during tunnel-

Table 10.1.5. Tunnelman's Ground Classification

Classification		Behavior	Typical Soil Types
Firm		Heading can be advanced without initial support, and final lining can be constructed before ground starts to move.	Loess above water table; hard clay, marl, cemented sand and gravel when not highly overstressed.
Raveling	Slow raveling Fast raveling	Chunks or flakes of material begin to drop out of the arch or walls sometime after the ground has been exposed, due to loosening or to overstress and "brittle" fracture (ground separates or breaks along distinct surfaces, opposed to squeezing ground). In fast raveling ground, the process starts within a few minutes.	Residual soils or sand with small amounts of binder may be fast raveling below the water table, slow raveling above. Stiff fissured clays may be slow or fast raveling depending upon degree of overstress.
Squeezing		Ground squeezes or extrudes plastically into tunnel, without visible fracturing or loss of continuity, and without perceptible increase in water content. Ductile, plastic yield and flow due to overstress.	Ground with low frictional strength. Rate of squeeze depends on degree of overstress. Occurs at shallow to medium depth in clay of very soft to medium consistency. Stiff to hard clay under high cover may move in combination of raveling at excavation surface and squeezing at depth.
Running	Cohesive, running Running	Granular materials without cohesion are unstable at a slope greater than their angle of repose (± 30-35°). When exposed at steeper slopes, they run like granulated sugar or dune sand until the slope flattens to the angle of repose.	Clean, dry granular materials. Apparent cohesion in moist sand, or weak cementation in any granular soil, may allow the material to stand for a brief period of raveling before it breaks down and runs; termed cohesive running.
Flowing		A mixture of soil and water flows into the tunnel like a viscous fluid. The material can enter the tunnel from the invert as well as from the face, crown, and walls, and can flow for great distances, completely filling the tunnel in some cases.	Below the water table in silt, sand, or gravel without enough clay content to give significant cohesion and plasticity. May also occur in highly sensitive clay when such material is disturbed.
Swelling		Ground absorbs water, increases in volume, and expands slowly into the tunnel.	Highly preconsolidated clay with plasticity index 30, generally containing significant montmorillonite.

ing. Table 10.1.5 presents the classification, as proposed by Terzaghi (1950) and modified by Peck (1969) and Heuer (1974).

The classification is more than a description of the properties of the soil because it describes behavior under certain of the conditions encountered in a tunnel. Construction methods and equipment and tunnel geometry influence the ground behavior.

The stand-up characteristics of the soil are the primary factors that define the categories. The initial cohesion or cementing of the soil differentiates between *firm, raveling,* or *running* ground, whereas the presence of groundwater seepage can transform running ground into *flowing* ground. *Swelling* and *squeezing* ground are most likely in shales or clays of high plasticity. Protection of the shale from disturbance and access to water will minimize the swelling pressures that can develop. Thus early placement of a continuous lining against potentially swelling surfaces is usually desirable. If pressures still are high, then procedures to allow the ground to displace in a controlled manner can be devised.

10.1.6.3 Squeezing Ground

The severity of squeezing ground in a tunnel can be estimated from the ratio of overburden pressure gH to the undrained shear strength s_u of the clay (Peck, 1969). Ratios greater than 5 will produce the equivalent of a bearing capacity failure of the clay surrounding the tunnel, and result in rapid squeezing into unsupported areas in the tunnel face or into voids around the shield and lining. Compressed air pressures used in the tunnel are subtracted from the overburden pressure to reduce the ratio to acceptable levels. Although squeezing conditions are not as severe for ratios less than approximately 4, some time dependent movement and loading of supports will take place for ratios between 1 and 4.

Squeezing relationships cannot only be evaluated for soft clays in shallow tunnels but also for deep rock tunnels in shales and fault zones. In mine pillars or abutments where mining nearby has resulted in transfer of large stresses to the pillars, the stresses that cause failure or time-dependent movements will be greater than overburden and will be a function of the total stresses transferred to the pillar. The time rate of closure and build-up of pressures on linings in squeezing ground can be evaluated using creep relations, such as given in Eq. 10.1.12. Using this equation to describe the stress-strain-time relationships, the creep closure around a circular tunnel instantly excavated in a hydrostatic stress field was analyzed and compared with the results of field measurements of the Stillwater Tunnel in Utah, excavated at a depth of 2000 ft (600 m) in a shale containing fault zones and closely fractured zones subject to squeezing (Phienweja, 1987). Creep parameters were not only estimated from laboratory creep tests on shale and sheared clay gouge but were also estimated from measured closure and lining pressures in various rock mass classes. Table 10.1.6 illustrates the creep parameters and typical values of pressure and deforma-

Table 10.1.6. Back Calculated and Estimated Values of Strength and Creep Parameters for Various Classes of Shale

Ground Class	Description	ϕ, Degrees	q_u, psi	λ, %	$\dfrac{E_s}{(\sigma_1 - \sigma_3)_f}$	R_f	Diametral Closure, % at 3 months
CL I	Sandstone and siltstone, widely spaced joints	50	9000	—	200–500	0.1	
CL II	Siltstone to shale, moderately jointed	45	2000–4000	1–2	250–300	0.1–0.2	0.2 to 0.4 (steel ribs)
CL III	Shale, closely jointed with some thin shears	35	500–1000	5–7	250–300	0.5	1 to 2 (steel ribs)
CL IV	Shale, closely jointed and sheared, shear zones contain crushed and soft materials	25	200–300	7–10	200–250	0.8–0.9	0.8% (concrete segments)
CL IVb	Shale, wide shear zones of crushed and soft materials	25	0–100	12–15	150–200	0.9	4% (steel ribs)

Note: Steel ribs developed yield stresses (40 psi radial pressure) in Class III-IVb ground. Concrete segments were effectively stiffer and developed approximately 100 psi radial pressure. Conversion factor: 1 psi = 6.895 kPa.

tion that developed for the ground classes. The slope of the (log of tunnel closure) – (log time) plot was initially greater than λ as the headings advanced but approached λ with time. Pressures that develop on a lining installed soon after excavation are also strongly influenced by λ.

10.1.6.4 Estimating Ground Movements Around Shallow Tunnels.

In any tunnel, the ground in the heading must be kept under control to prevent instability prior to and during installation of the tunnel lining. When advancing shallow tunnels in soil in the vicinity of structures and utilities, a further requirement is to limit ground movements so that unacceptable movements and damage do not occur. Ground control measures must be selected with due consideration to the stand-up time of the soil. Two types of ground loss can occur: (1) large loss of ground, often in a sudden, uncontrolled, or catastrophic manner; and (2) the regular, smaller ground movements that occur throughout the length of the tunnel drive.

Large ground losses usually occur by means of running, flowing, or squeezing of soil into the tunnel face but can also occur due to flow of soil through the tunnel lining or by collapse of an unstable lining. Tunneling procedures should be selected to minimize the risk of large ground losses for the anticipated ground conditions.

Regular ground losses occur around the shield and around the lining, during excavation and installation of the lining, throughout the tunnel drive. For a shielded tunnel, a large portion of regular ground losses results when the shield cuts a perimeter that is several inches (tens of millimeters) outside the outer perimeter of the installed support. Ground losses are minimized by design of a shield that has minimum overcutting and is easily steered on line and grade, and by rapid and complete expansion or grouting of the lining behind the tail of the shield.

The volume of ground loss into the tunnel is distributed through the soil mass and results in a surface settlement trough of volume V_s that is less than the volume of ground loss V_1 by the amount of the volume expansion ΔV that takes place in the ground (Fig. 10.1.13). The denser the soil and the larger the ratio of tunnel-depth to diameter, the greater is the volume expansion, which reduces the volume of the surface settlement trough for a given volume of ground loss around the tunnel. In soft clays, V_s is approximately equal to V_l, but will exceed V_1 if drainage or disturbance of the clay surrounding the tunnel results in consolidation with time.

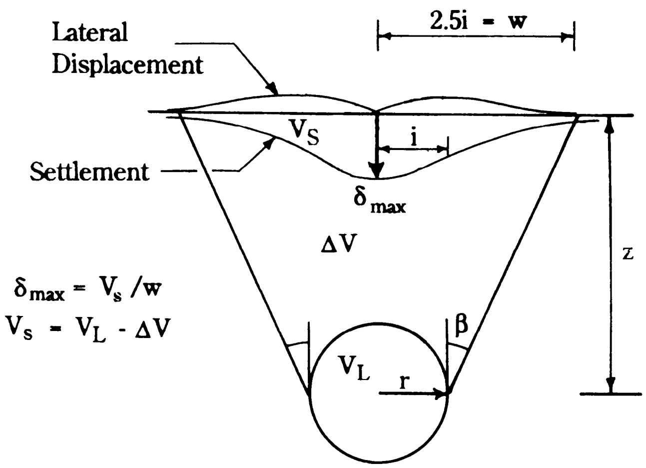


Fig. 10.1.13. Settlement trough for tunnel in soil.

The half width of the settlement trough *w* can be estimated using procedures described by Peck (1969) and Cording and Hansmire (1975). For a settlement trough shaped like a normal distribution curve with an inflection point *i*, then *w* is equal to 2.5 *i*. For sands, the angle *b* is in the range of 10 to 30°. For clays, the angle *b* is typically in the range of 25 to 45°. With these relationships, the magnitude and distribution of surface settlements and the damage to structures can be estimated for given volumes of ground loss.

10.1.6.5 Controlling Ground Movements.

A variety of tunneling procedures and shield designs can be used to control the face as well as limit movements around the shield and lining. Open face shields or shields using mechanical diggers support the face with the muck that is excavated and by additional breasting boards or plates positioned against the face. Wheeled excavators must control the face by only rotating the wheel sufficiently to excavate a volume of soil equivalent to the volume of the tunnel that is advanced. In both of these techniques, additional measures, such as compressed air or dewatering, may be used to prevent inflow of soil, and it may also be necessary to prevent running by use of techniques such as chemical grouting of the soils ahead of the face. Pressurized face shields may be used in lieu of compressed air to stabilize the face. They use air, water, a slurry, or the excavated muck to provide positive pressures in a chamber at the face of the shield to balance the water and earth pressures in the soil ahead of the face. Even with this technique, additional ground control measures may be necessary to prevent loss of ground in the face.

Procedures used to modify ground and groundwater conditions prior to tunneling include dewatering, chemical or cement grouting to penetrate the ground, jet grouting to form columns of soil-cement around the tunnel, and freezing. Structures that would be damaged by settlements can be directly supported by underpinning, or tied to reduce lateral extension of the structure. Compaction grouting is a process that can be used as required during tunneling to redensify the ground that has loosened above the tunnel before it causes surface settlement. It consists of a low slump grout placed at high pressures above the tunnel, usually after the shield has passed, in holes drilled down from the surface or up from the tunnel (Baker, et al., 1983).

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Chapter 10.2
ROCK MECHANICS

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The fundamental objective of rock mechanics is to predict the motion of rock. In mine engineering, the rock of interest is the mass adjacent to the open cuts, shafts, adits, stopes, entries, rooms, panels, and so forth that are excavated in the course of mining. Equations that are available for the purpose of calculating the motion of a rock mass follow from consideration of three basic sets of laws:

1. Physical laws.
2. Kinematics.
3. Material laws.

The prediction of rock mass motion is a formidable task. A less ambitious but more practical objective that is often associated with engineering design is to determine whether the anticipated displacements are within some acceptable limit. The limit may be implied or restricted outright to the elastic range of response. The tacit assumption is that beyond the elastic limit, a large increase in displacement may become possible with little increase in load. Roof falls, pillar spalls, and rock slides are examples. Under these circumstances, design analysis is essentially an analysis of safety and stability. The fundamental objective is then to calculate a factor of safety appropriate to the problem at hand.

10.2.1 PHYSICAL LAWS

The physical laws of most importance to rock mechanics are:

1. Conservation of mass.
2. Balance of linear momentum.
3. Balance of angular momentum.
4. Balance of energy.
5. Balance of entropy.

These laws are applicable to all rock types and, indeed, are independent of the nature of the material considered. Other physical laws such as the balance of charge could be added to the list, and would be required, for example, if electro-mechanical phenomena were to be considered. Application of physical laws taken in basic form from particle mechanics to the mechanics of bodies of finite size requires the introduction of continuum concepts including that of stress.

10.2.1.1 Concept of a Continuum

The great generality and simple form of the equations that express basic physical laws tend to mask certain conceptual difficulties that arise in their application to bodies of finite size in contrast with particle mechanics. In rock mechanics, the concept of a continuum seems particularly troublesome.

In fact, the concept of a continuum is straightforward: matter is assumed to be continuous to an infinitesimal subdivision; the discrete nature of matter at the molecular level is ignored. A cube 25 mm on edge (about 1 in.^{3}) of a monatomic gas at room temperature and pressure contains about one hundred billion billion atoms (10e + 20). Dense solids of similar volume contain 100 to 1000 times more atoms. For comparison, the grain size in granitic laboratory test specimens of rock is of the order of millimeters. A single quartz grain 1 mm (0.04 in.) on edge then contains about the same number of atoms (10e + 20). Imperfections in the crystalline structure of a grain in the form of dislocations are several orders of magnitude smaller than grain size but still involve very large numbers of atoms and are certainly amenable to a continuum mechanics analysis.

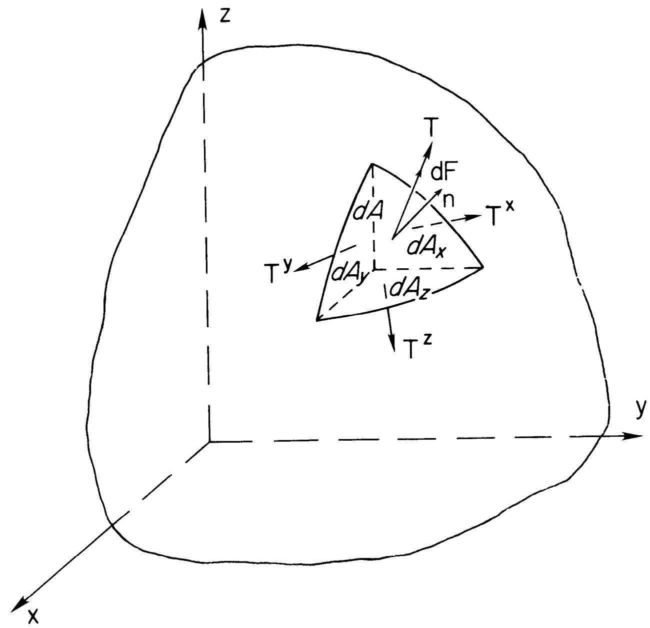

Fig. 10.2.1. Tetrahedral neighborhood of a surface point leading to the concept of stress.

The concept of a continuum does not require a uniform or even a smooth distribution of matter. Subgrain scale structural imperfections and heterogeneity in laboratory test specimens of rock are not excluded by the continuum concept. Geologic structures such as faults, joints, bedding planes, and so forth, and rock type variations at even larger scales of observations, are also entirely consistent with the continuum concept. The assumed continuous subdivision of matter does not preclude abrupt changes in its distribution.

10.2.1.2 Concept of Stress

The continuum concept of stress arises in consideration of the internal mechanical reaction of a body to deformation (Fung, 1965; Jaunzemis, 1967; Malvern, 1969). At the surface of a body in equilibrium, the external forces must be equal to the internal forces. Fig. 10.2.1 shows a small tetrahedral neighborhood of such a point. The external force dF is distributed over the surface dA of the tetrahedron to produce the traction T such that $dF = TdA$. The vectors dF and T have the same direction. The surface element dA has an associated outward unit normal vector n, as shown in Fig. 10.2.1. If the origin of coordinates is located at the internal vertex of the tetrahedron, then the internal surfaces of the tetrahedron are coordinate planes with areas dA_x, dA_y, dA_z. They are acted on by tractions T_x, T_y, T_z. Equilibrium requires

$$\begin{aligned} T_x &= \sigma_{xx}n_x + \tau_{yx}n_y + \tau_{zx}n_z \\ T_y &= \tau_{xy}n_x + \sigma_{yy}n_y + \tau_{zy}n_z \\ T_z &= \tau_{xz}n_x + \tau_{yz}n_y + \sigma_{zz}n_z \end{aligned} \tag{10.2.1}$$

where σ_{xx}, τ_{yx}, τ_{zx}, etc., are components of the tractions acting on the coordinate planes. The first subscript indicates the direction of the normal to the surface acted on, while the second subscript indicates the direction of action of the considered traction component. The direction cosines (components) of n are n_x, n_y, n_z.

When these three equations (10.2.1) are applied to all imaginary surfaces passing through an interior point of a body, the result is the famous Cauchy stress principle. By definition, the quantities σ_{xx}, τ_{yx}, etc., are the stresses. Stresses σ_{xx}, σ_{yy}, and σ_{zz} act normal to their associated surfaces and are direct stresses. The stresses τ_{xy}, τ_{yz}, τ_{zx}, etc., act parallel to their associated surfaces and are shear stresses.

For each direction n at a given point P, there corresponds a stress vector T with components given by Eq. 10.2.1. Formally, the set of stress vectors at P defines the state of stress at P. However, Eq. 10.2.1 shows that, as a practical matter, the stresses only need to be known relative to a given set of reference axes in order to specify the state of stress at a point.

The balance of angular momentum, in the absence of intrinsic angular momentum, shows an equality of shear stresses such that $\tau_{xy} = \tau_{yx}$, $\tau_{yz} = \tau_{zy}$, $\tau_{zx} = \tau_{xz}$. Thus the order of subscripts is not important, and only six stresses are necessary to define the state of stress at a point.

Introduction of the concept of stress into the balance of linear momentum leads to the stress equations of motion. Thus

$$\begin{aligned} \partial\sigma_{xx}/\partial x + \partial\tau_{yx}/\partial y + \partial\tau_{zx}/\partial z + \gamma_x &= \rho a_x \\ \partial\tau_{xy}/\partial x + \partial\sigma_{yy}/\partial y + \partial\tau_{zy}/\partial z + \gamma_y &= \rho a_y \\ \partial\tau_{xz}/\partial x + \partial\tau_{yz}/\partial y + \partial\sigma_{zz}/\partial z + \gamma_z &= \rho a_z \end{aligned} \tag{10.2.2}$$

Various representations of the state of stress are possible. The simplest is to use a list $(\sigma_{xx}, \sigma_{yy}, \sigma_{zz}, \tau_{xy}, \tau_{yz}, \tau_{zx})$. A one-by-six row vector is useful when matrix notation is convenient as is a six-by-one column vector. If the state of stress is represented by a row vector,

$$(\sigma) = (\sigma_{xx}\ \sigma_{yy}\ \sigma_{zz}\ \tau_{xy}\ \tau_{yz}\ \tau_{zx}), \tag{10.2.3}$$

then the same state is represented by the column vector,

$$\{\sigma\} = (\sigma)^t \tag{10.2.4}$$

where the superscript t means transpose. A three-by-three array is also a useful representation:

$$[\sigma] = \begin{bmatrix} \sigma_{xx} & \tau_{yx} & \tau_{zx} \\ \tau_{xy} & \sigma_{yy} & \tau_{zy} \\ \tau_{xz} & \tau_{yz} & \sigma_{zz} \end{bmatrix} \tag{10.25}$$

which represents the state of stress at a point referred to the (xyz) coordinate axes. It is often convenient to divide the stresses into spherical and deviatoric parts. The spherical part is the mean normal stress:

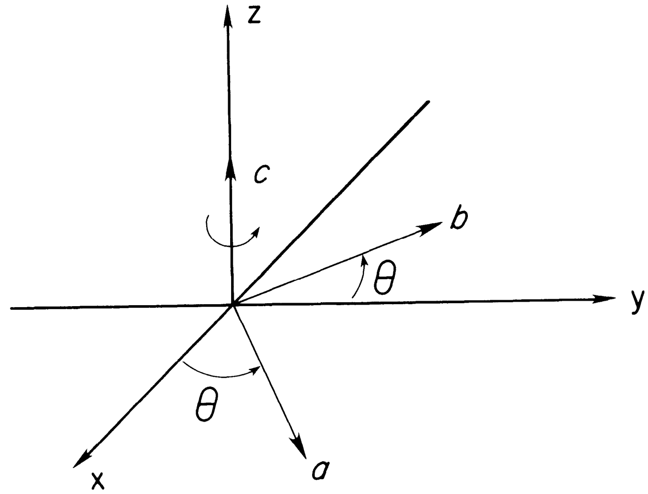

Fig. 10.2.2. Rotation of the reference axes about one of the given.

$$\sigma = (⅓)\,(\sigma_{xx} + \sigma_{yy} + \sigma_{zz}). \tag{10.2.6}$$

The deviatoric normal stresses are obtained by subtracting the mean normal stress, thus,

$$s_{xx} = \sigma_{xx} - \sigma,\ s_{yy} = \sigma_{yy} - \sigma,\ s_{zz} = \sigma_{zz} - \sigma, \tag{10.2.7}$$

are the deviatoric normal stresses. The deviatoric shear stresses are simply the given shear stresses; that is, $s_{xy} = \tau_{xy}$, etc. The state of stress in matrix form is then

$$[\sigma] = \begin{bmatrix} s_{xx} & s_{yx} & s_{zx} \\ s_{xy} & s_{yy} & s_{zy} \\ s_{xz} & s_{yz} & s_{zz} \end{bmatrix} + \begin{bmatrix} \sigma & 0 & 0 \\ 0 & \sigma & 0 \\ 0 & 0 & \sigma \end{bmatrix} \tag{10.2.8}$$

The spherical part of stress is also known as the hydrostatic component of stress in analogy with the pressure p at a point in a fluid at rest; that is, $p = \sigma$.

The symmetric array of real numbers (Eq. 10.2.5) can be transformed by a rotation of the reference axes to the diagonal form:

$$[\sigma(123)] = \begin{bmatrix} \sigma_1 & 0 & 0 \\ 0 & \sigma_2 & 0 \\ 0 & 0 & \sigma_3 \end{bmatrix} \tag{10.2.9}$$

where σ_1, σ_2, σ_3 are the major, intermediate, and minor principal stresses. The new reference axes (123) are the principal directions. Principal stresses are mutually perpendicular and act normal to principal planes. Principal planes are free of shear stresses; any plane free of shear stress is a principal plane. If any two principal stresses are equal, then the corresponding principal directions are not unique. The surfaces of naturally supported excavations in rock are free of shear stresses and are therefore principal surfaces. The normal to such a surface is a principal direction, and the associated normal stress is a principal stress whose value happens to be zero. If the excavation was an unlined water reservoir, the normal stress would still be a principal stress. However, its value would be equal to the water pressure.

Rotation of the reference axes about one of the given axes as shown in Fig. 10.2.2 is a very useful procedure. If the given axes are (xyz), and the new axes are (abc), then rotation through a positive counterclockwise angle Θ about the z axis is in matrix notation,

$$[\sigma(abc)] = [R][\sigma(xyz)][R]^t \tag{10.2.10}$$

where $[R]$ is the rotation matrix given by

$$[R] = \begin{bmatrix} \cos\Theta & \sin\Theta & 0 \\ -\sin\Theta & \cos\Theta & 0 \\ 0 & 0 & 1 \end{bmatrix} \tag{10.2.11}$$

After carrying out the multiplications in Eq. 10.2.10, the stresses referred to the (abc) axes are in algebraic form:

$$\begin{aligned}
\sigma_{aa} &= (\tfrac{1}{2})(\sigma_{xx} + \sigma_{yy}) + (\tfrac{1}{2})(\sigma_{xx} - \sigma_{yy})\cos(2\Theta) + \tau_{xy}\sin(2\Theta) \\
\sigma_{bb} &= (\tfrac{1}{2})(\sigma_{xx} + \sigma_{yy}) - (\tfrac{1}{2})(\sigma_{xx} - \sigma_{yy})\cos(2\Theta) - \tau_{xy}\sin(2\Theta) \\
\tau_{ab} &= -(\tfrac{1}{2})(\sigma_{xx} - \sigma_{yy}) + \tau_{xy}\sin(2\Theta) \qquad (10.2.12) \\
\sigma_{cc} &= \sigma_{zz} \\
\tau_{ca} &= \tau_{zx}\cos(\Theta) + \tau_{zy}\sin(\Theta) \\
\tau_{cb} &= -\tau_{zx}\sin(\Theta) + \tau_{zy}\cos(\Theta)
\end{aligned}$$

The first three equations of Eq. 10.2.12 are the usual two dimensional equations of stress transformation in the xy plane. The last three equations are the three dimensional part of the rotation about the z axis. Generally, three dimensional rotations can be obtained by a sequence of three simple rotations about an axis.

If the z direction is a principal direction, the z direction shear stresses vanish. The magnitudes of the principal stresses in the xy plane are then given by

$$\begin{aligned}
\sigma_1 &= (\tfrac{1}{2})(\sigma_{xx} + \sigma_{yy}) + [(\tfrac{1}{4})(\sigma_{xx} - \sigma_{yy})^2 + (\tau_{xy})^2]^{(1/2)} \\
\sigma_3 &= (\tfrac{1}{2})(\sigma_{xx} + \sigma_{yy}) - [(\tfrac{1}{4})(\sigma_{xx} - \sigma_{yy})^2 + (\tau_{xy})^2]^{(1/2)}
\end{aligned} \tag{10.2.13}$$

and the directions by

$$\tan(2\Theta) = \tau_{xy}/[(\tfrac{1}{2})(\sigma_{xx} - \sigma_{yy})]. \tag{10.2.14}$$

There are two solutions to Eq. 10.2.14. The solution that gives the direction of σ_1 can be decided by the fact that σ_1 will be nearest the largest given normal stress.

The xy shear stresses vanish on the principal planes associated with the xy principal stresses. In fact, rotation about the z axis through the angle given by Eq. 10.2.14 causes the xy shear stresses to vanish even if the z direction is not a principal direction. In that case, the stresses given by Eq. 10.2.13 are known as secondary principal stresses. Only if the z direction shear stresses are zero, are the stresses given by Eq. 10.2.13 true principal stresses. Since the z direction is arbitrary, there are an infinite number of secondary principal stresses at a point, but only three true principal stresses. A sequential determination of the secondary principal stresses allows for the determination of the true principal stresses, a fact that is used in the reduction of stress measurement data obtained in practice.

The shear stresses also have extremal values. There are six given in terms of the principal stresses by

$$\pm(\tfrac{1}{2})(\sigma_1 - \sigma_3), \quad \pm(\tfrac{1}{2})(\sigma_1 - \sigma_2), \quad \pm(\tfrac{1}{2})(\sigma_2 - \sigma_3) \tag{10.2.15}$$

The maximum and minimum shear stresses at a point are given by the first two equations of Eq. 10.2.15. They act on planes that are inclined at $\pm 45°$ to σ_1 and are parallel to the direction of σ_2.

Addition of stress is done by components. If the initial state of stress is $\{\sigma_i\}$, and stress changes $\{\Delta\sigma\}$ occur, the final state of stress $\{\sigma_f\}$ at the considered point is simply

$$\{\sigma_f\} = \{\sigma_i\} + \{\Delta\sigma\} \tag{10.2.16}$$

that is, $\sigma_{xx}(f) = \sigma_{xx}(i) + \Delta\sigma_{xx}$, $\tau_{xy}(f) = \tau_{xy}(i) + \Delta\tau_{xy}$, etc. It is important to note that all components are referred to the same reference axes. The principal stresses associated with the initial and final states are generally different and have different directions. It makes no sense, therefore, to write

$$\sigma_1(f) = \sigma_1(i) + \Delta\sigma_1 \tag{10.2.17}$$

In this regard, the stress changes also have principal components and directions; the latter are generally different from the principal directions associated with either the initial or final stress states. For example, in the gravity field, the vertical stress before mining is the major principal stress (compression considered positive). After excavation, say, of two stopes separated by a sill pillar in a vertical vein, the vertical stress on the sill floor is reduced to zero. The stress parallel to the sill floor remains compressive. The vertical stress thus becomes the minor principal stress after excavation, and the direction of the major principal stress rotates 90° from vertical to horizontal.

Closely associated with the concept of stress change is the concept of stress rate. If the state of stress is time dependent—that is, $\sigma = \sigma(x,y,z,t)$ where σ represents any particular component of stress, then the time rate of change of stress at a point in space is simply the partial derivative with respect to time, $\partial\sigma/\partial t$. However, if the stress rate for a given material element is desired, then the material derivative is used.

$$\begin{aligned} D\sigma/Dt = \partial\sigma/\partial t &+ (v_x\partial\sigma/\partial x) \\ &+ (v_y\partial\sigma/\partial y) + (v_z\partial\sigma/\partial z) \end{aligned} \tag{10.2.18}$$

Other stress rates are also of interest and are discussed in detailed treatments of continuum mechanics.

10.2.2 KINEMATICS

Kinematics deals with the geometry of motion. Like physical laws, kinematic relationships are independent of the constitution of the material and have great generality, especially when developed in general curvilinear coordinates. Only a brief outline of the more important kinematic relations including the concepts of motion, deformation, displacement, and strain are given here.

10.2.2.1 Motion of a Body

The concept of a motion of a material body in symbolic form relates the current position of all points in the body of interest to their initial position as a function of time. If x, y, z, and t denote the current position of a point that was initially at X, Y, Z, measured in the same Cartesian system, then the motion of the body may be described by

$$x = f(X,Y,Z,t), \quad y = g(X,Y,Z,t), \quad z = h(X,Y,Z,t) \tag{10.2.19}$$

where $X = f(X,Y,Z,0)$, $Y = g(X,Y,Z,0)$, and $Z = h(X,Y,Z,0)$. The functions f, g, and h are sufficiently smooth so that they can be inverted to give the initial positions as functions of the current positions. The motion described by Eq. 10.2.19 is a deformation of a body from an initial configuration to a final configuration. Spatial derivatives of Eq. 10.2.19 are deformation gradients.

Velocities and accelerations are the usual material time rates of change:

$$v_x = Dx/Dt, \quad v_y = Dy/Dt, \quad v_z = Dz/Dt, \qquad (10.2.20)$$
$$a_x = Dv_x/Dt, \quad a_y = Dv_y/Dt, \quad a_z = Dv_z/Dt.$$

The instantaneous motion in the neighborhood of a typical point in the body can be decomposed into two parts, a rigid body motion and a deformation. The length of a line segment between any two points of a body remains unchanged during rigid body motion, although its orientation may change. If x is the position vector with components (x,y,z), then the combination of rigid body translation and rotation in matrix rotation is

$$\{x(t)\} = \{a(t)\} + [R(t)]\,\{x(0)\} \qquad (10.2.21)$$

where $\{a(t)\}$ is the translation, $[R(t)]$ is the rotation matrix; $\{x(0)\}$ is the initial position, and $\{x(t)\}$ is the current position of a point of interest.

10.2.2.2 Displacements and Strains

Displacements relate the current positions of all points in the considered body to their initial positions:

$$x = X + u, \quad y = Y + v, \quad z = Z + w \qquad (10.2.22)$$

where u, v, and w are the time and position dependent Cartesian components of the displacement vector. Displacement should not be confused with distance traveled, which depends on the actual path followed between two points.

Elementary definitions of strains lead to complicated expressions in terms of displacement derivatives. However, when the strains are small in the sense that products of displacement derivatives are negligible, the usual definitions of infinitesimal strain apply. With reference to Fig. 10.2.3, if ϵ is the normal strain in the x-direction, then $\epsilon = \Delta L/L_0 \approx (\partial u/\partial x)$, or if γ is a shear strain in the xy-plane, then $\gamma = \tan(\alpha + \beta) \approx (\alpha + \beta) \approx (\partial u/\partial y + \partial v/\partial x)$, and in general,

$$\epsilon_{xx} = \partial u/\partial x, \qquad \epsilon_{yy} = \partial v/\partial y, \qquad \epsilon_{zz} = \partial \mathrm{w}/\partial \mathrm{z},$$
$$\gamma_{xy} = (\partial u/\partial y + \partial v/\partial x) = 2\epsilon_{xy},$$
$$\gamma_{yz} = (\partial v/\partial z + \partial w/\partial y) = 2\epsilon_{yz}, \qquad (10.2.23)$$
$$\gamma_{zx} = (\partial w/\partial x + \partial u/\partial z) = 2\epsilon_{zx},$$

where γ with double subscripts indicates the engineering shear strains. These are twice the tensorial shear strains indicated by ϵ with a double subscript.

In a rotation of the reference axes, the tensorial shear strains must be used. For example, if the state of strain is referred to a new set of axes (abc) rotated through an angle Θ about the z-axis, then

$$\epsilon_{aa} = (\tfrac{1}{2})(\epsilon_{xx} + \epsilon_{yy}) + (\tfrac{1}{2})(\epsilon_{xx} - \epsilon_{yy})\cos(2\Theta) + \epsilon_{xy}\sin(2\Theta)$$
$$\epsilon_{bb} = (\tfrac{1}{2})(\epsilon_{xx} + \epsilon_{yy}) - (\tfrac{1}{2})(\epsilon_{xx} - \epsilon_{yy})\cos(2\Theta) - \epsilon_{xy}\sin(2\Theta)$$
$$\epsilon_{ab} = -(\tfrac{1}{2})(\epsilon_{xx} - \epsilon_{xx}) + \epsilon_{xy}\sin(2\Theta)$$

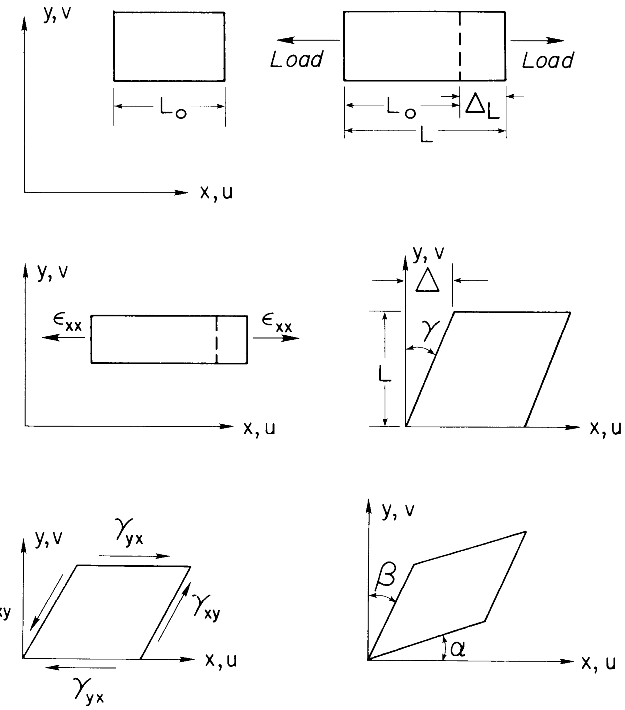

Fig. 10.2.3. Elementary concepts of normal and shear strain.

$$\epsilon_{cc} = \epsilon_{zz}$$
$$\epsilon_{ca} = \epsilon_{zx}\cos(\Theta) + \epsilon_{zy}\sin(\Theta)$$
$$\epsilon_{cb} = -\epsilon_{zx}\sin(\Theta) + \epsilon_{zy}\cos(\Theta) \qquad (10.2.24)$$

which are identical in form to the transformation of the stresses, Eq. 10.2.12.

The state of strain can be represented in the form of a symmetric three-by-three array:

$$[\epsilon] = \begin{bmatrix} \epsilon_{xx} & \epsilon_{yx} & \epsilon_{zx} \\ \epsilon_{xy} & \epsilon_{yy} & \epsilon_{zy} \\ \epsilon_{xz} & \epsilon_{yz} & \epsilon_{zz} \end{bmatrix} \qquad (10.2.25)$$

that has principal values ϵ_1, ϵ_2, and ϵ_3. In fact, the tensorial character of strain implies the same mathematical properties as for stress. All the formulas for principal stresses, directions, maximum and minimum shear stresses, stress changes, and so forth apply to the strains. The physical interpretation is quite different, of course. When the state of strain is decomposed into spherical and deviatoric parts, then the state of strain at a point has the representation,

$$[\epsilon] = \begin{bmatrix} e_{xx} & e_{yx} & e_{zx} \\ e_{xy} & e_{yy} & e_{zy} \\ e_{xz} & e_{yz} & e_{zz} \end{bmatrix} + \begin{bmatrix} \epsilon & 0 & 0 \\ 0 & \epsilon & 0 \\ 0 & 0 & \epsilon \end{bmatrix} \qquad (10.2.26)$$

where $\epsilon = (\tfrac{1}{3})(\epsilon_{xx} + \epsilon_{yy} + \epsilon_{zz})$, $e_{xx} = \epsilon_{xx} - \epsilon$, etc., are the deviatoric normal strains. The deviatoric shear strains are just the tensorial shear strains. The spherical part of strain is related to the dilatation Δ or change in volume per unit of original volume of material. Thus

$$\Delta = 3\epsilon \qquad (10.2.27)$$

is the dilatation. The deviatoric part of strain is associated with a change in shape.

10.2.3 MATERIAL LAWS

Material laws include such well-known relationships as Hooke's Law in elasticity theory, Darcy's Law in seepage analysis, and Fourier's Law of heat conduction. The need for material laws arises from the fact that physical laws and kinematics alone do not provide as many independent equations as there are unknowns for the description of the motion of a material body. Material laws also express relationships between dependent variables, for example, stress and strain, that arise in the description of the rock mass motion. The independent variables are position and time. Material laws such as Hooke's Law express broad categories of material behavior. In this regard, a particular rock type may respond elastically to an initial application of load, but with increasing load, reach an elastic limit and subsequently fracture and flow. Beyond the elastic limit, a more complicated material law is necessary. Of course, not all materials behave elastically even over a limited range of loading. Thus material laws, unlike physical laws and kinematical relations, are restricted in application by the nature of the material and the range of conditions of stress, temperature, moisture, and so forth that are encountered in a given mining environment.

Material laws may alternatively be viewed as simplified models of real material behavior. The three classic idealizations of material behavior are the elastic, plastic, and viscous models. Questions naturally arise as to what simplifications should be made and, once made, how well the idealized model corresponds to reality. Questions of this type relate more to the art than to the science of engineering design and have no final answers. Of course, experience gained against a background of insight and clear understanding of the fundamental concepts can be a great help.

10.2.3.1 Elasticity

In the linear elastic case (Love, 1944; Sokolnikoff, 1956; Lekhnitskii, 1963), the material law is

$$\{\sigma\} - \{\sigma(0)\} = [a]\,\{\epsilon\} \tag{10.2.28}$$

where the stresses and strains are six-by-one column matrices, and $[a]$ is a six-by-six material properties matrix. If the unstrained states of stress and strain coincide, then the initial stresses vanish, and

$$\{\sigma\} = [a]\,\{\epsilon\} = \begin{bmatrix} a_{11} & a_{12} & a_{13} & a_{14} & a_{15} & a_{16} \\ a_{21} & a_{22} & a_{23} & a_{24} & a_{25} & a_{26} \\ a_{31} & a_{32} & a_{33} & a_{34} & a_{35} & a_{36} \\ a_{41} & a_{42} & a_{43} & a_{44} & a_{45} & a_{46} \\ a_{51} & a_{52} & a_{53} & a_{54} & a_{55} & a_{56} \\ a_{61} & a_{62} & a_{63} & a_{64} & a_{65} & a_{66} \end{bmatrix} \{\epsilon\} \tag{10.2.29}$$

or in inverted form,

$$\{\epsilon\} = [b]\,\{\sigma\}. \tag{10.2.30}$$

where $[b]$ is the inverse of $[a]$. If the material is elastic, then 10.2.29 is the generalized Hooke's law, and the strains are recoverable upon unloading of the test specimen. Symmetry of the stresses and strains shows that $[a]$ and $[b]$ are also symmetric, $a_{12} = a_{21}, \ldots, a_{56} = a_{65}, b_{12} = b_{21}, \ldots, b_{56} = b_{65}$, so that there are at most 21 independent elastic constants. Such a material is highly directional or anisotropic even under a uniaxial test strain.

With increasing material symmetry, the number of elastic constants is reduced. An orthotropic material has three material axes and nine independent elastic constants. A gneiss or schist may be orthotropic; the three material axes correspond to the rift, grain, and hardway in quarry terminology. Many sedimentary rocks show pronounced differences in properties of samples parallel and perpendicular to the bedding and are transversely isotropic. Five elastic constants are needed for the description of transversely isotropic rock. Only two independent elastic constants are needed for the description of isotropic rock, which by definition lacks directional mechanical properties.

In the isotropic case, Hooke's law in extended form may be written as

$$\begin{aligned} E\epsilon_{xx} &= \sigma_{xx} - \nu\sigma_{yy} - \nu\sigma_{zz},\ G\gamma_{xx} = \tau_{xy}, \\ E\epsilon_{yy} &= \sigma_{yy} - \nu\sigma_{zz} - \nu\sigma_{yy},\ G\gamma_{zz} = \tau_{yz}, \\ E\epsilon_{zz} &= \sigma_{zz} - \nu\sigma_{xx} - \nu\sigma_{yy},\ G\gamma_{zx} = \tau_{zx}, \end{aligned} \tag{10.2.31}$$

where E is Young's modulus, v is Poisson's ratio, and G is the shear modulus (also known as the modulus of rigidity). The shear modulus, Young's modulus, and Poisson's ratio are related by the simple formula,

$$E = 2G(1 + \nu). \tag{10.2.32}$$

A number of other elastic constants are in use including the bulk modulus K and the Lame constants λ and μ. Any two constants can be used to derive a third constant. Table 10.2.1 shows the relationships between the more frequently used elastic constants.

Elastic constants such as Young's modulus and Poisson's ratio are properties that must be determined by physical measurement. The most common laboratory test is done on cylindrical samples prepared from diamond drill core. Samples are usually *NX* size (2⅛ in. diameter, or 54 mm) and are cut to a length about twice the diameter (4¼ in., or 108 mm). Application of an axial stress, say, in the z direction and simultaneous recordation of the axial and transverse strains allows for the determination of E and v.

Because of the cylindrical geometry of the test specimen, Hooke's law referred to cylindrical coordinates $(r\Theta z)$ are most convenient. Thus

$$\begin{aligned} E\epsilon_{rr} &= \sigma_{rr} - \nu\sigma_{\Theta\Theta} - \nu\sigma_{zz}, G\gamma_{r\Theta} = \tau_{r\Theta}, \\ E\epsilon_{\Theta\Theta} &= \sigma_{\Theta\Theta} - \nu\sigma_{zz} - \nu\sigma_{rr}, G\gamma_{\Theta z} = \tau_{\Theta z}, \\ E\epsilon_{zz} &= \sigma_{zz} - \nu\sigma_{rr} - \nu\sigma_{\Theta\Theta}, G\gamma_{zr} = \tau_{zr}. \end{aligned} \tag{10.2.33}$$

Inspection of Eq. 10.2.33 shows that the slope of a plot of the axial stress σ_{zz} (vertical axis) as a function of axial strain ϵ_{zz} (horizontal axis) is E. A consequence of the cylindrical geometry is equality of the radial and circumferential strains $(\epsilon_{rr}, \epsilon_{\Theta\Theta})$, and hence, Poisson's ratio is simply the ratio of the circumferential to axial strains. Fig. 10.2.4 (a) illustrates the uniaxial compression test for elastic moduli. An important feature of the idealized stress strain curve shown in Fig.10.2.4 (a) is the linear relationship between stress and strain.

The elastic moduli of a nonlinear elastic material are not constant but depend on strain. However, the shape of the material is recovered after a cycle of load application and release. Fig.

Table 10.2.1. Some Relations Among Elastic Constants

$E =$	$3K(1-2\nu) =$	$2G(1+\nu) =$	$9KG/(3K+G) =$	$(\lambda/\nu)(1+\nu)(1-2\nu)$
$G =$	$(E/2)/(1+\nu) =$	$(\lambda/2\nu)(1-2\nu) =$	$(3/2)(K-\lambda) =$	$(3KE)/(9K-E)$
$K =$	$\lambda + 2G/3 =$	$(E/3)/(1-2\nu) =$	$(GE/3)/(3G-E) =$	$(\lambda/3\nu)(1+\nu)$
$\nu =$	$(E/2G) - 1 =$	$\lambda/(3K-\lambda) =$	$(3K-E)/(6K) =$	$(\lambda/2)/(\lambda+G)$
$\lambda =$	$K - (2G/3) =$	$G(E-2G)/(3G-E) =$	$2G\nu/(1-2\nu) =$	$E\nu/[(1+\nu)(1-2\nu)]$

E = Young's modulus G = shear modulus K = bulk modulus

ν = Poisson's ratio λ = Lame's constant

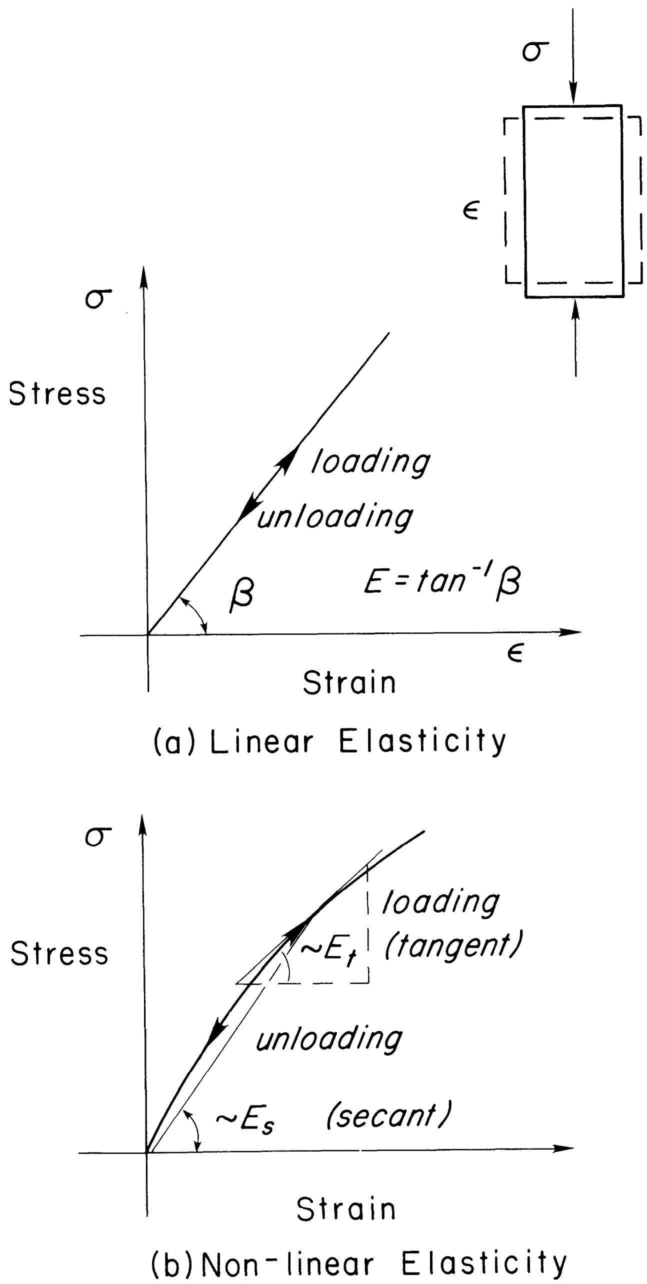

Fig. 10.2.4. Uniaxial compression test for elastic moduli (a) Linear elasticity, (b) Nonlinear elasticity.

10.2.4 (b) illustrates the concept of a nonlinear elastic material. The local slope of the uniaxial stress strain curve is known as the tangent modulus E_t. The slope of a straight line from the origin to a point on the uniaxial stress strain curve is the secant modulus, E_s.

STRAIN ENERGY. The work done on a body during deformation is stored in the body as strain energy. In one dimension, the area under the stress strain curve is the strain energy per unit volume of material. If A is the cross-sectional area of a prismatic bar and L is the length, then the stress $\sigma = F/A$ and the strain $\epsilon = (L - L_0)/L_0$, where L_0 is the original length of the sample. The work done $w = (\sigma^2/2E)$. In the case of one dimensional shear loading, $t = F/A$, and g is the corresponding shear strain. The work done $w = (t^2/2G)$. In three dimensions, the strain energy density may be written as a quadratic form,

$$W = (K\epsilon^2/2) + G[e^2_{xx} + e^2_{yy} + e^2_{zz}) + (1/2)(\gamma^2_{xy} + \gamma^2_{yz} + \gamma^2_{yz})] \qquad (10.2.34)$$

which partitions the strain energy into hydrostatic and deviatoric strain contributions. Use of the stress strain law allows for expression of the strain energy density as a quadratic form in terms of the stresses and as a bilinear form in terms of stresses and strain.

ELASTIC STIFFNESS. The concept of elastic stiffness is closely associated with the one dimensional stress-strain curve through a plot of force vs. displacement. From the one dimensional form of Hooke's law, $\sigma = E\epsilon$, the one dimensional force displacement relation, $F = K_nU$, may be obtained, where the normal stiffness $K_n = EA/L_0$. In the case of shear loading, the shear stiffness $K_s = GA/L_0$. Stiffnesses depend on the geometry of the structure and may be much less than the elastic moduli, especially when the structure is a frame or has holes in it. An example is a timber crib. Shear stiffness of a timber crib is likely to be much less than the shear modulus of individual timber pieces in the crib.

10.2.3.2 Plasticity

Plastic or permanent, irreversible strain occurs beyond the elastic limit. The simplest plasticity theory is an extension of time-independent elastic behavior. If the plastic deformation is time-dependent so that creep or relaxation phenomena occur, then a more complex plasticity theory is needed. In either case, a well-defined elastic limit is characteristic of the material. By contrast, viscous flow occurs in materials that lack a well-defined yield point, fluids, for example. However, compressible fluids may respond elastically to a uniform compression. Indeed, elastic, plastic, and viscous mechanisms of deformation may all be simultaneously present to a significant degree in some geologic media. Nevertheless, deformation beyond the elastic limit is plastic deformation (Hill, 1950; Kachanov, 1971).

Strength may be conceptually defined as the stress at the limit to a purely elastic response to load. Strength plays a central role in plasticity theories. While unloading from the elastic limit is an elastic process, continued loading at the elastic limit leads to inelastic deformation. The range of post-elastic deformation of a laboratory test specimen may be quite small and terminated by brittle rupture. However, ductile deformation of laboratory test specimens is observed as well as brittle failure. In this regard, post-elastic strain is composed of a recoverable elastic part and a nonrecoverable plastic part. Ductile deformation of rock is often associated with diffuse microcracking throughout a laboratory test specimen in contrast to the formation of a macroscopic rupture surface that transects the entire sample. In large rock masses, the formation of rupture surfaces localizes deformation onto the rupture surface but, of course, does not terminate the deformation process; motion of the rock mass must continue until a new equilibrium position is reached under the action of the applied loads. Thus jointed rock masses often appear to yield in a ductile manner, even though samples of intact rock between the joints may be brittle when tested in the laboratory at room temperature, pressure, and quasi-static loading rates.

EFFECTIVE STRESS. A concept that is essential to any discussion of rock strength is the concept of effective stress (Terzaghi and Peck, 1948; Taylor, 1948). The concept applies to porous media in general. Most rocks contain void spaces in the form of pores between grains and in the form of microcracks across grains and at grain boundaries. When the voids are interconnected, the presence of fluids changes the rock strength. By definition, the effective stress is just the difference between the total stress and the pore fluid pressure. The concept applies only to the normal stresses. Thus the effective stresses are

$$\sigma'_{xx} = \sigma_{xx} - p_f,\ \sigma'_{yy} = \sigma_{yy} - p_f,\quad \sigma'_{zz} = \sigma_{zz} - p_f \qquad (10.2.35)$$

where the prime indicates effective stress and compression is considered positive. According to the concept of effective stress, strength is a function of the effective stresses only, although the total stresses determine equilibrium. Effective stresses are also known as the intergranular stresses; that is, the stresses transmitted through the solid skeleton of the porous sample. The solid skeleton provides resistance to failure. For this reason, it is the effective stresses that determine the strength of porous media. If the material lacks interconnected void spaces or pore fluids are absent, then the effective and total stresses are the same. Effective stresses are implied in any discussion of rock strength.

PLASTIC POTENTIAL. The strength of rock formally enters plasticity theory in two ways: (1) as a plastic potential function and (2) as a loading criterion. In the first way, a yield function or failure criterion, when satisfied by the state of stress at a point, is differentiated with respect to the stresses obtained from the plastic contribution to the total strain. In the second way, the total differential of the failure criterion, when satisfied, is set equal to zero in order to maintain a smooth transition from one plastic state to a neighboring state. In this regard, the material properties may depend on the amount of plastic deformation that has occurred, so that the failure criterion may depend implicitly on the plastic strains as well as the stresses. If the plastic potential and loading function are the same, then the resulting plastic stress strain relations are known as associated rules of flow. However, the two need not be the same, and the rules of flow may be nonassociated. In either case, the stress strain relations have the incremental form,

$$\{d\sigma\} = ([E_e] - [E_p])\,\{d\epsilon\} \qquad (10.2.36)$$

where $[E_e]$ is the elastic material properties matrix, and $[E_p]$ is the plastic contribution to the overall material properties matrix. Generally, $[E_p]$ depends on the current state of stress and strain, so that Eq. 10.2.36 is nonlinear even when the plastic strains are

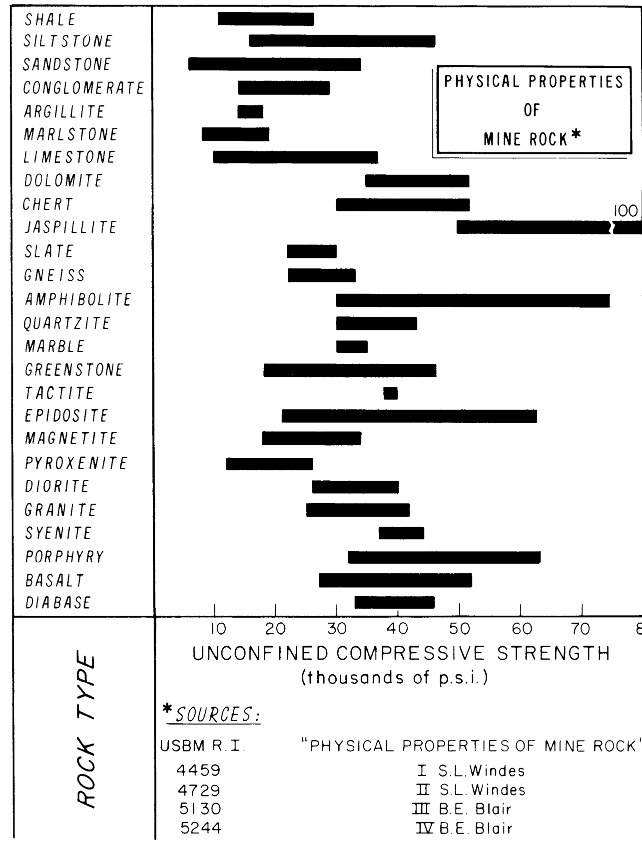

Fig. 10.2.5. Uniaxial strength data for several rock types. Conversion factor: 1 psi = 6.895 kPa.

the same order of magnitude as the elastic strains, which is often the case in rock mechanics.

In the realm of plasticity theory, care should be exercised in consulting the literature because of the frequent specialization of theory for applications to metals and the omission of a more general treatment applicable to geologic media. Ductile metals typically yield regardless of the hydrostatic component of stress; they are pressure-independent. Geologic media are characterized by pressure-dependent failure criteria.

STRENGTH OF INTACT ROCK. If a small test cube of rock is loaded to the elastic limit under a number of different combinations of principal stresses (σ_1, σ_2, σ_3), the resulting locus of points plotted in stress space defines the strength of the material. The strength of the rock is then given by the function $F(\sigma_1, \sigma_2, \sigma_3)$. The function F is a criterion for failure, and its form may be suggested according to some empirical rule (Nadai, 1950; Jaeger, 1967, 1969).

Any such rule should be consistent with experimental data and, for practical use, have a simple mathematical form. Fig. 10.2.5 shows some uniaxial strength data obtained from a variety of rock types. In this regard, the tensile strength of rock T_0 is generally only a small fraction of the unconfined compressive strength C_0, usually less than 10% of the compressive strength, and often much less. Fig. 10.25 shows there is a wide range of strengths for a given rock type; there appears to be no correlation with a genetic classification of rock types. Another feature is great variability within a given set of samples; the coefficient of variation often shows a standard deviation, that is, 30% of the mean strength. Variability is the rule rather than the exception.

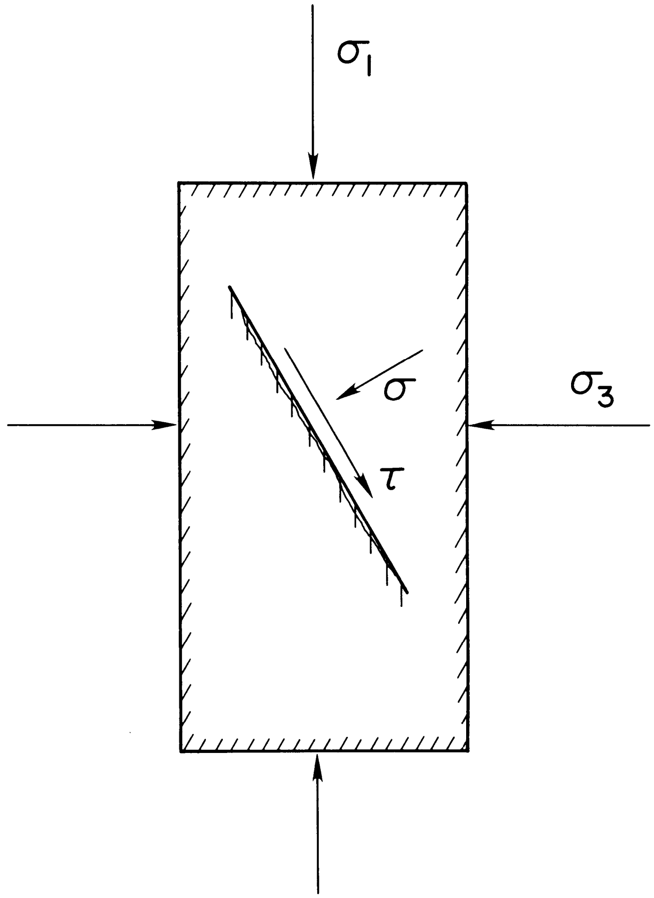

Fig. 10.2.6. Mohr concept of failure.

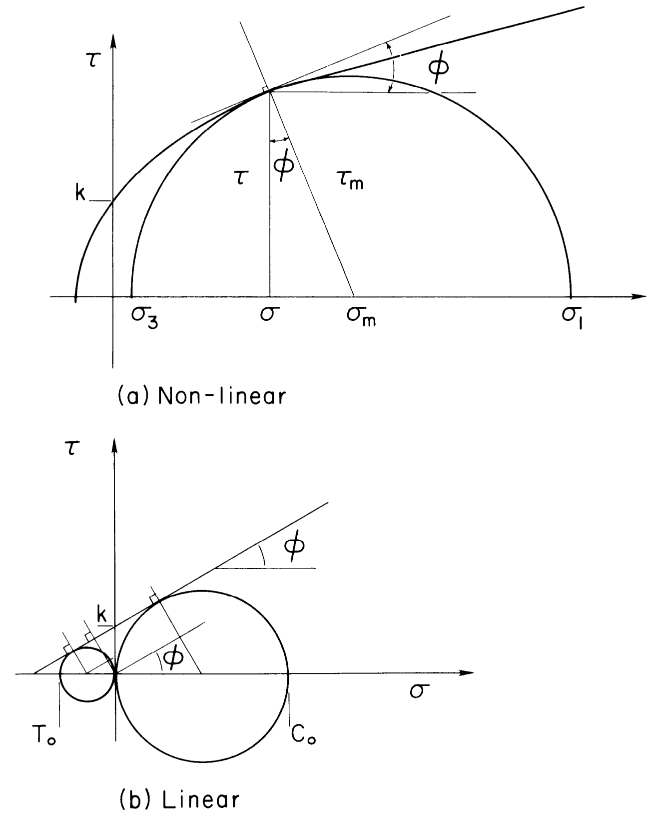

Fig. 10.2.7. Mohr failure criterion in graphical form (a) Non-linear, (b) Linear.

A number of failure criteria that have been suggested include (Nadai, 1950; Jaeger, 1969):

1. Maximum principal stress.
2. Maximum shear stress.
3. Maximum principal strain.
4. Maximum shear strain.
5. Maximum strain energy.
6. Maximum shear strain energy.
7. Maximum octahedral shear stress.
8. Griffith theory.
9. Mohr-Coulomb theory.
10. Drucker-Prager theory.
11. Nonlinear forms of (9) and (10).

None of the simple rules (1) to (7) are consistent with experiment, nor is the Griffith theory (8), and a more sophisticated approach to defining the form of a failure criterion for rock is needed.

According to the Mohr description (Nadai, 1950, 1963), there is a functional relationship between the normal and shear stresses acting on a potential failure surface, as shown in Fig. 10.2.6. Thus, at the onset of failure, $\tau = f(\sigma)$. When applied to the usual laboratory test for compressive strength under confining pressure, an extension of the Mohr criterion leads to $\tau_m = f(\sigma_m)$ where $\tau_m = (\frac{1}{2})(\sigma_1 - \sigma_3)$ and $\sigma_m = (\frac{1}{2})(\sigma_1 + \sigma_3)$. The simplest forms of $f(\sigma_m)$ are

1. Constant: $\tau_m = A$
2. Linear: $\tau_m = A + B\sigma_m$
3. Quadratic: $(\tau_m)^2 = A + B\sigma_m$
4. *n-type:* $(\tau_m)^n = A + B\sigma_m$

where *A*, *B*, and *n* are different experimental constants, depending on the form that best fits the data. They are the strength properties of the test specimen and are related to the uniaxial compressive strengths. Criterion 1 is the well-known Tresca criterion used in metal plasticity; 2 includes the famous Mohr-Coulomb criterion that finds great use in soil mechanics and to some extent in rock mechanics; and 3 is associated with the name of Torre, while the *n*-type condition is due to writer.

A plot of the Mohr-Coulomb criterion and a typical Mohr's circle representing the stress state at failure in the usual laboratory test for compressive strength under confining pressure is shown in Fig. 10.2.7, where compression is considered positive. Under these conditions, the intermediate and minor principal stresses are equal, and the Mohr-Coulomb criterion is

$$\tau = \sigma \tan(\phi) + k \qquad (10.2.37)$$

where k is the shear (t) axis intercept and is known as the cohesion; t is the inclination to the normal (σ) stress axis, and is the angle of internal friction. The criterion expressed by Eq. 10.2.37 has the interesting property of being an envelope of Mohr circles representing the stress states at failure. With a change in variables, Eq. 10.2.37 can be expressed as

$$\tau_m = k \cos(\phi) + \sigma_m \sin(\phi) \qquad (10.2.38)$$

The angle of internal friction is quite different from the ordinary friction angle associated with sliding surfaces in contact. In this regard, the failure surface referred to in Mohr theory is a virtual surface. If failure occurs by fracture, then an actual surface is created during the failure process. Subsequent slip on such a surface or on an existing failure surface would be analogous to frictional slip. However, if failure occurs by yielding, then the surface remains a virtual surface.

The strength parameters, cohesion k and angle of internal friction ϕ of the Mohr-Coulomb criterion, are related to the uniaxial compressive and tensile strengths C_0 and T_0 as can be seen from the geometry of the Mohr's circle representing the stress state at failure and the failure criterion. Inspection of Fig. 10.2.7 shows that in the linear case,

$$\begin{aligned} C_0 &= (2k\cos(\phi))/(1 - \sin(\phi)) \\ T_0 &= (2k\cos(\phi))/(1 + \sin(\phi)) \qquad (10.2.39) \\ \sin(\phi) &= [(C_0/T_0) - 1] \,/\, [(C_0/T_0) + 1] \\ k &= (\tfrac{1}{2})[C_0T_0]^2. \end{aligned}$$

The geometry of Fig. 10.2.7 also shows that the angle m between a Mohr-Coulomb failure surface and the major compression is $\pm(\pi/4 - \phi/2)$. In an actual test, the state of stress in the sample, while axially symmetric, is generally not uniform. As a consequence, the failure surface even when it manifests itself as a fracture is curved, and only a very rough measurement of m is possible. There is also a possibility that the Mohr envelope does not exist for some stress states near the origin.

Quadratic criteria lead to a variable failure surface angle and are generally more suitable for describing the failure of rock. There are several quadratic criteria that have been proposed in rock mechanics that are parabolas when plotted in the plane of (σ_3, σ_1) or in the plane of (σ_m, τ_m). These are (1) Fairhurst, (2) Hoek-Brown, (3) standard, and (4) n-type, (super-parabola). The first two have special properties and contain tacit assumptions about the material behavior. For example, No. 2 does not cross the σ_m axis at 90°, although close fits to strength data obtained under confining pressure are obtained.

A peculiarity of Mohr theory when applied to the standard laboratory test for strength is the absence of the intermediate principal stress in the failure criterion. A criterion that lacks one of the principal stresses leads to edges on the failure surface when plotted in principal stress space $(\sigma_1, \sigma_2, \sigma_3)$ that when combined with a difference in uniaxial compressive and tensile strengths leads to an irregular-shaped pyramid as shown in Fig. 10.2.8 (a) (Shield, 1955). There is a dearth of experimental data concerning the importance of the intermediate principal stress on the strength of rock because of the difficulties associated with the experimental access to arbitrary, three dimensional stress states.

However, there are alternatives to Mohr theory that also have the advantage of leading to smooth yield surfaces in stress space. The conical surface shown in Fig. 10.2.8 (b) is a graphical representation of the widely used Drucker-Prager criterion (Drucker and Prager, 1952):

$$(J_2)^{(1/2)} = A(I_1) + B \qquad (10.2.40)$$

where A and B are strength properties that are related to the uniaxial strengths by

$$\begin{aligned} A &= (1/\sqrt{3})(C_0 - T_0)/(C_0 + T_0), \\ B &= (2/\sqrt{3})(C_0T_0)/(C_0 + T_0), \end{aligned}$$

and (10.2.41)

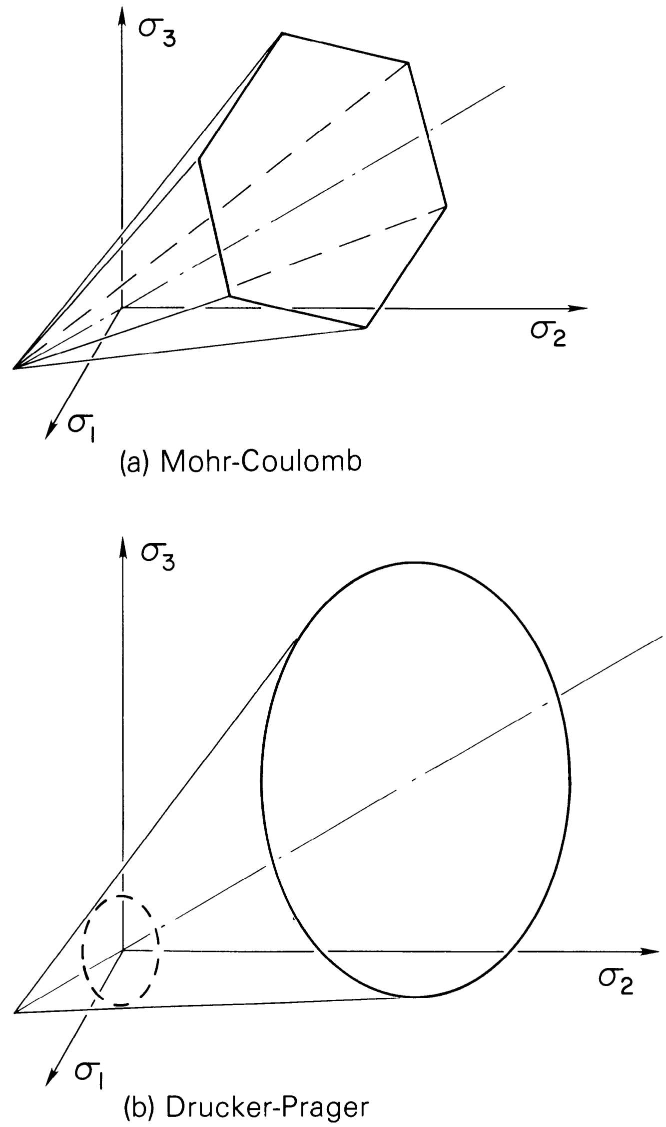

Fig. 10.2.8. Mohr-Coulomb and Drucker-Prager yield criteria in principal stress space.

$$\begin{aligned} (J_2)^{(1/2)} &= \{(\tfrac{1}{6})[(\sigma_1 - \sigma_2)^2 + (\sigma_2 - \sigma_3)^2 \\ &\quad + (\sigma_3 - \sigma_1)^2]\}^{(1/2)}, \\ I_1 &= (\tfrac{1}{3})(\sigma_1 + \sigma_2 + \sigma_3). \end{aligned}$$

The quantity $(J_2)^{(1/2)}$ is akin to a root-mean-square measure of shear stress and is analogous to the maximum shear stress t_m, while I_1 is a true mean normal stress analogous to the average normal stress σ_m. In fact, J_2 and I_1 are invariant with respect to rotation of the reference axes, and thus meet a basic requirement for the description of failure criteria appropriate to isotropic materials. If the compressive and tensile strengths are equal, then $A = 0$, and the yield condition is independent of the hydrostatic component of stress. Eq. 10.2.40 then reduces to the famous Von Mises yield function that is in widespread use in metal plasticity.

Quadratic and n-type forms of $J_2 - I_1$ yield functions or failure criteria that include the effects of the intermediate principal stress are possible. The n-type form is

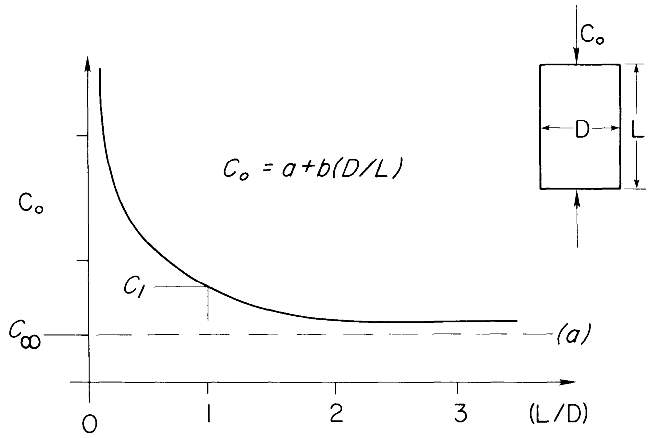

Fig. 10.2.9. Size effect on compressive strength.

$$(J_2)^{(n/2)} = A(I_1) + B \qquad (10.2.42)$$

and is a smooth surface in stress space.

Anisotropic forms that take into account directional properties of certain rock types are also in use (Pariseau, 1972). In this regard, shear properties that are distinctly different from values computed on the basis of isotropy play a major role in the analysis of ground motion where a distinct bedding, foliation, or schistosity is present. Examples include subsidence analysis or soft coal mines in flat-lying strata and the stability of underground excavations in metamorphic rocks such as phyllites and schists (Pariseau, Duan, and Schmuck, 1984; Pariseau et al., 1986). In this regard, all yield functions known to the writer are, in fact, polynomial functions of the stresses.

SIZE EFFECTS. An interesting feature of laboratory test results for compressive strength is a so-called size effect (Obert and Duvall, 1967; see also Chapter 10.5). The effect is most often observed in uniaxial compressive strength test data where the strength is computed by dividing the axial force at failure by the cross-sectional area of the test specimen. A different result is obtained depending on the height of the sample for a given diameter. The same phenomenon is observed in testing prisms. The compressive strength apparently depends on the "size" of the sample. Several simple formulas have been fit to such data; the most common one has the form,

$$C_0 = a + b(1/r) = a + b(D/L) \qquad (10.2.43)$$

where r is the ratio of sample height to diameter (L/D ratio). Fig. 10.2.9 shows a plot of Eq. 10.2.43. The curve is a hyperbola. According to the Eq. 10.2.43, the compressive strength approaches a minimum value a, as the height of the sample becomes very large at fixed diameter. This observation coupled with the presence of friction shear stresses across the ends of the specimen and the fact that rock compressive strength is generally confining-pressure dependent suggests an "end effects" explanation of the observed results. A simple equilibrium analysis shows this to be the case. With reference to Fig. 10.2.10, the shearing forces at the ends of the test specimen must be equilibrated by an equal but opposite horizontal force distributed over the midplane of the sample. The required force effectively confines the test specimen by a pressure p such that $P = (p)(LD)$. The average shear stress over the sample ends is $t = T/(pD^2/4)$. Hence $p = (tp/4)(D/L)$. According to the Mohr-Coulomb criterion,

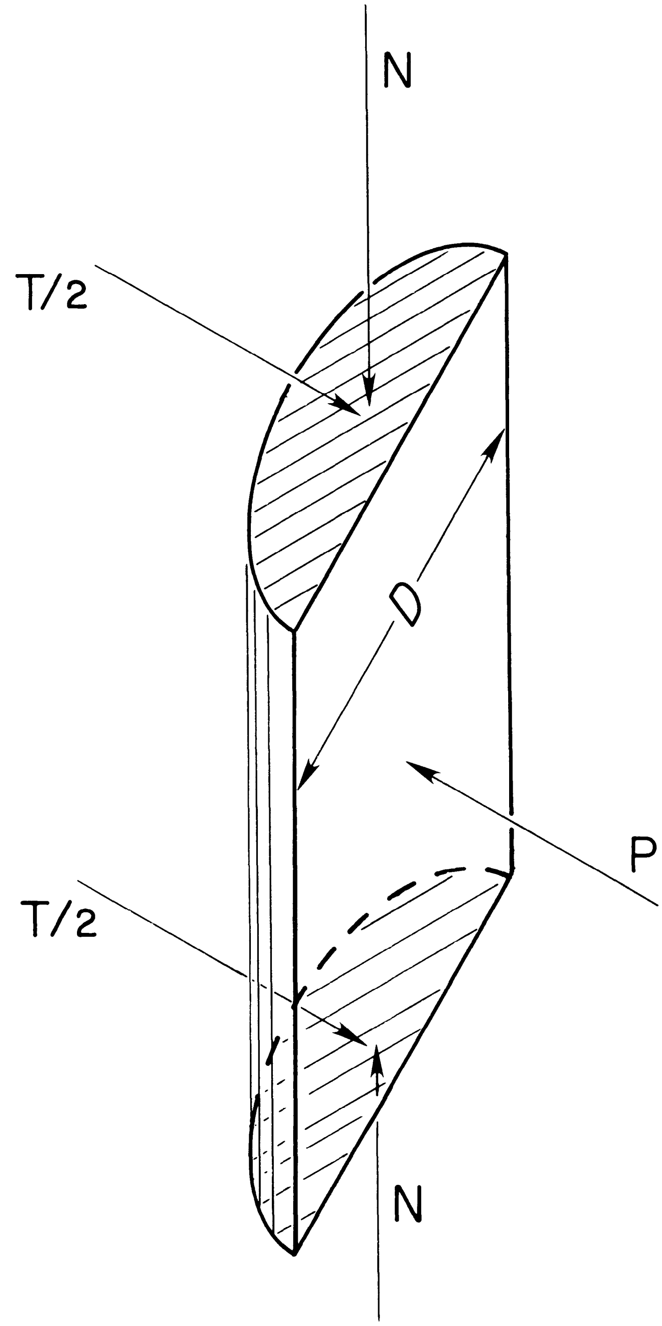

Fig. 10.2.10. End effects forces.

$$C = C_0 + (C_0/T_0)p \qquad (10.2.44)$$

and thus,

$$C = C_0 + (C_0/T_0)(\tau\pi/4)(D/L) \qquad (10.2.45)$$

which has the same form as the size-effects relation, Eq. 10.2.43. Since t is expected to be independent of the ratio L/D, Eq. 10.2.45 can be written as

$$C = C_1[a' + b'(D/L)] \qquad (10.2.46)$$

where $a' = (C_0/C_1)$, $b' = (1 - C_0/C_1)$, and C_1 = compressive

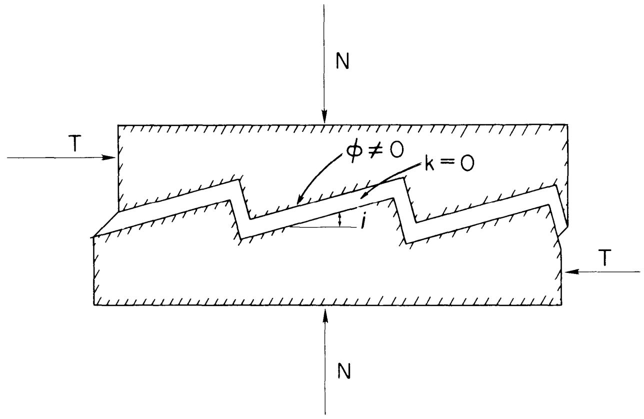

Fig. 10.2.11. Conceptual model of joint asperities.

strength at $L/D = 1$. Detailed computer simulations of the laboratory test for compressive strength support the end-effects explanation of Eq. 10.2.43 (Pariseau, 1979).

However, statistical theories of strength may also be used to explain size effects (Jaeger and Cook, 1969), although the evidence is not decisive because of the natural variability in rock, the need to test enormous numbers of samples, and the difficulties associated with test interpretation in the presence of end effects and possibly stress gradients as occur in a beam bending test for tensile strength. In any case, size effects in laboratory test data should not be extrapolated beyond the range of experiment, in particular, not to the scale of excavations in rock masses.

STRENGTH OF ROCK JOINTS. Joints refer to a large variety of discontinuities commonly present in rock masses. Examples are faults, bedding planes, and contacts between different rock types. The description of the strength of rock joints is similar in many respects to intact rock strength (Goodman, 1980; Brady and Brown, 1985; Einstein and Dowding, 1989). Both are based on experimental observations and the practical need for simple mathematical forms. Joints are solid surfaces in contact. Although referred to as surfaces, joints are not very smooth. Moreover, additional material may be present along a joint surface in the form of fault gouge or cementatious chemical precipitates of various kinds. Joint topography and filling suggests that they may be viewed as a thin layer of material whose properties differ from those of the adjacent intact rock. The thickness of a joint is intuitively related to joint roughness and to the height of the surface asperities. If the asperity height is δ, perhaps the joint thickness is 2δ. Thus joints may be assigned a thickness and a volume, and then described in a conventional manner. All the various failure criteria applied to intact rock may be used for the description of joint strength. However, there are phenomena associated with joint mechanics that are not observed in intact rock.

The strength of a clean tensile fracture that is free of extraneous material and lacks cohesion is entirely frictional. However, the topography of such a joint influences the apparent angle of friction (Patton, 1966) when first sheared. With reference to Fig. 10.2.11, if shearing loads are applied to the sawtooth-shaped joint surface, and the top block slides upwards parallel to the forward slope of the sawtooth-like asperities, which are in frictional contact, then resolution of forces parallel and perpendicular to the joint plane shows that

$$T = N \tan(\phi + i) \tag{10.2.47}$$

where ϕ is the angle of sliding friction, i is the inclination of the asperity face to the joint plane, and $(\phi + i)$ is the apparent angle of sliding friction on the joint. The expansion of the joint under the action of the shearing force is known as shear dilatancy. When several joints in parallel are tested simultaneously, any mismatch in the peaks and valleys of the asperities tends to obscure shear dilatancy (Brechtel, 1978).

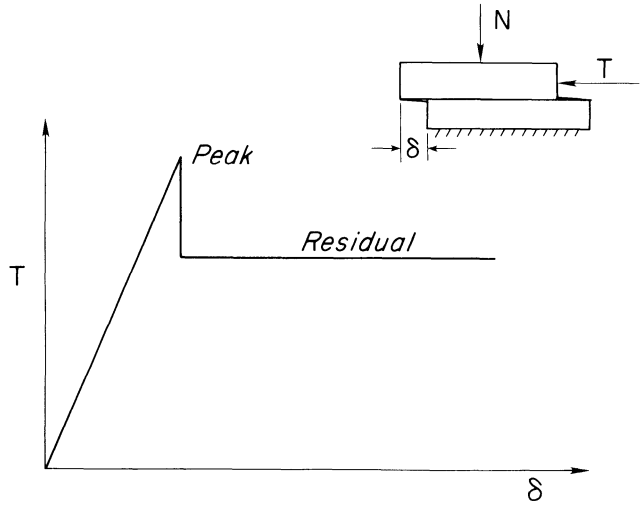

Fig. 10.2.12. Idealized shear force shear displacement test of a cohesionless joint showing peak-residual behavior.

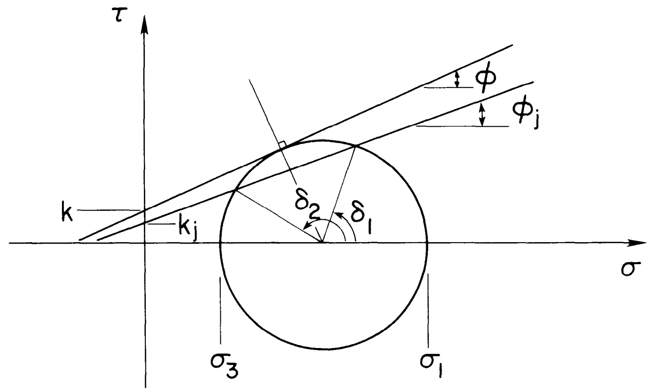

Fig. 10.2.13. Mohr-Coulomb joint failure criterion when the joint strike is parallel to the intermediate principal stress.

Under continued shearing, the asperity tips will fail and form a gouge that tends to fill the valleys between asperity peaks. As a consequence, the joint surface friction angle tends to decrease towards a residual value. Fig. 10.2.12 shows an idealized plot of shear force as a function of shearing displacement obtained during a direct shear test of a cohesionless joint that shows peak residual behavior. Repetition of the test under higher normal loads changes the outcome of the test. The transition from peak to residual frictional sliding is potentially unstable. Peak residual behavior may be suppressed entirely at very high normal loads.

The most commonly used joint strength criterion in practice is the Mohr-Coulomb criterion, which attributes frictional and cohesive strengths to the joint. The frictional component is dependent on the normal stress, of course. Fig. 10.2.13 shows a plot of the Mohr-Coulomb criterion for a joint inclined at an

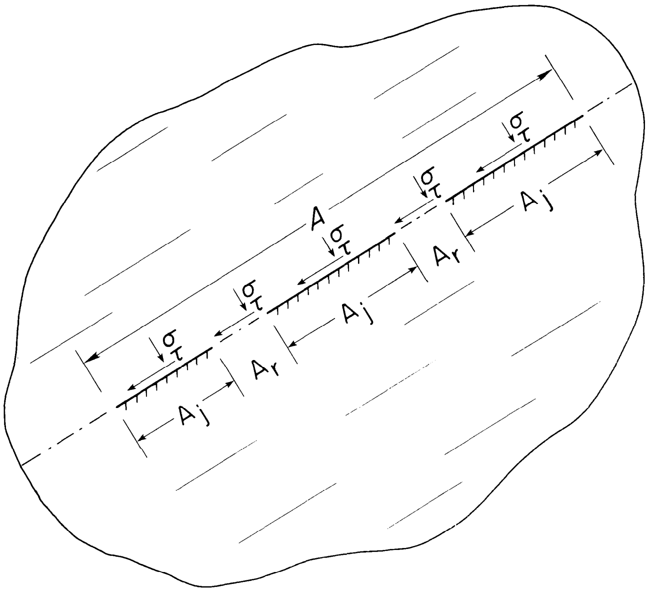

Fig. 10.2.14. Planar failure in a jointed rock mass.

angle δ to the horizontal and subject to applied stresses σ_1 and σ_3 (compression positive). The intermediate principal stress σ_2 is assumed to be parallel to the strike of the joint. The Mohr circle that represents that state of applied stress extends beyond the joint failure line. This is possible in the case of joints because the failure surface is fixed in advance of the test, unlike the case of intact rock where the failure surface is free to orient itself relative to the applied stresses. With reference to Fig. 10.2.13, only joints within the range (δ_1,δ_2) are liable to slippage and failure. In three dimensions where the principal stresses are not aligned with the joint plane geometry, the normal and shear stresses acting on the joint surface must be determined by a rotation of the reference axes before the strength of the joint can be determined. A comparison of joint shear strength with joint shear stress can then be made to determine the potential for slip.

STRENGTH OF WELL-JOINTED ROCK MASSES. Although the mechanics of well-jointed rock masses is an area of active research and is central to engineering design of excavations in rock, procedures for rational estimation of rock-mass strength from test data obtained on small samples of intact rock and rock joints remain to be firmly established within the context of plasticity theory (Pariseau, 1989). However, an empirical formula of longstanding can be obtained by combining the strength of intact rock and joints according to a simple area-weighting formula due to Terzaghi (1962).

Mohr-Coulomb failure criteria are assigned to the intact rock and the joint. Failure is assumed to occur along a plane as shown in Fig. 10.2.14. The failure surface area A is composed of joint area A_j and the area A_r of the intact rock bridges between the joints. The applied normal force N acts perpendicular to the failure surface; the resisting shear force T is parallel and composed of joint shear resistance T_j and rock shear resistance T_r. The forces are simply stresses multiplied by areas. Thus,

$$\tau = [\sigma_j \tan(\phi_j) + k_j](A_j/A) + [\sigma_r \tan(\phi_j) + k_r](A_r/A) \qquad (10.2.48)$$

where the subscripts j and r refer to joint and intact rock, respectively. If the assumption that the rock and joint friction angles are equal and the joint is cohesionless, then the composite strength is

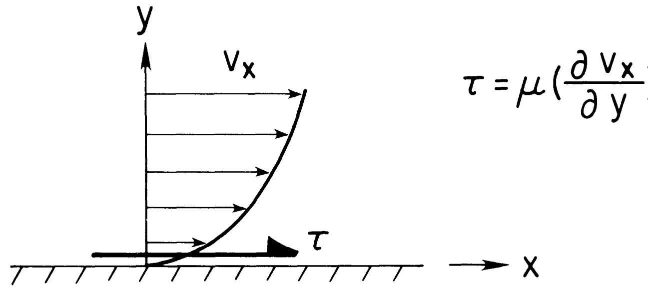

Fig. 10.2.15. Elementary viscous shear.

$$\tau = \sigma \tan(\phi) + (k_r)(A_r/A). \qquad (10.2.49)$$

Eq. 10.2.49 shows that the cohesion of a jointed rock mass is likely to be only a small fraction of the cohesion of the intact rock. Stress concentrations at joint ends and progressive failure are likely to lower the rock mass strength even more. Progressive failure is associated with extension of existing joint surfaces at the expense of the rock bridges and thus leads to a displacement dependent cohesion that decreases with successive episodes of motion associated with periodic blasting, cycles of rainfall, or even heating and cooling. The process may be unstable in the sense that it may lead to triggering of catastrophic motion (Pariseau and Voight, 1979).

10.2.3.3 Time-dependent Material Laws

Time may enter material laws in several ways. A material may age with time, meaning that its material properties change in time. The increase of strength as concrete cures is an example. The loss of strength by some shales upon exposure to air in coal mines is another. Properties that are sensitive to moisture content and variation of moisture content in time also may be interpreted as time dependent material behavior. Fluid flow in porous rock imparts a time dependency to the motion. Perhaps the most important source of time-dependent material behavior is the presence of viscous phenomena.

Although usually associated with fluid motion, viscous behavior is also observed in solids. A characteristic feature of purely viscous behavior is that flow occurs with the application of shear stress. Flow stops with the removal of shear stress. However, the material does not return to its original configuration; the deformation is not elastic. Incompressibility is often assumed with viscous behavior, although it is not necessary to do so; density may be allowed to depend on pressure. An elementary model of viscous behavior shown in Fig. 10.2.15 assumes a proportionality between the shear stress t and a derivative of the flow velocity taken normal to the shear surface. The latter is a shear strain rate (strictly speaking, a rate of deformation), and the constant of proportionality is the viscosity:

$$\tau = \mu d \qquad (10.2.50)$$

where m is viscosity of the material, and d is the engineering rate of shear deformation = $\partial v/\partial y$.

The viscosity of silicate glass at room temperature (68°F or 20°C) is about 10^{22} poise (dyne-sec-cm^{-2}) or about 10^{17} psi (kg/m^2) sec (Jaeger and Cook, 1969). An intact silicate rock having the same viscosity and subject to a 1000 psi (7-Mpa) load would creep about 10 microstrain units in 100 years. Thus true creep of intact hardrock is likely to be insignificant relative to engineering design. However, jointed rock masses may show significant creep

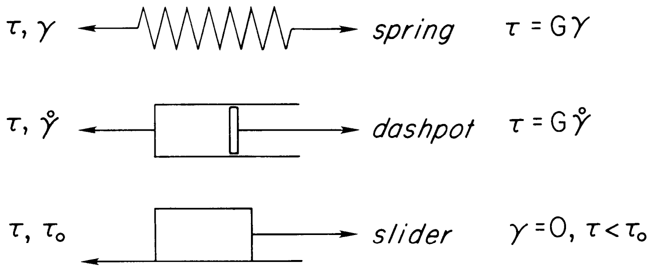

Fig. 10.2.16. Rheologic elements in time-dependent material models.

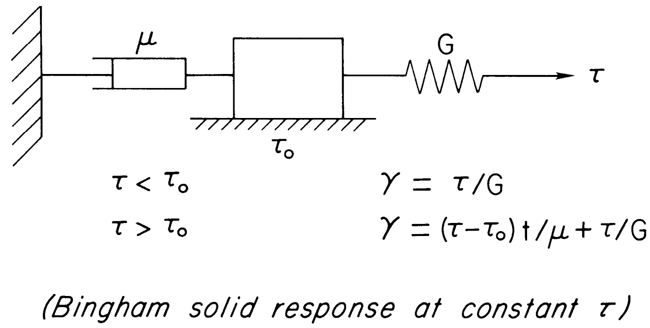

Fig. 10.2.17. Bingham solid response.

because of viscous joint filling or locally high stress on joint asperities which leads to fracture and flow. Soft rock such as shales and rock types that are composed of minerals that crystallize in the cubic system (e.g., evaporites) may also show significant creep.

The linear three dimensional case is obtained by applying Eq. 10.2.50 to the deviations of stress and rates of deformation (Malvern, 1969). Thus

$$\begin{aligned} s_{xx} &= 2\mu\, d_{xx}, \quad s_{xx} = 2\mu\, d_{xx}, \quad s_{xx} = 2\mu\, d_{xx}, \\ \tau_{xy} &= 2\mu\, d_{xy}, \quad \tau_{yz} = 2\mu\, d_{yz}, \quad \tau_{zx} = 2\mu\, d_{zx}, \end{aligned} \tag{10.2.51}$$

where the deviators of the deformation rates are

$$\begin{aligned} d_{xx} &= (\partial v_x/\partial x) - d, \\ d_{yy} &= (\partial v_y/\partial y) - d, \\ d_{zz} &= (\partial v_y/\partial z) - d \\ \tau_{xy} &= (\partial v_x/\partial y + \partial v_y/\partial x), \\ \tau_{yz} &= (\partial v_y/\partial z + \partial v_z/\partial y), \\ \tau_{zx} &= (\partial v_z/\partial x + \partial v_x/\partial z), \\ d &= (1/3)(\partial v_x/\partial x + \partial v_y/\partial y + \partial v_y/\partial z) \end{aligned} \tag{10.2.52}$$

and m is the shear viscosity of the material. Materials may also have a bulk viscosity that relates the departure of the mean pressure from the equilibrium pressure to the rate of change of volume.

The viscous deformation is still only part of the total deformation of a solid that shows significant time-dependent strain. In this regard, rheological models guide the development of material laws for such solids. There are many such models, but all are based on the behavior of just a few rheologic elements. One element is a spring that represents elastic behavior as shown in Fig. 10.2.16. Another element is a dashpot or shock absorber that represents viscous behavior described in Eq. 10.2.50. A third element is a slider that represents a yield point. These elements are combined in various one dimensional series and parallel arrangements. The three dimensional material law is then obtained by viewing the one dimensional result as being applicable to deviatoric relations. An example is the Bingham model shown in Fig. 10.2.17. The Bingham material is elastic until the yield point is reached; flow occurs beyond the yield point.

Unfortunately, most rocks that exhibit measurable viscous deformation also show nonlinearity with respect to viscosity. Creep of salt and potash are examples (Obert, 1964). Fig. 10.2.18 shows a model creep curve and three stages of creep. Creep is time-dependent deformation under constant load. Relaxation is

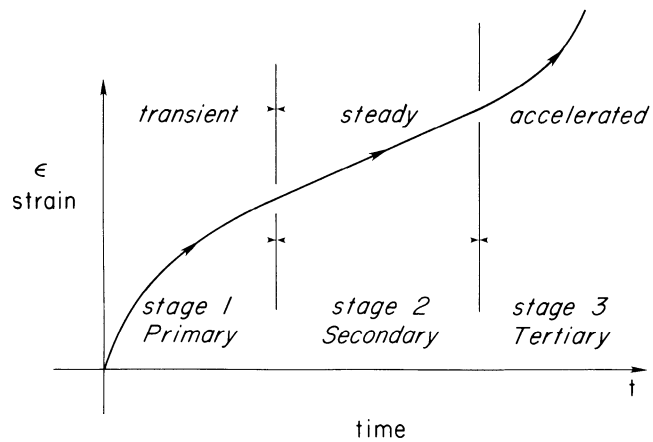

Fig. 10.2.18. Stages of creep.

time-dependent stress at constant strain. The first stage of creep shown in Fig. 10.2.18 is transient creep. The second stage is steady-state creep. Tertiary creep occurs prior to rupture. Elevated temperatures and pressures complicate the picture. A number of empirical relations have been proposed on the basis of experimental observations, but no generally accepted time-dependent material laws have been formulated. A simple form that fits much uniaxial steady state creep data is cited by Carter and Gnirk (1979):

$$(d) = A[\exp(-Q_c/RT)](s^n) \tag{10.2.53}$$

where A, Q_c, and R are constants, and T is the absolute temperature. The exponent n ranges between 1 and 9, but is usually between 3 and 6. Comparison of Eq. 10.2.53 and 10.2.50 suggests that the steady state viscosity is indeed stress dependent:

$$1/\mu = A\,[\exp(-Q_c/RT)](s^{n-1}). \tag{10.2.54}$$

Note that under uniaxial stress, say, in z-direction, deviations of stress and corresponding deformation rates are given by

$$s_{zz} = (2/3)\sigma_{zz}, \qquad d_{zz} = (2/3)(\partial v_z/\partial z) \tag{10.2.55}$$

In many cases, strain rates, for example, $(D\epsilon_{zz}/Dt)$ are used instead of deformation rates $(\partial v_z/\partial z)$.

10.2.4 SOLUTION APPROACHES

A proper combination of physical laws, kinematic relations, and material laws leads to a complete system that contains as many unknowns as equations, and therefore allows the motion of the rock mass to be estimated in advance of actual excavation. Excavation processes are equivalent to the application of load. Of course, there are other ways of loading a rock mass, for example, by the addition of heat. In the absence of inertia forces caused by wave propagation, the resulting system of differential equations is composed of the stress equations of equilibrium, the strain displacement equations, and the equations describing the material response. Integration of the system in the region of interest is generally subject to the specification of stresses and displacements at the boundary of the considered region and throughout the region at the beginning of the excavation processes being investigated. Analytical solutions obtained under a given set of specifications are functions that allow for the calculation of the displacements, strains, and stresses that would result if the excavation was carried out according to plan. Such solutions and associated calculations provide valuable design guidance in advance of actual excavation. However, analytic solutions to three dimensional problems in rock mechanics are generally not feasible. Even in two dimensional linear elasticity, analytic solutions, with few exceptions, are not easy to obtain.

Although there are relatively few analytic solutions available, they provide valuable insight concerning stress concentration about underground excavations such as tunnels and shafts and the bending and flexure of stratified ground. The results of analytic solutions to stress concentration problems and to beam and sheet bending analyses are immediately applicable to opening design based on safety factor criteria. Pillars between openings are amenable to analytic treatment based on average stresses. Approximate solutions to slope stability problems associated with surface mine excavations may be obtained from consideration of limiting equilibrium states. In this regard, the factor of safety is by definition the ratio of strength to stress. Of course, appropriate measures of strength and stress must be stated for the problem at hand. In all cases, numerical methods may be used for detailed study of stress distributions, strains, displacements, and safety factors.

10.2.4.1 Analytic Solutions

An example problem of great technological importance that demonstrates several interesting features of stress concentration about excavations in rock is that of determining the distribution of stress in a thick-walled cylinder subject to uniformly distributed radial pressures inside and outside the cylinder. As the outer radius of the cylinder becomes indefinitely large, the problem amounts to the determination of stress about a circular opening in an medium of indefinite extent. The latter is applicable to the analysis of unlined shaft wall safety, while the first is applicable to the analysis of shaft liners.

Another example problem amenable to analytic study and of considerable importance is beam analysis. In stratified ground where the immediate roof layers tend to separate from strata in the remote roof, analytic solutions are readily obtained from technical beam bending theory.

STRESS CONCENTRATION CONCEPT. Cylindrical coordinates are a natural choice for the analysis of circular openings as shown in Fig. 10.2.19. The opening is considered long in the sense that variations with z are negligible, so that derivatives with respect to z are zero. In particular, the z direction strains are zero, and the analysis is plane strain.

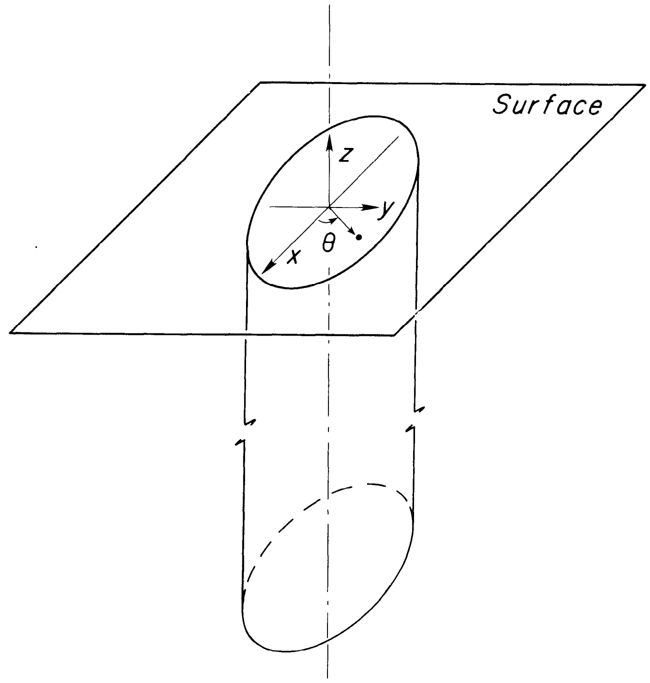

Fig. 10.2.19. Circular shaft analysis.

Use of Hooke's law for linearly elastic and isotropic rock in the stress equations of equilibrium together with the strain displacement relations and the axial symmetry of the problem leads to the solution for the radial displacement (Obert and Duvall, 1967):

$$u(r) = Ar + B/r \tag{10.2.56}$$

where the constants of integration A and B are determined from the applied loads.

The shaft in Fig. 10.2.19 is excavated in initially stressed ground, and the displacement given by Eq. 10.2.56 is the radial displacement of the shaft wall induced by excavation. The induced displacement decreases away from the shaft, so that as r becomes indefinitely large, A must be zero.

At the shaft wall, the change in radial stress caused by excavation must be equal but opposite to the preexcavation stress. The stresses about the shaft after excavation are simply the preexcavation stresses plus the stress changes. Hence

$$\begin{aligned} \sigma_{rr} &= (\sigma_h^0)[1 - (b/r)^2] \\ \sigma_{\Theta\Theta} &= (\sigma_h^0)]1 + (b/r)^2] \\ \sigma_{zz} &= \sigma_v^0 \\ \sigma_v^0 &= \gamma h \\ \sigma_h^0 &= K_0\sigma_v^0 \end{aligned} \tag{10.2.57}$$

where σ_{rr}, $\sigma_{\Theta\Theta}$, σ_{zz} are the post-excavation stresses in the radial, circumferential, and vertical directions, respectively; σ_h^0 and σ_v^0 are preexcavation horizontal and vertical stresses related by the constant K_0; h is depth; g is specific weight of rock; and b is the unlined shaft radius.

Stress concentration K is by definition the ratio of the actual stress $\sigma_{\Theta\Theta}$ to a reference stress. The reference stress is generally

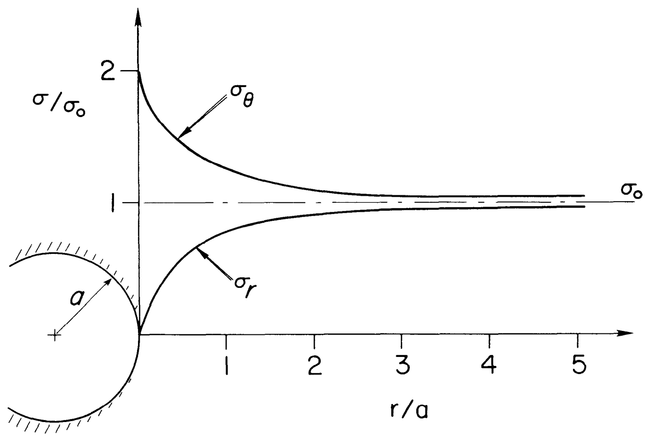

Fig. 10.2.20. Stress concentration around a circular hole.

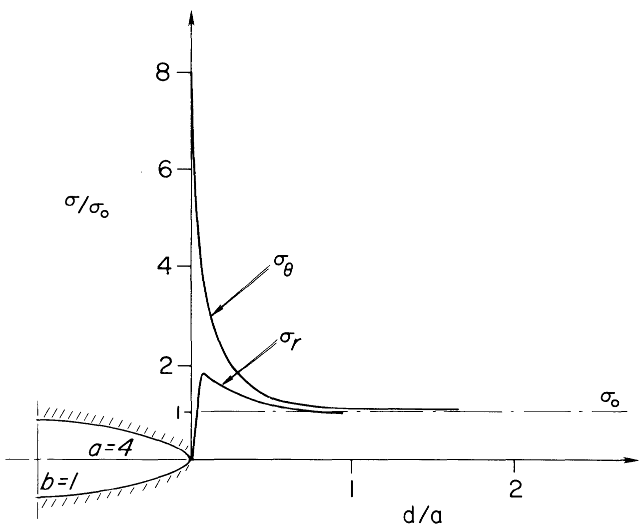

Fig. 10.2.21. Stress concentration around an elliptical hole.

the major principal stress before excavation σ_h^0. A plot of stress concentration vs. distance from the shaft wall is shown in Fig. 10.2.20. Inspection of Eq. 10.2.57 and Fig. 10.2.20 shows that the greatest concentration of stress in the horizontal direction occurs at the shaft wall. The compression at the shaft wall is double the preexcavation stress.

Eq. 10.2.57 and Fig. 10.2.20 demonstrate an important feature of stress concentration in two dimensional analyses and that is its rapid decrease with distance from the excavation wall. The stress concentration is within 4% of the preexcavation stress at a distance of two shaft diameters away from the shaft wall and within about 11% at a distance of only one shaft diameter. A rough rule of thumb is that excavations more than one-diameter (1D) apart do not interact to a significant degree.

In this regard, shaft diameter is the characteristic linear dimension of the excavation in the example analysis. Excavations of other shapes are characterized by the *greatest* dimension of the opening for the 1D rule of thumb. Fig. 10.2.21 shows the decrease of stress concentration with distance from the end of an elliptical excavation that has a ratio of major to minor axis of four and subject to the same initial state of stress as the circular shaft. Again the stress concentration decreases rapidly with distance from the excavation wall, and at about a distance equal to the length of the *major* axis of the excavation, the circumferential stress is within 10% of the preexcavation stress. The characteristic linear dimension of the opening in this case is indeed the greatest dimension of the opening.

The zone of significant stress concentration is the zone of influence of the excavation. An envelope extending a characteristic dimension from the excavation wall into the adjacent rock defines a zone of influence of the opening. The rock mass beyond the zone of influence is relatively unaffected by the excavation. However, if the wall of the opening is stressed to the elastic limit, plastic deformation may occur with the development of a yield zone in the vicinity of the opening. The effect of yield zone development is to reduce the peak stresses that would occur in a purely elastic response and to raise the average stress in the region of yielding. Beyond the yield zone, the rock is elastic where again the stress concentration decreases rapidly with distance (away from the elastic-plastic boundary).

Another important rule of thumb that arises from analysis of stress and the study of stress concentration around shaft, and tunnel-like excavations is that most favorable orientation of a single opening is with the long axis of the opening aligned parallel with the major compression. Such an orientation exposes the long dimension of the opening to the minimum preexcavation stress. A similar rule is appropriate to three dimensional, stope-like openings. Stated another way, the rule is as follows: orient the smallest dimension of the excavation perpendicular to the greatest compression, the largest dimension perpendicular to the least compression, and necessarily the intermediate dimension perpendicular to the intermediate preexcavation principal stress. This rule minimizes the exposure of a stope to stress concentration. Fig. 10.2.22 illustrates the most favorable opening orientation rule of thumb.

In the case of a row of openings of similar size and shape, the row axis should be aligned parallel to the major compression. The least compression is then perpendicular to the row axis. Spacing of the openings according to the zone of influence rule would require the pillars between the openings to be as wide or wider than the openings themselves, as shown in Fig. 10.2.22.

SHEET AND BEAM BENDING. Consideration of the equations of equilibrium, Hooke's law, and the geometry of strain under the assumption that sections perpendicular to the neutral surface of a sheet remain plane while deforming into a cylindrical shape leads to the equations of technical beam and sheet bending theory (Timoshenko and Woinowsky-Geiger, 1959). The two equations of most importance are the flexure formula,

$$\sigma_{max} = Mc/I, \tag{10.2.58}$$

where σ_{max} is peak stress in the sheet or beam, M is maximum moment, c is distance from the neutral axis to outermost fiber, and I second moment of area about the neutral axis; and the Euler-Bernoulli bending formula,

$$d^2w/dz^2 = M/EI \tag{10.2.59}$$

where $w(x,z)$ is displacement of the beam from its unloaded position, and E is Young's modulus. In the case of a sheet, E is replaced by $E/(1 - \nu^2)$, where ν is Poisson's ratio. The difference between sheet and beam bending in most cases is therefore small, about 4%. Mine roof strata are rectangular in cross section, and thus $I = bH^3/12$, where b is the breadth of the section and h is the stratum thickness.

Analytic solutions to the problem of a beam loaded on its upper surface by a uniformly distributed pressure p with built-

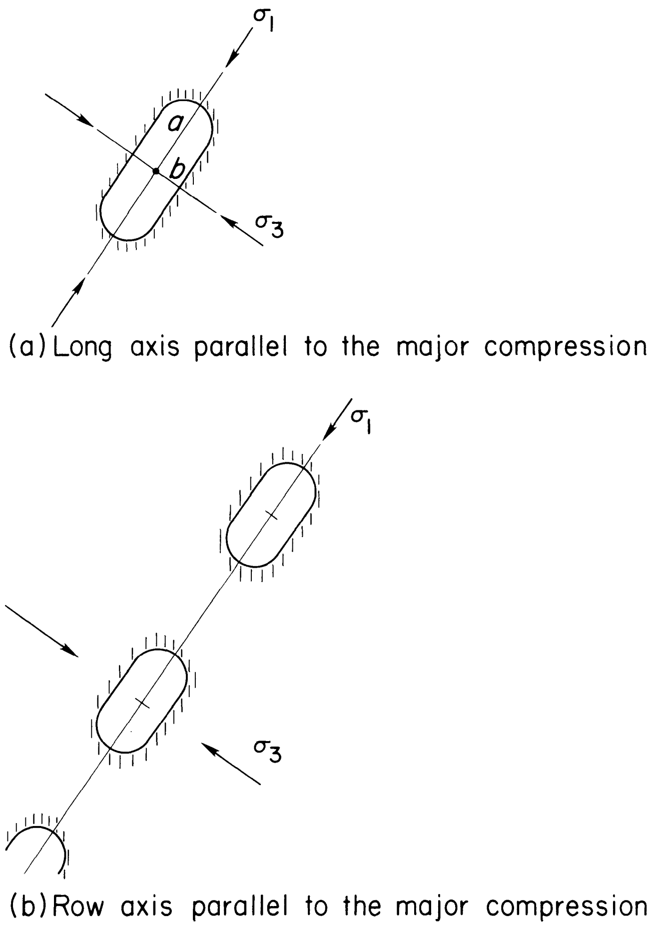

Fig. 10.2.22. Favorable orientation of holes and rows of holes (a) Long axis parallel to the major compression, (b) Row axis parallel to the major compression.

Table 10.2.2. Beam Analysis Results

Case	Max. Tension	Max. Sag
Simply supported	$(3/4)p(L/h)^2$ at center span beam bottom	$[5p(L/h)^2]/[32E]$ at center span
Built-in ends	$(1/2)p(L/h)^2$ at beam ends beam top	$[p(L/h)^2]/[32E]$ at center span

in ends and simply supported ends are given in Table 10.2.2, including the maximum tension and sag. The two cases in the table are extremes; all other cases that allow some rotation of the beam ends lead to stresses and displacements that are in between the simply supported and built-in end cases. Inspection of the results in Table 10.2.2 shows that the built-in end case is optimistic in the sense that the peak stress in the roof beam is less than in the simply supported case. The peak shear stresses are the same in both cases; the sag in the simply supported case is five times the sag in the built-in end case.

In some instances, several roof strata may separate in a cluster. The sag is the same for each stratum within a cluster,

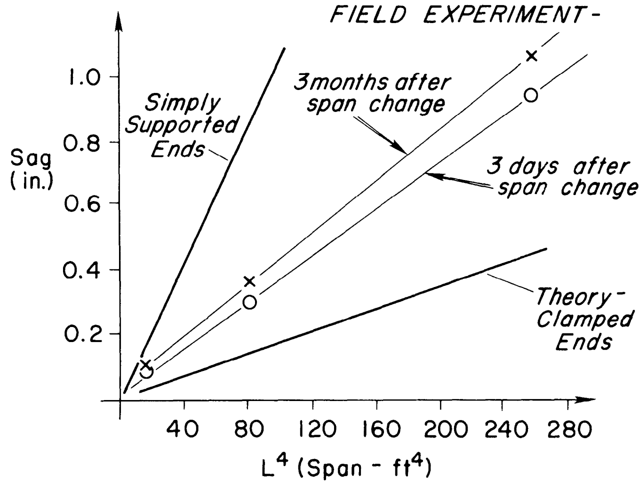

Fig. 10.2.23. Comparison of measured roof sag with beam theory (after Merrill, 1957). Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

and the net "pressure" $p(i,n)$ acting on the ith layer in a cluster of n layers is given by

$$p(i,n) = \{(E_i h_i)[p_g - p_b + \textstyle\sum(\gamma_j h_j)]\}/[\textstyle\sum(E_j h_j^3)] \tag{10.2.60}$$

where E_i, h_i, g, are Young's modulus, thickness, and unit weight of the ith roof layer, respectively; p_g and p_b are gas and bolting pressures; and summation is from the first to the nth roof layer in the separated cluster. The gas (or water) pressure term is included only if gas or water pressure develops at the separation horizon. Eq. 10.2.60 can be used to determine where bed separation may occur by a sequential computation of $p(1,n)$ that load on the first roof layer above the opening. As n is increased, $p(1,n)$ will increase up to the separation horizon where $p(1, n + 1)$ will be found to be less than $p(1,n)$. Separation occurs between the roof layer n and $n + 1$ counted up from the first roof layer.

Results of a full-scale test (Merrill, 1957) of the applicability of beam theory to mine roofs is shown in comparison with theory in Fig. 10.2.23. The measured data fall between the simply supported and built-in end cases, as might be expected in actual practice. In this regard, an essential feature of the analysis is the occurrence of bed separation. If bed separation occurs in the immediate roof, then beam analysis is applicable. The immediate roof strata are the layers within the zone of influence of the excavation. In this regard, a roof beam should have a span to thickness ratio greater than five and preferably eight to insure applicability of beam theory. Beam theory also finds application to cantilevered strata over longwall support.

PILLARS. Rock left between multiple openings forms pillars. There are many different types of pillars in hard-rock and soft-rock mines, for example, crown pillars, sill pillars, shaft pillars, rib pillars, chain pillars, and barrier pillars. Where pillars are numerous and occur in some repetitive pattern in a flat tabular ore body, a rigorous calculation of the average vertical stress in a typical pillar is possible. The calculation is based on the tributary area concept (Obert, Duvall, and Merrill, 1960). The tributary area A is simply the area defined by half the distance to the adjacent pillar. The ratio of the area mined A_m to the tributary area is the extraction ratio R, and the method is also known as extraction ratio pillar design. Excavation transfers to the pillar the weight of the overburden in an imaginary prism extending from seam to surface. Thus the force on the pillar after mining is equal to the overburden load, and the average vertical stress in the pillar is given by the formulas,

$$\text{(a) } S_p = S_v/(1 - R), \quad \text{(b) } R = A_m/(A_m + A_p) \qquad (10.2.61)$$

where $S_v = gH$, g is unit weight of overburden, H is seam depth, and R is extraction ratio.

In dipping seams, the average normal and shear stresses in the pillar are given by similar extraction ratio formulas (Pariseau, 1982):

$$\text{(a) } S_p = S_n/(1 - R), \quad \text{(b) } T_p = T_n/(1 - R) \qquad (10.2.62)$$

where S_n and T_n are the preexcavation normal and shear stresses, respectively, and are obtained through a rotation of the reference axes from the horizontal and vertical to directions parallel and perpendicular to the seam. Eq. 10.2.62 is valid at all seam dips. Pillar stresses S_p and T_p are the post-excavation stresses.

Where progressive failure of pillars needs to be taken into account in the design process, for example, in the design of yielding pillars, then a detailed analysis of the evolution of the stress distribution about the openings and in the pillars as time passes and mining progresses as necessary.

SLOPES. Surface excavation in rock masses generates slopes that must remain stable for safety of operations (see Chapter 10.4). Failure of slopes in rock masses almost always results from an unfavorable combination of slope height, slope angle, and geologic structure (Hoek and Bray, 1977). Water seepage is often an important contributing factor. From the kinematic viewpoint, there are four basic types of rock slope failures: (1) planar block slides, (2) wedge failures, (3) rotational, and (4) toppling, as shown in Fig. 10.2.24. The planar block slide is the simplest and serves as a conceptual model for more complicated failures.

With reference to Fig. 10.2.25, the downhill forces D tend to drive the slide mass, while the uphill forces R resist the motion. According to Newton's second law,

$$F = D - R = (W/g)a \qquad (10.2.63)$$

where W is slide weight, g is acceleration of gravity, and a is downhill acceleration. Since the slide mass is initially at rest, negative accelerations imply stability. Resisting forces arise from cohesion and friction on the slide failure surface and reinforcement, say, from pretensioned cable bolts. Frictional resistance depends on the effective or net normal force that is calculated from the requirement for equilibrium in the direction normal to the potential failure surface. The water force P is the integral of water pressure over the wetted portion of the failure surface. Provision is also made for seismic load S in Fig. 10.2.25, usually calculated as the product of slide mass (W/g) and a horizontal acceleration equal to a small fraction of g. For example, in a high-risk seismic zone, an acceleration of $0.15g$ may be dictated, so that $S = 0.15W$. Bolting forces in surface installations may range to over 1.8 MN (400,000 lb/ft) per bolt hole, but are generally quite small in relation to other forces acting on the slide mass (Seegmiller, 1975). An interesting feature of Eq. 10.2.63 is that it also describes the motion of the mass center of the slide mass even though the mass center is a point in space rather than a mass point. Mass center motion calculations are important for estimating dynamic features of rock slides, including mass center velocity and distance traveled. Such calculations are germane to practical questions such as zoning against geologic hazards.

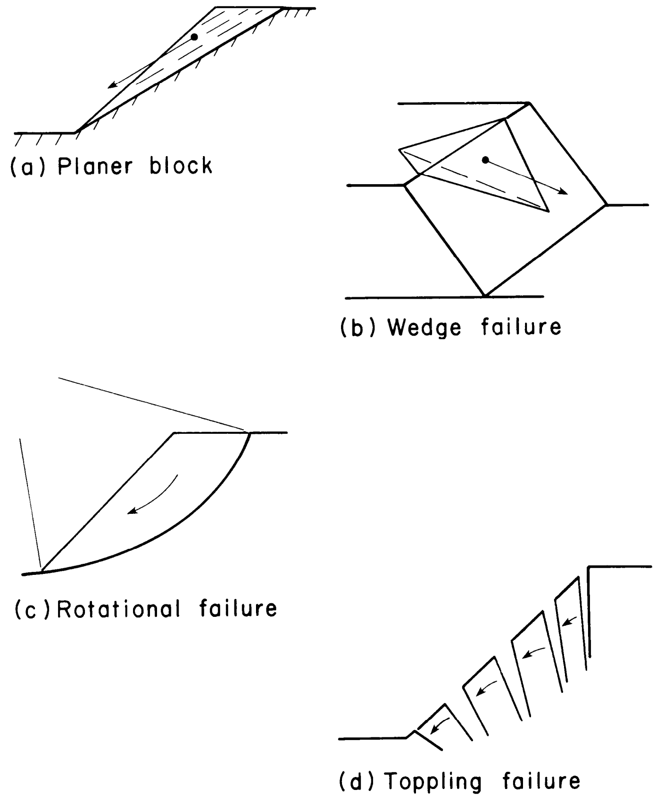

Fig. 10.2.24. Basic types of rock slope failures. (a) Planar block, (b) wedge failure, (c) rotational failure, (d) toppling failure.

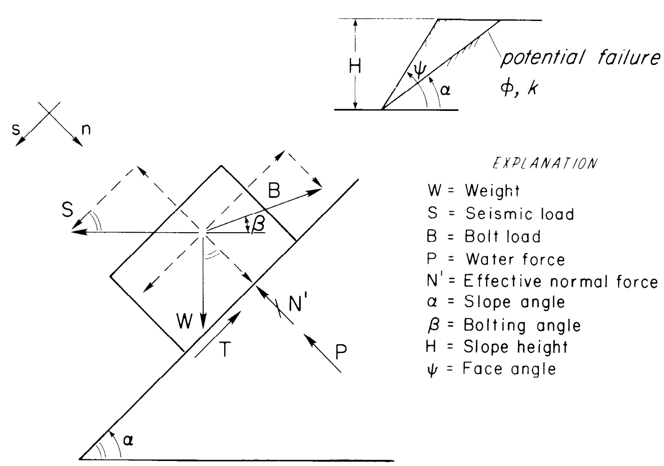

Fig. 10.2.25. Planar block slide model.

10.2.4.2 Factor of Safety Concept

The safety factor concept is a practical design criterion that has been in engineering use for many years. A global safety factor is used in design against collapse of a structure as a whole such as an open pit mine slope. A local factor of safety is more appropriate to elastic design of underground openings.

SAFETY FACTOR AGAINST COLLAPSE. The most commonly used safety factor FS in slope stability analysis is a ratio of resisting forces R to driving forces D:

$$FS = R/D \tag{10.2.64}$$

Safety and stability is indicated by $FS > 1$. If $FS < 1$, then the design is unsatisfactory—slope failure would occur if excavated accordingly to plan. Only the downhill forces enter into D; uphill forces are included in R. Negative resisting and driving forces are inappropriate concepts; R and D are strictly sums.

Eq. 10.2.64 is entirely equivalent to the negative acceleration criterion described previously as can be seen by substitution of Eq. 10.2.64 into Eq. 10.2.63. Thus

$$D(1 - FS) = (W/g)a \tag{10.2.65}$$

shows that stability is indeed achieved with $FS > 1$. Safety factors near one are common in open pit mines, but higher safety factors are needed for special cuts and embankments.

In the case of rotational failure, the global factor of safety is the ratio of resisting to driving moments. Thus

$$FS = (M_R/M_D) \tag{10.2.66}$$

where M_R and M_D are sums of the resisting and driving moments, respectively. When the potential failure surface is curved, then strictly speaking, the forces and moments must be obtained by integration of the stresses over the failure surface and throughout the slide mass volume. A detailed analysis of stress is required for such calculations. However, there are a number of approximate methods in use, many in computer code form, that allow for slope stability analysis and global safety factor estimation without the need for a rigorous stress analysis.

SAFETY FACTOR AGAINST LOCAL YIELDING. By definition, the local safety factor FS = "strength"/"stress." As a ratio of strength to stress, the local safety factor is a measure of the nearness of the stress state at a point to the elastic limit there.

In the problem of stress concentration at the wall of an excavation, in roof beam analysis, and in pillar stress calculation by the tributary area method, the stress state is essentially one dimensional. The appropriate measure of strength is then the unconfined compressive strength for compressive stress and tensile strength for tensile stress. The factors of safety in (a) compression, (b) tension, and (c) shear are

$$\text{(a) } FS_c = C_0/\sigma_c, \quad \text{(b) } FS_t = T_0/\sigma_t, \quad \text{(c) } FS_s = T_s/\sigma_s \tag{10.2.67}$$

where T_s and σ_s are shear strength and stress, respectively.

In the problem of stress concentration, the peak (a) compressive and (b) tensile stress are given by

$$\text{(a) } \sigma_c = K_c\sigma_1^0, \qquad \text{(b) } \sigma_t = K_t\sigma_1^0 \tag{10.2.68}$$

where K_c is peak compressive stress concentration factor, and K_t is peak tensile stress concentration factor.

In roof beam analysis, the peak tension for use in Eq. 10.2.67(b) is obtained from Table 10.2.2 in the case of a single, separated roof beam and from Eq. 10.2.60 for a multi-layered roof that has separated from the strata above.

In tributary area pillar design, the peak compression for use in Eq. 10.2.67(a) is obtained from the extraction ratio formula Eq. 10.2.62(a), while the peak shear stress is obtained from Eq. 10.2.62(b). The shear strength T_s is often calculated from the Mohr-Coulomb criterion.

10.2.4.3 Numerical Solutions

Declining costs of hardware and improved software for implementing advances in numerical solution techniques have made very powerful stress analysis tools available at the desk top of rock mechanics engineers. Where the complexities of excavation geometry, geology, and material behavior preclude analytic solutions, design guidance can often be obtained with the help of these new tools. The most important numerical methods are

1. Finite difference (FD).
2. Finite element (FE).
3. Boundary element (BE).
4. Distinct element (DE).

There are advantages and disadvantages associated with each, although in the hands of the sophisticated user, the distinctions tend to blur.

The finite difference method is a traditional approach to numerical solution of differential equations. The method is well developed and is especially suited for solution to problems in homogeneous rectangular domains that do not contain holes. Wave propagation analysis, temperature distribution calculations, and quasi static analysis of groundwater seepage are examples of problems often studied by finite difference methods. An advantage of finite difference methods is the large accumulation of knowledge about such techniques. A disadvantage is the difficulty often encountered in handling boundary conditions for regions containing holes of irregular shape. For this reason, the method is not popular for stress analysis of excavations in rock.

The finite element method (Cook, 1974; Zienkiewicz, 1977) is based on a spatial subdivision of the region or rock mass of interest. The method is well suited to analysis of irregularly shaped openings and is perhaps the most popular method in rock mechanics. It is relatively easy to use, and a number of engineering colleges now offer introductory finite element courses at the undergraduate level. The technique allows for a variety of rock types and material laws to be used in the same analysis and also allows for sequential excavation and filling of surface and underground excavations. The method has great versatility and has been used in the analysis of stress, temperature, and groundwater flows about mines. Analysis of two dimensional problems is almost routine. Three dimensional problems represent a sizable increase in computing effort; although feasible in principle, they are far from routine.

Boundary element methods are based on discretization of the bounding surface of the rock mass of interest (Crouch and Starfield, 1983; Banerjee and Butterfield, 1981). They are similar to finite element methods in some respects and have the advantage of reducing the dimensionality of the problem, say, from three (volume) to two (surface) dimensions. However, the apparent advantage of reduced dimensionality does not necessarily translate into reduced computer size and time requirements. The real advantage of boundary element methods relative to finite element methods is in the preparation of input data, especially in three dimensional analysis where volume discretization is a major task. An important disadvantage of the boundary element method is that it is difficult to apply to the analysis of other than linear, homogeneous, isotropic elastic rock masses.

The distinct element technique was originally developed by Cundall and treats rock masses as assemblages of rock blocks (Cundall, 1971). The emphasis in the distinct element technique is on the important role that rock mass discontinuities such as joints play in the mechanics of field-scale rock masses. In the distinct element method, individual blocks are defined by joint sets that are usually assumed to be continuous and persistent throughout the rock mass. Blocks may be rigid or deformable. A major advantage to the distinct element technique is the capability for following large displacements of rock blocks on joints. Applications range from conventional slope stability analysis, fault slip stability in an underground mine, and bin flow analysis, to throwout calculations associated with blasting of craters, to blast casting of overburden in a strip mine. An important disadvantage to the distinct element technique is the rather large amount of computer storage and time needed for problem analysis.

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Chapter 10.3
GEOMECHANICS INSTRUMENTATION

W.A. HUSTRULID AND B.P. BOISEN

10.3.1 INTRODUCTION

Two geomechanics *instruments* used by miners throughout time to assess the state of their mine are their eyes and their ears. For many applications, these are still the most important and appropriate. These sensors have the unique feature of being able to scan a wide horizon and also to focus on a single place or event. They are connected to a powerful data acquisition system that provides interpretation and issues instructions based on that interpretation. However, just as one would not think of running a modern mill without a battery of instruments to provide quantitative data regarding the state of the process, the same is true—albeit to a lesser level—in a mining operation. A number of devices have been developed throughout the years that can be used to provide quantitative information regarding rock/soil behavior and/or the response of the mine structure. In this chapter, the major types of instruments for underground and surface applications are discussed.

Call (1982) has provided some very useful guidelines for the selection of geomechanics instrumentation. Although they deal specifically with the monitoring of slopes in open pits, they have general applicability. His general guidelines for decision making are

1. Measure the obvious things first. Surface displacement is the most direct and most critical aspect of slope instability.
2. Simpler is better. The reliability of a series system is the product of the reliability of the individual components. A complex electronic or mechanical device with a telemetered output to a computer has significantly less chance of being in operation when needed than do two stakes and a tape measure.
3. Precision costs money. The cost of a measuring device is often a power function of the level of precision. Measuring to ½ in. (10 mm) is inexpensive compared to measuring to 4×10^{-5} in. (0.001 mm). A micrometer is unnecessary for monitoring slope movement that has a velocity of 2 in. (50 mm)/day.
4. Redundancy is required. No single device or technique tells the complete story. A single extensometer or survey point cannot indicate the area involved in the instability, and, if it is destroyed, the continuity of the record is lost.
5. Timely reporting is essential. Data collection and analysis must be rapid enough to provide information in time to make decisions. Reducing last week's survey data and telling the mine superintendent that the slope was moving Thursday when a shovel was buried Sunday does not lead to pay raises!

The questions that must be asked—in sequence—in any instrumentation program are these:

1. Why measure?
2. What measure?
3. Where measure?
4. How measure?

If the question, "Why measure?", cannot be answered satisfactorily; then there is no reason to continue through the list. Some typical instrumentation uses include:

1. Determination of rock and soil mass properties such as strength, deformability, anisotropy, and resistance to some types of alteration.
2. Observation of the state of stress in the masses, including in situ stress and redistributed stress.
3. Observation of the response of the rock or soil masses due to disturbances induced by the excavation and operation of the structure.
4. Measurement of the reaction of the mine structure to conditions imposed by the geologic mass, and to conditions during subsequent operation.
5. Observation of the behavior of associated or adjacent structures or services either affected by or at risk as a result of mining.
6. Recognition of hazards, either actual or incipient.
7. Identification of remedial measures, i.e., reinforcement.
8. Verification of the effectiveness or ineffectiveness of remedial measures.
9. Evaluation of safety factors.
10. Short- and long-term monitoring, with alarm capability as required.

Regardless of the application, all instruments have certain requirements in common (Dutro, 1986).

1. *Range:* The provision of adequate range is a consideration often overlooked in choosing instruments for a specific application. On the other hand, excessive range should also be avoided, as it often can be obtained only at the expense of some other desirable characteristic, such as resolution.
2. *Resolution*: Resolution denotes the smallest numerical change an instrument can discriminate, but not necessarily repeat. Adequate resolution is important in identifying low-level phenomena, particularly those indicative of hazards. Excessive resolution, however, should also be avoided, as it is usually attainable only at the expense of other characteristics, and as it may lead to a degree of scatter in the data that obscures other trends and relationships.
3. *Repeatability*: Repeatability is a measure of the least reading which can be repeated consistently. Repeatability and resolution, together with range, indicate the general suitability of a particular instrument for a specific application.
4. *Accuracy*: Absolute accuracy is a difficult requirement to adequately define. All instruments are accurate under some conditions, but no instruments are likely to be absolutely accurate under all conditions. It is important to understand the advantages and limitations of the instruments proposed, and to relate these to the requirements of the proposed measurements. Accuracy is expensive and preoccupation with excessive accuracy may divert attention from other instrument characteristics and requirements. In a majority of instrument applications, the data obtained relate to a component of force, deflection or displacement, and not a resultant. Unless the other vector quantities are known with the same precision required of the instruments, the accuracy of the instruments is not at issue. Instrument accuracy should, therefore, be specified in terms of the probable overall precision of the entire application.
5. *Survivability*: The adversity of mining environments is usually underestimated. Instrument failures are not only expensive and inconvenient, but they typically result in lost data that are almost always irretrievable. Mining instruments must, therefore, have the highest order of survivability, if acceptable levels of operation and data continuity are to be maintained.

Many instrumentation programs ultimately are unsuccessful because the information was not acquired, processed, and transmitted in a timely manner. Readout intervals, levels of processing and interpretation, and chains of authority for the transmittal of data should be clearly established at the outset of programs. The usual tendency is to overdo at the outset, with too short an interval between readouts, too much data, and too elaborate initial processing and interpretation. Consequently, data do not become available to the appropriate personnel in time to be really useful.

When dealing with instrumentation data, several points should be kept in mind: (1) Significant measurement trends, particularly adverse trends, are generally evident in the raw data, without any processing. Readout personnel should scan all data as they are acquired, and take whatever action is indicated without awaiting further processing. (2) If potential hazards to life and property are involved, "timely" means right now! (3) In many situations and almost invariably in hazard situations, the appropriate personnel are the lowest level of authority for the necessary action, or the individual or individuals in danger.

Some of the advantages/disadvantages of automatic data acquisition and processing vs. manual readout are implicit in the foregoing comments. Manual instrument readout is costly and time consuming. On the other hand, a person is present, and that person can be trained to scan the data, recognize significant trends or deviations, and take the indicated action. The necessary skill can be readily developed by mine personnel. Automatic and semiautomatic readout apparatus are expensive, as is the extra cabling usually necessary to route the signals to common points. The systems, however, provide data at more frequent intervals and at lower unit readout cost, so the cost factors tend to balance out, particularly on programs of considerable duration. The ideal readout plan is an automatic system with an alarm capability, supplemented by periodic manual readout and data scan by qualified personnel. This would also relieve a persistent problem of automatic systems, whereby alarm thresholds are either set too low, resulting in annoying false alarms, or too high, resulting in increased hazard exposure.

There are a variety of costs involved in a geomechanics instrumentation program:

1. Hardware.
2. Installation.
3. Readout.
4. Data processing.
5. Interpretation.

Sometimes, there are also costs or at least lost savings by not having such a program. Presuming that such a cost-benefit analysis has been done and the decision has been made to measure, then the tendency is to try and minimize cost. The most obvious way is to economize on the hardware. Although "homemade" will sometimes suffice, there are many examples where this decision has turned out to be penny wise but pound foolish. The costs add up very quickly when trying to interpret "suspect" data. *Quality* hardware—whether homemade or "store bought"—is a good investment. Furthermore, quality installation is also a must. The acquisition of poor data is sometimes worse than having no data at all.

10.3.2 INSTRUMENTATION FOR SURFACE MINING APPLICATIONS

10.3.2.1 Introduction

The use of rock mechanics instrumentation in open pit mining can in general be placed in one of the following three categories:

1. Initial slope design.
2. Slope design assessment.
3. Slope monitoring.

Monitoring the condition of critical or suspect slopes in open pit mines is the most obvious and currently widespread use of rock mechanics instrumentation in open pit mines. Much of the available instrumentation, as a result, deals with this aspect. However, because of the large effect slope angle has both on short- and long-term pit economics, increasing emphasis is being placed upon the development of techniques upon which recommendations for optimum bench, inter-ramp, and overall slope angles can be based. Some instruments and instrumentation techniques for assisting in this assessment both prior to and during mining are available. The discussion in this segment will be organized along these lines of mining application.

10.3.2.2 Initial Slope Design

Generally, a major part of the evaluation of a mineral property is carried out through a development drilling program. Diamond drilling yielding core is common, although rotary drilling is used as well. The type of information that is sought according to Waterman and Hazen (1968) is

1. Geology of the mineralized zone.
2. Quantitative data on grade and tons.
3. Physical size and shape of the deposit.
4. Mineralogical and metallurgical characteristics of the ore.
5. Physical characteristics of the ore.
6. Data on other factors that could affect mining operations, such as groundwater, ground conditions, etc.

Unfortunately, the last category is sometimes added as more or less an afterthought. These factors, however, are extremely important in the design of today's surface mining operations. Maximum slope angles as determined by structural characteristics and groundwater have a very large impact on mining economics. A change in slope angle by 1° can mean millions of dollars in stripping cost (Stewart and Kennedy, 1971). Fig. 10.3.1 is a plan view of a large open pit in the design stage. As can be seen, there are 10 sectors in which the slope angle has been prescribed.

To be able to make such a detailed design requires a significant amount of information regarding rock structure, rock mass properties, and groundwater. Therefore, consideration must be given as to how to extract such information during the development and subsequent drilling phases of a property.

Core Orientation: The orientation of the structures that will make up the walls of a pit are important for design. Figs. 10.3.2 and 10.3.3 show one such device, the Craelius core orienter (CCO), that can be used (Seegmiller, 1979).

Originally, the device was fitted to a core barrel which in turn was attached to the drill string. The string is lowered down the hole without rotation. The pins make a multipoint impression of the bottom configuration. A marking ball dents a soft steel plate thereby indicating the vertical plane. The CCO slides up in the core barrel, undisturbed as drilling proceeds. Drilling continues as long as there is space in the core barrel to take both the CCO and the core. The string must then be removed from the hole to extract the oriented core. Seegmiller (1990) has modified the device to work in wireline systems.

The orienter can be used, in dipping, in upwards inclined, and in horizontal drillholes without having to orient the drill string. It can be used in vertical holes after the drill string has been oriented. Instructions for using the orienter and logging oriented core have been given by Seegmiller (1976).

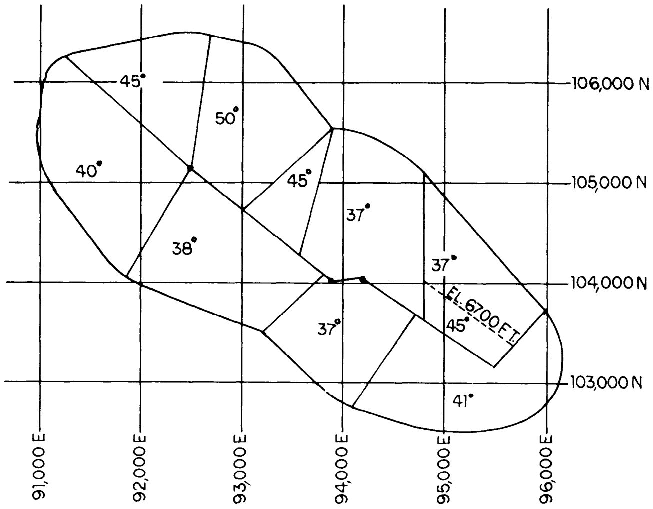


Fig. 10.3.1. Plan view of open pit showing slope sectors (after Crawford and Davey, 1979).

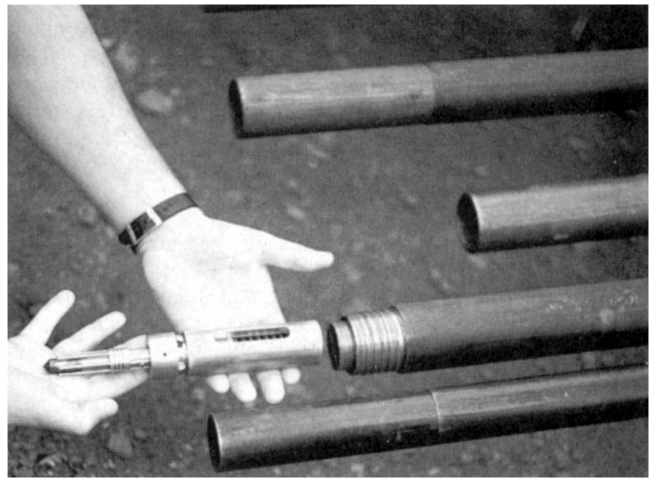


Fig. 10.3.3. CCO wireline orienter (*courtesy:* Seegmiller International).

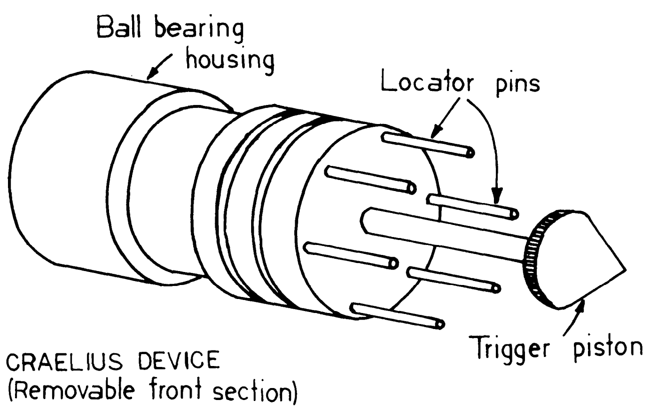


Fig. 10.3.2. Isometric drawing of Craelius core orienter (after Seegmiller, 1990).

The CORO, another core-orienting device operating along somewhat similar principles to the CCO, has been developed in Australia and Chile (Seegmiller, 1990).

Call et al. (1982a) have described another system for obtaining oriented core.

Borehole Television Cameras: The development of small-diameter, high-resolution television cameras over the past few years have made them into practical tools for viewing borehole walls. By observing the same structure at three known angular orientations in the borehole, the strike and dip can be calculated. The resolution is quite good even in water-filled holes.

Borehole television cameras are commercially available for lowering on a cable into vertical or near-vertical holes up to 1000 ft (300 m) deep. The smallest camera from one manufacturer (Fig. 10.3.4) has a diameter of 1 l/8 in. (29 mm) and can be used in boreholes with a length up to 500 ft (150 m). The image is transmitted through the lowering cable to the surface where it can be viewed and recorded. Various techniques are used for positioning purposes. Some systems use a rotating mirror or prism when viewing the side wall. In others, the camera is rotated (Stenning, 1990). Television borehole cameras producing both color and black and white images are available.

Borehole Seismic Tools: *Seismic Surveys*—Seismic surveys can be made in drillholes using commercially available equipment (Anon, 1977a). Because the strength of rock is roughly proportional to the sonic velocity, velocity surveys can provide estimates of the in situ strength of a slope wall or changes in the strength with time. In general, the higher the seismic velocity, the stronger the rock mass and vice versa. Large defects such as faults, massive discontinuities, or other large anomalies can be located through geophysical seismic techniques (Anon., 1988; Elkington et al., 1982). Fig. 10.3.5 shows the results of a series of different logs run in a borehole at a coal property (Kowalski and Fertl, 1976). Radar has recently been used to help define rock conditions (Holloway et al., 1990; Pittman et al., 1984; Anon., 1990l).

Borehole Radar—The borehole radar system (RAMAC), shown in Fig. 10.3.6, has recently been developed by the Swedish Geological Co. (Anon.,1990l) and applied to fracture detection in crystalline rock.

The system works in principle in the following manner: short current pulse is fed to a transmitter antenna, which generates a radar pulse. The pulse, made as short as possible to obtain high resolution, propagates through the rock. It is received by the same type of antenna, amplified, and registered as a function of time. From the full wave record of the signal, the distance (travel time) to a reflector, the strength of the reflection, and the attenuation and delay of the direct wave between transmitter and receiver may be deduced.

Optical fibers are used for transmission of the trigger signals from the computer to the borehole probes and for transmission of data from the receiver to the control unit. The advantages with optical fibers are (1) they have no electrical conductivity and will not support waves propagating along the borehole, and (2) they cannot pick up electrical noise. Since the signal is digitized down-hole there will be no deterioration of the signal along the cable. The quality of the results will thus be independent of cable length.

Borehole radar measurements can be performed as single-hole reflection measurements and/or cross-hole measurements. In single-hole reflection measurements, the transmitter and re-

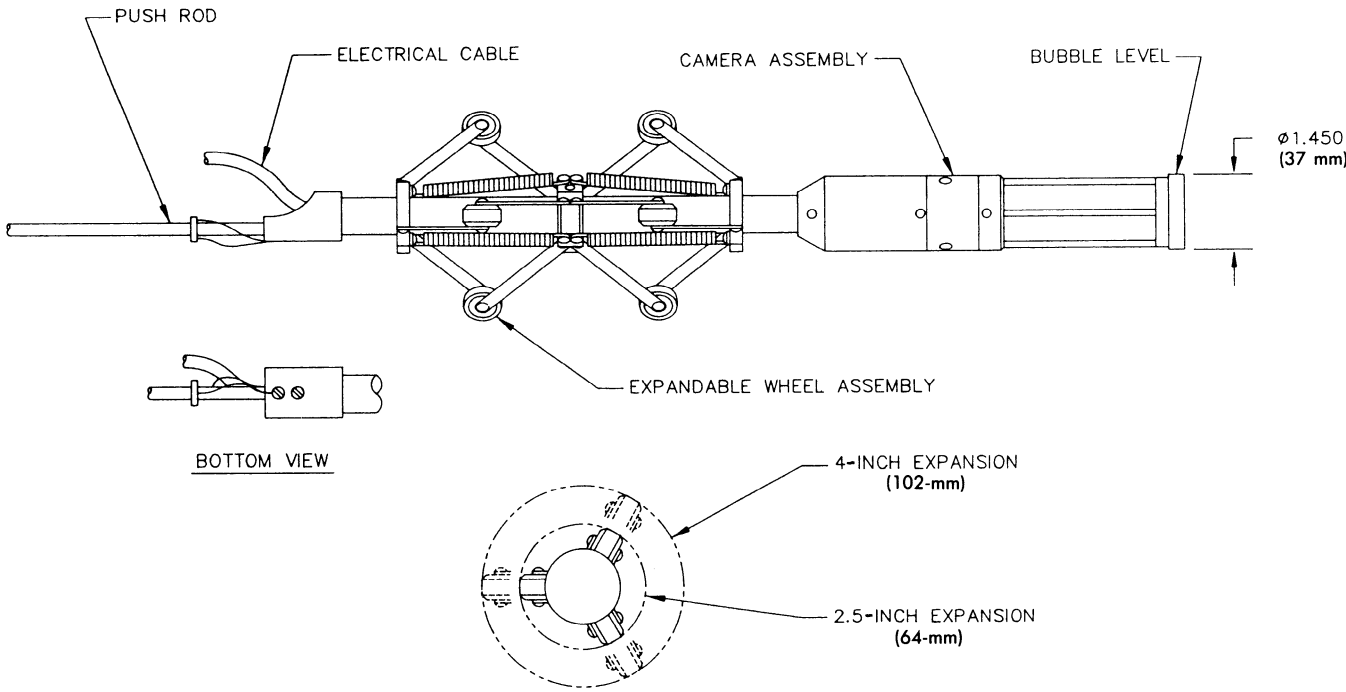


Fig. 10.3.4. Diagrammatic representation of small-diameter borehole television camera with guide arrangement (*courtesy:* Stenning Instruments).

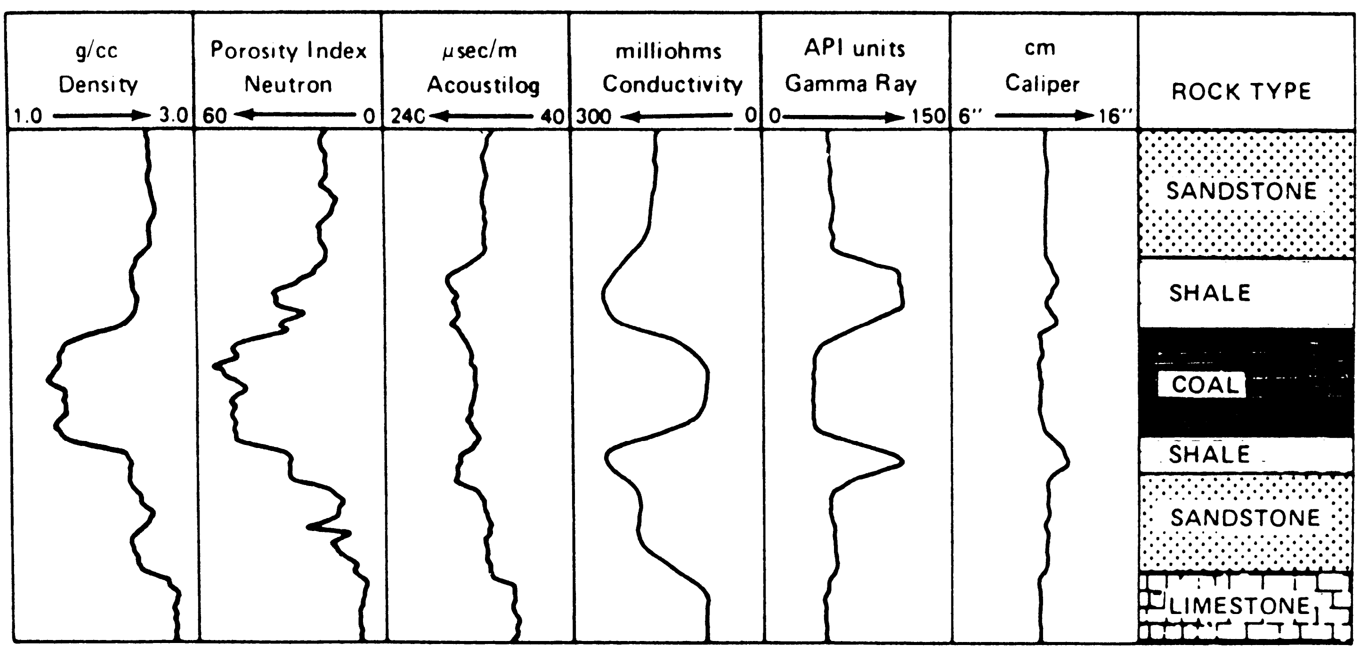


Fig. 10.3.5. Generalized response of geophysical logs in various lithologies (after Elkington et al., 1982). Conversion factors: 1 in. = 2.54 cm, 1 ft = 0.3048 m, 1 lb/ft^3 = 16.018 kg/m^3 = 0.016 g/cm^3.

ceiver are located in the same borehole, with the fixed separation distance maintained by glass fiber rods. The transmitter-receiver array is moved along the borehole and measurements are made at fixed intervals. Each measurement, including the movement to the next measuring position, takes about 30 to 50 seconds. Measurements can be performed with antennas ranging from 20 to 80 MHz. A large penetration (300 to 500 ft, or 100 to 150 m) from the borehole in combination with a high resolution (7 to 16 ft, or 2 to 5 m) is obtained.

In a cross-hole measurement, the transmitter and receiver are placed in separate boreholes. One of the probes is kept in a fixed position in one of the boreholes while the other probe is

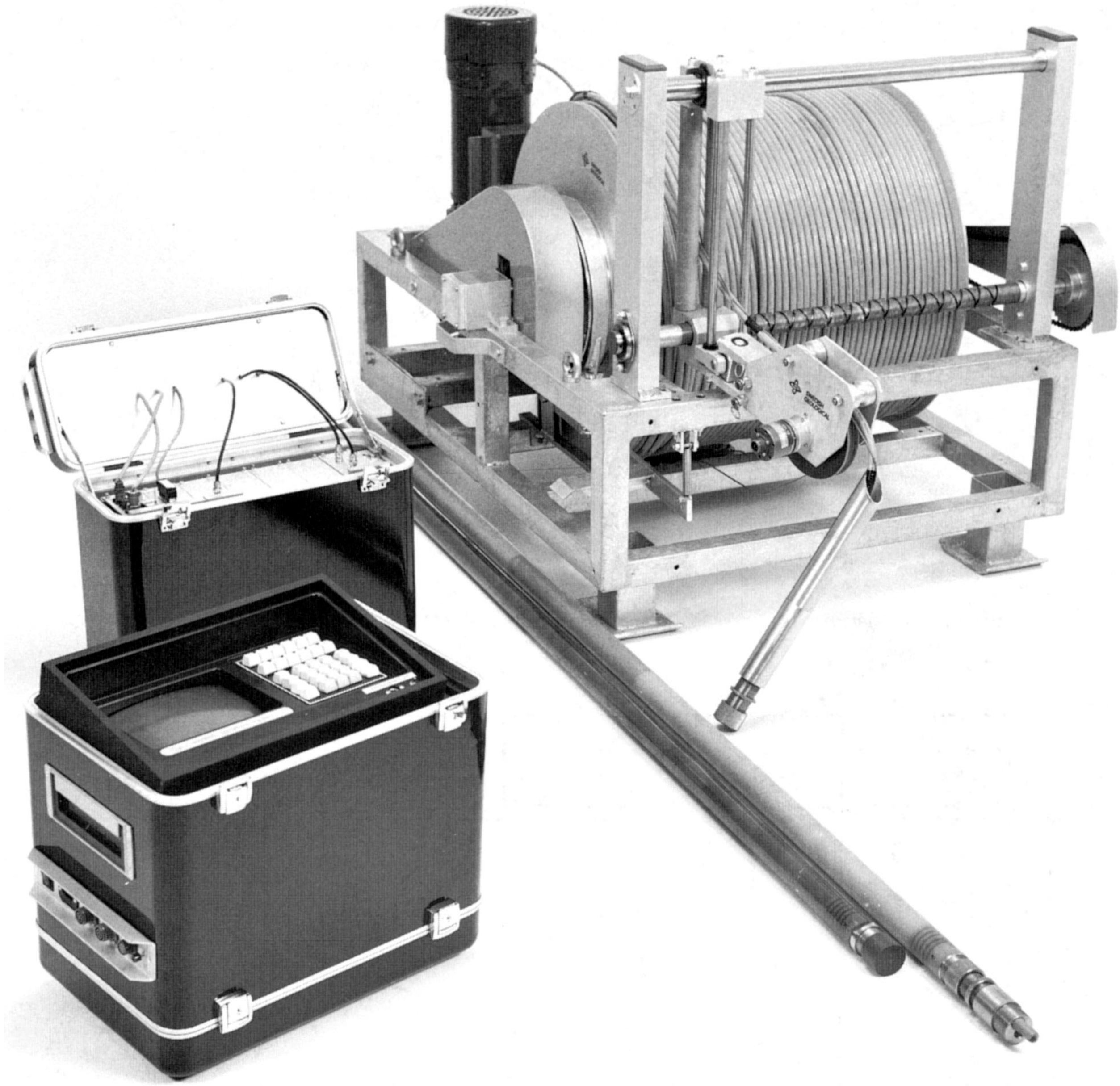


Fig. 10.3.6. Borehole radar equipment (*courtesy:* Swedish Geological Co., SGAB).

moved in the other hole where measurements are made at fixed intervals.

Fig. 10.3.7 is an example of a measurement made in granite. In the radar plot, fractures are shown as lineaments; the drift located about 130 ft (40 m) from the borehole is shown as a hyperbola, and a pair of 2.2-in. (56-mm) boreholes intersecting the measured borehole at 0 ft (0 m) are shown as line reflectors.

Borehole Water Level: The determination of water conditions is an important input (Anon., 1977b) into slope design as well as the evaluation of slope stability. The most common way of obtaining such information is simply through the observation of the water level in an uncased borehole or observation well. A disadvantage of this system is that a perched water table or an artisan pressure in one of the intersected strata can make the readings meaningless.

The simplest technique for determining the basic water table level in a drillhole is through the use of an open-well piezometer (Fig. 10.3.8). The hole is drilled to the layer of interest, a PVC casing with a filter tip at the lower end is installed, and the annulus between the PVC pipe and the hole wall is backfilled with bentonite to prevent flow of water into the filter from other locations along the hole. When the probe, shown in the figure, touches the water surface, an electrical circuit is completed, and the lamp lights. The depth to the surface is read from the cable tape. Knowing the difference in elevation between the water surface and the filter tip, the pressure at the tip can be calculated. Flow rate can also be calculated by pumping down the water level in the hole and observing the time of recharge.

Water Pressure Measurements: When the water pressure is to be measured directly, pneumatic and hydraulic piezometers

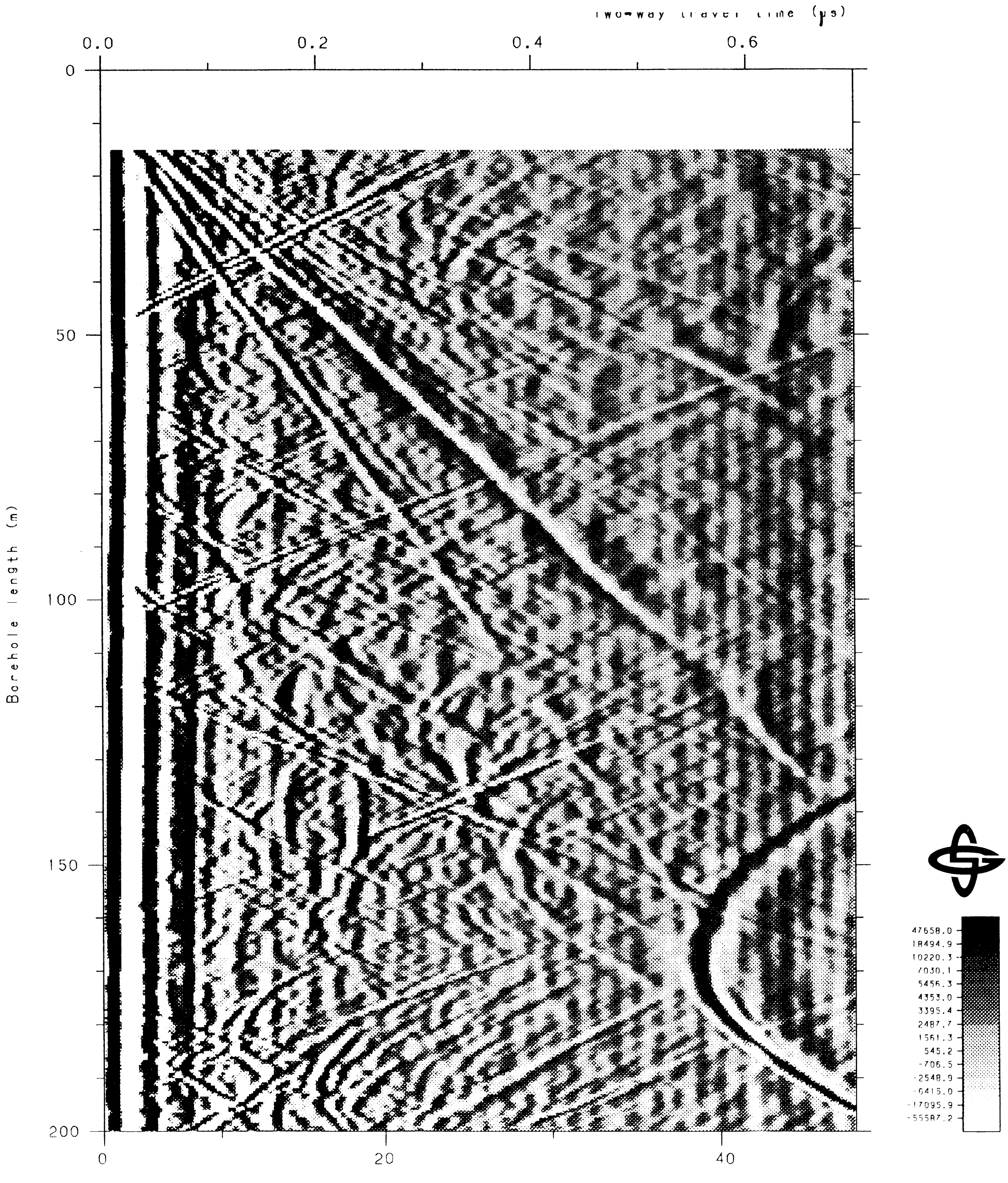


Fig. 10.3.7. Radar map produced in crystalline rock from borehole F4 60 MHz antenna (*courtesy:* Swedish Geological Co., SGAB). Conversion factor: 1 ft = 0.3048 m.

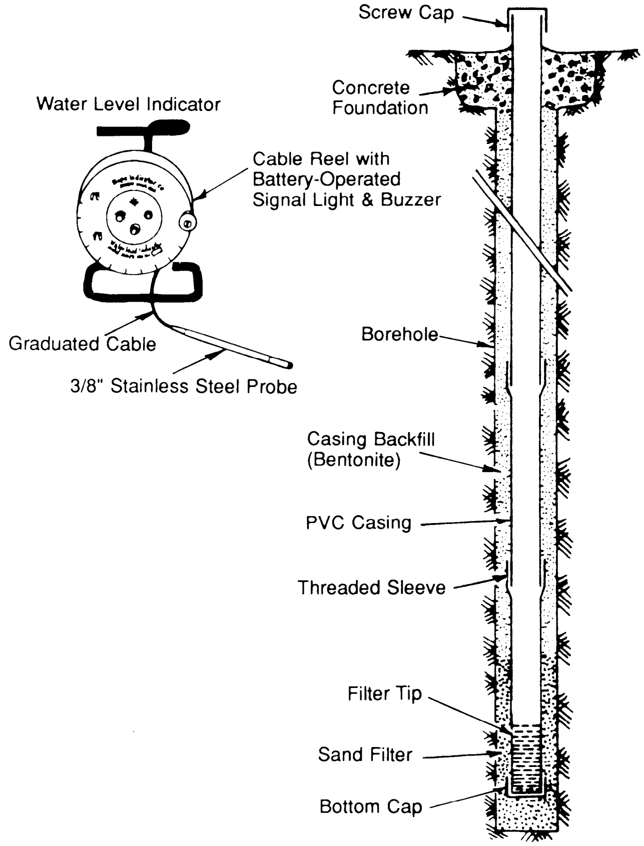


Fig. 10.3.8. Borehole water level device (*courtesy:* Slope Indicator Co.). Conversion factor: 1 in. = 25.4 mm.

employing a special sealed tip containing a pressure-sensitive valve are often used. The valve opens or closes the connection between two tubes that lead to the surface, the slope face, or any other convenient location and elevation. In the piezometer shown in Fig. 10.3.9, compressed nitrogen flow through the outlet tube is established as soon as the inlet-tube pressure equals the pore-water pressure. In the hydraulic piezometer, hydraulic fluid is used instead of gas, but the basic principle remains the same. Pneumatic piezometers have the following advantages: (1) negligible time lag because of the small volume change required to operate the valve, (2) simplicity of operation, (3) capability of purging the lines, (4) minimum interference with construction, and (5) long-term stability. Their main disadvantage is the absence of a de-airing facility.

Electric piezometers have a diaphragm that is deflected by the pore pressure against one face. The deflection of the diaphragm is proportional to the pressure and is measured by means of various electric transducers.

Because they may be effected by the environment or they may have various life spans, the longevity record of electric piezometers should be investigated prior to selection for installations in which reliable readings are required during an extended period of time.

10.3.2.3 Slope Design Assessment

Strike-dip Information: The collection of strike and dip information from exposed rock surfaces remains one of the most important tasks for the geologist/rock mechanician (Anon., 1978b). Fig. 10.3.10 shows a pocket transit that is commonly used for making such measurements (Anon., 1990k).

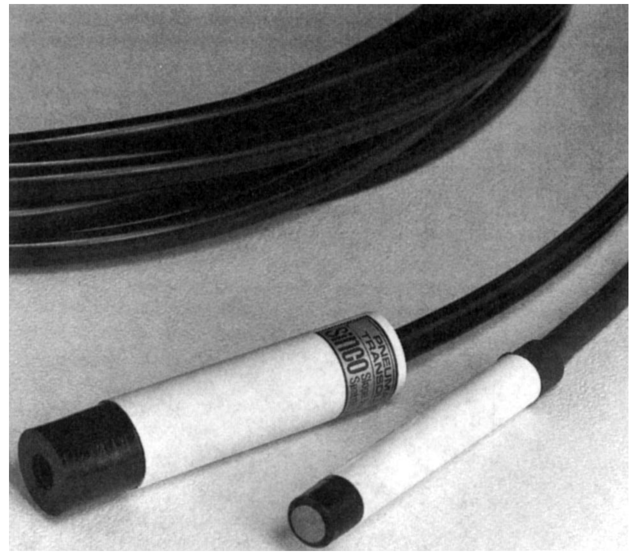


Fig. 10.3.9. Borehole piezometer for water pressure measurements (*courtesy:* Slope Indicator Co.).

Electronic Notebook: Although the data collected are generally still entered by hand into a paper notebook for later manual transfer onto maps or into a computer, electronic notebooks are available that greatly simplify the process. These notebooks can be programmed for data collection in the form desired, do some intermediate processing, and then transfer the contents of its memory to a personal computer, mainframe computer, or printer. Fig. 10.3.11 shows one such electronic notebook being used in the data collection mode (Anon., 1990b).

10.3.2.4 Slope Monitoring

Introduction: Stability monitoring of slopes and dumps is important at many open pits and strip mines (Barron et al., 1971; Call, 1982b; Stewart et al., 1971; Weir-Jones and Bumala, 1975; John, 1977). There are many techniques that can be used. Typical measurement objectives (Wilson et al., 1978) are

1. To determine absolute lateral and vertical movements of a sliding surface.
2. To determine the rate of sliding (accelerating or decelerating) and thus warn of impending dangers.
3. To determine the depth and shape of the sliding surfaces.
4. To determine the relative movements within a slope.
5. To monitor the effectiveness of various control measures.
6. To monitor groundwater levels and pore pressures so that analyses can be made.

There are a number of reasons why measurements are made. The obvious one is for safety. By monitoring the movements and the rate of change of movements, warning can be provided in advance of a slide. With such a warning, certain actions ranging from restricting area access to the unloading of the slope can be planned and taken. Design parameters such as friction angle, cohesion, groundwater levels, and pore pressures can be obtained through slope monitoring. These can lead to better slope designs. The measurement of groundwater parameters has already been discussed. The focus in this segment will be on the measurement of deformation.

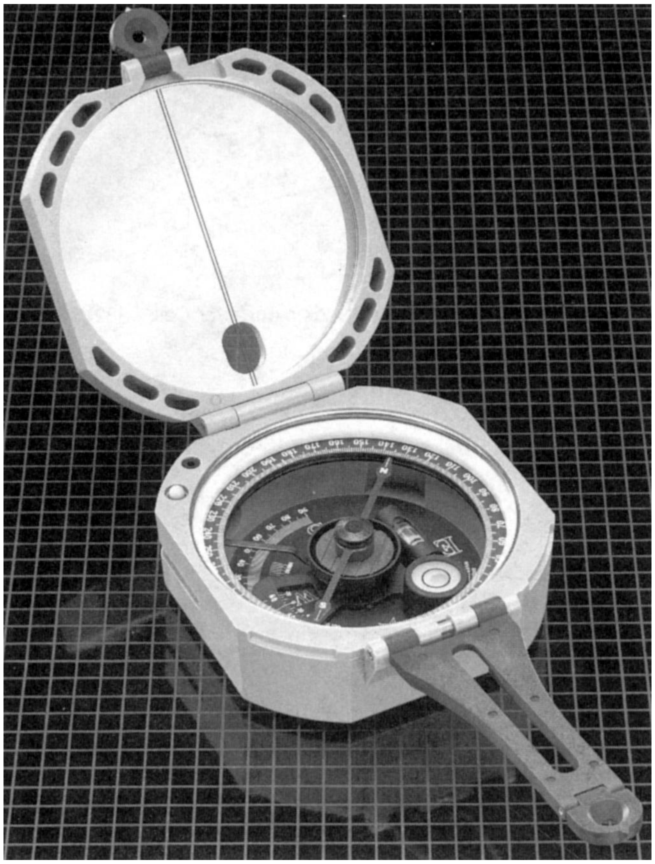


Fig. 10.3.10. Brunton compass (*courtesy:* Brunton, USA).

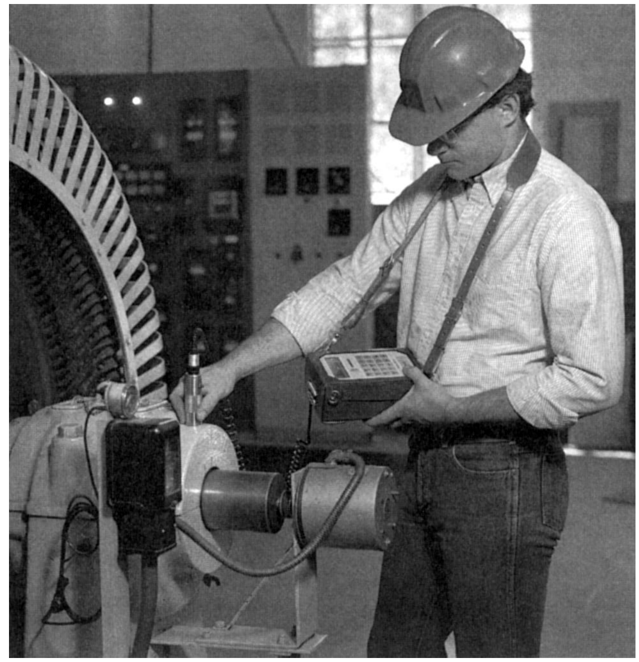


Fig. 10.3.11. Electronic notebook/recorder in use (*courtesy:* Omnidata).

The overall objectives of a pit slope monitoring program (Call, 1982) are

1. To maintain safe operational procedures for the protection of personnel and equipment.
2. To provide advance notice of instability so that mine plans can be modified to minimize the impact of slope displacement.
3. To provide geotechnical information for analyzing the slope failure mechanism, for designing appropriate remedial measures, and for conducting future redesign of the slope.

Conventional Survey Networks (Call, 1982): Surface displacement measurement using conventional survey equipment and extensometers has been and continues to be the most widely used and cost-effective technique.

Fig. 10.3.12 shows a survey network consisting of targets on the pit slopes and instrument stations from which angles and distances to the targets are measured. Either triangulation with a theodolite or trilateration with an electronic distance measuring (EDM) instrument can be used.

In planning the network, care should be taken to ensure that a sufficient number of targets are established for collecting the necessary data. If a total station EDM instrument is used, approximately 3 min will be required for each reading; thus about 30 to 40 prism targets can usually be surveyed in a half day. If a distance-meter EDM and a theodolite are used in combination, each reading would take about 5 min.

The survey network has several primary functions:

1. It establishes a surveillance system to detect initial stages of slope instability.

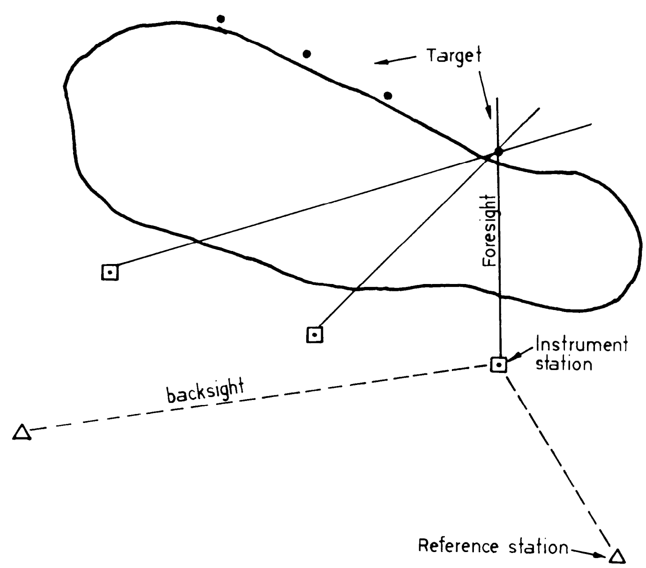


Fig. 10.3.12. Survey network for open pit mine (after Call, 1982).

2. It provides a detailed movement history in terms of displacement directions and rates in unstable areas.
3. It defines the extent of the failure areas.

Since deviations in survey data can result from the inability to repeatedly set up exactly in the same position at the stations, the observation (instrument) station should have stable bases. These are best established with concrete or metal monuments. An instrument base plate affixed to the top of the monument serves as an instrument platform.

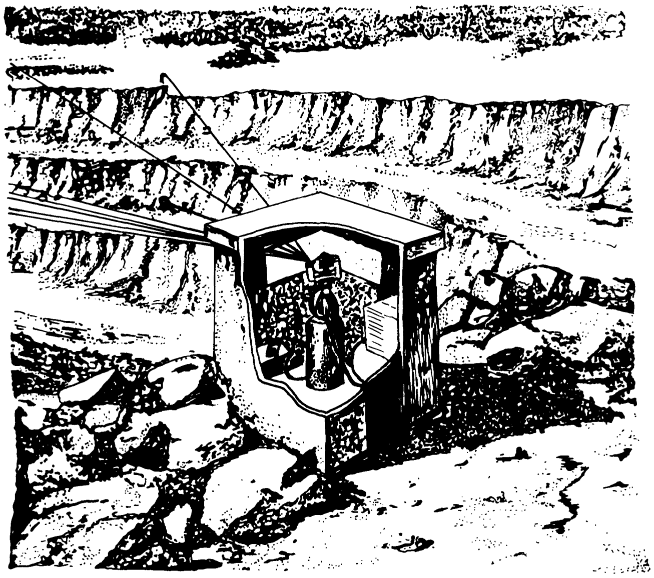


Fig. 10.3.13. Computer-directed surveying (*courtesy:* Geotronics).

Primary survey points, used to tie the observation stations to the mine grid baseline, should be located on stable ground, beyond the influence of the pit excavation.

Prism targets should be attached to bench faces, if possible. A location 6 to 8 ft (1.8 to 2.4 m) above the bench toe is usually preferred. Minor raveling may dislodge the prism if it is located near the crest. If the target is near the toe, it could be covered relatively quickly by raveled rock debris. In some areas, the prism reflectors will have to be mounted on sturdy tripods at selected monitoring points. It is best to allow newly installed targets to stabilize for one week before readings begin. Initially, the readings may be somewhat erratic, but an overall trend should soon become apparent.

An adjustment capability is usually needed for each target on a moving slope because it may be necessary to adjust the prism for proper instrument alignment if significant slope displacement occurs or if the instrument station is relocated. The prism can tolerate a misalignment up to 14º and still return the signal to the EDM.

Automated Surveying: A computer-controlled measurement system (Anon., 1989c) that can be programmed to measure distances and horizontal and vertical angles at predetermined intervals to a set of prism targets, anchored in the pit wall, is currently available (Fig. 10.3.13).

An analysis of the measurements can be performed by the computer and the results, in terms of target movement, velocity, and acceleration, can be presented numerically or graphically on the computer display or a printer. The measurement data can also be transferred to a mainframe computer over a data link for further processing.

Surface Mounted Extensometers: Surveying procedures can be used to obtain absolute displacements (with respect to a fixed reference) or relative movements. These may be considered as a type of surface extensometer on a very large scale. Other types of surface extensometers of varying ranges, precisions, and costs are also used for monitoring.

One early, obvious indication of slope instability is the development of tension cracks. By systematic mapping of these cracks, the extent of the unstable area can be established (Fig. 10.3.14).

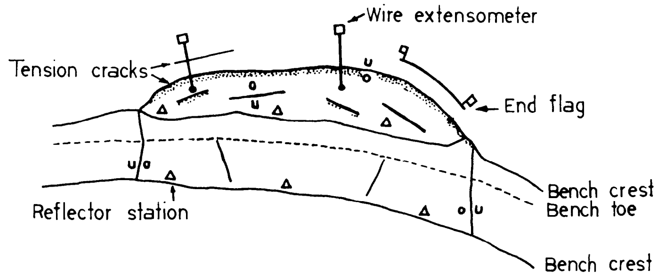


Fig. 10.3.14. Tension crack map (after Call, 1982).

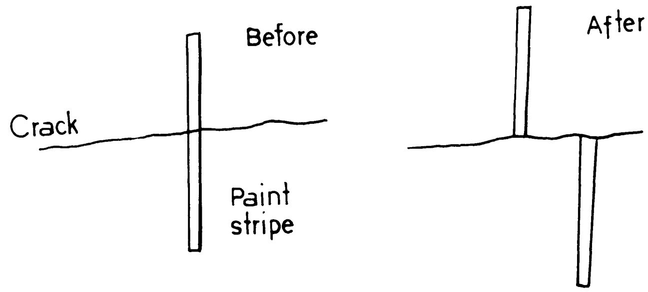


Fig. 10.3.15. Paint stripe technique (after Call, 1982).

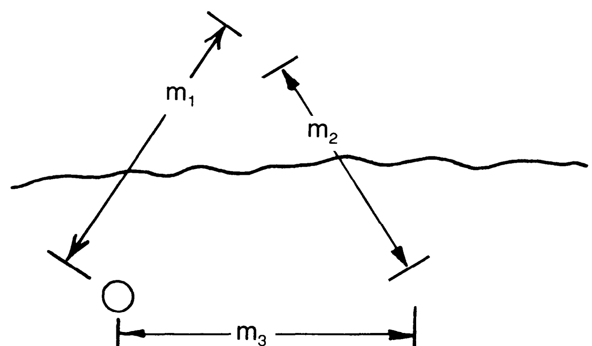


Fig. 10.3.16. Simple triangulation system for crack monitoring (after Call, 1982).

The ends of the cracks should be flagged so that on subsequent visits new cracks or extensions of existing cracks can be identified.

It is often useful to monitor on a very small scale (Anon., 1984), such as is shown in Fig. 10.3.15. The displacement of the simple paint stripe gives a good idea of movement.

The use of three rebar posts installed as shown in Fig. 10.3.16 provides triangulation possibilities. By measuring the distances between marks in the tops with a tape, one can determine relative motion and movement direction.

Another simple indicating device is a hardwood wedge lightly forced into the joint crack with a mark placed on it at the level of the rock. If the joint opens, then the wedge will fall to a deeper level in the rock. However, if the joint is closing or

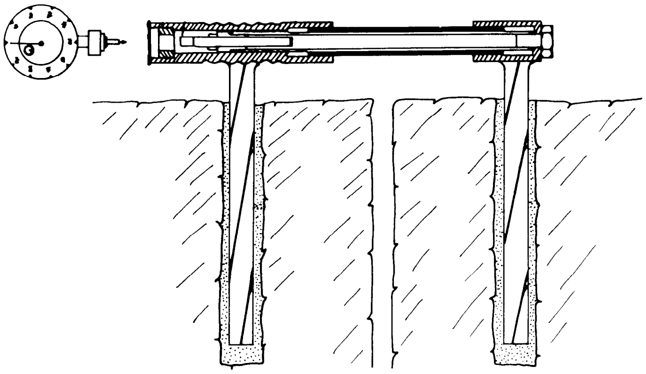


Fig. 10.3.17. Micrometer-based crack monitoring device (*courtesy:* Interfels).

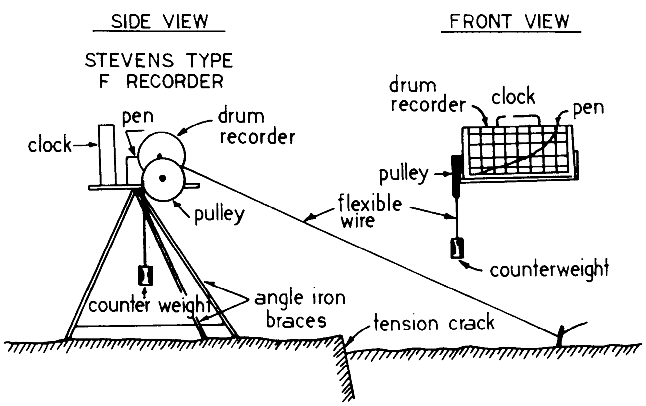


Fig. 10.3.18. Wire extensometer with drum recorder (after Call, 1982).

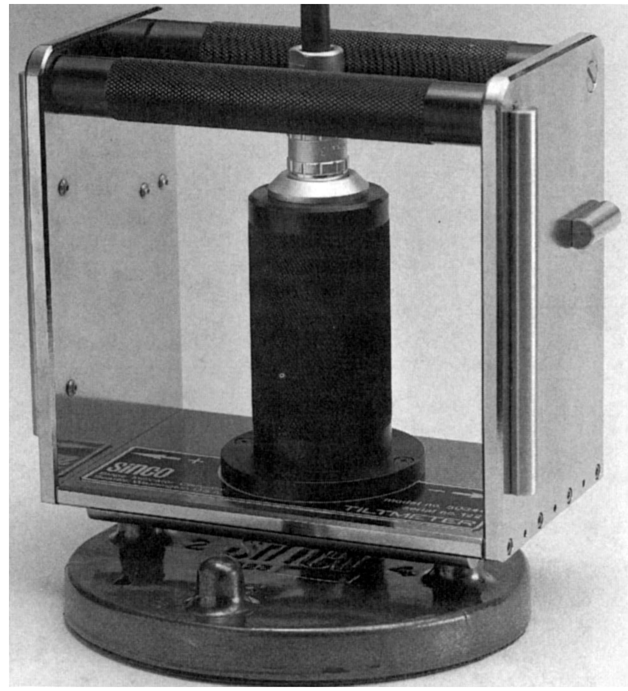


Fig. 10.3.19. Tiltmeter (*courtesy:* Slope Indicator Co.).

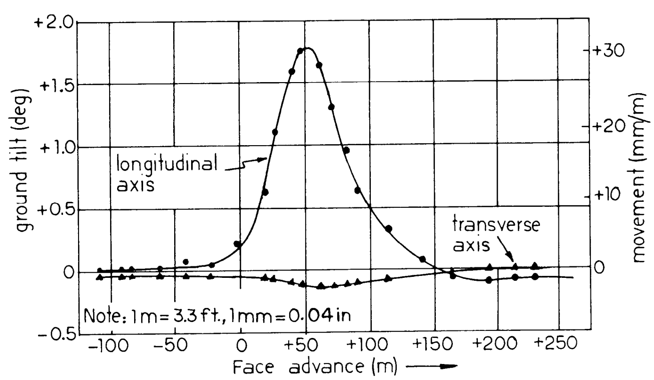


Fig. 10.3.20. Output from tiltmeter used in subsidence zone (after Wilson et al., 1978).

subject to shear movement, such a wedge indicator may not be helpful.

More elaborate systems (Fig. 10.3.17) are also available.

Portable wire extensometers can be used to provide monitoring in areas of active instability and to provide backup for the survey system. These monitors can be quickly positioned and easily moved. A simple extensometer, which can be fabricated in the mine shop, appears in Fig. 10.3.18.

For backup to the survey system, an extensometer should be positioned on stable ground behind the last visible tension crack, and the wire should extend out to the unstable area. Wire extensometers can be placed at strategic locations where anyone working in the area can make an immediate check on slope movement by inspecting the instruments.

A wire extensometer can be set up as a warning device by affixing a switch several inches (millimeters) above the displacement weight; significant displacement will trip the switch. Lights or sirens powered with a 6-V dry cell battery wired to the switch will warn of slope activity. A continuous drum-type recorder adapted to an extensometer can provide a continuous record of slope movement (Fig. 10.3.18). It will provide excellent data regarding the sensitivity of the slope to blasting, production, and rainfall.

The length of the extensometer wire should be limited to approximately 200 ft (60 m) because sag can produce inaccurate readings. Usually 35 to 50 lb (17 to 24 kg) of counterweight are needed for such a length, but this depends on the tensile strength of the wire. Aircraft control cable or similar wire, which is manufactured to have very little stretch, is recommended for this type of monitoring device. The flexibility and durability of steel cable, compared to the rigidity and brittleness of INVAR wire, outweigh the benefits of the thermal properties of INVAR wire. Temperature fluctuations can be measured, and corrections can be made if necessary (Call, 1982).

Tiltmeters: Tiltmeters provide another means of monitoring movement (Anon., 1977d; McCarter, 1976). They may be used in any area where the failure mode of a mass of soil or rock can be expected to contain a rotational component. One type of tiltmeter in which servo-accelerometer sensors are used is shown in Fig. 10.3.19. Sample tiltmeter data collected from a subsidence area overlying an underground mine appear in Fig. 10.3.20. A simpler tiltmeter contains a mercury switch which, when tilted,

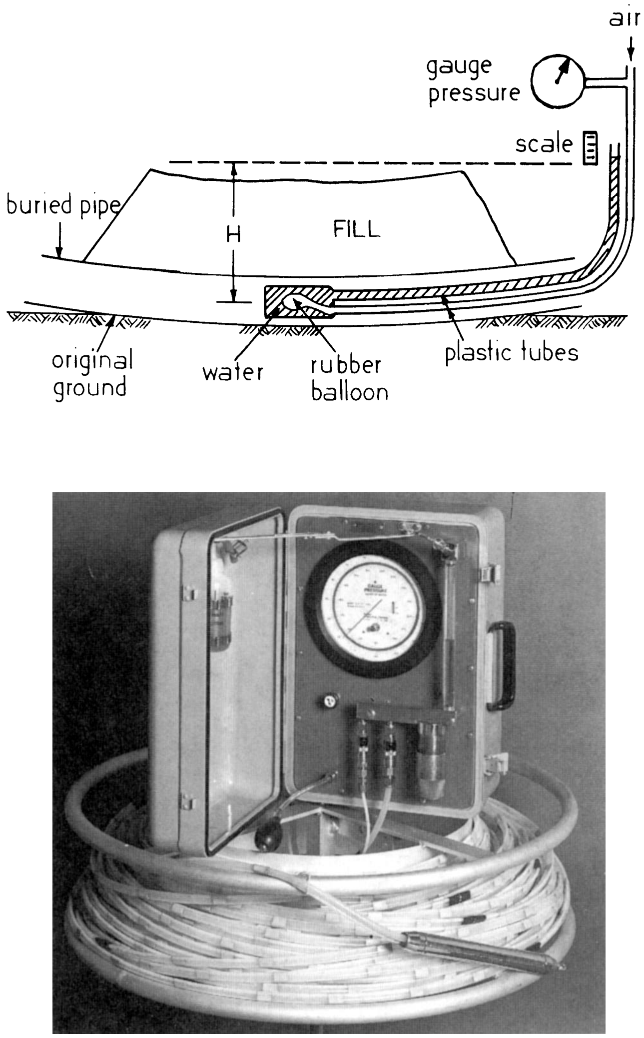


Fig. 10.3.21. Liquid-level displacement gage (*courtesy:* Walter Nold Co.).

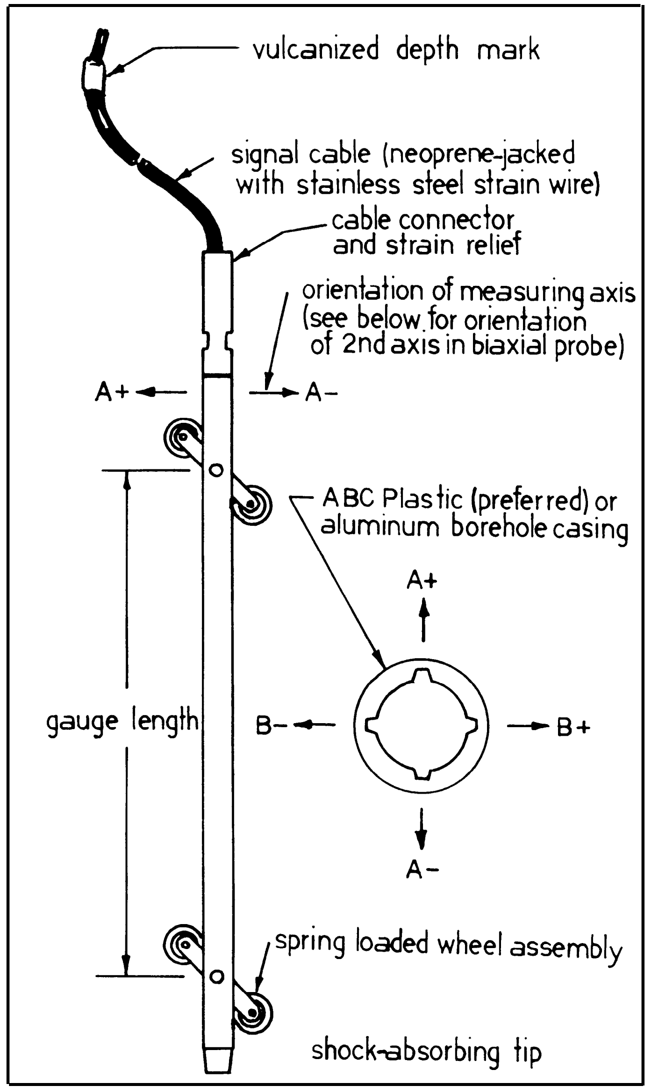


Fig. 10.3.22. Diagrammatic representation of inclinometer (after Wilson, 1970).

causes an electrical circuit to be closed. This is an off/on type of device that can be connected to a warning system. It does not provide information regarding rate of movement, etc.

When tension crack displacement is predominantly vertical, a tiltmeter, consisting of a bar across the crack with protractor and pendulum, can be used to measure displacement. A mercury switch on the bar can be attached to a warning light.

Liquid-level Displacement Gages: A liquid-level vertical displacement gage operates on the physical principle that the liquid level in two vessels joined by a connecting tube is at the same elevation, provided the pressure on the liquid surface is the same in both vessels. The change in elevation between the two reservoirs can be determined by micrometers attached to each reservoir.

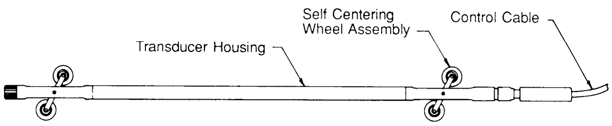
Fig. 10.3.21 shows a diagrammatic, horizontal, full-profile settlement gage that works on a somewhat similar principle. A water-filled tube terminated by a torpedo is pulled or pushed through a 1 l/2-in. (38-mm) diameter PVC pipe. Elevation differences of up to 20 ft (6 m) can be evaluated over lengths of 500 ft (150 m). The settlement accuracy is from l/4 to 3/4 in. (6 to 19 mm), depending upon the amount of care used while taking a reading.

Sub-surface Instruments: Although it is possible to make inferences about the subsurface extent of instability from surface displacement measurements, there are situations where subsurface data are needed. Access is generally through the use of boreholes. Because of this, they involve considerably more expense. Four devices fall into this category:

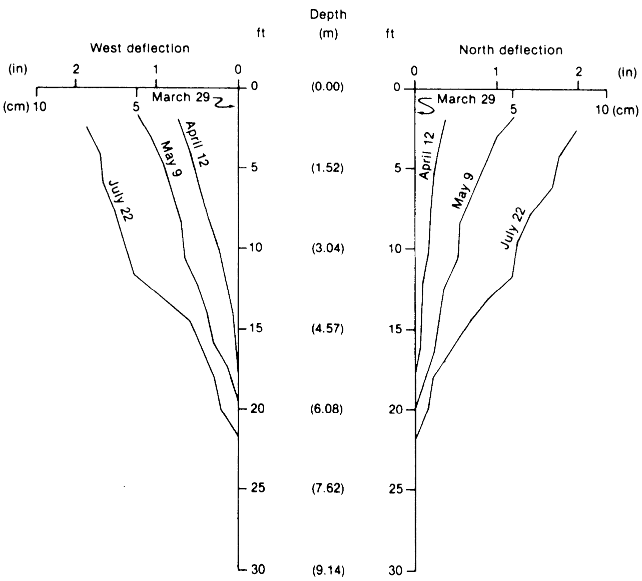
1. Inclinometers.
2. Shear strips.
3. Time-domain reflectometry.
4. Borehole extensometers.

Only the first three are discussed here since extensometers are covered in the underground segment (10.3.3).

Inclinometers—An inclinometer (Anon., 1977d; Anon., 1981; Muller et al., 1977) measures the change in inclination (or tilt) of a casing in a borehole (Fig. 10.3.22) and thus allows the distribution of lateral movements to be determined as a function of depth below the ground surface and as a function of time. Application of the inclinometer to slides is readily apparent, namely, to define the slip surface or zones of movement relative to the stable zones. Inclinometers have undergone rapid develop-



A Portable Borehole Inclinometer
with Self Centering-Type Wheel Assemblies. (Courtesy
of Slope Indicator Co.)



Plotted Traverses of a Portable Borehole Inclinometer.

Fig. 10.3.23. Typical output from inclinometer installation (after Wilson, 1970).

ment to improve reliability and accuracy, reduce weight and bulk of instruments, lessen data acquisition and reduction time, and improve versatility of operations under adverse conditions. Automatic data-recording devices, power cable reels, and other features are now available.

Most inclinometers measure the inclination of the casing in two mutually perpendicular near-vertical planes. Thus horizontal components of movement, both transverse and parallel to any assumed direction of sliding, can be computed from the inclinometer measurements.

The function of the inclinometer is to detect the change in inclination of the casing from its original near-vertical position. Readings taken at regular preestablished depths inside the casing allow the change in slope at various points to be determined; integration of the slope changes between any two points yields the relative deflection between those points. Repeating such measurements periodically provides data on the location, magnitude, direction, and rate of casing movement (Fig. 10.3.23). The integration is normally performed from the bottom of the hole, since the bottom is assumed to be fixed in position and inclination.

Shear Strips—The shear-strip failure locator (Fig. 10.3.24) is a safety device used to detect and locate up to two failure zones in soil, rock, or concrete. It can be installed in boreholes as small as 1 1/2 in. (38 mm) in diameter, bonded to surfaces, or embedded in slots or trenches. The strip consists of a series of resistors mounted on a brittle substrate that breaks with shear movements. The shear-strip indicator is used to make the read-

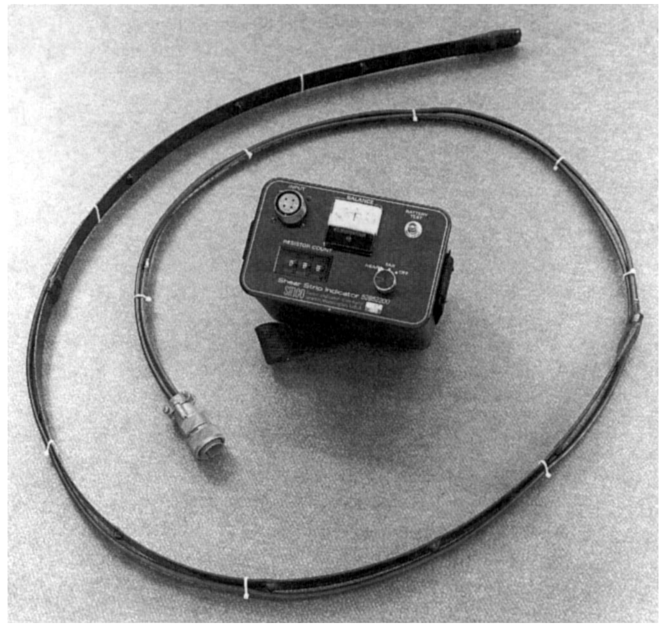


Fig. 10.3.24. Shear strip failure locator (*courtesy:* Slope Indicator Co.).

ings. These systems have the limitation of being go/no-go devices.

Time-Domain Reflectometry—This technique (Alexander, 1990; Anon., 1983; Anon., 1989e) was initially developed to locate discontinuities in coaxial transmission cables. In mining, the technique has been employed (Wade and Conroy, 1980; Panek and Tesch, 1981) to identify zones of rock mass deformation. When the ground deforms, a coaxial cable grouted into a borehole will also be deformed, thereby modifying cable impedance. A step voltage pulse transmitted along the cable experiences a partial reflection at the points where impedance changes exist. Pulse reflections from all changes in impedance along the cable are superimposed on the input pulse to form a series of TDR signatures. The composite signature observed on a TDR cable tester screen consists of many individual reflections generated by localized deformations along the cable.

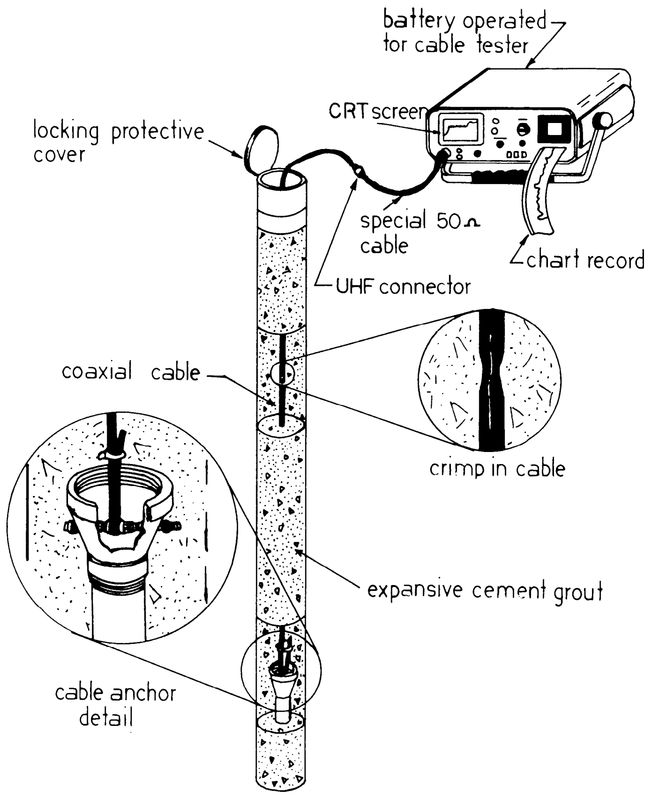
The time delay between a transmitted pulse and the reflection from a cable deformity (cable fault) uniquely determines the fault location. The time, sign, length, and amplitude of the individual reflection pulses define the location, type, and severity of the deformity.

Cable crimps are made at known distances during installation to serve as references for distance along the cable. A schematic TDR installation (Dowding et al., 1989; O'Connor and Dowding, 1984) is shown in Fig. 10.3.25. A readout instrument is shown in Fig. 10.3.26. Some results in which the TDR technique was used to monitor caving over a coal longwall face (Wade and Conroy, 1980) are shown in Fig. 10.3.27. Obviously, the technique can be used to monitor rock slopes and even the development of caving in underground mines.

10.3.3 INSTRUMENTATION FOR UNDERGROUND MINING APPLICATIONS

10.3.3.1 Introduction

Underground mining systems in the future will be based upon four fundamental concepts:



SCHEMATIC TDR INSTALLATION

Fig. 10.3.25. Time-domain reflectometry installation (after Dowding et al., 1989).

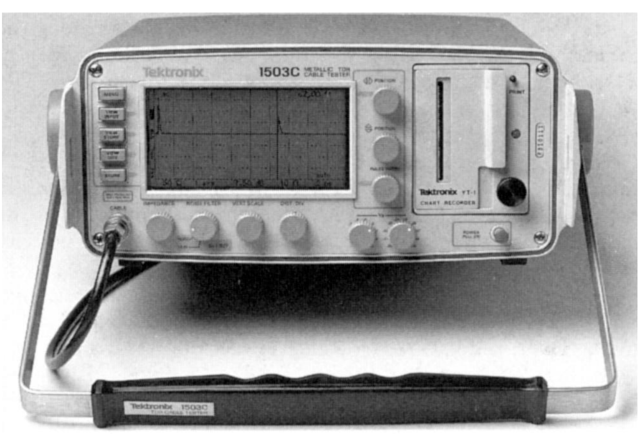


Fig. 10.3.26. Readout device for TDR (*courtesy:* Tektronix).

1. Low specific development.
2. High degree of automation.
3. High level of equipment utilization and availability.
4. Minimum time lag between development and production.

In order to achieve these objectives, a much greater emphasis and level of responsibility than in the past will be placed on mine engineering, and particularly on the mine design component. The engineering of underground openings requires a knowledge

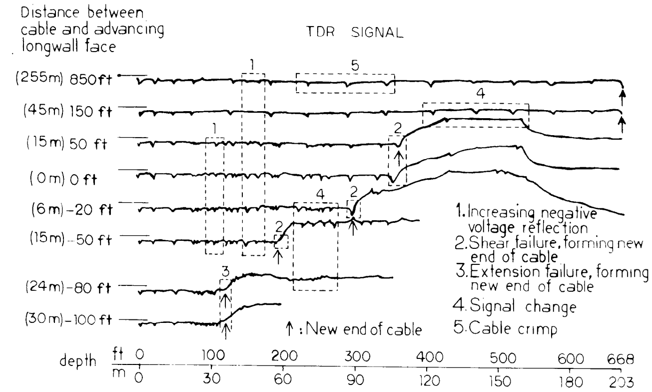


Fig. 10.3.27. Results from TDR monitoring of a longwall (after Wade et al., 1980).

of rock mass properties, the in situ stress state, the interaction between the rock mass and various stabilizing elements, etc. As in surface mining, instrumentation plays an important role in

1. Developing the basic underground design.
2. Developing design improvements.
3. Short- and long-term design monitoring.

Advances in underground mining have tended to outstrip the state of the art in the prediction of geologic conditions and rock behavior. It has often been necessary to evaluate conditions as they are encountered, and adjust mining procedures accordingly. It is in this area of timely identification and evaluation of unforeseen rock and soil mass conditions that field instrumentation can help improve both the safety and economy of underground mine.

Mining instrumentation involves measurements to (1) observe the behavior, and hopefully the stabilization, of geologic masses disturbed by operation, (2) evaluate the effectiveness and efficiency of support systems and structure designs, (3) provide early detection of hazards, (4) indicate remedial action for hazards, and (5) indicate the effectiveness of remedial measures. The measurements are typically of stress, load, pressure, strain, and displacement.

Measurements (Bauer, 1985; Boisen, 1990; Dunnicliff, 1988; Franklin, 1977) can be made on exposed surfaces of structures or masses, in drillholes, at interfaces, between surfaces (tunnel walls, roof/floor), and internally, typically in structures. Measurements distributed over an entire mass or structure are preferable to those at isolated locations or test sections. Instruments should be selected and distributed to provide a reasonable degree of data redundancy so that important interpretations are not made on the basis of single measurements. Other important considerations are that sample distribution and redundancy be achievable even under the conditions of limited site accessibility that are typical of most field locations, and that costs be tolerable.

Achieving any rational basis for designing an underground opening or system of openings requires knowledge of both the mechanical properties and the state of stress in the surrounding rock. This information, together with knowledge of the deformations produced by or following excavation, are essential for evaluating the stability of underground openings in rock. The development of instrumentation and procedures for measuring rock properties, the state of stress, and structural deformations has been the subject of many investigations. Some of the instruments and more generally accepted procedures are described in the following.

10.3.3.2 Absolute Stress Measurements

Over the years, a number of different techniques (Obert, 1962; Aggson and Hooker, 1982; Dunnicliff, 1988) for determining the state of stress in the ground have been developed. Under ideal conditions, many will give satisfactory results. Unfortunately, many mining applications are far from ideal, and hence the results obtained are commensurate. Even under the best circumstances, successful stress determinations require careful instrument installation and interpretation by experienced people. Available space allows no more than a brief introduction to some of the methods. The Commission on Stress Measurements of the International Society of Rock Mechanics has prepared detailed materials regarding the major techniques (Anon., 1987a).

When considering which of the stress determination techniques is best suited to a particular site or problem under investigation, attention should be given to the following selection factors: (1) the soundness of the theoretical basis of each of the available methods; (2) the limitations, practicability, or ease of use of each of the available methods in the particular situation; and (3) the direct cost of equipment and the time required (Aggson and Hooker, 1982). These three factors cannot be considered independently. For example, the rate at which data can be gathered using a particular method must be considered when evaluating both the second and third selection factors. Data are usually obtained at a slower rate when using techniques that require the use of epoxy or grout. Trade-offs between the various selection factors must also be considered. However, if stress information is a necessity, reliability of results should be a prime consideration (Aggson, 1972).

Hydraulic Fracturing: The hydraulic fracturing method of rock stress determination was first demonstrated in 1957 (Hubbert and Willis, 1957). Since then, numerous investigators (Haimson 1974, 1977, 1988; Kehle, 1964; Ljunggren, 1990) have examined hydraulic fracturing as a means of in situ rock stress determination. This method of rock stress determination consists of sealing off a section of borehole with packers and then increasing the fluid pressure in the section of the hole until the surrounding rock is fractured (Fig. 10.3.28).

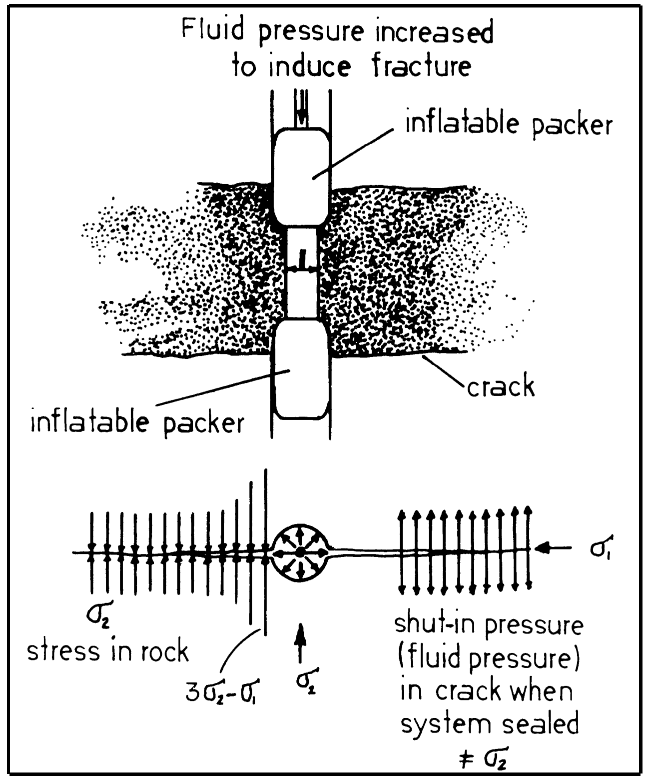


Fig. 10.3.28. Diagrammatic representation of hydraulic fracturing test (*courtesy:* Mindata).

The magnitude of the minor secondary principal stress component can be estimated directly from the shut-in pressure. The magnitude of the major secondary principal stress is calculated from relationships involving the crack initiation pressure, the crack reopening pressure, and the hydraulic fracture strength of the rock. The fracture strength can be determined from miniature hydraulic fracture tests carried out on the rock in question in the laboratory. The mini-frac equipment developed by the CSIRO (Anon., 1990d; Enever et al., 1989; Wold et al., 1989) is shown in Fig. 10.3.29.

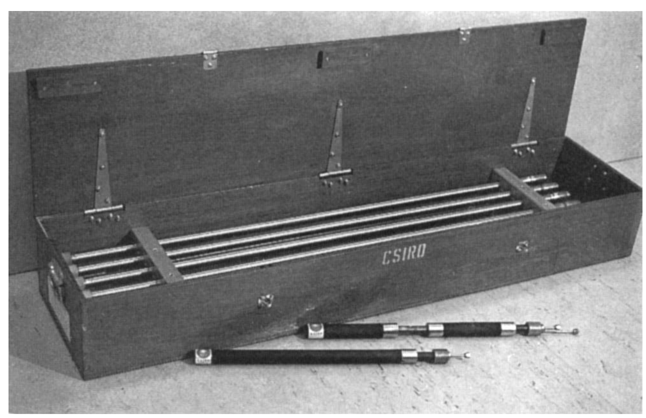


Fig. 10.3.29. Mini-frac equipment (*courtesy:* CSIRO).

An impression packer consisting of an inflatable element wrapped with a replaceable, soft rubber film can be used to determine the orientation of the fracture. This, in turn, gives the orientation of the major secondary principal stress in the plane normal to the hole axis. Recently there has been the commercial development of a small-diameter (1½-in. or 38-mm) cored-hole tool for mine use.

Because the hydraulic fracturing method of stress determination does not require some type of stress or strain relief by overcoring, this method is the only method that has been successful in deep (up to 6300 ft or 1920 m) boreholes (Haimson, 1974).

Borehole Deformation Gage: For many years, the most popular device for measuring in situ stresses has been the US Bureau of Mines (USBM) borehole deformation gage (Fig. 10.3.30) (Merrill and Peterson, 1961; Hooker et al., 1974; Hooker and Bickel, 1974; Bickel, 1985). This device consists of three pairs of strain-gaged cantilevers, each pair oriented at 120° to the next; these cantilevers are held within a sealed, stainless-steel housing, and their tips are deflected by means of tungsten-carbide-faced plungers.

The borehole gage is inserted inside an EX diamond drillhole so that the plungers are depressed. The initial deflection of the cantilevers is measured by connecting the signal cable to a conventional strain indicator readout box.

With the gage still in the EX borehole, a 6-in. (150-mm) concentric hole is drilled around the gage so that it is eventually isolated from the stressed rock mass inside a 5 5/8-in. (143-mm) diameter core about 12 in. (300 mm) long.

Fig. 10.3.30. USBM borehole deformation gage (*courtesy:* Geocon).

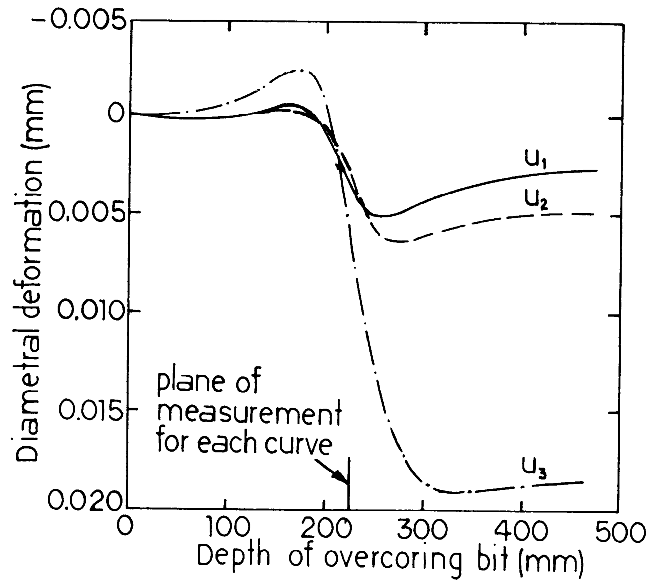


Fig. 10.3.31. Typical output from USBM gage (after Anon., 1987a). Conversion factor: 1 in. = 25.4 mm.

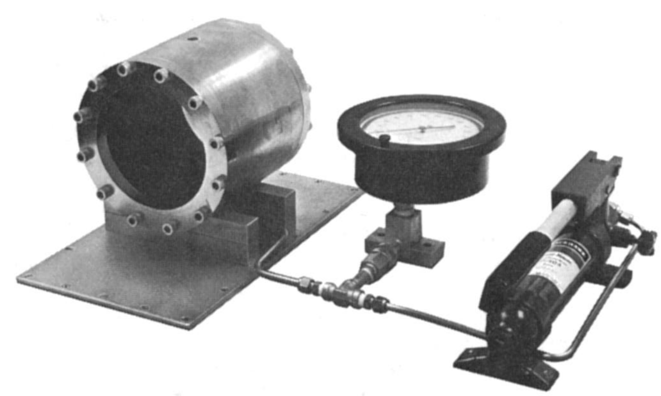


Fig. 10.3.32. Biaxial chamber for determining rock properties (*courtesy:* Geocon).

As this core is liberated from the stress field, it expands, and the resultant change in the diameter of the EX hole is measured by the strain-gaged cantilevers. Fig. 10.3.31 is a typical output.

After each overcore, the core is removed from the hole and placed inside a biaxial chamber (Fig. 10.3.32). This chamber incorporates a neoprene membrane that permits a hydraulic pressure to be exerted radially on the rock core while the resultant change in the EX borehole diameter is once again measured by means of the borehole gage. The stress/strain relationship so measured is used to calculate the elastic modulus of the rock, which then permits the rock stress to be computed from the strains measured during overcoring.

This procedure, repeated in three differently oriented boreholes, permits the magnitude and direction of the three principal stresses of the three dimensional state of stress to be calculated using an appropriate computer program.

In rock that tends to fracture easily, "disk," or "poker chip" during overcoring, the borehole gage can be modified by replacing the standard housing with a "reverse case" that allows the

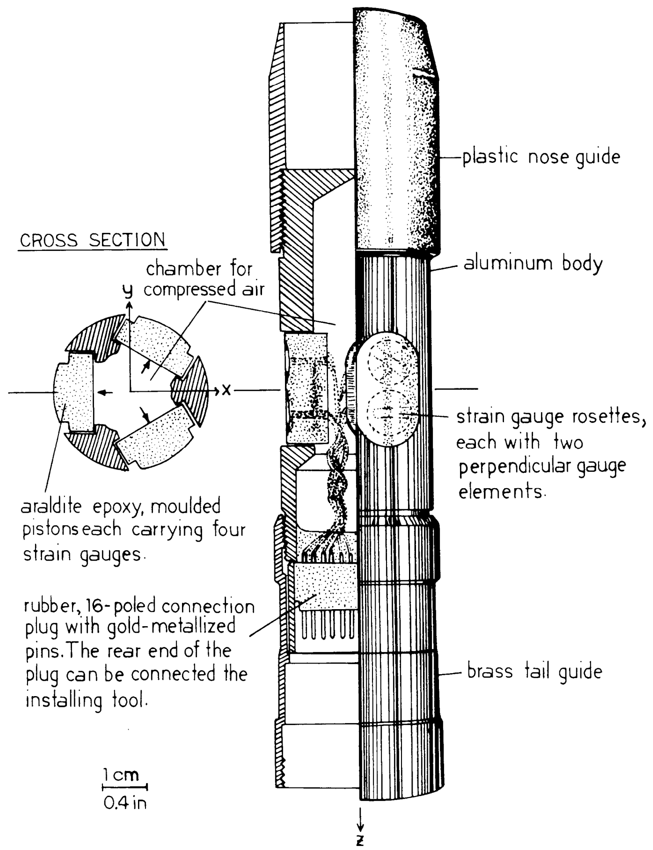


Fig. 10.3.33. Cross section through LuH cell (after Stillborg and Leijon, 1982).

Fig. 10.3.34. LuH cell (*courtesy:* Leijon, 1990).

cantilever plungers to be positioned very close to the start of the EX hole.

A calibration jig is available for periodic checks on the stability of the gage calibration.

CSIR Triaxial Strain Cell: The South African Council for Scientific and Industrial Research (CSIR) has developed a gage that can be used to obtain the complete stress tensor at one point in a borehole (Leeman 1964, 1969a, 1969b; Van Heerden, 1976). The triaxial strain cell consists of three groups of three strain gages mounted on the circumference of a tubular body.

A 1 l/2-in. (38-mm) diameter pilot hole is drilled and the gages glued directly to the borehole wall using a special positioning/installing tool. After the cement has dried, the initial strain readings are taken. The installing tool is then removed from the hole. After overcoring is completed, the installing tool is reconnected, and the new strains are read. The core, together with the included gage, is removed from the hole and placed in a biaxial chamber. From the pressure-strain readings, the elastic properties can be determined.

The strain changes induced by overcoring and the elastic properties of the rock are used to calculate the complete in situ state of stress from overcoring.

LuH Cell: Researchers at the University of Lulea (LuH) in Northern Sweden have made a number of improvements to the original CSIR triaxial strain cell (Stillborg and Leijon, 1982). Strain gages are attached to movable pistons contained in the body of the cell (Figs. 10.3.33 and 10.3.34). A rapid-setting cement is applied to the gages, and the cell is inserted in a small-diameter pilot hole. When the correct location and orientation is obtained, compressed air is used to force the pistons against the wall of the hole. After the cement has dried, initial strain readings are taken and the installing tool removed. The cell is then overcored and the resulting core removed from the hole. The installing tool is reattached and final strain readings are taken. The data acquisition system contains a computer which calculates the stresses from the measured strain changes. Elastic properties are determined using the biaxial chamber.

CSIRO Hollow Inclusion Cell: The "Hollow Inclusion" (HI) stress gage developed by the Australian Commonwealth Scientific and Industrial Research Organization (CSIRO) can also be used to determine the complete stress tensor at a measuring site (Worotnicki and Walton, 1976; Walton, 1980).

The CSIRO cell incorporates nine (or twelve) strain gages in three rosettes (Fig. 10.3.35) encased within an epoxy shell attached to a 12-core readout cable.

The cell is grouted into a borehole and overcored.

Fig. 10.3.36 shows typical output from a good test. The retrieved core containing the cell is placed in a biaxial chamber. From the resulting strains the elastic properties can be calculated. The strain gage results and rock material properties are processed by computer to provide principal stress magnitudes and orientations plus resolved vertical and horizontal stresses and relevant statistical information.

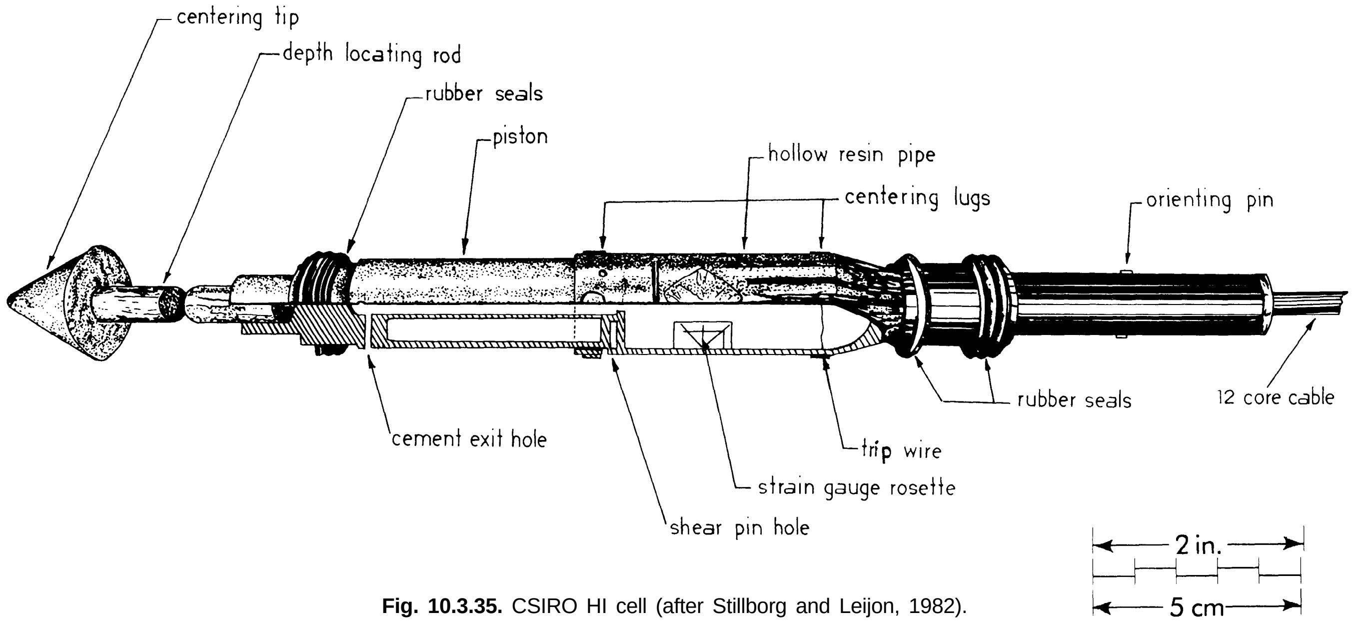


Fig. 10.3.35. CSIRO HI cell (after Stillborg and Leijon, 1982).

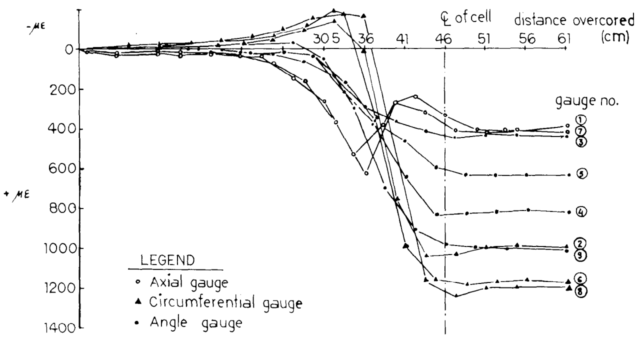


Fig. 10.3.36. Typical output from HI cell (after Walton, 1980). Conversion factor: 1 in. = 2.54 cm.

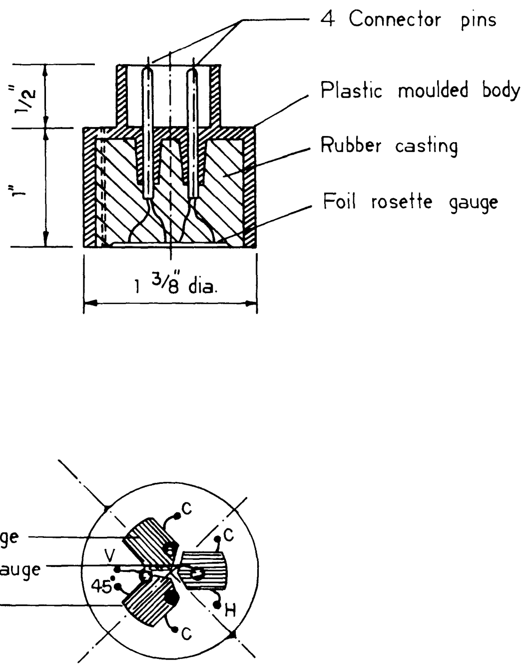


Fig. 10.3.37. Diagrammatic view of doorstopper (after Anon., 1966).

Doorstopper: The South African CSIR has also developed a device that has come to be known as the "doorstopper" (Leeman, 1964, 1969a, 1969b). It consists of a strain gage rosette potted in the base of a low-modulus cylindrical solid (Fig. 10.3.37 and 10.3.38). The doorstopper is glued to the end of a borehole that has been ground flat and smooth with a flat-faced, diamond-impregnated bit (Fig. 10.3.39). Strain readings taken before and after overcoring give a measure of the stress relief. The in situ state of stress within the rock mass is related to the stress relief measured at the end of the borehole through the use of stress concentration factors. Elastic properties (Young's modulus and Poisson's ratio) of the rock are used together with the overcoring data to calculate the stress. Readings in three boreholes are required to calculate the complete three dimensional stress field.

Flatjacks: The flat-jack method of rock stress determination was first described in 1952 by Habid and Marchand. This method basically consists of first installing strain devices oriented to measure in the line of intended stress determination (Merrill et al., 1964; Panek and Stock, 1964). An initial strain reading is made. A slot is cut between the strain gages, and a thin hydraulic cell called a flat jack is grouted into the slot. After the grout has cured, the flat jack is pressurized to a value such that the strain gages indicate their initial value. The flat-jack pressure is then considered to be equal to the rock stress that existed normal to the plane of the flat jack before the slot was cut. The flat-jack technique has the disadvantage of being limited to measurements near the surface of an underground opening and thus influenced by the geometry of the opening and resulting stress concentrations. However, it has the advantage of not requiring a knowledge of the elastic properties of the rock. The averaging effect due to the comparatively large area of the flat jack can also be a significant advantage. This method can be used in highly stressed

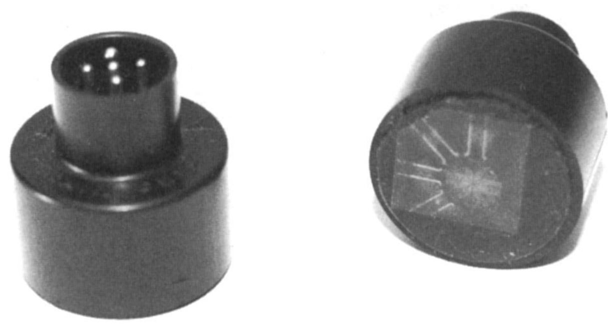


Fig. 10.3.38. Doorstopper (*courtesy;* Roctest).

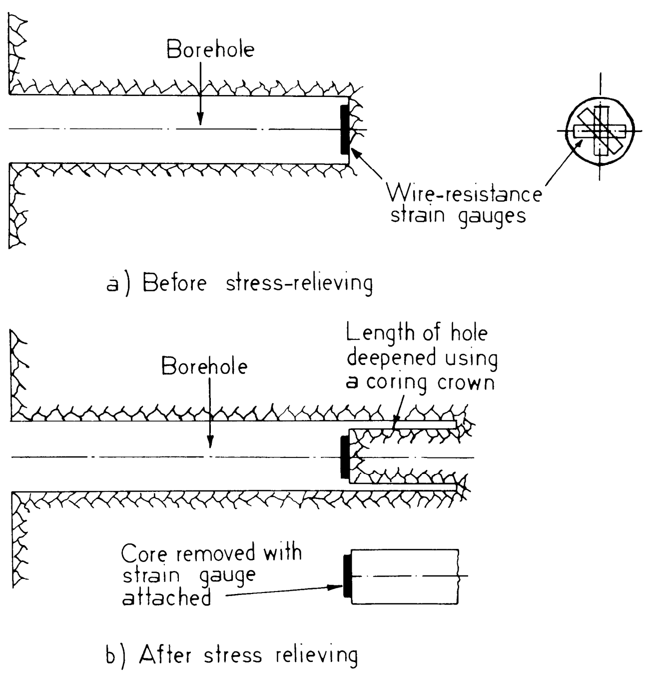


Fig. 10.3.39. Overcoring of doorstopper (after Leeman 1969a, 1969b).

areas or rock in which poor core recovery would eliminate those methods that require drill core recovery.

Core Disking (Aggson, 1982): The phenomenon known as core disking may be used to infer in situ rock stress magnitude. Core disking is the formation of disks or wafers of relatively uniform thickness that fracture or rupture on the surface approximately normal to the axis of the drill core. Usually, the surfaces of the disks are concave-convex, with the concave side toward the collar of the hole. Core disking is independent of core diameter and has been observed in drill cores 5 ft (1.5 m) in diameter down to less than 1 in. (25 mm). The presence of a center hole in the core, such as would be required for overcoring, does not influence the disking process.

Core disking, often called "poker chipping" by diamond drillers, occurs when the in situ stress field is high in relation to the strength properties of the rock involved. The relationships between the in situ stress field, the strength properties of the rock, and the stress concentrations caused by the borehole and the kerf of the drilling bit are rather complex. However, laboratory investigations (Obert and Stephenson, 1965), modeling studies (Durelli, Obert, and Parks, 1968), and field observations (Hooker, Bickel, and Aggson, 1972) have provided various relationships that allow estimation of the stress field magnitudes.

The drill core from a vertical hole near the earth's surface or from an underground borehole near the opening from which it is drilled will disk if the average compressive stress in the plane normal to the borehole is approximately equal to one-half of the unconfined compressive strength of the rock. At this threshold stress level, the thickness of the disks that are formed will be roughly one-fourth of the core diameter. As the magnitude of the stress field increases above the threshold stress, the disks will become thinner (shorter in the axial direction). At yet higher stress levels, the disks become very thin and may have sharp edges. Empirical relationships have been developed that allow stress field estimation from rock strength parameters and disking that occurs in deep holes under triaxial loading conditions (Obert and Stephenson, 1965). Core disking, when observed in exploration holes from the surface or in drillholes underground, should be taken as an indication that stress levels are high relative to rock strength and that ground control problems may be anticipated.

Explosive Fracturing (Aggson, 1982): The orientation of the maximum and minimum normal compressive stress in rock can be determined near a rock surface without the need of absolute stress measurement by one of the methods described previously. This can be done by drilling a hole to a depth that is below the critical crater depth and then detonating an explosive charge in the borehole. If the stress field components are of a biaxial nature, a fracture will propagate in the direction of the maximum normal compressive stress. The direction of fracture propagation is determined by the principle of least work. This process has been shown consistently to produce fractures that are in line with the measured maximum compressive stress (Hooker, Nicholls, and Duvall, 1964).

10.3.3.3 Stress Change Measurement

Investigations into the stability of underground structures often require the evaluation of stress changes. Stress change measurements are easier to perform than absolute measurements since overcoring is not required. They are often more difficult to interpret, however, due to uncertainties regarding rock properties. All of the techniques described under the heading of "absolute stress determination" can be used for examining stress changes (Myrvang and Hansen, 1990). The borehole deformation gage, for example, can be placed in a borehole, and as stress conditions change, the change in shape of the borehole, as measured by the gage, can be used to directly calculate stress change in the plane normal to the hole. Specialized instrumentation and techniques have been developed that are intended for use in those situations where stress change is of concern.

Borehole Pressure Cells (Sellers, 1990): Hydraulic borehole pressure cells have been in use for many years for the measurement of rock stress changes in both elastic and viscoelastic (i.e., plastic) rock types. A great deal of the development and use was a result of efforts made by the US Bureau of Mines (Panek, 1961).

The standard borehole pressure cell (BPC) is manufactured from two steel plates welded together around their periphery. The plates are deformed into a "dogbone" configuration so that they can be expanded easily without damage to the welds (Fig. 10.3.40).

The space between the two plates is filled with deaired antifreeze solution and is connected via a high-pressure stainless

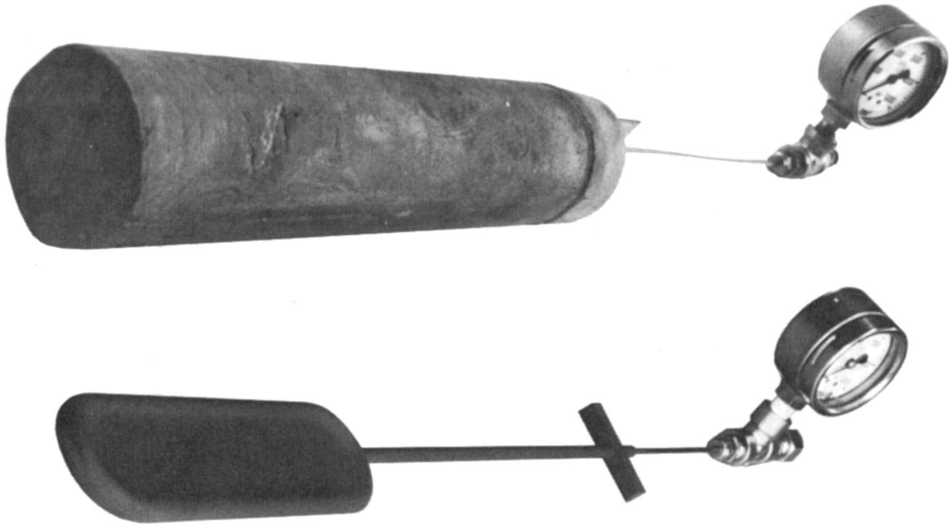


Fig. 10.3.40. Borehole pressure cell (*courtesy:* Geocon).

steel tube to a stainless steel pressure gage and/or a pressure transducer.

In use, the BPC is grouted inside a borehole drilled into the rock. The BPC is either pushed into a hole already filled with grout or grout is pumped in after the BPC has been positioned.

Alternatively, the BPC may be preencapsulated inside a cylinder of mortar of such size to permit it to slide inside the borehole and then to fit snugly when the BPC is pumped up.

Usually the BPC is pumped up until the initial hydraulic pressure is equal to the estimated rock stress. In elastic rocks, the relationship between observed fluid pressure change (as shown by the pressure gage) and the actual rock stress change can be calculated if a special pressurization technique is used and the elastic modulus of the rock is known (Sellers, 1970).

The BPC, because of its flat configuration, responds mainly to stresses acting at right angles to its plane. It is only slightly sensitive (about 6%) to stresses acting in the plane of the BPC.

In strongly biaxial stress fields, two BPCs oriented at right angles to each other in the same borehole will enable the principal stresses in the plane perpendicular to the borehole to be analyzed. In rocks which exhibit plastic creep, such as salt, potash, trona, etc., the BPC will measure not only rock stress changes but will also measure the actual in situ stress.

Vibrating Wire Stressmeter (Sellers, 1990): The stressmeter (Hawkes and Bailey, 1973; Hawkes and Hooker, 1974) consists essentially of a high-strength steel proving ring wedged tightly across one diameter inside a borehole drilled into the rock (Fig. 10.3.41). The distortion of the proving ring, caused by changing rock stresses, is measured by means of a vibrating wire that is tensioned across another diameter. Changes in rock stress cause changes in the resonant frequency of vibration of the tensioned wire. The two are related by means of calibration data supplied with each stressmeter.

The stressmeter is installed in boreholes up to 100 ft (30 m) long by means of a setting tool that is used to drive a wedge so that a platen is expanded against the side of the borehole (Fig. 10.3.42). Although diamond drillholes are preferable, those drilled percussively can be used. They should have their walls smoothed by incorporating a reaming shell with the bit.

The stressmeter behaves as a rigid inclusion in that the calibration varies by only a factor of 2 if the rock modulus varies

Fig. 10.3.41. Vibrating wire gage (*courtesy:* Geocon).

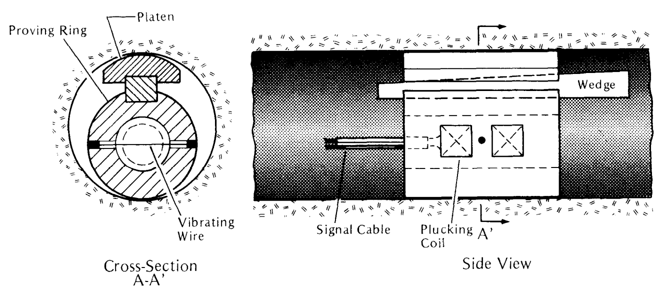


Fig. 10.3.42. Cross-section through vibrating wire gage (*courtesy:* Geocon).

by a factor of 10. The relationship between calibration factor and rock modulus is provided by the manufacturer. A thermistor can be incorporated in the stressmeter if temperatures are to be measured.

Several stressmeters can be installed in the same hole when both magnitude and orientation of the stress changes are re-

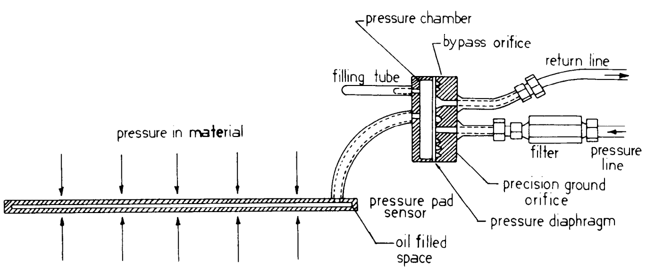


Fig. 10.3.43. Cross section through Glöztl cell (*courtesy:* Slope Indicator Co.).

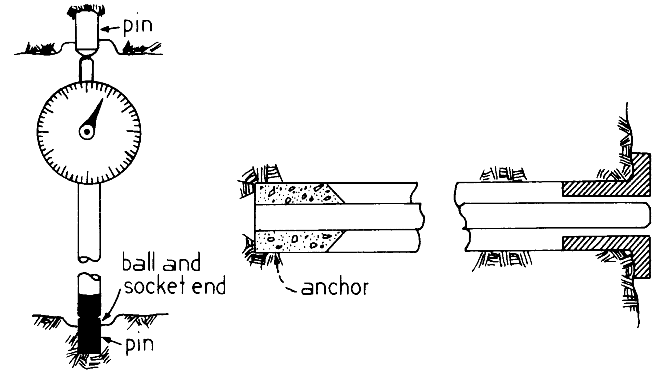


Fig. 10.3.44. Tube extensometer setup (after Obert, 1973).

quired. Special gages incorporating multiple vibrating wire sensors in one body are also available.

Glötzl-Type Pressure Cells: In underground mining, there are a number of applications in which it is of interest to measure ground pressure. Hydraulic-cemented rock fills are important ground control components in several mining systems. Although fill pressures can be measured using several different techniques, one of the most common devices is the Glötzl cell shown diagrammatically in Fig. 10.3.43.

All Glötzl cells utilize a unique bypass valve assembly which permits the isolation of the sensing system of the various cells from the measuring (readout) medium. Separating sensing and readout circuitry allows control of the sensing component characteristics to provide maximum response in diverse material types and applications.

The bypass valve's primary components are a stiff metal plate and a synthetic seal that is essentially undeformable. The plate supports the seal to create a diaphragm, which is loaded by pressure transmitted by the sensing system. When loaded, the diaphragm prevents circulation between the two sides (input line and return line) of the measuring system until the pressure introduced into the input line is sufficient to exceed the load pressure. When this slight excess pressure is reached, the diaphragm lifts slightly, permitting the measuring medium to circulate into the return line. The diaphragm will close the input-return line circuit at a pressure very closely approximating the pressure in the sensing system. Initial diaphragm lift approximates 5 µm, and the final pressure determination is made at a lift which approaches zero.

Glötzl earth pressure cells feature a standard thickness-to-width ratio of approximately 0.05.

Cells for the measurement of earth pressures are filled with oil, while those for the measurement of concrete stresses are filled with mercury. This ensures that the modulus of the cell assembly is higher than the modulus of the material surrounding the cell. The characteristics of the pad and the interaction of the pad and bypass valve assembly are such that the cell acts as a rigid inclusion in the pressure medium and thus responds directly to pressure changes in the medium.

The satisfactory operation of Glötzl cells requires that they be installed in a manner that (1) fully and continuously exposes them to material typical of the pressure medium, and (2) does not appreciably modify load or pressure conditions typical of the medium.

When in use, cells may be imbedded directly in the material in which measurements are to be made. In coarse fill, imbedment in a lens of finer-grained material allows proper load distribution of the measuring surfaces. Generally, the diameter of the largest grain bearing directly on the measuring surface should be 1/10 or less of the minimum plane dimension (length or width) of the cell.

Earth pressure cells may be installed in any attitude. They may also be installed in oriented arrays to provide data for complex pressure envelope analysis.

Readout facilities are usually remote from the cell locations, the cells being connected to the readout instrument by means of hydraulic tubes extending from the cell locations to the readout enclosure. The tube array for one Glötzl cell comprises an individual pressure (input) tube, and a zero pressure discharge (return) tube that may be shared in common by a number of cells.

This same principle has been used in pore water pressure cells and concrete stress cells.

10.3.3.4 Displacement Measurements/ Deformation Measurements

The deformeters described here are used for measuring both relative and absolute deformations produced by excavation, for example, roof-floor convergence, deformations in the walls, or structural parts of a mine. Included are optical levels and theodolites, as they can be used for deformation measurements over large distances.

Roof-Floor Convergence: *Tube Extensometer*—The tube extensometer (Haas, 1982; Obert, 1973) consists of telescoping tubes, a dial indicator, and contact seats on each end to measure the change in distance between anchor points in the roof and floor (Fig. 10.3.44). An initial zero reading is taken. Subsequent readings are taken at daily, weekly, or monthly intervals. The difference between one of the subsequent readings and the zero reading is the convergence during the intervening time period.

The indicator accuracy is approximately 0.001 in. (0.025 mm). The overall length of the telescoping tube is typically adjusted in l-ft (0.3-m) and l-in. (25-mm) increments by tapered pins that fit into a series of precisely spaced holes in the tubes. Instruments are available with measurement lengths of 3 to 6 ft (0.9 to 1.8 m), 6 to 11 ft (1.8 to 3.4 m), 11 to 17 ft (3.4 to 5.2 m), and 17 to 25 ft (5.2 to 7.6 m). Threaded hemispherical contact seats may be threaded into holes tapped into the heads of mine roof bolts, which results in an inexpensive anchor station. Floor stations must be protected by a pipe to prevent loose rock from contacting the sides of the bolt. The floor anchor points must be in solid rock or else the readings will be affected by movements of the floor due to subsequent blasting vibrations and possible movements of mine equipment near the station.

A necessary accessory for the tube extensometer is the master bar. The master bar is made of the same material as is the tube extensometer and is used as a reference standard against which the extensometer is checked periodically. The magnitudes of

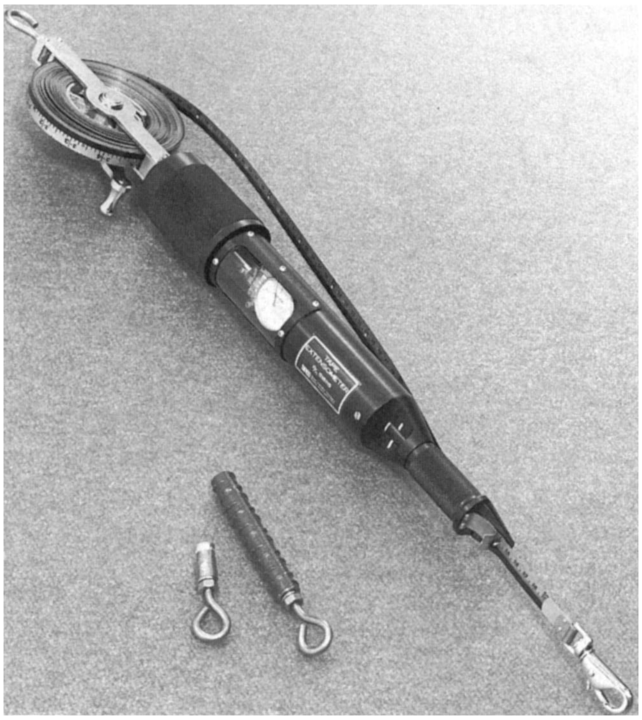


Fig. 10.3.45. Tape extensometer (*courtesy:* Slope Indicator Co.).

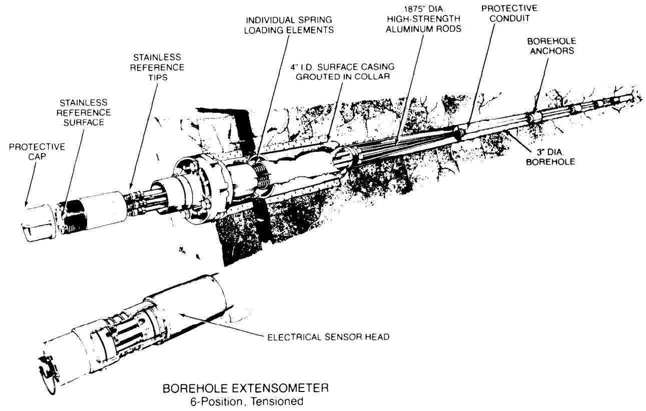
errors introduced by wear of the contact seats and tampering with the zero setting of the dial indicator are easily measured with the master bar. Errors due to temperature are not determined with the master bar, however. A thermometer on the side of the extensometer should be read at each measurement location in the mine. A temperature correction can then be made if necessary.

The two principal applications of the tube extensometer in underground mining are (1) to measure the convergence between roof and floor as a function of time (rate of convergence gives an indication of stability and instability of the roof), and (2) to measure separations between roof layers by placing anchors at several horizons in the roof.

While these two applications appear quite simple in concept, the technique gives reliable data that are easy for the mine engineer to interpret. Increasing rates of convergence or bed separation give warning of an approaching unstable condition so that preventive action may be taken to stabilize or withdraw from an area. On the other hand, convergence data showing little or no change between readings give some assurance that the opening will remain stable.

Tape Extensometer—The tape extensometer is a portable instrument which is hand-held while making readings (Fig. 10.3.45). It uses a steel engineer's tape with punched holes to measure changes in distance between two points with precision and speed. Measurements are made in any direction—vertical to horizontal—between the two reference points. The changes in distance between these two points over a period of time can be monitored with accuracy, reliability, and repeatability. These features are important to an instrumentation program measuring relative convergence of tunnels, mine roof sag, deformations in excavations and structures, or ground movements.

Tape tension is controlled by means of a compression spring. To adjust the tape tension, the knurled collar is rotated until the scribed lines are precisely aligned. This alignment causes a known force to be exerted on the tape.



Multiple Position Borehole Extensometer

Fig. 10.3.46. Components of borehole extensometer system (after Boisen, 1982).

The dial caliper often used as a readout has a resolution of 0.001 in. (or 0.025 mm).

The alignment and positioning of the anchor points for tape extensometer measurements are not critical. The hook fittings on the tape extensometer are not affected by the direction angle of the tape, or the positioning of the anchor.

Borehole Extensometers: *Mechanical Extensometer*—Measurement of rock deformation is essential to stability and behavioral monitoring of underground openings. The difficulties of obtaining adequate sample distribution, density, and orientation can frequently be overcome by the use of single- and multiple-position borehole extensometers (Boisen, 1982; Anon., 1978a; Windsor et al., 1987).

In its simplest form, a borehole extensometer consists of an instrument head, usually anchored in the collar of a borehole, and a number of in-hole anchors, each of which is fixed in place at a known distance from the collar. As rock adjacent to the borehole deforms, the component parallel to the borehole direction causes a distance change between adjacent in-hole anchors, as well as a distance change between individual anchors and the instrument head. These changes are measured by means of a mechanical link (usually wire or rod) extending to the instrument head from each anchor. The resulting data can be used to compute the distribution, magnitude, rate, and acceleration of deformation in the material intersected by the extensometer's borehole.

Borehole extensometers are made up of at least the following five principal components (Fig. 10.3.46): the instrument head, the collar anchor assembly, the downhole system of anchors and rods or wires, the signal cable if the head is equipped for electronic readout, and the readout apparatus, whether mechanical or electronic.

Extensometers are designed to measure axial strain components in the instrumented drillhole. As the drillhole is axially deformed, the distances between individual down-hole anchors and the instrument head change, as do the distances between adjacent anchors. The distance changes measured by a multiple position borehole extensometer, for example, are reduced to functions of time, hole depth, anchor position, or other parameters of significance in terms of the structure under investigation.

In order to establish an optimum plane of reference, the boreholes should be long enough to extend into rock zones (Fig.

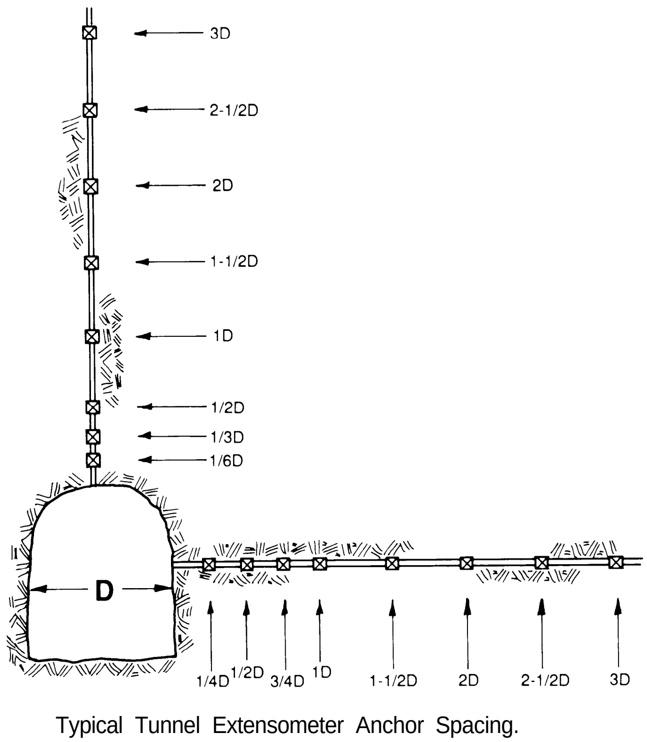


Fig. 10.3.47. Borehole extensometer anchor layout (after Boisen, 1982).

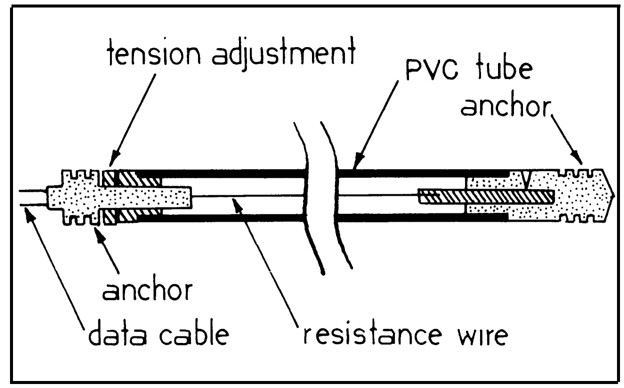


Fig. 10.3.48. Diagrammatic representation of resistance wire extensometer (*courtesy:* Carlson/RST).

10.3.47) in which the stress field is in an original state (i.e., where the stress field is relatively unaffected by the presence of the structure). Assuming that one or more of the deepest down-hole anchors is in substantially stable ground, the instrument head is analogous to a yo yo moving up and down, and in or out, at the end of a wire or rod fixed in place at a stable anchor. For interpretive purposes, the stable anchor occupies a point that is fixed, or relatively fixed, in space, and in relation to which the instrument head is displaced. The shallower anchors, on the other hand, are located in rock zones exhibiting varying degrees of instability, and may themselves be undergoing displacement at varying rates and in varying directions, relative to the instrument head, the stable anchor, and adjacent anchors. An analysis of the reduced data permits the identification of zones that are undergoing strain at varying rates and in varying directions in relation to the structure, and the measurement of axial strain rates and magnitudes.

Since the equipment is designed to measure rock strain components acting in an axial direction with respect to the borehole, the boreholes should be oriented to measure axial components that are meaningful to the problem or structure under investigation. In tunnel instrumentation, the normal orientation is in a plane transverse to the long axis of the tunnel. In fault-zone and slide-plane instrumentation, holes may be oriented either transverse to the principal planes of movement to measure separation or at a small angle to the principal planes of movement to measure a component of translatory displacement.

Data processing may consist of the reduction of field data to indicate displacement of the instrument head relative to each down-hole anchor, the displacement of each down-hole anchor relative to the deepest anchor, and rock strain acceleration either between adjacent down-hole anchors (and between the shallowest down-hole anchor and the instrument head) or between the deepest down-hole anchor and the instrument head, etc.

Intermediate computation steps involve the calculation of anchor-anchor displacement and anchor-anchor strain. Other parameters, such as anchor-instrument head strain, anchor-instrument head strain rate, anchor-instrument head strain acceleration, and displacement of each anchor relative to the instrument head, may be calculated as required.

Resistance Wire Extensometer—A resistance wire extensometer (RWE) is a simple extensometer (Anon., 1990m) for monitoring rock movement (Fig. 10.3.48). It basically consists of an electrical resistance wire element situated inside a 3- to 6-ft (l- to 2-m) long, PVC hollow tube. The wire is attached, under mild tension, to each end of the tube and is configured to have a resistance of 120 ohms. Thus configured, the RWE can be monitored with a resistance meter, conventional strain gage measuring equipment, or a data logger. No additional displacement transducer is required to convert movement of the extensometer into a proportional electrical signal. An RWE is installed by grouting into a percussive- or diamond-drilled borehole. A number of RWEs can be installed end-to-end into a borehole by attaching them to a suitable carrier (such as PVC conduit) to facilitate installation. The resistance wire is pretensioned within the hollow tube enabling up to 0.5% compressional strain to be measured. However, up to 18 to 20% tensile strain can be measured, with high sensitivity. Due to the hollow tube construction, the RWE has low sensitivity to moderate displacements across a borehole.

Magnetic Extensometer—In this design (Fig. 10.3.49), ring magnets sliding on a central access tube and within a collapsible outer tube are fixed in the ground at locations where movement is to be monitored (Anon., 1990j). These magnets may be installed in a borehole or placed subsequently during construction of earth works. Location of the targets is accomplished by passing a reed switch probe through the access tube. When the probe enters the magnetic field generated by the target, an audible signal is emitted at ground level. Measurements made with a steel tape may then be related to any convenient datum. The targets are designed to move independently of the access tube and within the protection of the outer collapsible tube. Features of this type of extensometer are

1. Reliable, accurate, simple to install and read.
2. No limits to number of points that can be located in the one borehole, with little extra cost.
3. Can be extended simply.

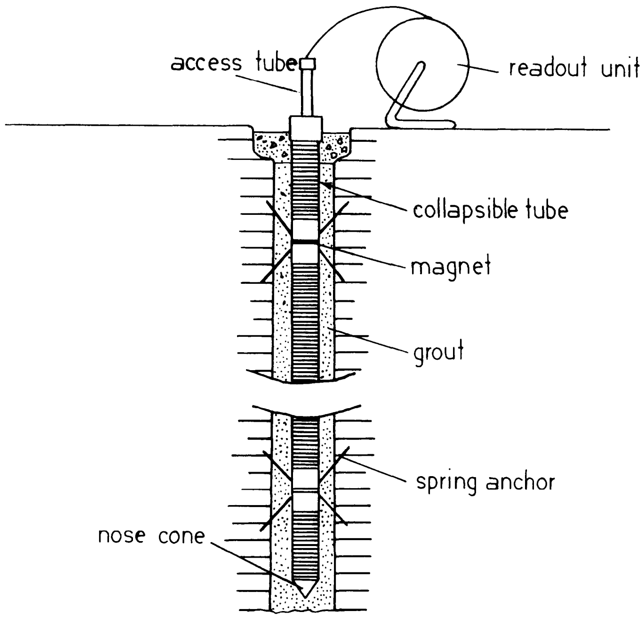


Fig. 10.3.49. Diagrammatic representation of magnetic extensometer (*courtesy:* Geotechnical Systems).

Fig. 10.3.50. Sonic probe extensometer (*courtesy:* Geocon).

4. Only one probe required for many installations.
5. Very economical.

Sonic Probe Extensometers—The advantage of sonic probe extensometers (Sellers, 1990) lies in their ability to provide electronic readout with maximum simplicity at minimum cost (Fig. 10.3.50). They are particularly suited to the measurement of rock movements around mines and underground openings.

Rigid sonic probe types use a portable sonic probe in the form of a "wand." The probe can be inserted into each extensometer head in turn to measure the relative position of a series of magnets attached to the ends of the measurement rods leading to each of the borehole anchors. A readout box, when connected to the probe, displays movements of the magnets. Various types of borehole anchors are available. The sonic probe may be left permanently in position for remote readout.

A probe 10, 20, or 25 ft (3, 6, or 7.5 m) long is inserted inside a guide tube that is located along the borehole axis and that passes through from 1 to 20 snap-ring, or hydraulic, borehole anchors. A magnet is located within each anchor. The position of each of the magnets relative to either the head of the extensometer, or to a neighboring anchor, is measured and displayed directly on the readout box.

The principle of the sonic probe relies on the magnetostrictive properties of the probe material. An electrical pulse in the probe creates a magnetic field that interacts with the field of a permanent magnet located either in a borehole anchor (flexible probe) or on the end of a rod attached to the borehole anchor (rigid probe). The change in the magnetic field generates, in the probe material, a torsional stress pulse that travels along the probe at the speed of sound. The arrival time of the stress pulse at the end of the probe is detected and becomes a measure of magnet position. Changes in position of as little as 0.001 in. (0.025 mm) can be detected. A switch determines whether movements are measured between each anchor and the mouth of the borehole or between each succeeding pair of anchors.

The maximum cable length between readout box and probe should not exceed 800 ft (244 m). Distances greater than this require the use of an in-line repeater to boost the signal.

Differential Sag Stations (Haas, 1982): The measurement system shown in Fig. 10.3.51 was developed by the Mine Safety and Health Administration (MSHA). Several spring anchors are installed at various depths in a single drillhole in the roof of the mine. Movement of the anchor is transmitted to the hole collar by a wire and is measured relative to the collar anchor. A 5-lb (2.3-kg) weight is used to keep the wire taut as the measurement is being taken. A dial indicator with 0.001-in. (0.025-mm) graduations measures between a brass button attached to the wire and a brass pipe cap secured to the anchored collar pipe. The system has the advantage of having several measurement points in one hole.

The two principal applications of the wire-type differential sag station in underground mining are

1. To measure strata separations in a mine roof as a function of time. Usually one anchor is placed quite deep, say 15 ft (4.6 m). Other anchors are placed at 2, 4, and 8 ft (0.6, 1.2, and 2.4 m), or as desired to span expected separation planes.

2. Measurement of roof deflection when, because of unstable floor conditions or vehicular traffic, one cannot install floor stations for use with a tube extensometer. In this application, the uppermost anchor should be placed above the zone of influence of the opening. The zone of influence above the opening normally extends a distance of at least one opening width.

Precision Leveling: Targets such as shown in Fig. 10.3.52 can be suspended from the roof of a drift on the ball-headed measuring bolts. This facilitates precision level surveys. If the same type of bolts are installed in the floor, the convergence surveys can also be run.

Stope and Drift Profilers: Over the past several years there have been some remarkable developments in surveying instruments (see Chapter 8.2) (Anon., 1989d; Anon., 1990e; Anon., 1990h; Anon., 1990i; Darling, 1988). Today there are instruments that can provide profiles of both drifts and stopes without the need for reflecting targets other than the rock wall itself. The obvious application is for improved mine surveying. However, as stope dimensions are increased, stability evaluation and control will require the use of such instruments on a routine basis. Stope roofs and pillar walls can be examined to determine their locations, dimensions, and gross deformations.

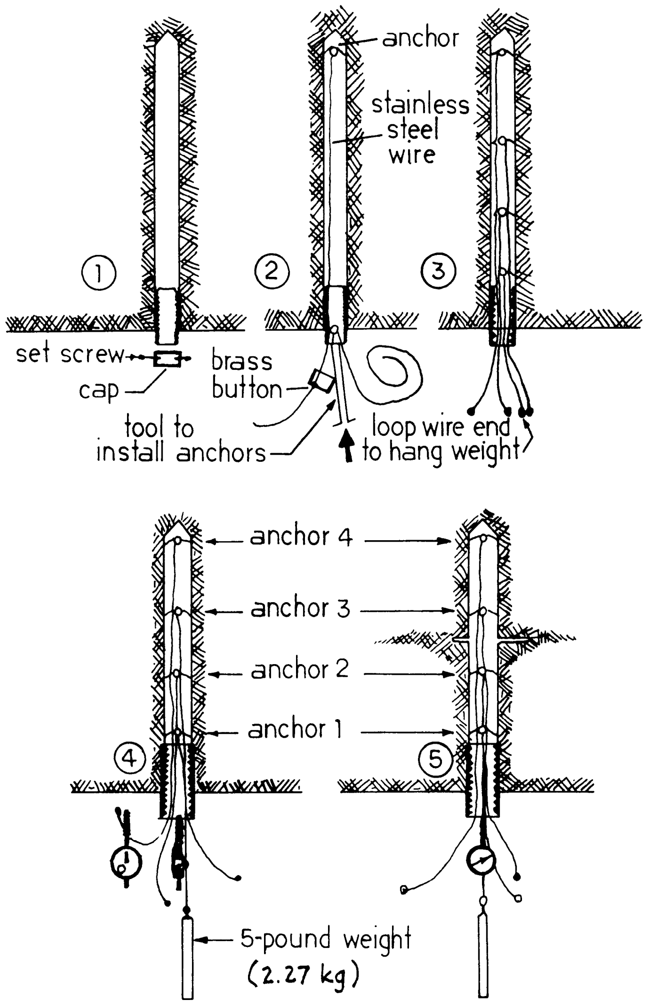


Fig. 10.3.51. Differential sag measurement system (after Haas, 1982).

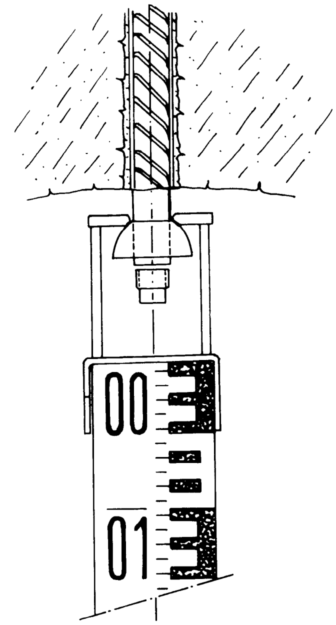


Fig. 10.3.52. Precision leveling (*courtesy:* Interfels).

Fig. 10.3.53. Distance measuring system using two laser beams (*courtesy:* Afgen).

In one technique, developed specifically for mine use (Anon., 1990e), two laser beams are emitted from the measuring unit. One beam is fixed and the other movable. The movable spot is adjusted to coincide with the fixed line at the point to be measured. The distance is read off from the instrument scale. By using the horizontal and vertical circles, outlines can be plotted.

The accuracy for this very inexpensive system, shown in Fig. 10.3.53, is dependent on the preset operating range. Some typical published values are given below:

Preset range		Typical accuracy at full range	
(ft)	(m)	(in.)	(mm)
1.5 to 15	0.5 to 5	±0.2	±5
3 to 30	1 to 10	±0.5	±10
6 to 60	2 to 20	±1.5	±40
10 to 100	3 to 30	±4	±100
15 to 150	5 to 50	±10	±250

Recent results for the 5- to 150-ft (5- to 50-m) range have suggested accuracies of the order of ± 1 in. (± 30 mm). For very rapid distance measurements, where the plotting facility is not required, the unit may be hung from a strap around the user's shoulders.

Other units of varying levels of sophistication and prices are available from several manufacturers.

10.3.3.5 Visual Inspection in Boreholes (Emmanuel, 1990)

There are several methods for visual examination of boreholes (Anon., 1990f; Anon., 1990g).

Fiberoptic Borescopes (Flexible): These instruments are manufactured in lengths to 200 ft (60 m). Some can be equipped to provide a scan capability at right angles to the shaft axis. These devices transport illuminating light to the distal or insertion end and present an image to the operator that is transmitted through a spatially correlated bundle of glass or quartz fibers. In the longer lengths, diameters generally run about 0.4 in. (10 mm). Cost of these devices typically runs between $500 and $1000/ft ($1500 to $3000/m).

The flexible fiber optic borescope is relatively portable, requiring only a light source as additional equipment. High-powered sources at about $3000 are recommended at long lengths, and cameras may be adapted to the eyepiece for recording purposes. Color video cameras are probably of marginal utility due to low light conditions at the eyepiece. Resolution is limited to the pixel count of the image bundle which is typically about 10,000. The instruments are relatively delicate and should not be torqued or kinked.

Video Borescopes (Flexible): Currently available instruments include devices of 0.25-in. (6-mm) diameter to lengths of 50 ft (15 m). These systems carry a charge-coupled-device sensor in the distal end that is sequentially illuminated with red, green, and blue reflection from the scene. This light is transmitted through the instrument shaft. The resulting fields are combined in a digital processor and presented as color video. Prices are typically about $1000/ft ($3000/m).

The video borescope typically offers improved resolution, over that of the fiberoptic technique, with 30,000 pixels a typical resolution. The optical path is very short, thus providing much greater latitude as regard lighting conditions. The video format permits all kinds of recording capabilities, and the video screen allows group observation. It is, however, both expensive and "high tech."

Geometric Borescopes (Rigid): The long-established rigid borescope technology, which transmits light through a carefully designed series of refractive (lens) elements, enables observation to depths of about 65 ft (20 m). These instruments are constructed in sections 3 to 5 ft (1.0 to 1.5 m) in length. Light is furnished by a halogen lamp which is carried in the head and powered through the body of the scope. They are relatively heavy, and unwieldy at long lengths. Cost is typically about $300/ft ($1000/m).

The rigid extendable borescope provides excellent resolution, is modest in cost, but is difficult to use at lengths exceeding the dimensions of the work space, since the operator must unscrew the assembly to add additional 3- to 5-ft (1.0- to 1.5-m) extension sections. Lighting tends to become a problem with added length as the losses increase with the addition of every lens section. This may make it extremely difficult to recover data in video format.

Direct Video Imaging: The current generation of commercially available remote head video cameras has permitted the construction of direct imaging color video systems that easily operate up to insertion depths of about 100 ft (30 m). Units with head diameters of 1.10 in. (28 mm), remote focus, remotely controlled lighting, and a remotely switchable directions of view (forward and side) have been provided. Cost (for a 100-ft, or 30-m, system) is about $300/ft ($1,000/m). Longer reaches are attainable for small incremental cost increases; shorter lengths do not change significantly from the 100-ft (30-m) baseline numbers.

Direct video imaging offers an excellent compromise in all characteristics, since resolution is typically about 330 to 350 thousand pixels (400 line horizontal resolution), insertion depth is limited by cable length, which is comfortable at 100 ft (30 m) and may, with current technology, extend to twice that distance. Much longer distances are possible but require different transmission approaches if the system is constrained to fit the 1 l/4-in. (32-mm) borehole. Care must be taken with the cable since it is a complex assembly. The system shown in Fig. 10.3.54 features an environmentally sealed 1.1-in. (28-mm)-diameter optical head with remote focus and light control supported by a 100-ft (30-m) cable.

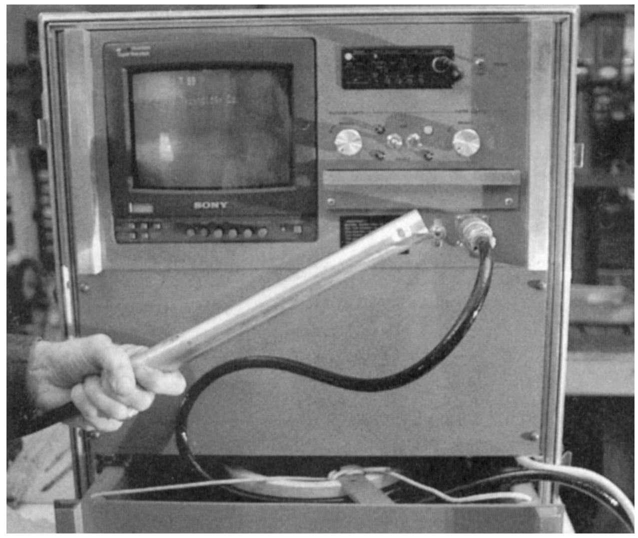


Fig. 10.3.54. Borehole video camera system (*courtesy:* Inspection Technologies).

10.3.3.6 Microseismic Monitoring (Brady, 1990)

Literally, a microseism is a "small earthquake." (The word is derived from the Greek words "micros," meaning small, and "seismos," meaning earthquake.) As detected in a mine, a microseism is sometimes an audible, sometimes subaudible, noise resulting from vibrations introduced into the mine structure by rock fracturing or by slippage along preexisting planes of weakness (joints, faults) in the rock mass.

The microseismic method to detect regions of structural instability and high-stress zones in underground mines was developed in the early 1940s by the USBM. Since that time, the microseismic method has been improved upon and it has been used throughout the world by mining and construction companies as well as by governments in assessing the stability of underground and surface excavations (Blake, 1982; Leighton, 1982; McMahon, 1988).

Microseisms or acoustic emissions are readily detected in mines by securely attaching velocity gages or accelerometers to the rock mass surrounding the area of interest within the mine. These are transducers that "listen" for rock vibrations, and they are known as "geophones." Acoustic vibrations travel through the mine at the speed of sound in rock, typically several miles (kilometers) per second, which usually varies along the path of the sound wave. When a rock noise is detected by several geophones, the origin of the noise can be calculated from relative arrival times of the signal at different geophones. Plots of event locations are superimposed on mine maps for given periods of time. Events in a given area are counted to produce rate plots. For example, the number of microseismic events per day may be plotted against the days of the month. An anomalous increase in the daily event rate may forecast a failure. However, mining progress must also be considered because there is normally more microseismic activity during drilling, blasting, and extraction than during nonmining cycles.

There are three types of microseismic monitoring systems currently in use. In the first type, a surface seismograph, similar to those used by the US Geological Survey in worldwide seismic networks, is installed with its sensor in a quiet area of the mine near the surface in competent rock. Its usefulness lies in allowing mine management to determine that a large event of an approximate magnitude occurred somewhere in the mine at a given time. For location purposes, an observer using only one surface seismograph can differentiate between an event in the host mine, an event in a nearby mine, or a distant earthquake.

A mine-wide system whose geophone spacings are at intervals of 1000 ft (300 m) or more is the second type of system. A typical mine-wide system will detect and locate, within 100 ft (30 m), events whose magnitudes are on the order of routine blasting or larger. These systems are useful for determining the approximate location of large events and what the physical mechanism is that is causing these events. An auxiliary use for a mine-wide system is the identification of unstable areas that warrant detailed investigation.

The third type of system is a local microseismic system where geophone spacings are as close as 30 ft (10 m). Such a network is designed to monitor a small section of a mine. In a mine with several working sections, complete coverage requires several local systems which can be networked such that data can be shared, effectively making a mine-wide system. These data can then be transmitted to central surface analysis stations via phone modem or satellite.

10.3.3.7 Remote Sensing (Peters, 1990)

Remote sensing data and analysis methods have proven to be useful for identification of zones of rock mass instability in both underground and open pit mines (see Chapter 4.4).

Lineaments are linear or curvilinear features of the earth's surface that are definable on aerial and satellite photographs and images. A number of researchers (e.g., Dimick and Barnum, 1978; Elder et al., 1974; Hylbert, 1981; Jansky and Valane, 1983; Milici et al., 1982; Overbey et al., 1973; Peters and Speirer, 1986; Peters et al., 1988) have demonstrated the ability to define geologic structures (faults, joints, and fracture zones) that extend to at least the depth of the mine (as deep as 2000 ft or 670 m) through use of lineament analysis. Figs. 10.355 through 10.357 provide an example of a site at an underground coal mine in central Utah where such structures have created instability in the roof rock. For this small mine, it was found that 82% (27 out of 33) of the ground-control problems (primarily roof falls, bad roof, and water inflows) were within the lineament-related, potential-hazard zones defined through interpretation of a Landsat Multispectral Scanner image (Peters et al., 1988). Although remote sensing analysis has proven useful in many cases, some researchers have had difficulty in using and/or verifying lineaments (e.g., Peng and Haddad, 1981; Moebs and Sames, 1987). Therefore, lineament analysis can be useful as an indication of degree of risk for potential ground control problems, but such analysis is not a panacea and must be backed up by further surface and underground (where possible) geologic investigations and by monitoring of mine and rock mass conditions over time.

Some work also has been done using remote sensing analysis for surface mine stability and subsidence monitoring. For example, Stahl (1983) gives examples of the use of aerial photography for lineament analysis for identification of structures that can destabilize pit walls in uranium mines. Structures or other unstable areas so identified then could be instrumented to focus monitoring efforts on the areas most likely to have slope stability problems. Lineament analysis of a Landsat Thematic Mapper image also was used to define areas of potential adverse breakage and material property changes (such as clay and iron content) in a propant and glass sand quarry in western Wisconsin. Advances in the spatial resolution of satellite systems (e.g., the 30-ft, or 10-m, ground resolution with stereo capability of the French SPOT satellite imagery, and the 15-ft, or 5-m, or better resolution of the Soviet KFA-1000 satellite photography) hold promise for the use of these systems for slope monitoring of surface mines or of glory holes above caving mines.

Modern photogrammetric techniques and instruments, in conjunction with aerial photography, have been shown to be usable in place of ground surveying to monitor subsidence over coal longwall mines (Cousins, 1986). Using aerial photographs from 1980, prior to mining of the underlying panel, and from 1984, following mining of the panel and during subsidence activity, and known control points from surface monuments surveyed over the same period, Cousins was able to derive an elevation stereomodel (through use of an analytical stereoplotter) of the subsiding area for both 1980 and 1984 that matched the ground surveys to within 2 in. or 50 mm. The Office of Surface Mining Reclamation and Enforcement subsequently approved the use of this subsidence monitoring technique, in place of repetitive ground surveys, for a few coal mines on an experimental basis. The method promises to be a less costly monitoring method for rugged and relatively inaccessible sites where ground surveys would be time-consuming and overly labor intensive.

Current and improving Global Positioning Satellite (GPS) techniques of vertical and horizontal location determination also offer the possibility of continuous, automatic monitoring of slopes and subsidence. This technique takes advantage of a carefully arranged array of Navy satellites to determine the latitude, longitude, and elevation (relative to the orbit of the satellite) of a GPS ground station to within as little as 0.5 in. (10 mm). Automated GPS stations (probably solar-powered) would allow both accessible and inaccessible sites to be monitored for ground movement with minimal labor.

10.3.4 SUMMARY

As in all measuring programs, the overall effectiveness of the instrumentation depends primarily on the care with which the instruments are installed and the measurements made, processed, and interpreted.

The distribution of instruments in each measurement station, and the exact nature of the instruments, is also an extremely important consideration and one that usually involves a considerable amount of judgment based on good geologic and engineering information. As a rule, an appreciable degree of redundancy should be designed into the measurement program whenever and wherever possible. In other words, each measurement should serve not only to provide specific information, but also to at least partially substantiate some other measurement. The end result should be an accumulation of measurements, which, by virtue of their redundancy, comprise an aggregate of information to which a high intrinsic level of practical value both for construction and for information relative to the subsequent operation and maintenance of the completed structure.

In conclusion, it should be noted that measurements without purpose are without value. The value of the data obtained can be realized only in proportion to the extent that the data are ultimately applied to the interpretation or solution of actual problems.

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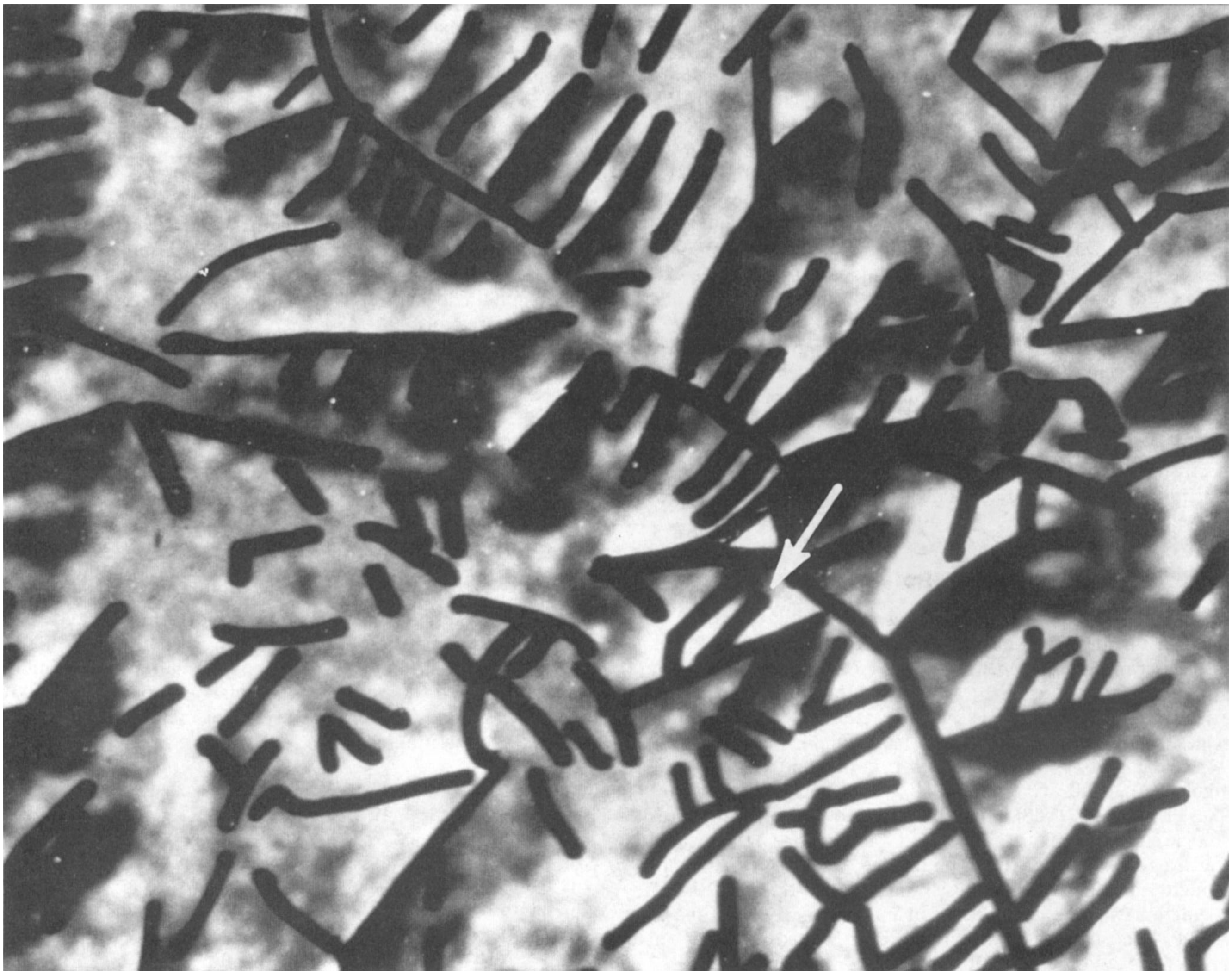

Fig. 10.3.55. Close-up view of portion of Landsat Multispectral Scanner Image of central Utah and Wasatch Plateau Coal Field showing lineaments (dark lines) interpreted from the image. North is approximately the top of the picture (after Peters, 1990).

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Fig. 10.3.56. View of hillside and valley, looking west-southwest, at location of arrow in Fig. 10.3.55. The lineament indicated by the arrow in Fig 10.3.55 is a joint-controlled cliff parallel to, but above, the cliff face in the upper right of this picture (after Peters, 1990).

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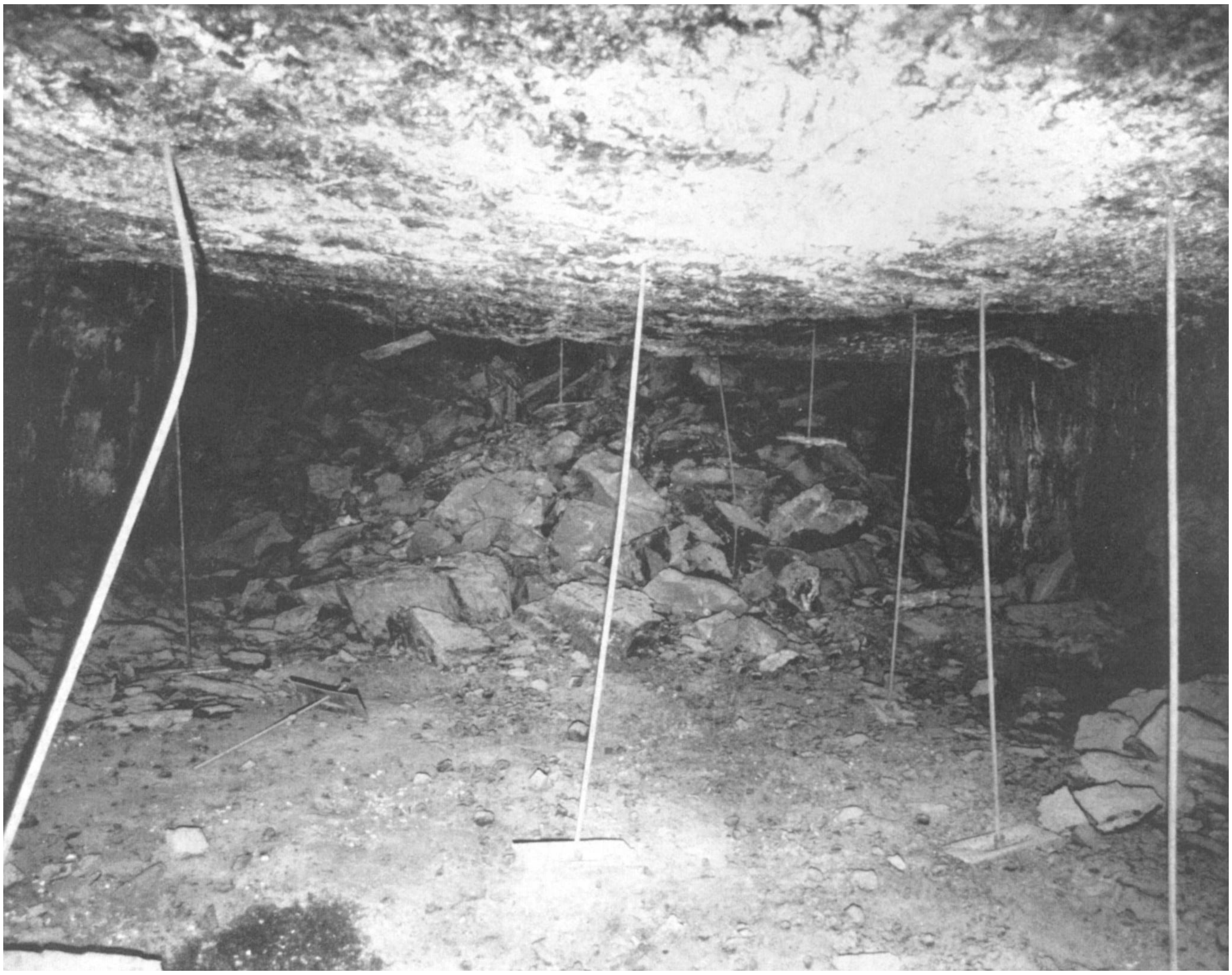

Fig. 10.3.57 View of area of coal mine directly under lineament indicated by arrow in Fig. 10.3.55. This section of the mine has experienced severe bad roof. A growing roof fall extends approximately 100 ft (30 m) back from area seen here (after Peters, 1990).

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Chapter 10.4
SLOPE STABILITY

RICHARD D. CALL

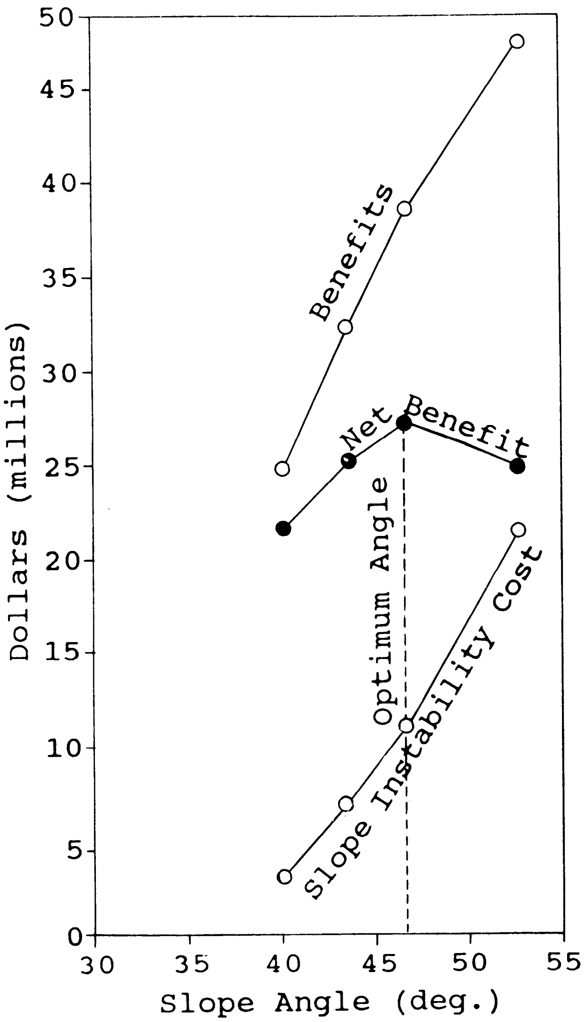


Fig. 10.4.1. Cost benefit curves.

10.4.1 INTRODUCTION

10.4.1.1 Design Approach

In the design of a typical open pit, increasing the slope angle decreases the stripping and/or increases the recoverable ore, which produces a higher benefit or return on investment (Fig. 10.4.1). However, increasing the slope angle decreases the stability of the slope. Because of the variability of geologic structure and rock properties, there is not a unique angle below which there is no slope instability and above which massive failure occurs. More typically, as the slope angle is increased, the number, size, and movement rate of slope failures increases. These slope failures result in operating costs such as the expense of cleaning up failed material, lost production, and unrecovered ore.

At steeper slope angles, the cost of slope instability increases more rapidly than the benefits. Thus the net benefit curve obtained by subtracting the cost of instability from the gross benefit has a maximum. The slope angle at which this maximum occurs is the *optimum angle,* since mining at a flatter angle results in higher stripping costs and reduced ore recovery. Conversely, mining steeper than the optimum results in slope instability costs greater than the increased ore recovery.

Slope design is the process of determining this optimum angle for input into pit design. The slope stability portion of slope design is the prediction of the slope instability as a function of slope angle.

10.4.1.2 Stability Criteria

From the standpoint of simple mechanics, the *stability of a slope* is the ratio of the strength of the material to the stresses in the slope. If the stress exceeds the strength, the slope is unstable; conversely, if the strength exceeds the stress, the slope is stable. This ratio is termed the *safety factor* and has been the basis for stability analysis in civil engineering for many years. Because of the variability of rock properties, uncertainty in the measurement of these properties, and the influence of quasi-random events, such as earthquakes and rainfall, the stresses and strengths used in stability are estimates of populations with significant distributions rather than single values. For this reason, safety factors greater than one have been used for slope design. An alternate approach to defining stability is to use the reliability method, whereby the probability of whether or not a slope will be stable is calculated from the distribution of input values.

Slope instability does not necessarily mean slope failure from the operational standpoint. It is not uncommon for a slope to become unstable, with the resulting displacement being less than 3 ft (1 m). Whether an unstable slope results in significant cost to the operation depends on the rate of movement, the type of mining operation, and the relationship of the unstable material to the mining operation. Unstable areas with displacement rates of over 4 in. (100 mm)/day have been successfully mined by truck and shovel operations. On the other hand, a few inches (millimeters) of displacement of the rock under a crusher, conveyor, or building may require extensive repair. When the rate of displacement is such that it disrupts the operation or the movement produces damage to mining facilities, it is considered an operational slope failure. A similar economic concept was used by Varnes (1958) to distinguish between creep and landslides. He restricts the lower limit of the rate of movement of landslide material "..to that actual or potential rate of movement which provokes correction or maintenance."

10.4.1.3 Safety

Another aspect of slope stability is *slope management.* In an optimized slope, some slope failure can be expected but the specific location and time of instability cannot be predicted with any certainty. Also the stability analyses utilized in design, with very few exceptions, are static solutions that do not provide

Table 10.4.1. Comparative Approximate Fatality Rates (per 10^6 hours of exposure)

Highway travel		1.9
Air travel		2.4
Motorcycle travel		4.4
Cigarette smoking		2.6
Open pit mining		0.42
Falls of rock	0.01	
Runs of muck, stockpiling, etc.	0.03	
Fall of person	0.05	
Vehicle accident	0.10	
Miscellaneous	0.23	
Logging		0.94
Construction		0.26

Source: Coates, 1977

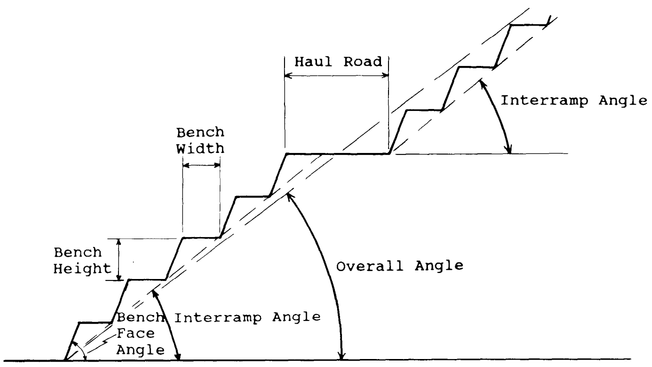


Fig. 10.4.2. Typical design cross section.

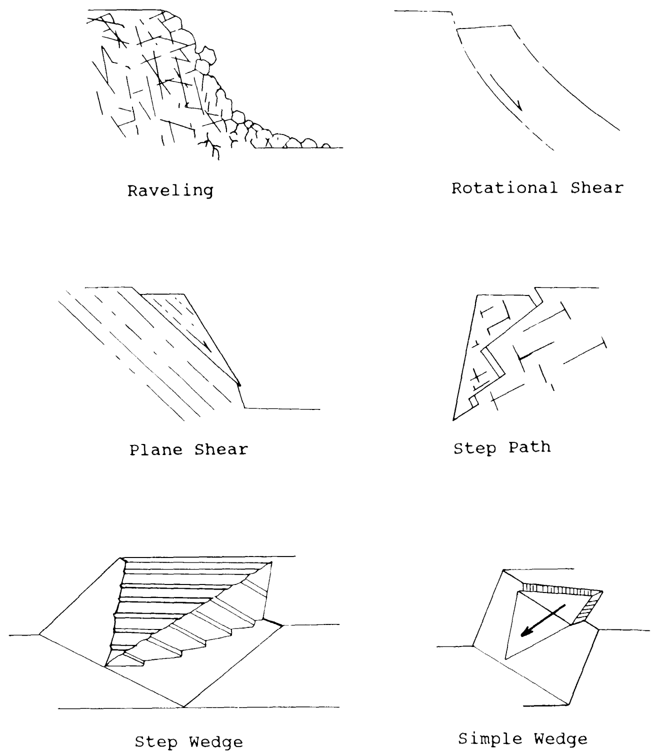


Fig. 10.4.3. Typical failure models.

estimates of the rate or magnitude of displacement. Therefore, to provide safe working conditions and minimize the economic impact of slope instability, there should be a program of displacement monitoring to provide advance warning of major slope displacement, accompanied by design of remedial measures. In spite of the uncertainty in slope stability, the safety record has been excellent compared with mining in general and other activities (Table 10.4.1). With an appropriate slope management program, it should be possible to mine steep slopes with an equal or greater safety record.

10.4.1.4 Slope Geometry

There are three major components of a pit slope: bench configuration, interramp slope, and overall slope (Fig. 10.4.2). The *bench configuration* is defined by the bench face angle, the bench height, and bench width. The *interramp angle* is the slope angle produced by a number of benches. Where there are haul roads, working levels, or other wide benches, the *overall slope angle* is the angle of the line from the toe to the crest of the pit; the slope angle will be flatter than the interramp angle. It is important in slope design to consider these components. For example, in the case of bedding dipping into the pit at 40°, the daylighting plane shear criteria would result in a design angle of 40°. If this angle were used for the overall slope angle, haul roads cut into the slope would undercut the bedding and result in interramp instability. In addition, there would be almost no catch benches left, as the bench face angle would be steeper than the bedding.

10.4.2 INSTABILITY MODELS

In order to make a quantitative estimate of the stability of a slope, analytical models amenable to mathematical solutions must be used. The requirements of these models are the failure geometry and assumptions regarding material properties and stress distributions. (In the following discussion the term "failure" is used for simplicity and to be consistent with prior usage.)

10.4.2.1 Geologic Model

The rock of a slope can be considered to consist of the following components.

Intact Rock: The primary unbroken rock as determined from a piece of core cut for compression testing. The term rock substance has also been used for the unbroken rock.

Fractures: The geologic structures such as joints, bedding, foliation, and minor faults that break the intact rock into more or less discrete blocks. *Discontinuity* is a term that is also used for fractures.

Rock Mass: The combination of intact rock and fractures considered as a unit. Soil could be considered a special case of rock mass.

Major Structures: Geologic features such as faults that are large enough to be mapped and located as individual structures. There is actually a continuum between fractures and major structures, but the differentiation is useful for design purposes.

10.4.2.2 Sliding Block Geometries

The sliding block failure mode refers to a situation in which displacement occurs along one or more geologic structures, and the failure mass is considered to be a rigid block or a number of blocks. These geometries are shown in Fig. 10.4.3.

Plane Shear: The plane shear is the simplest geometry consisting of a single plane striking nearly parallel to the slope. The structure must have a dip flatter than the slope angle (daylighted) and must be long enough to reach the surface or a tension crack. Since the stability analysis is two dimensional, the width of the failure must be great enough that the end results are negligible or there are boundary structures that define the lateral extent of the failure.

Step Path: The step path geometry occurs where there is a fracture set dipping into the pit in the plane shear orientation, but no individual fracture is long enough to form a plane shear geometry. Sliding is assumed to occur along fractures in the plane shear orientation (the master joint set) and separation along fractures approximately perpendicular to the master joint set or tensile failure of the rock between the master joints.

Wedge: The wedge failure geometry is the result of two planar geologic structures intersecting to form a detached prism of material. Sliding can occur down the intersection or on one plane with separation on the other plane. In some cases the sliding on one plane will be a rotation rather than simple translation.

Step Wedge: The step wedge is similar to the simple wedge except that one or both of the failure surfaces are step paths.

Two-block: The two-block is a two dimensional plane shear geometry where there are two plane shear structures dipping into the pit, with a third structure dipping back into the wall that divides the failure into an active and a passive block.

Slab: Where there is bedding or foliation parallel to the pit, slope instability can occur even though the structures are not daylighted. The possible failure mechanisms are crushing at the toe, a two-block geometry formed by joints at the toe, and buckling.

10.4.2.3 Nonplanar Failure Surfaces

Rotational Shear: In a soil or weak rock mass slope where there are no geologic structures that control the failure, the most unstable failure surface is approximately a circular arc. The radius and location of the most unstable circle (the critical circle) depends on the material properties and must be found by iterative solutions of trial circles. The stability of the circular arc is usually analyzed by the method of slices. The failure is divided into a series of vertical slices so that the failure surface can be approximated by planar segments. The driving forces and resisting forces on the failure surface at the base of the slice, as well as the interslice, forces are summed up over the slices. The method of slices is normally a two-dimensional analysis.

General Surface: The general surface is a mixed mode failure in which part of the failure surface is structurally controlled and part is failure through the rock mass. An example would be a nondaylighted plane shear. The method of slices can be used to analyze the stability of the general surface.

10.4.2.4 Other Models

Block Flow: Compared with underground rock mechanics, the stresses in a pit slope are low and do not exceed the rock mass strength. Thus most slope instability is controlled by geologic structure. However, in deep pits, there is the possibility that the stresses in the toe of the slope would be sufficient to result in the crushing failure of the rock mass, particularly if there was a high horizontal stress. This mode of instability was referred to as *block flow* by Coates (1981).

A conceptually possible variation of the block flow would be a situation where the rock mass under confinement in the slope wall yields plasticly. The resulting deformation would be plastic flow such as occurs in a glacier. At the surface of the pit slope, where there is no confinement, secondary sliding block failure would occur similar to the calving of a glacier. This is a possible explanation for situations where instability occurs in a relatively flat slope, and the back analysis indicates an anomalously low shear strength.

Toppling: Where there are steeply dipping structures that result in blocks with a large height-to-thickness ratio, the *toppling* failure mode has been postulated. For toppling to occur, the center of gravity of the block must be outside the toe of the block. Therefore, sliding or crushing of the toe must occur before toppling is initiated unless the slope is mined steeper than 90º. Because of this, toppling is most commonly observed as a secondary failure mechanism resulting from displacement caused by another mode of instability.

An exception to this generalization is where ice wedging or pressure from water-filled cracks causes toppling, as in the case of the Hells Gate Bluffs failure in Fraser Canyon, British Columbia (Piteau et al., 1976).

Rockfalls and Raveling: Bench faces are normally cut as steeply as the loading equipment can dig them. As a result, individual blocks in the face are at or close to limiting equilibrium, and disturbing forces can dislodge them. The primary disturbing forces are freeze/thaw and water from rainfall. The action of these disturbing forces can dislodge individual blocks, producing a rockfall. The dislodging of large numbers of blocks is termed *raveling.* Weathering can also produce raveling by the deterioration of the material supporting the blocks. Although in principle, the stability of individual blocks could be analyzed, there is no practical method of conducting stability analyses for raveling on a pit scale. The design approach is to provide for adequate catch benches.

10.4.3 STRESSES IN A SLOPE

Although most stability analyses assume simple gravitational body loading to calculate the stress on a failure surface, it is recognized that the actual stress magnitude and orientation is affected by the in situ stress field, the geometry of the pit, and the variation in material properties.

10.4.3.1 In Situ Stress

Simple gravitational loading would produce a vertical stress equal to the weight of the overlying material, and according to elastic theory, the horizontal stress would be a function of the vertical stress and Poisson's ratio. For the common value of 0.25 for Poisson's ratio, the horizontal stress would be 1/3 the vertical stress. Measurements of in situ stress in underground mines have demonstrated that the horizontal stress can be greater than the vertical stress, as a result of active or residual tectonic stress. The horizontal stress is not equal in all directions, either. In the absence of in situ stress measurements or other indications of a high horizontal stress, the most reasonable assumption is that the horizontal stress is equal to the vertical stress.

10.4.3.2 Slope Geometry

There is a stress concentration at the toe of a slope that is a result of the deflection of stresses around the toe. A high horizontal stress produces a greater toe stress than simple gravity loading. The effect of in situ stress and slope geometry for a plane strain analysis is shown in Fig. 10.4.4. It should be noted that the toe stress is much more dependent on the pit depth and the ratio of horizontal to vertical stress than on the slope angle.

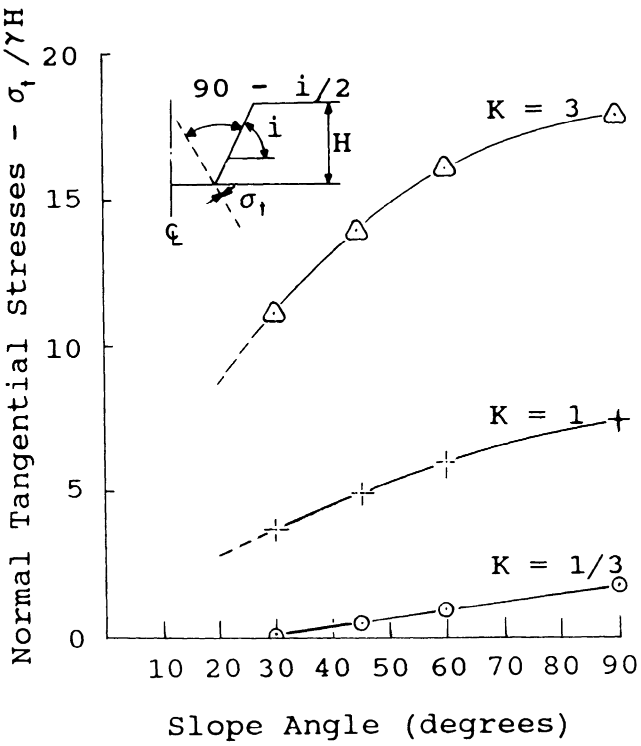


Fig. 10.4.4. Variation in plane strain of the toe stress.

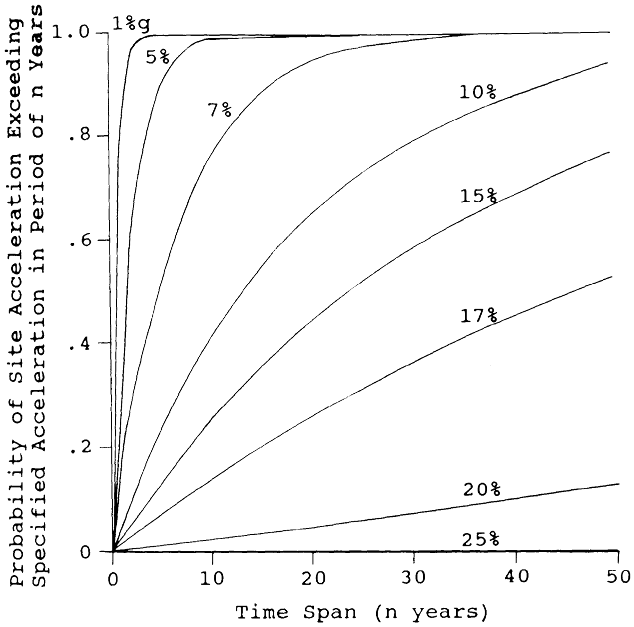


Fig. 10.4.5. Site acceleration probabilities as a function of time. Attenuation calculated according to Patwardhan (1978).

10.4.3.3 Material Properties

Finite element analyses have shown that major stress concentrations can be produced where there are rocks of differing stiffness in the slope. Stiffer rock units carry more load and thus have a stress concentration. Of particular concern for slope stability is the development of high shear stresses in the vicinity of the contact between rocks of differing stiffness.

10.4.3.4 Seismic Acceleration

The shock wave from an earthquake exerts a temporary additional stress on a slope that can cause instability. This has been demonstrated by the number of landslides triggered by earthquakes (Glass, 1982), although this record is misleading with regard to rock slopes; as saturated soil slopes are subjected to liquefaction, which would result in much greater displacement at lower seismic loading. Thus it is appropriate to include the affect of dynamic stresses in the stability analysis of slopes.

The classic method of including the effect of earthquakes in stability analysis is the pseudo-static approach whereby the maximum site acceleration that could be produced by an earthquake is input into the stability analysis as a horizontal force. This approach is excessively conservative when applied to pit slopes, for the reasons listed in the following.

Probability of Occurrence: The maximum earthquake may have a very low probability of occurrence during the critical exposure time of a pit slope. Although the life of a pit may be 20 years or more, the maximum height and angle only exists at the end of the mine life. Therefore, when analyzing the stability of the final slope, the exposure time for that slope geometry is only a few years.

For the cost-benefit approach to slope design, a probabilistic risk analysis of seismic dynamic loading can be used. From the historical earthquake record, a time history of acceleration at the site can be obtained by using empirical attenuation relationships to convert the magnitude and distance for each earthquake to a site acceleration, and calculating the probability of occurrence of site accelerations for time intervals with Gumbel's extreme value theory (Fig. 10.4.5).

Slope Response: The response of a specific failure geometry is a function of the frequency and duration of the ground motion as well as the maximum acceleration. Even if a failure geometry is unstable at an acceleration below the maximum, resulting from ground motion, the amount of total movement may be only a few inches (tens of millimeters). Although a pseudo-static stability analysis would give a safety factor less than one, the displacement would be below the range of what would be considered an operational slope failure.

To estimate the displacement resulting from seismic acceleration, the linear acceleration dynamic response (LADRS) technique developed by Glass (1982) can be used. In this technique the displacement is calculated for small time steps using a digitized model accelerogram and the displacement summed over the duration of the accelerogram. The failure criteria can be expressed as a maximum permissible displacement specific to the slope situation being analyzed. For slopes without facilities such as crushers or conveyors, a maximum displacement of 8 to 12 in. (200 to 300 mm) would be appropriate. Where facilities are present, the displacement tolerance of the structure would be the criteria.

10.4.4 DATA COLLECTION

Collecting adequate and appropriate data for stability analysis is a key aspect of slope design. Obtaining incorrect results from slope stability analysis is predominantly the result of failing to analyze the critical failure mode or not having the suitable estimates of the input parameters such as rock strength or the geometry of geologic structures. With the use of computers, our ability to construct mathematical models and perform the calculations exceeds our ability to collect adequate input data for the models. There are two aspects to the problem of data collection: sampling and measurement.

To illustrate the problem, let us take the specific task of determining compressive strength. This is usually measured by conducting a compressive test on a cylinder of rock 2 to 3 in. (50 to 75 mm) in diameter and 5 to 8 in. (125 to 200 mm) in length. The population of interest (referred to as the target population by statisticians) includes all the cylinders in the volume of rock that could be involved in slope instability. Based on slope behavior and stress considerations, this volume would extend one pit depth back from the design pit and one half the pit depth below the bottom of the design pit. It is obvious that all of the target population could not possibly be tested, so the strength distribution must be estimated by inference, using the test results from some small, hopefully representative, fraction of the target population.

The availability of samples for testing is determined by access, which would be the ground surface, the pit wall, underground workings, and drillholes. Where there is no preexisting pit or underground workings, and the ground surface is covered by alluvium, access to samples is restricted to drillholes. These accessible samples are referred to as the sampled population. The samples that are actually collected and tested are referred to as the sample population. To make valid statistical inferences about a population, every member of the population in question must have an equal likelihood of being sampled, and the tested samples must be an unbiased representation of the population. This can be true regarding the sampled population, but making the step from the sampled population to the target population is more difficult. Because of the restricted access, not all members of the target population are available for sampling even if a specific rock mechanics drilling program is conducted, as there are surface topographic restrictions on locating drill sites, and holes may not be completed because of bad ground. As pointed out by Cochran et al. (1954), "No statistical processes can make the step from sampled population to target population. It can only be done by judgment, intuition, and subject matter knowledge."

Sample disturbance and the difficulty of reproducing the in situ field conditions introduce measurement uncertainties. In the case of rock mass strength, the sample size required makes direct testing prohibitive. Indirect methods, such as modeling the rock mass by compositing the intact rock strength and joint strength, or a rock mass classification with subsequent correlation to empirical behavior are generally employed.

In the case of geologic structure data collection, parameters such as orientation, length, and spacing are geometric rather than scalar, and cannot be measured at a point. This results in a window problem, particularly in the case of fracture length. If the fracture is larger than the observation window, such as a bench face, the length cannot be directly measured. This is why surface mapping is preferable to drillhole data where the core diameter is the window. There is also an orientation bias, as a linear sampling window such as a drillhole does not intersect fractures parallel to the window.

Data collection should be well organized, with specific objectives regarding the use of the data and the quantity required (Table 10.4.2). Collecting data for data's sake should be avoided, as it will result not only in information that is not used, but the possible omission of information needed. Ongoing data reduction is important in order to determine whether a sufficient quantity of appropriate data is being collected.

10.4.4.1 Geology and Major Structure

Conventional geology provides the distribution of rock types and alteration, and the location of major structures. Geologic data should be in the form of a surface map, cross sections, and level maps. It is preferable to have two sets of documents—the factual sheets that show only the actual observations and a set of interpreted maps and cross sections.

For the design of final pits, a geologic map of a trial pit design and cross sections normal to the pit wall should be constructed.

10.4.4.2 Rock Fabric

Rock fabric is the orientation, length, and spacing of fractures. These are the geometric attributes used in stability analysis and in characterizing the rock mass. On a pit scale, the number of fractures such as joints are too numerous to map. Fracture mapping, therefore, consists of measuring the attributes of a subset of the total fractures and characterizing the population with distributions of the attributes.

It has been found from detailed mapping that the orientation of fracture sets has a normal or bivariate normal distribution. Since the orientation is vector quantity, it is properly a spherical normal distribution. However, for the limited range of attitude for a specific fracture set (100), the simple normal distribution is adequate. In the case of folded rock, the poles of the bedding planes fall along a great circle.

The measurable aspect of joint size is the trace length, which is the intersection of the joint and the mapping surface. The negative exponential appears to be the best distribution for trace lengths on the basis of fit to mapping data and theoretical considerations. Models such as the circular disc and the Poisson flat have been postulated to describe joints in three dimensions. These models can be used to correct for the observation window limitation.

Several common mapping methods are available.

Fracture Set Mapping: This is a modification of conventional joint mapping where fracture sets are identified by eye, and the orientation, length, and spacing are recorded. If joints or other structure orientations have been recorded during regular geologic mapping, they can be compiled and used in slope design.

Detail Line: The detail line method is a systematic spot-sampling method in which a measuring tape is stretched along the bench face or outcrop to be measured. For all the fractures along the tape, the point of intersection with the tape, orientation, length, roughness, filling type, and thickness are recorded. To get an adequate representation of the fabric, 100 to 150 fractures need to be mapped. This is the least subjective method, as individual fractures are recorded, and it provides the most detailed length and spacing data. It is relatively inefficient, however, as more observations are made on closely spaced fracture sets than are required for adequate statistical representation.

Cell Mapping: In this method, mapping surfaces such as a bench face are divided into cells. Normally, the width of the cells is made equal to the height of the cells. Within each cell, the fracture sets are identified by eye, and the orientation, length, and spacing, characteristics are recorded. Cell mapping is a combination of fracture set mapping and detail line, with the efficiency of visual identification of fracture sets and some of the more rigorous measurements of detail line.

Oriented Core: To obtain subsurface fabric, oriented core can be used. In inclined holes excentrically weighted, imprinting devices can be used to determine the orientation. In vertical holes, a scribing technique coupled with a downhole compass must be used. Oriented core provides information on fracture

Table 10.4.2. Checklist for Preliminary Slope Stability Data

MAPS AND SECTIONS	
Regional (1:50,000 or USGS quadrangles)	
Surface topography	( )
Surface geology	
Outcrop	( )
Interpretation	( )
Typical geologic cross-sections	( )
Hydrology	
Drainage areas and surface flow	( )
Groundwater	
Water levels in drillholes	( )
Contour map of piezometric surface	
Mine Area (1:5000 or 1:10,000)	
Surface topography	( )
Surface geology	
Outcrop and drillhole locations	( )
Interpretations	( )
Geologic cross-sections	
Surface and drillhole data	( )
Interpretations	( )
Groundwater	
Water levels in drillholes	( )
Contour map of piezometric surface	( )
Pit (1:1000 or 1:2000)	
Surface topography	( )
Trial pit plans	( )
Geology	
Surface outcrop	( )
Surface interpretations	( )
Grid cross-sections	
Drill data	( )
Interpretations	( )
Radial cross-sections	( )
Drill data	( )
Interpretations	( )
Level maps	( )
Structure contour maps	( )
GRAPHS	
Precipitation	( )
Water levels in drillholes	( )
Stream flow	( )
RQD of drill core	( )
TABLES	
Surface fracture data	( )
Drill core fracture attitude	( )
RQD of drill core	( )
SAMPLES	
Type samples for each rock type	
Uniaxial compression (6 per rock type)	( )
Triaxial compression (6 per rock type)	( )
Disc tension (3 per rock type)	( )
Direct shear (3 per rock type)	( )
PHOTOGRAPHS	
Drill core before splitting	( )
General views of pit	( )
Typical outcrops	( )
Major structure exposures	( )
Aerial photographs	( )
OPERATING PITS SHOULD ADD THE FOLLOWING:	
MAPS AND SECTIONS	
Unstable areas (pit scale)	
Topography	
Before movement	( )
After movement	( )
Plan showing unstable areas	( )
Tension cracks and displacement vect	( )
Geologic cross-sections through slide	( )
Groundwater	
Surface seepage and water levels in drillholes and blastholes	( )
GRAPHS	
Displacement measurements	( )
Bench face angle vs. mine coordinates	( )
TABLES	
Displacement measurements	( )
Bench face angle data	( )
Blasthole data	( )
Production data	( )
Mining Costs	( )
Stockpile capacities	( )
Rock strength data	( )
PHOTOGRAPHS	
Unstable slopes	( )

orientation and spacing, but the length of fractures cannot be directly measured.

10.4.4.3 Rock Properties

Since the spatial variability of rock properties is large, the potential for sampling error is greater than the measurement error. For this reason, it is preferable to use simple test methods for a number of samples than to use an expensive precise method on one sample.

For the shear strength of fractures and fault gouge, the direct shear test is recommended as it is a simulator of field conditions. Since the shear/normal failure curve may be nonlinear, it is important to use normals that represent the expected range of normals for potential failure geometries in the slope. The tests at each normal should be run with sufficient displacement to obtain a residual shear strength, as the residual shear strength usually is a better estimate of in situ strength than the peak strength. To obtain the shear/normal relationship, a curve can be fitted to the shear normal values for a range of normals. Some stability analyses, such as the modified Bishop method of slices, require a linear failure curve of the classic relation,

$$S = c + n \tan \phi \tag{10.4.1}$$

where ϕ is friction angle, c is cohesion, n is normal stress, and S is shear stress. This is a linear failure curve and is often a good fit, particularly for fault gouge. The more general curve is the power with an intercept, that is

$$S = c + kn^m \tag{10.4.2}$$

where c, k, and m are constants.

Commonly, fractures have the simple power curve,

$$S = kn^m \tag{10.4.3}$$

The linear is a special case of the power with intercept where

$m = 1$, in which case k becomes tan and c is cohesion. The linear fit to an actual power curve can be an adequate predictor of shear strength except at low and high normals where the curves diverge. When using these strength estimates, it is useful to think of cohesion as a mathematical intercept rather than an intrinsic property of the material.

For intact rock, unconfined compression and Brazilian disc tension tests are recommended. In addition to obtaining the compression and tension strengths, the intact rock shear strength can be approximated by a fit to the tension and compression Mohr circles using the relationships,

$$\phi = \arcsin[(U\text{-}T)/(U+T)](0.85) \qquad (10.4.4)$$

$$C = 0.5T\tan\phi(1/\sin\phi + 1)(0.98) \qquad (10.4.5)$$

where U is uniaxial compression, and T is tensile strength.

The constants 0.85 and 0.98 are factors developed from comparison between triaxial testing and the simple linear fit to the uniaxial and disc tension strengths. For most stability analyses, the failure surface is not under high confinement, so triaxial testing is not necessary.

The uniaxial compression tests can be gaged to obtain the Young's modulus and Poisson's ratio for the intact rock.

Index tests such as the point load can also be used to evaluate the spacial variability of intact rock strength.

For the rock mass where direct testing is not possible, indirect methods such as the rock mass rating (RMR) classification and back analysis must be used.

10.4.4.4 Hydrology

Standard hydrologic procedures such as piezometers and pump tests can be used to obtain the current pore pressure distribution and the permeability for predicting changes in pore pressure with time and changes in pit geometry. Simple techniques, such as measuring the water level in drillholes, are effective procedures. Two factors need to be considered, however:

1. Water behavior in rock slopes is a fracture flow phenomenon, and porous media analysis, while useful at a regional scale, may be a poor predictor of pore pressure at pit slope scale.

2. The critical factor in slope design is the pore pressure rather than the quantity of water. A low permeability rock mass may yield very little water and appear "dry," yet have significant pore pressure.

10.4.4.5 Stress Measurements

The most cost-effective stress measurement techniques are overcoring methods such as the "door stopper" or the triaxial gage. Because of the practical limitation of most current overcoring techniques to hole depths of 100 ft (30 m), underground openings are needed to penetrate far enough into the slope to get away from the surface effects. Also overcoring is usually not successful where the rock quality designation (RQD) is less than 50%, which is often the case with deposits such as porphyry copper. Alternates to current overcoring techniques, such as hydrofracturing, have potential where overcoring is not feasible.

It is useful to conduct a finite element analysis using assumed stress to evaluate the mud for in situ stress.

Detailed measurement techniques are discussed in Chapter 10.3.

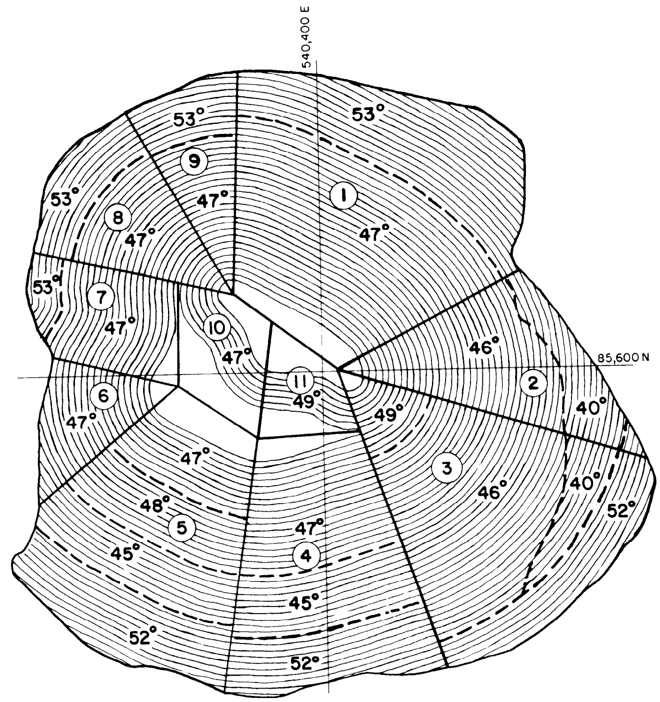


Fig. 10.4.6. Cuajone design sectors and recommended interramp angles.

10.4.5 DESIGN

Steps in slope design are the following:

1. Define design sectors.

2. Conduct a bench design analysis to determine the maximum interramp slope.

3. Conduct interramp design analysis using economic criteria for the selection of interramp angles.

4. Evaluate the resulting overall slope for potential instability, and modify the design if required.

Slope design is an interactive process as a trial pit is required to select design sectors, but the development of a trial pit requires slope angles.

10.4.5.1 Design Sectors

To conduct stability analyses and develop optimum slope angles for input into pit design, the proposed pit must be divided into design sectors that are sections of the pit with similar geologic and operational characteristics (Fig. 10.4.6).

The first criterion for the selection of design sectors is the structural domain, which is an area within which the rock properties and fabric are consistent. Typical structural domain boundaries are lithologic contacts and major structures which separate areas of dissimilar fabric.

The second criterion is wall orientation. Since rock is usually anisotropic, different wall orientations within the same structural domain can have significantly different modes of instability and different optimum angles. An extreme example of this is a dipping coal deposit where slab slides occur in the footwall at a slope angle of 18°, whereas 150-ft (45-m) high benches in the same lithologic sequence in the highwall orientation are stable at 70° with only minor step path raveling.

A third criterion for defining design sectors is operational considerations. Because of the higher cost of slope failure, sections of the pit wall that will contain in-pit crushers, conveyors, or haul roads require different stability criteria than the same wall orientation in the same structural domain.

Since a pit geometry is required to define design sectors, slope design is iterative with mine planning. A preliminary set of slope angles must be provided so that a trial pit can be developed. After the optimum angles are selected and a pit designed, the pit plan must be reevaluated to determine whether the design sectors need to be changed because of changes in the pit geometry.

For each of the design sectors, the rock fabric and major structure orientation data can be plotted on a stereographic projection. This diagram is used to determine failure modes and select structure sets for stability analyses.

10.4.5.2 Bench Design

Bench faces are normally mined as steeply as possible so that some bench-scale rockfalls and raveling can be expected. Thus it is customary, and in many cases mandated by mining regulations, that catch benches be left in the pit wall to retain rockfalls and raveling. Bench design is the process of conducting stability analyses to estimate the minable bench face angles, selecting the bench width, and, to a limited extent, the bench height. The bench height is controlled by the height of the mining levels, but it is possible to increase the height by leaving catch benches on every other level (double benching) or every third level (triple benching).

Based on an analysis of rockfall mechanics, Ritchie (1963) developed width and depth criteria for a ditch at the toe of a slope to protect highways from rockfalls. Falling rocks impact close to the toe of the slope, but, because of horizontal momentum and spin, can roll considerable distances from the toe. The concept of Ritchie's design was that the rock would impact in the ditch, and the side of the ditch would stop the horizontal roll.

It is not practical to excavate a ditch in an open pit catch bench, but a berm can be substituted for the ditch. A modification of Ritchie's design that can be used to determine the minimum bench width for bench heights from 30 to 100 ft (9 to 30 m) is

$$\text{minimum bench width} = 4.5 \text{ ft} + 0.2 \text{ bench height} \qquad (10.4.6)$$

with a minimum 4-ft (1.2-m) high berm on the edge of the bench.

Recent work with mathematical simulation of rockfalls has indicated that this criterion may be conservative, and the simulation method has the potential for more site-specific bench width criteria (Evans, 1989).

For a given bench height and corresponding bench width, the upper limit of the interramp angle becomes a function of the bench face angle. The bench face angle, however, is not a unique value, as variability of the rock fabric results in varying amounts of backbreak. Backbreak is defined as the distance from the design crest to the as-mined crest. Because of this variability, it is preferable to use a reliability approach rather than using the mean bench face angle. (Calculating an interramp slope using the minimum bench width and the mean bench face angle results in 50% of the benches being too narrow.) The procedure is to select a percentage reliability and use the cumulative frequency distribution of the bench face angle to find the angle where the percentage greater is equal to the reliability (Fig. 10.4.7). This gives the design bench face angle to use, with the minimum bench width and the bench height, to calculate the interramp slope angle.

The percent reliability represents the percentage of the bench along a given level that would be wider than the minimum required bench width to catch rockfalls. The reliability should be selected on the basis of the potential for rock fall and the exposure of personnel and equipment. For example, the catch bench in raveling ground above a haul road requires a greater reliability than catch benches in a stripping area with more competent ground. In practice, reliabilities from 60 to 90% have been satisfactory.

In an operating property, the actual bench faces can be measured, and the measured bench face angle distribution can be used in design. Where existing bench faces are not available, a bench-face angle distribution can be obtained by running a stability analysis of a vertical face. For this analysis, the plane shear, wedge, and step path analyses are run using the fracture data. The height analysis should be incremented in steps up to the bench height, and the resulting backbreak composited, as short fractures that would not result in full bench failure can still cause crest backbreak. This bench face angle distribution is referred to as the theoretical bench face distribution, as the effect of blasting and digging is not included. If there is a strong geologic control such as bedding or foliation, the measured and theoretical bench face angles are the same. Where no strong structure exists, the theoretical bench face angles should be reduced to include the effect of blasting. Based on comparisons that have been made between measured and theoretical angles, the reduction should be between 10 and 20°, depending on the controlled blasting to be used.

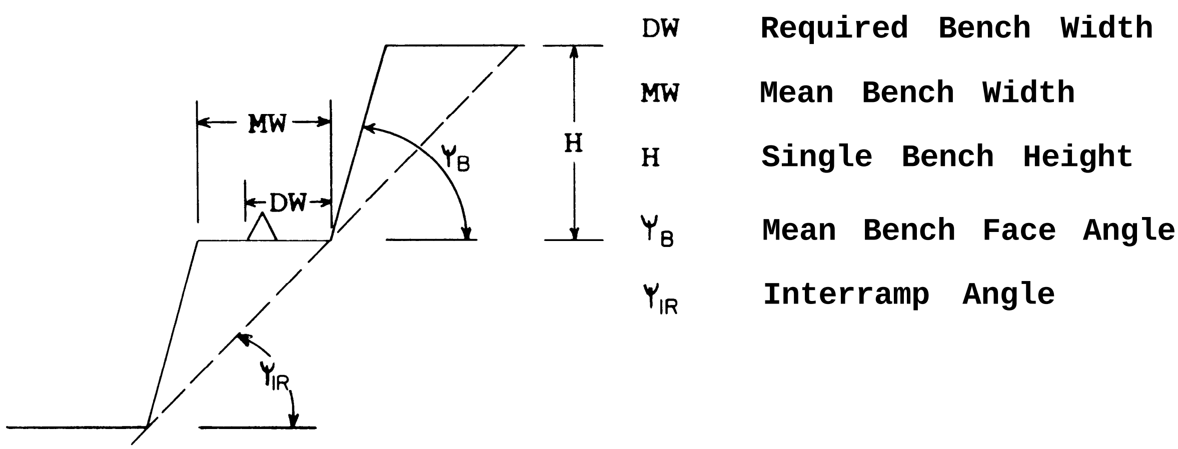
10.4.5.3 Interramp Design

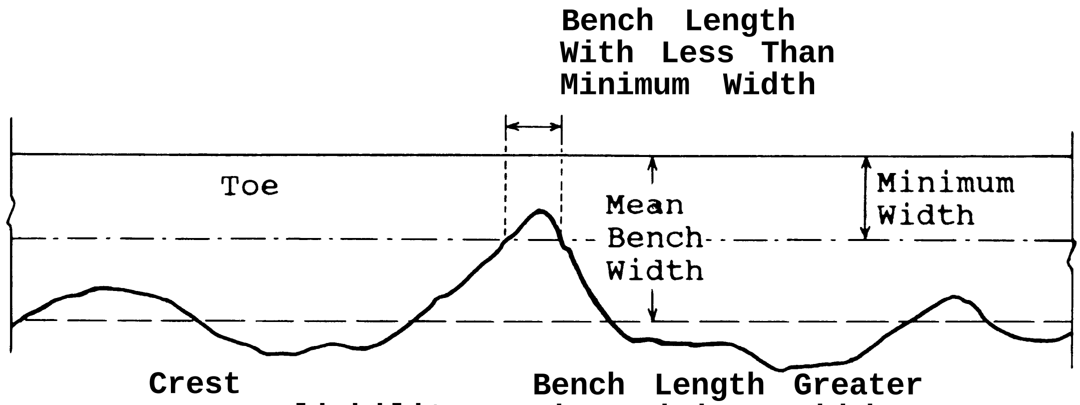
The stability of interramp slopes is primarily controlled by intermediate and major structure failure geometry. Where major structures can be specifically located in space, the geometry relative to the slope can be defined and a discrete stability analysis can be conducted. Commonly, however, the number of mapped structures is large and the distance between the mapping sites and the design wall is greater than the length of the structures. In this case, the structural data must be considered a statistical representation of the structures that will occur in the design slope, and a probabilistic analysis is required.

To obtain the input for stability analysis, the wall orientation can be plotted on a lower hemisphere stereographic plot of the poles of the fractures and the major structures. The fractures and major structures are sorted into design sets based on their orientation relative to the orientations for failure modes, as shown on Fig. 10.4.8; and the distribution of orientation, length, and spacing can be computed for the design set. These design sets may not correspond to geologic sets, although the boundaries of the sets may be adjusted to avoid splitting a geologic set. An advantage of this approach is that it is based on kinematic tests for viable failure geometry and makes it unnecessary to test all the structures for each failure mode.

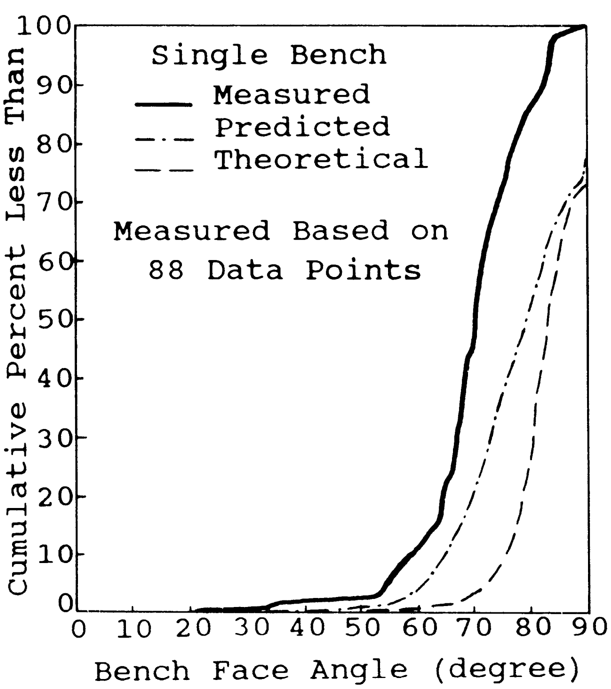
Major Structures: In the case of through-going major structures, where the geometry is known, a safety factor can be calculated for specific slope angles and slope heights using analytical models described in the references for the appropriate failure model. For a deterministic design, the slope angle with the desired safety factor would be selected.

In the reliability method, the probability of sliding can be calculated by Monte Carlo sampling of the shear strength distribution to obtain a distribution of safety factors; and computing the area of the safety factor distribution that is less than one (Fig. 10.4.9). Other techniques can be used, such as the point estimate method (Harr, 1984) or calculating the probability that





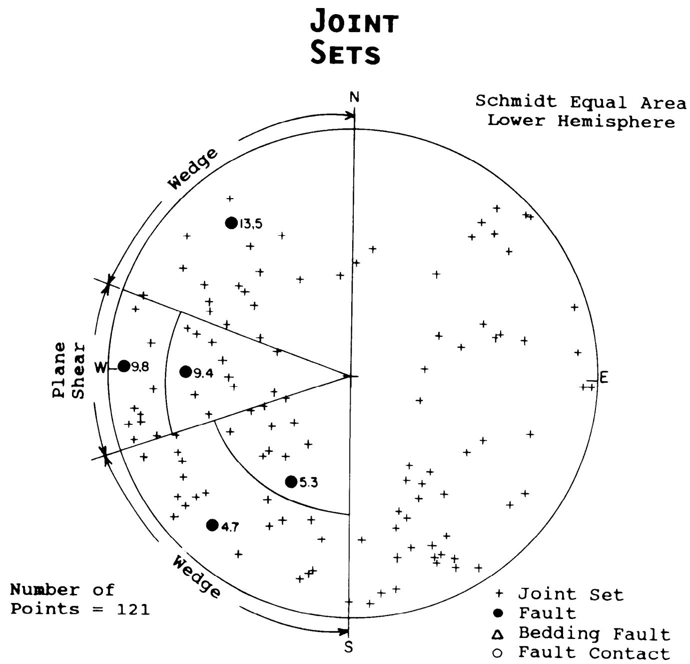
$$\text{Reliability} = \frac{\text{Bench Length Greater Than Minimum Width}}{\text{Total Bench Length}}$$



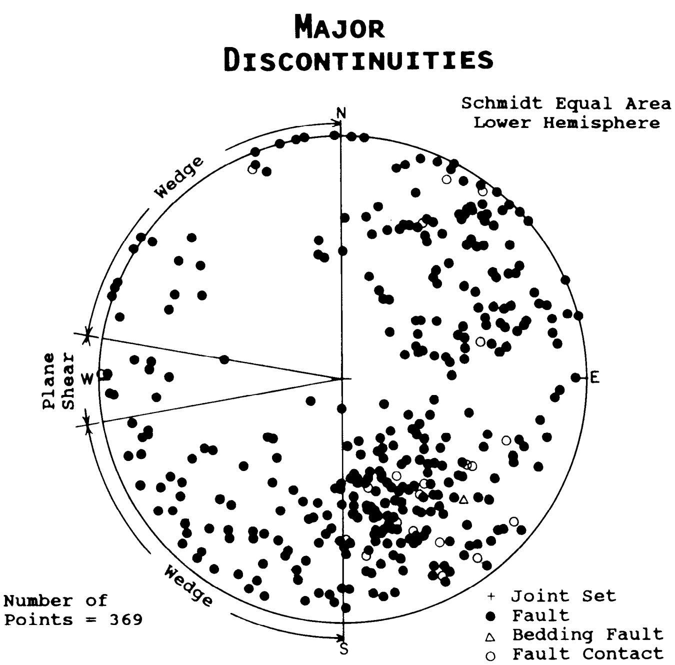
Single Bench Measured Bench Face Angles
SB Mean Face Angle: 69.9
DB Mean Face Angle: 69.9

Single Bench (Ht=50ft)			Double Bench (Ht=100ft)		
Slope Angle deg	% Reliability:Min. Bch Width of 25 ft percent	Mean Width ft	Slope Angle deg	% Reliability:Min. Bch Width of 35 ft percent	Mean Width ft
30	98	68.3	30	98	136.6
31	98	64.9	31	98	129.8
32	98	61.7	32	98	123.4
33	98	58.7	33	98	117.3
34	98	55.8	34	98	111.6
35	98	53.1	35	98	106.2
36	98	50.5	36	98	101.0
37	98	48.0	37	98	96.1
38	98	45.7	38	98	91.4
39	97	43.4	39	98	86.9
40	94	41.3	40	98	82.5
41	91	39.2	41	98	78.4
42	90	37.2	42	98	74.4
43	89	35.3	43	96	70.6
44	87	33.5	44	93	66.9
45	85	31.7	45	91	63.4
46	78	30.0	46	90	59.9
47	71	28.3	47	89	56.6
48	60	26.7	48	88	53.4
49	55	25.1	49	86	50.3
50	40	23.6	50	83	47.3
51	33	22.2	51	78	44.3
52	29	20.7	52	71	41.5
53	24	19.4	53	61	38.7
54	19	18.0	54	55	36.0
55	15	16.7	55	43	33.4
56	13	15.4	56	36	30.8
57	10	14.2	57	32	28.3
58	3	12.9	58	28	25.8
59	1	11.7	59	23	23.4
60	1	10.5	60	19	21.1

Fig. 10.4.7. Catch bench design. Conversion factor: 1 ft = 0.3048 m.



Design Set	Dip Direction Mean (degrees)	Dip Direction S.D. (degrees)	Dip Mean (degrees)	Dip S.D. (degrees)	Length Mean	Length S.D.	Spacing Mean	Spacing S.D.
4.7	38.1	22.3	71.7	10.2	6.7′	4.1′	.54′	.41′
5.3	52.7	13.5	34.6	7.2	4.8′	5.2′	.65′	.55′
9.8	86.1	13.9	79.6	5.6	3.9′	3.3′	.51′	.34′
9.4	91.1	12.9	45.5	11.5	5.2′	3.9′	.49′	.16′
13.5	133.7	20.3	52.6	13.6	6.8′	5.3′	.61′	.49′



Length Mean	Spacing (Mean) LW	Spacing (Mean) RW	Spacing (Mean) PS
233′	417′	1139′	2963′

Conversion factor: 1 ft = 0.3048 m.

Fig. 10.4.8. Design set determination.

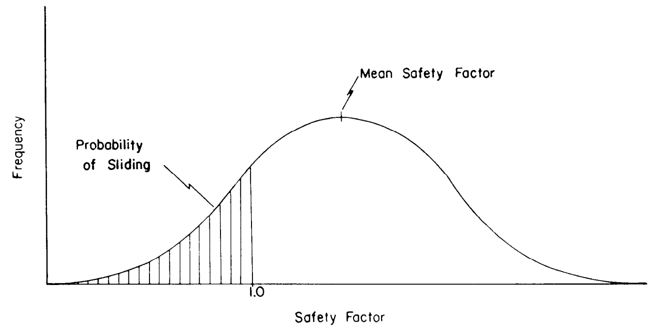


Fig. 10.4.9. Example distribution of safety factors used to calculate probability of sliding.

the shear strength is less that the strength required for a safety factor of one.

Because of the variability of the shear strength, a safety factor greater than one is used to reduce the risk of instability to an acceptable level. One problem with this is that a given safety factor will have a different level of risk depending on the dispersion of the input parameters. The advantage of the reliability approach is that it deals directly with the risk.

Failure Volume Estimation: Where the geologic structures compose a statistical population, the probability of failure for the single occurrence of a specified failure mode is a function of the probability that the structures exist and form a viable failure geometry, as well as the probability of sliding. The probability of existence is calculated from the orientation, length, and spacing of the structures.

To calculate the expected number of failures and the expected failure volume for input to a cost-benefit analysis, the probability of failure for the possible failure modes must be calculated for a range of heights and angles and then composited.

Fig. 10.4.10 is an example of the number of failures and failure volume as a function of slope angle for the design sector of a large pit.

Cost of Failure: Given the expected number of failures and the expected failure tonnage, the cost of slope failure can be estimated. Failure costs consist of cleaning up failure material, repairing haul roads, repair of facilities, lost production due to disruption of operations, the value of lost ore buried by a failure, and engineering costs. The method used to estimate failure cost is a "what if" mine planning procedure. A failure is postulated for a design sector, a plan for responding to the failure is made, and the cost of the plan is estimated. These exercises are useful whether or not a full cost-benefit optimization is done, as they can lead to modifications of the mine plan that will reduce the impact of slope instability.

10.4.5.4 Overall Slope

The overall slope is usually flatter than the interramp slope because of ramps or other step-outs. Thus the overall slope normally will be more stable than the interramp except for stress-induced failure or failure modes not analyzed for the interramp.

Block flow potential can be analyzed by the finite element technique. Finite element has been a time-consuming and expensive analysis in the past. However, with the faster computers and better software currently available, it has become more feasible. A quick check can be made for block flow potential using the charts developed by Coates (1981) (see Fig. 10.4.4). If the charts do not indicate block flow potential with any regional stress assumption, finite element is not needed unless there is a high contrast in stiffness between adjacent materials in the slope. The charts assume a homogenous material and would therefore not indicate stress concentrations produced by stiffness contrasts.

Changes in the overall slope angle have relatively little effect on the stress concentration at the toe of the slope, where a greater concentration could produce block flow. Therefore, block flow potential would not be a suitable method for selecting overall slope angles. A more effective design approach would be to design the slope based on other criteria, and to make provision in the mine plan for step-outs, if needed in the toe area of the pit to reduce the stress concentration produced by the notch effect of the bottom of the slope. The loss of ore from step-outs at the toe would have less economic impact than the amount of stripping required to have the same effect on block flow potential.

Rotational shear analysis should be run for the overall slope, even on rock slopes, to verify that it would not be a critical failure mode. Rotational shear would be a primary method of analysis for both interramp and overall slopes in alluvium and low rock-mass strength slopes such as soft coal measures.

The general surface analysis should be used for the overall slope to evaluate mixed mode failure types where part of the failure is structurally controlled and part is failure of low rock-mass strength. Nondaylighted wedge and plane shear failures in which the weak rock at the toe fails are becoming recognized as a more significant failure mode. This is in part because pits are becoming steeper and deeper, and partly because more pits have been designed for the simpler sliding block failure modes.

10.4.5.5 Slope Support

Ground support techniques such as cable bolting have not had wide application in open pit mining, although the effectiveness is well established in underground mining and in civil construction. One reason is the uncertainty of the ultimate pit geometry. Where the ultimate pit is defined by an economic cutoff, changes in the price of the commodity or in operating costs change the location of the pit slope. It is difficult to justify the expense of cable bolting when there is a reasonable chance that the there will be a new pushback and the bolts will have to be mined out. Corrosion is another problem with bolting, particularly in copper mines where acid mine water is very corrosive. In large pits where the bolt length would have to be in excess of 500 ft (150 m) to include potential failure surfaces, bolting would be difficult and expensive.

A current application of support, employed in Australia in particular, is the bolting of bench faces to reduce ravel and steepen the faces. By reducing raveling, catch benches can be made narrower, which can increase the interramp angle if multibench failure geometries are stable. In some cases, catch benches could be eliminated by using a combination of bolting and meshing to prevent raveling.

There are special situations where bolting is warranted. Even small-scale failures could damage in-pit facilities such as crushers and conveyors, resulting in expensive repair costs and a long period of lost production while the equipment is being repaired or replaced. In this case, bolting to improve the reliability of the slopes that affect the facility would be appropriate. An example of this is the bolting of the haul road in the Ertsberg Pit (Mealey and Nicholas, 1986). The haul road was the only access to ore, and a bench-scale wedge failure of the haul road would stop production. Because of the steep interramp slope, repair of the haul road would have been difficult, so the face below was cable-bolted to increase the reliability of the haul road.

The contribution of a bolt to the shear resistance is composed of three parts:

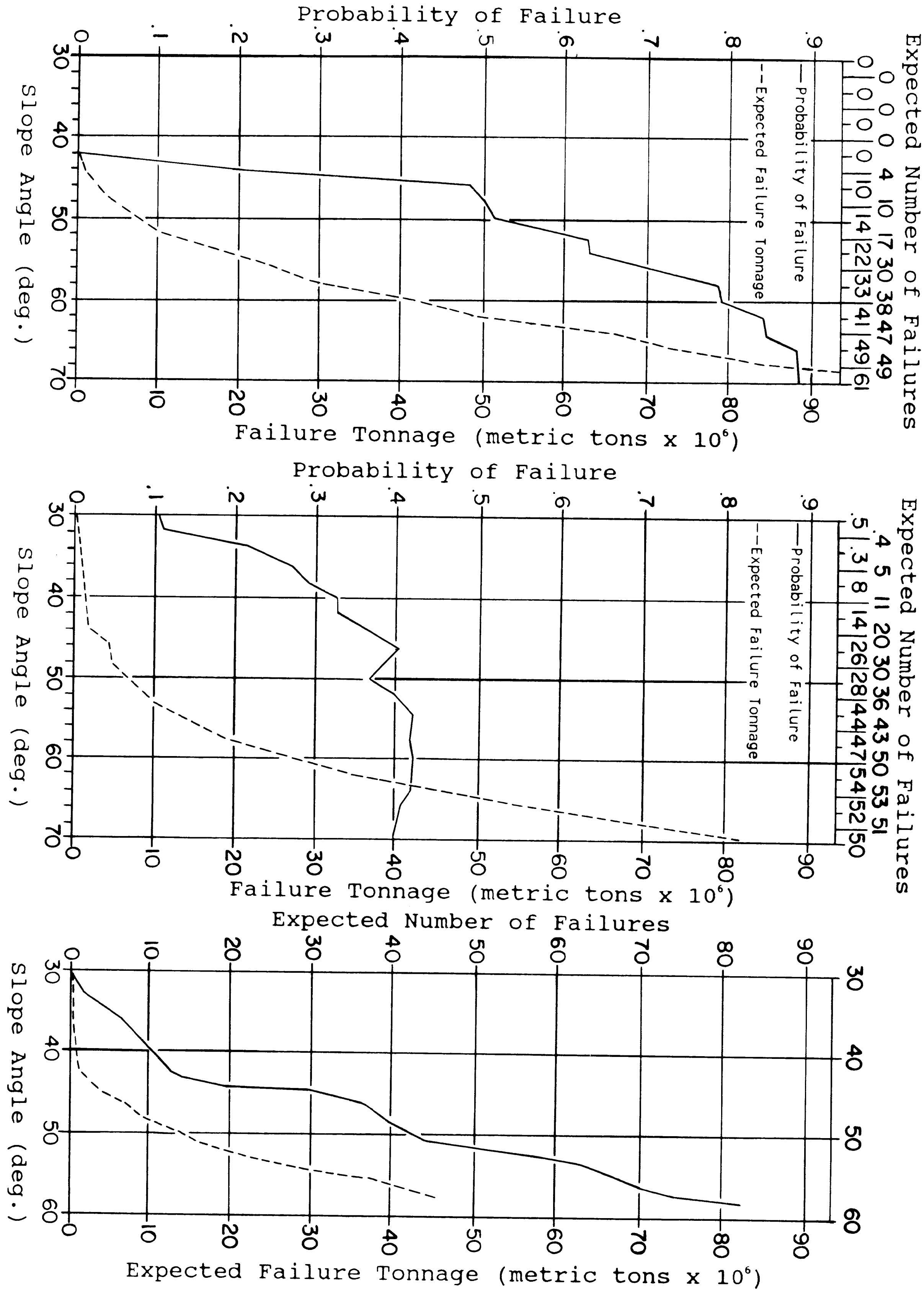


Fig. 10.4.10. Expected failure tonnage curves. Conversion factor: 1 ton = 0.9072 t.

$$\text{tension} = B\cos\alpha \quad (10.4.7)$$

$$\text{dowel} = 0.5B \quad (10.4.8)$$

$$\text{friction} = B\sin\alpha\ \tan\phi \quad (10.4.9)$$

where B is the bolt tensile strength, and a is the angle between the bolt and the shear surface. The dowel strength, which is the strength of the bolt in shear, is taken as one-half the tensile strength. Assuming the bolt acts as a Tresca material (Dight, 1982), the tensile and dowel strengths are not mutually exclusive, and the net shearing resistance of the bolt is

$$\text{net resistance} = (\text{tensile}^2 + \text{dowel}^2)^{1/2} + \text{friction} \quad (10.4.10)$$

Untensioned, fully grouted bolts are easier to install and less expensive than tensioned bolts, and there is considerable evidence that a fully grouted bolt reaches full tension with a very small displacement. The argument for tensioned bolts is that displacement required to tension an untensioned bolt may result in loss of the peak shear strength of the rock and may cause cracking of the grout, which would expose the bolt to corrosion. It is doubtful, however, that the benefit of tensioned bolts justifies the additional cost.

Shotcrete has also been used successfully on a small scale to stabilize progressive raveling failure affecting haul roads.

10.4.5.6 Controlled Blasting

Production blasting is designed to fragment rock for loading. At the slope wall, this fragmentation results in backbreak, which reduces the bench face angle and results in flatter slope angles or narrower catch benches. To reduce this backbreak the fragmentation of the final wall must be reduced by controlled blasting.

It has been found that a blast shock wave with a peak particle velocity greater than 25 in./sec (625 mm/s) initiates cracks in rock and produces significant damage above 100 in./sec (2.5 m/s). The peak particle velocity is a function of the charge weight and the distance from the charge. The relationship is

$$V = k(D/W)^b \quad (10.4.11)$$

where V is peak partial velocity, D is distance, and W is instantaneous charge weight (see Chapter 9.2.2). The constants k and b are a function of the rock and the type of blasting, so are site specific. Typical values for open pit blasting are $b = -1.6$ and $k = 26$ to 260 (Oriard, 1982). This relationship can be used to determine the maximum charge weight per delay required to keep the peak particle velocity below 25 in./sec. (625 mm/s).

Reducing the number of holes per delay will reduce the peak particle velocity, but for the perimeter row of holes and the buffer row, a production hole charge is usually too large, and must be reduced. To maintain the same powder factor, the hole spacing must be reduced concurrently with the reduction in hole charge. In practice, this method of controlled blasting has been shown to increase the measured bench face angle by 5° (Savely, 1986).

Presplitting, where a closely spaced line of holes with a light powder charge is shot before the main blast, can produce a smooth face with minimum damage. Presplitting is usually not necessary and is not effective in closely jointed rock.

10.4.6 SLOPE MANAGEMENT

With an economically optimized slope design, some degree of slope instability can be expected. Minimization of the adverse effects of slope instability must be accomplished through judicious mine planning and the establishment of operational contingencies.

There are several principles of slope mechanics that should be kept in mind when dealing with slope instability.

1. *Slope failures do not occur spontaneously.* A rock mass does not move unless there is a change in the forces acting on it. The common changes that lead to instability in an open pit are removal of support by mining, increased pore pressure, and earthquakes.

2. *Most slope failures tend toward equilibrium.* It is an observed phenomenon that as a slide displaces, the toe pushes out and the crest recedes. Such displacement reduces the driving force and increases the resistance force so that the displacement rate is reduced until movement stops. When high pore pressures are involved, a similar balance is attained. Displacement causes dilation of the rock mass. As a result, pore pressures drop, and the effective shear strength increases. This mechanism explains the stick slip movement of some slides, in which recharge increases the pore pressure in tension cracks, resulting in renewed displacement. There are exceptions to this generalization, but they are usually the result of reduction of shear strength due to shearing of asperities or changes in the forces acting on the rock mass.

3. *A slope failure does not occur without warning.* Prior to major movement, measurable deformation and other observable phenomena such as development of tension cracks occur. These phenomena occur from hours to years before major displacement. However, single bench sloughing directly associated with mining does occur rapidly. While a slope failure does not occur without warning, the converse is not always the case. Deformation and tension cracks can occur without major displacement.

10.4.6.1 Detection of Instability/Monitoring

The first step in slope management is the identification of potential failure areas such as faults, breccia dikes, and/or jointing with attitudes that would form a failure geometry. Data for this identification would come from geologic pit mapping. Areas of higher water levels are also potentially unstable and should be identified.

The second step is monitoring areas that are potentially unstable and/or show evidence of instability by displacement and tension cracks. On the basis of monitoring and mapping, the geometry of a failure can be determined and predictions made of future behavior.

The objectives of a pit slope monitoring program should be

1. To maintain safe operational procedures for the protection of personnel and equipment.

2. To provide advance notice of instability so that mine plans can be modified to minimize the impact of slope displacement.

3. To provide geotechnical information for analyzing the slope failure mechanism, for designing appropriate remedial measures, and for conducting future redesign of the slope.

An effective slope monitoring program consists of the systematic detection, measurement, interpretation, and reporting of evidences of slope instability. Measurements are normally made of both surface and subsurface displacement in order to provide an accurate assessment of slope instability (see Chapter 10.3).

Surface Displacement: Surface displacement measurement by means of tension crack mapping, extensometers, and survey points is still the most cost-effective monitoring method. All three procedures should be used as no one method would give the entire picture. Fig. 10.4.11 shows a typical surface monitoring layout.

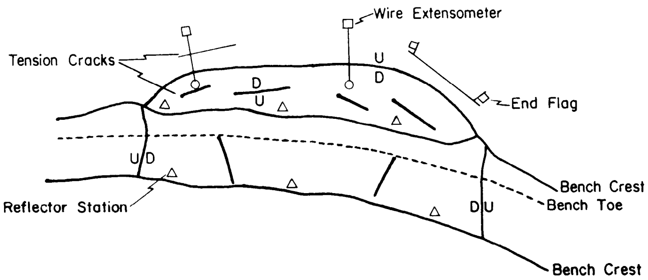


Fig. 10.4.11. Tension crack map.

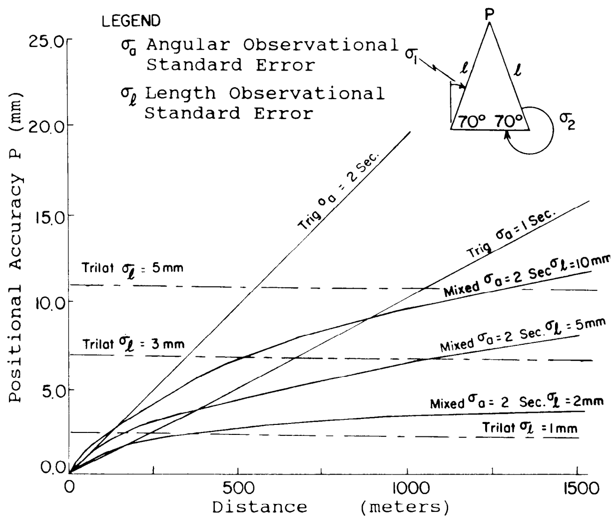


Fig. 10.4.12. Positional accuracy of P by triangulation, trilateration, and triangulateration (Ashkenazi, 1973). Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

1. Tension crack mapping: Tension cracks are an early, obvious indication of instability. By systematically mapping the cracks, the geometry of a failure can be better defined. All cracks should be mapped regardless of apparent cause. Often cracks which appear to be the result of local bench failure or blasting form a pattern showing an impending larger failure when plotted on a pit map.

The ends of the cracks should be flagged or marked so that on subsequent visits new cracks or extensions of existing cracks can be identified.

2. Extensometers: Portable wire extensometers should be used to provide monitoring in areas of active instability across tension cracks. These monitors can be quickly positioned and easily moved. The extensometer should be positioned on stable ground behind the last visible tension crack, and the wire should extend out to the unstable area. For warning devices, or for information on deformation within a sliding mass, wire extensometers can be placed at any strategic location. Anyone working in the area can make an immediate check on slope movement by inspecting the instruments.

3. Survey monitoring: Monitoring prism targets with the geodimeter total station continues to provide the most detailed movement history in terms of displacement directions and rates in the unstable areas. To keep within an accuracy of 0.05 ft (15 mm), the two-second geodimeter should have a maximum range of 5000 ft (1500 m) (Fig. 10.4.12).

Backsights can be taken on other instrument stations or on reference points outside the pit for calibration. In addition to a backsight, each instrument station should have a reference point on stable ground. This reference point is used to check the stability of the instrument station and to calibrate the EDM. Since the displacement measurements are relative, reproducibility is often more important than absolute distances.

Subsurface Displacement: Surface displacement measurements do not determine the subsurface extent of instability, although it is possible to make inferences from displacement vectors. There are many situations where measurement of subsurface displacement is needed. These measurements are commonly made utilizing shear strips, borehole inclinometers, and borehole extensometers.

1. Shear strips: Shear strips in a borehole will help to locate the position where the hole is cut off. Either commercial segmented strips or a coaxial cable with a fault finder can be used. These systems have the limitation of being go/no-go devices.

2. Borehole inclinometers: A borehole inclinometer that measures the angular deflection of the hole will give the deformation normal to the hole.

3. Borehole extensometers: Borehole extensometers will give the deformation parallel to the borehole.

Precision, Reliability, and Cost: The number of different devices that can be used for monitoring, as well as the precision and sophistication of the devices, are a function of the ingenuity, time, and budget of the engineer in charge of monitoring. Since none of these factors is infinite, hard choices must be made. Some general guidelines for decision making follow.

1. Measure the obvious things first: Surface displacement is the most direct and most critical aspect of slope instability.

2. Simpler is better: The reliability of a series system is the product of the reliability of the individual components. A complex electronic or mechanical device with a telemetered output to a computer has significantly less chance of being in operation when needed than do two stakes and a tape measure.

3. Precision costs money: The cost of a measuring device is often a power function of the level of precision. Measuring to 0.4 in. (10 mm) is inexpensive compared to measuring to 0.0004 in. (0.001 mm). A micrometer is unnecessary for monitoring slope movement that has a velocity of 2 in. (50 mm)/day.

4. Redundancy is required: No single device or technique tells the complete story. A single extensometer or survey point cannot indicate the area involved in the instability, and, if it is destroyed, the continuity of the record is lost.

5. Timely reporting is essential: Data collection and analysis must be rapid enough to provide information in time to make decisions.

Monitoring Schedule: A definite monitoring schedule should be established. The frequency of monitoring is a function of the precision of the system, the rate of movement, and how critical the area is. Table 10.4.3 is a suggested schedule. If there is heavy rain or a large blast in the area, additional measurements should be made.

Cooperation between operations and engineering is important. Equipment operators often have an intuitive feel for ground conditions. Any changes in the condition of an area observed by operators should be reported to engineering for followup.

Data Reduction and Reporting: The following measurements or calculations should be made for each survey reading:

1. Date of reading, incremental days between readings, and total number of days the survey point has been established.
2. Coordinates and elevation.
3. Magnitude and direction of horizontal displacement.
4. Magnitude and plunge of vertical displacement.

Table 10.4.3. Suggested Monitoring Schedule

Mining	Velocity Ft/Day	Velocity Mm/day	Visual Inspection	Extension	Crack Map	Survey[3]	Piezometers
Active	0	0	Daily[1]	—	Monthly	Monthly	Monthly
	< 0.05	<15	Daily[1]	Daily[2]	Weekly	Monthly	Weekly
	0.05–0.17	15–50	Each Shift[1]	Each Shift[2]	Daily	Weekly	Daily
	0.17–0.30	50–100	2 × Shift	2 × Shift	Daily	Daily	Daily
Inactive	0	0	Monthly	—	Monthly	Quarterly	Monthly
	< 0.05	<15	Monthly	Monthly	Monthly	Monthly	Monthly
	0.05–0.17	15–50	Daily	Daily[2]	Weekly	Weekly	Weekly
	0.17–0.30	50–100	Daily	Daily[2]	Daily	2 × Week	Daily
	< 0.30	<100	2 × Day	2 × Day[2]	Daily	2 × Day	Daily

Note:
1. Some mining codes require inspection of working face at beginning of each shift.
2. Extensometers should have warning lights.
3. If extensometers are not installed, survey observations should be on extensometer schedule.

5. Magnitude, bearing, and plunge of resultant (total) displacements.

Both incremental and cumulative displacement values should be determined. Calculating the cumulative displacement from initial values rather than from summing incremental displacements minimizes the effects of occasional survey aberrations.

Slope displacements are best understood and analyzed when the monitoring data are graphically displayed. For engineering purposes, the most useful plots are

1. Horizontal position.
2. Vertical position (elevation vs. change in horizontal position, plotted on a section oriented in the mean direction of horizontal displacement).
3. Displacement vectors.
4. Cumulative total displacement vs. time.
5. Incremental total displacement rate (velocity, usually in ft or m/day) vs. time.

All graphics should be kept up-to-date and should be easily reproducible, for ease of distribution. By studying several graphs simultaneously, the movement history of a particular slope can be determined.

Precipitation data should also be recorded in order to evaluate possible correlations with slope displacement. A gage located at the mine site can be used to measure occurrences and amounts of precipitation. In addition, measurement of the average daily temperatures will provide some indication of freeze and thaw periods.

The location of mining areas and the number of tons mined should also be recorded on a regular basis, because slope displacements are often associated with specific mining activity. One method of cataloging this information is to plot the mining area and then note the number of tons mined and the date on a plan map of the pit. A histogram can be made of tons mined vs. time, and this plot can then be compared to the total displacement graphs.

A formal monthly slope stability report should be prepared, containing the data listed in Table 10.4.4 and recommendations on the appropriate response to current instability. This ensures that mine management receives the appropriate information and provides the discipline to document slope behavior. Direct informal communication should also be maintained with pit operations on a daily basis.

10.4.6.2 Slide Management

When instability occurs, there are a number of response options:

Table 10.4.4. Monitoring Data Presentation

Graphs
Cumulative Displacement vs. Time
Velocity vs. Time (ft or m/day, semi-log plot)
Precipitation vs. Time
Water Levels vs. Time
Mining vs. Time
Maps and Sections
Pit Map with Location of Unstable Areas
Location of Monitoring Points with Displacement Vectors
Tension Crack Map
Horizontal Plot of Location with Time
Vertical Plot of Location with Time
Map of Piezometric Surface
Cross Section of Unstable Area

1. Leave the unstable area alone.
2. Continue mining without changing the mine plan.
3. Unload the slide through additional stripping.
4. Leave a step-out.
5. Partial cleanup.
6. Mine out the failure.
7. Support the unstable ground with cable bolts.
8. Dewater the unstable area.

The choice of options or combination of options depends on the nature of the instability and the operational impact. Each case should be evaluated individually and cost-benefit comparisons conducted. The following is a list of guidelines on the choice of options.

1. When instability is in an abandoned or inactive area, it can be left alone.

2. If the displacement rate is low and predictable and the area must be mined, living with the displacement while continuing to mine may be the best action.

3. Even though unloading has been a common response, in general it has been unsuccessful. In fact, there are situations involving high water pressure where unloading actually decreases stability.

4. Step-outs have been used successfully in several mines. The choice between step-out and cleanup is determined by the trade-off between the value of lost ore and the cost of cleanup.

5. Partial clean-up may be the best choice where a slide blocks a haul road or fails onto a working area. Only that material necessary to get back into operation need be cleaned up.

6. Where the failure is on a specific structure and there is competent rock behind the structure, mining out the failure may be the optimum choice.

7. Mechanical support may be the most cost-effective option when a crusher, conveyor, or haul road must be protected.

8. Where high water pressure exists, dewatering is an effective method of stabilization that may be used in conjunction with other options.

10.4.6.3 Contingency Planning

Mine planning should have the flexibility to respond to slope instability. Rather than an after-the-fact crisis response to forced deviation from a rigid mine plan, contingency plans should be prepared in advance so that the response to slope instability is well thought out.

Operational flexibility should be built into the mining plan. For example:

1. Adequate ore should be exposed and accessible so that production is not dependent on a single location.

2. There should be more than one access road into the pit for service vehicles.

3. Whenever possible, double access to working benches should be maintained.

4. Production scheduling should have a provision for slide cleanup.

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Chapter 10.5
GROUND CONTROL

Z.T. BIENIAWSKI

The terms ground control and strata control, which here are used interchangeably, date back some 60 years to the early 1930s when the use of rock bolts was pioneered in the United States. Records show that in 1936, the St. Joseph Lead Co. in Missouri introduced rock bolting in its mines. But it was not until 1943 that systematic use of rock bolts for ground control was described by Weigel. The first international conference on rock pressure and ground control was held in Liege, Belgium, in 1951. Originally, ground/strata control were concerned primarily with coal mining in bedded rock formations (strata), but since the early 1960s, the term has been generalized and systematically applied in all types of underground mining and specifically also in hard-rock mining. More recently, ground control found its way to civil engineering tunneling, due particularly to the importance of mined underground repositories for nuclear waste storage, and to petroleum engineering with the emergence of the mining-for-oil concept (Bieniawski, 1987).

In essence, *ground control* means maintaining rock mass stability by controlling the movement of excavations in the ground, which can be either rock or soil. This is a specialty engineering field that is important because the design of underground excavation is, to a large extent, the design of underground support systems such as pillars and rock reinforcement (rock bolting).

Although today, ground control is no longer solely a mining art since much of it can qualify as an engineering science, the application of rock and soil mechanics principles and techniques in day-to-day mine planning and design is still not common. It is hoped that the up-to-date knowledge digest contained in this chapter will contribute to better dissemination and application of ground-control concepts and techniques, some of which have been available for many years but of which mining engineers have not availed themselves fully.

10.5.1 COMMON GROUND-CONTROL PROBLEMS IN UNDERGROUND MINES

Ground-control failures in underground mines are characteristic of the mining methods employed, opening shape, and support methods. Types of failure include pillar failure, roof falls, floor heave, rock bursts, strata cavings in longwall mining, and support failure.

Moreover, there are many types of failure mechanisms in underground excavations. These include loosening of the rock mass due to body forces and geologic conditions; in situ stresses exceeding the rock strength and resulting in slabbing, swelling, and squeezing of rock; or rock failure due to water, gas, and temperature effects. In essence, failure of an underground excavation may be caused by (1) movement of blocks or wedges of rock into an excavation under the action of gravity, water pressure or in situ stresses, and/or (2) failure of intact rock from overstressing. At shallower depths, the stability of an excavation is generally structurally controlled.

10.5.2 DESIGN PRINCIPLES FOR GROUND CONTROL IN UNDERGROUND MINING

The following definition of *engineering design* may be quoted after the Accreditation Board for Engineering and Technology (Anon., 1987): "Engineering design is the process of devising a system, component, or process to meet desired needs. It is a decision-making process (often iterative), in which the basic sciences, mathematics, and engineering science are applied to convert resources optimally to meet a stated objective. Among the fundamental elements of the design process are the establishment of objectives and criteria, analysis, synthesis, construction, testing and evaluation. Central to the process are the essential and complementary roles of analysis and synthesis. In addition, sociological, economic, aesthetic, legal and ethical considerations need be included in the design process."

The ten distinguishable stages in the design process are as follows (Bieniawski, 1984):

1. Recognition of a need or a problem.
2. Statement of the problem, including identification of performance objectives and design issues.
3. Collection of information.
4. Concept formulation in accordance with design criteria.
5. Analysis of solution components.
6. Synthesis to create alternative solutions.
7. Evaluation and testing of the solutions.
8. Optimization.
9. Recommendation and communication.
10. Implementation.

In mining, the process of designing an excavation may be conceptually visualized, as depicted in Fig. 10.5.1. The design methods available for assessing the stability of underground excavations can be broadly categorized as follows:

1. Analytical methods.
2. Observational methods.
3. Empirical methods.

Analytical methods utilize the analyses of stresses and deformations around openings and include numerical modeling techniques, such as the finite element method. They are effective in ground control because they enable comparisons of a few variations of a mining situation and serve as a relative design procedure.

Observational methods rely on actual monitoring of ground movement during excavation to detect measurable instability, and on the analysis of ground-support interaction.

Empirical methods assess the stability of mines and tunnels by the use of statistical analyses of underground observations and experience. Engineering rock mass classifications are the best known empirical methods.

Note that only some factors are within the control of the design engineer. In the case of coal mining these include opening width, pillar width and length, pillar layout, panel width and length, excavation sequence, panel sequencing, and orientation. Nevertheless, the range within which these factors may be varied is limited; for example, one cannot mine an opening narrower than that needed for operating mining machinery.

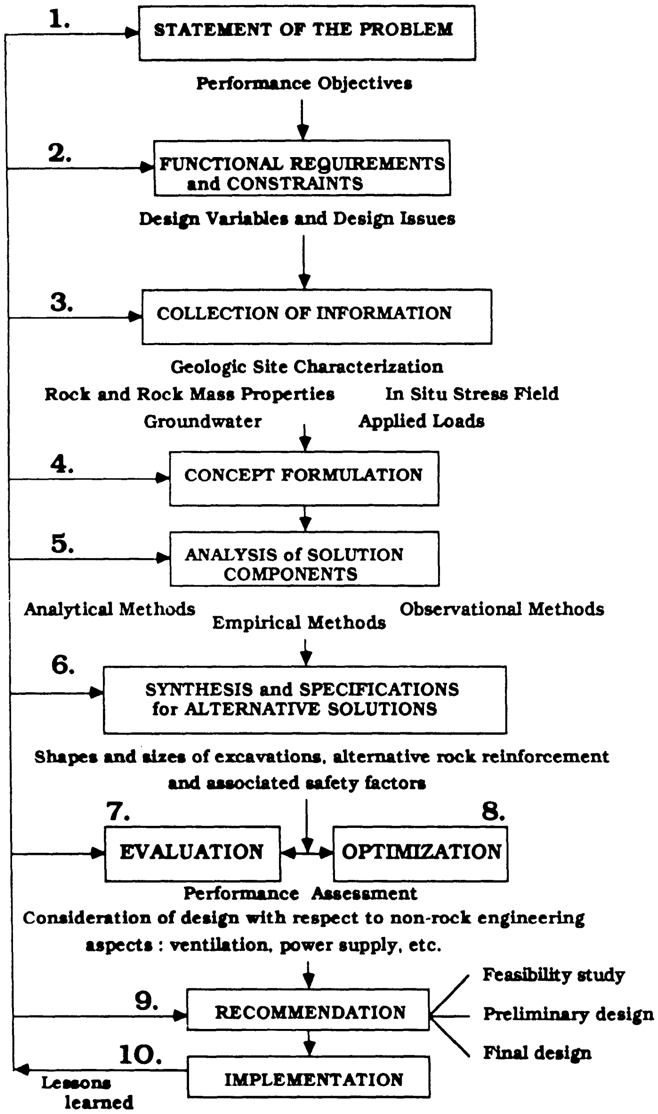


Fig. 10.5.1. Simplified chart of a design process in mining (Bieniawski, 1988).

10.5.3 GEOLOGIC SITE CHARACTERIZATION AND ENGINEERING CLASSIFICATIONS OF ROCK MASSES

Provision and interpretation of reliable input data for ground-control design is one of the most difficult tasks facing geologists and engineers. It is extremely important that the quality of the input data matches the sophistication of the design methods. It has been contended often that some design methods, such as numerical techniques, have outpaced our ability to provide the input data necessary for the application of these methods. Obviously, it must be realized that if incorrect input parameters are employed, incorrect design information will result.

The input information needed for design purposes generally includes geologic characterization of the rock masses and evaluation of the virgin ground stresses and the mechanical properties that characterize the rock mass in its natural state. Whatever procedures and techniques are employed to obtain the input parameters, it is necessary to emphasize that any such procedures and techniques should only be used if they can be fully justified for the purposes of a given project. In other words, measurements and investigations should be carefully planned and matched with the purpose of a project, full justification being given for any investigations and tests performed.

Furthermore, determination of the input parameters for design should be so planned that as much quantitative data as possible is obtained rather than relying on qualitative descriptions.

In essence, the following three important messages emerge:

1. The eventual quality of the engineering design is directly proportional to the quality of the input parameters.
2. Any procedures or methods employed in providing input data should be fully justified and carefully planned.
3. Quantitative rather than qualitative information is required for design purposes.

Determination of the input data for design, involving geologic site characterization, ground stress conditions, groundwater conditions, and mechanical properties of rock masses, is a wide subject. In particular, see Goodman (1976), Hoek and Brown (1980), and Hansen and Lachel (1980), with special reference to mining applications.

10.5.3.1 Geologic Site Characterization

"The first fact which must be recognized when planning a site investigation program is that there is no such thing as a standard site investigation" (Hoek, 1982). The scope of a detailed site exploration is outlined in Fig. 10.5.2. In essence, site characterization involves the discovery, correlation, and analysis of geologic data such as:

1. Rock types to be encountered.
2. Depth and character of overburden.
3. Microscopic scale discontinuities such as major faults.
4. Groundwater conditions.
5. Special problems, such as weak ground or swelling rock.

The initial site assessment can utilize a number of sources of information, of which two are particularly relevant:

1. Available geologic maps, published literature, and possibly local knowledge.
2. Photogeologic images (aerial photographs) of the area. The photogeologic study includes information on topography, drainage, lithology, geologic structures, and discontinuities.

One of the purposes of the initial site exploration is to determine the regional geology of the vicinity of the project. While determination of the regional geology is based mainly on studies of reports, maps, and publications involving the geologic history of the area, some limited investigations may also be conducted. These would include mapping of the surface outcrops, physical exploration, and a limited program of drilling and groundwater investigations. Some laboratory tests on rock samples and index field tests on rock cores may also be performed. Based on these investigations, preliminary geologic maps and sections should be prepared showing favorable and unfavorable regions in the rock mass. These maps and sections are important for planning the next stages of the site characterization program.

Where outcrops and geologic structures are not easily deduced by either photogeologic or ground investigations, geophysical methods may be used to locate large discontinuities such as faults. The most effective means of doing this would be by seismic or resistivity methods (see Chapter 4.4).

Based on initial site exploration, detailed site investigation will be conducted once the feasibility of the project has been established. This stage of site characterization will include detailed exploratory drilling, geologic mapping, geophysical surveys, and rock mechanics testing.

Drilling Investigations and Core Logging: The purpose of a drilling investigation is to

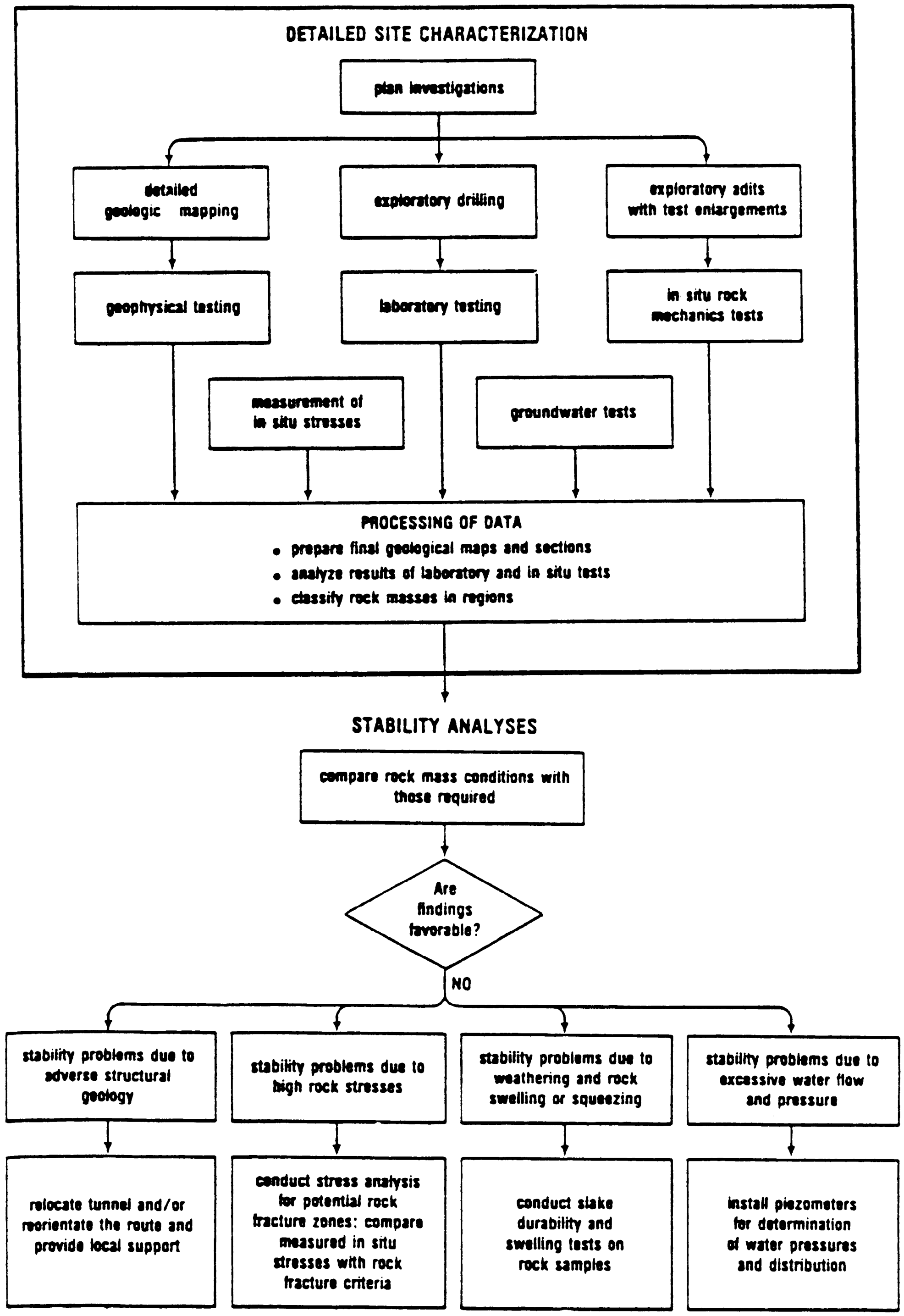


Fig. 10.5.2. Stages of a site characterization program (Bieniawski, 1984).

1. Confirm geologic interpretations.
2. Examine cores and boreholes to determine the quality and characteristics of the rock mass.
3. Study groundwater conditions.
4. Provide cores for rock mechanics testing and petrographic analyses.

As the object of drilling is to obtain cores for interpretation and testing, it is essential to obtain as near 100% core recovery as possible. To ensure a successful drilling operation, the following aspects should be remembered:

1. The cost of a drilling investigation for geotechnical purposes is much higher, sometimes by a factor of two, than the cost of drilling for mineral exploration purposes. Geotechnical drilling necessitates good-quality equipment and extra care, but it can provide high-quality information.

2. The drilling equipment should feature diamond core drilling facilities permitting core of at least NX size ($2\frac{1}{8}$ in. or 54 mm diameter) and featuring split double-tube core barrels to minimize drilling vibrations. Also included should be equipment for performing water pressure tests.

3. The purpose of drilling investigations is to obtain not only the core logs, but also the logging of the borehole itself. Hence examination of the borehole walls by borehole cameras or by other systems should be considered.

4. For meaningful interpretation of the orientation of geologic features, core orientation procedures may be employed during geotechnical drilling. A number of techniques are available for that purpose (see Chapter 5.2). Boreholes should be angled so that vertical discontinuities can be sampled.

5. Good care should be taken of the core recovered from the boreholes. This means that the cores should be photographed as soon as possible, should be carefully marked, placed in protective wrapping in the core boxes, and stored in properly provided storage sheds. Core samples removed for testing should be appropriately marked in the core boxes.

A systematic method should be used for geotechnical logging of the rock cores. There is a difference between a geologic core log for general purposes and a geotechnical core log for engineering purposes. The geotechnical core log provides a format to record both the geologic and engineering characteristics of the rock core and the results of any field tests. The core log should systematically record all the information available from the core. An example of a geotechnical core log is given in Fig. 10.53. It should be noted that there is no rigid standard format for a geotechnical core log, and the amount of detail used will depend on the actual purpose of the project.

Engineering Geologic Mapping: The purpose of engineering geologic mapping is to investigate the significant features of the rock mass, especially the discontinuities such as naturally occurring joints. It is also important to determine the geologic structure especially in stratified rock that may have been subjected to faulting. Detailed procedures for engineering geologic mapping have been described in a number of publications (e.g., see Chapter 5.2).

It should be noted that while engineering geologic mapping is fairly frequently found on tunneling projects, this is not the situation on mining projects. Engineering geologic mapping in underground mines is a fairly recent innovation in mapping procedures for mines (Chapters 5.2 and 5.5).

Finally, one should emphasize that one of the purposes of engineering geologic mapping is to provide input data for a rock mass classification to be used at the site for estimating the stability of underground structures and support requirements. Clearly, engineering geologic mapping will provide the most reliable input data for a rock mass classification, although it is also possible to obtain reasonable data from interpretations of the borehole and core logs.

Geophysical Investigations: Geophysical techniques involving seismic refraction and reflection, electrical resistivity, gravimetric, and magnetic measurements form an accepted part of engineering-geologic investigation procedures. Detailed descriptions of these methods, together with their applications, limitations, accuracy, and costs, may be found in many textbooks and articles (e.g., Hoek and Brown, 1980), as well as in Chapter 4.4.

Geologic Data Presentation: If determination of geologic data for site characterization is a difficult problem, presentation of these data for engineering purposes is sometimes even more difficult. Communication between the engineering geologist and the design engineer would be greatly enhanced if the format for data presentation could be established in the early stages of an engineering project. The following suggestions are useful:

1. Borehole data should be presented in well-executed geotechnical logs.

2. Mapping data derived from joint surveys should be presented as spherical projections, for example, of the Schmidt or Wolff type (Goodman, 1976; Hoek and Brown, 1980).

3. A summary of all geologic data, including groundwater conditions, should be entered in the input data sheets for rock-mass classification purposes, such as depicted in Fig. 10.5.4. One such sheet is required for each geologic zone in which certain rock-mass features are more or less uniform, for example, type of rock or discontinuity spacing.

4. Longitudinal and cross sections of structural geology at the site should form an integral part of a geologic report.

5. Consideration should be given to constructing a geologic model of the site.

In summary, the following geologic parameters are considered important in the stability of underground excavations and should be included in a site characterization report:

1. Orientation of discontinuities in the rock mass.
2. Spacing of discontinuities in the rock mass.
3. Condition of discontinuities, including roughness, separation, continuity, weathering, and infilling (gouge).
4. Groundwater conditions.
5. Major faults.
6. Properties of intact rock material.

10.5.3.2 Rock Mass Classifications

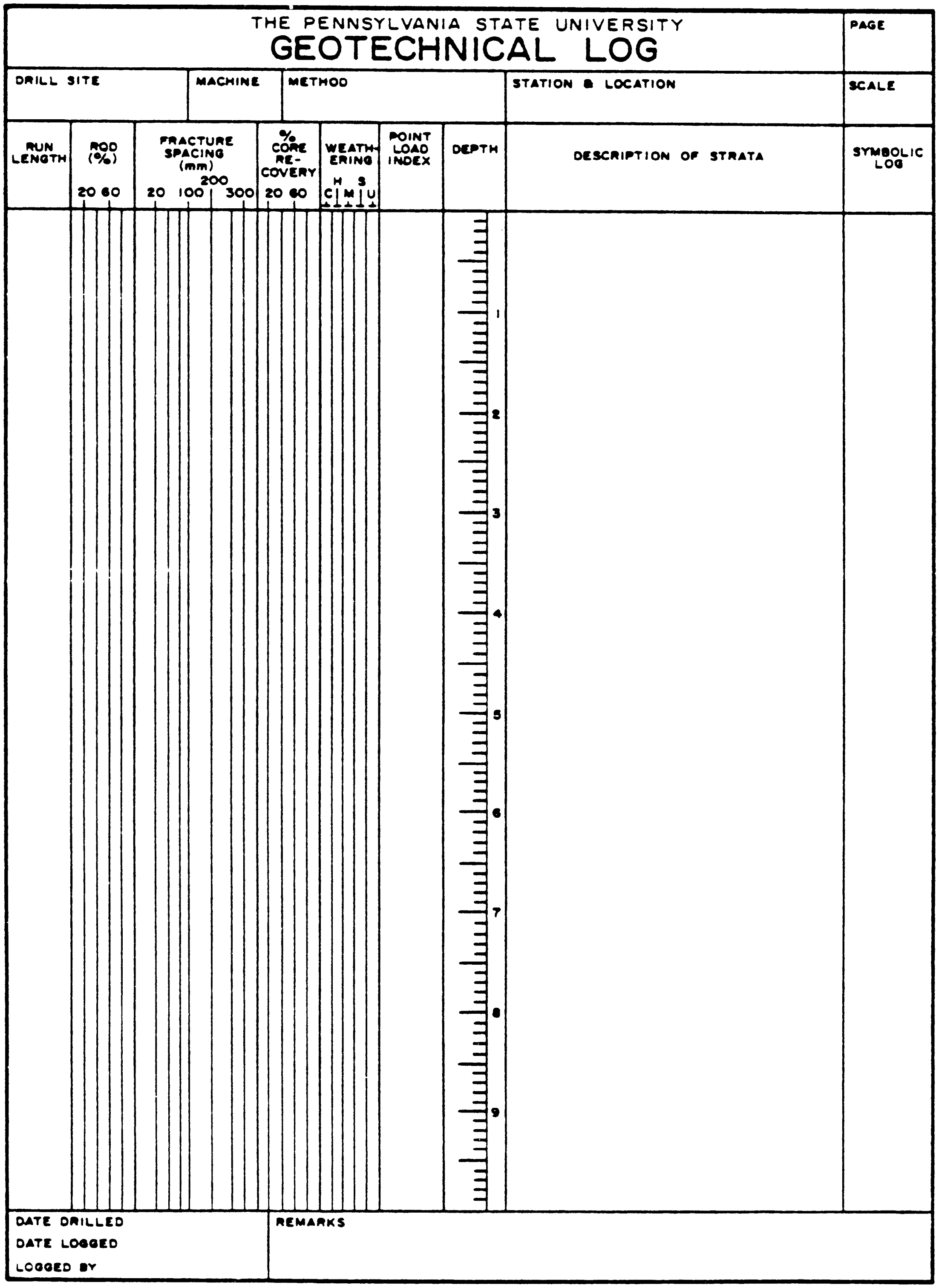
Empirical design methods relate practical experience gained on previous projects to the conditions anticipated at a proposed site.

Rock mass classifications form the backbone of the empirical design approach and are widely employed in rock engineering (Bieniawski, 1989). In fact, on many projects, the classification approach serves as the only practical basis for the design of complex underground structures. Most of the tunnels and mines constructed at present make use of some classification system. The most used and the best known of these is Terzaghi's rock load classification introduced over 45 years ago (Terzaghi, 1946). Since then, this classification has been modified (Deere et al., 1970), and new rock classification systems have been proposed. These systems took cognizance of the new advances in rock support technology, namely, rock bolts and shotcrete, as well as addressed different engineering projects: tunnels, chambers, mines, slopes, and foundations. Today there are so many different rock classification systems in existence that it is useful to tabulate the more common ones (Table 10.5.1).

Rock mass classifications have been applied throughout the world, in the United States (Deere et al., 1967; Wickham et al., 1972; Bieniawski, 1979); Canada (Franklin, 1976); Western Europe (Lauffer, 1958; Pacher et al., 1974; Barton et al., 1974); South Africa (Bieniawski, 1973; Laubscher, 1977; Olivier, 1979); Australia (Baczynski, 1980); New Zealand (Rutledge and Preston, 1978); Japan (Nakao et al., 1983); India (Ghose and Raju, 1981; Venkateswarlu, 1986); USSR (Protodyakonov, 1974); and Poland (Kidybinski, 1979).

Major Classifications Currently in Use: Of the many rock-mass classification systems in existence today, six should be cited because they are especially important contributions, namely, those proposed by Terzaghi (1946), Lauffer (1958), Deere et al. (1967), Wickham et al. (1972), Bieniawski (1973), and Barton et al. (1974).

The rock load classification of Terzaghi (1946) was the first practical classification system introduced and has been dominant in the United States for over 40 years, proving very successful for tunneling with steel supports. Lauffer's classification (1958) was based on the work of Stini (1950) and was a considerable step forward in the art of tunneling since it introduced the concept of the stand-up time of the active span in a tunnel, which is highly relevant in determining the type and amount of tunnel support. Deere's classification (1967) introduced the rock quality designa-


THE PENNSYLVANIA STATE UNIVERSITY

GEOTECHNICAL LOG

PAGE

DRILL SITE

MACHINE

METHOD

STATION & LOCATION

SCALE

RUN LENGTH

RQD (%) 20 60

FRACTURE SPACING (mm) 20 100 200 300

% CORE RE-COVERY 20 60

WEATH-ERING H S C M U

POINT LOAD INDEX

DEPTH 1 2 3 4 5 6 7 8 9

DESCRIPTION OF STRATA

SYMBOLIC LOG

DATE DRILLED

DATE LOGGED

LOGGED BY

REMARKS

Fig. 10.5.3. Geotechnical core log for ground-control applications (Bieniawski, 1984).

tion (RQD) index, which is a simple and practical method of describing the quality of rock core recovered from boreholes.

The concept of rock structure rating (RSR), developed in the United States by Wickham et al. (1972, 1974), was the first system featuring classification ratings for weighing the relative importance of classification parameters. The geomechanics classification (RMR system) proposed by Bieniawski (1973) and the Q system proposed by Barton et al. (1974) were developed independently; both provide quantitative data for the selection of modern tunnel reinforcement measures such as rock bolts and shotcrete. The Q system has been developed specifically for tunnels and chambers while the RMR system, although also

INPUT DATA FORM : GEOMECHANICS CLASSIFICATION (ROCK MASS RATING SYSTEM)

Name of project:
Site of survey:
Conducted by:
Date:

STRUCTURAL REGION	DEPTH, m	ROCK TYPE

STRENGTH OF INTACT ROCK MATERIAL

Designation	Uniaxial compressive strength, MPa	OR	Point-load strength index, MPa
Very High:	**Over 250**.........		**>10**.............
High:	**100-250**..........		**4-10**............
Medium High:	**50-100**...........		**0-4**.............
Moderate:	**25-50**............		**1-2**.............
Low:	**5-25**.............		**<1**..............
Very Low:	**1-5**		

DRILL CORE QUALITY R.Q.D.

Excellent quality:	90-100%	....
Good quality:	75-90%	....
Fair quality:	50-75%	....
Poor quality:	25-50%	....
Very poor quality:	< 25%	....

R.Q.D. = Rock Quality Designation

STRIKE AND DIP ORIENTATIONS

	Strike (average)		Dip (angle)	(direction)
Set 1	Strike..................	(from .... to ....)	Dip:...............	....
Set 2	Strike..................	(from .... to ....)	Dip:...............	....
Set 3	Strike..................	(from .... to ....)	Dip:...............	....
Set 4	Strike..................	(from .... to ....)	Dip:...............	....

NOTE: Refer all directions to magnetic north.

SPACING OF DISCONTINUITIES

		Set 1	Set 2	Set 3	Set 4
Very wide:	Over 2 m	....	....	....	....
Wide:	0.6 - 2m	....	....	....	....
Moderate:	200 - 600 mm	....	....	....	....
Close:	60 - 200 mm	....	....	....	....
Very close:	< 60 mm	....	....	....	....

GROUND WATER

INFLOW per 10 m of tunnel length liters/minute ....
or
WATER PRESSURE kPa ....

GENERAL CONDITIONS (completely dry, damp, wet, dripping or flowing under low/medium or high pressure) : ....

IN SITU STRESSES

CONDITION OF DISCONTINUITIES

		Set 1	Set 2	Set 3	Set 4
PERSISTENCE (CONTINUITY)					
Very low:	< 1 m	....	....	....	....
Low:	1 - 3 m	....	....	....	....
Medium:	3 - 10 m	....	....	....	....
High:	10 - 20 m	....	....	....	....
Very high:	>20 m	....	....	....	....
SEPARATION (APERTURE)					
Very tight joints:	< 0.1 mm	....	....	....	....
Tight joints:	0.1 - 0.5 mm	....	....	....	....
Moderately open joints:	0.5 - 2.5 mm	....	....	....	....
Open joints:	2.5 - 10 mm	....	....	....	....
Very wide aperture:	> 10 mm	....	....	....	....
ROUGHNESS (state also if surfaces are stepped, undulating or planar)					
Very rough surfaces:		....	....	....	....
Rough surfaces:		....	....	....	....
Slightly rough surfaces:		....	....	....	....
Smooth surfaces:		....	....	....	....
Slickensided surfaces:		....	....	....	....
FILLING (GOUGE)					
Type:		....	....	....	....
Thickness:		....	....	....	....
Uniaxial compressive strength, MPa		....	....	....	....
Seepage:		....	....	....	....
WALL ROCK OF DISCONTINUITIES					
Unweathered		....	....	....	....
Slightly weathered		....	....	....	....
Moderately weathered		....	....	....	....
Highly weathered		....	....	....	....
Completely weathered		....	....	....	....
Residual soil		....	....	....	....

GENERAL REMARKS AND ADDITIONAL DATA

MAJOR FAULTS specify locality, nature and orientations.

NOTE: For definitions and methods consult ISRM document: *'Quantitative description of discontinuities in rock masses.'*

Fig. 10.5.4. Data sheet for engineering classification of rock masses (Bieniawski, 1989). Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 psi = 6.895 kPa.

Table 10.5.1. Major Engineering Rock Mass Classifications Currently in Use

Name of classification	Originator and date	Country of origin	Applications
1. Rock loads	Terzaghi, 1946	USA	Tunnels with steel support
2. Stand-up time	Lauffer, 1958	Austria	Tunneling
3. NATM	Rabcewicz, Pacher Müller, 1964	Austria	Tunneling
4. RQD index	Deere et al., 1967	USA	Core logging, tunneling
5. RSR concept	Wickham et al., 1972	USA	Tunneling
6. **RMR system** (geomechanics classification	Bieniawski, 1973 [last modified, 1979, USA]	South Africa	Tunnels, mines, slopes, foundations
RMR System Extensions			
	Laubscher, 1977	South Africa	Mining
	Ghose and Raju, 1981	India	Coal mining
	Kendorski et al., 1983	USA	Hard rock mining
	Serafim and Pereira, 1983	Portugal	Foundations
	Gonzales de Vallejo, 1983	Spain	Tunneling
	Unal, 1983	USA	Roof bolting/coal
	Romana, 1985	Spain	Slope stability
	Newman, 1985	USA	Coal mining
	Venkateswarlu, 1986	India	Coal mining
	Robertson, 1988	Canada	Slope stability
7. **Q system**	Barton et al., 1974	Norway	Tunnels, chambers
8. Strength—Size	Franklin, 1975	Canada	Tunneling
9. Basic geotechnical description	International Society for Rock Mechanics, 1981		General, communication

initially developed for tunnels, has been applied to rock slopes and foundations, ground rippability assessment, as well as to mining problems (Laubscher, 1977; Ghose and Raju, 1981; Kendorski et al., 1983).

Today the RMR system and the Q system are the two rock mass classifications most commonly used throughout the world, and they will be discussed in detail.

Geomechanics Classification (Rock Mass Rating System): The rock mass rating (RMR) system, otherwise known as the geomechanics classification, was developed during 1972–73 and modified over the years as more case histories became available and to conform with international standards and procedures (Bieniawski, 1973, 1979). Over the past 15 years, the RMR system has stood the test of time and benefited from extensions and applications by many authors throughout the world. These varied applications, contained in some 350 case histories, point to the acceptance of the system and its inherent ease of use and versatility in engineering practice, involving tunnels, chambers, mines, slopes, and foundations. Nevertheless, it is important that the RMR system is used for the purpose for which it was developed and not as the answer to all design problems.

Because the RMR system has been modified several times, it is important to note that the system has remained essentially the same in principle in spite of the changes. Thus any modifications and extensions were the outgrowth of the same basic method and should not be misconstrued as new systems. To avoid any confusion, the following extensions of the system were valuable new applications, but were still a part of the same overall RMR system: mining applications, Laubscher (1977, 1984); rippability, Weaver (1975); hard-rock mining, Kendorski et al. (1983); coal mining, Unal (1983), Newman (1984), Venkateswarlu (1986), Newman and Bieniawski (1987); dam foundations, Serafim and Pereira (1983); tunneling, Gonzales de Vallejo (1983); and slope stability, Romana (1985).

Classification Procedures—The following six parameters are used to classify a rock mass using the RMR system:

1. Uniaxial compressive strength of rock material.
2. Rock quality designation (RQD).
3. Spacing of discontinuities.
4. Condition of discontinuities.
5. Groundwater conditions.
6. Orientation of discontinuities.

To apply the RMR system, the rock mass is divided into a number of structural regions such that certain features are more or less uniform within each region. Although rock masses are discontinuous in nature, they may, nevertheless, be uniform in regions when, for example, the type of rock or the discontinuity spacings are the same throughout the region. In most cases, the boundaries of structural regions will coincide with major geologic features such as faults, dykes, shear zones, etc. After the structural regions have been identified, the classification parameters for each structural region are determined from measurements in the field and entered onto the input data sheet (Fig. 10.5.4).

The geomechanics classification (RMR system) is presented in Table 10.5.2. In section A, five parameters are grouped into five ranges of values. Since the various parameters are not equally important for the overall classification of a rock mass, importance ratings are allocated to the different value ranges of the parameters, a higher rating indicating better rock mass conditions. The importance ratings are assigned to each parameter, according to section A of Table 10.5.2. In this respect, the average typical conditions are evaluated for each discontinuity set and the ratings are interpolated, using the classification charts A through E. The charts are helpful for borderline cases and also remove an impression that abrupt changes in ratings occur between categories. Chart D is used if either RQD or discontinuity spacing data are lacking. Based on the correlation data from Priest and Hudson (1976), the chart enables an estimate to be made of the missing parameter. Furthermore, it should be noted that the importance ratings given for discontinuity spacings apply to rock masses having three sets of discontinuities. Thus, when only two sets of discontinuities are present, a conservative assessment is obtained. In this way, the number of discontinuity sets is considered indirectly. Laubscher (1977) presented a rating concept for discontinuity spacings as a function of the number of joint sets. It can be shown that when less than three sets of discontinuities are present, the rating for discontinuity spacing may be increased by 30%.

Table 10.5.2. Rock Mass Rating System (Geomechanics Classification of Rock Masses).

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS

<table>
<tr><th colspan="3">Parameter</th><th colspan="7">Ranges of Values</th></tr>
<tr><td rowspan="3">1</td><td rowspan="2">Strength of intact rock material</td><td>Point-load strength index (MPa)</td><td>>10</td><td>4 – 10</td><td>2 – 4</td><td>1 – 2</td><td colspan="3">For this low range, uniaxial compressive test is preferred</td></tr>
<tr><td>Uniaxial compressive strength (MPa)</td><td>>250</td><td>100 – 250</td><td>50 – 100</td><td>25 – 50</td><td>5 – 25</td><td>1 – 5</td><td><1</td></tr>
<tr><td colspan="2">Rating</td><td>15</td><td>12</td><td>7</td><td>4</td><td>2</td><td>1</td><td>0</td></tr>
<tr><td rowspan="2">2</td><td colspan="2">Drill core quality RQD (%)</td><td>90 – 100</td><td>75 – 90</td><td>50 – 75</td><td>25 – 50</td><td colspan="3"><25</td></tr>
<tr><td colspan="2">Rating</td><td>20</td><td>17</td><td>13</td><td>8</td><td colspan="3">3</td></tr>
<tr><td rowspan="2">3</td><td colspan="2">Spacing of discontinuities</td><td>>2 m</td><td>0.6 – 2 m</td><td>200 – 600 mm</td><td>60 – 200 mm</td><td colspan="3"><60 mm</td></tr>
<tr><td colspan="2">Rating</td><td>20</td><td>15</td><td>10</td><td>8</td><td colspan="3">5</td></tr>
<tr><td rowspan="2">4</td><td colspan="2">Condition of discontinuities</td><td>Very rough surfaces
Not continuous
No separation
Unweathered wall rock</td><td>Slightly rough surfaces
Separation < 1 mm
Slightly weathered walls</td><td>Slightly rough surfaces
Separation < 1 mm
Highly weathered wall</td><td>Slickensided surfaces
or
Gouge < 5 mm thick
or
Separation 1 – 5 mm
Continuous</td><td colspan="3">Soft gouge > 5 mm thick
or
Separation > 5 mm
Continuous</td></tr>
<tr><td colspan="2">Rating</td><td>30</td><td>25</td><td>20</td><td>10</td><td colspan="3">0</td></tr>
<tr><td rowspan="4">5</td><td rowspan="3">Groundwater</td><td>Inflow per 10 m tunnel length (L/min)</td><td>None</td><td><10</td><td>10 – 25</td><td>25 – 125</td><td colspan="3">>125</td></tr>
<tr><td>Ratio $\frac{\text{Joint water pressure}}{\text{Major principal stress}}$</td><td>or
0</td><td>or
<0.1</td><td>or
0.1 – 0.2</td><td>or
0.2 – 0.5</td><td colspan="3">or
>0.5</td></tr>
<tr><td>General conditions</td><td>or
Completely dry</td><td>or
Damp</td><td>or
Wet</td><td>or
Dripping</td><td colspan="3">or
Flowing</td></tr>
<tr><td colspan="2">Rating</td><td>15</td><td>10</td><td>7</td><td>4</td><td colspan="3">0</td></tr>
</table>

B. RATING ADJUSTMENT FOR DISCONTINUITY ORIENTATIONS

Strike and Dip Orientations of Discontinuities		Very Favorable	Favorable	Fair	Unfavorable	Very Unfavorable
Ratings	Tunnels and mines	0	−2	−5	−10	−12
	Foundations	0	−2	−7	−10	−25
	Slopes	0	−5	−25	−50	−60

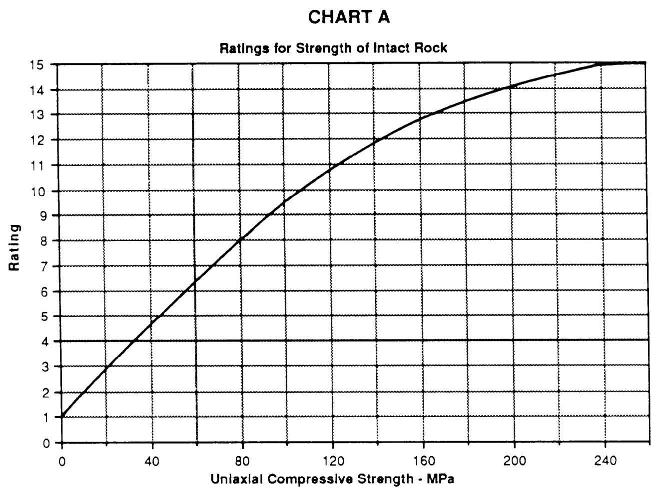
C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS

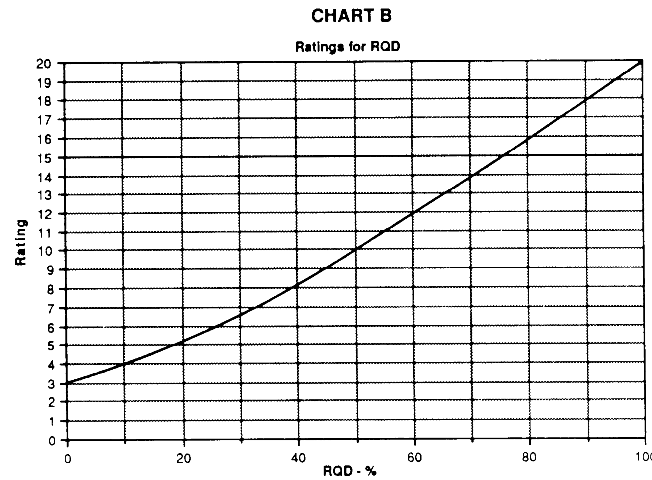
Rating	100 ← 81	80 ← 61	60 ← 41	40 ← 21	<20
Class no.	I	II	III	IV	V
Description	Very good rock	Good rock	Fair rock	Poor rock	Very poor rock

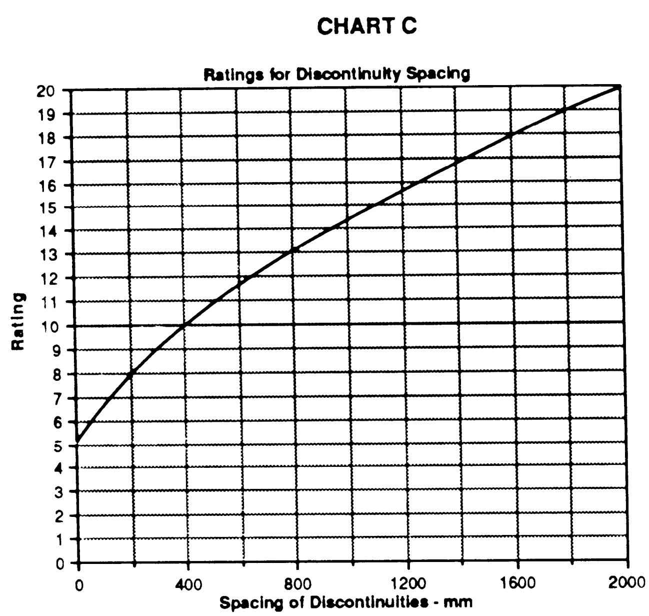
D. MEANING OF ROCK MASS CLASSES

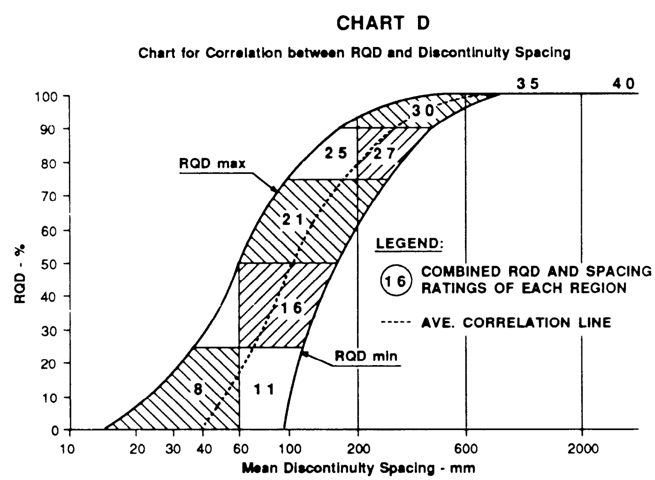
Class no.	I	II	III	IV	V
Average stand-up time	20 yr for 15-m span	1 yr for 10-m span	1 wk for 5-m span	10 h for 2.5-m span	30 min for 1-m span
Cohesion of the rock mass (kPa)	>400	300 – 400	200 – 300	100 – 200	<100
Friction angle of the rock mass (deg)	>45	35 – 45	25 – 35	15 – 25	<15

Table 10.5.2.—cont.









Source: Bieniawski, 1979. Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 psi = 6.895 kPa, 1 gpm = 3.785 L/min.

Table 10.5.2.—cont.

CHART E
Guidelines for Classification of Discontinuity Conditions[a]

Parameter	Ratings				
Discontinuity length (persistence/continuity)	<1 m 6	1 – 3 m 4	3–10 m 2	10–20 m 1	>20 m 0
Separation (aperture)	None 6	< 0.1 mm 5	0.1 – 1.0 mm 4	1–5 mm 1	>5 mm 0
Roughness	Very rough 6	Rough 5	Slightly rough 3	Smooth 1	Slickensided 0
		Hard filling		Soft filling	
Infilling (gouge)	None 6	<5 mm 4	>5 mm 2	<5 mm 2	>5 mm 0
Weathering	Unweathered 6	Slightly weathered 5	Moderately weathered 3	Highly weathered 1	Decomposed 0

[a] *Note:* Some conditions are mutually exclusive. For example, if infilling is present, it is irrelevant what the roughness may be, since its effect will be overshadowed by the influence of the gouge. In such cases, use Table 10.5.2.

Source: Bieniawski, 1989.

Table 10.5.3(a). Effect of Discontinuity Strike and Dip Orientations in Tunneling

Strike Perpendicular to Tunnel Axis			
Drive with Dip		Drive against Dip	
Dip 45–90°	Dip 20–45°	Dip 45–90°	Dip 20–45°
Very favorable	Favorable	Fair	Unfavorable

Strike Parallel to Tunnel Axis		Irrespective of Strike
Dip 20–45°	Dip 45–90°	Dip 0–20°
Fair	Very unfavorable	Fair

Source: Bieniawski, 1984.

After the importance ratings of the classification parameters are established, the ratings for the five parameters listed in section A of Table 10.5.2 are summed to yield the basic (unadjusted for discontinuity orientations) rock mass rating for the structural region under consideration.

The next step is to include the sixth parameter, namely, the influence of strike and dip orientation of discontinuities by adjusting the basic rock mass rating according to section B of Table 10.5.2. This step is treated separately because the influence of discontinuity orientations depends upon the engineering applications, for example, tunnel, mine, slope or foundation. It will be noted that the value of the parameter, *discontinuity orientation,* is not given in quantitative terms but by qualitative descriptions such as "favorable." To facilitate a decision whether strike and dip orientations in mining and tunneling are favorable or not, reference should be made to Table 10.5.3(a), which is based on studies by Wickham et al. (1972). For slopes and foundations, the reader is referred to appropriate papers by Romana (1985) and by Bieniawski and Orr (1976).

The parameter discontinuity orientation reflects on the significance of the various discontinuity sets present in a rock mass. The main set is usually designated as set number 1 and controls the stability of an excavation; for example, in tunneling it will be the set whose strike is parallel to the tunnel axis. The summed up ratings of the classification parameters for this discontinuity set will constitute the overall rock mass rating. On the other hand, in situations where no one discontinuity set is dominant and of critical importance, or when estimating rock mass strength and deformability, the ratings from each discontinuity set are averaged for the appropriate individual classification parameter.

In the case of civil engineering projects, an adjustment for discontinuity orientations will generally suffice. For mining applications, other adjustments may be called for, such as the stress at depth or a change in stress, and these were discussed by Laubscher (1977) and by Kendorski et al. (1983). The procedure for these adjustments is depicted in Table 10.5.3(b).

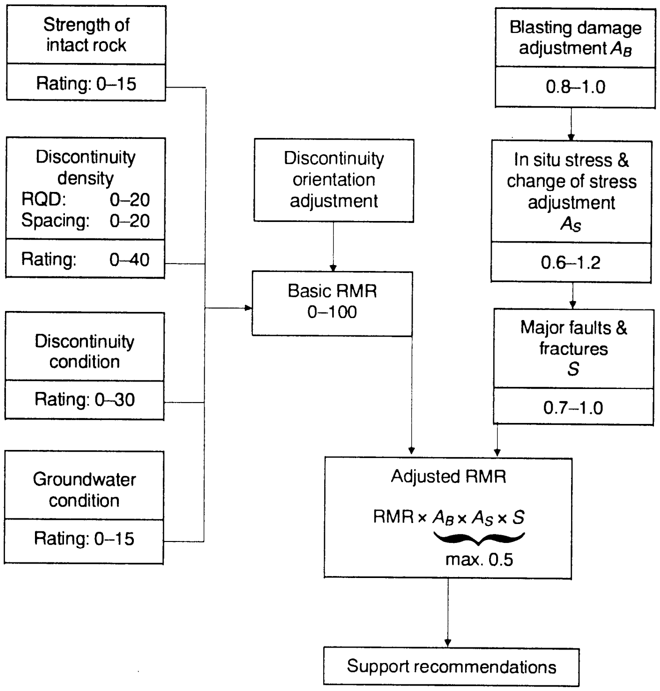
After adjustment for discontinuity orientations, the rock mass is classified according to section C of Table 10.5.2, which groups the final (adjusted) rock mass rating into five rock mass classes, the full range of the possible RMR values varying from 0 to 100. Note that the rock mass classes are in groups of 20 ratings each. This concept of rating a rock mass on a maximum value of 100 has a distinct advantage over an open-ended system in that it allows the engineer or geologist to get the sense of a relative quality, or the lack of it, of a given rock mass in terms of its maximum potential.

Next, section D of Table 10.5.2 gives the practical meaning of each rock mass class by relating it to specific engineering problems. In the case of tunnels, chambers, and mines, the output from the geomechanics classification is the stand-up time and the maximum stable rock span for a given rock mass rating, as shown in Fig. 10.5.5 (a) and (b).

When mixed-quality rock conditions are encountered at the excavated rock face, such as good quality and poor quality being present in one exposed area, it is essential to identify the most critical condition for the assessment of the rock strata. This means that the geologic features that are most significant for stability purposes will have an overriding influence. For example, a fault or a shear in high-quality rock face will play a dominant role irrespective of the high rock material strength in the surrounding strata.

It is recommended that when there are two or more clearly different zones in one rock face, one approach to adopt is to obtain RMR values for each zone and then compute the overall

Table 10.5.3(b). Adjustments to the Rock Mass Rating System for Mining Applications



Source: Bieniawski, 1984.

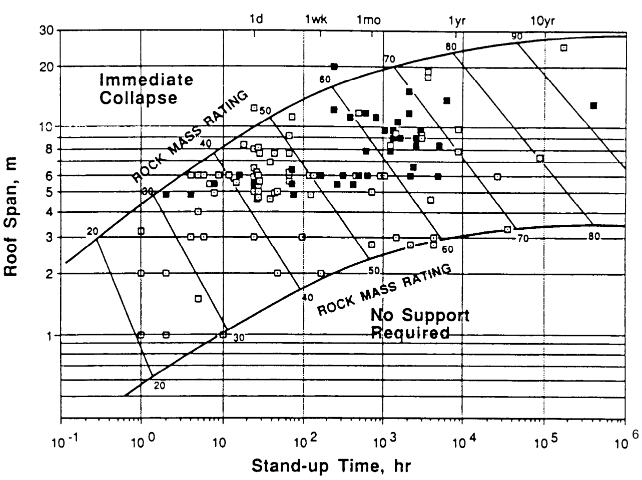


Fig. 10.5.5a. Relationship between the stand-up time and span for various rock mass classes, according to the RMR classification system—output for mining and tunneling. The plotted data points represent the roof falls studied: black squares for mines, open squares for tunnels. The contour lines are limits of applicability. (Bieniawski, 1989). Conversion factor: 1 ft = 0.3048 m.

weighted value by the surface area corresponding to each zone in relation to the whole area, as well as by the influence that each zone has on the stability of the whole excavation.

The RMR system provides guidelines for the selection of rock support for tunnels and mines, examples of which are given in Tables 10.5.4, 10.5.5, and 10.5.6. These guidelines depend on such factors as the depth below surface (in situ stress), tunnel size and shape, and the method of excavation. Note that the

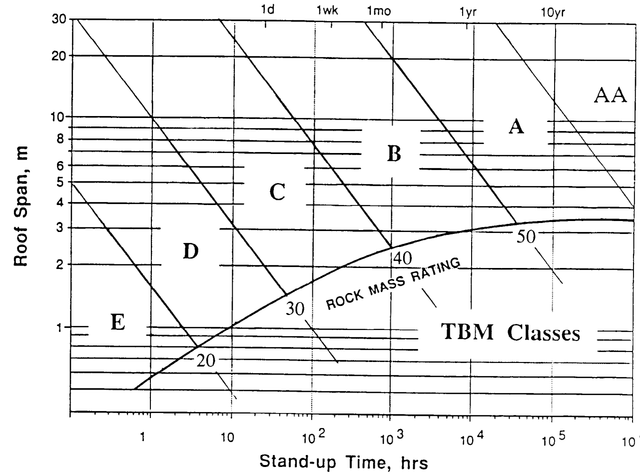


Fig. 10.5.5b. Modified 1988 Lauffer diagram depicting boundaries of rock mass classes for tunnel boring machine applications (Lauffer, 1988). Equivalent RMR values are estimates.

support measures given in Table 10.5.4 represent the permanent and not the primary support for tunnels. All these guidelines should be tested for applicability to the local mining conditions.

Support load can be determined from the RMR system as proposed by Unal (1983):

$$P = [(100 - \mathrm{RMR})/100]\gamma B \qquad (10.5.1)$$

where P is support load in lb (kN), B is tunnel width in ft (m), and g is rock density in lb/ft^3 (kg/m^3).

It must be emphasized that for all applications such as those involving the selection of rock reinforcement, determination of rock loads or rock mass strength and deformability, it is the actual RMR value that must be used and not the rock mass class in which this RMR value falls. In this way, the RMR system is very sensitive to individual parameters because within one rock mass class, for example, "good rock", there is much difference between RMR = 80 and RMR = 61.

Rock Mass Deformation—In the case of rock foundations, knowledge of the modulus of deformation of rock masses is of prime importance. The RMR system proved a useful method for estimating in situ deformability of rock masses (Bieniawski, 1978). As shown in Fig. 10.5.6, the following correlation was obtained:

$$E_M = 2\ \mathrm{RMR} - 100 \qquad (10.5.2)$$

where E_M is in situ modulus of deformation in GPa and RMR > 50.

More recently, Serafim and Pereira (1983) provided many results in the range RMR > 50 and proposed a new correlation:

$$E_M = 10^{(\mathrm{RMR}-10)/40} \qquad (10.5.3)$$

In the case of slopes, the output is given in section D of Table 10.5.2 as the cohesion and friction of the rock mass. Romana (1985) has applied the RMR system extensively for determination of rock slope stability.

Rock Mass Strength—Hoek and Brown (1980) proposed a method for estimating rock mass strength that makes use of the

Table 10.5.4. The RMR System Guidelines for Excavation and Support in Rock Tunnels

SHAPE: HORSESHOE, WIDTH: 10 m; VERTICAL STRESS: BELOW 25 MPa; CONSTRUCTION: DRILLING AND BLASTING

Rock mass class	Excavation	Support		
		Rock bolts (20 mm, dia., fully grouted)	Shotcrete	Steel sets
Very good rock I RMR:81–100	Full face 3 m advance.	Generally no support required except for occasional spot bolting.		
Good rock II RMR:61–80	Full face 1.0–1.5 m advance. Complete support 20 m from face.	Locally bolts in crown 3 m long, spaced 2.5 m with occasional wire mesh.	50 mm in crown where required.	None
Fair rock III RMR:41–60	Top heading and bench 1.5–3 m advance in top heading. Commence support after each blast. Complete support 10 m from face.	Systematic bolts 4 m long, spaced 1.5 m–2 m in crown and walls with wire mesh in crown.	50–100 mm in crown and 30 mm in sides.	None
Poor rock IV RMR:21–40	Top heading and bench 1.0–1.5 m advance in top heading. Install support concurrently with excavation 10 m from face.	Systematic bolts 4–5 m long, spaced 1–1.5 m in crown and walls with wire mesh.	100–150 mm in crown and 100 mm in sides.	Light to medium ribs spaced 1.5 m where required.
Very poor rock V RMR: <20	Multiple drifts. 0.5–1.5 m advance in top heading. Install support concurrently with excavation. Shotcrete as soon as after blasting.	Systematic bolts 5–6 m long, spaced 1–1.5 m in crown and walls with wire mesh. Bolt invert.	150–200 mm in crown, 150 mm in sides and 50 mm on face.	Medium to heavy ribs spaced 0.75 m with steel lagging and fore-poling if required. Close invert.

Source: Bieniawski, 1979. Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

RMR classification. The criterion for rock mass strength is as follows:

$$\frac{\sigma_1}{\sigma_c} = \frac{\sigma_3}{\sigma_c} + \sqrt{m\frac{\sigma_3}{\sigma_c} + s} \qquad (10.5.4)$$

where σ_1 is major principal stress at failure, σ_3 is applied minor principal stress, σ_c is uniaxial compressive strength of the rock material, and m and s are constants dependent upon the properties of the rock and the extent to which it has been fractured by being subjected to σ_1 and σ_3. For intact rock, $m = m_i$ is determined from a fit of the above equation to triaxial test data from laboratory specimens, taking $s = 1$ for rock material.

For rock masses, the constants m and s are related to the basic (unadjusted) rock mass rating RMR, as follows (Hoek and Brown, 1988):

For *undisturbed rock masses* (smooth-blasted or machine-bored excavations):

$$m = m_i \exp[(\text{RMR} - 100)/28] \qquad (10.5.5)$$

$$s = \exp[(\text{RMR} - 100)/9]$$

For disturbed rock masses (slopes or blast-damaged excavations):

$$m = m_i \exp[(\text{RMR} - 100)/14] \qquad (10.56)$$

$$s = \exp[(\text{RMR} - 100)/6]$$

Significant values are summarized in Table 10.5.7.

An alternative criterion for estimating rock mass strength also involves the use of RMR. A rock failure criterion proposed by Bieniawski (1974) was modified by Yudhbir (1983) with respect to the constant A, as follows :

$$\frac{\sigma_1}{\sigma_c} = A + B\left[\frac{\sigma_3}{\sigma_c}\right]^{\alpha} \qquad (10.5.7)$$

where a = 0.75 and B depends on rock type as follows: shale and limestone, B = 2; siltstone and mudstone, B = 3; sandstone and quartzite, B = 4; norite and granite, B = 5.

In the Bieniawski criterion, A = 1 (for *intact rock*), but for *rock masses* Yudhbir determined that A is a function of the rock-mass quality as follows:

$$A = 20\,m\,(0.0765\ \text{RMR} - 7.65) \qquad (10.5.8)$$

Database: The database used for the development of a rock mass classification may indicate the range of its applicability. For example, the RMR system involved originally 49 case histories followed by 62 case histories added by Newman and Bieniawski (1987) and a further 78 cases collected between 1984 and 1987. For the database represented by all 351 case histories, see Bieniawski (1989).

Q System: This system of rock mass classification was developed in Norway in 1974 by Barton et al., all of the Norwegian Geotechnical Institute. Its development represented a major contribution to the subject of rock mass classification for a number of reasons: (1) the system was proposed on the basis of an analysis of 212 tunnel case histories, mostly from Scandinavia, (2) it is a quantitative classification system, and (3) it is an engineering system facilitating the design of tunnel supports.

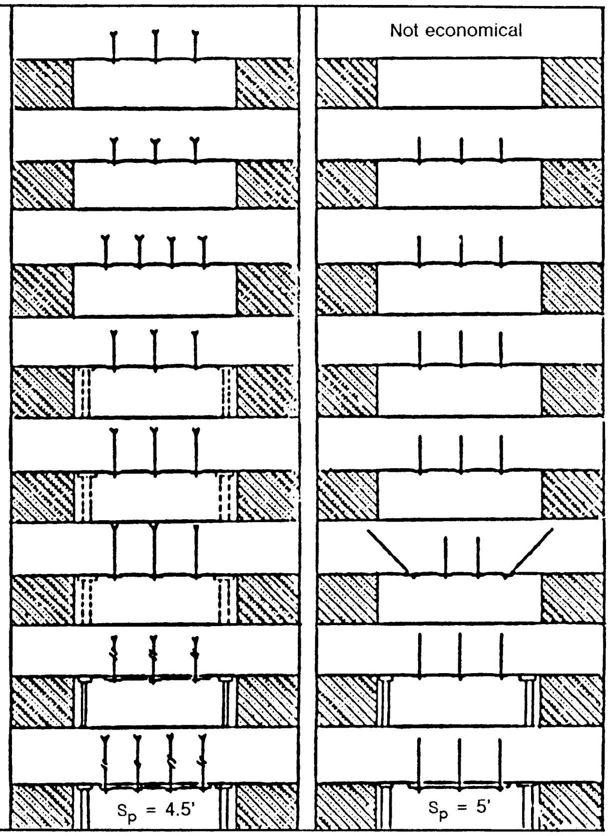
The Q system is based on a numerical assessment of the rock mass quality using six different parameters: (1) RQD, (2) number

Table 10.5.5. Rock Support Selection Chart for Coal Entries 20 ft wide

Roof rock class	Rock mass rating (RMR)	Rock load height (ft)	Support specifications: Mechanical bolts	Support specifications: Resin bolts
I Very good	90	2.0	L: 2.5' S: 5' x 5' G: 40 ø: 5/8" C: 6.2 tons	
	80	4.0	L: 2.5' S: 5' x 4.5' G: 60(40) ø: 3/4"(7/8") C: 11 tons	L: 2.5' S: 5' x 5' G: 60 ø: 3/4" C: 12 tons
II Good	70	6.0	L: 3.0' S: 4' x 4' G: 60 ø: 3/4" C: 10 tons	L: 3.0' S: 5' x 5' G: 60 ø: 3/4" C: 18 tons
	60	8.0	L: 4.0' S: 5' x 5' G: 60 ø: 5/8" C: 9 tons	L: 4.0' S: 5' x 5' G: 60 ø: 1" C: 23.7 tons
III Fair	50	10.0	L: 5.0' S: 5' x 5' G: 40 ø: 3/4" C: 8 tons	L: 4.0' S: 5' x 4' G: 60 ø: 1" C: 23.7 tons
	40	12.0	L: 6.0' S: 5' x 5' G: 40 ø: 3/4" C: 7 tons	L: 4.0' S: 4' x 4' G: 60 ø: 1" C: 23.7 tons
IV Poor	30	14.0	L: 7.0' S: 5' x 5' G: 40 ø: 5/8" C: 6 tons	L: 5' S: 5' x 5' G: 60 ø: 3/4" C: 12 tons
	20	16.0	L: 8.0' S: 4' x 4.5' G: 40 ø: 5/8" C: 5 tons	L: 5' S: 5' x 5' G: 60 ø: 3/4" C: 12 tons

Alternate support patterns

Mechanical bolts/posts | Resin bolts/posts



L = bolt length
S = bolt spacing
G = grade of steel
ø = bolt diameter
C = bolt capacity
$ø_p$ = post diameter
S_p = post spacing

Source: Unal, 1983. Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 ton = 0.9072 t.

of joint sets, (3) roughness of the most unfavorable joint or discontinuity, (4) degree of alteration or filling along the weakest joint, (5) water inflow, and (6) stress condition. These six parameters are grouped into three quotients to give the overall rock mass quality Q as follows:

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF} \qquad (10.5.9)$$

where RQD is rock quality designation, J_n is joint set number, J_r is joint roughness number, J_a joint alteration number, J_w is joint water reduction number, and SRF is stress reduction factor.

The rock quality can range from Q = 0.001 to Q = 1000 on a logarithmic rock mass quality scale.

In Table 10.5.8, the numerical values of each of the classification parameters are given. They are interpreted as follows: the first two parameters represent the overall structure of the rock mass, and their quotient is a relative measure of the block size. The quotient of the third and the fourth parameters is said to be an indicator of the interblock shear strength (of the joints). The fifth parameter is a measure of water pressure, while the sixth parameter is a measure of (1) loosening load in the case of shear zones and clay bearing rock, (2) rock stress in competent rock, and (3) squeezing and swelling loads in plastic incompetent rock. This sixth parameter is regarded as the *total stress* parameter.

Table 10.5.6. Support Recommendations for Mine Drifts

Spot bolting: Bolting to restrain limited areas or individual blocks of loose rock, primarily for safety.

Wide pattern bolting: Bolt spacing on 1.5 m to 1.8 m, or wider in very large openings.

Medium pattern bolting, with or without mesh or straps: Bolts spaced 0.9 m to 1.5 m, 23 cm wide straps or 100 mm welded wire mesh.

Close pattern bolting mesh, or straps: Bolt spacing less than 0.9 m, 100 mm welded wire mesh, 0.3 m straps, or chain link.

Medium pattern bolting with shotcrete: Bolts spaced 0.9 m to 1.5 m and 80 mm (nominal) of shotcrete. Light mesh for wet rock to alleviate shotcrete adherence problems.

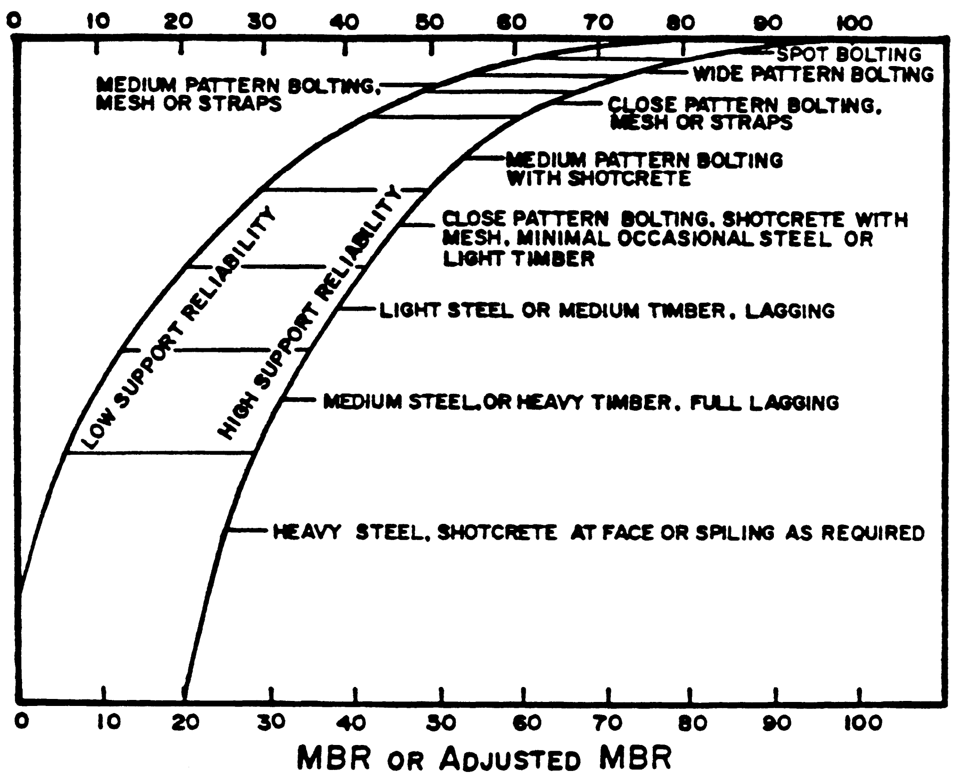
Close pattern bolting, shotcrete with mesh, minimal occasional steel or light timber: Bolt spacing less than 0.9 m with 100 mm welded wire mesh or chain link throughout, and nominal 100 mm of shotcrete. Localized conditions may require light wide-flange steel-sets or timber sets.

Light steel, medium timber, lagging: Bolting as required for safety at the face—full contact (grouted or Split Set) bolts only. Light wide-flange steel-sets or 0.25 m timber sets spaced 1.5 m, with full crown lagging and rib lagging in squeezing areas.

Medium steel, heavy timber, full lagging: Medium wide-flange steel-sets or 0.3 m timber sets spaced 1.5 m, fully lagged across the crown and ribs. Support to be installed as close to the face as possible.

Heavy steel, shotcrete at face or spilling as required: Heavy wide-flange steel-sets spaced 1.2 m, fully lagged on crown and ribs, carried directly to face. Spilling or shotcreting of face as necessary.

General: Bolting: bolts in spot bolting through close pattern bolting are considered to be 19 mm in diameter, fully grouted or resin-anchored standard rockbolts; mechanical anchors are acceptable in material of MBR > 60. Split-Set use is at the discretion of the operator.



Source: Kendorski et al., 1983. Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

The quotient of the fifth and the sixth parameters describes the *active stress.*

Barton et al. (1974) consider the parameters, J_n, Jr, and J_a, as playing a more important role than joint orientation, and if joint orientation had been included, the classification would have been less general. However, orientation is implicit in parameters J_r and J_a, because they apply to the most unfavorable joints.

The Q value is related to tunnel support requirements by defining the equivalent dimensions of the excavation. This equivalent dimension, which is a function of both the size and the purpose of the excavation, is obtained by dividing the span, diameter, or the wall height of the excavation by a quantity called the excavation support ratio (ESR). Thus,

$$\text{equivalent dimension} = \text{span or height (meters)/ESR} \tag{10.5.10}$$

The ESR is related to the use for which the excavation is intended and the degree of safety demanded, as now shown:

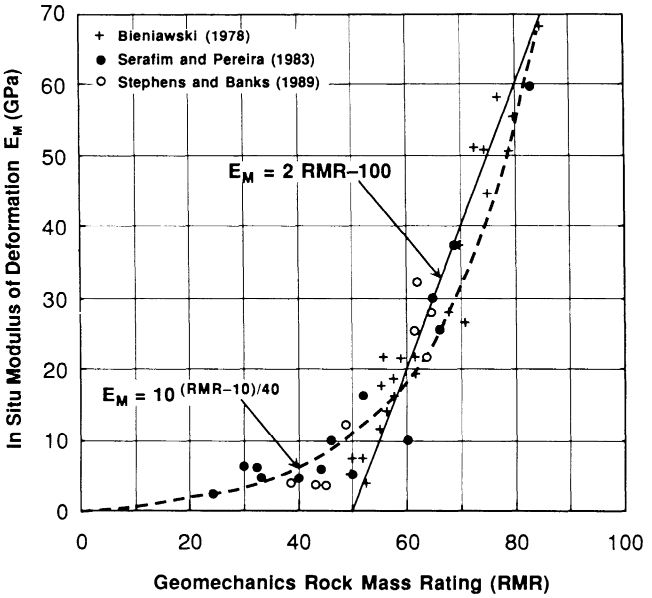


Fig. 10.5.6. Correlation between the in situ modulus of deformation and rock mass rating RMR (Serafim and Pereira, 1983). Conversion factor: 1 psi = 6.895 kPa.

Excavation category	*ESR*	*No. of cases*
A. Temporary mine openings	3–5	2
B. Vertical shafts:		
circular section	2.5	—
rectangular/square section	2.0	—
C. Permanent mine openings, water tunnels for hydropower (excluding high-pressure penstocks), pilot tunnels, drifts, headings for large excavations	1.6	83
D. Storage caverns, water treatment plants, minor highway and railroad tunnels, surge chambers, access tunnels	1.3	25
E. Power stations, major highway or railroad tunnels, civil defense chambers, portals, intersections	1.0	73
F. Underground nuclear power stations, railroad stations, factories	0.8	2

The relationship between the index Q and the equivalent dimension of an excavation determines the appropriate support measures, as depicted in Fig. 10.5.7 (a) and (b). These 38 support categories specify the estimates of permanent support. For temporary support determination, either Q is increased to 5 Q or ESR is increased to 1.5 ESR.

It should be noted that the length of rock bolts is not specified in the support guidelines in Fig.10.5.7, but the bolt length L is determined from the equation,

$$L = 2 + 0.15B/\mathrm{ESR} \qquad (10.5.11)$$

where B is the excavation width.

The maximum unsupported span can be obtained as follows:

$$\text{Maximum span (unsupported)} = 2(\mathrm{ESR})Q^{0.4} \qquad (10.5.12)$$

The relationship between the Q value and the permanent support pressure P_{roof} is calculated from the following relation:

$$P_{\mathrm{roof}} = (2.0/J_r)Q^{-1/3} \qquad (10.5.13)$$

If the number of joint sets is less than three, the equation is expressed as:

$$P_{\mathrm{roof}} = 2/3\, J_n^{1/2} J_r^{-1} Q^{-1/3} \qquad (10.5.14)$$

Case histories used to develop the Q system consisted mainly of tunnels and chambers from Scandinavia (Sweden and Norway), including 97 cases reported by Cecil (1970).

Correlations: A correlation was proposed between the RMR and the Q index (Bieniawski, 1976). Based on 111 case histories analyzed for this purpose (involving 62 Scandinavian cases, 28 South African cases, and 21 case histories from the United States, Canada, Australia, and Europe), the following relationship was found for civil engineering tunnels:

$$\mathrm{RMR} = 9\, \ln Q + 44 \qquad (10.5.15)$$

For mining tunnels, Abad et al. (1983) analyzed 187 coal mine roadways in Spain arriving at this correlation:

$$\mathrm{RMR} = 10.5\, \ln Q + 42 \qquad (10.5.16)$$

Moreno Tallon (1982) in Spain, Jethwa et al. (1982) in India, and Trunk and Honish (1989) in Germany further substantiated the correlation by Bieniawski (1976) on the basis of 59 projects.

Example 10.5.1. Find rock mass quality by a classification system.

Given: Rock mass in horizontal bedding, 500 ft (152 m) depth.
Rock material: shale.
Uniaxial compressive strength: 5800 psi (50 MPa).
Strata conditions: separation < 0.04 in. (1 mm); slightly weathered, slightly rough surfaces, no infilling.
RQD = 60%. Three discontinuity sets.
Spacing of main discontinuities: 6 in. (150 mm).
Groundwater conditions: damp.
In situ stresses: horizontal stress = 2.5 vertical stress.

Solution.

RMR System

Rating due to uniaxial compressive strength	5
Rating due to RQD	12
Rating due to discontinuity spacing	7
Rating due to condition of discontinuities	17
Rating due to ground water conditions	10
Rating due to strike and dip of discontinuities (Horizontal bedding = "fair" orientation)	− 5
Rock Mass Rating (RMR)	46
Factor due to in situ stresses: 0.9 Adjusted RMR is	41

Q system

RQD = 60
J_n = 9
J_r = 1.25
J_a = 2
J_w = 0.66
SRF = 1

Q = 60/9 × 1.25/2 × 0.66/l = 2.75

Table 10.5.7. Approximate Relationship between Rock Mass Quality and Material Constants (after Hoek and Brown, 1988)

Disturbed rock mass m and s values		Undisturbed rock mass m and s values				
EMPIRICAL FAILURE CRITERION $\sigma_1 = \sigma_3 + \sqrt{m\sigma_c\sigma_3 + s\sigma_c^2}$ σ_1 = major principal effective stress σ_3 = minor principal effective stress σ_c = uniaxial compressive strength of intact rock, and m and a are empirical constants.		Carbonate Rocks with Well Developed Crystal Cleavage *dolomite, limestone and marble*	Lithified Argillaceous Rocks *mudstone, siltstone, shale and slate (normal to cleavage)*	Arenaceous Rocks with Strong Crystals and Poorly Developed Crystal Cleavage *sandstone and quartzite*	Fine Grained Polyminerallic Igneous Crystalline Rocks *andesite, dolerite, diabase and rhyolite*	Coarse Grained Polymineralic Igneous & Metamorphic Crystalline Rocks *amphibolite, gabbro gneiss, granite, norite, quartz-diorite*
INTACT ROCK SAMPLES						
Laboratory size specimens free from	m	7.00	10.00	15.00	17.00	25.00
discontinuities	s	1.00	1.00	1.00	1.00	1.00
RMR = 100	m	7.00	10.00	15.00	17.00	25.00
Q = 500	s	1.00	1.00	1.00	1.00	1.00
VERY GOOD QUALITY ROCK MASS						
Tightly interlocking undisturbed rock with	m	2.40	3.43	5.14	5.82	8.56
unweathered joints at 1 to 3m.	s	0.082	0.082	0.082	0.082	0.082
RMR = 85	m	4.10	5.85	8.78	9.95	14.63
Q = 100	s	0.189	0.189	0.189	0.189	0.189
GOOD QUALITY ROCK MASS						
Fresh to slightly weathered rock, slightly	m	0.575	0.821	1.231	1.395	2.052
disturbed with joints at 1 to 3m.	s	0.00293	0.00293	0.00293	0.00293	0.00293
RMR = 65	m	2.006	2.865	4.298	4.871	7.163
Q = 10	s	0.0205	0.0205	0.0205	0.0205	0.0205
FAIR QUALITY ROCK MASS						
Several sets of moderately weathered	m	0.128	0.183	0.275	0.311	0.458
joints spaced at 0.3 to 1m.	s	0.00009	0.00009	0.00009	0.00009	0.00009
RMR = 44	m	0.947	1.353	2.030	2.301	3.383
Q = 1	s	0.00198	0.00198	0.00198	0.00198	0.00198
POOR QUALITY ROCK MASS						
Numerous weathered joints at 30–500mm,	m	0.029	0.041	0.061	0.069	0.102
some gouge. Clean compacted waste rock	s	0.000003	0.000003	0.000003	0.000003	0.000003
RMR = 23	m	0.447	0.639	0.959	1.087	1.598
Q = 0.1	s	0.00019	0.00019	0.00019	0.00019	0.00019
VERY POOR QUALITY ROCK MASS						
Numerous heavily weathered joints spaced	m	0.007	0.010	0.015	0.017	0.025
< 50mm with gouge. Waste rock with fines.	s	0.0000001	0.0000001	0.0000001	0.0000001	0.0000001
RMR = 3	m	0.219	0.313	0.469	0.532	0.782
Q = 0.01	s	0.00002	0.00002	0.00002	0.00002	0.00002

10.5.4 ROCK MASS PROPERTIES AND IN SITU STRESSES

The ground-control engineer is confronted with rock as a rock mass, that is, an assemblage of blocks of rock material separated by various types of geologic discontinuities such as joints, bedding planes, shears, and faults. Consequently, both the engineering properties of intact rock and of rock masses must be considered. The tests and properties for intact rock are dealt with in Chapter 10.2.

Unfortunately, laboratory-measured quantities obtained from tests conducted on small rock specimens generally do not yield data that are directly applicable to the in situ rock mass from which the specimens were taken. If laboratory test results could be scaled reliably to field conditions, then small scale, easily controllable tests would provide a convenient means for estimating rock mass characteristics.

10.5.4.1 Rock Mass Properties

Rock mass characteristics of particular significance in ground control are as follows:

1. Modulus of deformation or elasticity, essential for the design of drifts, chambers, entries, and mine floors.
2. Compressive strength, important in the design of mine pillars.
3. Shear strength, important in rock slopes, foundations, and tailings dams.
4. Tensile strength, important in mine roofs.
5. Frictional properties (cohesion and friction angle) important in fractured masses, yield zones, and mine floor design.
6. Bearing capacity, important for mine floors.

The concept of the *critical size* (Bieniawski, 1967; see also 10.2.3.2), depicted in Fig. 10.5.8, represents the phenomenon of strength reduction with increasing specimen size. It will be noted

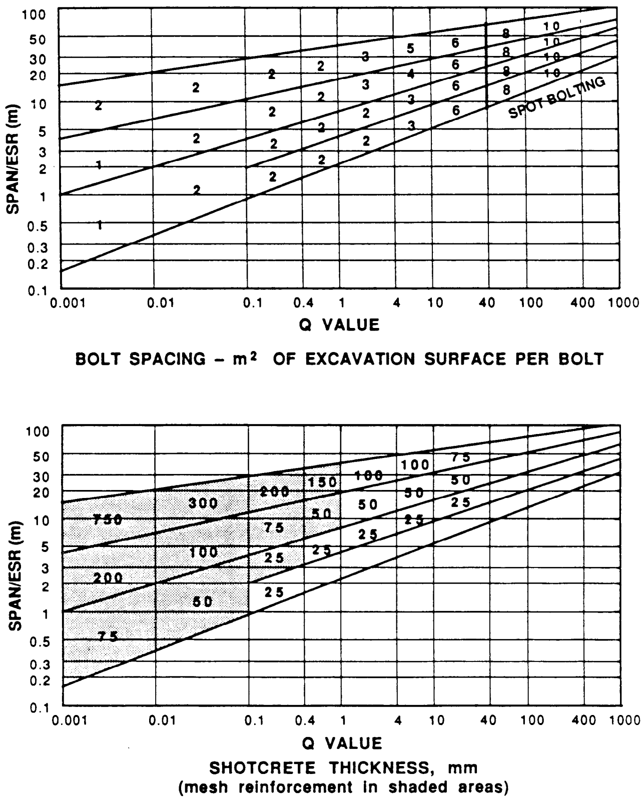


Fig. 10.5.7. Tunnel support requirements in accordance with the Q system (Barton et al., 1974). Conversion factor: 1 ft = 0.3048 m.

that from a certain size onward, the in situ strength remains constant, irrespective of specimen size. The significance of the phenomenon of critical size is that the strength values at the critical specimen size are directly applicable to full-size rock pillars.

10.5.4.2 In Situ Stresses

When an excavation is made in a rock mass, the initial in situ stresses are disturbed and redistributed in the vicinity of the excavation. In situ stresses can therefore be categorized into the virgin (original) stresses and induced (due to excavation) stresses.

The virgin stresses can be of gravitational, tectonic, or residual type. Gravitational stress is due to the effect of gravity on the overburden rock. Tectonic stress is due to previous or present day straining in the earth's crust, for example, regional faulting. Residual stress is the stress remaining after the cause has been removed, for example, thermal stresses or stresses caused by swelling or heat.

Rock stress measurements featuring a variety of techniques have been conducted throughout the world. These techniques may involve either overcoring methods, flat jack techniques, or the hydrofracturing method. Full details of these methods are given in Chapter 10.3.

Fig. 10.5.9 shows the accumulated results of stress measurements obtained in certain parts of the world, featuring a variation of the vertical stress with depth. A clear trend in the data is apparent. However, there is little consistency if a similar figure is constructed featuring the change of the horizontal stress with depth. As depicted in Fig. 10.5.10, there is no unique correlation between the ratio of the horizontal to vertical stresses and depth. It can also be seen that at a depth of about 1500 ft (450 m), the horizontal stresses are greater than the vertical stress. This is a worldwide phenomenon, but a difference of opinion exists as to the actual relationship governing the horizontal stresses. A conclusion that can be reached from this discussion is that while the vertical stresses may be predicted on the basis of the depth of the overburden with an accuracy sufficient for engineering purposes, no simple method exists for estimating the horizontal stresses. Consequently, in situations where such stresses are of great importance, stress measurements should be conducted. Note that Aggson (1978), Haimson (1978), and Linder and Halpern (1978) have compiled information on the principal stress magnitudes and directions in the United States.

10.5.5 DESIGN OF ROCK BOLTING REINFORCEMENT

Rock reinforcement must utilize the structural properties of the rock mass to improve the stability of underground excavations. The principal objective in the design of underground support is to help the rock mass support itself. The mechanics of rock reinforcement includes the concept of rock-support interaction as the main principle. The design of rock reinforcement systems depends on the geotechnical properties of the discontinuities and of the intact rock, the size and shape of the excavations, the magnitudes of redistributed stresses, and the degree of deformation acceptable in the completed excavation.

10.5.5.1 Mechanism of Rock-Support Interaction

The behavior of an opening and the performance of the support system depend upon the load-deformation characteristics of the rock and of the support, as well as of the manner and timing of support installation. The interaction between support and rock mass is qualitatively illustrated by the *ground reaction curve* given in Fig. 10.5.11. This concept was developed in detail by Deere et al. (1970), but was discussed by Fenner in Austria as early as 1938. More recently, the concept was studied by Brown et al. (1983).

When a tunnel is excavated, the rock moves inwards. The ground reaction curve displays the load that must be applied to the roof or the walls of the tunnel to prevent further movement. The movement which occurs before support can be installed is denoted by the line OA. If the support were perfectly incompressible, the load of the support would be represented by the line AA'. Support, however, deforms and, with the walls of the tunnel also deforming, an equilibrium is attained at point C at a radial displacement of the walls equal to OB and a support deformation equal to AB, at which stage the support load is BC.

However, the equilibrium at point C is only reached if the support is properly designed and placed timely. Line AeE in Fig. 10.5.11 depicts support that yields before stabilizing the opening, line AF represents support that is too flexible, while line GH is support too delayed in installation, hence ineffective. Accordingly, it is important to note that support should be installed as soon as possible so that the early rock deformation could stress the support at the same time as the rock mass is generating interior arching movement and shear stress in an attempt to be self-supporting. Furthermore, the less competent the rock, the earlier the support should be installed. Clearly, then, active rock reinforcement will be more effective and will require less capacity than passive support, but installation must take place as soon as possible after each face advance. Active rock reinforcement will require less support because the ability of the rock to support

Table 10.5.8. Classification of Individual Parameters Used in the Q System

	Description	*Value*	*Notes*
1.	**Rock Quality Designation**	(RQD)	
A.	Very poor	0– 25	(I) Where RQD is reported or measured as ≤ 10, (including 0) a nominal value of 10 is used to evaluate Q.
B.	Poor	25– 50	
C.	Fair	50– 75	
D.	Good	75– 90	
E.	Excellent	90–100	(II) RQD intervals of 5, i.e., 100,95,90, etc. are sufficiently accurate.
2.	**Joint Set Number**	(J_n)	
A.	Massive, no or few joints	0.5–1.0	
B.	One joint set	2	
C.	One joint set plus random	3	
D.	Two joint sets	4	
E.	Two joint sets plus random	6	
F.	Three joint sets	9	(I) For intersections use ($3.0 \times J_n$)
G.	Three joint sets plus random	12	(II) For portals use ($2.0 \times J_n$)
H.	Four or more joint sets, random, heavily jointed, "sugar cube" etc.	15	
J.	Crushed, rock, earthlike	20	
3.	**Joint Roughness Number**	(J_r)	
	(a) Rock wall contact and *(b) Rock wall contact before 10 cm shear*		
A.	Discontinuous joints	4	
B.	Rough or irregular, undulating	3	
C.	Smooth, undulating	2	
D.	Slickensided, undulating	1.5	(I) Descriptions refer to small scale features and intermediate scale features, in that order.
E.	Rough or irregular, planar	1.5	
F.	Smooth, planar	1.0	
G.	Slickensided, planar	0.5	(II) Add 1.0 if the mean spacing of the relevant joint set is greater than 3 m.
	(c) No rock wall contact when sheared		
H.	Zone containing clay minerals thick enough to prevent rock wall contact	1.0	(III) J_r = 0.5 can be used for planar slickensided joints having lineations, provided the lineations are orientated for minimum strength
J.	Sandy, gravelly or crushed zone thick enough to prevent rock wall contact	1.0	
4.	**Joint Alteration Number**	(J_a)	**(ϕ_r approx.)**
	(a) Rock wall contact		
A.	Tightly healed, hard non-softening, impermeable filling i.e. quartz or epidote.	0.75	(—)
B.	Unaltered joint wall, surface staining only.	1.0	(25–35°)
C.	Slightly altered joint walls. Non-softening mineral coatings, sandy particles, clay-free disintegrated rock etc.	2.0	(25–30°)
D.	Silty-, or sandy-clay coatings, small clay fraction (non-soft).	3.0	(20–25°)
E.	Softening or low friction clay mineral coatings, i.e. kaolinite or mica. Also chlorite, talc, gypsum, graphite etc., and small quantities of swelling clays.	4.0	(8–16°)
	(b) Rock wall contact before 10 cm shear		
F.	Sandy particles, clay-free disintegrated rock etc.	4.0	(25–30°)
G.	Strongly over-consolidated non-softening clay mineral fillings (continuous, but < 5mm thickness).	6.0	(16–24°)
H.	Medium or low over-consolidation, softening, clay mineral fillings, (continuous but < 5 mm thickness).	8.0	(12–16°)
J.	Swelling clay fillings, i.e. montmorillonite (continuous, but < 5 mm thickness) Value of J_a depends on percent of swelling clay-size particles, and access to water etc.	8–12	(6–12°)
	(c) No rock wall contact when sheared		
K. L. M.	Zones or bands of disintegrated or crushed rock. and clay (see G,H,J for description of clay condition).	6,8, or 8-12	(6–24°)
N.	Zones or bands of silty-or sandy-clay, small clay fraction (non-softening).	5.0	(—)
O,P. R.	Thick, continuous zones or bands of clay (see G,H,J, for description of clay condition).	10,13, or 13-20	(6–24°)

Table 10.5.8.—cont.

	Description	*Value*		*Notes*
5.	**Joint Water Reduction Factor**	(J_w)	**Approx. water pressure (kp/cm^2)**	
A.	Dry excavations or minor inflow, i.e. < 5 l/min. locally.	1.0	< 1.0	
B.	Medium inflow or pressure, occasional outwash of joint fillings.	0.66	1–2.5	
C.	Large inflow or high pressure in competent rock with unfilled joints	0.5	2.5–10	(I) Factors C to F are crude estimates. Increase J_w if drainage measures are installed.
D.	Large inflow or high pressure, considerable outwash of joint fillings	0.33	2.5–10	
E.	Exceptionally high inflow or water pressure at blasting, decaying with time	0.2–0.1	>10	(II) Special problems caused by ice formation are not considered.
F.	Exceptionally high inflow or water pressure continuing without noticeable decay	0.1–0.05	>10	

6.	**Stress Reduction Factor**		
	(a) Weakness zones intersecting excavation, which may cause loosening of rock mass when tunnel is excavated	(SRF)	
A.	Multiple occurrences of weakness zones containing clay or chemically disintegrated rock, very loose surrounding rock (any depth).	10	(I) Reduce these values of SRF by 25–50% if the relevant shear zones only influence but do not intersect the excavation.
B.	Single weakness zones containing clay or chemically disintegrated rock (depth of excavation < 50 m).	5	
C.	Single weakness zones containing clay or chemically disintegrated rock (depth of excavation > 50 m).	2.5	
D.	Multiple shear zones in competent rock (clay-free), loose surrounding rock (any depth).	7.5	
E.	Single shear zones in competent rock (clay-free) (depth of excavation < 50 m).	5.0	
F.	Single shear zones in competent rock (clay-free) (depth of excavation > 50 m).	2.5	
G.	Loose open joints, heavily jointed or "sugar cube" etc. (any depth).	5.0	

	(b) Competent rock, rock stress problems	σ_c/σ_1	σ_t/σ_1	(SRF)	
H.	Low stress, near surface	> 200	>13	2.5	(II) For strongly anisotropic virgin stress field (if measured): when $5 \leq \sigma_1/\sigma_3 \leq 10$, reduce σ_c and σ_t to 0.8 σ_c and 0.8 σ_t. When $\sigma_1/\sigma_3 > 10$, reduce σ_c and σ_t to 0.6 σ_c and 0.6 σ_t, where: σ_c = unconfined compression strength, and σ_t = tensile strength (point load), and σ_1 and σ_3 are the major and minor principal stresses.
J.	Medium stress	200–10	13–0.66	1.0	
K.	High stress, very tight structure (usually favourable to stability, may be unfavourable for wall stability)	10–5	0.66–0.33	0.5–2	
L.	Mild rock burst (massive rock)	5–2.5	0.33–0.16	5–10	
M	Heavy rock burst (massive rock)	2.5	0.16	10–20	(III) Few case records available where depth of crown below surface is less than span width. Suggest SRF increase from 2.5 to 5 for such cases (see H).

	(c) Squeezing rock: plastic flow of incompetent rock under the influence of high rock pressure	
N.	Mild squeezing rock pressure	5–10
O.	Heavy squeezing rock pressure	10–20
	(d) Swelling rock: chemical swelling activity depending on presence of water	
P.	Mild swelling rock pressure	5–10
R.	Heavy swelling rock pressure	10–15

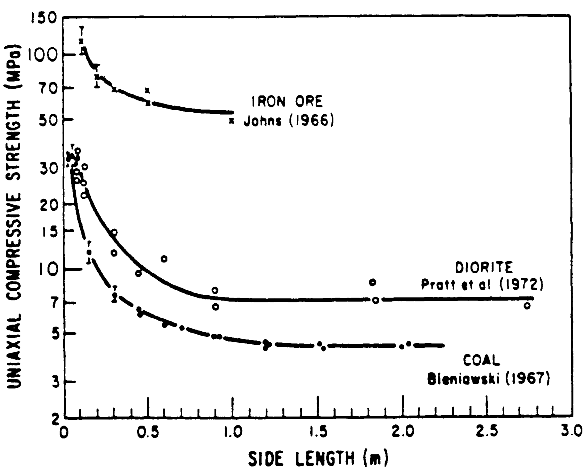
Table 10.5.8.—cont.

Additional Notes on the Use of the Above Table

When making estimates of the rock mass quality (Q) the following guidelines should be followed, in addition to the notes listed in the tables:

1. When borehole core is unavailable, RQD, can be estimated from the number of joints per unit volume, in which the number of joints per metre for each joint set are added. A simple relation can be used to convert this number to RQD for the case of clay free rock masses:
   $RQD = 115 - 3.3\ J_V$ (approx.) where J_V = total number of joints per m^3
   ($RQD = 100$ for $J_V < 4.5$)
2. The parameter J_n representing the number of joint sets will often be affected by foliation, schistosity, slaty cleavage or bedding etc. If strongly developed these parallel "joints" should obviously be counted as a complete joint set. However, if there are few "joints" visible, or only occasional breaks in the core due to these features, then it will be more appropriate to count them as "random joints" when evaluating J_n.
3. The parameters J_r and J_a (representing shear strength) should be relevant to the *weakest significant joint set or clay filled discontinuity* in the given zone. However, if the joint set or discontinuity with the minimum value of (J_r/J_a) is favourably oriented for stability, then a second, less favourably oriented joint set or discontinuity may sometimes be more significant, and its higher value of J_r/J_a should be used when evaluating Q. *The value of J_r/J_a should in fact relate to the surface most likely to allow failure to initiate.*
4. When a rock mass contains clay, the factor SRF appropriate to *loosening loads* should be evaluated. In such cases the strength of the intact rock is of little interest. However, when jointing is minimal and clay is completely absent the strength of the intact rock may become the weakest link, and the stability will then depend on the ratio rock-stress/rock-strength. A strongly anisotropic stress field is unfavourable for stability and is roughly accounted for as in note II in the table for stress reduction factor evaluation.
5. The compressive and tensile strengths (σ_c and σ_t) of the intact rock should be evaluated in the saturated condition if this is appropriate to present or future in situ conditions. A very conservative estimate of strength should be made for those rocks that deteriorate when exposed to moist or saturated conditions.

Source: Barton, Lien, and Lunde, 1974. Conversion factors: 1 in. = 25.4 mm = 2.54 cm, 1 ft = 0.3048 m, 1 psi = 6.895 kPa.



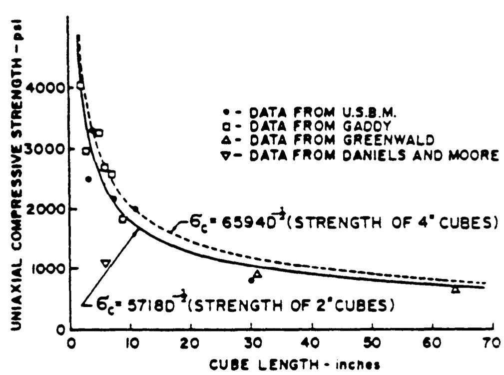


Fig. 10.5.8. Strength reduction in coal as a function of the size of cubical specimens tested in the laboratory and in situ (Bieniawski, 1967; Hustrulid, 1976). Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 psi = 6.895 kPa.

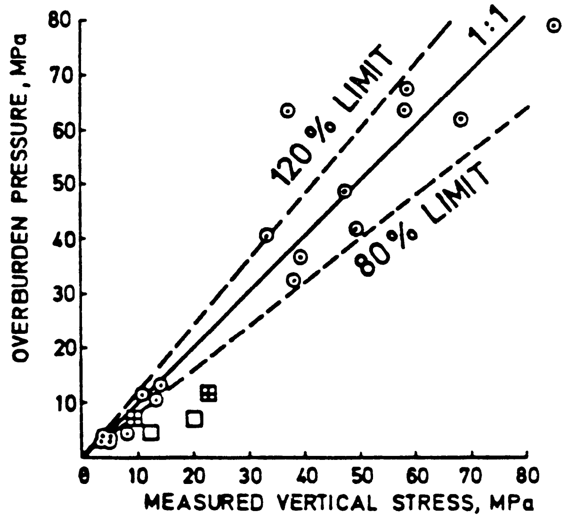
itself is being utilized, while with massive support the full weight of the rock is being supported.

Although the concept of the ground reaction curve has been studied extensively by Deere et al. (1970), Rabcewicz (1964), Pacher (1977), and Brown et al. (1983), the ground reaction curve cannot as yet be theoretically defined for rock masses. Furthermore, even if the theory could be used to predict the curve, Deere believes that the large local variations in construction procedures would inhibit the usefulness of the curve for the practical design of supports, not to mention that the load-deformation characteristics of some supports are also not clearly understood. The only possibility of obtaining quantitative data on the required support resistance and ground deformation behavior lies in measurements in situ. From measurements of the radial displacement of excavation surface and of the displacement inside the rock mass as a function of time during mining, the stabilization process as well as the load of the support can be established (Bieniawski, 1984).

Among the various methods of rock reinforcement, rock bolting stands out as the most effective technique. It was mechanical bolts that revolutionized to a great extent ground control in the American mining industry during the 1950s. The United States is credited with pioneering the use of rock bolts in underground excavations as early as 1930.

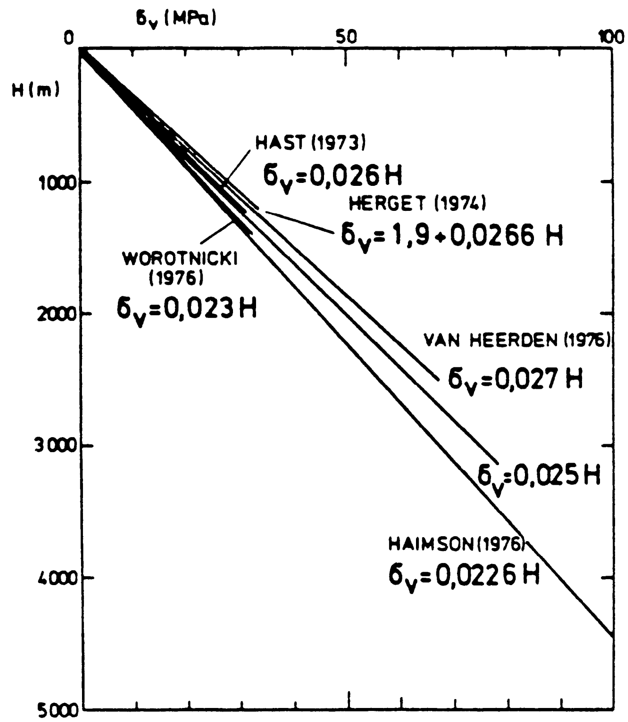
Although the first use of rock bolts is lost in antiquity, it was not until 1943 that their systematic use on a planned basis was recorded. Weigel (1943) described a system introduced by the St. Joseph Lead Co. in 1936 in Missouri (Panek, 1973).

Rock bolting first became popular in 1947 in the United States where it was promoted by the US Bureau of Mines (USBM) in an attempt to reduce the number of accidents caused by roof falls. In under two years, it had come into general use in the US mining industry. In 1949, the method was in use at over 200 mines, and by 1952, annual consumption had reached 25 million bolts. Growth of rock bolting was rapid. In 1968, the USBM reported that 912 coal mines used 55 million roof bolts annually and 60% of underground coal production was mined under bolted roof. In 1989, the Bureau of Mines estimated that about 120 million rock bolts were used annually in American mines, and over 90% of underground coal production was mined under bolted roof.



(a)

Correlation between calculated and measured vertical stresses.



(b)

Plot of vertical stresses versus depth below surface.

Fig. 10.5.9. Variations of vertical in situ stress with depth (Bieniawski, 1984). Conversion factors: 1 ft = 0.3048 m, 1 psi = 6.895 kPa.

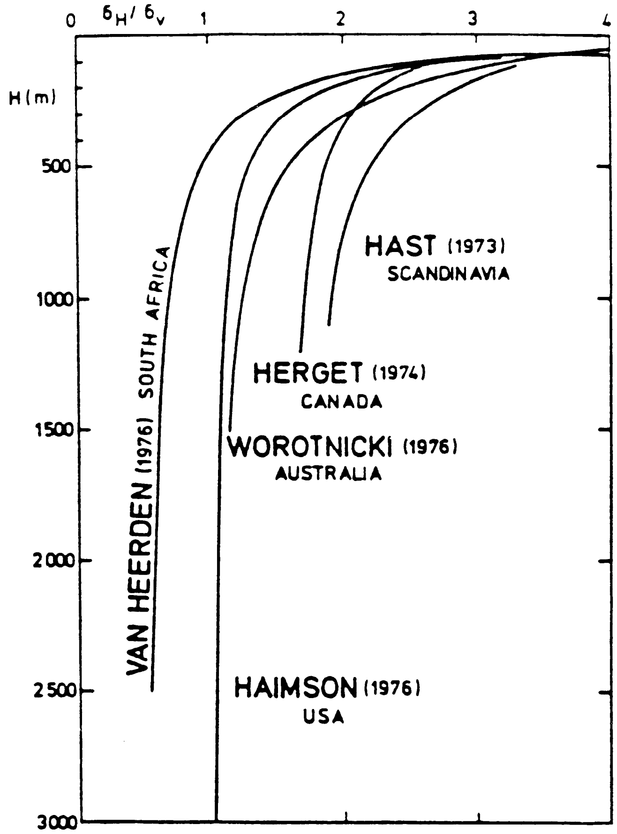
While there are many types of rock bolts, some of which are depicted in Fig. 10.5.12, it is estimated that the most popular type (in about 60% of mining) is the mechanical bolt, followed by a grouted bolt used for 30% of installations. The remaining 10% use other methods of rock bolting such as friction stabilizers (split sets), roof trusses, etc.

10.5.5.2 Mechanical Rock Bolts

Fig. 10.5.12 features a commonly used rock bolt that is mechanically anchored and tensioned. It should be noted that a large number of expansion-shell anchor designs are available, and these work well in all but very soft rocks. Failure of the system generally occurs in the threaded portions of the bolt at either the nut or the anchor ends.

Point anchors produce high stress concentrations against the rock. This results in rock fracture and a tendency for the anchor to slip or creep with time. This has led to the emergence of another version of mechanically anchored rock bolts, that is, one involving anchors imbedded in resin or grout. This system is similar to a standard mechanical rock bolt with the exception that the shell is embedded in resin which sets up around it, after which the bolt is tightened. Anchor creep is virtually eliminated with this system, and long-term anchorage is excellent.

After tensioning, grouting of mechanical rock bolts may be undertaken to ensure long-term corrosion protection.



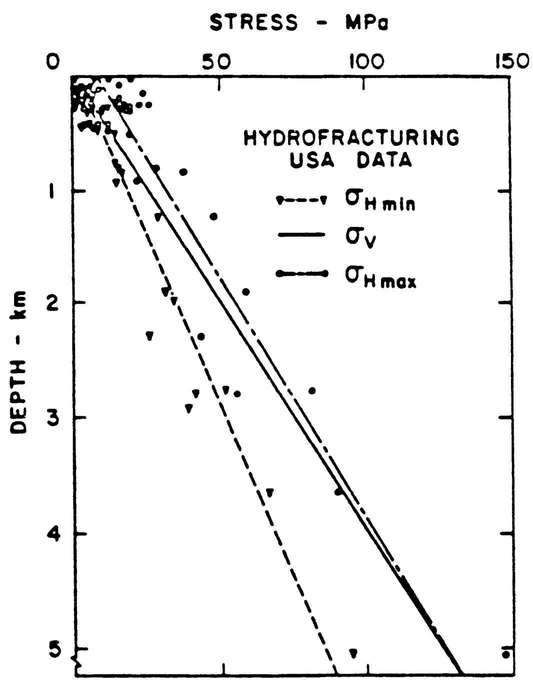


Fig. 10.5.10. Variations of horizontal in situ stresses with depth (Bieniawski, 1984). Conversion factors: 1 ft = 0.3048 m, 1 psi = 6.895 kPa.

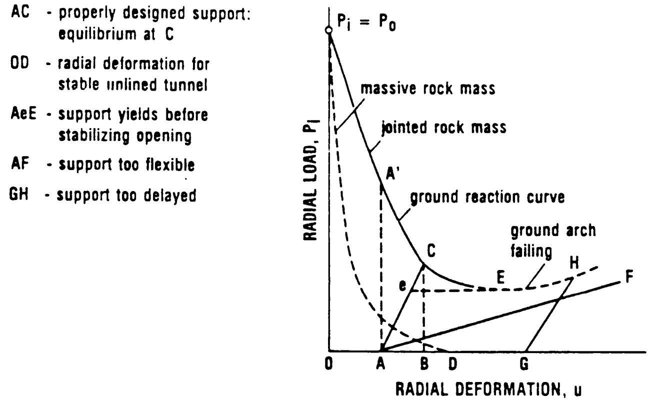


Fig 10.5.11. Concept of the ground reaction curve for rock tunnels (Deere et al., 1970).

Other Types of Mechanical Fixtures: The split-set rock reinforcement system (friction rock stabilizers), developed by Scott (1980) in conjunction with the Ingersoll-Rand Co., is widely used in American metal mines. The present range of usage is about 50% of the total fixtures in metal mines, or approximately 3.5 million bolts a year. The reinforcement element consists of a thin-walled steel tube of 1½-in. (38-mm) diameter, which is forced into a 1⅜-in. (35-mm) diameter drillhole. The spring action of the compressed steel tube induces a friction force along the length of the tube, and this frictional force anchors the reinforcing element in the rock. Assuming that the borehole diameter is accurately controlled, this system can be used very effectively. Installation is very rapid and support performance is good, provided that the split sets are installed close to a face and that stresses imposed are not very high. These devices have not been used to any significant extent in civil engineering tunnels because they cannot be grouted, are susceptible to corrosion, and hence cannot be considered as permanent support. They have also not been used much in coal mining because of problems with installing these rock bolts in low coal seams. However, equipment is under development for use in coal mines.

The "Swellex" rock reinforcement system, developed by Atlas Copco, is gaining popularity in Europe. This untensioned rock reinforcement system offers a number of advantages (Mattila and Boyd, 1985). The reinforcing element consists of a thin-walled tube of approximately 1⅝-in. (42-mm) diameter that has been folded into a collapsed shape of about l-in. (25-mm) diameter. This collapsed fixture can be inserted easily into a borehole of about 1½-in. (39-mm) diameter and, once in place, is expanded by injection of water at a pressure of about 3000 psi (20 MPa), which is generated by a small portable pump unit. Expansion of the fixture results in an overall length reduction, and this pulls the face plate right against the rock and induces a small tension in the device. The anchoring force is very high, and the strength of the system is limited by the strength of the tube. Although the system cannot be grouted, rusting is inhibited by the presence of a sealed volume of water inside the expanded fixture and by a protective coating that can be applied to the outside of the fixture.

Tensioning of Mechanical Rock Bolts: Mechanical bolts are always tensioned at the time of installation to a load level not exceeding 60% of their anchorage capacity or the yield strength of the bolt. With mechanical bolts, any differential rock sag stretches the bolt, introducing a tensile stress in the bolt and a compressive force on the rock countering the direction of movement. The magnitude of this force depends upon the length

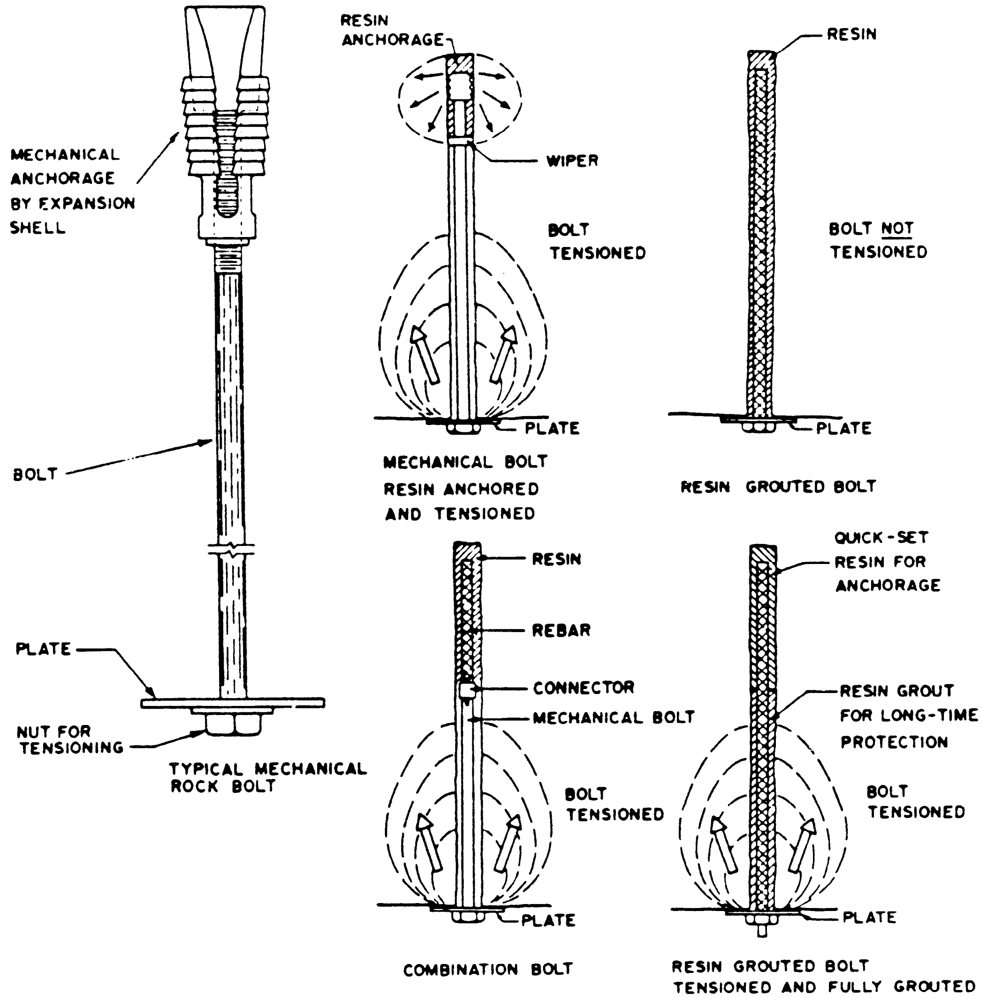


Fig. 10.5.12. Types of rock bolts commonly used (Peng and Tang, 1984).

of the bolt, and without tensioning, a large deformation would occur before the bolt mobilized sufficient support resistance. The purpose of tensioning is to prevent rock movement, and hence mechanical bolts are an active method of rock reinforcement. The bolts are tensioned to specified loads by means of a torque wrench or a hydraulic tensioning device. Grouting the bolt after tensioning ensures that the loads will be maintained in the bolts.

Mechanical bolts support the roof both by a suspension effect (Snyder et al., 1979) as well as through a friction effect (Panek, 1973). Lamination of the roof in coal mines requires that the individual layers slide across one another as they deform. This sliding action is opposed by friction, and the mechanical bolts act to increase the friction resistance by applying a compressive force normal to the friction surface.

10.5.5.3 Grouted Rock Bolts

If the circumstances are such that the installation of the rock reinforcement can take place before significant deformation of the excavation has occurred, the use of untensioned reinforcement is appropriate. With the introduction of fast-setting, high-strength resins in the early 1970s, it became possible to integrate fully grouted bolting into the high-speed mining cycle of US coal mines. Today approximately one-quarter of all rock bolts installed are resin-grouted bolts (Mahyera et al., 1981). Two other types of fully grouted nontensioned bolts are cement-grouted bolts and "pumpable" bolts (Zink and Wang, 1977; Serbousek and Bolstad, 1981). The grout may be either a resin or inorganic cement.

Grouted bolting is simple in concept (Fig. 10.5.12): a hole is drilled in the rock, and a rebar is inserted in it and grouted. The rebar should be threaded at one end, and the face plate should be fastened with a nut for added effectiveness. Grouted bolts are a passive reinforcement support since they apply little load to the rock unless movement occurs.

An advantage of resin bolts is their resistance to transverse shear (Fairhurst and Singh, 1974; Pells, 1974). The grouted bolt acts only where a crack in the rock tends to dilate, and all stretching in the bolt is confined to a very short section. Thus high unit stresses are developed in response to movement (Reed, 1974). In addition, resin bolts are effective in maintaining the initial friction between layers (Snyder et al., 1979).

The basic mechanism of resin bolting is a combination of the principles of suspension and beam building, although there is no consensus of opinions as to which effect is prevalent (Snyder et al., 1979).

Karabin and Debevec (1976) agree that the suspension mechanism applies to either a mechanical rock bolt or a grouted bar. Fully grouted resin bolts employ a full-length bond to bind the strata into an effective beam while mechanical bolts achieve this by relying on frictional interfaces developed through bolt tensioning. Moreover, an advantage of resin bolts is their resistance to transverse shear. They provide physical barriers to resist movement along the planes of discontinuities.

However, by comparison with mechanical bolts, untensioned resin-grouted bolts are a passive means of rock reinforcement. Hoek and Brown (1980) suggest that grouted rock bolts reinforce a rock mass in a fashion similar to that of steel bars in reinforced concrete.

In soft rock, the resin bolt assembly fails at the rock interface, indicating that the resin is stronger than the rock. In hard rock, a resin-grouted rebar fails by failure of the steel, but a smooth bolt pulls out, indicating that mechanical interlock is the primary means of bonding.

Types of Grouted Bolts: There are two main types of grouted bolts. The most common type features synthetic resin, which is introduced into the borehole in sealed capsules. These capsules contain the resin and a catalyst in two separate compartments, and mixing of these takes place when the plastic capsule is broken by insertion and rotation of the reinforcing steel rod. The chemical composition of the resin and catalyst can be varied to suit different ambient temperatures and to control setting times.

An advantage of resin bolts is that they are effective in weaker rock masses, but a disadvantage is that resin grouting is expensive. Recently, however, the USBM has developed an inorganic cement grout that is cost effective (Serbousek and Bolstad, 1981).

The other type is the Perfobolt system developed in Europe for grouting and anchoring rebars in boreholes. Perforated half tubes are packed with mortar, and the halves are wired together and inserted in the drillhole. The mortar is extruded when the rebar is pushed down the center of the tube. For a l-in. (25-mm) rebar diameter, the sleeve diameter is 1¼ in. (32 mm) while the borehole diameter is 1½ in. (38 mm).

The latest Mine Safety and Health Administration (MSHA) safety standards for rock bolts (MSHA Draft Regulation 1983-K75.202j) specify that if the tested grouted bolt does not withstand 150 ft-lb (203 N-m) of torque without rotating in the hole, corrective action must be taken or supplemental support installed.

10.5.5.4 Design Guidelines For Rock Bolt Systems

Design of rock bolt systems involves the selection of the following parameters: length of rock bolts, spacing of rock bolts, bolt capacity, bolt diameter, and bolt tension (if applicable).

Although the systematic use of rock bolts for strata stabilization in mining goes back nearly 50 years, the development of a rational basis for the design of rock-bolting systems proved to be elusive. Nevertheless, much progress has been made.

In early rock bolt research by the USBM, analytical studies and laboratory investigations with physical models were directed primarily towards clamping action incorporating the friction effect by tensioned bolts in laminated coal mine roof (Panek, 1973).

Concurrently with Panek, several rock behavior studies were conducted by Lang (1961). The emphasis was on heavily fractured ground rather than laminated mine roofs, and an extensive analysis of bolting patterns across various types of discontinuities was presented. It was found that even highly fragmented rock could be stabilized by rock bolting and form a load-carrying structural number that would span an opening. Bolts spaced closely enough produced a zone of uniform compression within the rock strata and were very effective at supporting the roof. The thickness of the beam of compressed rock was approximately the bolt length minus the spacing.

A major contribution to a rational design procedure for rock-bolt systems was provided by Lang and Bischoff (1982). They developed a rock bolting analysis to incorporate the shear strength developed by the rock mass on the vertical boundaries of the rock unit reinforced by a single rock bolt. The rock is assumed to be destressed to a zone width D, but variable vertical stresses σ_v and horizontal stresses $k\sigma_v$ are assumed to be induced with the destressed zone. Typically, k may be taken as 0.5. The shear strength developed at any point on the perimeter of the reinforced rock unit is given by $c + \mu k\sigma_v$, where c is the cohesion, and $\mu = \tan\phi$ is the coefficient of friction for the rock mass. Lang and Bischoff's analysis leads to the result,

$$\frac{T}{AR} = \frac{\alpha\gamma}{\mu k}\left(1 - \frac{c}{\gamma R}\right)\left[\frac{1 - exp(-\mu kD/R)}{1 - exp(-\mu kL/R)}\right] \qquad (10.5.17)$$

where T is rock bolt tension, A is area of roof carrying one bolt (equals s^2 for $s \times s$ bolt spacing), R is shear radius of the reinforced rock unit $= A/P$, where P is the shear perimeter (i.e., $4s$ for $s \times s$ bolt spacing), $a = 0.5$ for active support, and $a = 1.0$ for passive reinforcement, and L is bolt length that will often be less than D, the height of the destressed zone of rock, and g is rock density.

Lang and Bischoff suggest that, for preliminary analyses, the cohesion c should be taken as zero. Design charts based on Eq. 10.5.17 show that, particularly for low values of ϕ, the required bolt tension T increases significantly as L/s decreases below about two, but that no significant reduction in T is produced when L/s is increased above two. This result provides some corroboration of Lang's empirical rule that the bolt length should be at least twice the spacing for civil engineering applications.

In mining, the bolt length/bolt spacing ratio is acceptable between 1.2 and 1.5. The length of rock bolts may be selected as a function of the roof span, as depicted in Fig. 10.5.13.

Empirical Rules: Sources such as Hoek and Brown (1980), the US Corps of Engineers (Anon., 1980), and the USBM (Lang and Bischoff, 1982) provide information on general empirical rules that are useful as a check for bolt lengths and spacings. These rules are as follows:

1. *Minimum bolt length*
   Greatest of: (a) Twice the bolt spacing.
   (b) Three times the width of critical and potentially unstable rock blocks defined by the average discontinuity spacing in the rock mass.
   (c) For spans less than 20 ft (6 m), bolt length of one-half the span. For spans from 20 ft (6 m) to 60 ft (18 m), interpolate between 10 ft (3 m) and 15 ft (5 m) lengths, respectively. For excavations higher than 60 ft (18 m), sidewall bolts are one-fifth of wall height.
2. *Maximum bolt spacing*
   Least of: (a) One-half the bolt length.
   (b) One-and-one-half the width of critical and potentially unstable rock blocks.
   (c) 6 ft (2 m); greater spacing than 6 ft (2 m) makes attachment of wire mesh difficult.
3. *Minimum bolt spacing:* 3 ft (0.9 m).

NOTE: Where discontinuity spacing is close and the span is relatively large, the superposition of two bolting patterns may be appropriate, e.g., long heavy bolts on wide centers to support the span and shorter,

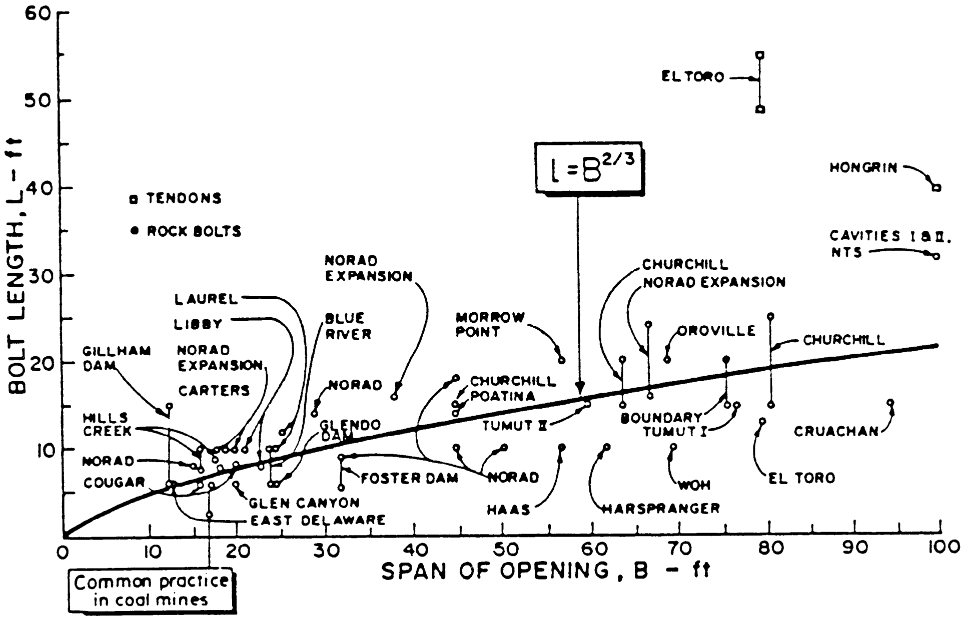


Fig. 10.5.13. Relationship between bolt length and roof span (Lang and Bischoff, 1982). Conversion factor: 1 ft = 0.3048 m.

thinner bolts on closer centers to stabilize the surface against raveling due to close jointing.

The above empirical rules should be considered only as general guidelines because the appropriate support system for a particular underground excavation depends on many factors such as the characteristics of the rock mass, in situ stress field, the expected loading history, safety regulations, availability, and cost of different types of support.

Hoek and Brown (1980) recommend that, once support estimates have been made, a working sketch should be prepared depicting the layout of rock bolting superimposed on the excavation cross section and the structural geologic features.

For coal mining applications, Title 30 of the *Code of Federal Regulations* (CFR) (Anon. 1989) states that openings should not exceed 20 ft (6.1 m) in width, where roof bolting is the sole means of support, nor should they exceed 30 ft (9.2 m) when roof bolts and other support, such as timber posts, are used. Section 75 of CFR 30 stipulates that in no case should the length of roofbolts be less than 30 in. (0.76 m) plus 1 ft (0.3 m) if anchored in the stronger strata to suspend the immediate roof. The bolt spacing and the distance between the bolt and the rib or the face should not be more than 5 ft (1.5 m). Mechanical bolts should be tensioned to 50% of the yield point of the bolt or the anchorage capacity. Miners may not work under an unsupported roof.

Support Selection Procedure: Hoek and Brown (1980) suggest the following procedure for estimating support requirements:

1. During site exploration, classify the rock mass by means of either the RMR system or the Q system. Check stability from Fig. 10.5.14.

2. Make a preliminary evaluation of the support system on the basis of the recommendations summarized in Tables 10.5.4, 10.5.5, or 10.5.6.

3. Estimate the in situ stress conditions from the measurements of stresses at the site or at adjacent sites in similar rock. If no measurements are available, obtain a crude first estimate

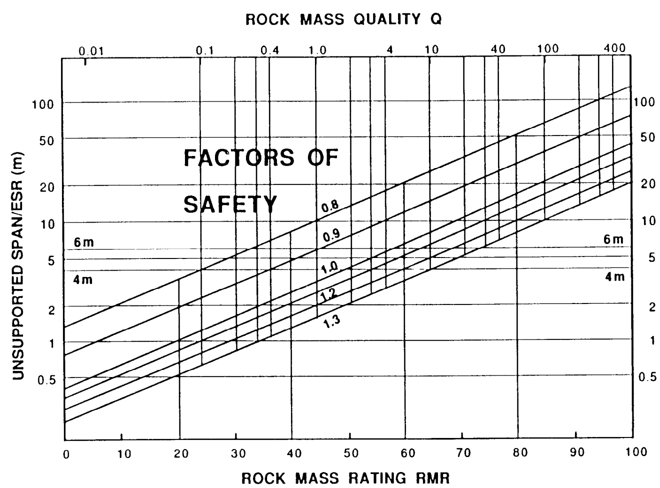


Fig. 10.5.14. Estimated factors of safety as a function of rock mass conditions and unsupported spans (Houghton and Stacey, 1980). Conversion factor: 1 ft = 0.3048 m.

based on the depth below surface and the compiled data, as shown in Fig. 10.5.9 and 10.5.10.

4. From analytical studies, estimate the maximum boundary stresses in the rock surrounding the proposed excavation.

5. If tensile stresses can occur on the excavation boundary, consider the extent of the tensile stress zone (from the minor principal stress contours), and explore the extent to which this zone could be eliminated or minimized by small changes in the excavation shape. If no such change is possible, consider the length of rock bolts that would be necessary to stabilize this zone, and estimate the capacity of this support from the volume of material subjected to tensile stress.

6. If only compressive stresses are present around the opening, compare the maximum boundary stress (excluding sharp corners) with the unconfined compressive strength of the rock mass given by $\sigma_{cM} = \sigma_c\sqrt{s}$ in accordance with Eq. 10.54. If the boundary stress exceeds the unconfined compressive strength of the rock mass, carry out a more detailed analysis of the extent of the potential failure zone and the directions of fracture propagation.

7. Whatever the outcome of the analyses described in steps 5 and 6, make an examination of the potential for structurally controlled instability. Using whatever geological information is available, examine the possibility of wedges or blocks forming by means of the sterographic techniques or the key-block theory recently developed by Goodman and Shi (1984). If the potential for this type of failure exists, consider the length and capacity of rock bolts required to stabilize the excavation.

8. Taking all these analyses into account, consider the consequences of excavation sequence and possible variations in the timing of support installation.

9. Check the rock bolt pattern design against preceding experience using the empirical rules described earlier and ensure that the various requirements do not conflict with one another.

10. Once these estimates have been made, prepare a drawing of a typical cross section of the excavation, superimposing an approximate pattern of structural features, the possible final excavation profile, the size and shape of the potential failure zone around the excavation, and the support system (draw to scale). Check to see if it appears correct and that there is enough room available to drill holes and install bolts.

Specifications: Peng and Tang (1984) offer these guidelines for mechanical bolts:

1. Bolt the roof as soon after mining as feasible to prevent strata separation.
2. Determine the optimum expansion shell and the optimum anchorage horizon through onsite pull-out tests.
3. Employ substantial structural components (i.e., high-strength bolts and plates with low deformation characteristics) to reinforce the strata effectively.
4. Ensure that all bolts work together by installing them with equal tension.

For fully grouted resin bolts, it is recommended to:

1. Employ quality components (i.e., resins, bolts, and bearing plates) that meet ASTM specifications.
2. Determine the optimum bolt-hole combinations and necessary length for best support.
3. Bolt newly exposed rock strata as soon as possible.
4. Closely follow the manufacturer's recommended installation procedures concerning borehole length, number of cartridges required, mix time, and holding time.
5. Conduct torque and pull-out tests to verify curing and system strength.
6. Store resin in cool places (underground) for maximum shelf life.

In general, a higher in situ horizontal stress improves the performance of roof bolts (Peng and Tang, 1984), which may be due to the apparent increased compressive strength of the rock with increased confining pressure and improved anchorage efficiency.

The spacing of the bolts should satisfy the criterion that the ratio of bolt length to bolt spacing should be about 1.5 generally in mining and 2.0 minimum in fractured rock and civil engineering tunneling (Lang and Bischoff, 1982). A bolt length-to-spacing ratio of 1:2 is considered the acceptable minimum for coal mine roofs reinforced by tensioned rock bolts.

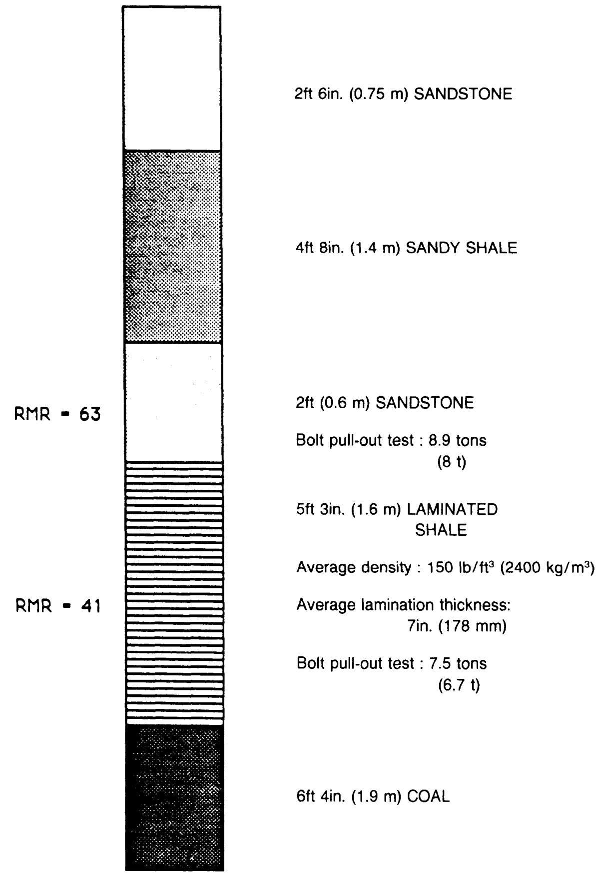


Fig. 10.5.15. Stratigraphic column (Ex. 10.5.2)

If roof bolting provides mainly a suspension effect, the roof bolts should be equally spaced. If there is no competent layer in the immediate roof, the function of roof bolting is to provide a friction effect and, in this case, the bolt density should be larger near the ribs with a smaller bolt density at the center of the excavation. The spacing between the bolt sets does not have to be on the usual square grid pattern but depends on the anchorage capacity of the anchoring layer.

Fully grouted resin bolts were suggested as being superior in ground control to mechanical bolts due to the following advantages (Peng and Tang, 1984): (1) anchorage is virtually guaranteed; (2) the bolt is permanently effective throughout its length; (3) fully grouted bolts resist both vertical and lateral strata movements; (4) the hole length is not critical; (5) fully grouted bolts seal wet holes and exclude air, thus reducing corrosion of the bolt assembly and weathering of rock; (6) resin-grouted bolts can absorb blast vibrations without bleed-off of the bolt load; and (7) time and labor for tensioning the bolts are not needed.

Example 10.5.2. A roof support system is to be designed for a 20-ft (6.1-m) wide coal mine entry. Consider the use of mechanical and resin-grouted bolts. The simplified stratigraphic column in Fig. 10.5.15 illustrates the coal seam and the overlying and underlying strata. The RMR value of the roof strata is 41. Depth of coal seam below surface is 500 ft (152 m).

Solution.

Step 1. Calculate the rock-load height.

The rock-load height h_t is given by:

$$h_t = \left(\frac{100 - \text{RMR}}{100}\right) B = \left(\frac{100 - 41}{100}\right) 20$$
$$= 11.8 \text{ ft} \cong 12 \text{ ft (3.7m)}$$

(If the RMR value is not known, the rock-load height can be estimated from roof-fall data by taking the roof-fall height as the rock-load height).

Step 2. Find the bolt length.

$$L = \frac{h_t}{2} = 6 \text{ ft}$$
$$\text{or } L = \text{SPAN}/3 = 20/3 = 6.7 \text{ ft}$$
$$\text{or } L = (\text{SPAN})^{2/3} = 7.2 \text{ ft}$$

Use L = 6 ft (1.78 m) as a practical bolt length. This would provide anchorage in the sandstone (Fig. 10.5.15).

Step 3. Find the bolt capacity.

The anchorage failure load P_f from pull-out tests is given as 8.9 tons (8.0 t). The yield load P_y of bolts can be checked for bolt diameter ϕ and grade of steel G from manufacturers' data, e.g.,

ϕ	G	P_y	G	P_y
⅝ in.	60	9.3 tons	40	6.2 tons
¾ in.	60	13.2 tons	40	8.8 tons

Select ¾-in. bolt, G = 60. Thus, bolt capacity is 8.9 tons.

Step 4. Calculate bolt spacing.

The bolt spacing is calculated using the suspension principle:

$$S = \sqrt{\frac{0.6\ P}{\gamma h}} = \sqrt{\frac{0.6 \times 17800}{150 \times 5.25}} = 3.68 \text{ ft}$$

Thus: S = 3 ft 7 in. (1.12 m). Use S = 4 ft (1.2 m).

Check: L/S = 1.5. Use four bolts per row.

Step 5. Find the bolt tension (mechanical bolts only) = 60% of bolt capacity.

Step 6. Compare mechanical bolting with resin bolting.

The final choice between the two types of roof bolting should be made on economic grounds. Select mechanical bolts.

Step 7. General considerations.

Always prepare a sketch of the rock bolt layout, to scale, to check whether it is practicable. It should be noted that in Ex. 10.5.2, no use is made either of the beam theory nor of the Panek chart for friction effect. Both these approaches are considered out of date: the first one because it needs the unrealistic assumption of the "modulus of rupture" (tensile strength), which is an unreliable parameter and rarely available at that; and the second, because it is oversimplified and has been abandoned by the Bureau of Mines where it was originally developed (Panek, 1973).

It should further be noted that no distinction is made in the example between the tensioned mechanical bolts and untensioned grouted bolts when it comes to the selection of the bolt length. This is so because the bolt length is the function of the excavation size and the rock mass quality. However, due to better anchorage characteristics, grouted bolts result in larger bolt spacing; they are also preferable for longer applications (e.g., in the main entries).

10.5.6 DETERMINATION OF PILLAR STRENGTH

Experimental results from tests on rock and coal show that there is a strength-reduction effect with increasing specimen size (Fig. 10.5.8).

The concept of *critical-size strength* (Bieniawski, 1968) for rock masses is very important in practical design. The critical size is defined as that specimen size at which a continued increase in specimen width causes no significant decrease in strength. Other authors (Jahns, 1966; Lama, 1971; Pratt et al., 1972) have confirmed that this critical-sized phenomenon exists in various rock types.

For coal, it was concluded by Bieniawski (1968) that 5-ft (1.5-m) cubic specimens constitute the critical-size value. Pariseau (1977) reported that the critical size for US western coal is 3 ft (0.9 m). Hustrulid (1976) pointed out that a critical size of 3 ft (0.9 m) would be generally applicable for coal for practical engineering purposes. This is evident from Fig. 10.5.8(b) where his data for the Pittsburgh coal seam are depicted.

The significance of the phenomenon of critical size is, of course, that the strength values at the critical size are directly applicable to full-sized pillars.

The size effect characterizes the difference in strength between the small-sized specimens tested in the laboratory and the large-sized pillars mined in situ. Research has shown (Hustrulid, 1976) that the scaling of coal properties from laboratory-measured data to field values can be satisfactorily achieved by the following equations (in customary English units):

$$\sigma_1 = \frac{k}{\sqrt{36}} \tag{10.5.18}$$

applicable to cubical pillars having a height h > 36 in. (0.9 m), or

$$\sigma_1 = \frac{k}{\sqrt{h}} \tag{10.5.19}$$

applicable to cubical pillars having a height less than 36 in. (0.9 m).

In the above equations, the constant k must be determined for the actual pillar material and is obtained as shown by Gaddy (1956):

$$k = \sigma_c \sqrt{D} \tag{10.5.20}$$

where σ_c is uniaxial compressive strength of rock specimens tested in the laboratory having a diameter or cube size dimension D (in inches). It should be noted that although there is a difference in laboratory results depending on whether cylindrical or cubical specimens are used, for practical engineering purposes this difference is not significant within the range of D between 2 to 4 in. (50 to 100 mm) (see Fig. 10.5.8b).

Typical k values for different coal seams are listed below:

Seam	k	Seam	k
Cameo (CO)	3200–7970	MaryLee (AL)	3000
Clintwood	4230–5200	Pittsburgh (PA, WV)	5550–5860(av.5580)
Elkhorn No. 4	6000–6250	Pocahontas	4310–4825
Harlan	8860–9460	Springfield #5 (IL)	4930
Herrin #6 (IL)	5500	Upper Freeport (PA)	1640
Marker	10120–10600	Winifrede (WV)	6510

10.5.6.1 Pillar Strength Formulas

Numerous pillar strength formulas have been proposed, but five formulas are used most commonly (Bieniawski, 1984; Peng, 1986). Each formula specifies its own appropriate factor of safety.

1. Obert-Duvall/Wang Formula: Obert and Duvall (1967) derived from laboratory tests on hard rock and elasticity considerations the same relationship as did Bunting in 1911. Greenwald et al. (1939) mention that this form of an expression for pillar strength was proposed in 1900 for anthracite after laboratory tests made for the Scranton Engineers Club. This formula is given as

$$\sigma_p = \sigma_1 \left(0.778 + 0.222 \frac{w}{h}\right) \qquad (10.5.21)$$

where σ_p is pillar strength, σ_1 is uniaxial compressive strength of a cubical specimen ($w/h = 1$), and w and h are pillar dimensions.

According to Obert and Duvall, this equation is valid for w/h ratios of 0.25 to 4.0, assuming gravity-loading conditions. Through back calculations from mining case histories and utilization of laboratory rock properties, safety factors of 2 to 4 were derived for short- and long-term pillar stability, respectively. Essentially, this safety factor accounts for strength scaling from laboratory (or rock-material) strength to in situ (or rock-mass) strength for hard rock.

In 1975, Wang, Skelly, and Wolgamott of the Colorado School of Mines (CSM) conducted in situ tests on a coal pillar located in West Virginia (Wang et al., 1977). The tests consisted of reducing pillar dimensions until failure occurred and then determining the pillar strength. The authors proposed the same formula as above and defined σ_1 as the ultimate strength of a cubical specimen of critical size or greater. The recommended factor of safety is 2.0, although $F = 1.5$ is acceptable if mining conditions are well known.

The CSM research was important for a number of reasons. First, Eq. 10.5.21, was applied to coal strata. Second, the term σ_1 was defined acknowledging the existence of a critical sized phenomenon. Third, the equation was stated as being valid for w/h ratios up to 8.

2. Holland-Gaddy Formula: Holland (1964) extended the work by Gaddy (1956) and proposed the following formula:

$$\sigma_p = \frac{k\sqrt{w}}{h} \qquad (10.5.22)$$

where k is the Gaddy factor from equation 10.5.20, w and h are pillar dimensions in in., and σ_p is pillar strength in psi. Holland specified a safety factor between 1.8 and 2.2 for the design of coal pillars, with a recommended value of 2.0. The width-to-height ratio, for which the Holland formula is valid, ranges from 2 to 8. Although popular in the 1970s, the Holland-Gaddy formula is no longer recommended because it was found to be overly conservative at higher $\frac{w}{h}$ ratios (> 5).

3. Holland Formula: In a paper published in 1973, Holland provided a different expression for the strength of coal pillars, namely:

$$\sigma_p = \sigma_1 \sqrt{\frac{w}{h}} \qquad (10.5.23)$$

where σ_1 is the strength of cubical pillars ($w = h = 1$). In effect, σ_1 can be interpreted as the strength at the critical size of coal specimens and is to be determined from Eq. 10.5.18. The recommended factor of safety is 2.0.

4. Salamon-Munro Formula: Salamon and Munro (1967) conducted a survey of failed and standing coal pillars in South Africa. Based on the studies of Holland (1964) and Greenwald et al. (1939), they selected the following form of pillar strength to apply to square pillars:

$$\text{strength} = Kh^{\alpha}w^{\beta} \qquad (10.5.24)$$

The constants for the above equation were derived from a statistical survey of data reflecting actual mining experience. In all, 125 case histories were used, of which 98 were standing pillars and 27 were failed pillars (collapsed at the time of the analysis). In deriving a pillar strength formula, it was assumed that those pillars that were still intact had safe dimensions, while the collapsed pillars were too small. The following pillar strength formula was proposed:

$$\sigma_p = 1320\frac{w^{0.46}}{h^{0.66}} \qquad (10.5.25)$$

where the strength σ_p is in psi, and the pillar dimensions w and h are in feet. The recommended safety factor for this formula is 1.6, the range being 1.31 to 1.88.

In SI units, the above equation becomes:

$$\sigma_p = 7.2\,\frac{w^{0.46}}{h^{0.66}} \qquad (10.5.26)$$

where the strength σ_p is in MPa while w and h are in meters.

This statistical formula is applicable to South African conditions, and it represents the average strength data for coal pillars in that country. Since there are considerable variations in coal strength between the various mines in South Africa (Bieniawski and van Heerden, 1975), the Salamon-Munro formula is currently being modified in South Africa in two respects: (1) by incorporating the actual strength of coal in a mine rather than the average coal strength in the country, and (2) by extending its use for a w/h ratio of 5 and above (Wagner, 1982). The first aspect can be simply achieved by the use of Eq. 10.5.19 when working in English units or using Eq. 10.5.18 and converting σ_1 to SI units for substitution in place of factor 7.2 in Eq. 10.5.26. Thus in English units, the Salamon-Munro formula is of the form,

$$\sigma_p(\text{psi}) = \frac{k}{\sqrt{12}}\,\frac{w^{0.46}}{h^{0.66}} \qquad (10.5.27)$$

5. Bieniawski Formula: This formula is based on large-scale in situ tests on coal pillars. Such tests were first undertaken in the United States by Greenwald et al. (1939) during the period 1933–1941. Extensive tests were conducted in South Africa during 1965–1973 by Bieniawski (1968, 1969), Wagner (1974), and Bieniawski and van Heerden (1975). Wang et al. (1977) conducted in the United States the largest test of all involving one full-sized coal pillar measuring 80 ft (24 m) in width. All these investigations examined the various pillar-strength formulas.

To make the in situ test results generally applicable (i.e., not only to the locality where the actual tests were carried out), the pillar-strength formula can be expressed in a normalized form. For example, the original formula for the Witbank coalfield (Bieniawski, 1967) was of the form,

$$\sigma_p = 400 + 220\,\frac{w}{h} \qquad (10.5.28)$$

where σ_p is in units of psi. This can be represented dimensionlessly as

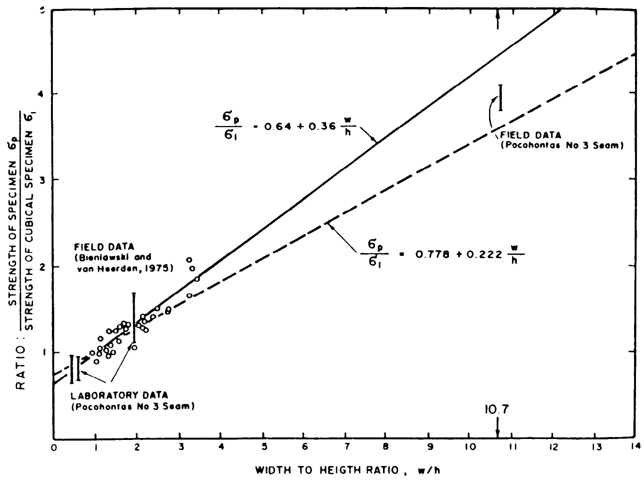


Fig. 10.5.16. The effect of specimen width-to-height ratio on the uniaxial compressive strength of coal pillars (Wang et al., 1977; Bieniawski and van Heerden, 1975).

$$\sigma_p = 620\left(0.64 + 0.36\frac{w}{h}\right) \tag{10.5.29}$$

where σ_1 = 620 psi is the critical-size strength for the Witbank coalfield. Thus the general normalized form of the Bieniawski equation is

$$\sigma_p = \sigma_1\left(0.64 + 0.36\frac{w}{h}\right) \tag{10.5.30}$$

where σ_p is pillar strength, w is pillar width, h is pillar height, and σ_1 is the strength of a cubical specimen of critical size or greater (e.g., about 3 ft or 1 m for coal).

Bieniawski (1969) and Bieniawski and van Heerden (1975) confirmed this relationship by large-scale in situ tests on 66 coal specimens of width-to-height ratios from 0.5 to 3.4.

The formula is particularly realistic for *w/h* ratios up to 10, after which it provides conservative estimates (Fig. 10.5.16). However, for high *w/h* ratios, it is the least conservative formula by comparison with the other four formulas. As this formula is applicable to any mine pillar with a value of σ_1 characterizing the in situ strength of the rock strata, Holland (1973) suggested that a safety factor of two would be generally adequate for US coal mining applications.

To clarify this point, a national survey of coal pillar and roof span dimensions and design procedures in the United States was reported by Bieniawski (1983) that features 171 cases of standing pillars, 23 cases of failed pillars, and 58 cases of roof failures (see Fig. 10.5.17). It was shown that factors of safety ranging from 1.5 to 2.0 would be applicable to coal mining in the United States using the Bieniawski pillar strength formula (Eq. 10.5.30). The value of F = 1.5 is recommended for short term applications (e.g., in the panels) while F = 2.0 should be used in the mains and when pillar recovery on retreat is contemplated. Nevertheless, these recommendations should be regarded as a guide only, and local mining experience should be taken into consideration.

The application for pillar strength formulas to room and pillar mining is covered in 10.5.6.4 and Chapter 18.1.

10.5.6.2. Pillar Load Determination

A number of approaches are available for estimating the *pillar load* or, more correctly, the average pillar stress. The

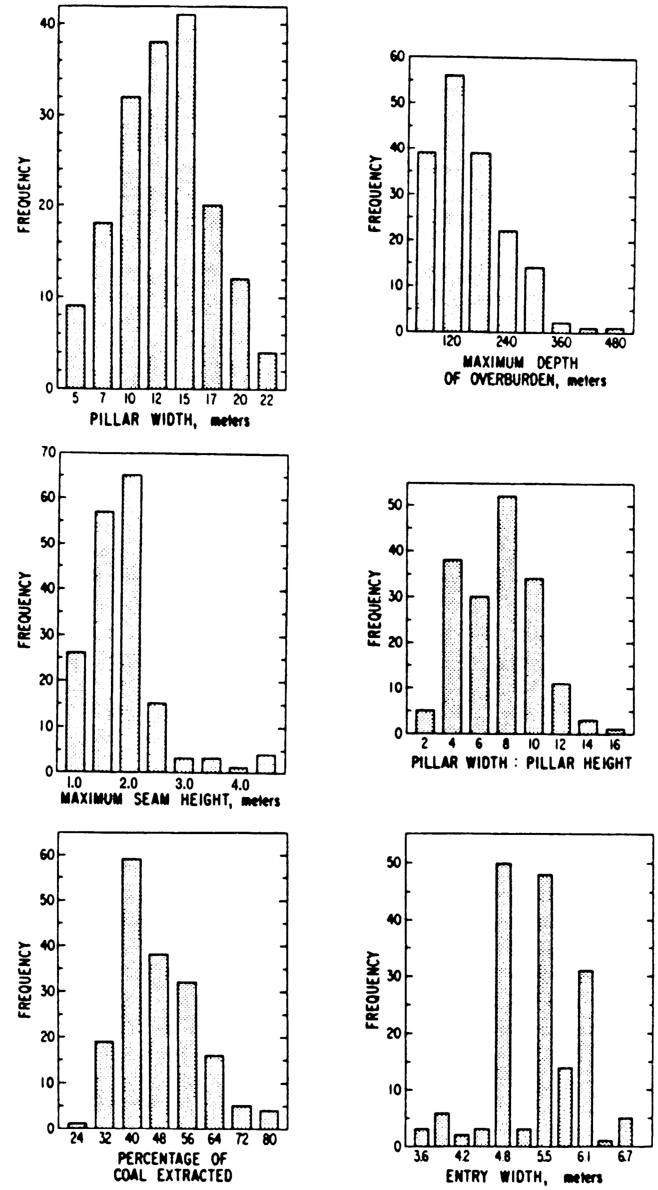


Fig. 10.5.17. Results of survey of room-and-pillar parameters in US coal mining (Bieniawski, 1984). Conversion factor: 1 ft = 0.3048 m.

two major ones are the tributary area approach and the elastic deflection theory.

The simplest approach to determine the pillar load is by the tributary area theory. If a number of well-known simplifying assumptions are made, the pillar load can be calculated from:

$$S_p = \frac{1.1\,H\,(w + B)\,(L + B)}{w \times L} \tag{10.5.31}$$

where S_p is pillar load or the average pillar stress in psi, H is depth below surface in ft, w is pillar width in ft, L is pillar length in ft, and B entry width in ft. The term 1.1 H can be replaced by the virgin vertical pressure S_v derived from the overburden weight above the seam gH, where g is the unit weight of the overburden. The pressure can be considered to increase at a rate of 1.1 psi/ft of depth.

For square pillars, that is, when $w = L$, Eq. 10.5.31 becomes

$$S_p = 1.1H \frac{[w + B]^2}{w} \tag{10.5.32}$$

If the term *extraction e* is introduced (100*e* is percentage extraction), which is defined as the ratio of the mined-out area to total area, then for rectangular pillars the extraction

$$e = 1 - \left[\frac{w}{w + B}\right]\left[\frac{L}{L + B}\right] \tag{10.5.33}$$

Thus Eq. 10.5.31 may also be rewritten as:

$$S_p = \frac{1.1H}{1 - e} \tag{10.5.34}$$

This approach incorporates the following assumptions:

1. The seam is subjected only to vertical pressure, which is constant over the mined area. However, stress transfer occurs where stiff abutments exist in underground workings. Thus this vertical pressure may be relieved partially.

2. Each pillar supports the column of rock over an area that is the sum of the cross-sectional area of the pillar plus a portion of the room area, the latter being equally shared by all neighboring pillars. However, this is certainly not valid if the area of development is small since the pillars in the center of the excavation are under more stress than the pillars near the sides. It is usually only accepted as valid if the mined-out area is greater than the depth below surface.

3. It is assumed that the load is uniformly distributed over the cross-sectional area of the pillar. However, research has shown that:

a) The stress is not evenly distributed over the cross section of an individual pillar, the maximum stress occurring at the corners formed by the intersection of three orthogonal planes, namely, two sidewalls of the pillar and the roof or the floor.

b) The stress on pillars increases with percentage extraction.

c) The stress distribution in pillars depends upon the ratio of pillar width to pillar height.

Clearly, the assumptions made in the formulation of this approach lead to a conservative estimate of the pillar load. Therefore, it represents the upper limit of the average pillar stress. In fact, measurements have shown (Hustrulid and Swanson, 1981) that this approach overestimates the pillar load by about 40%. The simplicity and conservatism of this approach results in its present popularity.

10.5.6.3 Comparisons of Pillar Strength Formulas

From all the available pillar strength formulas, five empirical expressions are most commonly used.

In Fig. 10.5.18, the five selected formulas are plotted (using the Pittsburgh coal seam properties) as the strength ratio vs. the width-to-height ratio. It is apparent from these figures that for higher width-to-height ratios, the Holland-Gaddy formula predicts the lowest strength while the Bieniawski formula predicts the highest strength. At the same time, the form of the Holland formula is such that it will become very conservative at large width-to-height ratios. The higher strength values predicted by the Bieniawski formula are consistent with the fact that for high width-to-height ratios, there is a very rapid strength increase. In fact, pillars are thought to be almost indestructible for width-to-height ratios greater than 10 (Cook and Hood, 1978).

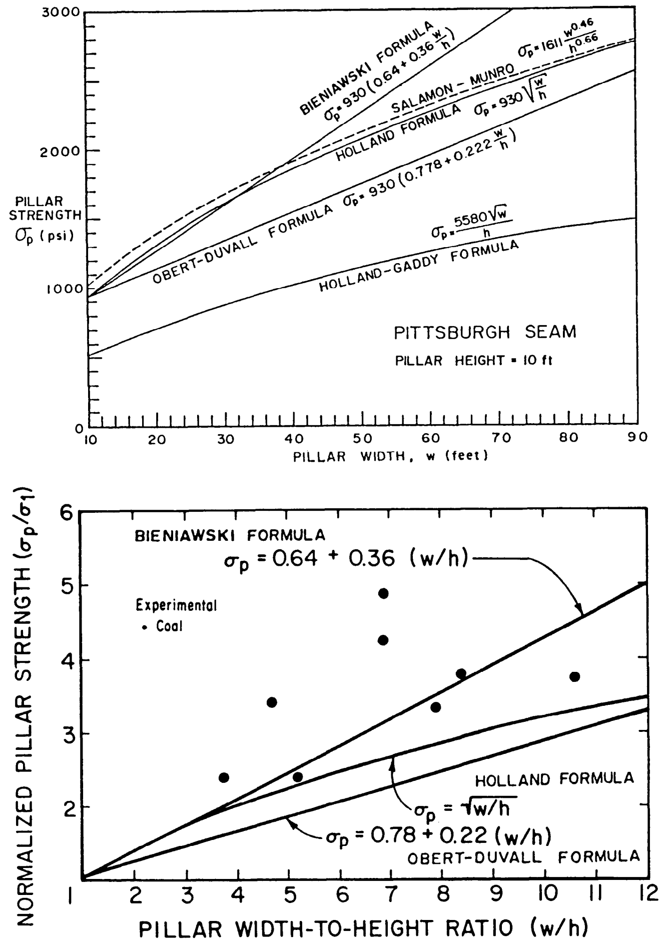


Fig. 10.5.18. Comparison of pillar strength formulas with respect to width-to-height ratio (Bieniawski, 1987). Conversion factor: 1 psi = 6.895 kPa.

A detailed study of this aspect (Bauer, 1980) revealed that the theoretical strength of coal pillars is considerably higher even than that predicted by the Bieniawski formula. Accordingly, it was proposed (Belesky, 1981) that an exponent could be added to the Bieniawski formula thus incorporating a higher rate of strength increase with increasing width-to-height ratios. Currently, the Salamon-Munro formula is being modified along this line (Madden, 1988).

In addition, it is also obvious from Fig. 10.5.18 that the Holland formula and Salamon and Munro formula are quite close in their predictions. This is not surprising since Salamon-Munro used the format of the Holland formula to derive their expression for the strength of coal pillars in South Africa. It is also evident that the Holland-Gaddy formula is very conservative by comparison with the original Holland formula as well as with the other three formulas.

An important point of consideration when comparing pillar strength formulas is the value of the recommended factor of safety, which varies for different formulas. This is demonstrated in Ex. 10.5.3.

10.5.6.4 Design Procedure

The following step-by-step pillar design procedure is recommended (Bieniawski, 1984) when planning new room and pillar coal mines (equations identified):

Step 1. From coal specimen testing $1\frac{1}{8}$-in. (54-mm) core or cubes, tabulate the uniaxial compressive strength of coal σ_c and specimen size (edge dimension or core diameter) D.
Step 2. Based on the uniaxial compressive strength of coal σ_c, determine the value of k for the pillar locality:

$$k = \sigma_c \sqrt{D} \tag{10.5.35}$$

where D is specimen diameter or cube size.
Step 3. Select a pillar strength formula to estimate the pillar width w for a known seam height h. For example, for best economy and safety use:

$$\sigma_p = \sigma_1 (0.64 + 0.36\, w/h) \tag{10.5.36}$$

where

$$\sigma_1 = \frac{k}{\sqrt{36}} \tag{10.5.37}$$

Step 4. Select the roof span B.
Step 5. Determine the pillar load (the average stress on the pillar). In the case of room and pillar coal mining, use the formula based on the tributary area approach:

$$S_p = 1.1H \left[\frac{w + B}{w}\right] \left[\frac{L + B}{w}\right] \tag{10.5.38}$$

where S_p is pillar load in psi, H is depth below the surface in ft, B is entry span in ft, w is width of pillars in ft, and L is length of pillars in ft. For longwall mining application, the pillar load is determined as discussed in 10.5.8.
Step 6. Select a factor of safety F (usually ranging between 1.5 and 2.0) and equate $\sigma_p/F = S_p$. Solve for pillar width w.
Step 7. For economic considerations, check whether the percentage extraction (100e) is acceptable for profitable mining:

$$e = 1 - \left[\frac{w}{w + B}\right] \left[\frac{L}{L + B}\right] \tag{10.5.39}$$

Step 8. If the percentage extraction is not acceptable and needs to be increased by decreasing the pillar width w, select from step 7 the pillar width and length that would give the required coal extraction and determine whether this is acceptable for mine stability. This requires calculation of the factor of safety,

$$F = \sigma_p/S_p \tag{10.5.40}$$

where σ_p is the pillar strength from step 3 while S_p is the pillar load from step 5. The safety factors should be F = 1.5 for short term pillars (in the panels) and F = 2.0 for long-term pillars (in the mains) and for panel pillars to be retreated.
Step 9. Exercise engineering judgment, by considering a range of mining and geologic parameters, to assess the various options for mine planning. Consider the effect of the floor condition.

The above design procedure is not only applicable to room and pillar coal mining but also facilitates the determination of the strength of rock pillars in other rock engineering applications, including longwall mining. In such cases, to establish pillar sizes, special consideration needs to be given to the pillar load since the tributary area formula will not be applicable to tailgate design in view of the abutment pressures. Accordingly, design procedures for chain pillars and yielding pillars are discussed under longwall mining in 10.5.8.

Example 10.5.3. Check an existing coal mining operation in the Pittsburgh seam and improve its performance. Given:

Depth	H =	500 ft (152 m)
Entry width	B =	18 ft (5.5 m)
Pillar width	w =	60 ft (18.3 m)
Pillar length	L =	80 ft (24.4 m) (L/w = 1.33)
Seam height	h =	7 ft (2.1 m)

Pittsburgh seam characteristics: k = 5580 based on σ_c = 3822 psi from NX cores.

Solution. First find σ_ℓ = 5580/6 = 930 psi
Then determine the current factor of safety:

$$F = \sigma_p/S_p = \frac{930\,(0.64 + 0.36 \times 60/7)}{1.1 \times 500 \times 78/60 \times 98/80} = 3.95$$

Clearly, this is not an efficient operation with $F \cong 4.0$ (w/h = 8.57), as evident from the low-percentage extraction:

$$e = 1 - 60/78 \times 80/98 = 0.372 = 37.2\%$$

To improve coal extraction, determine the minimum pillar sizes recommended by the five pillar strength formulas, retaining the ratio of pillar length to pillar width of 1.33.

The calculations provide these results for the appropriate factor of safety recommended by each formula:

Formula	Pillar width (ft)	Pillar length (ft)	Extraction e	Appropriate factor of safety, F
Obert-Duvall	44.2	58.8	0.46	2.0
Holland-Gaddy	52.8	70.2	0.41	1.8
Holland	38.6	51.3	0.50	2.0
Salamon-Munro	28.9	38.5	0.57	1.6
Bieniawski	28.8	38.3	0.58	1.5

Based on the above tabulation, pillar dimensions of 29 by 39 ft (8.8 by 11.9 m) are selected, resulting in an acceptable coal extraction of 59% with a factor of safety of 1.52.

Example 10.5.4. A coal mine in the Pittsburgh seam is mining at 500 ft (152 m) with stable pillars and roofs, and mining is planned at 1000 ft (305 m). What square pillar sizes should be selected at that depth?

Given: H = 500 ft (152 m)
h = 10 ft (3 m)
w = 40 ft (12.2 m)
B = 18 ft (5.5 m)

Solution. (a) Determine the factor of safety found satisfactory at the mine:

$$S_p = 1156 \text{ psi}$$
$$\sigma_p = 930\,(0.64 + 0.36\, w/h) = 1934 \text{ psi}$$
$$F = 1.67$$

(b) Check on the current percentage extraction:

$$e = 0.52 = 52\%$$

(c) At the planned depth of 1000 ft and the same roof span B = 18 ft, the new square pillar width is calculated as:

Equation:	Obert and Duvall	
F	Extraction	Width
2.0	0.26	109.6 ft (33.4 m)

Equation:	Holland	
F	Extraction	Width
2.0	0.27	105.3 ft (32.1 m)

Equation:	Holland and Gaddy	
F	Extraction	Width
1.8	0.15	210.2 ft (64.1 m)

Equation:	Salamon and Munro	
F	Extraction	Width
1.6	0.33	80.0 ft (24.4 m)

Equation:	Bieniawski	
F	Extraction	Width
1.5	0.39	63.5 ft (19.4 m)

10.5.7 ANALYSIS OF FLOOR STABILITY

The existence of floor stability problems in mines and tunnels has been recognized for many years, yet little effort has been made to develop a rational floor stability analysis. This neglect has led to numerous difficulties affecting miner safety, working area and equipment access, transportation, and ventilation. Knowledge of the bearing capacity of mine floors is also particularly important when designing face supports in longwall coal mining.

The bearing capacity approach as practiced in soil mechanics is applicable since room and pillar mining reduces the surcharge due to overburden pressure and leaves a pillar acting in a fashion similar to a vertically loaded shallow foundation. The term *bearing capacity* is defined as the maximum load that may be applied by a foundation to the surrounding soil without causing failure.

Brady and Brown (1985) recommend the following expressions for a cohesive, frictional material such as soft rock:

$$q_u = \frac{1}{2}\gamma\, BN_\gamma + cN_c \tag{10.5.41}$$

$$N_c = (N_q - 1)\cot\phi \tag{10.5.42}$$

$$N_\gamma = 1.5\,(N_q + 1)\tan\phi \tag{10.5.43}$$

$$N_q = e^{\pi\tan\phi}\tan^2\left(\frac{\pi}{4} + \frac{\phi}{2}\right) \tag{10.5.44}$$

where g is density, B is width of footing or pillar, L is length of footing or pillar, c is cohesion of floor strata, and ϕ is friction angle of the floor strata.

Eq. 10.5.41 is applicable to determining the bearing capacity developed under a long rib pillar. For pillars of length L, the bearing capacity is given by

$$q_u = \frac{1}{2}\gamma\, BN_\gamma S_\gamma + c\cot\phi\; N_q S_q - c\cot\phi \tag{10.5.45}$$

where S_y and S_q are shape factors defined as:

$$S_\gamma = 1.0 - 0.4\,(B/L) \tag{10.5.46}$$

$$S_q = 1.0 + \sin\phi\,(B/L) \tag{10.5.47}$$

The factor of safety against bearing capacity failure q_u/S_p should be greater than 2.0 in view of the uncertainty of assuming that the average pillar stress S_p is applied as a uniformly distributed normal load to the floor area (Eq. 10.5.31).

Example 10.5.5. Assess floor stability in a coal mine entry under the following conditions:

Longwall panel at the depth of 640 ft (195 m). The floor of the 18-ft (5.5-m) wide entry is next to a yield pillar with B = 22 ft (6.7 m) and L = 130 ft (39.7 m). Entry height h = 6 ft (1.8 m). The pillar load, as determined from the ALPS procedure, is σ_p = 2660 psi (18.3 MPa). The properties of the fractured grey shale, which is the floor material, are: density g = 70 lb/ ft^3, cohesion c = 30 psi (4320 psf) (0.2 MPa), friction angle ϕ = 19°, uniaxial compressive strength σ_c = 1680 psi (11.6 MPa). The rock mass rating RMR = 44.

Solution.

$$S_\gamma = 1 - 0.4 \times 0.169 = 0.932$$
$$S_q = 1 + 0.2789 \times 0.169 = 0.953$$
$$\gamma = 70\ \text{lb/ft}^3$$
$$N_q = 5.80 \text{ for } \phi = 19°$$
$$N_\gamma = 4.68 \text{ for } \phi = 19°$$
$$q_u = \tfrac{1}{2} \times 70 \times 22 \times 4.68 \times 0.932 + 4320 \times 2.31 \times 5.8 \times 0.953 - 4320 \times 2.91$$
$$= 3358.55 + 69486.0 - 12571.0 = 60{,}273 \text{ psf} = 418 \text{ psi}$$
$$\text{F} = 418/2660 = 0.16 < 2.00 \text{ (flcor failure expected).}$$

Following is a microcomputer program output (SI units) for an alternative determination of the ultimate stress on floor strata of underground coal mines (after Faria and Bieniawski, 1989).

Mine Data and Geomechanical Parameters of the Floor Strata:

Lithological type of the floor: sandy shale
Uniaxial compressive strength (in MPa): 11.6
Point of critical energy release (in MPa): 8.5
Tensile strength (in MPa): 1.5
Unit weight of the floor (in kN/m^3): 12
RMR (rock mass rating): 44
Depth of overburden (in meters): 195
Unit weight of the overburden (in kN/m^3): 25
Pillar width (in meters): 6.7
Pillar length (in meters): 40
Pillar stress (in MPa): 18.3
Entry width (in meters): 5.5
Entry height (in meters): 1.85
Value of m for intact rock: 12.5

Stability analysis—pillar punching into floor

Parameter m: 1.69
Parameter s: 0.002
Geostatic stress (in MPa): 4.88
Cohesion of floor strata (in MPa): 3
Equivalent angle of internal friction—floor: 19°
Ultimate stress on floor strata (in MPa): 15.34
Factor of safety of floor: 0.84. Expect floor failure.

10.5.8. GROUND CONTROL IN LONGWALL MINING

Good ground control is the most important element for successful longwall mining; proper roof control in longwall faces is needed not only to control effectively the immediate roof but also to control and adapt to the behavior of the main roof. This requires a complete understanding of the behavior of the overburden strata. Two of the more frequent problems encountered in US longwall mining are related to panel and entry design and to the development method used. These problems are (1) a panel can be mined out faster than a new one can be developed, and (2) strata control problems occurring in the tail entry, particularly shearing of the roof strata at the rib line, floor heave, and pillar sloughing.

10.5.8.1 Strata Mechanics

It has been observed from measurements of rock pressures around longwall faces that beyond a certain depth—approximately 1000 to 1150 ft (300 to 400 m)—one may achieve control of the roof at the face by using supports with a load-bearing capacity ranging from 220 to 550 tons/ft^2 (20 to 50 t/m^2) of exposed roof. This holds true even at depths down to 4600 ft (1400 m), as is the case in Europe. This figure of support capacity is very small when compared with the weight of the overlying strata; in fact, it is no more than 1 to 2% of that weight. An explanation lies in the rock strata transferring a considerable part of the load ahead of the face, thus creating a zone of high pressure which advances with the face. This creates a bridge over the mining area in which the abutment on one side is the mass of coal in situ and on the other side is the caved waste in the gob. Both "legs" of the bridge are being compressed, and the face supports are therefore carrying only a very small thickness of the rock strata of the order of 33 to 50 ft (10 to 15 m).

To understand this phenomenon better, it may be noted that a redistribution of the stress field in the presence of a longwall face causes high stress concentrations at various points around the longwall face. Three types of such abutment pressures can be categorized, namely, front abutment pressure, side abutment pressure and rear abutment pressure (Peng and Chiang, 1984; see also Chapter 20.1). The front abutment pressure may reach a peak pressure of up to five times the original overburden pressure at about 3 to 10 ft (1 to 3 m) in advance of the coal face. This peak pressure causes a significant strata control problem in both headgate and tailgate entries, even without the interaction with the adjacent previously mined-out panel. However, when interaction does occur, the tailgate entry suffers more than the headgate entry. This is due to the superposition of pressure from the old panel because the side abutment pressure of the previously mined panel meets with the front and side abutment pressures of the presently mined panel.

However, the location of the peak front abutment pressure varies from area to area due to differences in strata characteristics. For US conditions, measurements show that the peak front abutment pressure occurs at about 13 to 30 ft (4 to 9 m) ahead of the face line (Peng and Chiang, 1984).

It should be noted that the generalized positions of the front-abutment pressure profile for British and German coal measure rocks is different from that in the United States. The reason for this is that mining practices in America are different, necessitating a multiple entry tailgate and headgate as well as bleeder entries, which change the stress distributions.

The side abutment pressure, which occurs along both sides of the panel in the gob area, is the largest at the rib sides, reaching its peak some 10 to 30 ft (3 to 10 m) from the longwall side, decreasing exponentially with distance from the ribs to a maximum distance of one-fourth to one-third of the overburden thickness, or typically 200 ft (60 m).

Generally, the magnitudes of the side-abutment pressure for the first row of chain pillars range from 0.4 to 3.5 times the overburden pressure, depending on the location inside the pillar. Moreover, depending on the total pillar width and the zone of influence of side abutment, the stress change in the pillar when the face of the second panel is passing by may reach up to seven times that when the first panel is being mined. The magnitude due to second panel mining ranges from 1.6 to 10 times the overburden stress. As far as the rear abutment pressure is concerned, this is the pressure in the gob area, and its maximum can equal the overburden pressure.

10.5.8.2 Sizing of Chain Pillars

The optimum size of chain pillars is affected by such parameters as the overburden depth, stratigraphic thickness in the roof, working height, entry width, coal strength, and face width. The assessment of the chain pillar stability requires a knowledge of the pillar strength, the pillar stress, and the safety factor.

The strength of coal pillars can be determined using the procedures described in an earlier part of this chapter (10.5.6). Determination of the stress state in chain pillars is particularly crucial in longwall mining because of the presence of the abutment stresses. In the longwall retreating phase, the high abutment stresses can damage entries ahead of the longwall face and the tail entry for the next panel. An optimum arrangement of chain pillars can lead to favorable strata control conditions in the gate entries.

Choi and McCain (1980) reported on the use of the yield pillar concept in a coal mine in West Virginia at a depth of 700 to 800 ft (213 to 244 m). The longwall panels were 450 ft (137 m) wide and were developed using three-entry headings, 16 ft (4.9 m) wide on 100 ft and 50 ft (30.5 m and 15.2 m) centers. The width of the resulting chain pillars (stiff pillars) in the development headings was 80 ft (25.6 m). It was reported that the yield pillars of 34 ft (10.4 m) to 44 ft (13.4 m) width have been successfully used where the strength of the Pittsburgh coal was 2200 to 2500 psi (15.2 to 17.2 MPa). Note that the size of the yield pillars is determined by the strength of the roof in the tailgate, that is, the pillar must yield before the roof of the tailgate breaks. Because the yielding pillars are considered to be independent of supporting the abutment pressure, the size of the chain pillars is obtained from the pillar stress (including the abutment pressures) vs. pillar strength considerations. This is given by Choi and McCain, (1980) as

$$L = 0.6H - 1.2\left[\frac{H^2}{4} - \frac{5}{3}\left(\frac{AW}{W + C} \times \frac{\sigma_p}{24.9\ SF} - AH - \frac{SH}{2}\right)\right]^{0.5} \qquad (10.5.48)$$

where L is width of the panel, H is overburden depth, A is chain pillar width, S is entry width, C is crosscut width, W is chain pillar length, all in ft (m), and SF is safety factor for pillar strength.

Hsiung and Peng (1985) proposed a chain pillar design formula under weak roof conditions by analyzing statistically the results from the three-dimensional, finite-element, parametric analyses. The parameters such as mechanical properties of the roof and floor strata, overburden depth, panel width and length, and coal strength were incorporated. The formula was expressed as follows:

$$\begin{aligned}\log W_p = &-4.676 \times 10^{-3}\, E_i/E_c - 4.04 \times 10^{-3}\, E_m/E_c \\ &- 3.33 \times 10^{-2} \log (E_f/E_c) - 0.0789 \log \sigma_{oc} + 0.5144 \log h \\ &+ 0.0494 \log (L_p/2) + 0.1941 \log P_w\end{aligned} \qquad (10.5.49)$$

where W_p is pillar width in ft, h is overburden depth in ft, L_p is panel length in ft, P_w is panel width in ft, σ_{cc} is uniaxial compressive strength of coal in situ in psi, E_i is Young's modulus of the immediate roof in psi, E_c is modulus of the coal in psi, E_m is modulus of the main roof in psi, and E_f is modulus of the floor in psi.

More recently, Mark and Bieniawski (1986) developed at Penn State a new method for the design of longwall chain pillars, called ALPS (analysis of longwall pillar stability). The method provides estimates of the pillar abutment loads during all stages of their service life. The key issue addressed by ALPS is the magnitude and time-of-arrival of the longwall abutment loads applied to chain pillars as demonstrated in the example which follows. Field studies were conducted, and over 100 case histories were analyzed to verify the method.

10.5.8.3 Analysis of Longwall Pillar Stability (ALPS)

The Penn State ALPS method is a *stiff pillar* design approach. Its goal is to size longwall pillars capable of carrying the abutment loads to which they will be subjected. ALPS consists of three basic elements:

1. Estimation of the load applied to the pillar system.
2. Estimation of the strength of the pillar system.
3. Application of a safety factor.

Before using ALPS, it is necessary to collect mine and coal-strength data used in the design:

1. Maximum depth of cover (H).
2. Unit weight of the overburden (g, typically 162 lb/ft^3).
3. Width of the panel, or face length (P).
4. Entry width (B).
5. Pillar length (L).
6. Height of the coal seam (h).
7. In situ strength of the coal seam (σ_1).

Fig. 10.5.19 defines most of the geometric parameters listed above. The in situ coal strength σ_1 is the same parameter discussed in 10.5.61.

To use ALPS, it is also necessary to have estimates of the individual pillar widths w and the total width of the pillar system W_t. The pillar widths are needed because both the development load and pillar-strength equations are functions of w. If ALPS is being used to size pillars, it is necessary to perform several iterations, adjusting the pillar widths each time as required.

The length of the pillars is normally based on the requirements of ventilation and operations. From a strata control standpoint, the pillars should be as long as possible to increase the available load bearing area. It is particularly important that the pillar lengths equal or exceed the pillar widths. Pillar strength is determined by the least pillar dimension.

The heart of the ALPS method is the estimation of the load applied to longwall pillars. The load estimation procedure begins with an estimate of the development load per ft of gate entry L_t:

$$L_t = (H)\,(W_t)\,(\gamma) \tag{10.5.50}$$

The total pillar load is the sum of the development and the abutment loads. Three parameters are needed in order to estimate the magnitude of the abutment load. These are the magnitude of the side abutment (L_s or L_{ss}), the percentage of the side abutment applied to the chain pillars (R), and the front abutment factors (F_h and F_t). Two equations are available for calculating the side abutment. For critical and supercritical panels, where $P > 0.77\,H$, the magnitude of the side abutment per ft of gate entry L_s can be estimated as

$$L_s = 0.38\,(H^2)\,(\gamma/2) \tag{10.5.51}$$

For panels where $P < 0.77\,H$, the expression for the side abutment (L_{ss}) is

$$L_{ss} = \left(\frac{H\,P}{2} - \frac{P^2}{3.1}\right)\gamma \tag{10.5.52}$$

The percentage of the abutment load applied to the chain pillars is the *abutment fraction R:*

$$R = 1 - \left(\frac{D - W_t}{D}\right)^3 \tag{10.5.53}$$

In Eq. 10.5.53, D is the extent of the side abutment influence zone, which is equal to $9.3\sqrt{H}$.

The front abutment factor is used to estimate the fraction of the total side abutment that is present at the T-junctions. Two front abutment factors are necessary, one (F_h) for the first front abutment experienced in the headgates (and the first tailgate), and the other (F_t) for second front abutment applied to the tailgate. The proposed front abutment factors are F_h = 0.5 and F_t = 0.7, as determined from field measurements.

The maximum loading L_{max} to which the pillar system is subjected depends on the services for which the pillar system will be used. Three possible loading conditions may be defined.

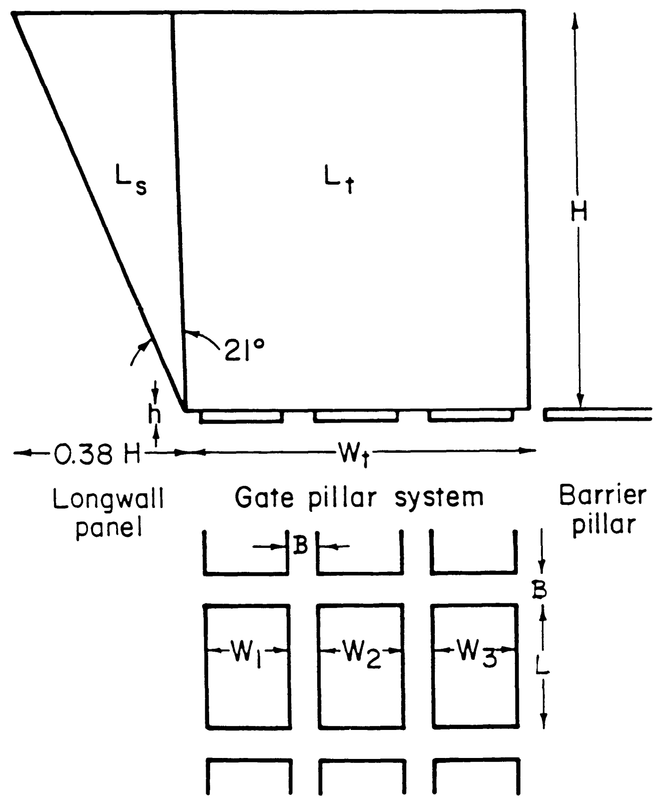


Fig. 10.5.19. Definition of geometric parameters used in the analysis of longwall pillar stability (Mark and Bieniawski, 1986).

The loading experienced by pillars at the T-junctions in the headgate, or in the tailgate during first panel mining, is called headgate loading. *Headgate loading* L_H consists of the development load plus the first front abutment:

$$L_H = L_t + (L_s)(F_h)(R) \tag{10.5.54}$$

Pillars which are expected to protect bleeder entries will be subjected to the development load and the first full side abutment or *bleeder loading* L_B:

$$L_B = L_t + (L_s)(R) \tag{10.5.55}$$

The loads on barrier pillars may also be determined from Eq. 10.5.55 by setting $R = 1$.

The most severe longwall loading is *tailgate loading* (L_T), experienced during the mining of the second and subsequent panels. Tailgate loading consists of the development load, the first side abutment, and the second front abutment:

$$L_T = L_t + (L_s)(1 + F_t) \tag{10.5.56}$$

Once the design pillar loading is established, the next stage is the estimation of the load-bearing capacity of the pillar system. First, the strength of each individual pillar is estimated using the Bieniawski formula:

$$\sigma_p = \sigma_1 (0.64 + 0.36\, w/h) \tag{10.5.57}$$

Then the load-bearing capacity of the pillar system per ft of entry LB is calculated as the sum of the individual pillar resistances:

$$LB = \Sigma[(\sigma_p)(w)(L)][144/(L + B_e)] \tag{10.5.58}$$

Once both the load and the resistance of the pillars have been determined, a safety factor SF may be calculated as:

$$SF = LB/L_{\max} \tag{10.5.59}$$

The final step in the analysis is the comparison of the safety factor determined in Eq. 10.5.59 to a recommended safety factor. If no previous longwall experience is available, $SF = 1.3$ is considered adequate for sizing chain pillars for the gate entries. For barrier pillars that are expected to protect the main entries for a long period of time, a higher safety factor should be used ($SF = 2.0$ to 2.5).

In many cases it is possible to calibrate ALPS with actual field experience. Where several case histories are available from a given mine or mining area, safety factor may be calculated for each pillar design and compared to the observed ground conditions. The SF which corresponds to acceptable conditions may then be used for sizing pillars in future panels.

One major advantage of the ALPS method is that it may be applied to two-, three-, and four-entry systems employing any combination of pillar widths. Although there does not appear to be sufficient justification for specific recommendations regarding the use of unequal size pillars, ALPS does predict that using a single large pillar flanked on one or both sides by small pillars results in a more efficient design. The reason that a design employing a combination of small and large pillars is more efficient than one using equal sized pillars is that the pillar strength increases rapidly as the w/h ratio increases. It appears that many mines could minimize the coal lost in chain pillars without compromising gate entry stability by shifting from equal to unequal sized longwall pillars.

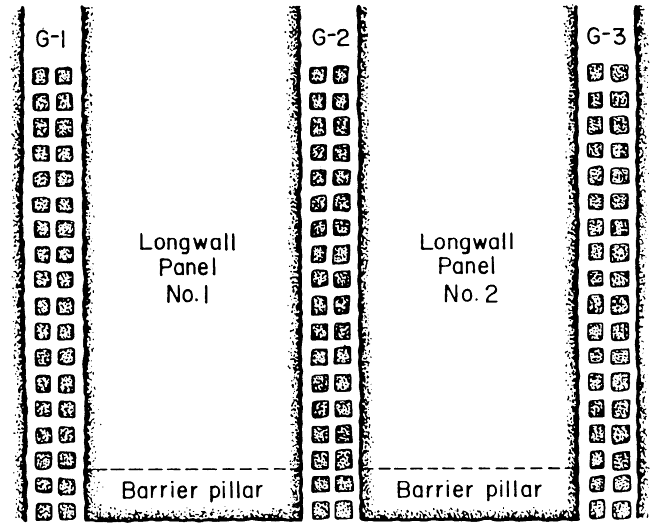


Fig. 10.5.20. Sample problem illustrating the Penn State ALPS method (Mark, 1987).

If small pillars are used in combination with large ones, it is suggested that they be sized to be weaker than either the roof or the floor to reduce stress concentration in the entries. Moreover, as shown in 10.5.7, information on pillar stresses is required for floor stability. The Penn State ALPS method may be used to estimate the average pillar stresses for this purpose.

Example 10.5.6. Chain Pillar Design by the ALPS method. A mine is developing its first two longwall panels in a block of coal reserves. The seam is known to be about 6 ft (1.8m) thick and is considered to be of an average strength for the Pittsburgh seam. The depth of cover over the proposed panels is 1000 ft (305 m). The face length is fixed by the available longwall equipment at 800 ft (244 m). The entries will be 18 ft (5.5 m) wide, and three-entry gate systems will be used. MSHA approval has been obtained to drive the crosscuts with a center-to-center spacing equal to 110 ft (33.5 m) or the entry spacing, whichever is greater. It is desired to size the chain pillars for the three sets of gate entry systems, numbered G-1, G-2, and G-3 in Fig. 10.5.20. Determine the required pillar sizes.

Solution. Each of the three-gate systems will be subjected to different maximum service loadings. Therefore, in order to minimize the total amount of coal used in the chain pillars, the pillars will be sized for each gate system separately. A step-by-step solution for G-1 is given below.

1. Collect the required data:

H = 1000 ft (305 m)
g = 162 lb/ft^3 (25.1 kg/m^3)
P = 800 ft (244 m)
B = 18 ft (5.5 m)
L = 92 ft (28 m) or the pillar width if w > 92 ft (28 m)
h = 6 ft (1.8 m)
σ_1 = 930 psi (6.4 MPa)

2. Provide initial estimates of the pillar width w and the total width of the pillar system W_t:

w = 72 ft
W_t = 180 ft

3. Estimate the development load using Eq. 10.5.50:

L_t = (1000) (180) (162)/2000
= 14,600 tons/ft

4. Estimate the side abutment load. Because the panel is supercritical ($P > 0.77\ H$), Eq. 10.5.51 is used:

L_s = (0.38) (1000^2) (162/2)/2000
= 15,500 tons/ft

5. Calculate the extent of the side-abutment influence zone D and the abutment fraction R, using Eq. 10.5.53.

$$D = 9.3\sqrt{1000} = 294 \text{ ft}$$

$$R = 1 - \left(\frac{294 - 180}{294}\right)^3 = 0.94$$

6. Estimate the maximum service loading L_{max}. Since gate system G-1 will be used as a bleeder system, the maximum pillar loading is L_B, represented by Eq. 10.5.55:

L_B = 14,600 + (0.94) (15,500)
= 29,170 tons/ft

7. Calculate the strength of the individual pillars in the system using Eq. 10.5.57. As the pillars in G-1 are of equal size, only one calculation is necessary:

σ_p = 930 [0.64 + 0.36 (72/6)]
= 4613 psi

8. Calculate the load bearing capacity of the pillar system using Eq. 10.5.58:

LB = (2) (4613) (72) (92) (144)/(110)(2000)
= 39,970 tons/ft

9. Calculate the safety factor (Eq. 10.5.59):

SF = 39,970/29,170
= 1.37

(Thus the maximum abutment pressure is 3367 psi).

10. Compare the SF to a design criterion. In this example, we assume longwall experience is available, and a criterion of SF = 1.5 is used. Since the SF calculated in step 9 is less than 1.5, another iteration of steps 2 to 10 is necessary using an increased pillar size. In order to obtain SF = 1.5, 82-ft (25-m) wide pillars must be used.

The same procedure must be followed to size the pillars for G-2 and G-3, considering the different loading conditions in each case. The pillars in G-2 must protect the tailgate of the second panel, and should be designed using the maximum load defined by Eq. 10.5.56. If the entries in GS-3 will not be used as bleeders for any length of time, then the headgate loading defined by Eq. 10.5.54 can be used. Otherwise, Eq. 10.5.55 would be more appropriate.

The advantage of using unequal-sized pillars may be illustrated by this example problem. If equal-sized pillars are used in G-2, each pillar must be 100 ft wide (30.5 m) for a total pillar width of 200 ft (61 m). The same SF of 1.5 could also be obtained using a 130 ft (40 m) pillar in combination with a 20 ft (6.1 m) pillar, thereby reducing the total pillar width by 25%.

Computer Program for ALPS: Although the application of the ALPS method is straightforward, it can be time-consuming if a number of designs must be analyzed. A computer program is available from Penn State and the USBM.

Longwall Chain Pillar Sizing by Different Methods: There are three approaches to longwall chain pillar design from the point of view of stress distribution/abutment pressure determination. In contrast, in room and pillar mining, only one method is used: the tributary area approach. In both room and pillar mining and longwall mining, the same pillar *strength* formulas are applicable: Obert-Duvall/Wang, Holland, Holland-Gaddy, and Bieniawski.

The three approaches to longwall chain-pillar design are

1) Analytical (Wilson).

2) Empirical, from field measurements (Mark and Bieniawski).

3) Numerical, from finite element method modeling (Hsiung and Peng).

The Wilson method provides an analytical solution of stress distribution due to abutment loading (see 10.5.8.4), and for pillar strength it simply assumes that the in situ coal strength is one-fifth of the laboratory strength. It also assumes that the horizontal and the vertical in situ stresses are equal.

The Mark-Bieniawski method (ALPS) is an empirical approach providing estimates of the abutment loads based on field measurements, for three phases of longwall mining: development, headgate loading, and tailgate loading. For pillar strength, the Bieniawski formula is used (Eq. 10.5.30).

The Hsiung-Peng method is based on finite element method modeling of abutment loading, but it is not directly comparable to the other two methods because it requires assumptions of the modulus of deformation of coal in pillars, mine roof (immediate and main), and mine floor. The pillar strength, like in the Wilson method, is again assumed to be one-fifth of the laboratory coal strength.

It is obvious that the determination of the pillar strength (as one-fifth of the laboratory strength) in the Wilson method and the Hsiung-Peng method is open to criticism. On the other hand, if one of the five common strength formulas is included, care must be exercised that the factor of safety appropriate to each formula is used because they are not the same, as discussed earlier.

Example 10.5.7. It is required to determine the relationship between the overburden depth and the resulting chain pillar sizes as predicted by the various design methods. The following input data are used from a base study by Peng (1986):

Longwall panel width	460 ft (140 m)
Longwall panel length	4000 ft (1219 m)
Entry width	20 ft (6.1 m)
Entry height	6.5 ft (2 m)
Laboratory strength of coal	4500 psi (31 MPa) = σ_c
Specimen size tested	$2\frac{1}{8}$ in. (54 mm) = D
Gaddy k factor	6560
Triaxial strength factor	3.75
Angle of draw	17°
Abutment angle	21°
E_i/E_c = 1	
E_m/E_c = 10	
E_f/E_c = 1	

Modulus of deformation of coal $E_c = 0.5 \times 10^6$ psi (3448 MPa).

Solution. In situ coal strength for the Wilson and Hsiung-Peng methods: $\sigma_o = \sigma_c/5 = 900$ psi (6.2 MPa).

In situ strength of coal for the Mark-Bieniawski method: $\sigma_1 = k/6 = 1093$ psi (7.5 MPa). Note that for comparison with the other two methods, factor of safety of unity will apply.

In situ strength of coal for the Bieniawski formula (also for Obert-Duvall/Wang and Holland formulas): $\sigma_1 = k/6 = 1093$ psi (7.5 MPa).

Pillar strength factor for the Salamon-Munro formula:

$$K = k/\sqrt{12} = 1893 \text{ psi (not convertible to SI units)}$$

In undertaking a comparison, it may be appropriate not only to compare the three main longwall chain pillar design approaches mentioned above but also to include the results of the Wilson stress distribution calculation combined with the common pillar strength formulas instead of the arbitrary assumption that $\sigma_o = \sigma_c/5$. In this case, each pillar strength formula must include its corresponding safety factor *F*, so that the appropriate values will be as follows:

Obert-Duvall/Wang formula: $F = 2.0$, thus $\sigma_o = \sigma_1/2 =$ 546 psi (3.77 MPa)
Holland-Gaddy formula: $F = 1.8$ thus $k_o = k/1.8 = 3644$ psi (2513 MPa)
Holland formula: $F = 2.0$ thus $\sigma_o = \sigma_1/2 = 546$ psi (3.77 MPa)
Salamon-Munro formula: $F = 1.6$ thus $K_o = K/1.6 = 1183$ psi (8.16 MPa)
Bieniawski formula: $F = 1.5$ thus $\sigma_o = \sigma_1/1.5 = 728$ psi (5.02 MPa)

If, in the above example, $F = 1.0$ is chosen for all five formulas, the results will be comparable with those predicted by the three main chain pillar design approaches.

Fig. 10.5.21 shows a comparison of the three main design approaches. It is concluded that the Mark-Bieniawski method provides the most economical, yet safe, pillar sizes.

Special Considerations: These considerations include chain pillars in multi-entry development, pillars at greater depths (over 1500 ft or 458 m), rectangular pillars, barrier pillars, and pillars exhibiting brittle behavior.

It should be noted that for *multi-entry development* as practiced in US longwall mining, one can use the Wilson method to obtain a "stress decay" profile over the entry, that is, stress decrease from the gob size to the unmined panel. Once this is done, pillar sizes may be selected, including yielding pillars, based on any of the five pillar strength formulas.

For *pillars at greater depths* (over 1500 ft or 458 m), which means higher width-to-height ratios, the formulas of Obert-Duvail/Wang, Holland-Gaddy, Holland, and Salamon-Munro are all too conservative and, in fact, were never intended for this purpose. Only the Bieniawski formula is still realistic up to width-to-height ratios of 10 to 12. Realizing this, Madden (1988) reported on a modified Salamon-Munro formula for squat pillars, which is of the following form:

$$\sigma_p = \sigma_1 \frac{5^{0.5933}}{V^{0.0667}} \left\{ \frac{0.5933}{2.5} \left[\left(\frac{w}{5h} \right)^{2.5} - 1 \right] + 1 \right\} \tag{10.5.60}$$

where σ_1 is in situ strength of coal = 7.2 MPa, *V* is pillar volume, *w* is pillar width, and *h* is pillar height, all in meters.

This formula should be compared with the Bieniawski formula as well as the original Salamon-Munro formula given in Fig. 10.5.18. Note that the factor $K = 7.2$ MPa is applicable to South African conditions and for US applications should be replaced by the value of σ_1 in MPa.

For *rectangular pillars,* three expressions are available. It is no longer accepted that the strength of rectangular pillars equals the strength of square pillars, the width of which is the same as the smallest dimension (width) of the rectangular pillars.

The expressions for the effective pillar width for rectangular pillars are as follows:

(Salamon and Oravecz, 1970)

$$w_{eff} = (w_w \times w_L)^{0.5} \tag{10.5.61}$$

(Wagner, 1974)

$$w_{eff} = 4A/C \tag{10.5.62}$$

(Peng, 1986)

$$w_{eff} = w_w^{0.85} \times w_L^{0.15} \tag{10.5.63}$$

where w_w is pillar width, w_L is pillar length, *A* is pillar area, and *C* is pillar circumference. It is believed that the Wagner formula has been most tested in practice.

For *barrier pillars*, the old "mine inspector's formula" of 1930 is considered too conservative. It is of the form, with units in ft,

$$W_{barrier} = 20 + 4h + 0.1H \tag{10.5.64}$$

A more realistic approach is to select a barrier on the basis of the width-to-height ratio being $w/h > 20$ since pillars with $w/h > 12$ are virtually indestructible, and hence $w/h > 20$ gives a sufficient margin of safety.

Finally, for *pillars exhibiting brittle behavior,* the elastic-plastic analysis of Wilson may not be applicable. Instead, a stress analysis incorporating the Hoek-Brown rock mass strength criterion could be more appropriate (Eq. 10.5.4). It should be noted that the Wilson method was developed for British soft-coal strata and, up to a depth of 2300 ft (700 m) and for $k = 3$, it conforms closely to a British empirical formulation that the rib pillar width should equal one-tenth of the depth below surface plus 15 yd (13.7 m). Note that in Britain, a single rib pillar separates longwalls mined on advance with single-way entries (called drivages or roadways).

Distribution of Vertical Stress Around a Longwall Face: Wilson (1977, 1983) has developed an instructive analysis of the distribution of vertical stress around a longwall extraction. This analysis uses a stress balance method in which the total vertical force applied over a large plan area must remain equal to that caused by the overburden, even after part of the seam has been removed. It is assumed that, compared with the total vertical force to be redistributed, the vertical forces transmitted by the roadway supports are small and may be ignored.

Fig. 10.5.22 shows Wilson's approximations to the vertical stress distributions acting at cross sections. The distribution of vertical stress postulated in the caved waste is based on an assumption, derived from field observations, that the stress reaches the overburden stress at a distance of 0.3h from the rib side (Fig. 10.5.22a). The distribution of stress within the yielded zone at the rib side is calculated using an elastic-plastic analysis. As shown in Fig. 10.5.22 c, the resulting nonlinear stress distribution is approximated by a triangular distribution. Table 10.5.9 sets out the values calculated by Wilson for the vertical stress, the peak abutment stress, the width of the rib-side yield zone, and the total vertical force carried by the yield zone for two sets of strata conditions. In Fig. 10.5.22 and Table 10.5.9, the following symbols are used:

σ_{zz} = vertical stress
σ_y = peak abutment or yield stress
p = vertical stress remote from the excavation = gh
C_o = in situ uniaxial compressive strength of the strata
b = a constant in the principal stress form of the Coulomb shear strength equation, $\sigma_y = C_o + b\sigma_3$

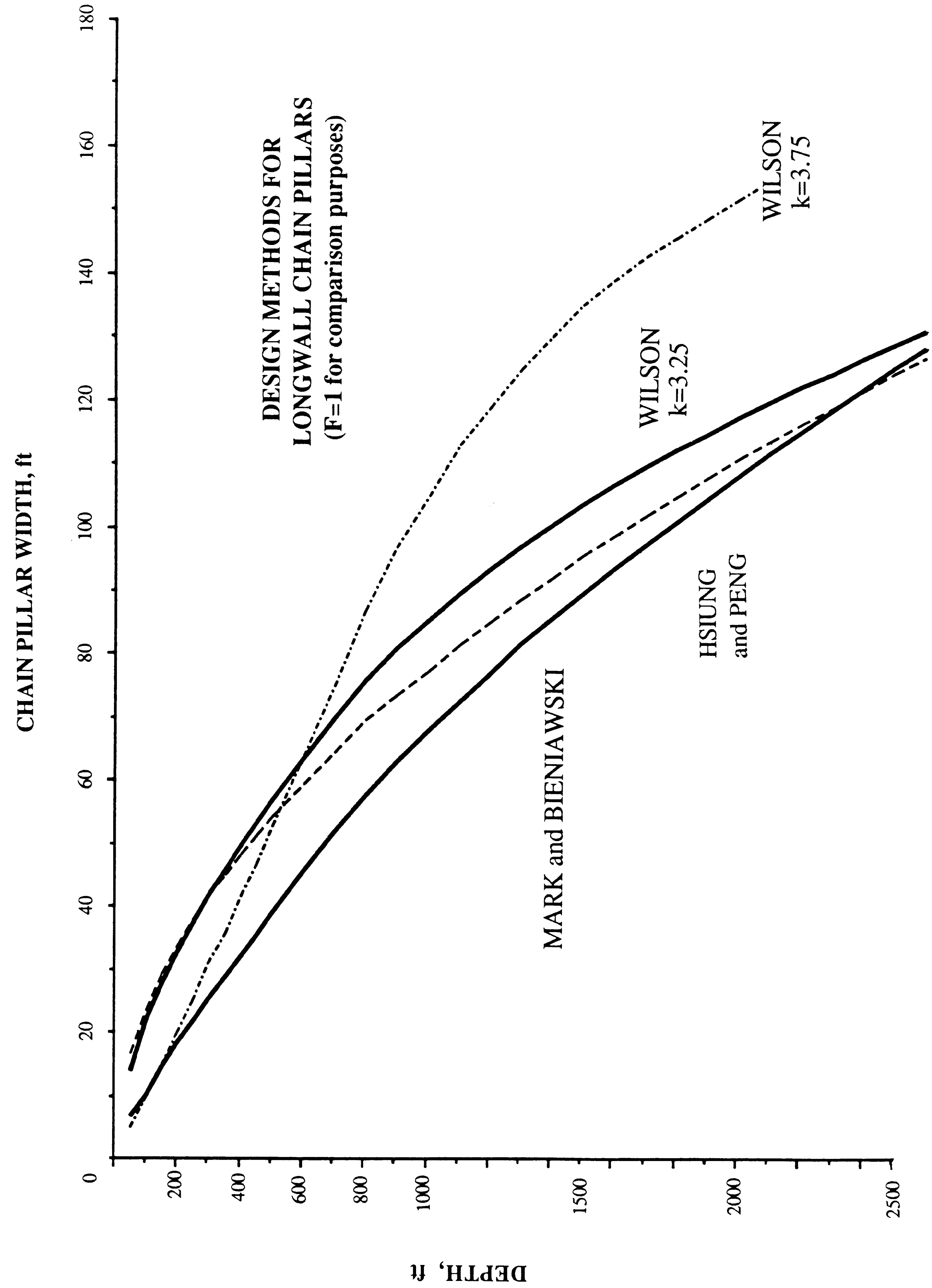


Fig. 10.5.21. Comparison of longwall pillar design methods. Conversion factor: 1 ft = 0.3048 m.

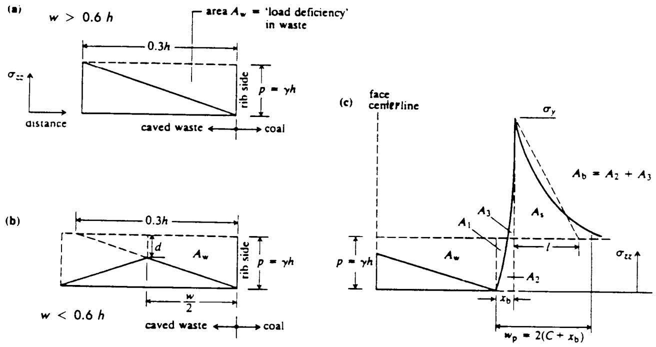


Fig. 10.5.22. Hypothetical decay of stress around a longwall face (Wilson, 1983).

Table 10.5.9. Expressions for Calculating Distributions of Vertical Stress at Rib Sides (Roof, seam and floor of similar strength).

vertical stress	$\sigma_{zz} = bp^*\left(\frac{2x}{m} + 1\right)^{b-1}$
peak abutment or yield stress	$\sigma_y = C_o + bp$
width of yield zone	$x_b = \frac{m}{2}\left[\left(\frac{p}{p^*}\right)^{1/(b-1)} - 1\right]$
vertical force carried by the yield zone	$A_b = \frac{m}{2}p^*\left[\left(\frac{p}{p^*}\right)^{b/(b-1)} - 1\right]$

Source: Wilson, 1983.

m = height of the extraction
x = distance from the rib side
x_b = width of yield zone
p^* = support pressure p_i plus the unconfined compressive strength of the broken material at the rib side, taken as 0.1 MPa, or $p^* = p_i + 0.1$ MPa
$F = \frac{b-1}{\sqrt{b}}\left(1 + \frac{b-1}{\sqrt{b}}\tan^{-1}\sqrt{b}\right)$, where $\tan^{-1}\sqrt{b}$ is expressed in radians

The stress in the zone beyond the peak stress decays asymptotically towards the overburden stress p. The stress decay curve is assumed to be of the form,

$$(\sigma_{zz} - p) = (\sigma_y - p)\exp\left(\frac{x_b - x}{C}\right)$$

where C is a constant having the units of distance. By equating areas under the curves in Fig. 10.5.22c so as to maintain a vertical stress balance, the value of C is calculated as

$$C = \frac{A_w + px_b - A_b}{\sigma_y - p} \tag{10.5.65}$$

where A_w is the *load deficiency* associated with each rib side as shown in Figs. 10.5.22 a and b.

For $w > 0.6h$,

$$A_w = 0.15\gamma h^2 \tag{10.5.66}$$

and for $w < 0.6h$,

$$A_w = \frac{1}{2}w\gamma\left(h - \frac{w}{1.2}\right) \tag{10.5.67}$$

where w is the width of the extracted area.

It must be emphasized that Wilson's method of calculating the distribution of vertical stress in the mining domain following extraction is mathematically not rigorous but is based on a number of assumptions and approximations.

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Chapter 10.6
MINE SUBSIDENCE

MADAN M. SINGH

10.6.1 INTRODUCTION

Subsidence is an inevitable consequence of underground mining— it may be small and localized or extend over large areas, it may be immediate or delayed for many years. During recent years, with the expansion of urbanization and increased concern for the environment, it is no longer possible to ignore its aftermath. In the United States, mining companies have, therefore, begun to devote attention to the subject and study it in a methodical manner. Appropriate regulations have also been promulgated by various government agencies, depending on the needs of the region, in order to protect the public interest.

The problems associated with subsidence have been recognized since the inception of mining and mentioned in the literature as far back as Agricola's *De Re Metallica* in 1556. The initial "vertical theory" was modified by the researches of Coulomb, Toillez, Gonot, Rziha, and Fayol (Peele, 1952). Significant contributions may also be credited to Rucloux, Durmond, Callon, Goupilliere, Schulz, von Sparre, von Dechen, Hausse, and Jicinsky (Peele 1952). In the early part of the century, Briggs (1929) presented his comprehensive treatise on the subject. In the United States, the early investigations of Richardson (1907), Young and Stoek (1916), Rice (1923), Rutledge (1923), Crane (1925, 1929, 1931), and Allen (1934) are notable. The motivation behind these studies was the severe damage caused to structures, communications, and agricultural lands; the victims (mostly property owners) demanded compensation and restitution from the mine operators, and frequently resorted to court action. In order to defend against unjustified claims, measurements of ground movements were made. These data, along with theoretical concepts on the development of these movements, gradually evolved into the subject of *mine subsidence engineering.* Courses specifically devoted to the subject have been taught in German academies since 1931 and in US universities only since 1963. The major objectives of subsidence engineering are

1. Prediction of ground movements.
2. Determining the effects of such movements on structures and renewable resources.
3. Minimizing damage due to subsidence.

Thus it is evident that subsidence engineering not only entails the study of ground movements, structural geology, and geomechanics (both soil and rock mechanics), but also encompasses a knowledge of surveying, mining and property law, mining methods and techniques, construction procedures, communications technology, agricultural science, hydrology and hydrogeology, urban planning, and socioeconomic considerations.

Although the mining of all underground minerals may result in subsidence, most studies to date have concentrated on the extraction of flat bedded deposits—primarily coal. Hence throughout this discussion, reference to coal mining is made frequently. The information presented herein is, therefore, most pertinent to coal mining, although it generally applies to other bedded deposits. The principles may be also extrapolated to other mining methods, but the conclusions need validation by actual experience.

The pumping of geofluids, such as petroleum, natural gas, geothermal brines, and water, constitute "mining" in the strict sense and also cause subsidence. The effects of fluid withdrawal have been investigated at some length, although they are beyond the scope of this chapter. But the lowering of the water table in the region adjoining mining activity also induces ground movements, thereby causing surface damage, which must not be overlooked.

The term *subsidence,* as used in this chapter, implies the total phenomenon of surface effects associated with the mining of minerals and not only the vertical displacement of the surface as is sometimes inferred in the literature.

10.6.2 PRINCIPLES OF SUBSIDENCE

10.6.2.1 Development of Subsidence

Whenever a cavity is created underground, due to the mining of minerals or for any other reason, the stress field in the surrounding strata is disturbed. These stress changes produce deformations and displacements of the strata, the extent of which depends on the magnitude of the stresses and the cavity dimensions (Chapter 10.2). With time, supporting structures deteriorate and the cavity enlarges, resulting in instability. This induces the superjacent strata to move into the void. Gradually, these movements work up to the surface, manifesting themselves as a depression. This is commonly referred to as subsidence. Thus *mine subsidence* may be defined as ground movements that occur due to the collapse of the overlying strata into mine voids. Surface subsidence generally entails both vertical and lateral movements.

Surface subsidence manifests itself in three major ways:

1. Cracks, fissures, or step fractures.
2. Pits or sinkholes.
3. Troughs or sags.

Surface fractures may be in the form of open cracks, stepped slips, or cave-in pits and reflect tension or shear stresses in the ground surface.

When the area of surface collapse into the mine void is relatively small, the subsidence is termed a *pit* or *sinkhole;* generally, these are associated with shallow room and pillar mining. In Britain, the terms "crownholes," "chimneys," or "pipes" are also used to describe this phenomenon. In time, these pits may enlarge and coalesce to form trenches. Frequently, the walls of the pit intersect the surface precipitously, and the pit diameter increases with depth. The depth of pits is generally limited: 100 ft (30 m) in Pennsylvania (Gray et al., 1977), 165 ft (50 m) in Illinois (DuMontelle et al., 1981), or 10 to 15 times seam thickness, based on studies primarily in Colorado, Utah, and Wyoming (Dunrud and Osterwald, 1980).

When the mine void is of larger size due to longwall mining or eventual collapse of pillars, the collapsed strata fall into the excavation and bulk (i.e., broken material occupies a larger volume than in situ rock). This process continues until a height is reached of about three to six times the mined seam thickness (Singh and Kendorski, 1981), unless the material spreads or is transported to other parts of the mine by water. Cyclical wetting and drying of the debris could also induce greater compaction.

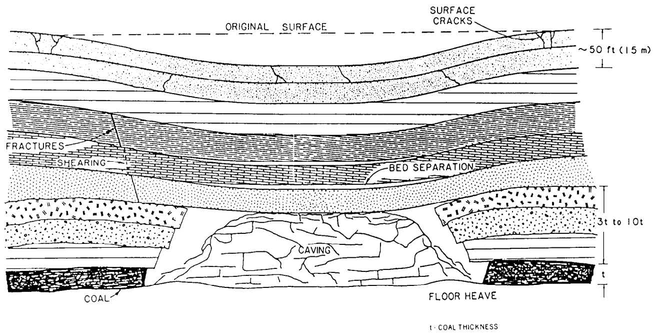


Fig. 10.6.1. Strata disturbance and subsidence caused by mining (Singh and Kendorski, 1981).

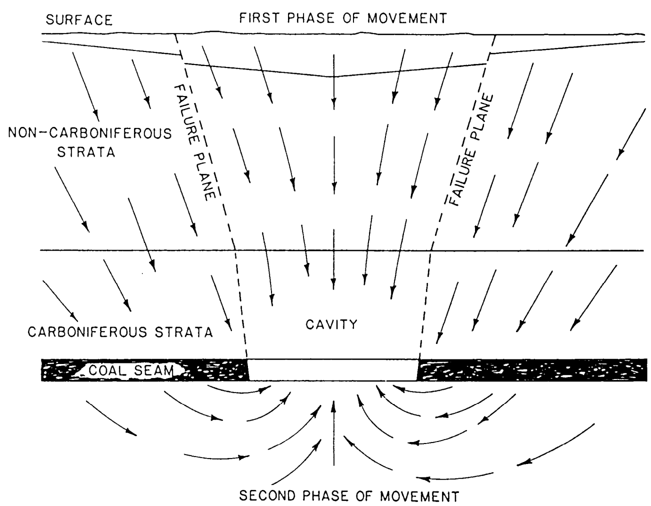


Fig. 10.6.2. Schematic representation of ground movements due to subsidence (Grond, 1957).

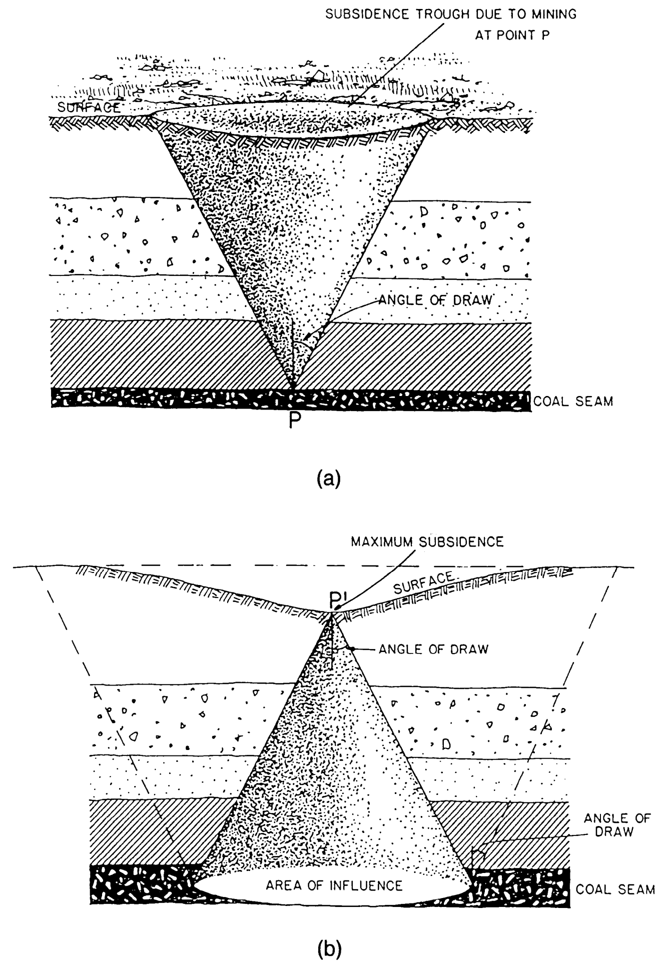


Fig. 10.6.3. Sketch depicting area of influence. (a) Effect on surface by mining at P. (b) Maximum subsidence at P′ by mining entire area of influence.

When the cavity is essentially filled with broken rock, the debris offers some support to the superjacent beds (Fig. 10.6.1). As these strata settle or sag, bed separation may occur because of the tendency for lower strata to subside more than the higher beds. Overall, as the various strata settle or subside, they sag rather than break and produce a dish- or trough-shaped depression on the surface. This type of feature normally covers a larger area than sinkholes, and is referred to as *trough* or *sag subsidence.* Trough subsidence may occur due to mining at any depth. The overall movements of the ground around the opening are depicted in Fig. 10.6.2; the direction of motion is not only vertically downward but also horizontal and, in some locations, upward.

The mining of a single point P (Fig. 10.6.3a) at seam level will affect a circular area on the surface, defined by the base of an inverted cone with P as the apex and the limit angle g as the semi-angle of the cone. If this cone is turned upright, then the mining of any part of its base will influence the subsidence of its apex P (Fig. 10.6.3b). Hence, this circular area is termed the *area of influence.* This implies that the diameter of the area of influence is given by $2D \tan g$, where D is the depth of the seam below the surface and g is the limit angle. (It may be noted that some authors use the complement of the limit angle, often termed the "angle of major influence.") This diameter also defines the *critical width* of the workings, which is the minimum width that needs to be mined before the maximum possible subsidence is observed at the center of the trough. If the mined width is less than critical, it is termed *subcritical,* and the amount of subsidence that occurs will be less than the maximum. If a *supercritical* (i.e., larger than critical) width is excavated, the central portion of the trough will attain maximum subsidence, and a flat-bottomed depression will be produced (see Fig. 10.6.4).

If the vein being extracted is relatively flat and nearly horizontal, as is generally the case with coal and potash, the overburden and surface collapses or subsides forming a *depression* or *trough.* The surface area affected by mining is generally larger than the vein area excavated. Hence the angle of inclination between the vertical at the edge of the workings and the point of zero vertical displacement at the edge of the trough is termed the *limit angle* or *angle of draw.* It is evident that the limit of surface subsidence depends upon the precision with which the subsidence is measured. By convention, this is taken to be the contour of points that have subsided vertically by 0.01 ft (3 mm). It is also a function of the vein dip and the geology of the area.

10.6.2.2 Movement of the Subsidence Curve

As subsidence occurs, there is a movement of surface points towards the center of the mined area. The amount of vertical

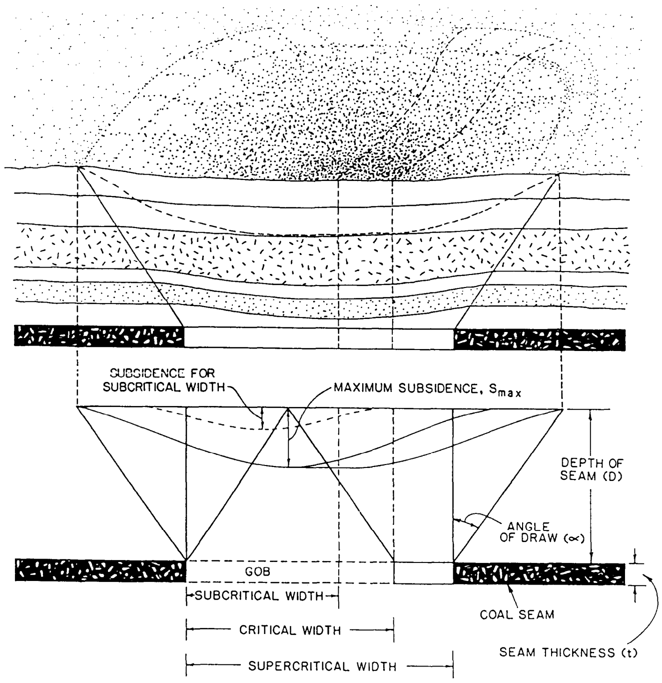


Fig. 10.6.4. Influence of extraction width on subsidence.

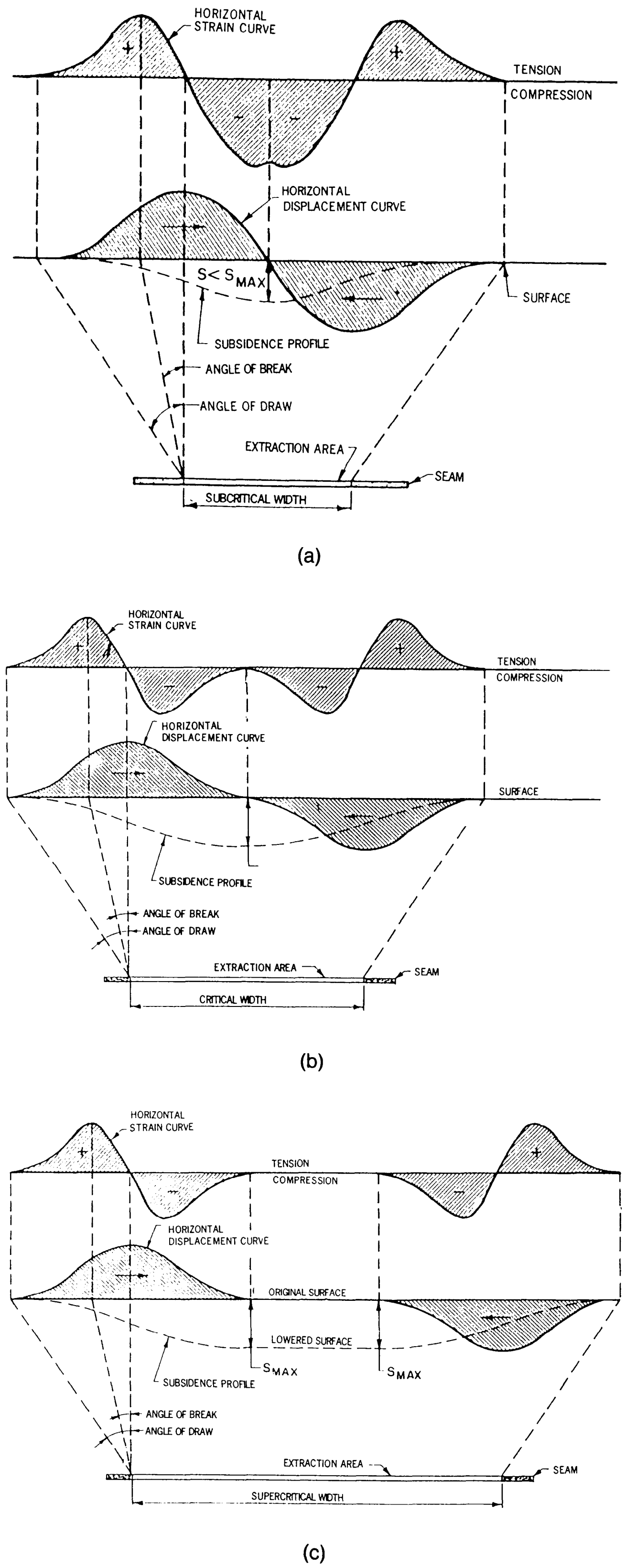


Fig. 10.6.5. Schematics of displacement and strain curves for various working widths. (a) Subcritical width. (b) Critical width. (c) Supercritical width.

displacement experienced is greatest at the center while the horizontal displacements are least at the center and edges of the trough and maximum at, or close to, the edges of the mined area. Since these lateral movements are not uniform, there are changes in the lengths per unit length (i.e., strains), along any cross section of the mined panel. These strains tend to stretch the surface near the edges of the trough (i.e., these are tensile), and push inward within the boundaries of the extracted area (compressive strains; see Fig. 10.6.5). Both the tensile and compressive horizontal strains disappear at the center of the subsidence trough in the case of critical- and supercritical-width workings. Fig. 10.6.6a shows the progressive development of the mine working and the subcritical, critical, and supercritical widths being formed. As the mine workings progress, the horizontal tensile and compressive strain regions also move along (Fig. 10.6.6b). Hence these are also referred to as traveling strains.

The inclination to the vertical of the line connecting the edge of the mined area with the surface point exhibiting the maximum tensile strain is called the *angle of break* or *angle of fracture* (Fig. 10.6.7). These terms should not be confused with the draw or limit angle defined earlier. Generally, the angle of break is somewhat higher with subcritical-width workings than with critical- or supercritical-width workings for a given region.

10.6.2.3 Components of Subsidence

Subsidence consists of five major components, which influence damage to surface structures and renewable resources (see Fig. 10.6.7):

1. *Vertical displacement* (settlement, sinking, or lowering).
2. *Horizontal displacement* (lateral movement).
3. *Slope* (or tilt), i.e., the derivative of the vertical displacement with respect to the horizontal.
4. *Horizontal strain,* i.e., the derivative of the horizontal displacement, with respect to the horizontal.

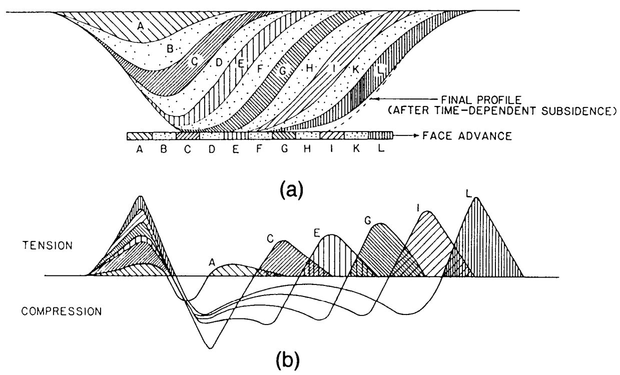


Fig. 10.6.6. Development of subsidence trough and strains with face advance (Rellensman and Wagner, 1957). (a) Subsidence development. (b) Traveling strain curve.

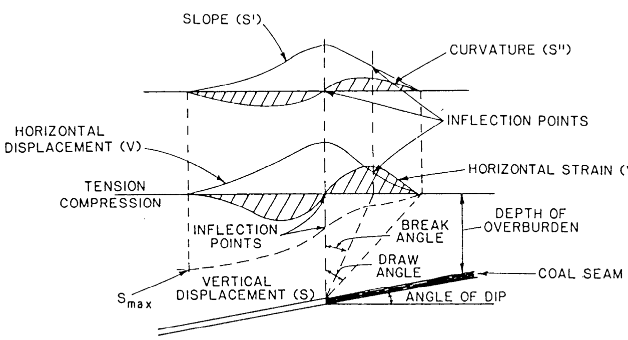


Fig. 10.6.7. Schematic of ground movements caused by subsidence (Singh, 1978).

5. *Vertical curvature* (or flexure), which may be approximated by the derivative of the slope, or the second derivative of the vertical displacement with respect to the horizontal.

Vertical displacements alone cause little structural damage. Examples of an observation tower having sunk 30 ft (9.2 m) in a coalfield, mining structures subsiding a similar amount around the sulfur mining areas off the coast of Louisiana, and a church in a potash mining district having settled 20 ft (6.2 m), all without significant damage, have been quoted in the literature. However, lowering of the land may cause some parts to be inundated, so that drainage patterns in agricultural areas may require redesign, flow through pipes may be disrupted or reversed, the groundwater circulation may be dislocated, and the grade of roads or railways may be altered.

Uniform horizontal movements of the ground surface also cause little damage to structures. But breaks in pipes, electric or communications lines, roads, and other features may occur.

Differential vertical settlements cause slopes to form and induce tilting. Maximum slopes in a subsidence trough generally range between 2×10^{-3} to 20×10^{-3}, but may reach 150×10^{-3} for multiple-seam extraction (Brauner, 1973). The formation of slopes may cause structures to tilt, the gradients of railroad tracks and highways to change, and tanks to overflow, and interfere with gravity drainage.

Surface horizontal strains cause most of the damage to structures located above mined areas. They cause tensile or shear cracks and buckling. Masonry structures withstand compression much better than tension. Strains may induce distortion, fractures, or failure. The weaker parts of the structure (e.g., openings) and the lower part of the frame are the first to show tension cracks. Pipes, cables, roads, railways, walls, and other building components buckle readily under compressive strains. Generally, the strains due to mining range between 1×10^{-6} and 10×10^{-6} (Brauner, 1973). The extent of strain transfer from the ground to foundations is not well established.

Curvature causes two types of distress on structures: (a) *shear strain,* which induces angular changes and tends to distort buildings out of square (distortion is generally proportional to structure height): and (b) *flexure* or *bending,* which causes strains in long load-bearing members. Concave curvature causes tension along the bottom and compression along the top of the building. The dimension of curvature is the inverse of length. Hence generally, the reciprocal of the curvature (i.e., radius of curvature) is quoted. For mine subsidence, this normally varies between 3200 ft (1000 m) and 6600 ft (2000 m), and seldom falls below 1600 ft (500 m) (Brauner, 1973).

10.6.2.4 Factors Affecting Mine Subsidence

Several geologic and mining parameters and the nature of the structure affect the magnitude and extent of subsidence that occur due to coal mining (Henry, 1956; King and Whetton, 1957; Sinclair, 1963; Cortis, 1969; Brauner, 1973; Chen et al., 1974; Anon., 1975).

Effective Seam Thickness: The thicker the seam extracted, the larger the amount of surface subsidence that is possible. In some cases, the entire seam may not be mined or some pillars or other nonminable coal may be left in place. Hence the effective seam thickness should be considered. In thick beds, the slenderness (height-to-width ratio) of the pillars is higher for a given extraction ratio. Slender pillars are normally more prone to failure.

Multiple Seams: Where multiple worked-out mining horizons exist, collapse could be initiated from any one of several levels, thereby increasing the likelihood of subsidence events, because the adjacent strata are disturbed. This is especially true when the prior mining was in an overlying seam.

Seam Depth: A school of thought exists that at greater depths, an arch is formed over the mine cavity, preventing surface subsidence. In recent years, this has been gradually refuted. Perhaps the time period that elapses before subsidence effects are observed at the surface is prolonged, but the total amount of subsidence does not appear to be changed; that is, subsidence is independent of depth (Orchard, 1964). Pit depths generally do not exceed certain limits, as discussed previously (see 10.6.2.1).

Dip of Seam: When the coal seam being mined is inclined, an asymmetric subsidence trough is formed that is skewed toward the rise; that is, the limit angle is greater on the dip side of the workings. The strains are also smaller toward the dip direction. Pillars in steeply dipping seams tend to be less stable.

Competence of Mine Roof and Floor: Since subsidence propagates from the mine level, the characteristics of the mine roof and floor are vital in the initiation of subsidence movements. Soft fireclay floors, especially if susceptible to further weakening due to moisture, induce pillar punching or heave. Weak roofs, composed of shales, siltstones, and limestones, permit falls that are accentuated if punching also occurs. Competent roof beds tend to support the overlying strata longer and hence delay the subsidence. Also, when these fracture, they occupy a greater bulk volume than weaker strata (which compact more). When both the roof and floor are strong, the pillars tend to spall and crush.

Nature of Overburden: Strong massive beds above the mine level tend to prop the overburden for a prolonged period and defer the occurrence of subsidence.

Near-surface Geology: The soils and unconsolidated rocks near the surface tend to accentuate subsidence effects. The geologic materials are less homogeneous and isotropic than the underlying strata, and often behave in an inconsistent manner. Cracks and fissures may initially form in a 50-ft (15-m) thick layer from the surface (Singh and Kendorski, 1981). Later, these may be filled by plastic deformation or material transportation by water. Occasionally, however, water flow may accentuate these fissures and form gullies. Structures and renewable resource lands are thereby adversely affected.

The composition of the rock/soil cover is important; if the material is of a fine, sandy nature containing large amounts of water, it may flow to a rock fracture and drop into the underground workings. Besides, water accumulating in the abandoned mine may seep upwards into the unconsolidated strata above through natural fissures and cracks in the rock and increase the potential for soil collapse.

Geologic Discontinuities: The existence of faults, folds, and the like may increase subsidence potential. Mining disturbs the equilibrium of forces in the strata and may trigger movement along a fault plane, due to ease of slippage, causing either settlement or upthrust at the surface, which may appear as a series of step fractures. The effects of the other parameters may need to be discounted in areas of adverse geological conditions.

Lateral movements concentrate near the fault, but the strains may become immeasurable on either side. Structures that straddle fault planes tend to be severely damaged, but nearby buildings remain relatively intact. Joints and fissures in the strata affect subsidence behavior in a manner similar to faults but on a smaller scale.

Fractures and Lineaments: Natural fractures and lineaments affect surface subsidence, but a strong correlation has not been established to date.

In Situ Stresses: High horizontal stresses tend to restrain surface subsidence by forming a ground arch in the immediate mine roof (Lee and Abel, 1983). The arch height and stability are sensitive to the ratio of vertical to horizontal stresses. Highly stressed arches may fail violently (e.g., the Urad molybdenum mine, Colorado). Roof instability and floor heave, resulting from high horizontal stresses and their orientation, need to be taken into account when laying out coal mines in the Allegheny Plateau (Aggson, 1978).

Degree of Extraction: Lower extraction ratios tend to delay subsidence. It is less prevalent in areas superjacent to first mining, since sufficient pillar support is generally available without crushing of pillars. In second mining, the cross section of the pillars is reduced by splitting and slicing. Localized stress buildups promote crushing, and excessively wide roof areas exposed between pillars stimulate roof failure. Third mining is almost invariably followed by roof collapse in the workings. Surface manifestations are a function of time, dependent on the rate of upward propagation of settlement.

Surface Topography: As may be anticipated, sloping ground tends to emphasize downward movements because of gravity. Tensile strains may become more marked on hilltops and decrease in valleys. Surface effects are influenced accordingly.

Groundwater: Deformation of the strata around mined areas may alter drainage gradients, resulting in the formation of surface or underground reservoirs (in aquifers). Low-lying areas, such as in central Illinois, may become flooded. Rocks may be weakened by saturation. Erosion patterns could change, and in limestone areas, caverns or karst areas may be created over a period of time.

Where surface runoffs from precipitation or water from leaky mains are allowed to accumulate, water may percolate down through the soil to the fractures and fissures in the bedrock, and finally into the mine openings. The erosion and lubrication effects induce failure.

Water Level Elevation and Fluctuations: Water reduces the strength and stiffness of pillars and the roof and floor markedly. Periodic changes in mine humidity promote deterioration of all these members. Floor softening permits punching, resulting in instability and subsidence. Flow through fissures cause seepage pressures, endangering the stability of the rock mass. Cleavage and bedding planes are lubricated by water, inducing movements.

Mined Area: The critical width needs to be exceeded along both the lateral and longitudinal axes to achieve maximum subsidence. This is especially important if competent strata present in the overburden tend to bridge across the panel and decrease subsidence when the panel width is less than the critical width, even though the length of the panel is greater.

Method of Working: The type of initial subsidence experienced, namely pit or trough, depends on whether room and pillar or longwall mining is being practiced. With room and pillar mining, the eventual collapse of pillars may lead to trenching or sagging of the surface. The displacements and strains over short distances, when they start appearing on the surface, are significant. Nearly immediate but predictable subsidence occurs with longwall mining. Harmonic mining, either by working adjacent longwalls in the same seam or superposed panels in different seams, can be effectively utilized to neutralize compressive and tensile strains and thereby protect surface structures. However, the method is not readily applied and is restricted for use only where mining costs become subservient to historical or social demands.

Rate of Face Advance: Surface subsidence follows the face as it progresses in the panel. If the coal extraction rate varies markedly, the traveling strains also fluctuate. This results in large differential settlements. A fairly rapid, even rate of face advance is best (Legget, 1972).

Backfilling of the Gob: Partial or complete backfilling of the gob reduces, but does not eliminate, subsidence. The amount of subsidence that occurs depends upon the type and extent of backfilling adopted. Thus, for example, hand packing is not as good as pneumatic stowing or hydraulic backfilling.

Time Elapse: The amount of subsidence observed is a function of time. In room and pillar operations, no surface effects may be noted for some time after the mining is complete until the pillars deteriorate or punch into the floor. In longwall mines, the surface may start sagging almost immediately after the face passes below an area. However, the occurrence of massive beds in the overburden could delay this. With longwalls, surface movements are complete within a few years, but when pillars are left intact for support, this may take decades. Room and pillar mining with removal of pillars may produce surface effects similar to longwall mining, with the degree of similarity dependent upon the amount of coal left as fenders or stumps (see also 10.6.3.4).

Structural Characteristics: The extent of damage to a structure is dependent on the type of structure and its size, shape, age, foundation design, construction materials and techniques used, standards of maintenance and repair, and purpose (Chen et al., 1974). The surcharge due to building loads may induce soil compaction, generating instability. Tall structures cannot tolerate much tilt, poorly constructed or older buildings are more readily damaged, and a large edifice is more liable to crack because of the strains and curvature induced by subsidence.

10.6.3 SUBSIDENCE PROFILES

10.6.3.1 Measurement of Subsidence

In order to make subsidence measurements, it is essential to erect monuments that will undergo the same vertical or horizontal displacements as the ground. Several designs of monuments have been used, some of which are depicted in Fig. 10.6.8. Whichever design is selected for use, it should be stable and firmly anchored about 5 ft (1.5 m) below the ground surface so as not to be affected by frost or other surface effects.

The choice of measuring instruments that may be used for subsidence measurements depends on a number of parameters (Panek, 1970):

1. Objectives of the investigation.
2. Area to be covered.
3. Topography of the region.
4. Profiles along which monuments are installed.
5. Spacing and number of monuments or observation stations.
6. Total cost that can be tolerated.
7. Duration of the investigation; survey frequency.
8. Labor requirements for surveying and data reduction.

The horizontal distance between monuments depends on the subsidence gradient. Generally, however, a compromise has to be reached between placing the monuments too close, which increases installation and measurement cost, and too far apart, which does not give enough readings to depict the measured variables adequately. The British National Coal Board (NCB) (Anon., 1975a) has recommended a monument spacing of $0.05D$, where D is the depth of the mined bed. In the United States, the tendency is to increase this distance (Deere, 1961; Gentry and Abel, 1978), and spacings of $0.05D$ to $0.1D$ have been advocated (Panek, 1970). The accuracy of the measurements should be such as to detect strains of 10^{-4}, which is about 1/10th the strain-level for structural damage. The method of measurement and the precautions necessary depends on the distance between monuments and may be obtained from any text on surveying (see also Chapter 8.2; Panek, 1970; O'Rourke et al., 1977).

Vertical displacements may be measured by trigonometric leveling (precision optical or laser), differential leveling, or tilt measurement. When using the theodolite, vertical angles must be measured correct to ½ second of arc. With precise leveling, a micrometer direct reading to about 0.005 ft (1.5 mm) should be employed. An inclinometer with a sensitivity of 10 seconds of arc is generally adequate for subsidence measurements. Second-order subsidence surveys with geodetic level and invar scale or equivalent, should close within $0.035(M)^{1/2}$ feet, where M is the circuit length in miles (Moffitt and Bouchard, 1975), an accuracy of 0.01 ft (3.3 mm) over a 1000-ft (305-m) line. In mountainous or swampy areas, third-order control with a closure of $0.052(M)^{1/2}$ ft may be used, an accuracy of 0.02 ft (6.5 mm) in 1000 ft (328 m). Tiltmeters for use in long-term (several years) subsidence surveys have been developed (e.g., Jacobsen et al., 1975; Holzhausen, 1986).

Automatic data acquisition systems (ADAS) for subsidence have also been used. One device employs one angular displacement transducer to measure tilt and another with an invar wire between monuments to ascertain linear displacement (Schmechel et al., 1977).

10.6.3.2 Subsidence Prediction

Existing subsidence prediction techniques fall under two basic categories, empirical and phenomenological (Voight and Pariseau, 1970; Brauner, 1973; Singh, 1978). The empirical theories are principally based on observations and experience from field subsidence studies. Some of the empirical methods have proved sufficiently reliable for subsidence prediction, at least for a given region. Many of these have been successfully applied in a number of countries, especially in Europe. Phenomenological techniques are based on equivalent material modeling principles where the subsiding strata are mathematically represented as idealized materials that obey the laws of continuum mechanics. Unlike empirical methods, the procedures used in the latter category have not achieved much success to date, mainly due to the difficulty of representing complex geologic properties of the strata in simple mathematical terms.

Promising empirical methods for prediction of subsidence over underground mines consist of the following:

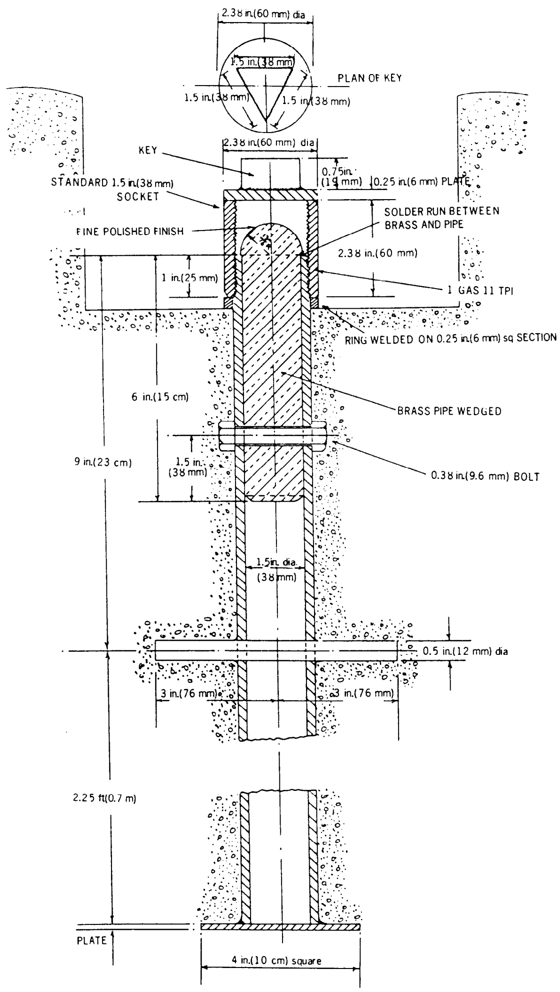
1. Graphical.
2. Profile functions.
3. Influence functions.

Graphical Method: This simply involves displaying subsidence data in the form of graphical charts or nomographs, whereby subsidence magnitude and the associated parameters may be directly obtained for a specified set of mine parameters. This method is adaptable in areas where considerable subsidence data exist, and its applicability is generally restricted to relatively few, geologically similar regions. This technique has seen considerable use in the United Kingdom (Anon., 1975a).

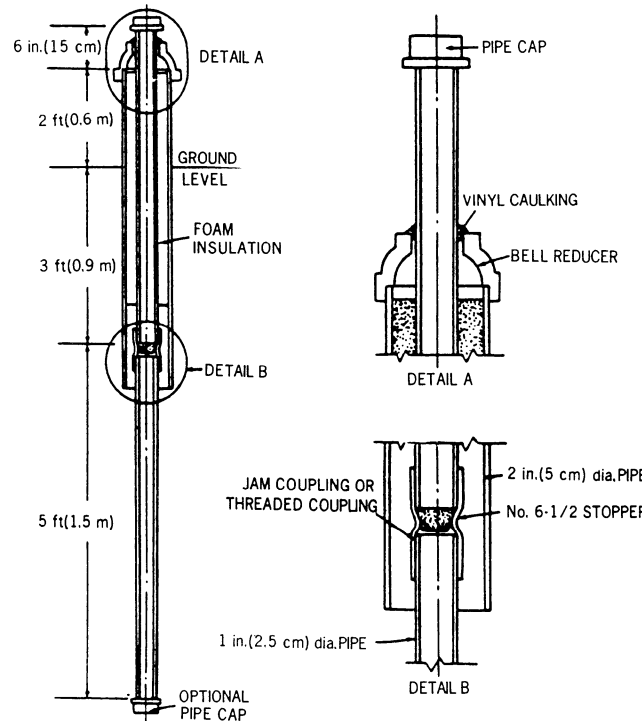
Profile Functions: This involves the derivation of a mathematical function that can be used to plot a complete profile of the subsidence trough at the surface. It differs from a phenomenological approach in that the constraints employed in the profile function are empirically derived from observed data. This method can be readily applied to geologically dissimilar conditions by modifying the constant values. Profile functions have been successfully applied in several countries abroad such as Poland, Hungary, the Soviet Union, and currently in the United States (Gill, 1971; Brauner, 1973; Munson and Eichfeld, 1980; Adamek and Jeran, 1981; Hood et al., 1981; Peng and Cheng, 1981; Wardell, 1982). Selected profile functions are given in Table 10.6.1.

Influence Functions: This principle for subsidence prediction is based on the extraction of infinitesimal elements of area. Subsidence at any point on the surface is obtained from the sum of the influence of each extracted element, using the principle of superposition. Unlike profile functions, influence functions cannot be found directly by measurement. In addition, this method assumes a homogeneous, isotropic overburden material and, therefore, has limited accuracy. In general, influence functions have been found to be especially suitable for subsidence prediction over underground workings with irregular or complex geometries. This method has received considerable attention in Europe, and to a limited extent in this country (Sinclair, 1963; Gill, 1971; Brauner, 1973; Adamek and Jeran, 1981; Hood et al., 1981; Karmis et al., 1981; Peng et al., 1986). Table 10.6.2 depicts a few influence functions.

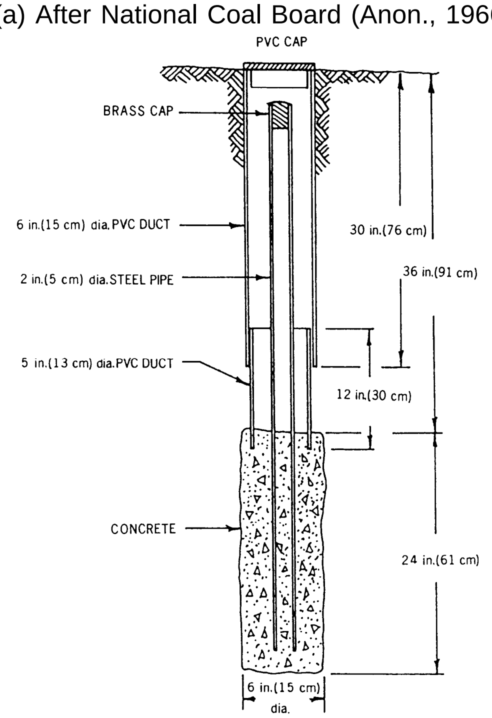
Phenomenological Methods: These are primarily based on the principles of continuum mechanics and assume the media to be elastic (Salamon 1963–4; Berry, 1969; Plewman et al., 1969; Crouch, 1973), viscoelastic (Marshall and Berry, 1967), plastic (Pariseau and Dahl, 1970), and elastic-elastoplastic (Dahl and Choi, 1974). Only the elastic-plastic model has been used with success in the United States (Dahl and Choi, 1974; 1981). Recently an elastic, frictionless, laminated model has been proposed (Salamon, 1989). Subsidence predictions using these various approaches are demonstrated in the following example.



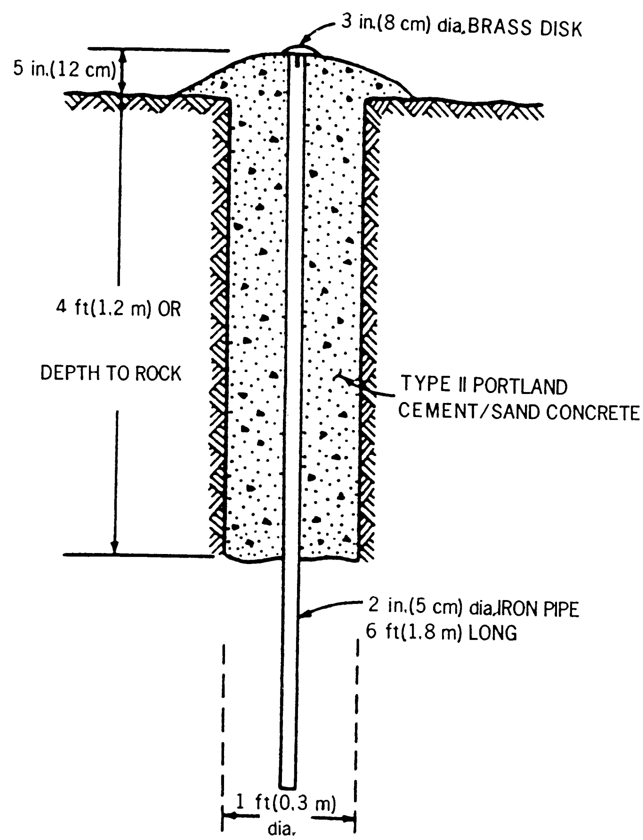
(a) After National Coal Board (Anon., 1966)



(b) After Conroy and Gyarmaty (1982)



(c) After Huck and Bhattacharya (1988)



(d) After O'Rourke et al. (1982)

Fig. 10.6.8. Various subsidence monument designs.

Table 10.6.1. Profile Functions

Name	Function	Country/Area	Reference
Critical Extraction:			
Hyperbolic	$S(x) = \frac{1}{2} S_{max} \left[1 - \tanh\left(\frac{cx}{B}\right)\right]$	UK	King and Whetton (1957) Wardell (1965) Cherny (1966)
Error	$S(x) = \frac{1}{2} S_{max} \left\{1 - \left[\frac{2}{(\pi)^{1/2}} \int_{O}^{(\pi x/B)} exp(-u^2)du\right]\right\}$	Poland/ Upper Silesia	Knothe (1953)
Exponential	$S(x) = S_{max}\, exp\left[-\left(\frac{1}{2}\right)\frac{(x+B)^2}{B^2}\right]$	Hungary	Martos (1958) Marr (1958–59)
	$S(x) = S_{max}\, exp\left[-\left(\frac{cx}{B}\right)^d\right]$	US/Appalachia	Peng and Cheng (1981)
Trignometric	$S(x) = \frac{1}{2} S_{max} \left[1 - \left(\frac{x}{B}\right) - \left(\frac{1}{\pi}\right)\sin\left(\frac{\pi x}{B}\right)\right]$	USSR/Donets	General Institute of Mine Surveying (Anon., 1958)
	$S(x) = S_{max} \sin^2\left[\left(\frac{\pi}{4}\right)\left(\frac{x}{B} - 1\right)\right]$		Hoffman (1964)
Subcritical Extraction:			
Trignometric	$S(x) = S_{max}\,(n_1, n_2)^{1/2}\left[n^2\left(1 - x + \frac{\sin 2\pi x}{2\pi}\right) + \frac{1-n^2}{4}(1 + \cos \pi x)^2\right]$	USSR/Donets	General Institute of Mine Surveying (Anon., 1958)
Hyperbolic	$S(x) = \frac{1}{2} S_{max}\left[\tanh\frac{2(x+w)}{B} - \tanh\frac{2x}{B}\right]$	Poland/ Upper Silesia	Knothe (1957) Wardell and Webster (1957)
		US/Appalachia	Peng (1978)

x = horizontal distance
c = arbitrary constant
B = radius of critical area of excavation
u = integration variable
w = panel width
$S(x)$ = profile function
S_{max} = maximum possible subsidence
n_1, n_2 = coefficients related to width/depth
n = n_1 or n_2 depending on side of panel

Source: Updated from Brauner (1973) and Hood et al. (1981).

Example 10.6.1. Mine M, located in the Appalachian coalfield, is worked by the longwall method and has the following dimensions for a panel:
Depth of seam D = 213 m (700 ft)
Extraction height of seam h = 1.83 m (6 ft)
Panel width w = 152 m (500 ft)
Panel length L = 1219 m (4000 ft)

Solution.

(1) *Graphical Method (NCB)*

The National Coal Board (NCB) procedure has been selected as an example of the graphical method because of its wide use in this country in the past. The method is, however, becoming less popular in the United States, and thus the graphs presented by the National Coal Board (Anon., 1975a) are not reproduced in the interest of space. Reference is made instead to the figures in the original publication.

Step 1: From NCB, p. 9, Fig. 3, for w = 152 m (500 ft) and D = 213 m (700 ft), the subsidence factor, a = 0.68.

Hence the maximum possible subsidence,

$$S_{max} = ah = 0.68 \times 1.83 \qquad (10.6.1)$$
$$= 1.24 \text{ m } (4.1 \text{ ft})$$

Step 2: Since $\frac{w}{D} = \frac{152}{213} = 0.71$, from NCB, p. 13, Fig. 5, the values of $\frac{x}{D}$ (i.e., distance x from the center of the panel, in terms of the depth D) can be read off at various subsidence ratios, 1.00 S, 0.95 S, 0.090 S, . . . (see Table 10.6.3a).

For D = 213 m (700 ft), the values of distance d may be computed, as in Table 10.6.3a.

Step 3: From these data, the subsidence profile, $S(x)$, may be plotted (Fig. 10.6.9).

Step 4: Based on prior experience, the method assumes that the point of inflection occurs at 0.5 S_{max} = 0.62 m (2.05 ft), i.e.,

Table 10.6.2. Influence Functions

Function	Reference
$\phi(r) = \dfrac{S_{max}}{\pi\{\sin\gamma \cos\gamma + [(\pi/2) - \gamma]\}} \dfrac{B^3 \tan^3 \gamma}{r(r^2 + B^2 \tan^2 \gamma)^2}$	Bals (1932–33)
$\phi(r) = \dfrac{3\, S_{max}}{\pi\, B^2}\left[1 - \left(\dfrac{r}{B}\right)^2\right]^2$	Beyer (1945)
$\phi(r) = \dfrac{n\,(2)^{\frac{1}{n}}\, S_{max}}{\pi B \Gamma\ (1/2n)\ r}\, exp\left[-4\left(\dfrac{r}{B}\right)^{2n}\right]$	Sann (1949)
$\phi(r) = \dfrac{2\, S_{max}}{(\pi)^{3/2}\, Br}\, exp\left[-4\left(\dfrac{r}{B}\right)^2\right]$ when $n = 1$	
$\phi(r) = 0.216\,\dfrac{S_{max}}{Br}\, exp\left[-4\left(\dfrac{r}{B}\right)^6\right]$ when $n = 3$	
$\phi(r) = \dfrac{n\, S_{max}}{B^2}\, exp\left[-n\,\pi\left(\dfrac{r}{B}\right)^2\right]$	Litwiniszyn (1957)
$\phi(r) = \dfrac{S_{max}}{B^2}\, exp\left[-\pi\left(\dfrac{r}{B}\right)^2\right]$ when $n = 1$	
$\phi(r) = \dfrac{2\, S_{max}}{B^2}\, exp\left[-2\,\pi\left(\dfrac{r}{B}\right)^2\right]$ when $n = 2$	
$\phi(r) = \dfrac{4.6\, S_{max}}{\pi\, B^2}\, exp\left[-4.6\left(\dfrac{r}{B}\right)^2\right]$	Ehrhardt and Sauer (1961)
$\phi(r) = \dfrac{n\, S_{max}}{2\pi\, r_o^2\, \Gamma(2/n)}\, exp\left(-\dfrac{r}{r_o}\right)^n$	Kochmanski (1959)
$\phi(r) = \dfrac{7\, S_{max}}{B^2}\, exp\left(-6.65\,\dfrac{r}{B}\right)$ when $n = 1$ and $B = 6.65\, r_o$	

r = radial distance from reference point
B = radius of critical area of excavation
γ = angle of draw
n = parameter for characterizing strata conditions
$\phi(r)$ = influence function
S_{max} = maximum possible subsidence
Γ = gamma function
r_o = independent parameter

Source: Brauner (1973); Hood et al. (1981).

from Table 10.6.3a (or NCB, p. 13, Fig. 5), $\dfrac{x}{D}$ 0.36 so x = 0.36 × 213 = 76.7 m (251.6 ft).

Step 5: The slope curve may be drawn from the subsidence profile by taking slopes at various points and plotting.

From NCB experience, the maximum slope is given by

$$2.75\,\frac{S_{max}}{D} = \frac{(2.75 \times 1.24)}{213} = 16.0 \times 10^{-3} \quad (10.6.2)$$

Step 6: The maximum values of extension and compression may be read off from NCB, p. 29, Fig. 15.

For $\dfrac{w}{D} = 0.71$, these are

$+E = 0.67\,\dfrac{S}{D}$, i.e.,

$$+E = \frac{(0.67 \times 1.24)}{213} = 3.90 \times 10^{-3}, \text{ and}$$

$$-E = 0.83\,\frac{S}{D} = -4.83 \times 10^{-3}. \quad (10.6.3)$$

Now from NCB, p. 25, Fig. 12, the values of horizontal strain may be read off in terms of $\dfrac{x}{D}$ and then converted to x, as in Table 10.6.3b.

Step 7:

$$\text{Curvature } r = \frac{(\text{bay length})^2}{\text{differential of the strain}} \quad (10.6.4)$$

(i.e. second differential of subsidence)

Bay length is generally the distance between monuments. These are computed along with the curvature in Table 10.6.3b.

NOTE: All of the above calculations assume that both the seam and the surface are flat. If the surface is sloping, the adjustment in horizontal strain may be obtained from a nomograph (NCB, p. 31, Fig. 18). If, however, the surface is level, but the seam is dipping, the correction in the tensile strain is given in a table (NCB, p. 32, Table 6). The tensile strain on the dip side increases and that on the rise side decreases.

(2) *Profile Function*

A hyperbolic function has been demonstrated to fit well the subsidence characteristics of US mines (Peng and Chyan, 1981; Hood et al., 1981; Karmis and Jarosz, 1988), i.e.,

$$S(x) = 0.5\, S_{max}\left[1 - tanh\left(\frac{cx}{B}\right)\right] \quad (10.6.5)$$

where c = 1.8 for critical and supercritical widths, and c = 1.4 for subcritical widths (Karmis et al., 1986), and

$$B = D \tan\gamma \quad (10.6.6)$$

the radius of major influence.

For mine M, take g = 25°; hence tan g = 0.4663, D = 213 m (700 ft). So B = 99.3 m (325.7 ft). Since the panel width is 152 m (500 ft), it is subcritical.

$$G(x) = S'(x) = -\frac{1}{2}\, S_{max}\,\frac{c}{B}\, sech^2\left(\frac{cx}{B}\right) \quad (10.6.7)$$

$$\rho(x) = S''(x) = \frac{c^2}{B^2}\, S_{max}\left[sech^2\left(\frac{cx}{B}\right)\, tanh\left(\frac{cx}{B}\right)\right] \quad (10.6.8)$$

For a flat seam, the horizontal displacement $u(x)$ and horizontal strain $\epsilon(x)$ are similar but related by a constant b, so that

$$u(x) = -\frac{1}{2}\,\frac{bc}{B}\, S_{max}\, sech^2\left(\frac{cx}{B}\right) \quad (10.6.9)$$

$$\text{and } \epsilon(x) = \frac{bc^2}{B^2}\, S_{max}\left[sech^2\left(\frac{cx}{B}\right)\, tanh\left(\frac{cx}{B}\right)\right] \quad (10.6.10)$$

Table 10.6.3. Computations for Ex. 10.6.1. Graphical Method

$S(x)$	0.00	−0.05	−0.10	−0.20	−0.30	−0.40	−0.50	−0.60	−0.70	−0.80	−0.90	−0.95	−1.00
Subsidence, $S(x)$	0.000	−0.062	−0.124	−0.248	−0.372	−0.496	−0.62	−0.744	−0.868	−0.992	−1.116	−1.178	−1.24
x/D	1.05	0.60	0.49	0.39	0.33	0.29	0.25	0.21	0.18	0.14	0.10	0.06	
Distance, x	223.65	127.80	104.37	83.07	70.29	61.77	53.25	44.73	38.34	29.82	21.30	12.78	0.00

Distance, x	175.73	116.09	93.72	76.68	66.03	57.51	48.99	41.54	34.08	25.56	17.04	6.39
Differ Vert Displ	−0.062	−0.062	−0.124	−0.124	−0.124	−0.124	−0.124	−0.124	−0.124	−0.124	−0.062	−0.062
Spacing	95.85	23.43	21.3	12.78	8.52	8.52	8.52	6.39	8.52	8.52	8.52	12.78
Slope, $G(x) \times 100$	−0.065	−0.265	−0.582	−0.970	−1.455	−1.455	−1.455	−1.941	−1.455	−1.455	−0.728	−0.485

(a) Subsidence and Slope Data

e/E	0.0	0.2	0.4	0.6	0.8	1.0	0.8	0.0	−0.2	−0.4	−0.6	−0.8	−1.0	−0.8	−0.6	−0.4
Strain, $\epsilon(x) \times 100$	0.000	0.078	0.156	0.234	0.312	0.39	0.312	0.000	−0.097	−0.193	−0.290	−0.386	−0.483	−0.386	−0.290	−0.193
x/D	1.05	0.65	0.55	0.48	0.44	0.38	0.33	0.25	0.23	0.20	0.17	0.14	0.08			
Distance, x	223.65	138.45	117.15	102.24	93.72	80.94	70.29	53.25	48.99	42.60	36.21	29.82	17.04			

Distance, x	181.05	127.80	109.70	97.98	87.33	75.615	61.77	5.12	45.80	39.41	33.015	23.43	8.52
Strain Diff	0.078	0.078	0.078	0.078	0.078	−0.078	−0.312	−0.097	−0.097	−0.097	−0.097	−0.097	0.097
Spacing	85.20	21.30	14.91	8.52	12.78	10.65	17.04	4.26	6.39	6.39	6.39	12.78	17.04
Curvature, $\rho(x) \times 10$	9.31	0.58	0.29	0.09	0.21	−0.15	−0.09	−0.02	−0.04	−0.04	−0.04	−0.17	0.30

(b) Strain and Curvature Data

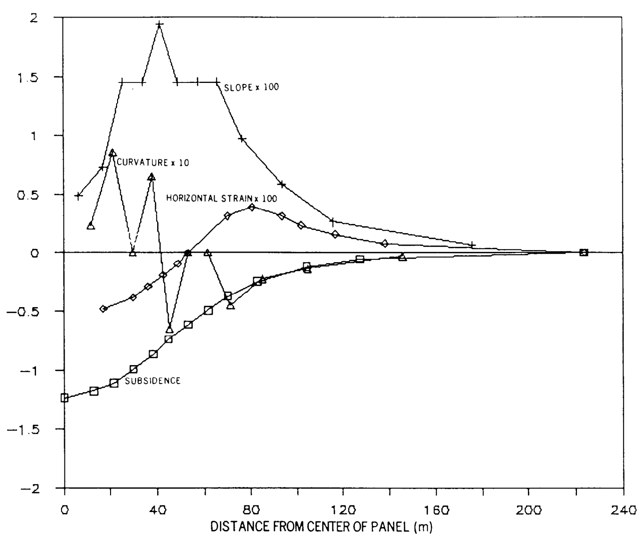


Fig. 10.6.9. Plots of results of Example 10.6.1, using the graphical method.

Peng et al. (1986) have found the horizontal displacement coefficient $b = 0.12$ in Appalachia.

Also, since $S_{max} = 1.24$ m (4.1 ft), $c = 1.4$, and $B = 99.3$ m (325.7 ft),

$$S(x) = 0.5\,(1.24)\left[1 - tanh\left(\frac{1.4x}{99.3}\right)\right]$$

$$G(x) = -0.5\left(\frac{1.4}{99.3}\right)(1.24)\, sech^2\left(\frac{1.4x}{99.3}\right)$$

$$\rho(x) = \left(\frac{1.4}{99.3}\right)(1.24)\, sech^2\left(\frac{1.4x}{99.3}\right) tanh\left(\frac{1.4x}{99.3}\right)$$

$$u(x) = -0.12 \times 0.5\left(\frac{1.4}{99.3}\right)(1.24)\ \mathrm{sech}^2\frac{1.4x}{99.3}$$

$$\epsilon(x) = 0.12\left(\frac{1.4}{99.3}\right)(1.24)\, sech^2\left(\frac{1.4x}{99.3}\right) tanh\left(\frac{1.4x}{99.3}\right)$$

These values are presented in Table 10.6.4 and plotted in Fig. 10.6.10.

(3) *Influence Function*

The influence function, commonly referred to as the Budryk-Knothe function (Knothe, 1957), has been successfully used in the United States for predicting mine subsidence, viz.,

$$f(x) = \frac{1}{B}\, exp\left[-\pi\left(\frac{x}{B}\right)^2\right] \tag{10.6.11}$$

where x is the distance from the origin, located at the inflection point, and B is the radius of major influence $= D \tan g$

Hence the subsidence at the surface point due to the extraction of a unit element Δx is

$$S_a = f(x) \cdot \Delta x \tag{10.6.12}$$

So when the extraction extends from $-x_1$ to $-x_2$ for an effective mining height h, the surface subsidence at any point A is given by

$$S_A = \frac{ah}{B}\int_{-x_1}^{+x_2} exp\left[-\pi\left(\frac{x}{B}\right)^2\right]dx \tag{10.6.13}$$

By definition, $S_{max} = ah$.

Integrating, using the probability density function, $\phi(x)$,

$$S\left(\frac{x}{B}\right) = \frac{S_{max}}{2}\left[\phi\left(\pi^{0.5}\,\frac{x}{B}\right) + 1\right] \tag{10.6.14}$$

Hence the surface slope

$$G\left(\frac{x}{B}\right) = S'\left(\frac{x}{B}\right) = \frac{S_{max}}{B}\, exp\left[-\pi\left(\frac{x}{B}\right)^2\right] \tag{10.6.15}$$

Table 10.6.4. Computations for Ex. 10.6.1, Profile Function Method

Distance, x	150	140	130	120	110	100	90	80	70	60	50	40	30	20	10
Subsidence, $S(x)$	0.0178	0.0235	0.0309	0.0407	0.0534	0.0698	0.0908	0.1175	0.1513	0.1929	0.2434	0.3032	0.3724	0.4497	0.5332
Slope, $G(x) \times 100$	−0.0495	−0.0650	−0.0851	−0.1110	−0.1440	−0.1857	−0.2374	−0.3002	−0.3745	−0.4592	−0.5515	−0.6460	−0.7347	−0.8081	−0.8570
Curvature, $\rho(x) \times 10{,}000$	0.1354	0.1752	0.2279	0.2923	0.3711	0.4646	0.5712	0.6859	0.7983	0.8921	0.9448	0.9306	0.8274	0.6250	0.3384
Horizontal displacement, $u(x) \times 10$	−0.0593	−0.0779	−0.1021	−0.1331	−0.1728	−0.2228	−0.2848	−0.3602	−0.4494	−0.5511	−0.6518	−0.7751	−0.9816	−0.9598	−1.0284
Horizontal strain, $\epsilon(x) \times 10$	0.1625	0.2115	0.2734	0.3508	0.4453	0.5575	0.6855	0.8231	0.9580	1.0705	1.1337	1.1167	0.9929	0.7513	0.4061
Subsidence $S(x) - S_{max}$	−1.2222	−1.2155	−1.2091	−1.1993	−1.1855	−1.1702	−1.1492	−1.1224	−1.0887	−1.0471	−0.9956	−0.9368	−0.8575	−0.7903	−0.7058

Distance, x	0	−10	−20	−30	−40	−50	−60	−70	−80	−90	−100
Subsidence, $S(x)$	0.6200	0.7068	0.7903	0.8676	0.9368	0.9966	1.0471	1.0887	1.1224	1.1492	1.1702
Slope, $G(x) \times 100$	−0.8741	−0.8570	−0.8081	−0.7347	−0.6460	−0.5515	−0.4592	−0.3745	−0.3002	−0.2374	−0.1857
Curvature, $\rho(x) \times 10{,}000$	0.000	−0.3384	−0.6260	−0.8274	−0.9306	−0.9448	−0.8921	−0.7983	−0.6859	−0.5712	−0.4646
Horizontal displacement, $u(x) \times 10$	−1.0489	−1.0284	−0.9698	−0.8816	−0.7751	−0.6618	−0.5511	−0.4494	−0.3602	−0.2848	−0.2228
Horizontal strain, $\epsilon(x) \times 10$	0.0000	−0.4061	−0.7513	−0.9929	−1.1167	−1.1337	−1.0705	−0.9580	−0.8231	−0.6855	−0.5575
Subsidence $S(x) - S_{max}$	−0.6200	−0.5332	−0.4497								

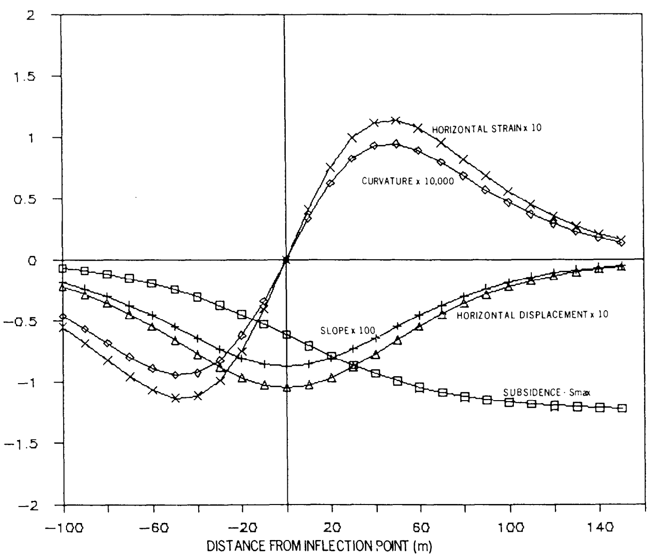


Fig. 10.6.10. Plots of results of Example 10.6.1, using the profile function method.

and the curvature

$$\rho\left(\frac{x}{B}\right) = S''\left(\frac{x}{B}\right)$$

$$= \frac{2\pi}{B^2} S_{max}\left(-\frac{x}{B}\right) exp\left[-\pi\left(\frac{x}{B}\right)^2\right] \qquad (10.6.16)$$

For a flat seam, the horizontal displacement profile $u(x)$ and the horizontal strain $\epsilon(x)$ are similar to the slope and curvature respectively, and related by the strain correlation coefficient b, i.e.,

$$u\left(\frac{x}{B}\right) = b\,\frac{S_{max}}{B}\,exp\left[-\pi\left(\frac{x}{B}\right)^2\right] \qquad (10.6.17)$$

$$\text{and } \epsilon\left(\frac{x}{B}\right) = b\,\frac{2\pi}{B^2}\left(-\frac{x}{B}\right) exp\left[-\pi\left(\frac{x}{B}\right)^2\right] \qquad (10.6.18)$$

Also we know that $B = D \tan g = 213 \tan 25° = 99.3$ m (325.7 ft) and $S_{max} = ah = 1.24$ m (4.1 ft).

For the Appalachian coalfield, $b = 0.12$ (Peng et al., 1986), so

$$S\left(\frac{x}{B}\right) = 0.5\,S_{max}\left[\phi\left(\pi^{0.5}\,\frac{x}{B}\right) + 1\right]$$

$$G\left(\frac{x}{B}\right) = \frac{1.24}{99.3}\,exp\left[-\pi\left(\frac{x}{B}\right)^2\right]$$

$$\rho\left(\frac{x}{B}\right) = \frac{2\pi}{(99.3)^2}\left(\frac{x}{B}\right) exp\left[-\pi\left(\frac{x}{B}\right)^2\right]$$

$$u\left(\frac{x}{B}\right) = 0.12 \times \frac{1.24}{99.3}\,exp\left[-\pi\left(\frac{x}{B}\right)^2\right]$$

$$\epsilon\left(\frac{x}{B}\right) = 0.12 \times \frac{2\pi}{(99.3)^2}\left(-\frac{x}{B}\right) exp\left[-\pi\left(\frac{x}{B}\right)^2\right] \qquad (10.6.19)$$

Since $B = 99.3$ m (325.7 ft), we generate Table 10.6.5 and plot these data as Fig. 10.6.11.

NOTE: In the NCB graphical method, the inflection point of the subsidence curve is assumed to be at the face. Generally, this is a short distance in the gob and can be empirically determined (Brauner, 1973; Kohli and Jones, 1986). In the solutions to the example above, subsidence is considered negative, extension is positive.

10.6.3.3 Prediction Methods Used in the United States

Many of the methods referred to in the earlier sections have been applied to US coalfields, yielding varying levels of success. The most favored technique until recently has been the use of the empirical charts developed by the National Coal Board (Anon., 1975a). Comparison of US subsidence data with NCB predictions highlight the following shortcomings:

1. With the possible exception of Illinois, maximum subsidence factors observed in US coalfields are less than predicted by NCB (O'Rourke and Turner, 1979; von Schonfeldt et al., 1979; Karmis et al., 1981). Probably the large proportion of competent strata in the overburden of most US coalfields is the cause of the discrepancy.

Table 10.6.5. Computations for Ex. 10.6.1, Influence Function Method

Distance, x	150	140	130	120	110	100	90	80	70	60	50	40	30	20	10
Subsidence, $S(x)$	−1.24	−1.24	−1.24	−1.24	−1.24	−1.23	−1.22	−1.21	−1.19	−1.16	−1.11	−1.04	−0.96	−0.86	−0.74
Slope, $G(x) \times 100$	−0.0010	−0.0024	−0.0057	−0.0127	−0.0264	−0.0516	−0.0946	−0.1625	−0.2621	−0.3966	−0.5631	−0.7500	−0.9374	−1.0993	−1.2096
Curvature, $\rho(x) \times 10{,}000$	0.0074	0.0174	0.0383	0.0783	0.1494	0.2652	0.4373	0.6681	0.9428	1.2228	1.4467	1.5417	1.4452	1.1298	0.6216
Horizontal displacement, $u(x) \times 10$	−0.0012	−0.0029	−0.0069	−0.0152	−0.0317	−0.0619	−0.1135	−0.1950	−0.3145	−0.4759	−0.6757	−0.9000	−1.1249	−1.3192	−1.4515
Horizontal strain, $\epsilon(x) \times 10$	0.0089	0.0209	0.0459	0.0940	0.1793	0.3183	0.5248	0.8018	1.1314	1.4674	1.7361	1.8501	1.7342	1.3558	0.7459

Distance, x	0	−10	−20	−30	−40	−50	−60	−70	−80	−90	−100
Subsidence, $S(x)$	−0.62	−0.50	−0.38	−0.28	−0.20	−0.13	−0.08	−0.05	−0.03	−0.02	−0.01
Slope, $G(x) \times 100$	−1.2487	−1.2096	−1.0993	−0.9374	−0.7500	−0.5631	−0.3966	−0.2621	−0.1625	−0.0946	−0.0516
Curvature, $\rho(x) \times 10{,}000$	0.0000	−0.6216	1.1298	−1.4452	−1.5417	−1.4467	−1.2228	−0.9428	−0.6681	−0.4373	−0.2652
Horizontal displacement, $u(x) \times 10$	−1.4985	−1.4515	−1.3192	−1.1249	−0.9000	−0.6757	−0.4759	−0.3145	−0.1950	−0.1135	−0.0619
Horizontal strain, $\epsilon(x) \times 10$	0.0000	−0.7459	−1.3558	−1.7342	−1.8501	−1.7361	−1.4674	−1.1314	−0.8018	−0.5248	−0.3183

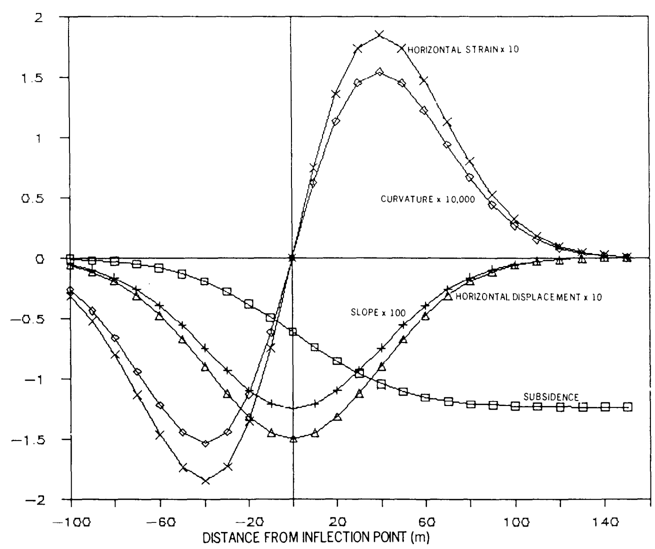


Fig. 10.6.11. Plots of results of Example 10.6.1, using the influence function method.

2. Limit (draw) angles in the US coalfields tend to be less than the 35º value generally accepted by NCB (Abel and Gentry, 1978; King and Gentry, 1979; von Schonfeldt et al., 1979; Bauer and Hunt, 1981; Karmis et al., 1981), because of the bridging effect of the strong strata in US coal measures. Typical draw angles for various countries and coalfields are given in Table 10.6.6.

3. The points of inflection of the subsidence profiles over US coal mines are generally closer to the panel centerline compared to the NCB profile (O'Rourke and Turner, 1981; Adamek and Jeran, 1981); this effect is dependent not only on the percentage of competent strata in the overburden but also on their locations relative to the ground surface and their thicknesses.

4. Surface strains and curvatures observed over US longwall panels have been shown to be significantly higher than NCB predictions, almost four times larger in many cases (O'Rourke and Turner, 1981; Adamek and Jeran, 1981; Moebs, 1982). It has been suggested that the presence of ancient workings, different in situ horizontal stress conditions, and the relatively unconsolidated nature of the surficial deposits above British mines may have resulted in the lower strain values (O'Rourke, 1983).

For the most part, the discrepancies in the observed values are a result of the differences in geologic characteristics between British and US coal measures. Most of the observations constituting the NCB charts were taken within relatively narrow geographical boundaries. Therefore, it may be expected that these data are inapplicable in the United States on a global basis. Only in areas where the geology is similar to that in the British coalfields can these charts apply. In other mining regions, these charts have to be modified prior to application.

In addition to the NCB-type subsidence predictive charts, studies have been undertaken to evaluate the potential of profile and influence functions for subsidence prediction in the United States. Among the profile functions that have been recommended (Munson and Eichfeld, 1980; Kohli et al., 1980a; Daemen, 1981; Hood et al., 1981; Adamek and Jeran, 1981), the hyperbolic tangent function appears the most promising. One drawback of profile functions is that their applicability is restricted to mine geometries that are simple, such as longwall panels. Most US coal mines are worked by room and pillar methods, rendering the profile function techniques less applicable. The use of influence functions is expedient in these instances since complex room and pillar layouts can be readily modeled. One application of influence functions is the zone area method, originally developed by Marr (1975). This method has been successfully used (Karmis and Haycocks, 1983; Karmis and Jarosz, 1988; Peng et al., 1986; Luo and Peng, 1989), since it lends itself to computer treatment.

Table 10.6.7 lists some computer programs currently available in the United States.

10.6.3.4 Time Effects

The duration of subsidence resulting from mining is composed of two distinct phases: (1) active and (2) residual. Active subsidence refers to all movements occurring simultaneously with the mining operations, while residual subsidence is that part of the surface deformation that occurs following the cessation of mining (or in the case of longwall mining, after an underground excavation has reached its critical width). The duration of residual subsidence is of particular importance from the standpoint of structural damage at the surface as well as from a legal perspective. The latter involves evaluating the extent of liability of underground mine operators for postmining subsidence.

Time spans during which surface subsidence may occur vary markedly with the mining method being used. Longwalls induce subsidence rapidly, beginning almost immediately after mining. With room and pillar systems, major occurrences of surface subsidence may be delayed for decades until the support pillars have substantially deteriorated and collapsed. The actual time involved depends on a number of factors such as the strengths of coal, roof, and floor; extent of fracturing; presence of water;

Table 10.6.6. Typical Values of Angle of Draw

Coalfield/Country	Reference	Angle of Draw (degrees)
Limburg/Netherlands	Brauner (1973)	35–45
Limburg/Netherlands	Pottgens (1978)	45
Northern France	Brauner (1973)	35
USSR	Brauner (1973)	30
Ruhr/Germany	Brauner (1973)	30–45
Ruhr/Germany	Kratzsch (1983)	55
Saar/Germany	Kratzsch (1983)	40
UK	ICE (Anon., 1977)	25–35
Midlands/UK	Orchard (1957), Wardell (1969), NCB (1975)	35
US:		
East — Anthracite	Montz and Norris (1930)	25
Southwestern, PA	Newhall and Plein (1936)	10–25
Appalachian	Cortis (1969)	15–27
Appalachian	Peng and Chyan (1981)	22–38
Northern Appalachian	Adamek and Jeran (1981)	12–17
Central — Illinois	Wade and Conroy (1977)	23–29
Illinois	Conroy (1979)	15–30
Illinois	Bauer and Hunt (1981)	12–26
Illinois	Hood (1981)	17–18 (long.)
Illinois		42–44 (trans.)
West — Raton, NM	Gentry and Abel (1977)	16
Deer Creek, Emery, UT	Allgaier (1988)	30
Somerset, Gunnison, CO	Dunrud (1984)	15–25
Salina, UT	Dunrud (1984)	8–20
Sheridan, WY	Dunrud (1984)	6–9

depth of workings; pillar size; and percentage extraction. Hence prediction of when or how much damage may occur becomes difficult.

Longwall Mining: The duration of residual subsidence movements above longwall panels is relatively short, typically varying between a few weeks and about 10 years. Further, the magnitude of these movements rarely exceeds about 10% of the total subsidence. The time spans reported in the literature are summarized in Table 10.6.8.

Empirical relations proposed to estimate the duration of residual subsidence include:

1. United Kingdom (Institution of Municipal Engineers, Anon., 1947):

$$\text{Time (mo)} = 6 + \frac{\text{mining depth (yd)}}{100} + \frac{\text{face advance (yd)}}{\text{rate of face advance (yd/mo)}} \qquad (10.6.19)$$

2. USSR, Germany (Brauner, 1973; Kratzsch, 1983):

$$s = 1 - e^{-ct} \qquad (10.6.20)$$

where s is ratio of instantaneous subsidence to final subsidence, c is overburden strata characteristic, t is subsidence time span, and e is base of natural logarithms.

3. USSR (Shadrin and Zamotin, 1977):

$$t = c[(1/q) - 1]^{1/2} \qquad (10.6.21)$$

where t is subsidence time span, c is overburden strata characteristic, q is a constant relating mining depth, panel width, and structure and hardness of overlying rock, i.e., $1/[1 + (D/w)^n]$, D is mining depth, and w is width of workings. Nomograms have also been developed and used for this purpose.

None of the above quantitative relations are immediately applicable to US conditions because of the site-specific constants contained within each expression.

Room and Pillar Mining: *Mechanism of Subsidence Development*—In room and pillar mining without pillar recovery, the magnitude of active subsidence is generally small, and the ground surface may experience a variable frequency of subsidence incidents during this period. The coal pillars and the surrounding rock are usually relatively sound at this time with only minor deflections of the roof being transmitted to the surface.

Some time after mining, however, complete collapse of the abandoned pillars and the adjacent strata may occur as a result of natural causes or human activities. These processes are likely to continue until all the voids created by mining excavation have been filled by the caved strata. Consequently, in the case of room and pillar mining, the residual subsidence can be the major subsidence measured on the surface.

There have been some misconceptions among the US mining community that surface subsidence may be avoided if certain conditions are fulfilled:

1. Sufficient coal is left unworked to serve as load-bearing pillars (generally over 50%).
2. Mining is conducted at great depths.
3. Strata overlying the workings contain competent beds.

Recent studies, however, have shown that no matter how well-designed a room and pillar layout might be, the additional weight transmitted to the pillars due to excavations will cause measurable deformation on the pillars, and these movements will eventually be transmitted to the surface. Depending upon the extent of pillar loading and the characteristics of the pillars and the superincumbent material, the surface deflection may vary from considerable to negligible, and sometimes is nearly undetectable. The long-term stability of mine pillars is extremely difficult to determine.

Table 10.6.7. US Computer Programs for Subsidence Prediction

Program	Year	Developer	Basis	Output	Merits	Limitations	Comments
INTEX	1982	international Exploration, Inc.	Empirical—finite element	Max vert subsidence max horiz comp strain break angle draw angle	Accommodates joints and faults. Can model local conditions.	Requires knowledge of specific coal and rock properties, and mine layout.	Developed for anthracite coalfield
SPASID (subsidence prediction and system identification)	1983	Pennsylvania State University	Influence function (Knothe and elastic)	Vertical subsidence horizontal subsidence profile horizontal strain grid lines percentage subsidence with respect to face	Can determine optimum influence function from measurements. Usable as batch program. Permits modification of input to suit geology. Considers dip, gates, edge effects, time, asymmetric profiles of adjacent panels. 2-D and 3-D plots. Math co-processor not required.	Command-driven, slow. FORTRAN and subsidence knowledge required. Needs graphical software. Default parameters are from UK experience.	
SUBPRO (subsidence professional)	1985	USBM	Combined profile and influence functions	Vertical subsidence inclination curvature horizontal strain	Has variable coefficient to suit geology. Can accommodate actual overburden data or average. Computer or subsidence knowledge not required. Special hardware not needed.	Maximum profile is 300 ft on either side of panel edge. Cannot store input or computed data.	Restricted to northern Appalachian coalfield. Parameters cannot be modified. Requires BASIC software.
SUBSIDE 2.0	1986	Buelah Engineering	Influence function (Bals)	Subsidence strain profiles	Permits changing input. Different parameters for room-and-pillar, varied overburden, multiple seams, dip. Generates design load/safety factor for pillars. Panel/profile data interchangeable. Operates batch jobs. AUTOCAD tie-in. Menu driven. Works with limited input data.	Cannot accommodate discrete points. Automatically generated intermediate points may not align with input data. Knowledge of subsidence required.	Developed for Appalachia.
SDPS (surface deformation prediction system)	1987	Virginia Polytechnic Institute and State University	Profile function	Vertical subsidence location of inflection point panel subcritical, critical, supercritical	Easy to use. Plots profiles.	Cannot predict subsidence at specific points.	Developed for Appalachian coalfield, but used elsewhere.
			Influence function	Subsidence slope curvature strains at discrete points	Displays 2-D and 3-D plots. Can accommodate room-and-pillar mining, dip, flexibility in input parameter.	Need to exit program to access deformation indicies.	
			Zone area	Maximum subsidence	Cannot compute subsidence at discrete points. Cannot adjust subsidence parameter	Plots are 2-D and 3-D. Can adjust subsidence parameters. Needs math co-processor and knowledge of subsidence.	
CISPM (comprehensive, integrated subsidence prediction model)	1988	West Virginia University	Influence function	Subsidence horizontal displacement slope strain curvature	Can take data directly from total survey station or electronic distance meter. Parameters automatically recommended, but can be adjusted for geology. Can graph all solutions. Accommodates profiles and discrete points. Computer and subsidence knowledge not required. Menu-driven.	Needs math co-processor. Has six programs—LWSUB, SUBSDNC, DYNSUB, SURVEY, SUBDED, CONSULT.	Developed for Appalachian coalfield, but used elsewhere. Assumes surface subsidence complies with normal probability distribution.

Sources: Anon. (1982), Ingram et al. (1989), Luo and Peng (1989), Martin (1990). **Author's Note:** Subsidence computer programs are continually being revised, modified, and developed. This information may not reflect the latest version, nor does this table list all available programs.

Table 10.6.8. Residual Subsidence Duration Over Longwall Mines

Reference	Country/ Coalfield	Residual Subsidence Duration
Institution of Municipal Engineers (Anon., 1947)	UK	2 to 10 years
Orchard & Allen (1974)	UK	Several months to 3 to 6 years (strong overburden)
Collins (1977)	UK	2 to 4.5 years
Grard (1969)	France	6 to 12 months
Brauner (1973)	Germany	1 year (Cretaceous overburden) 2 years (sandstone overburden)
Brauner (1973)	USSR	2 years (shallow mines) 4 to 5 years (deep mines, > 1300 ft or 400m)
Shadrin and Zamotin (1977)	USSR	2 to 25 months
Gray et al. (1977)	US/Appalachian	Few months to few years
Hood et al. (1981)	US/Illinois	12 months

The three basic mechanisms responsible for residual subsidence over room and pillar mines include:

1. Collapse of roof beds spanning adjacent pillars.
2. Pillar failures.
3. Squeezes or crushes.

1. *Roof Collapse.* Over remnant pillars, this is perhaps the most prevalent failure mechanism associated with abandoned room and pillar mines. Depending on certain geometric and geotechnical factors, the caving process may be arrested at some point in the overburden or it may extend upwards to the surface. The surface expression of this process is generally in the form of a localized depression or pit.

The height to which the collapse process can take place is a function of

a. Volume of the original mine opening or room.
b. Bulking factor of the strata material.
c. Location and thickness of overlying competent strata.

Two basic modes of roof failure have been recognized, namely, shear and flexural failure (Morgan, 1973). The former usually initiates diagonal tension cracks near the junction of the mine pillar and the roof, and the latter causes tension cracks near the midspan of the roof. Both result in voids above the mine level. Dependent on the mechanism of failure of the individual roof beds and their tensile and shear strengths, a variety of geometric forms of collapse are possible, ranging from conical through wedge to rectangular. For a given width of mine opening, it can be demonstrated that, for each type of collapse, the height of collapse is a function of the overlying strata. The influence of competent strata in the overburden has been neglected in this analysis (Piggott and Eynon, 1977).

2. *Pillar Failures.* These occur due to changes in the environment or surcharged loading and may take place at the time of mining or after considerable delay. They result in trough-like subsidence.

In general, subject to pillar geometry, pillar failure does not ordinarily occur at shallow depths since the size of coal pillars left behind are usually much greater than that required to support the overlying strata or any additional loading from surface development. However, where very small pillars or "stubs" exist within a given mining section, these may fail and cause sufficient loads to be transferred to adjoining pillars by arching, resulting in extended failure.

In most instances, pillar failures coincide with some phase of mining, such as pillar robbing on the retreat, abandonment of a particular mining area, or working other seams in close proximity. Another common cause of pillar failure is the action of concentrated foundation loads, from pile foundations or otherwise, being transmitted onto the remnant pillars (Piggot and Eynon, 1977).

3. *Squeezes or Crushes.* When abandoned pillars punch into either the immediate roof or floor that might have been weakened or altered by the action of water or other weathering processes, squeezes (crushes) may result. Generally, the surface settles as a trough or basin.

The mechanism of failure in this case is not unlike the failure of building foundations as the load carried by the mine pillar is transferred to the floor (or roof). If the bearing capacity of either the roof or floor is exceeded, squeezing may occur. The following factors (Gray et al., 1977) favor bearing capacity failure:

a. Underclay mine floor.
b. High pillar stresses.
c. Flooded mine conditions.

Factors Influencing Duration of Residual Subsidence—The factors in room and pillar mines that govern the duration of residual subsidence have not been quantified as yet. Probably the following parameters play a role:

1. *Depth of Working:* Increased depth implies a longer duration for subsidence movements. Any instability caused at the mine level has to propagate through the overburden in order to reach the surface.

2. *Mine Geometry:* This may be expressed in terms of the following attributes:

a. Seam thickness.
b. Pillar width-to-height ratio.
c. Extraction ratio.
d. Presence of multiple panels.
e. Presence of multilevel workings.

Increased *seam thickness* increases the potential for instability of pillars and speeds up the subsidence process.

Both the *pillar width-to-height ratio* and *extraction ratio* reflect upon the safety factor built into the mine design. Pillar width-to-height ratios greater than 0.1 and extraction ratios of less than 50% have both been claimed to permit no surface subsidence. Although mines designed to these standards have been known to be stable for long periods of time, sometimes more than a hundred years, this is not strictly true.

Presence of multiple panels and *multilevel workings* generally result in a shorter residual subsidence phase since they increase the volume of underground voids.

3. *Strength and Deformation Characteristics of the Roof Floor, and Pillar:* Over the long term, these affect the duration of surface subsidence depending on the interrelationships of these structural members.

4. *Types of Roof Control:* Roof control practices in a mine influence the relative susceptibility of the roadways to collapse; for example, bolted mines tend to subside faster than those with cribs, steel supports, or other types of bracing.

5. *Character of the Overburden:* Significant aspects which profoundly govern the duration of subsidence movements are

a. Thickness of surficial soil beds.

b. Lithology.
c. Structural geology.

Soil thickness is important since the fractures propagate through it rapidly. Also granular materials (e.g., sands) offer less bridging capacity than fine-grained soils (i.e., clays).

Although the effect of *lithology* is poorly understood, weaker rocks (i.e., shales and siltstones) are generally unable to support their own weight and the strata above, and transmit subsidence movements to the surface in a short time span. Competent rocks (e.g., sandstones and limestones) effectively bridge over excavations and delay the residual subsidence period. Besides their relative competency, the thicknesses of these strata govern the duration of subsidence; massive beds inhibit the propagation of subsidence movements longer than thin, laminated formations. Also affecting the process are facies changes, lensing, pinchouts, and other lateral variations of geology that may alter the character of the overburden from one place to another. Joints occur even in competent strata, and some slippage along these may be expected with time. Thus even though some investigators suggest that a competent rock layer of thickness greater than 1.75 times the width of the workings will arrest the collapse (Piggott and Eynon, 1977), other studies (Thornburn and Reid, 1977; Gray et al., 1977) show that such competent beds merely delay the subsidence process.

Structural geology impacts subsidence in the same manner as lithology, by varying the ability to bridge excavated spans. Generally, surface geologic features (e.g., faults, photolineaments, stream valleys), and underground features (e.g., bedding planes, joints, fissures, cleat, folds, or other inhomogeneities) tend to shorten the subsidence period.

6. *Presence of Old Mined-out Workings:* Old workings in the vicinity of an active mine accelerate the rate of residual subsidence, since the surrounding strata are disturbed.

7. *In Situ Stress Field:* The existence of high horizontal stresses impacts the time for subsidence since the structural integrity of the mine supports is affected.

8. *Water:* The presence of water reduces the strength and stiffness of mine pillars, roof, and floor in flooded mines. Further, softening of the floor (e.g., underclay) encourages pillar punching, resulting in instability and subsidence. Flow of water through fissures causes seepage pressures in the rocks, endangering the rock mass stability. Generally, the formation of pits in shallow mines is promoted by these factors. Dewatering of flooded mines accelerates coal pillar deterioration by exposing submerged pillars to the damaging effects of air and removing the buoyant support afforded by the water.

Periodic *changes of humidity* cause the slow deterioration of pillars, roof, and floor, with similar results.

9. *Nonmining factors:* Those that affect subsidence include
a. Mine fires.
b. Earthquakes.
c. Tectonic movements.
d. Surface precipitation.

Although not common, *mine fires* accelerate the subsidence process due to degradation of abandoned pillars (Dunrud and Osterwald, 1980). *Earthquakes* and *tectonic movements* may destabilize otherwise stable areas. Experience indicates a direct relationship between increased *rainfall* and greater subsidence activity (Anon., 1975b; Gray et al., 1977).

Prediction of Time of Subsidence—The wide variety of interrelated factors that may affect the duration of residual subsidence over room and pillar workings renders the task of accurate prediction of the time of subsidence difficult. Field observations of the time period of residual subsidence is further complicated by the fact that these movements generally continue over prolonged periods of time. Thorburn and Reid (1977) reported a case study of a subsidence event occurring about 118 years following mine abandonment. Ivey (1978) observed subsidence in Colorado 73 years after cessation of mining. Dunrud and Osterwald (1980) concluded that surface subsidence is likely to occur for several years or several decades after mining and reported subsidence events over workings in Wyoming that were driven 25 to 80 years earlier. Mahar and Marino (1981) made similar observations in Illinois where mining had taken place several months to more than 100 years prior to the occurrence of subsidence. Based on data collected from subsidence incidents in Pennsylvania and Illinois, a common trend is not evident (Fig. 10.6.12). Investigations in Illinois (Bauer and Hunt, 1981) showed no direct relationship between depth of workings and the time period of subsidence; Pennsylvania data show similar scatter. Kratzsch (1983) developed an analytical relation between the time factor and parameters such as pillar size, shape, deformation behavior, and degree of backfilling, based on data from room and pillar salt mines. Its applicability to coal mines is unverified. A generalized model for subsidence prediction over partial-extraction mining is not currently available.

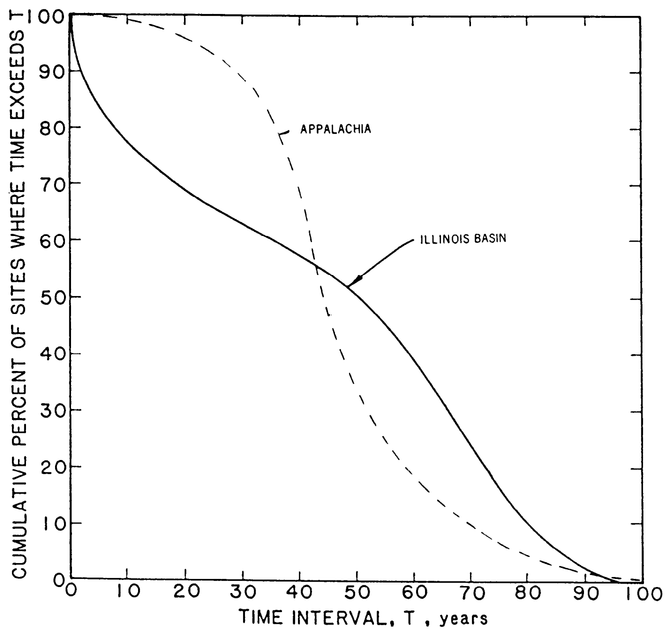


Fig. 10.6.12. Relationships for time interval between mining and subsidence. *Source:* Appalachia (Gray et al., 1977); Illinois data (Bauer and Hunt, 1981).

10.6.4 SUBSIDENCE-INDUCED DAMAGE

10.6.4.1 Surface Structures

Buildings: Damage to surface structures, especially buildings, is mainly caused by tilt, angular distortion, bending, and horizontal strain. Several distinct types of damage are evident as manifestations of tension, compression, angular distortion, shear, bending, and rigid body rotation and/or translation of the structure. Building deformations from ground movements usually begin at foundation level and propagate upward through the basement to the superstructure (Anon., 1975a; Bruhn et al., 1982). The transmission of the deformations from the foundation to the superstructure depends on the nature of the structure, in particular its continuity and attachment.

Tensile stresses developed by horizontal ground strain tend to produce vertical and steplike cracks in brickwalls, generally across and along the mortar-brick interfaces. Most of the extension cracks tend to be vertical, particularly near the base of the structure, and usually propagate from weak elements, such as doors, windows, or construction joints. These are generally uniformly open or more open at the bottom than the top. In the upper part of the structure, the cracks may become diagonal. Extension cracks on the basement floor are usually open and show little or no offset between the sides of the crack, usually appearing at right angles to the direction of the tensile stresses.

Compressive damage is characterized by bulging and bending failures of foundation cracks. Severe compression can cause the lower parts of the walls to move closer together and induce bending stresses in the structure that cause tensile cracks near the upper part of the walls and compressive failure at the lower part. In masonry structures, however, compressive stresses are usually not transmitted to the upper part of the walls due to the low shear resistance along mortar/brick contacts. In such cases, the induced cracks are generally restricted to the lower parts of the structure. Where the floor is especially weak, heaving of floor tiles or buckling failure may result.

Angular distortion related to differential settlement can cause vertical cracks in floor slabs that usually form closer to the side that has settled more. Walls subject to angular distortion exhibit diagonal cracks in the plaster or masonry walls which tend to extend toward the side with greater settlement. Angular distortion also causes binding of doors and windows.

Differential tilting or bending of a structure induces shear stresses in the walls and slabs. Under these circumstances, damage is characterized by horizontal and vertical cracks in the walls. Alternatively, shear and bending may act together to cause diagonal cracks in the walls.

These descriptions exemplify the effect of a single deformation mode acting alone on the structure. In actual practice, several of these forces may occur together and produce a complex pattern of cracking and distortion in various locations and directions. Furthermore, the damage descriptions provided above are limited to visible damage external to the structure. Damage caused to structural members or foundations is difficult to characterize in simple terms since the nature of the damage depends largely on the several variables including strength properties of the structural members, type of construction, zones of weakness, and previous deformation history.

In addition, the length of the structure has a major influence on the relative severity of the resulting damage. Studies have indicated that the longer the structure, the greater the damage severity (Anon., 1975a; Peng and Cheng, 1981a).

Damage Criteria—A number of damage classifications have been developed which correlate building damage to ground movements; only three are presented here. The scheme developed by the National Coal Board (Anon., 1975a) on the basis of direct observations of building damage in the United Kingdom is probably the best known (Table 10.6.9). A similar system has been developed for Northern Appalachian coalfields (Table 10.6.10), which relates damage to a severity index, based on the repairs required for building basements, instead of ground movements (Bruhn et al., 1982). Currently, a sufficient database does not exist to extend this nationwide. The results of a survey of a wide range of sources is presented in Table 10.6.11. A brief discussion of these and other classification systems is presented by Bhattacharya and Singh (1985).

Bridges: Damage may be caused to bridges by horizontal ground strain resulting in the movement of the supports of piers either towards or away from one another. Differential vertical settlement or distortions in the horizontal plane may bring about complex and often serious effects on the decking and arches (Anon., 1975a).

It is generally difficult to determine the location and nature of the ground movements based on visual observation of damage. Often compressive damage leads to crushing and spalling of concrete decks and plaster. Combined compression and extension due to bending may cause opening and closing of construction joints in abutments and plaster cracking. At the higher end of the scale, damage is characterized by distress in the superstructure, inward horizontal movement of abutments, jamming of beams and girders against the back wall of the abutments, and serious damage to the bearings (Moulton et al., 1982).

Damage Criteria—Few data on damage criteria for bridges are currently available. Results of a survey are presented in Table 10.6.12.

Miscellaneous Structures: Included in this category are dams, embankments, reservoirs, canals, locks, aqueducts, sewers, sewage disposal works, and other common types of structures. Very limited data are available on these structures.

Measurement of ground movements in the failure investigation of an earth dam due to extension cracks in the concrete embankment, revealed that the magnitude of horizontal strain developed was about 1×10^{-3} (Lee, 1976). Tensile strains greater than 1.5×10^{-3} resulted in damage to dams in the Kyushu coalfields of Japan (Nishida and Goto, 1969).

A failure criterion of 1×10^{-3} for limiting horizontal strain has been recommended in the design of two reservoirs at one location, and between 0.3 to 2.75×10^{-3} for another (Lackington and Robinson, 1973).

The consensus appears to converge upon a limiting horizontal ground strain of around 1×10^{-3}, applicable to a variety of surface structures. However, this should be regarded only as a first estimate, and site-specific analyses should be conducted prior to design recommendations.

10.6.4.2 Public Utilities and Communications

Roads and Airport Runways: The principal types of damage affecting roads and highways due to subsidence movements include:

1. Cracks on the road surface.
2. Deterioration of base course and/or subgrade.
3. Distortion of horizontal and vertical alignment.
4. Bumps or undulations on the road surface.
5. Damage to ancillary works such as sidewalks, drains, fences, curbs, and the like.
6. Water flooding.

Of these, the most common form of damage is the formation of tensile cracks on the road surface, usually coinciding with the position of the ribsides in the mine workings below. Altematively, compression ridges may occur near the center of the panels. Also common are severe local changes of gradient that may become a source of danger for high-speed traffic, especially if it causes surface water to stand in pools in affected areas.

Airport runways experience problems similar to those for high-speed highways.

Damage Criteria—It is generally accepted that highway bridges are more susceptible to ground movements than roadways. Hence damage criteria for bridges often are used to represent a conservative limit of such criteria for roads. Further, most public roads are protected by safety pillars and generate little interest for accumulating data. Besides, road damage can generally be repaired with tar or “cold patch” treatment, which is relatively inexpensive. Table 10.6.13 presents some available data.

Table 10.6.9. National Coal Board Classification of Subsidence Damage

Change of Length of Structure		Class of Damage	Description of Typical Damage
From	*To*		
	Up to 0.1 ft (30 mm)	Negligible or very slight	Hair cracks in plaster. Perhaps isolated slight fracture in the building, not visible on outside.
0.1 ft (30 mm)	0.2 ft (60 mm)	Slight	Several slight fractures showing inside the building. Doors and windows may stick slightly. Repairs to decoration probably necessary.
0.2 ft (60 mm)	0.4 ft (120 mm)	Appreciable	Slight fracture showing on outside of building (or one main fracture). Doors and windows sticking; service pipes may fracture.
0.4 ft (120 mm)	0.6 ft (180 mm)	Severe	Service pipes disrupted. Open fractures requiring rebonding and allowing weather into the structure. Window and door frames distorted; floors sloping noticeably; walls leaning or bulging noticeably. Some loss of bearing in beams. If compressive damage, overlapping of roof joints and lifting of brickwork with open horizontal fractures.
Over 0.6 ft (180 mm)		Very severe	As above, but worse, and requiring partial or complete rebuilding. Roof and floor beams lose bearing and need shoring up. Windows broken with distortion. Severe slopes on floors. If compressive damage, severe buckling and bulging of roof and walls.

Source: Anon., 1975a.

Table 10.6.10. Subsidence Damage Classification for Northern Appalachian Coalfield

Class	Characteristic Basement Damage	Severity Index
I Slight	• Hairline cracks in one or more basement walls and possibly floor slab. • Some cracks in perimeter walls causing loss of water tightness. • Repointing required in some or all walls.	0
II Moderate	• Cracks in one or more basement walls and floor slab. • Some wall/footing reconstruction and floor replacement required, as well as local repointing.	1
III Severe	• Cracks in one or more basement walls and floor slab. • Possible wall instability and loss of superstructure support, requiring shoring and bracing. • Extensive repair work involving wall/footing reconstruction and floor slab replacement.	2
IV Very severe	• Cracks typically in all basement walls, as well as floor slab. • Possible instability of several walls and loss of superstructure support, requiring extensive shoring and bracing. • Possible significant tilt to home.	4
	• General reconstruction of basement walls, footings and floor slab required.	5

Severity Index is the relative cost of repairing basement. *Source:* Bruhn et al. (1982).

Railroads: One of the first effects of subsidence on railroad tracks is rider discomfort, which sometimes requires the reduction of maximum permissible speeds. At higher levels of ground strain, rail tracks have a tendency to "snake" or bend, and in more extreme cases, entire rails may be forced out of the track. Sometimes where the mine workings are located on one side of the track, differential lateral strains may occur and cause displacement of the tracks relative to one another. Changes in ground slope may adversely affect track performance by formation of localized depressions, or creating gradients greater than permissible for a given type of traffic.

One effect commonly observed over longwall panels is that, depending on the location of the coal working faces, reversals of stress and strain occur on the surface. A railway line might be subjected to tensile stresses at first, followed by a neutral stress period, and then by compressive stresses. This transition is generally the most damaging phase of movements from the point of view of rail tracks (Anon., 1977).

The extent to which a railway line is affected by ground movements is related to

1. Types of traffic involved.
2. Speed limits.
3. Types and construction of track.
4. Preventive and remedial works.
5. Nature and magnitude of ground movements.

Damage Criteria—Damage to railroads may be classified in terms of interruption of use or failure. Interruptions of use would refer to excessive track gradients or bumps on the track resulting in rider discomfort, reduction of permissible speeds, increased propulsion and braking forces, reduced payloads and train lengths, or impairment of orderly traction and shunting operations. Quantitative limits for this level of damage are difficult to assign due to the many variables involved. A railroad track may be deemed to have failed if the deformations are of such magnitude that it is incapable of sustaining traffic due to risk of derailment. Available data are given in Table 10.6.14.

Pipelines. Generally, pipelines are laid some distance below the ground surface. Buried pipelines are known to be more susceptible to damage from ground movement than those laid above ground. These move in response to ground movements due to friction and soil pressure between the pipe material and the

Table 10.6.11. Damage Criteria for Buildings

Building Category	Damage Seventy Level	Movement Type	Movement Limits	Country	Reference	Suggested Value
Brick and masonry/ brick bearing walls/ low-rise structures	Architectural	Angular distortion	1.0–2.0 $\times 10^{-3}$	Germany	Niemczyk (1949)	1.0 $\times 10^{-3}$
			0.5–1.0 $\times 10^{-3}$		Meyerhoff (1953)	
			1.0–2.0 $\times 10^{-3}$		Skempton and McDonald (1956)	
			1.0 $\times 10^{-3}$	USSR	Polshin and Tokar (1957)	
			1.0–2.0 $\times 10^{-3}$	US	Sowers (1962)	
			1.0 $\times 10^{-3}$	US	O'Rourke (1976)	
			1.0 $\times 10^{-3}$	UK	Attewell (1977)	
			1.2 $\times 10^{-3}$	US	Boscardin (1980)	
Brick and masonry/ brick bearing walls/ low-rise structures	Architectural	Horizontal strain	0.6 $\times 10^{-3}$	Germany	Niemcyzk (1949)	0.5 $\times 10^{-3}$
			0.4 $\times 10^{-3}$	UK	Beevers and Wardell (1954)	
			0.5 $\times 10^{-3}$	USSR	Polshin and Tokar (1957)	
			0.8 $\times 10^{-3}$	UK	Priest and Orchard (1957)	
			0.5 $\times 10^{-3}$	Japan	Goto (1968)	
			0.4–0.5 $\times 10^{-3}$	India	Singh and Gupta (1968)	
			0.25 $\times 10^{-3}$	UK	Littlejohn (1975)	
			0.5–1.0 $\times 10^{-3}$	UK	National Coal Board (Anon., 1975a)	
			< 0.75 $\times 10^{-3}$	US	O'Rourke (1976)	
			0.5–1.0 $\times 10^{-3}$	UK	Attewell (1977)	
			1.0–1.5 $\times 10^{-3}$	US	Cording et al. (1976)	
			0.5 $\times 10^{-3}$	US	Yokel (1978)	
			0.5 $\times 10^{-3}$	US	Boscardin (1980)	
Brick and masonry/ brick bearing walls/ low-rise structures	Architectural	Deflection ratio	0.3–0.7 $\times 10^{-3}$	USSR	Polshin and Tokar (1957)	0.3 $\times 10^{-3}$
			1.0 $\times 10^{-3}$	US	Grant (1974)	
			0.4 $\times 10^{-3}$	UK	Burland and Wroth (1975)	
Brick and masonry/ brick bearing walls/ low-rise structures	Functional	Angular distortion	3.5 $\times 10^{-3}$	US	Meyerhoff/Terzaghi (1953)	2.5–3.0 $\times 10^{-3}$
			3.3 $\times 10^{-3}$	US	Skempton and McDonald (1956)	
			4.0–6.0 $\times 10^{-3}$	USSR	VNIMI (Anon., 1958)	
			2.0 $\times 10^{-3}$	US	Bjerrum (1963)	
			3.3 $\times 10^{-3}$	US	Grant (1974)	
			3.3–5.0 $\times 10^{-3}$	Poland	Starzewski (1974)	
			3.0 $\times 10^{-3}$		Ulrich (1974)	
			2.0–3.3 $\times 10^{-3}$	Sweden	Broms and Fredrikson (1976)	
			2.7 $\times 10^{-3}$	UK	Thorburn and Reid (1977)	
			2.5 $\times 10^{-3}$	Poland	Adamek and Jeran (1981)	
			3.0–6.0 $\times 10^{-3}$	Japan	Nishida et al. (1982)	
Brick and masonry/ brick bearing walls/ low-rise structures	Functional	Horizontal strain	2.0–4.0 $\times 10^{-3}$	USSR	VNIMI (Anon., 1958)	1.5–2.0 $\times 10^{-3}$
			1.0 $\times 10^{-3}$		Ulrich (1974)	
			2.5–3.5 $\times 10^{-3}$	US	Cording et al. (1976)	
			1.5 $\times 10^{-3}$	Poland	Adamek and Jeran (1982)	
Brick and masonry/ brick bearing walls/ low-rise structures	Functional	Deflection ratio	0.14–0.22 $\times 10^{-3}$		Rigby and Dekoma (1952)	0.5 $\times 10^{-3}$
			0.25 $\times 10^{-3}$		Wood (1952)	
			0.6 $\times 10^{-3}$	US	Horne and Lambe (1964)	
Brick and masonry/ brick bearing walls/ low-rise structures	Functional	Radius of curvature	1.9–12.4 mi (3–20 km)	USSR	VNIMI (Anon,, 1958)	12 mi (20 km)
			12.4 mi (20 km)		Ulrich (1974)	
			12.4 mi (20 km)	Poland	Adamek and Jeran (1982)	
			8.0 mi (13 km)	Japan	Nishida et al. (1982)	

Table 10.6.11. Damage Criteria for Buildings—cont.

Building Category	Damage Severity Level	Movement Type	Movement Limits	Country	Reference	Suggested Value
Brick and masonry/ brick bearing walls/ low-rise structures	Structural	Angular distortion	7.0–8.0 $\times 10^{-3}$	US	O'Rourke et al. (1977)	7.0 $\times 10^{-3}$
Brick and masonry/ brick bearing walls/ low-rise structures	Structural	Horizontal strain	3.5 $\times 10^{-3}$ 2.75 $\times 10^{-3}$	UK US	National Coal Board (Anon., 1975a) Boscardin (1960)	3.0 $\times 10^{-3}$
Steel and reinforced concrete	Architectural	Angular distortion	1.0–2.0 $\times 10^{-3}$ 2.0 $\times 10^{-3}$ 2.0–2.5 $\times 10^{-3}$ 2.2 $\times 10^{-3}$ 1.3 $\times 10^{-3}$ 2.0 $\times 10^{-3}$	US USSR US US UK	Skempton and McDonald (1956) Polshin and Tokar (1957) Sowers (1962) Breth and Chambrosse (1975) O'Rourke (1976) Attewell (1977)	1.3 $\times 10^{-3}$
Steel and reinforced concrete	Functional	Angular distortion	2.5–3.3 $\times 10^{-3}$ 3.3–6.6 $\times 10^{-3}$ 3.3–5.0 $\times 10^{-3}$	 US Poland	Thomas (1953) Skempton and McDonald (1956) Starzewski (1974)	3.3 $\times 10^{-3}$
Timber frame	Architectural	Angular distortion	2.0 $\times 10^{-3}$	US	Mahar and Marino (1981)	1.5 $\times 10^{-3}$
Timber frame	Architectural	Horizontal strain	1.0 $\times 10^{-3}$	Japan	Goto (1968)	1.0 $\times 10^{-3}$
Timber frame	Functional	Angular distortion	5.0–10.0 $\times 10^{-3}$ 3.3–5.0 $\times 10^{-3}$	Poland Sweden	Starzewski (1974) Broms and Fredriksson (1976)	3.3–5.0 $\times 10^{-3}$

Legend:

Architectural: Small scale cracking of plaster and sticking of doors and windows.
Functional: Instability of some structural elements, jammed doors and windows, broken window panes, building services restricted.
Structural: Impairment of primary structural members, possibility of collapse of members, complete or large-scale rebuilding necessary, may be unsafe for habitation.

No data available on rigid, massive structures/central core design.

Table 10.6.12. Damage Criteria for Highway Bridges

Damage Severity Level	Allowable Movement: Type of Movement	Allowable Magnitude	Source	Suggested
Architectural	Angular distortion	1.0×10^{-3}	Moulton et al. (1982)	1.0×10^{-3}
Functional	Angular distortion	$4.0–5.0 \times 10^{-3}$	Moulton et al. (1982)	3.0×10^{-3}
Architectural	Differential settlement	1.0 in. (25 mm) 2.0 in. (50 mm) 1.0 in. (25 mm)	Grover (1978) Bozozuk (1978) DiMillio (1982)	1.0 in. (25 mm)
Functional	Differential settlement	2.0–4.0 in. (25–50 mm) 2.5 in. (65 mm) 4.0 in. (100 mm) 2.0–3.0 in. (50–75 mm)	Moulton et al. (1982) Walkinshaw (1978) Bozozuk (1978) Grover (1978)	2.0 in. (50 mm)
Architectural	Horizontal movement	1.0 in. (25 mm)	Bozozuk (1978)	1.0 in. (25 mm)

Damage Level Legend:

Architectural: Minor cracking, opening and closing of construction joints in abutments, cracking and spalling of concrete decks.
Functional: Superstructure distress, horizontal displacement, bearing damage or damage to abutments, warping or tilt of bridge decks, bumps at compressed and open expansion joints.
Structural: Instability of primary structural members, possibility of collapse.

Table 10.6.13. Damage Criteria for Roads

Damage Severity Level	Movement Limits: Type of Movement	Movement Limits: Tolerable Range	Source	Suggested Value
Architectural (Minor pavement cracking)	Horizontal strain	$1.2–3.8 \times 10^{-3}$	Instn. Civil Engrs. (Anon., 1977)	1.0×10^{-3}
Architectural (minor pavement cracking)	Slope	$5.0–10.0 \times 10^{-3}$		5.0×10^{-3}
Functional (undulations and water accumulation)	Slope	5.0×10^{-3}	Kratzsch (1983)	5.0×10^{-3}
Structural (adverse effects on driving dynamics—large-scale cracking affecting base/subgrade; severe local gradients; potholes)	Slope	10×10^{-3} 10×10^{-3} $10.0–20.0 \times 10^{-3}$	Maize et al. (1941) Sowers (1962) Kratzsch (1983)	10×10^{-3}

Table 10.6.14. Damage Criteria for Railroads

Damage Severity Level	Movement Limits: Type of Movement	Movement Limits: Tolerable Range	Source	Suggested Value
Risk of derailment and rider discomfort	Horizontal strain	2.0×10^{-3} 3.0×10^{-3}	Kratzsch (1983) Saxena and Singh (1980)	2.0×10^{-3}
Risk of derailment and rider discomfort	Slope	12.5×10^{-3}* 10.0×10^{-3} *2.5×10^{-3} (for railway stations)	Kratzsch (1983) Saxena and Singh (1980) Kratzsch (1983)	10.0×10^{-3} or maximum permissible track gradient specified by design

ground. If the magnitude of the ground movements are such that either the pipeline or its joints (or couplings) are unable to accommodate the deflection or strain which is developed, they may fracture or fail.

Damage may be caused either by excessive strain along pipe lengths or excessive distortion at the joints or both. Three basic modes of failure may be identified (O'Rourke and Trautmann, 1982):

1. Strain in pipe material leading to rupture or intolerable deformation.
2. Rotation of the joints leading to leakage or loss of connectivity.
3. Axial slip at the joints leading to leakage or disengagement of adjacent pipe lengths.

The first two failure modes may be caused by differential settlement, and the first and third by lateral displacement.

The largest percentage of pipe failures are caused by compressional forces causing excessive telescoping at joints. Tensile failures are the next major mode, whereas failures due to ending or shearing rarely occur (Tilton, 1966). British experience recognizes the following types of fractures: beam, pull, shear, thrust, and leverage (Anon., 1975a).

The extent to which a pipeline can absorb ground deformations is said to be dependent upon the stress/strain behavior of the pipe material, the rotation and pull-out capacity of the couplings, connections to other structural elements, corrosion resistance of pipe and joints, and other general factors such as state of repair and installation technique (O'Rourke and Trautmann, 1980). The location of the joints with respect to the subsidence profile and the degree of rigidity of the pipeline will also significantly affect the nature and extent of damage. At times the joints constitute the weakest link in a pipeline system and are usually affected by ground movements long before the pipe length.

Couplings in water and gas distribution mains increase pipeline flexibility. Flexible couplings are generally equipped with a gasket that is compressed to prevent leakage. These joints are capable of sustaining rotations that vary from 1 to 7º. Mechanical joints can tolerate about 2 in. (50 mm) of horizontal slippage before leakage. When both horizontal strains and differential settlements must be sustained, pipeline joints can be designed to rotate and telescope. Welded pipelines are most susceptible to damage by compression because ground movements cause local wrinkling or buckling of the pipe wall. Once local wrinkling has initiated, all subsequent deformations will tend to concentrate at the location of the wrinkle. Local wrinkling may occur at compressive strains on the order of 0.4 to 0.6% (Bouwkamp and Stephen, 1973). Butt-welded steel pipelines are most capable of sustaining the differential soil movements caused by mining subsidence, but these must be high quality welds, free of significant corrosion.

Damage Criteria—Ground movement limits may be prescribed for two basic damage levels:

1. Interruption of use.
2. Failure or loss of use.

Pipelines are usually laid under fairly stringent grade requirements. A change of ground slope may affect power costs in the form of greater pumping requirements or may cause inconvenience to the users. These constitute impaired use of the application. Ground movement limits for this level are difficult to assign because of its site-specific nature. However, utility companies sometimes provide standards for a given type of application. If these are known, appropriate criteria can be developed for a given area. Gas line leaks pose a special hazard, because of the

Table 10.6.15. Damage Criteria for Pipelines

Type of Pipe	Damage Severity Level	Movement Limits: Type of Movement	Movement Limits: Range	Source	Suggested Value
Cast iron pipe with lead-caulked joints	Failure of pipes or couplings	Angular distortion	4.0×10^{-3}	O'Rourke and Trautman (1982)	4.0×10^{-3}
Cast iron pipe with lead-caulked joints	Failure of pipes or couplings	Horizontal strain	$0.5–2.0 \times 10^{-3}$	Grard (1969)	1.0×10^{-3}

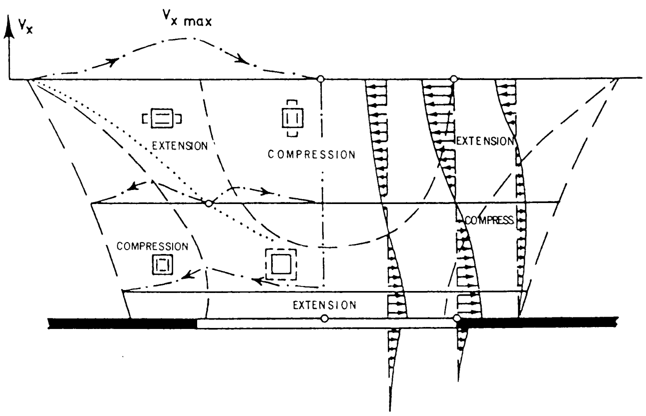


Fig. 10.6.13. Horizontal deformation of a continuum over workings in a bedded deposit (Kratzsch, 1983).

toxic and/or explosive effects of the constituents. Available data for failure are shown in Table 10.6.15.

10.6.4.3 Shafts and Subsurface Structures

Shafts: Mine subsidence damage affects not only surface structures but also those located within the rock mass. With shallow workings, it is possible to leave a protective shaft pillar, which is not mined. However, in deep mines, the size of pillars becomes excessive, and large amounts of ore tend to become sterilized within them. Hence it is impractical not to mine within the safety zone.

Fig. 10.6.13 depicts movements of the strata above the mine openings as if it were a continuum and not bedded. If the entire mine pillar were mined, a shaft through these strata would be subjected to compression near the surface and tension near the mine floor. If the pillar were partially retained, the type and magnitude of the ground strains experienced by the shaft would depend upon location of the shaft within the pillar, and would vary along the length of the shaft. Generally, the horizontal movements of the shaft tend to be towards the center of the workings near the surface and away from them near the floor and immediate roof. The net result is that the shaft tends to lean towards the gob as the face approaches the shaft, and then reverses itself as the face goes past. Finally, the shaft may become vertical again with time. During this period, hoisting equipment experiences increased wear in the inclined shaft. Rupture of the shaft, however, does not normally occur, since assuming the shaft experiences a compressive strain of 2×10^{-3}, the walls of a 20-ft (6-m)-diameter shaft close by only ½ in. (12 mm). The horizontal compression is largely absorbed by the pressure arch in the rock and compressible material surrounding the shaft. Although bending of the shaft is not large, bending stresses need to be considered in the design of rigid linings. Fractures in the shaft lining invariably accompany shear movements in severely bent strata.

With appropriate mining techniques, it is possible to use the shaft for hoisting as well as maintain integrity through water-bearing zones. Principles to follow include (Kratzsch, 1983):

1. *Compensating* axial tension and compression stresses along the shaft with the use of two or more faces.
2. *Working symmetrically* around the shaft, minimizing shaft tilt and shear.
3. *Minimizing subsidence* by backfilling or leaving a core pillar.
4. *Designing* the shaft lining to accommodate movements.
5. *Observation* of axial deformation, tilt, and bending regularly during the operation.

Designing flexible shaft linings implies

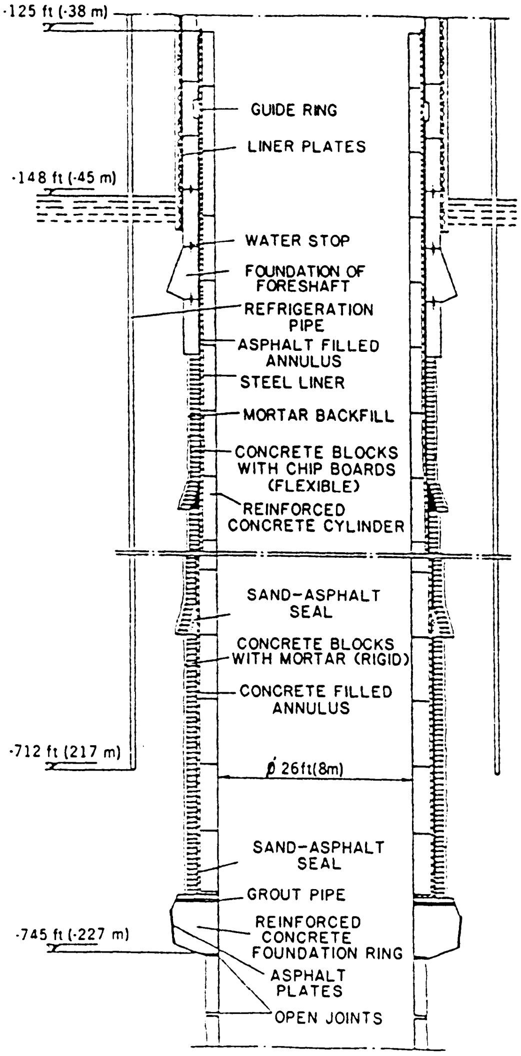
1. Introducing a *sliding joint,* which reduces friction and seals the shaft through water-bearing zones.
2. Providing an *expansion joint* in the lining at the mining horizon, which absorbs axial compression.
3. Placing *compressible courses,* 100 to 150 ft (30 to 50 m) apart, so that the shaft can yield or bend, but the shaft loses watertightness.
4. Using a *weak mortar,* which serves as a compressible course, and facilitates masonry repair.
5. Furnishing an *annular cushion* of fly ash at the mining horizon; this absorbs lateral expansion of the seam, and vertical compression.
6. Constructing an *outer ring of concrete,* which seals fractured strata and prevents shear movements from intersecting faults.
7. *Monitoring* movements so as to recognize danger in a timely manner.

A cross section of a shaft through water-bearing strata designed to absorb ground movements is shown in Fig. 10.6.14.

Damage Criteria—No specific values for damage can be provided, since this is dependent on the design of the lining, but the horizontal and vertical stresses in the shaft lining can be computed and checked against the strength values of the lining materials.

Subsurface Structures: The degree of damage experienced by subsurface structures depends upon their location with respect to the mine workings. These may be sited at various horizons:

1. *Below mine level:* the extent of damage incurred should be small, even if these are within the zone affected by movements.
2. *At mine level:* adequately designed and supported chambers should remain open, but the passageways leading to these may be difficult to maintain unobstructed, unless they are located in the shaft pillar. Openings sited in the shaft pillar would be subjected to minor movements.



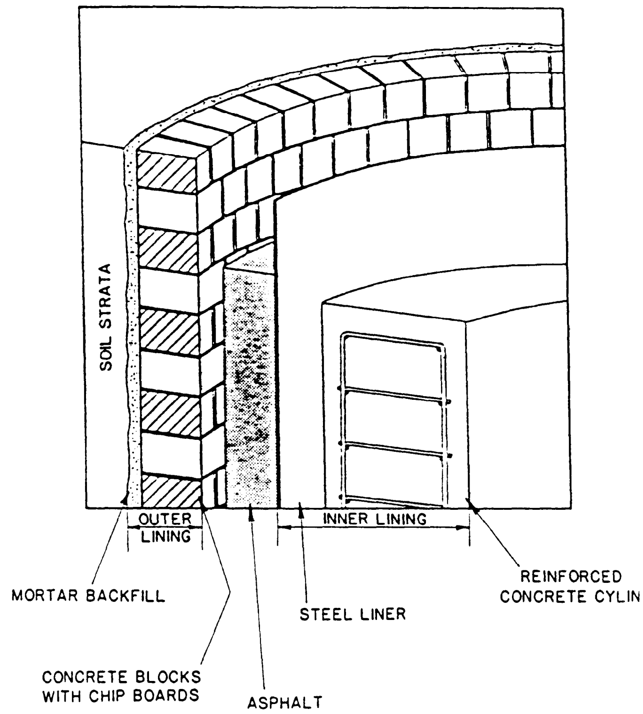


Fig. 10.6.14. Cross section through a shaft lining through water-bearing strata, designed to tolerate subsidence (Stoss and Braun, 1983).

3. *Above mine level, in the rock strata:* openings situated in this region may be subjected to large movements, and occasional shock loads or sudden falls; thus these would be costly to design and keep open. Structures located at a height above 60 times the seam thickness, and surrounded by thick plastic or viscoelastic strata (e.g., shales) may not experience sudden movements and could survive the subsidence effects with slight damage.

4. *Above mine level, in unconsolidated materials:* these openings are close to the surface and in deep mines, if properly designed and lined, may endure the ground movements due to the flow of the surrounding soils; some repairs would probably be required.

There are no data on structures within mine overburdens; validation of the above scenarios is needed.

10.6.4.4. Renewable Resource Lands

Renewable resource lands include aquifers and areas for the recharge of aquifers and other underground waters, areas for agricultural and silvicultural production of food and fiber, and grazing lands (Anon., 1979). In addition, significant springs, perennial streams, and other surface waters that support aquatic life or supply water to any public water system may be classed as renewable resources.

Agricultural Lands: Adverse effects of subsidence movements on agricultural lands include

1. *Formation of surface fissures* in the zones subjected to tensile stresses, which form pathways to drain the water away from the topsoil, thus being detrimental to plant growth. Flow of water in the cracks also causes erosion, thereby widening them (Schumann and Poland, 1970).

2. *Alteration in the ground slope.* Steepening increases flow velocity, which in turn enhances surface runoff and erodibility, whereas gradient reduction could lead to waterlogging of the soil. Crop yields may tend to decrease in either case.

3. *Disruption of surface drainage patterns* resulting from changes in ground elevation or slope. Lowering the topographic barriers to flow increases the potential for flooding (Moore and Nawrocki, 1980). In flat terrain, ponding may occur, especially if the water table is shallow and underlain by an impermeable stratum (Wohlrab, 1969). Changes in slope disturb the existing hydraulic regime.

4. *Modification of the subsurface hydrology* due to generally downward migration of the groundwater through cracks, and consequent decrease in soil fertility.

5. *Deterioration of groundwater quality* due to contact with sulfides and other minerals (Temple and Koehler, 1954; Lovell,

Table 10.6.16. Damage Criteria for Prime Farmlands

Damage Severity Level	Movement Limits		Source	Suggested Value
	Type of Movement	Tolerable Range		
Moderately reduced productivity	Horizontal strain	$2.0–3.0 \times 10^{-3}$	Inferred	$2.0–3.0 \times 10^{-3}$
Severely reduced productivity	Horizontal strain	5.0×10^{-3} 5.0×10^{-3} $<10.0 \times 10^{-3}$	Orchard (1969) Voight and Pariseau (1970) Jachens and Holzer (1982)	5.0×10^{-3}
Moderately reduced productivity	Slope	$2.0–3.0 \times 10^{-3}$	Inferred	$2.0–3.0 \times 10^{-3}$
Severely reduced productivity	Slope	6.0×10^{-3} $6.0–8.0 \times 10^{-3}$ $5.0–8.0 \times 10^{-3}$	Pierce et al. (1983) Fehrenbacher et al. (1978) U.S. Dept. Agriculture (Anon., 1951)	6.0×10^{-3}

1973; Kim et al., 1982) or changes in sediment loads (Moore and Nawrocki, 1980).

6. *Occurrence of subsidence pits* (sinkholes), which may result in upsetting the drainage system by either accumulation or loss of water and, in extreme cases, damage to equipment and life.

The effect of mine subsidence on farming depends on the type of crops, soil character, hydrology, topography, and other environmental factors. Essentially, subsidence damage caused to agricultural lands is characterized by the loss of use or reduced productivity. Increasing ground movements generally cause a decline in productivity, but the precise amount of decline is site specific and, thus, difficult to quantify in general terms.

Agricultural lands may be classified in various ways, but one that is commonly used in relation to mine subsidence is dividing such property into prime and nonprime farmlands. Prime farmland has "the best combination of physical and chemical characteristics for producing food, feed, forage, fiber, and oilseed crops. It is the land that gives the highest agricultural yield with minimum input when managed according to modern farming methods" (Guernsey et al., 1979). Generally, soils constituting prime farmland have slopes of less than 8 to 10×10^{-2} (Fehrenbacher et al., 1978; Guernsey et al., 1979). All lands not classed as prime farmland fall into the nonprime category.

Damage Criteria—Essentially, subsidence damage caused to agricultural lands is characterized by the loss of use or reduced productivity. Increasing ground movements generally cause a decline in productivity, but the precise amount is site-specific. In some instances, the extent of damage severely jeopardizes the capability of the land to economically produce crops, labeled "severely reduced productivity." Another level of damage, termed "moderately reduced productivity," accounts for less severe damage, such as decreased crop production potential because of loss of soil fertility. Ranges for these are presented in Table 10.6.16.

Forests and Grazing Lands: The types of damage caused by mine subsidence to forests and grazing lands is similar in nature to that for agricultural lands, but they are generally less susceptible to the damaging effects of subsidence movements than farmlands.

Damage Criteria—Table 10.6.17 depicts the current information. Small changes in slope are not likely to cause appreciable damage to forests and grazing lands.

10.6.4.5 Hydrologic Effects

Surface Water Bodies: The formation of open cracks, fissures, or pits at the bottom of surface bodies of water (streams, lakes, ponds) may lead to partial or complete loss of water by its drainage to lower strata or mine workings. Depletion of such water resources seriously impact public water systems and aquatic life forms. In extreme cases, catastrophic inundations of underground mine workings may occur.

Damage Criteria—Available data are given in Table 10.6.18; further guidance on the minimum depth required to maintain levels below the recommended limits is presented in Table 10.6.19.

Groundwater Aquifers: Mine subsidence affects groundwater aquifers in several ways:

1. *Lowering of groundwater* levels or hydrostatic head due to the formation of breaks and fractures in the strata above underground mine workings may decrease the groundwater supply.

2. *Changes in streamflow rates;* increases may occur due to faster movement through fractured strata, water accumulations in subsidence troughs, or reduced evapotranspiration because of a drop in the water table, whereas decreases are caused by water diversion.

3. *Alteration in water quality* produced by chemical interaction with the minerals or adjoining strata.

The prevailing hydrogeologic environment is significantly affected by subsidence-induced fissuring. Cessation of mining could bring about reconsolidation of the strata, but this may or may not result in the rebound of groundwater to premining levels. Water table depression is generally localized near the mine workings (Anon., 1981; Hill and Price, 1983), but the effects on the hydrologic regime could be highly variable and complex. Local structural features (e.g., faults, synclines, or anticlines) and the nature, continuity, and dip of the beds have a profound influence on subsidence damage. The significant presence of impermeable clay layers in the overburden inhibits groundwater drainage (Singh and Kendorski, 1981). The adverse impacts of water table lowering can be significant, especially in arid regions (Piper, 1933; Ward and Wilmoth, 1968; Subitzky, 1976; Rauch, 1978), and can reduce plant growth (Anon., 1981). Subsidence-induced streamflow rate variations have been noted (Growitz, 1978; Sgambat et al., 1980) as has the deterioration of water quality (Sgambat et al., 1980; Anon., 1981; Stoner, 1983), which is detrimental to water usability.

Damage Criteria—There is significant likelihood of damage to a water-bearing stratum if it is located within the caving or bed separation zones of an underground excavation. The strain limit suggested in Table 10.6.20 may be conservative since these strata are subjected to horizontal constraints, increasing their resistance to fracturing.

Example 10.6.2. Mine S is planning to develop several longwall panels and needs to obtain a permit, and hence to predict

Table 10.6.17. Damage Criteria for Forests and Grazing Lands

Surface Features	Damage Severity Level	Movement Limits: Type of Movement	Movement Limits: Tolerable Range	Source	Suggested Value
Pasture, woodland, range, or wildlife food and cover	Severe	Horizontal strain	$5.0–10.0 \times 10^{-3}$		5.0×10^{-3}
Wetlands	Severe	Horizontal strain	5.0×10^{-3}	Inferred	5.0×10^{-3}
Pasture, woodland, range, or wildlife food and cover	Severe	Slope	$450–660 \times 10^{-3}$ $250–350 \times 10^{-3}$	U.S. Dept. Agriculture (Anon., 1951; 1973) (Griffin, 1977)	300×10^{-3}
Wetlands	Severe	Slope	$30–80 \times 10^{-3}$	Griffin (1977)	30×10^{-3}

Table 10.6.18. Damage Criteria for Surface Water Bodies

Surface Features	Damage Severity Level	Movement Limits: Type of Movement	Movement Limits: Range	Source	Suggested Value
Natural: Sea or tidal waters, lakes, ponds, marshes, rivers, streams *Artificial:* Canals, impounded waters	Severe (implies partial or complete loss of water)	Horizontal strain	$5.0–10.0 \times 10^{-3}$	Wardell (1976)	5.0×10^{-3}

Suggested Vertical Distance Between Mine and Water Body = > 60 × (Mining Height)

Table 10.6.19. Minimum Cover For Total Extraction Under Water Bodies (with potential for causing catastrophic damage)

Seam Thickness (t): Feet	Seam Thickness (t): Meters	Minimum Cover Thickness (D): In terms of t	Minimum Cover Thickness (D): Feet	Minimum Cover Thickness (D): Meters
3	0.9	117 t	351	107.0
4	1.2	95 t	380	115.8
5	1.5	80 t	400	121.9
6	1.8	71 t	426	129.8
7	2.1	63 t	441	134.4
7.5	2.3	60 t	450	137.2
>7.5	>2.3	60 t	450	137.2

the surface damage expected due to subsidence. The mining height is 7.6 ft (2.3 m) and the depth of cover varies between 478 and 1022 ft (146 and 312 m). The mining company assumes that the angle of draw is 15° (probably low). Panel widths are expected to range between 500 and 700 ft (152 and 213 m).

Surface structures present in the permit area include buildings (e.g., a school), roads, and pipelines. Surface land use within the plan area is primarily pasture and woodland. There are no major aquifers or streams that serve as a significant water source for public water supply. There is one perennial spring fed by a perched aquifer, which is presently used for watering stock. This spring can be expected to have low flow rates during times of low precipitation or go completely dry during times of drought. The only persistent regional aquifer in the plan area is a sandstone, approximately 50 ft (15 m) below the valley level.

Solution: The extent of subsidence $S(x)$, slope $G(x)$, and horizontal strain $e(x)$ at the locations of the structures, and the expected worst-case sites for the renewable resource lands, may be determined by any of the methods discussed in Example 10.6.1 or by using one of the prevalent computer programs (see Table 10.6.7). These values are then compared with the ranges of slope and strains for various damage levels given in Tables 10.6.11 through 10.6.18 and 10.6.20. The expected damage levels can thus be obtained, as shown in Table 10.6.21. This information may be then provided in the permit application.

10.6.4.6 Nonmining Damage

When the time span for surface damage is prolonged, accounting for subsidence generated by nonmining causes is important and needs attention. In mining areas, local inhabitants have a tendency to blame mining activity for any damage that may be observed in local structures or lands. Such claims may not be justified in many instances. Types of nonmining damage that have been noted include the following.

Soil Settlement: Differential settlement of buildings can occur on fill material, especially when the fill is improperly compacted or part of the structure is on the fill and the rest on virgin ground. Such effects are also noted if some of the foundations are old and have already settled, whereas the rest of the foundation is relatively new. Inadequate bearing capacity of foundation soil and consolidation of soft clayey soil could also result in uneven settlement. The extent of damage caused depends upon the amount of settlement that develops and the orientation of the structure with respect to the settlement pattern.

Shrinking/Swelling of Soils: When soils shrink or swell due to the influence of water, damage may occur in surface structures that appears to be similar to subsidence damage. The change in moisture content could occur because of seasonal precipitation or drying, from leaks in water lines or sewers, or by trees and other vegetation in the vicinity. Roots tend to drain the clay; this is especially noticeable under paved areas since these areas are otherwise protected from precipitation and drying effects.

Table 10.6.20. Damage Criteria for Groundwater Aquifers

Features	Damage Severity Level	Movement Limits: Type of Movement	Movement Limits: Tolerable Range	Source	Suggested Value
Aquifers	Severe (implies partial or complete loss of water)	Horizontal strain	5.0×10^{-3}	Inferred	5.0×10^{-3}

Suggested Vertical Distance Between Mine and Aquifer = > 60 $\times$ (Mining Height)

Table 10.6.21. Damage Level Assessment for Ex. 10.6.2

Structure/ Resource	Overburden Depth m (ft)	Expected Strains ($\times 10^{-3}$) (Knothe)	Expected Strains ($\times 10^{-3}$) (NCB)	Expected Slopes ($\times 10^{-3}$) (NCB)	Strain Limit ($\times 10^{-3}$)	Slope Limit ($\times 10^{-3}$)	Expected Damage Level
Buildings	152–174 (500–570)	16.5–18.8	5.5–6.3	20.5–23.1	3.0	NA	Major structural damage
Roads	152–174 (500–570)	16.5–18.8	5.5–6.3	20.2–23.1	1.0	5.0–10.0	Surface cracking and considerable grade changes
Pipeline	152–274 (500–900)	10.4–18.8	3.5–6.3	12.9–20.2	1.0	NA	Widespread failure of pipe or couplings
Surface water	152–174 (500–570)	16.5–18.8	5.5–6.3	20.2–23.1	5.0	NA	Fissuring beneath stream bed, significant water loss
Aquifer	137 (450)	21.0	7.0	25.7	5.0	NA	Partial or complete dewatering
Forests/ grazing lands	146–311 (480–1,020)	9.2–19.6	3.0–6.6	11.3–24.0	5.0	300.0	Localized surface fissuring, little drainage effects

NA = Not available

Soil shrinkage generally induces vertical strains in the structure rather than the horizontal strains that commonly damage buildings due to subsidence. The building itself tends to shelter the inner portions so the outer walls are generally affected. These walls depict a tendency to pivot outward about the foundation. Swelling of the soil reverses the movement partially, but the cracks remain.

In cold areas, freezing and thawing effects may also cause expansion and contraction of poorly drained fine-grained soils in a manner similar to moisture-sensitive soils (DuMontelle et al., 1981).

Groundwater and Precipitation: Deviations in groundwater flows or significant changes in precipitation could cause damage to structures. Such changes may be a result of mining or due to other reasons. The effects of water drainage into mined excavations, leading to surface subsidence, must be recognized as mining-related and treated accordingly. However, alterations in groundwater flow may also be caused by building activity in the area, installation of wells for farm irrigation, or major excavations some distance away which affect an aquifer. These are entirely unrelated to the mining activities.

Inadequate Drainage or Waterway Regulation: Waterlogging occurs near buildings if proper drainage is not provided. This can induce seepage into building basements and general deterioration of structures, especially if these are made of wood or other materials susceptible to damage by water. Sometimes this type of damage is blamed on mining subsidence.

Faulty Construction, Inferior Materials, or Inferior Subsoil: Poor quality building materials and construction may increase the likelihood of damage to structures. Buildings on poor soil or improperly compacted fill may also suffer disastrous effects. Roof spread is a common form of damage that may occur due to an inadequate number of cross-ties or due to timber decay in older buildings. This permits the external walls to push outward at the top and induce cracking in them. This type of damage may be distinguished from subsidence damage when observed movements are in a direction opposite to that anticipated from mining.

Chemical Attack: Masonry or stone structures show cracking when embedded iron or steel members (such as dowels, hooks, or frames) begin to rust and corrode. Glass panes in metal frames may crack because of the pressure exerted on them by rust. This probably is the case when the window can still be opened, but the glass is cracked.

Dissolved sodium, calcium, or magnesium sulfates in surface water tend to react with Portland cement or lime causing the mortar to expand and deteriorate into a powder. The moisture initially affects the shrinkage cracks, but later induces further splitting. Chimneys with sulfurous gases make easy targets for such damage. If sulfur-containing shales (which are common in coal mining areas) are used for fill or as a foundation for buildings, the concrete floor is liable to show signs of distress from sulfate attack (Anon., 1975a).

Thermal Effects: The differential thermal expansion characteristics between tiles and the concrete surface to which they are bonded may cause them to separate or become loose (such as on the floor). Joints between different materials may also be affected (e.g., plaster board joints, wood and concrete or brick interfaces, concrete/brick and steel contacts). Pavements may crack or buckle because of thermal effects. Pipelines and railroad tracks show marked effects.

Natural Wear and Tear: All structures undergo wear and tear during normal usage, and sometimes it is difficult to isolate damage due to these causes, especially if the maintenance is not regular.

Vibrations: Industrial plants in the vicinity or heavy traffic on nearby roads may generate vibrations, which could induce cracking or cause settling of structures. This may be especially true if blasting is done for construction or seismic investigations.

Minor Earthquakes: Major earthquakes are readily noted and any damage resulting therefrom is clearly evident. In earthquake-prone areas where numerous minor earthquakes occur, however, this may not be the case. The cumulative effect of these can be seen as cracks in walls or other impairment to structures, which may be confused with subsidence damage.

Minor Landslides or Creep: As with earthquakes, the effects of major landslides are noticed. However, a number of small slides or gradual creep, especially on slopes, may induce deterioration that may not be readily visible for some time.

Tectonic Movements: Structural damage can result from long-term movements of the earth's crust. Such gradual motion has lifted pipelines in California and caused measurable vertical and horizontal displacements in Upper Silesia, Poland (Brauner, 1973).

Fluid Withdrawal: As mentioned in 10.6.1, this is considered a nonmining source of subsidence in this chapter. Notable instances include the withdrawal of water for industrial and public use in the Houston-Galveston area in Texas, for agricultural purposes in the San Joaquin Valley in California, land reclamation around New Orleans in Louisiana, production of oil from the Wilmington field near Long Beach, California, and geothermal fluid extraction near the geysers in California.

Nonmining Cavities: In situ coal gasification near Rawlins, WY, tunneling for infrastructure in several urban areas (e.g., Washington, DC; San Francisco, CA; Chicago, IL; Milwaukee, WI), and natural cavity formation in the karst areas of Florida and Tennessee have also resulted in subsidence.

These are, of course, well-publicized examples, but smaller amounts of subsidence continue to occur throughout the country as a result of these mechanisms. Coal mine operators and regulators need to be particularly aware of water and oil/natural gas withdrawal in the vicinity of their mines so that the correct cause of subsidence can be identified.

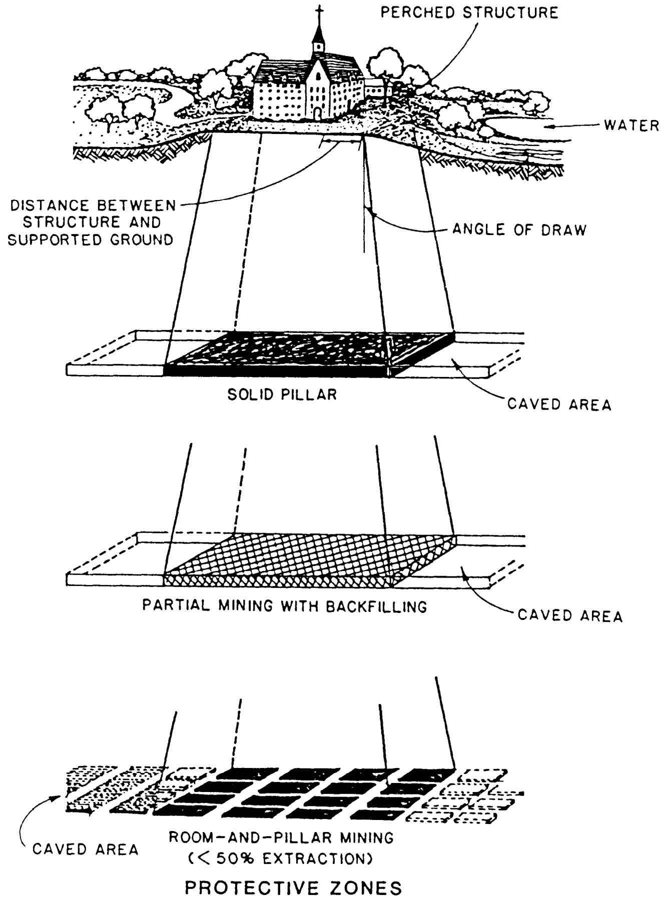


Fig. 10.6.15. Protective zones for surface structures (Singh, 1985).

10.6.5 CONTROL AND PREVENTION OF DAMAGE

There are four types of measures that may be adopted to control subsidence damage (Singh, 1985):

1. Alteration in mining techniques.
2. Postmining stabilization.
3. Architectural and structural design.
4. Comprehensive planning.

Each of these encompasses several methods.

10.6.5.1 Alteration in Mining Techniques

Partial Mining: This may be accomplished in a number of ways:

1. Leaving *protective zones,* which is the most commonly used procedure (Fig. 10.6.15). The zone may entail:

a. Leaving the *entire pillar unmined* beneath structures, such as factories, railroads, major highways, and bodies of water.

b. *Partially extracting the pillar* and backfilling

c. *Room and pillar mining,* with up to 50% extraction; a practice recommended in some states (e.g., Pennsylvania) by regulation. This method does not account for pillars deteriorating with time, especially if the mine is flooded.

It should be borne in mind that any structure supported by a protective zone is liable to become perched at a higher level than the surrounding ground, after it subsides. This may not affect the railroad or highway being protected, but could disturb the utilities to a building. An island may form if the water table is high.

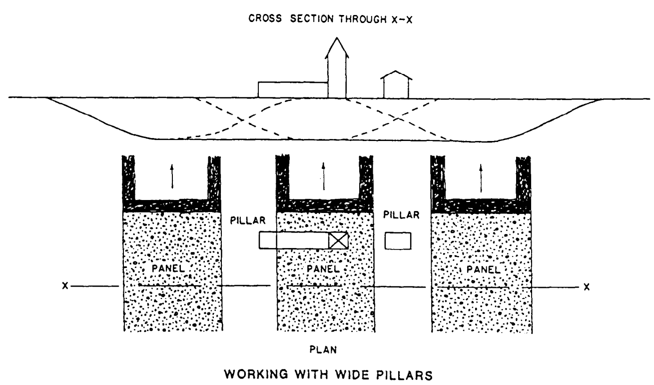


Fig. 10.6.16. Use of sized pillars for protecting surface structures (Singh, 1985).

2. Use of *sized pillars,* that is, the pillar width between panels is adjusted so as to uniformly lower the ground surface (Fig. 10.6.16)

3. *Mining subcritical widths,* so that the maximum subsidence is reduced.

Backfilling: This may be done using hydraulic or pneumatic techniques, which reduce the amount of subsidence, but do not eliminate it entirely. It is a very effective method of mitigating subsidence effects, since it not only minimizes the ground-defor-

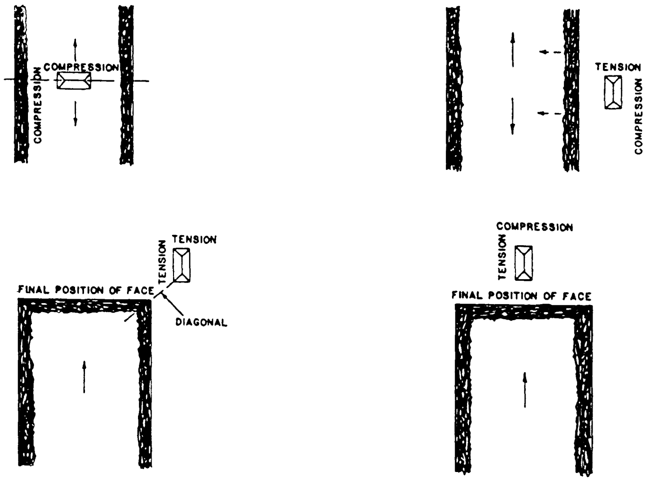


Fig. 10.6.17. Structural strains produced by basic face layouts (Kratzsch, 1983).

mation forces but also conserves the hydrologic regime. Cost effectiveness studies should consider the beneficial effect on the environment such as reduced acid-water drainage, savings in waste disposal and reforestation, prevention of refuse fires, reduced ground fissuring and escape of mine gases, as well as the advantages of long-term strata stability and decreased roof support. Railroads, canals, sewers, and streams experience smaller gradient changes. Hydraulic flushing may also cool the mine air, which is desirable in deep mines. Backfilling may become essential in flat regions with a high water table to prevent flooding, and in areas reclaimed from water bodies (e.g., in The Netherlands).

Harmonious Mining: The technique entails superimposing compressive surface strains on the tensile strains induced by another longwall face in a manner that they move along together. This may be accomplished by staggering two simultaneously worked faces that advance at the same rate, with (1) multiple seams, in which one face is superjacent over another, and (2) single seams, where the panels adjoin.

It is evident, of course, that total cancellation of the traveling strains can only occur if the displacement curves are congruent and symmetrical (i.e., the seam thickness, influence factors, width of compressive and tensile zones, and stowage density, if backfilling is adopted, are identical). Time factors for the mining sequence must also be available from prior experience.

Mine Layout or Configuration: Layout controls the strains experienced by the structure. It may be possible to locate the panel with respect to the building in such a manner as to expose it to deformations that it can withstand (Fig.10.6.17). In some cases, it may be best to stop short of taking all the coal; the loss may not be excessive if the seam is shallow.

Extraction Rate: Face advance cannot be readily altered in mining, and its range is generally limited with the available equipment. A faster rate is desirable in unfractured, viscoelastic strata because it lowers the tensile peak and moves it closer toward the working face. However, in fractured, clastic rocks (such as over previously mined beds), rapid face advance may accentuate displacements and strains and thus induce greater damage.

10.6.5.2 Postmining Stabilization

These techniques have been used in the United States and may extend over large areas (tens of acres or hectares) or be restricted to support a specific structure.

Areal: The four main methods used include:

1. *Backfilling*, which may be conducted
   a. Hydraulically.
   b. Pneumatically.

In either case the procedure may be (1) controlled, when the mine is accessible and barricades can be manually built, or (2) remote (blind), through boreholes when the openings cannot be entered, such as in abandoned mines.

With hydraulic stowing, the water level may rise temporarily in dry mines, acid water may be flushed out into the hydrologic system, and surface drainage may be affected by siltation, pollution, or flooding (especially in shallow mines). The technique is even usable in water-filled mines.

The Dowel1 process (Gray et al., 1974) is a special hydraulic blind-flushing technique, in which the slurry is pumped at a high velocity. The mixture deposits its load when the velocity drops on entering the mine cavity, forming a doughnut-shaped pile. As the pile height nears the mine roof the slurry velocity in the gap increases, keeping the solids in suspension longer, so that the pile grows outward.

Pneumatic stowing causes considerable sparking and may pose a hazard because of the potential for gas ignition.

A commonly used material for backfilling, both hydraulic and pneumatic, is fly ash because of its abundance at coal-fired power plants.

2. *Grouting* entails using a cementitious mixture and thus provides a stronger support. Additives used include Portland cement, pozzolanic mixtures, or organic compounds.

a. *Gravity grouting* is used to simply fill the mine void to whatever extent possible. There is little control, although a perimeter wall may first be built with a thick grout, which is then filled with an expansive grout to achieve good roof contact.

b. *Pressure grouting* is needed if a number of joints need to be filled or roof caving has occurred.

c. *Bag grouting* is a new development (Singh, 1985), and entails lowering a bag through a 6-in. (150-mm)-diameter borehole and filling that with grout until roof contact is obtained.

Grouting under important buildings requires special care (Scott, 1957).

3. *Excavation and fill placement* is only feasible in shallow abandoned mines, with no surface obstructions to excavation. The entire overburden and coal are removed and replaced with compacted fill. Flooded mines may yield large quantities of acid water.

4. *Blasting* of the roof and floor to fill the cavity is a patented technique (Patent No. 1 004 419), which has not been used recently (Gray et al., 1974). Over time, the broken rock compresses, but the movements may be expected to be gradual and evenly distributed.

Site Specific: These techniques are mostly used to support isolated structures.

1. *Grout columns* may be built remotely, but floor and column strengths are variable. Water may impede construction.

2. *Piers and cribs* may be constructed in mine openings that are accessible, if the mine floor and roof are competent.

3. *Deep foundations* may be used with shallow workings. They are, however, liable to damage by lateral shear forces that may be experienced.

4. *Groutcase supports* entail placing casing between the mine roof and floor and filling it with grout. These supplement existing coal pillars.

10.6.5.3 Architectural and Structural Design

Orientation: It is preferable to have the long axis of the building parallel to the subsidence contours. If a fault exists nearby, the shorter axis should be oriented perpendicular to the fault.

Location: Faults tend to concentrate ground strains, hence structures should be located at least 50 ft (15 m) away.

A single building should not be constructed on dissimilar soils, owing to the possibility of differential deformations or settlements.

Subsidence-Resistant Construction: This technique has received considerable attention in the literature. It may be discussed under four major construction categories:

1. *Rigid* in which both the foundation and superstructure are rigid in design. Often the foundations are highly reinforced concrete rafts or beams, capable of withstanding ground displacements and curvature. The structures generally span or cantilever over a subsidence wave. Foundations are of small plan area. Elevator shafts and the like are designed with extra clearances.

2. *Flexible design* permits slab foundations for small buildings such as houses. The slab should preferably be less than 60 ft (18 m) along the side, poured in a single operation, without joints, and finished close to ground level. It is generally underlain by granular material. Reinforcement should be near both the top and bottom so as to accommodate tensile and compressive strains.

If the building has a basement, there should be an open gap around it or filled with a compressible or granular material. Larger buildings may have rollers or slip-joints between the superstructure and foundation. Trenches around structures absorb some of the strains.

Flexible structures are designed to track the traveling subsidence wave without cantilevering, permit free ground movement below the foundation, provide sufficient superstructure support in spite of the ground flexing, and accommodate subsidence deformations that are larger than anticipated without jeopardizing structural stability.

3. *Semi-flexible designs* are used in instances where the structures can tolerate minor damage, such as some warehouses. These do not strictly adhere to the rigid or flexible criteria outlined above. It may be more cost effective to perform minor repairs as required than employ these more expensive designs.

4. *Use of releveling devices,* such as jacks, to prevent tilting. Excessive tilt may cause the gap between adjacent buildings to be reduced to the extent that they touch.

Gaps need to be provided between all buildings to allow for both compression and tilt. Other precautions that are helpful, depending on design philosophy, are (Chen et al., 1974; Anon., 1977)

a. Provide expansion joints to accommodate ground movements and thermal expansion.

b. Minimize the number of door and window openings and use flexible frames; their location should not significantly weaken the structure; do not position front and back doors opposite each other.

c. Avoid weak skin materials within rooms; partitions between building segments should be strong; instead of plaster on ceilings and walls, use plaster board.

d. Floors and roof should be secured to the walls.

e. Allow for tensile strains at all structural connections; movements should be possible for staircases.

f. Exclude masonry arches.

g. Do not have comer or bay windows or porches.

h. Detach outbuildings from the main building

i. Provide excessive falls for gutters.

j. Do not pave immediately adjoining buildings; use bituminous type materials for paving where necessary (e.g., driveways).

k. Employ flexible damp-proof courses (e.g., bitumen).

l. Use light fences around properties rather than walls.

m. Replace rigid retaining walls with earth banks.

Modification of Existing Structures: Total repair expenses may sometimes be reduced if a building is suitably modified prior to its experiencing ground movements. Possible alterations include (Chen et al., 1974; Kratzsch, 1983)

1. Cutting out a part of a house or removing an entire house from a row of buildings; unit lengths should be about 60 ft (18 m), with cuts extending into trenches, and gaps bridged with flexible materials; preferably locate cuts in connection corridors or unit divisions.

2. Digging trenches around a building (and filling with compressive material weaker than the surrounding soil) to below foundation level, without disturbing the foundation; trenches may be covered, if desired, with concrete slabs that do not butt.

3. Slotting rigid pavements or floors, and even superstructures (generally wood, brick, or stone do not present difficulties; concrete may).

4. Introducing slip planes, especially in new buildings.

5. Providing temporary supports and/or strengthening to parts susceptible to damage; support screens, partitions, and ornaments independently of the walls and floor.

6. Using tie rods, if it is anticipated that the roof trusses will be pulled out from their seats; however, indiscriminate use of tie rods may needlessly disfigure the building; stress concentrations at tie-rod bearing plates may pull these through the walls; often temporary corbels provide adequate support for trusses.

7. Installing pretensioned steel mesh around the exterior walls (this could be dismantled and reused later).

8. Taping windows, (especially with metal frames), to avoid flying glass.

9. Removing and storing stained glass windows, until subsidence is complete.

Remedial and Restorative Measures: Increasingly, structures are being constructed so as to be easily repaired after subsidence damage. Since a tension wave is usually followed by a compression wave, cracks should not be patched until all movements have stopped. Debris in the fractures should, however, be removed prior to the compression cycle.

In low-lying areas, the water table may create difficulties, necessitating the installation of drains and pumps.

10.6.5.4 Comprehensive Planning

It is desirable to plan both the surface land use and the mine with full knowledge of the requirements of each. Deep cuts for highways, railroads, or other structures, or excavations for utility tunnels or basements, may reduce the competent overburden thickness above the old workings to induce subsidence. This type of situation can be prevented with planning.

For planning or any other measure to be successfully implemented, it is paramount that everyone affected by subsidence fully comprehend what is being done and why. Therefore, an intensive effort of public education about the subject is in order. This should not only be directed towards the general populace, but also the mine operating personnel, builders and developers, government officials at all levels, and civic groups.

Four situations (Anon., 1977) may be identified, each of which requires a slightly different approach to planning:

1. Existing subsidence potential, existing development.
2. Existing subsidence potential, future development.

3. Future mining area, existing development.
4. Future mining area, future development.

Essentially all these approaches entail either coordination or control of both the surface and subsurface development.

Coordination of Surface/Underground Development: Although not a comprehensive list, typical of the principles that may be followed are

1. Avoid construction near outcrops or faults.
2. Build only specially designed structures over shallow workings; surface effects are magnified as the depth decreases.
3. Locate buildings above steeply dipping seams since the strains induced are reduced.
4. Erect communications or other significant structures in unmined or completely subsided areas.
5. Alter routes of highways, railroads, canals, and other structures to suit coal conditions (e.g., over want areas or near fault planes); subsequent costs for lowering may be thereby reduced.
6. Site linear structures (e.g., canals, railroads) so that they can be uniformly lowered along their entire length; locks may be located over unminable zones, although massive lock structures can be dropped without significant damage.
7. Avoid building important structures near mine boundaries since coordination with several mine operators and surface land owners is onerous; also boundary pillars may introduce higher stresses.

Collaboration between the mining companies and surface owners and developers is essential in regional or zonal planning, otherwise problems will arise.

Land Use/Development Control: Development of land areas overlying mines must be economically justifiable as well as socially and culturally acceptable. This implies that often regional plans should not only be discussed with mine and surface owners, but also be open to public comment prior to adoption. Changes in these plans also deserve an equally protracted treatment.

Federal, state, regional, county, and local government authorities exert considerable control over development of land that is potentially liable to damage due to subsidence through these means:

1. Surface Mining Reclamation and Control Act (SMRCA) of 1977 (Public Law 95-87).
2. Environmental impact requirements.
3. Zoning and subdivision regulations.
4. Building provisions (issuance of permits).
5. Mining regulations.
6. Safety requirements.
7. Insurance needs.
8. Investigative requirements for public buildings (e.g., Pennsylvania's Act 17 of 1972).
9. Special local ordinances.
10. Interagency coordination.

Perhaps in the future, it will be mandated that mine operators prepare plans that depict predicted subsidence locations, extent, trough centers, maximum subsidence, values and direction of tilt, compression and extension zones, and other pertinent data. These could then be circulated to building authorities, highway commissions, railroads, water supply and other utility agencies, pipeline operators, and others who may be affected for comments and suggestions (within strict time limitations). On the other hand, these groups as well as builders/developers should be required to incorporate proper precautions in the design of their respective structures. In extreme cases, construction may be barred from particularly risky areas, and these lands used for parks, forest preserves, and open spaces.

ACKNOWLEDGMENT

The author would like to thank Peter J. Huck of Engineers International, Inc. for his comments on the text and help with Example 10.6.1.

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Chapter 10.7
OTHER APPLICATIONS OF GEOMECHANICS

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Many areas of geomechanics are not covered elsewhere in this *Handbook.* In addition, there are uses for geomechanics that are peripheral to many other mining endeavors. This chapter introduces these peripheral areas and as yet uncovered topics. Applications of geomechanics should be discussed in a systematic fashion, so that the users can directly apply the concepts, principles, and information to their particular problems. In this chapter, geomechanics is discussed with respect to design at a planned, developing, or operating mine. Design as used here can include remediation of problem areas in the mine, design of a prototype mining method or layout, conceptualization of layout or support system alternatives, and calculation of stresses and deformations in an attempt to understand the behavior related to a particular observation, for example, pillar hour-glassing.

Application of geomechanics principles must follow a logical thought process; however, in solving particular problems, designers should not limit themselves to conventional techniques or design practices but should also consider findings in other disciplines, results of research and development, and conclusions from brainstorming sessions. A design philosophy and approach in geomechanics based on multiple working hypotheses has been found to be successful and workable in many cases. This approach has developed in response to the need for safe working conditions, sound design (consistent with precedent and accepted practice), cost effectiveness (to maintain profitability in economic downturns such as the early 1980s), and innovation (to maintain and enhance utility in the future). These four needs are driven by the respective requirements of (1) society and labor; (2) regulatory agencies (Mine Safety and Health Administration [MSHA], Office of Surface Mining, Reclamation, and Enforcement [OSMRE]); (3) the owner (mine management and shareholders); and (4) the design engineers themselves (for preservation, enhancement, and effective utilization of their skills).

In practice, this approach is achieved by developing an understanding of the true nature of the problem—for example, is the raveling slope a safety problem, or, more critically, the harbinger of an impending major failure—and by planning an effective step-by-step approach to the solution. Project management techniques (Martin, 1976) can aid in developing a systematic approach and have been found useful here. To Bieniawski's (1988) ten stages in the engineering design process (see Chapter 10.5) should be added another stage: observation and re-analysis.

Innovation can be included in the above process (Bieniawski, 1984a, 1988) through the use of brainstorming (a group throws out ideas for general consideration with no idea initially dismissed, no matter how fanciful, impractical, or illogical it may seem); programmed invention (Edison realized that a highly resistive filament would provide light when energized, and then his research group concentrated on either making the concept practicable or demonstrating its impracticability); or cross-fertilization (bringing together elements from different specialties).

Geomechanics design should be obtained from multiple analysis methods that converge to a desirable and mutually supportable conclusion. When the same solution is given by different analysis techniques, the engineer can have greater confidence in the result. Conversely, if the different techniques provide vastly different solutions, it is necessary to evaluate the data, methodology, and assumptions for the various techniques used. For instance, numerous formulas have been proposed for designing pillars in room and pillar mines: Holland and Gaddy (1957), Obert and Duvall (1967), Hedley and Grant (1972), Wilson (1972), Abel and Hoskins (1976), Bieniawski (1984b), Sheorey et al. (1987), and Salamon (1967). When designing a layout, try them all, determine which are most appropriate—some apply strictly for coal, some for hard rock, and some only to specific geometries —and optimize the results. For design in rock, one can use (1) basic statics methods (resisting forces vs. driving forces); (2) finite element, finite difference, boundary element, or discrete element numerical modeling methods; (3) physical models; (4) empirical methods (rules of thumb); or (5) whatever else can be brought to bear on the problem. In the design of a new level for a deep caving-method mine, the rock mass can be classified and support requirements estimated, load-shedding and driving forces can be calculated, numerical modeling can be performed to determine the strains to be accommodated, and lining requirements can be determined empirically based on comparisons of practices at other mines. The answer will be found from proper consideration of all these possibilities. No single method should be considered in preference to the others for it may be inapplicable or misleading.

10.7.1 GEOMECHANICS AS A TOOL

Like economics, time, labor, and other considerations such as personal or operational preferences, geomechanics is but one tool in a mine plan or design. It cannot be used in isolation, and must be used properly, with full awareness of the assumptions and limitations.

10.7.1.1 The Mine as a Structure

A mine is a system not only of functional components (mine, mill, shops, offices, transport) but also of structural components. For an open pit mine, these components could be benches, pit haul road, conveyor line, crusher station, and sump. Underground, they could be stopes, drifts, adits, ventilation drifts, crushing chambers, conveyor galleries, shafts, access and ventilation raises, orepasses and ore pockets, and the surface. These all interact structurally, and a failure in one could be catastrophic to the mine as a whole. Table 10.7.1 indicates the structural interactions likely for various components. These components cannot be designed in isolation, and the structural interactions of one on another must be considered.

10.7.1.2 Use in Mine Planning

Geomechanics is fundamental to mine planning. An open-pit or strip mine must have designed slopes and highwalls. Opening and stope stability are critical to any underground operation. At the exploration stage of a mining project or mine expansion project, geomechanical data must be collected to provide information for planning and design. For each core drilled, rock

Table 10.7.1 Mine Structure Interactions

Component	Interacts with
	SURFACE
Benches	Pit slope—bench stability dictates overall slope stability
	Haul roads—bench stability affects utilization of haul road above and below it
	Conveyor line—same as for haul roads
Pit slopes	Fundamental interaction with all components if located above, on, or near an unstable slope
	UNDERGROUND
Drifts	Stopes—instability in drifts may restrict access to stopes or lead to failure of overlying stopes
	Ventilation drifts—separate drifts for ventilation, common in some mining types, may be damaged or closed at intersections with unstable drifts
Stopes	Drifts—failure in stopes will lead to distress in overlying and underlying drifts of all types
	Shafts—instability in stopes may progress to ground around shafts, resulting in deformation or failure of the liner
	Surface—sinkholes or other subsidence features may develop at surface
Conveyor galleries	Crushing chambers at start or end of galleries may be impacted
	Shafts—the tail ends of galleries are often near shafts so that instability may lead to deformation or distress of shaft components
Ore pockets	Conveyor galleries, which often lead to or from ore pockets, may suffer distress from ore pocket failures
	Shafts may also suffer distress due to failure in nearby ore pockets
Surface	Stope—in caving or near-surface mines, surface deformations (slides, subsidence) may cause failure or filling of stopes
	Shafts—surface movements will distress shafts
Pillars	Drifts (rooms)—unstable pillars will result in closure of rooms or drifts
	Shafts—unstable pillars can lead to deformed and distressed shaft linings if the shaft is within the area of subsidence resulting from the unstable pillars
	Surface—failed pillars can lead to sinkholes or other subsidence effects

quality designation (RQD), fracture data, strength data (such as from a field point-load tester), core recovery, and a detailed geomechanical description must be provided. At large mines with aggressive exploration programs, two logging efforts are often made on all core—the first, geologic for ore control, the second, geomechanical. Separate logging forms are used to minimize clutter, and the core may be logged by different individuals. The geomechanical logging is usually performed before any splitting or sampling of the core, so it is often first. Obtaining correct and accurate geomechanical data early in the exploration phase is critical, because, as exploration proceeds and the prospect becomes a project, any redrilling required to obtain geomechanical data will be rushed. Moreover, it is very difficult to extract sufficient geomechanical information from split and assayed core (split core cannot be run in testing machines).

As mine plans are conceptualized, the geomechanical abilities of the rock mass to accommodate the plan must be assessed, as must the geomechanical consequences of the plan. For example, in room and pillar mining, the span of the rooms will initially be dictated by equipment clearance requirements and drilling equipment face coverage capabilities, whereas pillar size will be dictated by rock mass strength and deformation characteristics. After pillar size has been selected, room span, and crosscut span and spacing may be adjusted to meet the extraction ratio limits dictated by the pillar size and rock-mass characteristics. However, the limiting factor in room spans is the ability to support or reinforce the roof. It may be necessary to down-size equipment or put more effort into roof control to produce a safe design. Economic considerations of geomechanical designs (cost of rock reinforcement) must be balanced with equipment and with loading and haulage costs (size of units).

In caving mining systems, the geomechanical properties of the rock mass are all-important to the success of the system. Stewart (1981) contains numerous papers on rock mass descriptions for caving.

In open pit and strip mines, the required slope angle obviously dictates the stripping ratio.

10.7.1.3 Use in Mine Operations

Geomechanics programs should actively continue during mine operations, because more ground is now opened up, which facilitates sampling and mapping. In both underground and open pit mining, ongoing geomechanical programs should collect data on (among other things):

1. Rock strength properties and variations in strength.
2. Characteristics of fractures and faults.
3. Support performance (underground, and, where necessary, in open pits).
4. Slope performance (open pit).
5. Stress indicators (usually a greater concern underground).
6. Blasting performance.

The initial mine design should be fine-tuned to optimize return on investment and recovery of ore. Such data as listed above will allow reconsideration of the design and redesign, where necessary, to address changing and unforeseen conditions.

A balance must be struck between having, on one hand, a tailor-made design for every small mine section and, on the other hand, a standard design for the entire mine that is based on the expected worst conditions. A better solution is to design for average conditions, while making provision for alteration where necessary. For instance, a slope design that can accommodate wider benches or catch benches can have great flexibility for optimizing stability. In room and pillar mining, if pillars are initially left as ribs, crosscuts can be placed where roof or pillar conditions are best, rather than at the locations required by a regular, checkerboard pattern. In caving, design changes in small areas are difficult to achieve, and a redesign in an already-developed area is a disaster—once the drawpoints and undercuts are established, the mine is committed. (This is also true for stoping blocks in massive, steeply dipping ore bodies where methods other than caving—open stoping, shrinkage stoping, cut and fill, or vertical crater retreat mining—are used.)

10.7.1.4 When Things Go Wrong

It is all too common for geomechanics programs to be established only after serious problems have arisen. However, increasing requirements for regulatory permits are driving a demand for prediction of mining consequences. To do so requires the use of geomechanics. In any profitable mine, the very nature of natural geologic materials or human error will create a problem sooner or later. If a mine is designed to completely avoid any stability problems, or to be idiot-proof, it is probably uneconomic. (In jurisdictions where mines are government-owned, the economic viability of the mines may be secondary to other considerations; however, when mines are privately owned, mines will be operated or developed only if they can return a profit.)

The sorts of things that can go wrong geomechanically are

1. Surface: slope failures, bench failures, oversize in the blasts, poor road conditions, excess water.
2. Underground: roof falls, stope failures, pillar instability, drift closure, unsuccessful blasts (misfires, cutoff holes or short pulls), surface subsidence, runs of loose ore.

When such conditions occur, the original data collected must be reviewed carefully for the following

1. Was something missed (an unknown fault, a dike)?
2. Was something misinterpreted (why was that core lost)?
3. Was the laboratory testing independently verified?
4. Was the minimum core segment length changed when RQD was calculated for core diameters other than NX? (Four in.-long pieces apply to NX core [Deere, 1964] but not BX [Heuze, 1971], AX, or EXT.)
5. Were all holes drilled at the same orientation so that structures parallel or nearly parallel to the holes were missed? (Although this problem was recognized by Terzaghi [1965], it continues to be a major cause of incorrect structural interpretation.)
6. Were two distinct rock masses lumped in the same structural domain?
7. Is the stress field substantially different from that estimated or does it vary? (Changes in the orientation of the stress field within a mine can be inferred from the distribution and orientation of major structural features such as faults, folds, or dikes.)

All the design data and assumptions must be reconsidered. If there is no apparent cause for a problem, other investigations must be performed

1. Monitor deformations.
2. Instrument rock bolts.
3. Re-map the structural geology.
4. Monitor water levels.
5. Install pressure cells.
6. Install microseismic networks.
7. Drill additional exploratory coreholes.
8. Examine timing and sequencing of excavation steps (for example, in a stoping block with simultaneously mined parallel overhand stopes, the center stope should lead.
9. Determine whether support materials are appropriate and sufficiently long-lived.
10. Determine whether opening or bench use has changed from what was planned so that the opening or bench environment has altered.

Careful analyses of these and similar features and information will usually reveal the nature of the problem. Instrumentation is available that can monitor almost anything. Geophysical probing can be helpful, but low-velocity media will be obscured by adjacent high-velocity media when seismic methods are used (problem areas are usually low-velocity areas). All geomechanical aspects must be addressed before blaming operations personnel for problems, or accusing them of not doing their share. Unfortunately, human errors or errors in judgment cannot be overcome solely through improved geomechanical design. The operating problems that can (and have) led to mine failures include

1. Using rock bolts of incorrect length (usually too short, but roof failures can occur if bolts are all anchored at a uniform depth although the length is otherwise adequate).
2. Using insufficient (or all the same) delays for blasting rounds.
3. Using additional blastholes (or overloading holes) to use up powder.
4. Delaying or omitting roof support because "the roof always stands well."
5. Running drainage on the wrong bench.
6. Robbing pillars until they are grossly undersized, in order to get cheap muck.
7. Ignoring big voids discovered when blastholes are loaded.

10.7.2 IMPORTANCE OF PRESERVING ROCK INTEGRITY

The rock mass in which a mine is constructed is capable of great strength if its original strength is preserved in some way. "Weak" rocks are often considered to be those 8000 psi (50 MPa) or less in strength, whereas "strong" rocks are those with strengths in excess of 20,000 psi (140 MPa). Concrete for most purposes seldom exceeds 8000 psi (50 MPa) in strength, and in underground applications concrete strength is generally 3000 to 4000 psi (20 to 25 MPa). Hence concrete is usually weaker than the rocks encountered in most mining. Why then is it common (especially in tunnels and underground hydroelectric machine halls) to over-excavate and replace the rock with concrete? (Putting it another way, why tear out 20,000 psi, or 140 MPa, rock to replace it with 4000 psi, or 25 MPa, concrete?) There are two reasons:

1. The rock is flawed with joints, bedding, and other discontinuities, and, even when supported by passive external supports such as timber or steel sets, tends to detach or fall out as blocks.
2. Concrete can be engineered to give predictable properties and behavior, whereas the rock mass cannot.

Because of the greater inherent strength of the rock mass, any effort that preserves this strength is beneficial.

10.7.2.1 Rock Substance Compared to Rock Jointing

The strength of the rock substance is conventionally described in terms of the Mohr-Coulomb criterion (see Chapter 10.2):

$$\sigma_1 = S_c + \sigma_3 \tan^2\psi \qquad (10.7.1)$$

where σ_1 is maximum principal stress, σ_3 is minimum principal stress, S_c is uniaxial (unconfined) compressive strength, and ψ is failure angle (= 45° + $\phi/2$, where ϕ is the angle of internal friction). Failure is considered to occur if the actual maximum principal stress exceeds the value calculated from Eq. 10.7.1 when the actual minimum principal stress is substituted. The intermediate principal stress σ_2 is considered not to affect the strength. (This results from the fact that, in conventional "triaxial" testing, the intermediate principal stress is the same as the minimum principal stress. However, recent research with "true triaxial" testing machines has shown that the strength is affected by the magnitude of σ_2. Moreover, in real situations, σ_2 and σ_3 are usually different. Thus a failure criterion that incorporates all three principal stresses would be more applicable. Nevertheless, because the Mohr-Coulomb and other strength criteria that neglect the effect of σ_2 have been shown to give reasonable predictions, and because of their simplicity compared to triaxial failure criteria, their use is retained in these discussions.)

A "strength envelope" is obtained by plotting the results of several tests at different confining stresses on a graph of shear

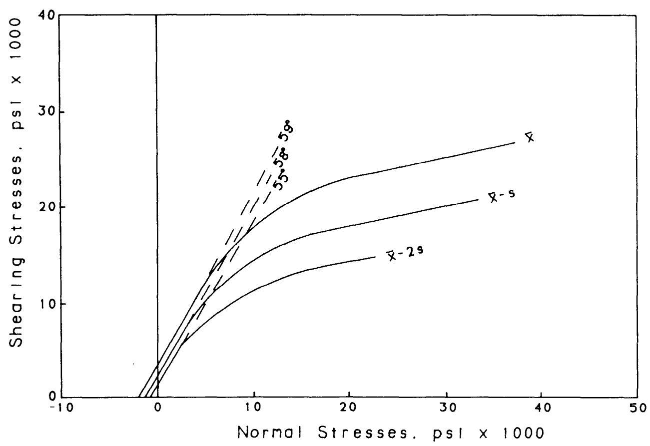

Fig. 10.7.1. Results of strength tests at various confining stresses for intact rock. Conversion factor: 1000 psi = 6.895 MPa.

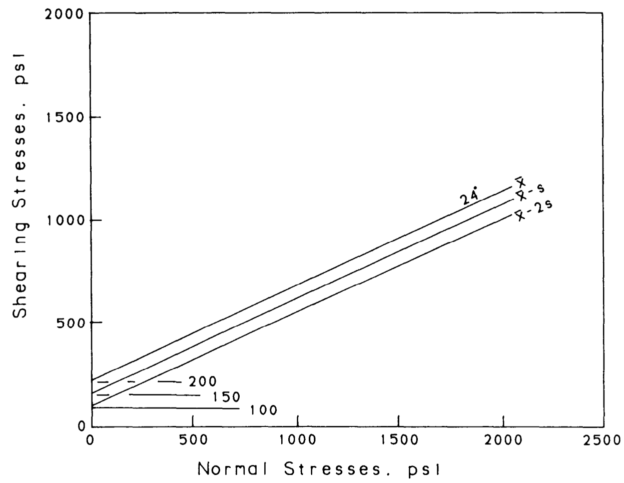

Fig. 10.7.2. Results of shear tests at various normal stresses for rock joints. Conversion factor: 1 psi = 6.895 kPa.

vs. normal stress (Fig. 10.7.1). The Mohr-Coulomb criterion gives a linear strength envelope. However, as shown, the strength envelope is actually curvilinear (parabolic) and approximately linear only for small confining stresses. Jaeger and Cook (1976) combine the Mohr circle construction for strength with a Griffith failure criterion to obtain the following parabolic failure criterion:

$$\tau^2 = 4T_o(\sigma + T_o) \tag{10.7.2}$$

where τ is shear stress, σ is normal stress, and T_o is tensile strength. Flaws or discontinuities can be tested to determine a similar strength envelope that will also usually be linear in the low-stress portion that is typically of interest. Fig. 10.7.2 illustrates a shear strength envelope for discontinuities.

Fig. 10.7.3 shows a graph of the combined rock substance (or intact rock) and discontinuities envelopes (Kendorski, 1977). In laboratory testing of intact rock, or in situations where the rock is not flawed by discontinuities, the mine rock will follow the upper envelope. When discontinuities totally control failure, such as the rock mass sliding along a large joint into a pit, the lower envelope is followed. Most rock masses, however, will behave in some intermediate fashion, with some failure along discontinuities, and some failure through intact rock. Fig. 10.7.3 also shows such an intermediate case where the rock has been restrained in some way to prevent loosening and block movement that would result in progressive failure along discontinuities only.

10.7.2.2 Rock Mass Strength

One of the greatest continuing problems in geomechanics is the manner in which to describe the strength properties of a rock mass consisting of intact rock (or rock substance) and pervasive discontinuities. Unfortunately, such rock masses are the norm in rock excavations. There is often considered to be a scale effect in rock mass strength—that is, the strength decreases as the volume of the rock mass increases. However, this scale effect is really a reflection of the fact that as the volume increases, the probability of discontinuities being incorporated into it increases. (Using the weakest link principle, the larger the number of discontinuities, the smaller the rock mass strength is likely to be.) A number of methods have been proposed for characterizing the rock mass strength (see Chapter 10.5 for a full discussion of these). They can be summarized as follows:

1. *Semi-empirical Scaling.*

For massive, elastic rock:

$$S_p = S_1(0.778 + 0.222\ W/H) \tag{10.7.3}$$

(Obert and Duvall, 1967)

For massive, brittle rock:

$$S_p = S_1 W^{0.5}/H^{0.75} \tag{10.7.4}$$

(Hedley and Grant, 1972)

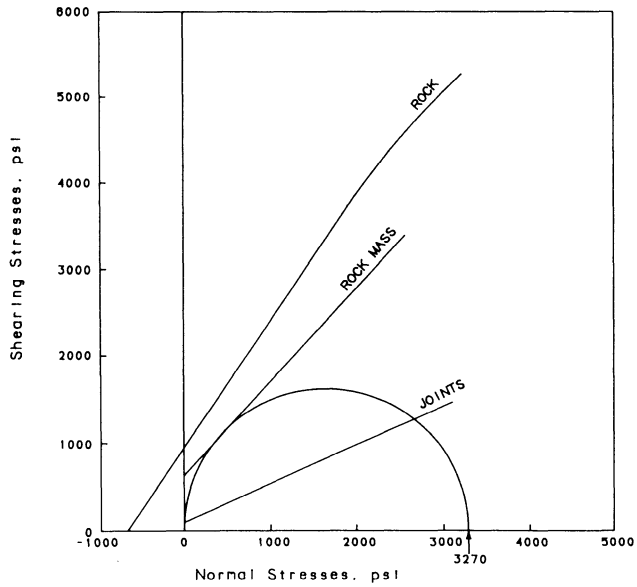

Fig. 10.7.3. Failure envelopes for intact rock and discontinuities and for failure intermediate between intact rock and discontinuities. Conversion factor: 1 psi = 6.895 kPa.

For coal:

$$S_p = S_1 W^{0.5}/H \qquad (10.7.5)$$
(Holland, 1974)

$$S_p = S_1 W^{0.46}/H^{0.66} \qquad (10.7.6)$$
(Salamon and Munro, 1967)

$$S_p = S_1(0.64 + 0.36W/H) \qquad (10.7.7)$$
(Bieniawski, 1984b)

$$S_p = S_1/H^{0.36} + amc\gamma h(W/H\text{-}1) \qquad (10.7.8)$$
(Sheorey et al., 1987)

where S_p is rock mass strength, S_1, is strength of a cube of intact rock (usually of a specified edge dimension), W is pillar width, H is pillar height, h is depth of cover, g is unit weight of the rock, m is ratio of the primitive stresses, c is a constant accounting for end effects, and a is a triaxial parameter. (Note that the formula given by Obert and Duvall for massive, elastic rock was initially applied by Bunting (1911) to design of pillars in anthracite coal mines.)

2. *Confinement Effect* (Wilson, 1972).

$$L = \sigma_v[WL - (W + L)y + 4y^2/3] \qquad (10.7.9)$$

a. $\sigma_v = \sigma_o + m\gamma h \tan\beta$
b. $y = H \ln(\sigma_v/\sigma_o)/[(\tan\beta)^{0.5}(\tan\beta\text{-}1)]$
c. $\tan\beta = (1 + \sin\phi)/(1\text{-}\sin\phi)$

where W is pillar width, H is pillar height, h is depth of cover, g is unit weight of the rock, ϕ is angle of internal friction of intact rock, m is ratio of the primitive stresses, L is length of the longer side of a rectangular pillar (= W for a square pillar), and σ_o is unconfined compressive strength.

3. *Partitioning* (Kendorski, 1977).

$$S_r = 2(c_f + R(c_r - c_f))\tan(45 + \phi_m/2) + \sigma_3\tan^2(45 + \phi_m/2) \qquad (10.7.10)$$

in which

$$\phi_m = \arctan(\tan\phi_r + R(\tan\phi_r - \tan\phi_f))$$

and where c_r is cohesion of intact rock, c_f is cohesion of discontinuity, ϕ_r is angle of internal friction of intact rock, ϕ_f is angle of internal friction of discontinuity, R is the fraction of intact rock present on a potential failure surface (balance is discontinuity), σ_3 is minimum principal stress, and S_r is rock mass strength.

(Note: The serious reader is strongly advised to consult the original references, or Chapter 10.5, before using these formulas.)

For some purposes, especially numerical modeling, it has been found useful and reasonable to "arbitrarily" discount the rock strength and other laboratory-derived properties, such as elastic moduli, by 40 to 60% to accommodate the rock-mass field situation. Models where this has been done give a reasonable approximation to the measured behavior in the field. The methods given here (Eqs. 10.7.3 through 10.7.10) also usually result in a discounted strength in the 40 to 60% range.

This discussion highlights the importance to the rock mass strength of the intact rock. The frictional properties of the discontinuities impart little strength to the rock mass and represent the lower bound to the rock mass strength. The rock mass integrity must be preserved as much as possible if the rock mass strength is to be effectively mobilized. The deterioration of rock-mass strength as the rock mass deforms is a time-dependent phenomenon, so that the integrity of the rock mass can be preserved by early application of reinforcement, restraint, support, or other measures. As the rock mass deformation increases in response to imposed loads or strains, the strength lessens, and it is essentially impossible to regain the lost strength or undo the deformations. Ultimately, an unrestrained rock mass will progressively deteriorate until it is composed of detached or loosened blocks bounded by discontinuities, and has reached the lower bound of its strength.

The so-called "New Austrian Tunneling Method" (NATM) uses this concept as its key in order to apply exactly the restraint needed at the right time, as determined by field measurements and observation. The "key block" concept (Shi and Goodman, 1981; Goodman, Shi, and Boyle, 1982) shown in Fig. 10.7.4 also uses the principle of preserving the rock mass integrity by determining the location and orientation of key blocks that, when restrained, prevent progressive deformation in the rock mass. The beauty of this concept is that the restraining effort is minimized, but the drawback is that it must be accurately determined at the moment of excavation. Yow (1985) discusses

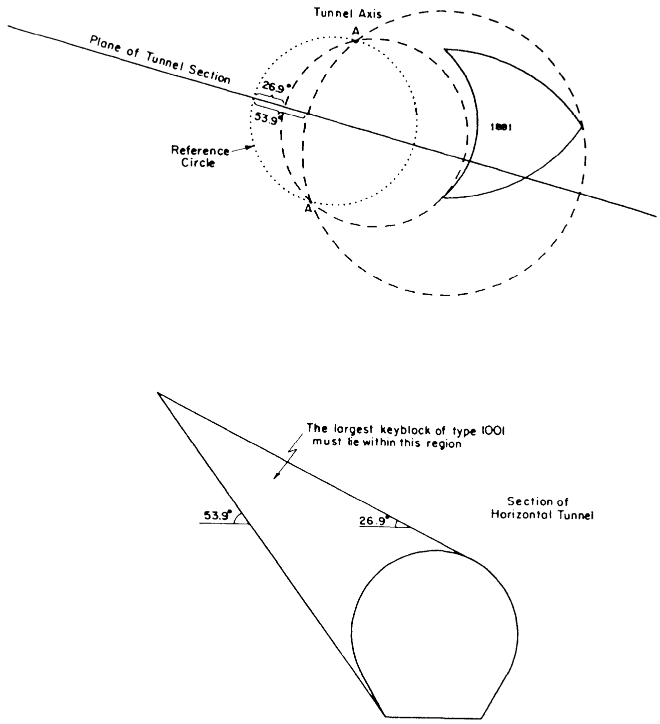

Fig. 10.7.4. The "key block" concept (Shi and Goodman, 1981).

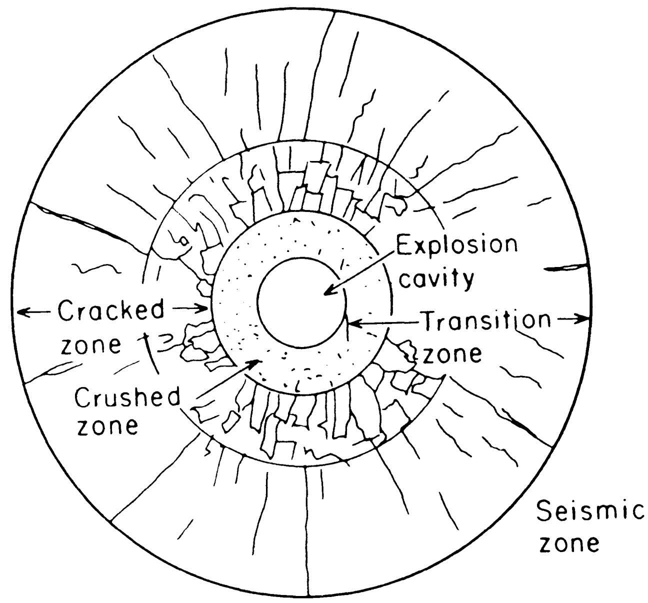

Fig. 10.7.5. Crushing of the rock mass adjacent to a blasthole by the blast-generated shock wave (Atchison, 1968).

key block stability calculations and presents computer programs for two- and three-dimensional analyses.

In underground situations, rock reinforcement restrains the rock mass and preserves its integrity through several mechanisms: pressure to keep loosening blocks in place, suspension of loosening strata, and reinforcement across discontinuities tending to slip. In surface excavations, rock anchors or tendons act in a similar manner, although long bars or tendons can also apply active forces to decrease sliding (Hoek and Bray, 1974; Seegmiller, 1982).

10.7.2.3 Blasting Effects

Mechanism of Fracture. The blasting process consists of several events that can damage the rock mass and lead to a loss of its integrity. When explosives or blasting agents are detonated, a shock wave is generated that progresses out from the blast location into the surrounding rock (see Chapter 9.2.1). This shock wave will locally crush the rock (Fig. 10.7.5) and was once thought to be solely responsible for fracturing near free surfaces. However, recent research has shown that such fracturing is caused by the expansion of compressed gases formed from the explosive reaction. The gases expand as they depressurize and, in doing so, create cracks and expand existing ones. Atchison (1968) has explained (Fig. 10.7.5) how the rock adjacent to the blasthole is damaged. Holmberg and Persson (1979) have shown how the rock much further away can be damaged by confined shots. The rock mass integrity can be preserved, when blasting, by two methods: presplitting and trim blasting.

Presplitting. In presplitting, small-diameter, lightly loaded holes are detonated at the perimeter of the desired excavation prior to the remainder of the blast, in order to presplit the rock mass and to provide a plane that the rock failure can progress to but not pass. The holes must be sufficiently loaded to allow gas-produced cracks from adjacent holes to connect. Cracks will, of course, extend in other directions, and damage to the rock mass from presplitting has been observed on numerous occasions—Hoek and Bray (1974), Herget (1972), and Oriard (1972) have all reported such instances. Thus great care must be taken in its design and application.

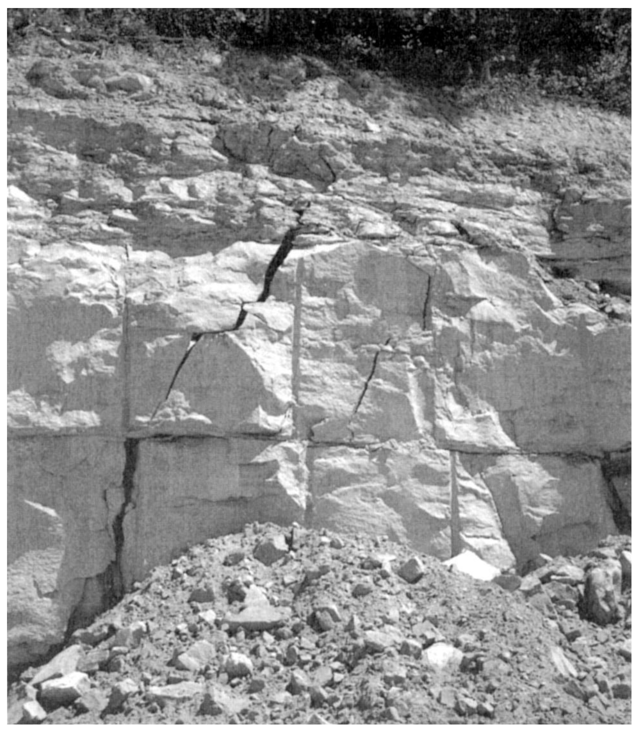

Fig. 10.7.6. Coal strip mine highwall severely damaged by blasting.

Trim Blasting. Trim blasting is essentially the reverse of presplitting. Perimeter holes (smaller or the same size as the holes for the main blast) are drilled and loaded, but are not detonated until after the main round. The holes are more closely spaced than in the main round; however, the explosives load in the holes is reduced and may be partially decoupled from the rock mass. Because of the light load, decoupling, and late firing, the rock slabs off the excavation perimeter, minimizing the damage to the surrounding rock.

Movement of Blasted Rock. The movement of the blasted rock can also damage the remaining rock mass, especially in surface mining. Fig. 10.7.6 shows a coal strip mine highwall severely damaged by the rock being forced to move along the slope face, rather than away from it. The dragging and tearing action has pulled apart the rock mass, weakening it, and causing it to lose its integrity.

10.7.3 STATE OF STRESS IN SITU

10.7.3.1 Causes and Estimation

The state of stress in the rock mass is extremely important, as it drives the rock deformations in response to excavation (see Chapter 10.2). Rock stress is a consequence of gravity, tectonic activity, or post-glacial isostatic rebound. The rock stress will be perturbed by excavation activity, and by rock damage from blasting or other destructive activities (e.g., hydraulic fracture stimulation for solution mining and oil and gas production).

Vertical Stress. In general, rock has a unit weight of about 160 lb/ft^3 (2560 kg/m^3), which results in a superincumbent load of 1.1 psi/ft of depth below the surface (0.025 MPa/m of depth). This relation provides a good estimator for the vertical stress in most cases—exceptions are areas with hill-and-valley topography, and, in some cases, areas of tectonic activity.

Horizontal Stress. For the horizontal, or confining, stress, the situation is considerably more complicated. According to elasticity theory, the confining stress is related to the superincumbent load (vertical stress) through the following formula:

$$\sigma_h = [\nu/(1 - \nu)]\sigma_v \qquad (10.7.11)$$

where ν is Poisson's ratio, σ_v is vertical stress, and σ_h is horizontal or confining stress. Although Eq. 10.7.11 has traditionally been used to estimate the horizontal stress, its use should be limited to areas where tectonic activity, post-glacial isostatic rebound, and major faulting are absent. Where tectonic activity or post-glacial isostatic uplift has occurred (or is occurring), and in areas where there is faulting, the horizontal stresses can considerably exceed the vertical stress. In bowl-shaped sedimentary basins, although the horizontal stress may be less than the vertical, it can, at the same time, be greater than the stress predicted by the above formula. Thus the situation assumed by Eq. 10.7.11 is the exception rather than the rule, and indiscriminate use of that formula, especially in designing tunnel supports, can lead to failures.

General Guidelines for In Situ Stress Estimation. Abel (1983) has provided a general guide for estimating the stress state in rock masses from which the following is synopsized:

1. Igneous and metamorphic rock types: high horizontal stresses probable.
2. Sedimentary rock type: high horizontal stresses possible.
3. Faulting nearby in igneous rocks: maximum stress approximately parallel to fault strike.
4. Faulting in nearby sedimentary rocks: maximum stress parallel to line of intersection between fault plane and bedding.
5. One dominant joint set: maximum stress lies in plane of joint set; minimum stress perpendicular to it.
6. Orthogonal joint sets: maximum stress parallel to line of intersection of the two most prominent joint sets; minimum stress perpendicular to most prominent set.
7. Pervasive foliation in metamorphic rocks: maximum stress lies in the plane of the foliation.

Formulas for In Situ Stress Estimation. Hoek and Brown (1980) have provided limits for the ratio of the horizontal to vertical stresses based on numerous field measurements in several continents, as follows:

$$100/h + 0.3 \leq K \leq 1500/h + 0.5 \qquad (10.7.12)$$

where K is ratio of horizontal to vertical stress, and h is depth in meters.

For the Canadian Shield, Herget (1980) has found the following relationships for the vertical and horizontal stresses:

$$\sigma_v = 0.0243h \text{ MPa } [\sigma_v = 1.074H \text{ psi}] \qquad (10.7.13)$$

a. 0-900 m depth: $\sigma_{ha} = 9.86 + 0.0371h$ MPa
[0-2953 ft depth: $\sigma_{ha} = 1430 + 1.64H$ psi]
b. 900-2200 m depth: $\sigma_{ha} = 33.41 + 0.0111(h\text{-}900)$ MPa
[2953-7218 ft depth: $\sigma_{ha} = 4844 + 0.486(H\text{-}2953)$ psi]

where h is depth in meters, H is depth in ft, σ_v is vertical stress, and σ_{ha} is average horizontal stress. Measurements by others have shown that Herget's relations are generally valid for formerly glaciated portions of eastern North America, including the Great Lakes Basin. (Note: Herget's relationship for vertical stress approximates the value using the superincumbent load. Also the lower limit for depth for the horizontal stress relations is the greatest depth at which measurements have been made.)

In situ stress is difficult to measure accurately, due to the basic physical principle that the act of measurement disturbs the measuring environment. The drilling of a hole to access the rock mass disturbs it and perturbs the stress field, as would driving a drift for access, or excavating a cut. Further, the fracturing usually present in a rock mass results in local rearrangements and variations in the stress field that are being measured by small instruments. Measurement of the in situ stress in the rock mass is, nevertheless, important for mine design.

Methods of measurement consist of (1) measuring the change in dimension of a stress-relieved rock mass, (2) measuring the pressure required to restore the rock mass to its near-initial position, or (3) measuring the pressure required to overcome the stress and fail the rock internally. The different methods include flat-jacks, borehole deformation strain cells, doorstopper gages, inclusion strain cells, inclusion stress meters, and hydraulic fracturing.

In situ stress measurements are dealt with in Chapter 10.3.

10.7.3.2 Effects in Surface Mines

Effects of in situ stress in surface mines arise from abnormally high stresses. As the rock is excavated, the weight of the rock is removed, and the remaining rock in the walls and the floor rebounds and deforms into the excavation. The result is loosening of the rock and, in competent strata, uplift and detachment of the immediate floor. The effect on the rock mass has also been noted as an increase in water-well yield near surface mines that is attributable to stress relief in the rock mass and the consequent opening of fractures in aquifers (Robertson et al., 1980). In mid-continental United States, it is common for the floors of quarries to be humped up and fractured in the center, due to this rebound effect. In New England granite quarries, the

Table 10.7.2. Stress Concentrations Around Openings Under Uniaxial Vertical Stress

		Stress Concentration (σ/σ_v)	
Shape	Height/Width	Roof	Side
Ellipse	0.5	−1.0	5.0
Oval	0.5	−0.9	3.4
Rectangle (round corners)	0.5	−0.9	2.5
Circle	1.0	−1.0	3.0
Ellipse	2.0	−1.0	2.0
Oval	2.0	−0.9	1.6
Rectangle	2.0	−1.0	1.7

floor may suddenly pop up due to high horizontal stresses and the relief of overlying weight (Obert and Duvall, 1967).

10.7.3.3 Effects in Underground Mines

In unsupported mines, particularly room and pillar mines, stress effects are manifested by stress-relief phenomena such as pillar slabbing and floor heave. In a 1200-ft (370-m) deep Gulf Coast underground salt mine, where the senior author was once employed, when subjected to little traffic, the floors of rooms would spring up, and a salt layer about 3 in. (7.5 mm) thick would detach leaving a small air-gap beneath. Walking on the floor was similar to walking on the skin of a drum, complete with booming steps and vibrations. High horizontal stresses can result in the roof failing by popping down or shearing—the roof strata act similar to a fixed-end beam subject to a lateral load, and deflect downward and fail.

Lack of stress due to the topographic or tectonic setting, or shallowness, can also lead to problems in unsupported mine roofs because roof strata tend to sag under their own weight when unsupported, and the stress level may be insufficient to counteract this downward movement. Natural fractures in the roof (or new fractures) will open because they are not held together, resulting in a roof fall. If stress is lacking in one direction, such as perpendicular to a mountainside, the roof fractures parallel to this direction will open and lead to failures.

In more highly stressed rock, or in rock that has had its stress increased by a high extraction ratio, deformation and failure will eventually occur. This is especially true near the corners of square and rectangular openings whose corners are sharp. Goodman (1980) and Duvall (1976) provide stress concentration factors for some typical opening shapes under a uniaxial, vertical stress. Although this stress state is unrealistic, it is, at the same time, instructive; hence, these factors are presented in Table 10.7.2.

Bray (1967) and Goodman (1980) provide a means of calculating the distance from the center of a circular opening in weak or ubiquitously fractured rock (subjected to an isotropically lithostatic stress field) to the boundary between the "plastic zone" of failed rock and the intact rock beyond, as follows:

$$R = a[(2\sigma_o - S_c + (1 + \tan^2(45 + \phi/2))S_j\cot\phi_j)/[1 + \tan^2(45 + \phi/2)(p_i + S_j\cot\phi_j)]^{1/Q} \qquad (10.7.14)$$

in which:

$$Q = [\tan(45 + \phi/2)/\tan((45 + \phi/2) - \phi_j)] - 1$$

and where a is radius of the circular opening, σ_o is primitive stress ($= \sigma_v$), S_c is unconfined compressive strength of the intact rock, S_j is cohesion of the joints, p_i is the support pressure provided to the walls of the opening by internal supports, ϕ is angle of internal friction of the intact rock, and ϕ_j is angle of internal friction of the joints. Although there are many idealized assumptions inherent in Eq. 10.7.14, it is useful for approximating the effects of stress state, rock strength, support pressure, and rock bolt lengths in fractured ground. Goodman (1980) provides estimates of support pressure p_i for typical support methods, as follows:

1. Rock bolts: 0 to 80 psi (0 to 0.6 MPa)
2. Shotcrete: 50 to 200 psi (0.3 to 1.4 MPa)
3. Steel sets: 0 to 400 psi (0 to 2.8 MPa)
4. Concrete lining: 100 to 500 psi (0.7 to 3.5 MPa)
5. Steel liner plates: 500 to 3000 psi (3.5 to 21 MPa)

Rock-bolt support pressures may be found by determining the yield strength of the chosen bolt in pounds and dividing by the square of the bolt spacing in inches (if a square pattern). For instance, number 5 (5/8-in. or 16-mm diameter), grade 36 rebar has a yield strength of 11,000 lb (76 MPa) ($36{,}000[\pi(0.625)^2/4]$) and with a 5-ft- (1.5-m-) square spacing has a support pressure of $11{,}000/(5 \times 12)^2$ or 3.06 psi (21 kPa).

10.7.4 SOIL/ROCK INTERACTIONS

10.7.4.1 Mixed Failure Modes

Soil and rock are very different materials. The key differences lie in the physical properties. Soils are generally unfractured—exceptions are near-surface clays. Most soils are cohesionless, or have small amounts of cohesion. Unlike rocks, soils do not "bulk" to a significant extent when disaggregated or dug. On the other hand, some "rocks" behave much as do soils. Deeply weathered rocks—sedimentary, igneous, or metamorphic—can behave just as soils do. Indeed, the authors are familiar with a "granite gneiss" in northwestern New Jersey that failed as a soil slope, but examination of the failure area backwall revealed all of the "crystal structure," "joints," and "veins" (Kendorski and Pincomb, 1991). In another instance in Colorado, triple-tube coring successfully cored highly altered rhyolite porphyry sills that looked just like wet rock cores when removed from the inner barrel, but could be squeezed in the hand like mud. Kendorski, Myers, and Grenot (1990) describe a highly weathered soil-like shale that underlies unaltered quartzite, yielding a mixed-mode situation.

The line between rock and soil is blurred. Neither properties alone nor genesis decide, because sedimentary rocks are consolidated soils whereas soils are decomposed or disaggregated rock. Farmer (1983) makes a distinction as follows:

> "Both rocks and soils are made up of mineral and organic particles. In the former, the particles are generally bonded or cemented together and an initial yield resistance must be overcome before they shear in an unconfined state. Soils exhibit no *real* resistance to shear in an unconfined state, and a very small energy input is required to precipitate breakdown." [emphasis in original]

Where soils overlie rock (the usual case), rock slope failures can trigger soil failures or extend into soil slopes. A wedge-type failure in rock becomes a slip-circle or single-plane failure when extended into soil.

In a streetcar tunnel in San Francisco, Kendorski (1982) found that a concrete-timber lining had failed in a peculiar way at the quarter-point on the upper arch as a result of a sloping rock/soil interface that dipped across the tunnel while the tunnel followed its strike. The two different materials applied two entirely different loads to the tunnel lining, which had been "designed" in 1913 to accept uniform loads. The failure occurred shortly after construction, but did not hinder the tunnel performance until federal funding in 1975 required the structure to be upgraded to current standards.

The case presented by Kendorski, Myers, and Grenot (1990) is an interesting one in that the formations dipped at 70° with quartzite overlying deeply weathered shale. When the quartzite was exposed, the rock slope behaved as a slope in rock, with planar failures. When the shale was exposed, usually after collapse of the quartzite, the rock slope behaved as a soil slope, with slip-circle failures. With both materials present, a certain minimum thickness of quartzite must be left in place to prevent the shale behind it from pushing out the quartzite and causing failure in both.

When drawing ore underground, or in sinkhole development during subsidence, the soil properties of minimal bulking and little or no shear strength lead to flooding of soil downward, because soil flow will not stop until the collapsed area has been filled. In rock, the 40% or so volume increase upon breakage will limit progressive collapse because the bulked rock fills the void, providing support against further collapse.

Mixed failure modes result from rock and soil in contact subjected to the same excavation or to the consequences of failure of one material affecting the other. Rock will behave as a rock; soil will behave as a soil. The effects of minimal bulking and little or no shear strength in soil as compared to substantial bulking and significant shear strength in rock is profound. As the previous case histories illustrate, the designer must recognize the likely failure modes and consequences of failure in the design. Recognition of the failure mode (or modes) is the key, because then design can proceed rationally.

10.7.4.2 Mixed-face Excavation

Mixed-face excavation represents an important problem because a combination of excavation and support techniques must be employed. The magnitude of the problem depends on the relative proportions of hard and soft materials encountered, and whether the soft material lies above or below the hard material. Moreover, the problems facing a geomechanics practitioner are different for surface and underground excavations, and will, in what follows, be dealt with separately.

Mixed-face Surface Excavations, In surface excavations, the critical factor is the stability of the excavated slope. In general, the soil will overlie the rock in the excavated face (unless the "soil" is gouge in a shear zone), in which case, the simplest strategy is to excavate the soil before beginning rock excavation.

The best method for excavating the soil in a given situation depends on a number of factors including the topography, type of soil, and groundwater conditions. A detailed discussion of the types of excavating equipment available and the advantages and disadvantages of each is beyond the present scope. For such information, the reader is referred to Nichols (1976), Church (1981), or Peurifoy and Ledbetter (1985). Design slopes should not exceed 1.5:1 (horizontal to vertical). In addition, a 50-ft (15-m) berm should be left between the toe of the soil slope and the crest of the rock slope at the top of rock.

In cases where the soil layer underlies a rock stratum, the situation is more complex, and a detailed slope stability investigation is advisable. The critical elements are the attitude and shear properties of the soil layer and the inclination of the final slope (Kendorski, Myers, and Grenot, 1990). For methods of slope stability analysis the reader is referred to Chapter 10.4 and to slope stability monographs such as *Rock Slope Engineering* (Hoek and Bray, 1974).

Mixed-face Underground Excavations. The underground excavations most likely to encounter mixed-face conditions are shallow tunnels. Mixed faces occur as a result of intersecting (1) seams of decomposed rock, (2) ancient erosion surfaces (usually river or stream beds that have filled with sand, gravel, or clay), (3) fault zones, and (4) different strata (if the dip of the strata is in the direction of tunnel advance). If the top of the excavation is in soft ground and the bottom is in rock, the result is one of the most difficult excavation situations that can be encountered—considerably more difficult than if the excavation were completely in soft or hard ground (Wilbur, 1982).

When a mixed face is encountered in a rock tunnel, excavation and support methods must usually be radically revised. The most difficult problems arise when a mixed face is encountered in a machine-driven excavation, because machines designed for hard rock generally do not work in soft ground. Furthermore, the very fact that a machine is being used implies that the mixed-face conditions were unexpected. As a result, long delays can be expected while the machine is removed and other methods of excavation are substituted. Worse, if the tunnel is being excavated by a contractor, large claims for changed conditions can be expected.

The situation is considerably simpler if mixed-face conditions are anticipated during the exploration phase. In this case, borings should be used to attempt to locate potential mixed-face zones. During excavation, exploratory drillholes should be carried ahead of the face as suspected mixed-face zones are approached.

Excavation and support of soft ground in the crown of a mixed-face tunnel should utilize the same techniques that would be used if the complete face were in soft ground. The critical consideration is whether the soft ground is dry or wet. If the ground is dry, temporary support consisting of steel poling plates, spiling, or a shield can be used to hold the crown until more permanent support such as liner plates or ribs and tight lagging can be installed. If the ground is wet, such methods can also be used if the soil can be consolidated by dewatering or grouting. Freezing may be practical in wet, running sands, whereas in wet, running, fine sands and silts, shield tunneling under compressed air may be necessary. In all cases, care must be taken when excavating the rock portion to ensure that the integrity of the soft-ground and its support is not compromised.

10.7.5 SPECIAL TOPICS

10.7.5.1 Problems at Great Depth

Definition of Great Depth. From a geomechanics standpoint, *great depth* refers to distances below surface where creation of an opening results in an overstressed condition in the surrounding rock, resulting in failure or yield. Thus the depth at which a mine may be considered deep depends on the strength of the rock mass. In very strong, brittle rocks such as metaquartzites or granites, mines are not generally considered deep until their workings extend 6000 ft (1800 m) or more below surface. Mines in this category include gold mines in South Africa, India, and Ontario, silver mines in Idaho, and nickel mines in Ontario. By contrast, coal and salt rock mines can be considered deep when they extend more than 1500 ft (450 m) below the surface. For mines in rock intermediate in strength, the criterion for a deep mine lies somewhere between these two extremes.

Geomechanics Behavior in Deep Mines. Openings in "deep" mines in weak, ductile (or pseudo-ductile) rocks, such as salt rock, shale, or bituminous coal, are characterized by viscoplastic (or pseudo-viscoplastic) deformation of the surrounding rock. This behavior serves to mitigate the effects of the high stresses. In strong, brittle rocks, however, strain energy resulting from excavation is not dissipated through viscous flow, but is stored in the rock until a limit is reached at which failure occurs in an explosive manner. Such explosive failures, which are called rockbursts, are small-scale seismic events—micro-earthquakes, if you like—and, like earthquakes, are accompanied by emission of acoustic energy. It is this behavior of strong, brittle rocks under large stresses that is generally associated with the phrase "problems at great depth." (Note: where extraction ratios approach 100%, rockbursts can occur at relatively shallow depths due to catastrophic failure of improperly designed pillars.) Hence the discussion hereafter will concern the causes and control of rockbursts.

Brief History of Rockburst Problems. As early as 1900, rockbursts had become a serious problem on the Kolar Gold Fields in India (Taylor, 1963). In South Africa, the first instance of earth tremors attributed to mining occurred in 1908 (Anon., 1977), although doubtless small bursts that were not felt at surface had occurred in the mines at a somewhat earlier date. Commissions were appointed to study the problem in South Africa in 1908, 1915, and 1924. The 1908 and 1915 Commissions attributed the cause to failures in support pillars and remnants. The 1924 Committee did not discuss the causes but rather directed attention to control methods and potential mechanisms. Even at such early dates, there was an awareness that the problem was related to the sudden release of strain energy (Anon., 1977).

In Ontario, rockbursts first occurred in 1928 or 1929, but their occurrence did not become a cause of anxiety until 1934. In 1940, R.G.K. Morrison, then superintendent of a Kolar gold mine in India, was engaged by the Ontario Mining Association to study the rockburst situation in Ontario. His report (Morrison, 1942) remains a classic discussion of the problem. In the Coeur d'Alene district in Idaho, rockbursting was first reported in 1956 (Blake, 1972a).

In spite of the early recognition of the problem, and the evident early qualitative understanding of the mechanism, it is only in the past 25 years that numerical stress analysis methods have been developed to evaluate the problem. Nevertheless, the early attempts to develop rockburst mitigation and control methods demonstrate considerable insight. The salient features of these attempts are the recommended modifications to mining methods and layouts, many of which remain valid. The use of longwall mining, and the minimization of the number of pillars and remnants, practices useful in mines having flat-lying or shallow-dipping orebodies, were proposed in South Africa as early as 1924 (Anon., 1977), and longwall mining was introduced on Crown Mines about 1930 (Pretorius, 1975) and on East Rand Proprietary Mines about 1940 (Ortlepp and Steele, 1975). For steeply dipping ore bodies, longwall mining is impractical, and other measures related to stope sequencing were proposed. These latter recommendations can be found in Morrison (1942). Oliver (1975) has demonstrated the effectiveness of these practical measures over a period of more than 30 years in the mines of INCO Metals Ltd. in the Sudbury District, Ontario.

Prediction of Rockburst Potential. The most important consideration in evaluating rockburst potential is the amount of energy released in connection with specific changes in mining geometry (Salamon, 1974, 1983). The energy quantity to be expended is the sum of the change of potential energy due to displacement of the perimeter of the opening, and the strain energy stored in the rock removed in the particular mining step. Part of this energy is accounted for by an increase in the energy stored in the surrounding rock. In supported excavations, some of this energy is also dissipated in deforming the supports. The change in excess energy due to a particular change in mining geometry is given by (Salamon, 1974, 1983):

$$\Delta W_r = \Delta U_m = (\textstyle\int_{\Delta A} T_i^{(p)} \Delta s_i dA)/2 \tag{10.7.15}$$

where ΔW_r is change in excess energy, ΔU_m is change in strain energy stored in rock removed by mining, $T_i^{(p)}$ is traction vector in the initial state acting on the surface A_m to be exposed by mining, Δs_i is change in the relative displacement (closure or ride) vector, and A is mined-out area of the stope.

The potential for rockbursting is minimized by minimizing the energy release rate, which is defined by (Salamon, 1984) as:

$$ERR = \Delta U_m/\Delta A \tag{10.7.16}$$

where ERR is the energy release rate, and the other variables are as defined previously. From Eq. 10.7.16, it is apparent that for a fixed volume of rock broken per round (ΔA), reduction in the energy release rate requires reduction of the change in excess energy associated with the mining step. From Eq. 10.7.15, this reduction can be accomplished by reducing either (1) the relative displacement or (2) the stress concentration at the surface to be exposed. The first step, therefore, is to evaluate the stresses and displacements due to several alternative mining sequences, choosing the one that minimizes the change in excess energy. Further reduction can be obtained through the use of backfill (provided the backfill is kept close to the face) as fill reduces the closure. In theory, the stiffer the fill (i.e., the higher the deformation modulus), the greater is the effectiveness; and a number of papers have been published (Van Eeckhout, 1984; Stout and Friel, 1980) that advocate stiff fills that are essentially lean, unreinforced concretes. Nevertheless, a contrast exists between the deformation modulus of the fill and that of the surrounding rock, and because of this contrast, very little of the stress is transferred from the surrounding rock to the fill, regardless of the stiffness of the fill. (A significant stress transfer might only occur if the opening were backfilled with fitted blocks of the original rock, as was once the practice in Cornwall, England, and on the Kolar Gold Fields). Moreover, because the rocks are elastic and brittle, the deformations are small unless failure has occurred so that greater reductions in energy are likely through reduction of the stress concentration by destressing. Destressing is as yet an art rather than a science, and is generally accomplished by extending drillholes some distance beyond the mining face, and blasting with spherical charges at the bottom of the holes. This practice has been used with apparent success at a number of mines including the Galena mine on the Coeur d'Alene (Blake, 1972b; Anon., 1979) and deep gold and nickel mines in Ontario (Anon., 1979). For more information on rockburst control, consult the various *Proceedings of the Symposium on Rockbursts: Prediction and Control.*

Detection of Strain Energy Buildup. Buildup of stored strain energy is accompanied by acoustic emissions. Thus microseismic monitoring systems can indicate if energy is building to potentially unsafe levels. Through the use of a multichannel system with accelerometers arrayed throughout the mine, the location of events can be determined. Event location algorithms have been discussed by Eccles and Ryder (1984), and recently a different approach based on the simplex algorithm has been proposed by Riefenberg (1989). Design criteria for monitoring systems have been discussed by Blake (1979, 1984), among others. For more information on rockburst detection, see the *Proceedings of the First International Symposium on Rock Bursts and Seismicity in Mines.*

10.7.5.2 Effects of Method of Excavation

The process of excavation results not only in the removal of rock, thereby creating an opening, but also in damage to the surrounding rock. Different methods of excavation have varying effects, and in situations where preservation of the integrity of the adjacent rock is of paramount importance, the method of excavation must be selected with the consequent damage in mind.

With normal blasting practices, the rock can be damaged (i.e., fractured and weakened) to a depth of 10 to 15 ft (3 to 4 m). This damage is caused by compressed gases in the blastholes that propagate cracks in the surrounding rock. By partially or fully decoupling the explosive from the blasthole, the detonation pressure is reduced, with a consequent reduction in the thickness of the damaged skin of rock. Blasting studies have shown (Hansen, 1976) that complete fracturing of a rock mass occurs for peak particle velocities (ppvs) in excess of 100 in./sec (2.54 m/s). For depths into the rock greater than about 3 ft (1 m), the ppv can be found from scaled-distance relationships. The following relationship has been found to give an upper bound to the ppv for blasts using cylindrical charges (Hendron and Oriard, 1972):

$$v = 130\ (R/W^{1/2})^{-1.57} \qquad (10.7.17)$$

where R is distance from the blasthole in ft, and W is charge weight per delay in lb. Similarly, a lower bound for ppvs from blasts in cylindrical holes is given by Dowding (1985):

$$v = 25\ (R/W^{1/2})^{-1.57} \qquad (10.7.18)$$

For most rocks, the actual ppv will fall within these limits. Substituting v = 100 in./sec (2.54 m/s) into Eqs. 10.7.17 and 10.7.18, and rearranging, we obtain a range of:

$$0.11\ W^{1/2} \leq R \leq 1.51\ W^{1/2} \qquad (10.7.19)$$

for the predicted limit of the depth of damaged rock. By substituting the charge weight per delay (in lb) for a particular blast into equation 10.7.19, the predicted depths of failed rock (in ft) for that blast can be found.

With mechanical excavation, the depth to which damage extends is much smaller. The reason is that the pressure applied to the rock is much smaller than the explosive detonation pressure, and there are no gases that need to expand to reduce their pressure. For roadheaders, the stress on the wall can be calculated from the Boussinesq solution for a force applied normal to a semi-infinite plane, if the normal component of the cutter thrust is known. In the case of tunnel boring machines (TBMs), and full-face and transverse drum continuous miners, the stress on the wall can be calculated from the Cerruti solution for a tangential force on a semi-infinite plane, if the cutter thrust is known. (The Boussinesq and Cerruti solutions can be found in elasticity texts such as Timoshenko and Goodier [1970] or Boresi and Chong [1987].) In either case, the depth to which the thrust results in significant stress is no more than a few inches (tens of millimeters). Moreover, the stress dissipates as soon as the cutterhead has advanced as little as 1 to 2 in. (25 to 50 mm). More important in the case of mechanically excavated openings is the limit of failed rock that results from the redistribution of stress around the opening. This limit can be found from elastic/plastic stress analysis, in the case of circular openings (see Eq. 10.7.14), if the failure criterion and failure parameters are known.

From the above discussions, it is clear that mechanical excavation is preferable to blasting from a geomechanics standpoint. However, the practicability of mechanical excavation depends upon other factors such as the rock strength, the size of the opening, and the length of the heading. With the exception of TBMs, mechanical miners are limited to rocks having compressive strengths less than 15,000 psi (100 MPa). Conversely, TBMs can excavate even the strongest rocks but are not cost-effective for headings shorter than about 3000 ft (900 m). Strong but fractured rock is very difficult to excavate by TBM because rock blocks tend to fall out of the face and get caught in the cutterhead, breaking cutters off. Thus drill-and-blast excavation may be the only practical alternative. However, controlled blasting techniques can be used to minimize damage. The available controlled blasting techniques include:

1. Presplitting.
2. Post-splitting.
3. Smooth blasting (using trim powder in the perimeter holes).
4. Dental excavation (blasting small, specially designed rounds over partial faces) in extreme cases.

These techniques were discussed in 10.7.2.3, and descriptions of some or all of these techniques are readily found in blasting handbooks as well as Chapter 9.2.1. In cases of contract mining, the use of these techniques will require close supervision of the contractor in order to assure that the desired result is achieved. (One can also hedge one's bets by putting close tolerances on the pay-line that defines payment for excavation.)

10.7.5.3 Evaporites

Before discussing the behavior of evaporites and the design of openings in evaporites, we shall first define what evaporites are, and introduce the minerals that fall into this classification.

Evaporites comprise all mineral deposits that are precipitated from solutions concentrated by evaporation. Although about 80 minerals have been identified in evaporite deposits, only about a dozen are sufficiently common to constitute important rock-formers (Boggs, 1987). Evaporites can be subdivided into those of marine origin and those of nonmarine origin. If carbonates are excluded, the most common marine evaporite minerals are anhydrite and gypsum. Halite is next in abundance followed by the potash salts sylvite, carnallite, langbeinite, polyhalite, and kainite, and the magnesium salt kieserite. The order of abundance derives directly from the order in which the minerals are precipitated.

Nonmarine evaporites are characterized by minerals not commonly found in marine evaporite deposits, although anhydrite, gypsum, and halite may also be present. The nonmarine evaporite minerals include trona, mirabilite, glauberite, borax, epsonite, thenardite, gaylussite, bloedite, and nahcolite (Boggs, 1987).

Design of Openings in Salt Rock. In the limited space available here, it is not possible to provide an exhaustive discussion of the rationale for, or derivations of, the design equations that are presented in this segment. For such information, the reader is referred to the references provided as well as references cited therein. It must also be noted that all the design equations in what follows assume that the primitive stress is uniform in all directions. This assumption is reasonable in the interiors of salt domes and thick, uniform, bedded deposits, but may not be accurate near contacts with surrounding strata in such deposits or in thin bedded deposits overlain and underlain by non-salt rocks.

To begin with, salt rocks are relatively weak. The compressive strength of halite in laboratory specimens—a meaningless parameter for design, but a useful parameter for comparisons with other rocks—typically ranges from 3000 to 6000 psi (20 to 40 MPa). Sylvite and carnallite, the most common potash minerals, are weaker still. Thus openings in salt rocks are surrounded by the following four zones (in order of increasing distance from the opening):

1. Brittle failed zone resulting from dynamic excavation stresses.
2. Yielded viscoplastic zone that may not exist at shallow depths.
3. Viscoelastic zone.
4. Undisturbed quasi-elastic zone.

Isolated single openings will be surrounded by all four zones. Properly designed yielding pillars contain the first two zones, and conventional pillars (and abutment pillars bordering panels with yielding pillars) contain the first three zones. The failed zone eventually separates from the surrounding rock and thus does not contribute to the stability. Mraz (1973, 1980, 1984) has shown that this failed zone can be ignored for calculation purposes if the opening is replaced by an effective equivalent elliptical opening that circumscribes the actual opening. The semi-width and semi-height of the ellipse is given by (Hambley, 1991):

$$d^2 = x^2 + xy \tag{10.7.20}$$

$$b^2 = xy + y^2 \tag{10.7.21}$$

where d is semi-width of the effective ellipse, b is the semi-height of the effective ellipse, and x and y are coordinates of a selected "corner point." The so-called corner point is either a point on the corner of the actual opening, or a point taken a small distance radially into the rock from it to account for the damaged zone. For a rectangular opening with rounded corners, the coordinates of the corner point on the perimeter of the opening are given by (Hambley, 1991):

$$x = (W/2) - R(1 - 1/\sqrt{2}) \tag{10.7.22}$$

$$y = (H/2) - R(1 - 1/\sqrt{2}) \tag{10.7.23}$$

where H is the height of the opening, W is the width of the opening, and R is the radius of curvature of the corner.

Considering a von Mises yield criterion (von Mises, 1913; Jaeger, 1969), which assumes that the deviatoric stresses are responsible for yield, the depth of the yielded zone in the roof is given by (Hambley, 1989):

$$M_{pv} = a_v\{\exp[(3^{1/2}\sigma_o/2k - 1)/2] - 1\} + b - H/2 \tag{10.7.24}$$

where σ_o is primitive stress, k is shear strength at zero mean stress (Prandtl limit), m_{pv} is depth of the yielded zone at the midpoint of the roof, and a_v is vertical radius of curvature of the ellipse. The formula for the vertical radius of curvature of the ellipse is given in mathematics handbooks (e.g., Tuma, 1987). The above formulas assume that the opening is in an infinite mass of salt rock. If there is a competent layer in the roof at a depth less than the depth of the yielded zone calculated from Eq. 10.7.24, it will be necessary to modify the shape of the effective ellipse. Procedures for so doing are given by Hambley (1989). The stress distributions along horizontal and vertical planes are estimated using a circular approximation to the effective ellipse.

In the yielded, viscoplastic zone, the radial and tangential stresses are given by the following expressions, assuming that the von Mises yield criterion is valid (Hambley, 1989):

$$\sigma_r = (4k/\sqrt{3})\ln(r/a) \tag{10.7.25}$$

$$\sigma_t = (4k/\sqrt{3})[1 + \ln(r/a)] \tag{10.7.26}$$

where σ_r is radial stress, σ_t is tangential stress, k is the Prandtl limit (shear stress at zero mean stress), a is radius of curvature of the ellipse, and r is radius to point at which stress calculated. The Prandtl limit can be estimated from laboratory creep tests but should be confirmed by underground stress and deformation measurements. For descriptions of laboratory creep testing procedures, Dusseault et al. (1985) should be consulted. For a description of the underground verification of the Prandtl limit, the reader is referred to Mraz and Dusseault (1986).

For a single opening, the stresses in the viscoelastic zone are given by (Hambley, 1989):

$$\sigma_r = \sigma_o(1 - r_f^2/r^2) + (4k/\sqrt{3})(r_f/r)^2 \ln(r_f/a) \tag{10.7.27}$$

$$\sigma_t = \sigma_o(1 + r_f^2/r^2) - (4k/\sqrt{3})(r_f/r)^2 \ln(r_f/a) \tag{10.7.28}$$

where r_f is the radius of the viscoelastic/viscoplastic interface. Other variables are as defined previously. The radius of the interface is found from:

$$r_f = a \exp[(3^{1/2}\sigma_o/2k - 1)/2] \tag{10.7.29}$$

where all variables are as defined previously.

For pillars between openings, the size should be selected so that the viscoplastic zones around the two openings just meet at the center of the pillar if a panel mining system with yielding pillars and bordered by barrier pillars is employed. If a conventional room and pillar system without barrier pillars is employed, then there should be viscoelastic cores in the pillars. Assuming that a viscoelastic core exists and that the steady-state value of Poisson's ratio is 0.5, the stresses in the viscoelastic zone are given by (Hambley, 1989):

$$\sigma_r = [\sigma'_v\, b^2/(b^2 - r_f^2)](1 - r_f^2/r^2) + 4k\ln(r_f/a)(b^2 - r^2)r_f^2/[3^{1/2}(b^2 - r_f^2)r^2] \tag{10.7.30}$$

$$\sigma_t = [\sigma'_v\, b^2/(b^2 - r_f^2)](1 + r_f^2/r^2) - 4k\ln(r_f/a)(b^2 + r^2)r_f^2/[3^{1/2}(b^2 - r_f^2)r^2] \tag{10.7.31}$$

where b is radius from the center of curvature to the center of the pillar, and σ'_v is average pillar stress determined by tributary area (Bieniawski, 1984b):

$$\sigma'_v = \sigma_v^0\,(W + B)(L + B)/(WL) \tag{10.7.32}$$

where σ'_v is average pillar stress, σ_v^0 is primitive vertical stress, W is pillar width, L is pillar length, and B is width of the opening.

In the case of pillars, the radius of the viscoplastic/viscoelastic interface must be found by trial and error or successive approximation, using the formulas for the tangential stresses in Eqs. 10.7.26 and 10.7.31. Some researchers claim that the Tresca criterion is more representative of yield in salt rock. If the Tresca criterion (Tresca, 1868; Jaeger, 1969), which assumes that yield occurs at a critical value of the maximum shearing stress($\sigma_1 - \sigma_3$), is chosen, Eqs. 10.7.24 through 10.7.31 do not apply. Instead, equations given by Hambley and Mraz (1987) can be used for single openings.

The equations presented above strictly apply only for long rib pillars. For rectangular or square pillars, some modifications are required as shown by Hambley (1989).

Steady-State Strain Rates. At steady state, the rate-controlling mechanism in the viscoelastic and viscoplastic zones appears to be Pressure-Solution creep. The strain rate is given by (Hambley, 1991a):

$$\dot{\epsilon}_{ij} = 21\ \pi\delta D_1\Omega s_{ij}/[kTd^3] \qquad (10.7.33)$$

where $\dot{\epsilon}_{ij}$ is strain rate in the *ij* direction, δ is grain boundary width, D, is effective diffusivity of solute in the fluid, Ω is molecular volume, s_{ij} is deviatoric stress in the *ij* direction, *k* is Boltzmann's constant, *T* is absolute temperature, and *d* is grain size (diameter).

Transient Creep Behavior. During transient creep, the strain rate decreases so that a plot of cumulative closure vs. time has a logarithmic shape (see Fig. 10.7.7). Empirical relationships can be used to estimate the closure once closure data from a prototype opening having identical dimensions are available. Conversely, predictions of closure behavior based on laboratory creep tests are unreliable. Hambley (1990b) has developed a method of predicting closures that provides reasonable agreement with measured values for opening perimeters but progressively poorer agreement with borehole extensometer data as the depth into the rock increases. Because of the similarity between physical properties of salt rocks from widely different locations, this method can be used even when parameter data for a specific location are unavailable. (There is insufficient space here to present the salient features of the method. The interested reader is referred instead to Hambley [1990b].) Closure predictions are not, however, a substitute for in situ measurements; however, after closure predictions for a given opening geometry have been confirmed by measurements, the prediction equations can be used with some confidence to predict the behavior of openings of other geometries.

Design of Shafts for Mines in Salt Rock. To this point, we have concerned ourselves with the design of rooms and pillars in salt rock. However, because of the behavior and properties of salt rock, shaft design also differs considerably from that in other rocks.

In the first place, because salt rocks are soluble in water, water must not be allowed to reach the salt rock via the shafts. Hence shafts must be watertight, especially because salt rocks are generally overlain by water-bearing sedimentary rock or sediments. The most effective linings for water-bearing zones consist of either (1) concrete-backed cast iron tubbing rings, or (2) a steel-and-concrete sandwich, which is sometimes separated from a primary lining of concrete blocks by a bitumen membrane. In both cases, the lining should be supplemented by curtain grouting behind it. Procedures for designing these linings are found in Ostrowski (1972) and Link et al. (1968). One note of caution, however: the design procedures of Link et al. are based on thin-shell theory and should be checked using thick-walled cylinder formulas for composite materials if the thickness of the lining exceeds 10% of the outside radius of the excavation.

If freezing is employed during shaft sinking, the temporary primary lining must be capable of resisting the deformations and stresses in the frozen material. Failure to ensure this can have tragic consequences if the material to be frozen consists of silts or clays.

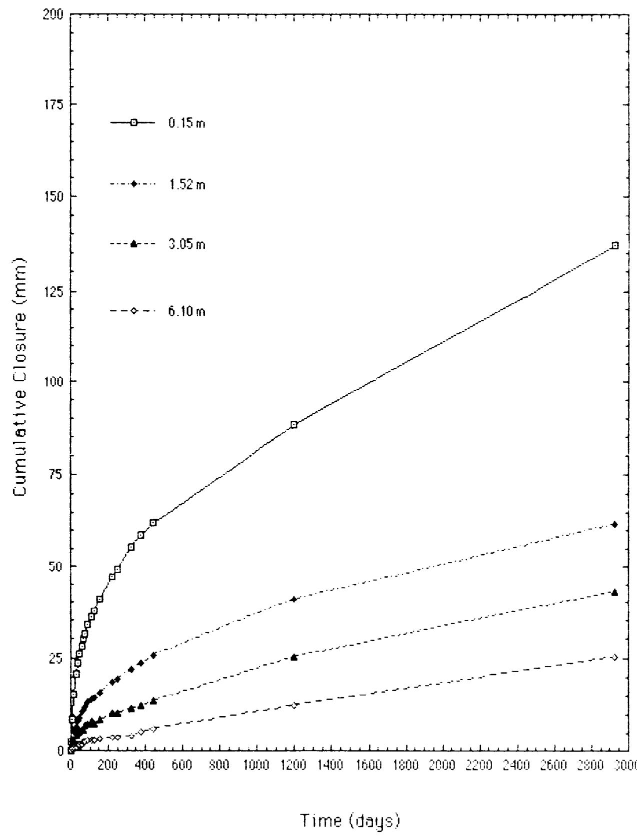

Fig. 10.7.7. Typical cumulative closure vs. time curves for extensometers in the wall of a single opening. Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

10.7.5.4 Nuclear Waste Repositories

The high-level nuclear waste repositories currently under consideration in the United States resemble room and pillar mines and should be considered under these special topics. The construction and initial high-level waste placement period will extend for about 60 to 80 years. However, the engineered barrier systems must perform their functions for more than 1000 years. From a geomechanics standpoint, the critical issues are the long-term performance of the backfill and seals and the integrity of the host rock. (Design of the waste packages and their integrity under repository conditions are problems for materials scientists and corrosion engineers.) The process of predicting the behavior of the barriers under repository conditions is called *performance assessment.*

Prior to licensing of a repository, performance assessments must be able to show with "reasonable assurance" that the barrier system will provide the containment and isolation required by the regulatory authorities. Because of the time periods involved, performance assessment necessarily entails numerical modeling. The use of numerical modeling in turn spawns other issues such as the validity of the models used. The biggest problem, however, is the need to run models in which thermomechanical, hydrogeological, and geochemical effects are coupled. Moreover, our understanding of the processes in each of these areas is incomplete. Thus even with decoupled models that address thermomechanical, hydrogeological, and geochemical conditions individually, there is considerable uncertainty regarding the validity of the predictions. In spite of all efforts and progress that have been made to improve our knowledge, research needs in these areas remain considerable.

Waste repository design is discussed in Chapter 24.3.

10.7.5.5 Design Uncertainty and Risk

Unlike structural engineers who can engineer a material to provide the properties required for a particular application, an engineer designing facilities in natural materials must utilize the host rock to best advantage, taking into account its properties. He or she must design around what nature has provided, which requires, first and foremost, a thorough knowledge and understanding of the properties of the material. This requirement is further complicated by the fact that natural materials are not homogeneous— the properties will vary spatially and even temporally. (Conversely, this lack of homogeneity is the origin for economic mineral deposits. If all the elements in the earth's crust were homogeneously mixed, mining would consist solely of stone quarrying.) As a result, uncertainty and risk always exist concerning the stability of facilities in soil and rock. Thus design involves reducing uncertainty and risk to an acceptable level.

Acceptable uncertainty and risk will vary according to the purpose of the facility. Nuclear waste repositories, for example, must be designed with very low consequent risk. In mine design, use of this same principle could result in inefficient exploitation of reserves and excessive cost, thus reducing the economic viability of an operation. Nevertheless, a well-conceived exploration and design strategy will reduce uncertainty and risk to a level appropriate for the particular facility.

Design Strategy for Reducing Risk and Uncertainty. A well-conceived design strategy comprises the following steps:

1. Site exploration.
2. Physical property testing.
3. Definition of property variability limits.
4. Identification of structural domains.
5. Development of competing designs.
6. Use of multiple support calculation methods.
7. Comparison of alternatives.

Site Exploration: A site exploration program is necessary to determine the range of geologic conditions that can be expected and should include core drilling, outcrop mapping, and possibly geophysical surveys. If there are existing facilities in the strata of interest, they, too, should be examined. For example, in the case of design of a highway tunnel in Colorado, detailed examination of the 50-year-old lining of a railway tunnel a few hundred yards (meters) away facilitated the calculation of the expected loads for the highway tunnel.

Core drilling should include sufficient holes and hole locations to characterize the complete area of the proposed facility. If the facility includes shafts, a core hole should be drilled along the centerline of each proposed shaft to the full depth to be reached by the shaft. Logging of the cores should include not only the stratigraphy but also locations and inclinations of joints, type of joint filling (if any), and the RQD. In the case of tunnels under large bodies of water, core drilling may not be feasible for the underwater sections. However, seismic reflection surveys can be used to provide the necessary information in conjunction with the available cores.

Any available outcroppings of formations traversed by the core drilling should be mapped. This mapping should include detail line surveys (Robertson and Piteau, 1970) in both horizontal and vertical directions within the member to sample the attitudes of the joints. At least 120 observations should be obtained along each detail line.

Another useful exploration technique is excavation of an exploratory adit into the formations of interest. Use of an adit allows a designer to test proposed support techniques and gives a better indication of expected conditions than either cores or outcrop mapping. Detail line surveys should be performed in the adit and compared with the results from outcrop mapping. (Care should be taken to ensure that blast fractures are ignored in the detail mine surveys in the adit.)

Physical Property Testing: Sufficient property tests should be performed on core samples from all formations of interest that the range of property values is determined. The properties of interest are the tensile strength and uniaxial and triaxial compressive strength of the rock substance as well as the shear properties of joints. In some rock types, special properties such as slake durability and creep deformation must also be determined. In porous rocks, permeability or hydraulic conductivity will also be important so that water inflows can be predicted. Sufficient tests of each type should be performed on each rock type that the mean and standard deviation (or perhaps range) of property values is established. It is very unusual in most rocks to have a coefficient of variation (the ratio of the standard deviation to the mean) of less than 25%. (For further information on physical property testing, see Chapters 10.2 and 10.3).

Definition of Property Variability Limits: Because design of rock support or reinforcement systems depends on the properties of the rock itself, it is important that the limits of the variability of the rock properties be known. Specifically, statistical parameters should be determined for each property for each geologic member. Strength properties and permeabilities have generally been found to be log-normally distributed. Spatial properties of the rock mass obey other distributions—joint lengths and apertures follow a negative exponential distribution, joint attitudes obey a Fisher or Bingham distribution, and joint spacings are Poisson-distributed.

After the various statistical parameters have been calculated, expected values and limits can be placed on the properties for design purposes. Design values for the properties can then be chosen to minimize the risk that the design does not perform to expectation. A method of tunnel design using such statistical principles has been used successfully by the authors (Kendorski, 1980; Hambley and Kendorski, 1984).

In slope-stability design practice, probabilistic design methods are widely accepted, and many papers have been presented regarding such techniques. Examples include Call, Savely, and Nicholas (1977); McMahon (1975); Major, Ross-Brown, and Kim (1978); and Glynn, Veneziano, and Einstein (1978).

Identification of Structural Domains: Structural domains are areas of the rock mass that have sufficiently similar properties that they can be considered to behave in a uniform manner. Also the boundaries of the structural domains need not coincide with stratigraphic boundaries—a shear zone that cuts across several different strata might constitute a structural domain. Structural domains may also repeat. For example, for a pumped storage project in Georgia, it was found that the 17 stratigraphic units could be lumped into just two structural domains: one for the stronger members, and another for the weaker members (Grainger et al., 1986).

Development of Competing Designs: A first stage in developing designs is a thorough literature search for case histories exposed to similar conditions. A simpler means of achieving the same end is to designate the rock using rock classification systems. Although a number of classifications have been developed over the years beginning with Terzaghi (1946), two systems are most commonly used: the Q System of the Norwegian Geotechnical Institute (Barton, Lien, and Lunde, 1974) and the rock mass rating (RMR) system developed by Bieniawski (1974, 1989). Both systems are based on a number of case histories (Chapter 10.5). However, the rock types represented by the case histories are quite different. Thus the classification based on the Q System may be more reliable for strong rocks such as gneisses, granites, and quartzites, whereas the RMR system is generally superior for weaker rock types. However, both provide estimates of support requirements for different classes of rock mass.

In many cases, several different support systems could be considered (e.g., rock bolts and wire mesh vs. shotcrete vs. rock bolts, mesh, and shotcrete). Also different types of bolts might be considered: mechanical bolts, grouted bolts, or grouted cables.

Another type of alternative that arises with tunnels, but not usually in mines, is different vertical alignments within the rock mass. For example, the initial alignment of a highway tunnel was located in a claystone with the crown about 10 to 12 ft (3 to 3.6 m) below a massive sandstone. For this alignment, a horseshoe tunnel with steel sets on 4-ft (1.2-m) spacings was considered viable. However, if the alignment were raised so that the crown fell at the base of the sandstone, a rectangular tunnel with rock bolt support was feasible. Based on these alternative alignments, a number of alternative excavation and support sequences were identified.

Another type of alternative design strategy is incorporated in the New Austrian Tunneling Method, discussed in 10.7.2.2.

Use of Multiple Support Calculation Methods: If the use of rock bolts is contemplated, the spacings suggested by the classification systems can be checked against empirical guidelines based on the RQD index developed by Deere et al. (1969). Bolt design is covered in Chapter 10.5.

Rock bolts are commonly installed on standard patterns, the most common being radial bolts where the roof is arched and bolts perpendicular to the roof where the roof is flat. However, such standard patterns may not effectively reinforce critical joint sets. Alternatively, the direction of the bolts can be based on the directions of the critical joint sets, using a kinematic analysis on a Schmidt equal-area stereonet containing joint attitude probabilities calculated by the method given in Goodman (1976). Bolts should also be checked for their resistance to sliding forces on the worst case joint inclination, and to en echelon failure along a series of joints. Examples of such analyses are given in Hambley and Kendorski (1984).

Comparison of Alternatives: A comparative examination of all the design calculations and support predictions should demonstrate a convergence of the designs. If not, discrepancies need to be critically examined and resolved. Upon achievement of the sought-after convergence of the designs, the designer can be reasonably certain that the risk of inadequacy of the selected design has been minimized. As a contingency against unforeseen conditions, designers should always prepare backup designs for poorer rock conditions than expected, and, in the case of facilities driven under contract, include these designs in the contract documents.

Because of the inherent variability of natural materials, risk and uncertainty can never be eliminated; however, through the use of a comprehensive design strategy, they can be reduced to an acceptable level.

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Section 11 Environmental Health and Safety

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Chapter 11.0
ENVIRONMENTAL ISSUES

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11.0.1 INTRODUCTION

Discussion in this section of the *Handbook* is limited to the working environment of miners, the mine workers themselves, and the machines used in mining (Fig. 11.0.1). The contents emphasize personnel health and safety aspects and engineering design and control practices to ensure a productive health and safety environment. Reference is to the health and safety of the individual miners as well as to the safety of the working environment. This *Handbook* also contains references to health and safety regulations in Chapter 3.2 and environmental protection in Chapters 3.4 and 7.3.

11.0.2 HEALTH AND SAFETY ISSUES

Conditions which have the potential to damage both the health and safety of the miners and the mining environment are numerous. In discussing occupational risks in the mining industry, it is common practice to identify health and safety hazards separately. Among possible threats to the health of miners are the following: exposure to toxic gases and dusts, exposure to excessive heat and humidity, inadequate illumination, noise and vibration problems, and oxygen-deficient atmospheres. Some of these environmental stresses may interact to produce a greater overall effect. For example, excessive vibration combined with low illumination may lead to a higher-order stress than either one being present alone. In combination or alone, if environmental stresses exceed human tolerance levels for prolonged periods of time, feelings of discomfort will arise, alertness will decrease, accidents will occur, and performance and productivity will drop. Threats to a miner's safety may arise from many sources: from falls of roof, face, rib, or side; from haulage or other machinery; from electrical equipment, explosives, or ignitions or explosions of gases and dust; from sudden inundations of water and gas; or from mine fires. Health and safety aspects are not necessarily unrelated; for example, a mine fire or an explosion is a safety problem that may pose a serious threat to the health of miners due to the resultant high concentrations of toxic gases in the mine atmosphere.

The importance of maintaining environmental stresses under control at all times cannot be overemphasized. Consequences of inadequate control can be sudden and catastrophic—such as injuries and loss of life through suffocation, heat strokes, and explosions—or slow and long-enduring—such as lung diseases including coal worker's pneumoconiosis (CWP) or black lung. The debilitation can be permanent. In addition to the toll of human suffering, environmental stresses are associated with insidious increases in social costs (see Chapter 3.1).

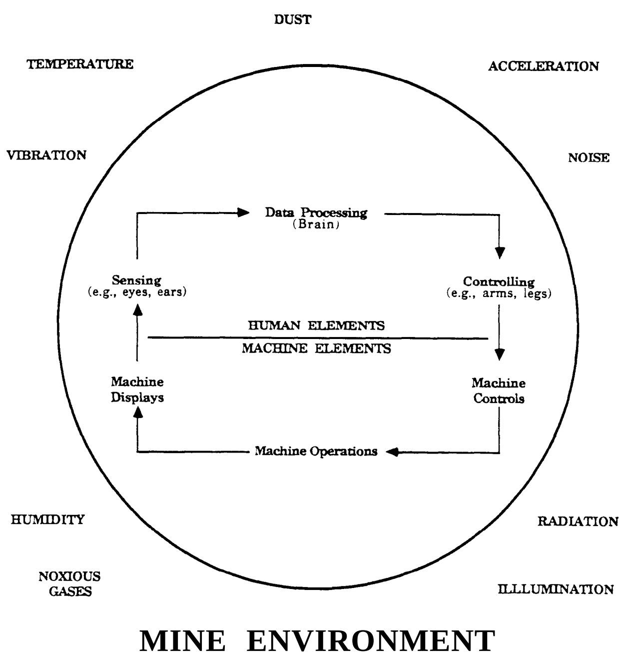

Fig. 11.0.1. Health and safety environment in mining operations.

11.0.3 ROLE OF GOVERNMENT

Today there is a strong measure of government control and inspection of mines under legislation specific to the mining industry that is intended to safeguard the health and safety of the miners (Anon., 1969; Anon., 1977). In general, however, mining laws have resulted due to the introduction of new and better mining methods by the industry and the development and implementation of new knowledge through research into the principles of accident prevention (Kenzy and Ramani, 1980). In this context, the establishment of the US Bureau of Mines in 1910, nearly 45 years after the first proposal to create one, is an event of major significance in the annals of health and safety. The Bureau's pioneering contributions in such areas as explosions, explosives, fires, and ventilation are widely credited for improved health and safety in mines worldwide.

Changes in the federal government's responsibility for mine safety have developed through a series of legislative actions in 1865, 1910, 1941, 1946, 1947, and 1952, culminating with the passage of the Federal Metal and Nonmetallic Mine Safety Act of 1966 (the 1966 Metal Act), the Federal Coal Mine Health and Safety Act of 1969 (the 1969 Coal Act), and the Federal Mine Safety and Health Amendments Act of 1977 (the 1977 Mine Act). The 1969 Coal Act represents yet another significant marker on the road to improved mine health and safety. In passing this Act, Congress declared that "the first priority and concern of all in the coal mining industry must be the health and safety of its most precious resource—the miner." According to Congress, the purpose of the Act is "to establish interim mandatory health and safety standards . . . to protect the health and safety of the Nation's coal miners."

Within eight short years of the enactment of the 1969 Coal Act, Congress passed the 1977 Mine Act by combining the health and safety programs of the 1966 Metal Act and 1969 Coal Act into one Act, and transferring the responsibility for enforcement

Table 11.0.1. Work Accidents in 1988 in the US

Industry	No. of workers	Fatalities	Disabling injuries
Agriculture	3,100,000	1,500	140,000
Mining, quarrying	800,000	200	30,000
Construction	6,500,000	2,200	210,000
Manufacturing	19,500,000	1,100	350,000
Transportation and public utilities	5,800,000	1,400	140,000
Trade	27,000,000	1,100	320,000
Services	34,600,000	1,500	330,000
Government	17,000,000	1,600	280,000
TOTAL	114,300,000	10,600	1,800,000

Source: Anon., 1989.

of the provisions to the Department of Labor from the Department of the Interior. Furthermore, new provisions were enacted to provide mandatory health and safety training to new miners and refresher training to all miners. The 1977 Mine Act is purported to have adopted the stronger features of the Occupational Health and Safety Act of 1970, perhaps the most significant piece of legislation with reference to workman's occupational health and safety in US history. The Mine Safety and Health Administration (MSHA) is the major federal agency responsible for the enforcement of the 1977 Mine Act. The agency also has provisions for technical support of the enforcement branch, training support for mining personnel, and testing and certification of new equipment for the mining industry.

11.0.4 ACCIDENT STATISTICS

According to the National Safety Council (NSC), in 1988, there were over 10,000 deaths on the job and 1.8 million disabling work-related injuries in the United States (Table 11.0.1). The total time lost due to these work-related injuries and fatalities was 35 million days. The total cost was estimated at $47.1 billion. The value of goods and services each worker must produce to offset the cost was approximately $410. These statistics vividly portray an enormous waste of talent and effort. In 1988, according to NSC figures, the mining and quarrying industry employed nearly 800,000 workers. There were 200 deaths and 30,000 disabling injuries (Anon., 1989). Despite these intolerable numbers and the need for higher safety, the 1988 statistics reflect an encouraging improvement over those of a few years ago.

The collection of mine accident statistics on an industry-wide basis and publication of reports of injury experiences, safety investigations, recommended practices, etc., are among the major functions of MSHA. Information reports on mining industry injury experiences and occupational illnesses are published on an annual basis. Separate reports are published for the coal, metallic mineral mining, nonmetallic mineral mining, sand and gravel, and stone segments of the mineral industry. Data used for the compilation of these reports are obtained from mine plant operators who are required to report the occurrences of injuries, occupational illnesses, and related data. The reported data are summarized by work location, accident classification, part of body injured, nature of injury, occupation, and principal type of mineral. Related information on employment, worktime, and operating activity also is presented. Data reported by independent contractors performing certain work at mining locations are also depicted separately in these reports. For ease of comparison with other segments of the mining industry, summary reference tabulations of other segments are included at the end of each report.

11.0.4.1 Definition of Terms

Detailed information on the source of data, scope of statistics, and definition of terms used can be found in the MSHA Information Reports on injury experiences (Anon., 1989a, 1989b, 1989c, 1989d, 1989e). Definitions of some of the important terms used in the Informational Report follow.

Occupational Injury: An occupational injury is any injury to a mine worker that occurs at a mine and for which medical treatment is administered, or which results in death or loss of consciousness, restriction of work or motion, inability to perform all job duties on any day after an injury, lost workdays, temporary assignment to other duties, transfer to another job, or termination. The injury must result from a recognizable single incident. For example, an explosion of a battery which splattered a worker with sulfuric acid would be a single incident, and the worker would be considered to have suffered an injury. Another example would be an injury where a worker was overcome by hydrogen sulfide gas released from an exploding vessel.

Occupational Illness: An occupational illness is an illness or disease of a mine worker that may have resulted from work at a mine or for which an award of compensation is made. To be classified as an occupational illness, the disability must result from repeated exposure to the condition or substance that caused the disability. A classic illness example is a pneumoconiosis disability that may have been induced by repeated exposure to the causative dusty condition. Dermatitis resulting from repeated exposure to lime dust or other material in the work environment also should be reported as an illness. In cases where the time of onset of illness is in doubt, the day of diagnosis of illness can be considered as the first day of illness.

Worktime: Worktime includes number of workers and number of employee-hours worked. "Average number of workers" is a summary of the average number of persons working at individual establishments during calendar quarters of active operations. "Employee-hours" is a summary of employee-hours reported.

Degree of Injury: Degree of injury indicates the seriousness of injuries. FATAL injuries are those occurrences resulting in death, NFDL (Non Fatal with Days Lost) injuries are nonfatal occurrences that result in days away from work, statutory days charged, or days of restricted work activity. NDL (No Days Lost) injuries are occurrences having no lost workdays, that is, nonfatal injury occurrences resulting only in temporary loss of consciousness or medical treatment other than first aid.

A *permanent total disability* is caused by work injury or illness other than death that permanently and totally incapacitates a worker from following any gainful occupation, or that results in the loss, or the complete loss of use, of both or any combination of: hands, arms, legs, feet, or eyes.

A *permanent partial disability* is caused by any work injury or illness other than death or permanent total disability that results in the loss, or the complete loss of use, of any member or part of a member of the body, or any permanent impairment of functions of the body or part thereof, regardless of any preexisting disability of the injured member of impaired body function.

Incidence Rate: Incidence rate is defined as the number of injuries per 200,000 employee-hours, rounded to two decimal places. The standard incidence rate formula is

$$\text{IR} = \frac{\text{number of injuries} \times 200{,}000}{\text{number of employee-hours}} \qquad (11.0.1)$$

The highest rate given is 99.99 even if the calculated rate is greater than 99.99.

Lost Workdays (LWD): Lost workdays consist of days away from work (DAW) and days of restricted work activity (DRA),

Table 11.0.2. Statutory Days Charged to Accidents

Disability	Lost Workdays Charged*
Death	6000
Permanent total disability	6000
Dismemberment or total loss of use	
Arm above elbow	4500
Arm at or below elbow and above wrist	3600
Hand at wrist	3000
Leg above knee	4500
Leg at or below knee and above ankle	3000
Foot at ankle	2400
Loss of sight:	
One eye (whether or not there is sight in the other eye)	1800
Both eyes (in one accident)	6000
Complete industrial loss of hearing:	
One ear (whether or not there is hearing in the other ear)	600
Both ears (in one accident)	3000
Unrepaired hernia	50

* Not more than 6000 lost workdays will be charged in a single accident that results in multiple injuries to a mine worker.
Source: Anon., 1989a.

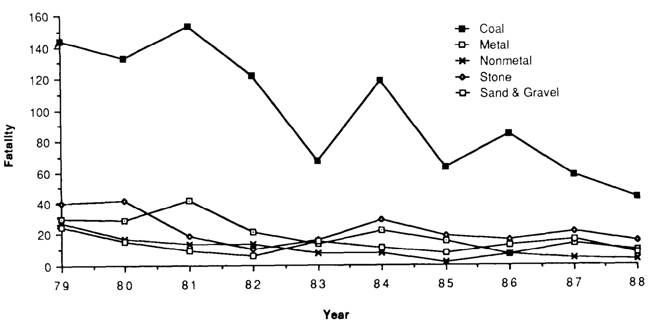

Fig. 11.0.2. Fatalities in coal, metal, nonmetal, stone, and sand and gravel over the period 1979–1988.

or statutory days charged as prescribed from a table of standard charges (see Table 11.0.2). For example, 6000 days are charged for a fatality or permanent total disability.

Severity Measure: Severity measure is the number of lost workdays per 200,000 employee-hours, rounded to whole numbers. The standard severity measure formula is

$$SM = \frac{\text{number of lost workdays} \times 200{,}000}{\text{number of employee-hours}} \qquad (11.0.2)$$

The highest rate given is 99,999 even if the computed rate is greater than 99,999.

Computer files of raw accident information data are available from MSHA. The raw data can be analyzed for studying the yearly variations in accident injury experiences for a particular mining commodity, job location, worker occupation, etc., as well as for several other comparative or absolute analyses.

11.0.4.2 Injury Experience

Summary statistics on the number of fatalities for each segment of the industry for the ten-year period 1979 to 1988 are presented in Fig. 11.0.2. While there have been fluctuations on a year-to-year basis, the long-term trend is characterized by a steady decrease in the number of deaths. Further, the frequency rates of incidences that involve lost working days (LWD injuries) and severity rates also show steady improvement for all segments of the mineral industry (Fig. 11.0.3 to 11.0.7). Even though there are year-to-year variations in either direction, the long-term trends are of decreasing incidence and severity rates. There is little doubt that mines are safer today than ever before. It is important to note that, each year, coal mining has experienced more fatalities than the other mining sectors. Also the incidence and the severity rates of the coal mining segment are the highest.

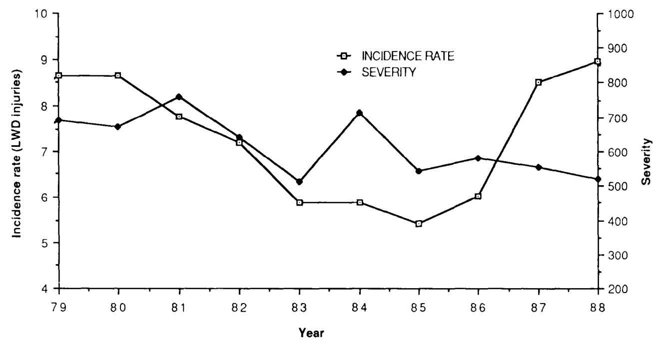

Fig. 11.0.3. Incidence rate and severity in coal mines over the period 1979–1988.

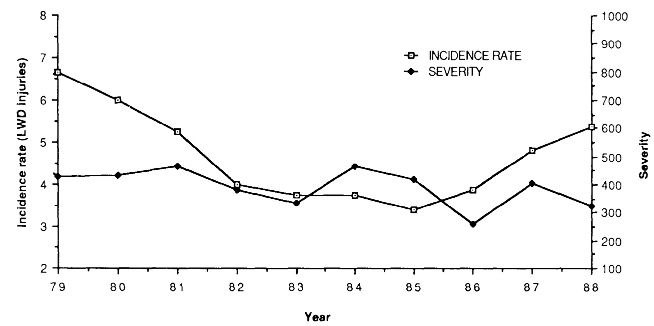

Fig. 11.0.4. Incidence rate and severity in metal mines over the period 1979–1988.

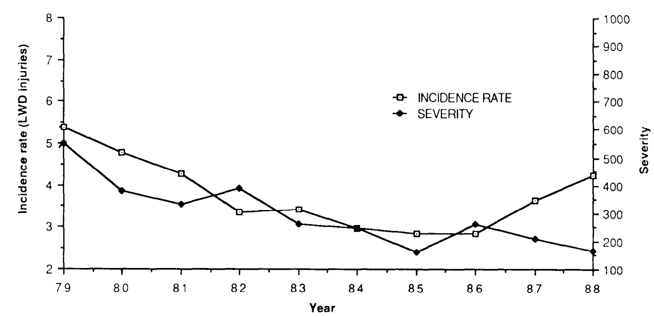

Fig. 11.0.5. Incidence rate and severity in nonmetal mines over the period 1979–1988.

11.0.4.3 Occupational Illnesses

The potential for permanent damage to the miner's health from the gaseous and particulate constituents in the mine atmo-

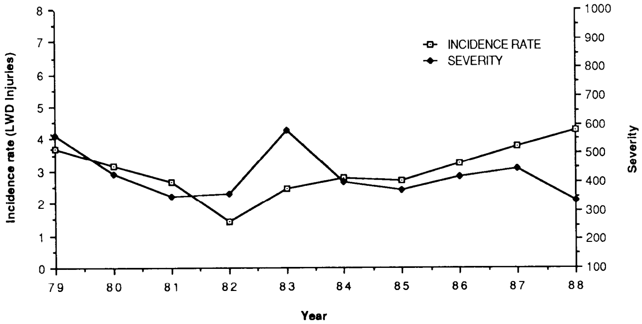

Fig. 11.0.6. Incidence rate and severity in sand and gravel over the period 1979–1988.

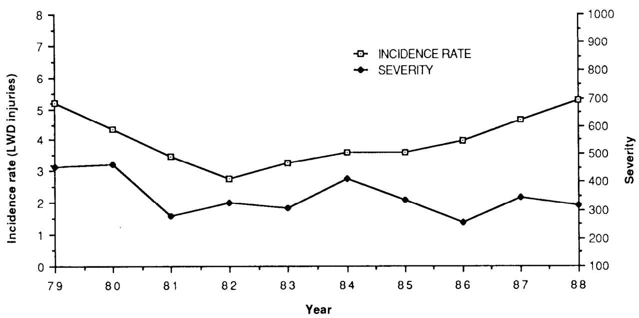

Fig. 11.0.6. Incidence rate and severity in sand and gravel over the period 1979–1988

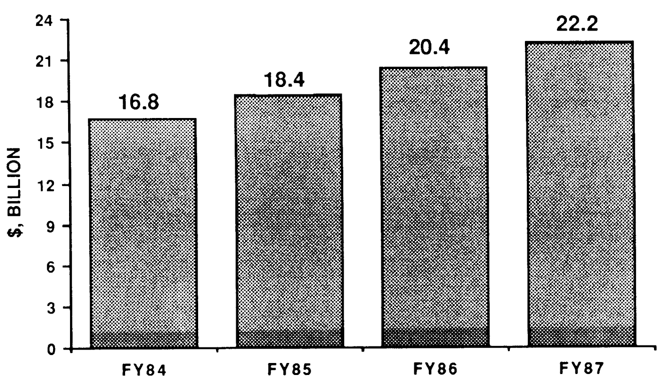

Fig. 11.0.8. Total compensation for black lung benefits.

sphere has been recognized from earliest times. Diseases such as silicosis, asbestosis, and pneumoconiosis are associated with long-term exposure to high concentrations of silica, asbestos, and coal dust, respectively. Bad water quality and the lack of adequate lighting have also been associated with such diseases as dermatitis and miners nystagmus, respectively. The payment of benefits to miners who suffer from occupational illness is a common practice. One such program is the black lung benefits program. Under this program, miners who are suffering from CWP are paid compensation. The cumulative benefits paid out of the program from 1969 until now total over $25 billion (Fig. 11.0.8). The number of claims processed in 1987 was nearly 300,000. The number of new claims per year averages between 400 to 500.

11.0.5 SECTION OUTLINE

It is now over two decades since the enactment of the 1969 Coal Act. This Act and the regulations contained therein defined in great specificity many standards for healthful and safe operation of a mine and, in some cases, what constitutes a good practice. Stringent standards have been presented for, among other things, respirable dust, noise, illumination, ventilation, roof control, electrical equipment, and escapeways. These have necessitated changes in mining layouts, mining operations, mine equipment, and miner training. Provisions for government inspection of mines and for assessment of penalties for violations have been revamped. Yet the fundamental requirements for a safe mining system continue to be (1) a well-engineered system in which all known hazards that can be eliminated by design are, in fact, eliminated; (2) a motivated work force that is trained to recognize hazards and to ensure that all operations are carried out in the safest manner; and (3) an effective management that not only maintains the safety integrity of the system but is constantly searching to improve on existing standards. Miner training must emphasize both general and job-specific health and safety aspects and improvement of production and maintenance skills. Management training must develop in its managers an ability to interpret safety statistics, perform cost benefit analyses on investments promising health and safety improvements, and understand human behavioral patterns. These are important issues in personnel health and safety (Chapter 11.1).

For maximum success, mine design must be compatible with sound engineering principles and practices and must incorporate the requirements of health and safety regulations. Tremendous changes have taken place in work procedures and practices to control mine atmospheric contamination from gases, dusts, heat and humidity, and radiation (Chapters 11.2, 11.3, and 11.4). Improved methods for controlling the hazards from diesel engine exhaust are also being actively pursued. The combination of better quality diesel fuel, well-maintained engines, new exhaust treatment systems, and changes in operational procedures has the potential to greatly reduce the gaseous and particulate components in the exhaust (Chapter 11.5). Ventilation is an important common denominator in most health and safety design considerations in mines. In addition to the supply of fresh air to the miners, the overall control of gas, dust, heat, and humidity problems is achieved through the proper design of the ventilation system (Chapters 11.6 and 11.7). Problems of noise and illumination, always a matter of concern in underground workings, are being addressed through the identification, development, and incorporation of specific design requirements for mine machinery and mining environment. All these have resulted in improved noise control and better lighting conditions (Chapters 11.8 and 11.9). A significant advance in aiding planning, designing, and controlling complex mining systems is the wide availability of computer-oriented mine atmospheric design simulators and mine-wide monitoring systems (Chapter 11.10).

The design of mines and mining equipment, and operational procedures that fit the mold of today's industrial and social environment is, however, a continuously evolving process. Experience world-over is that engineering controls and a knowledgable work force are the most effective preventive measures for occupation-related health and safety hazards. In fact, research and development of engineering solutions to the emerging problems along with enlightened legislation are the only realistic avenues to reach the goal of the complete elimination of health and safety hazards in the mine environment.

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Chapter 11.1
PERSONNEL HEALTH AND SAFETY

RAJA V. RAMANI

11.1.1 INTRODUCTION

The single most valuable asset in any organization is human resources. It is the ingenuity and creativity of people that lead to the effective and efficient utilization of all other available resources to achieve the economic and societal goals of the organization. Protection of the health and safety of employees from excessive or undesirable stresses in the occupational environment is all important. This must be so from both the humanitarian point of view and the overall welfare of the enterprise itself.

The enhancement of personnel health and safety in mines requires an understanding of the hazards and the requirements for their control. Further, it requires a critical evaluation and application of the various approaches to hazard control. In addition to the learning experience from the lamentable history of accidents and disasters in mines, there is a critical need to reduce the risks of mine hazards and resulting accidents through the application of such proactive analysis techniques as systems safety analysis and disaster simulations for the identification of new hazards. For rapid progress towards a hazard-free environment for the miners, combinations of several hazard control approaches need to be utilized. The vital role of management in focusing the miner's attention to personnel health and safety must be recognized. Finally, the search for new tools and techniques for mining hazard control must go hand in hand with transfer of successful practices from other industries.

11.1 2 HAZARDS, ACCIDENTS, AND DISASTERS

Unless there is a common understanding of terms, there can be considerable confusion in transmitting information and knowledge. This can be particularly vexing in the understanding of safety literature in which such terms as "injury" and "accident" are often (and mistakenly) used interchangeably. Furthermore, definitions of accidents and hazards adopted for classification purposes may not be detailed enough to use as a basis for hazard control.

The term *hazard* is used here to describe an unsafe situation in a mine (Anon., 1981). This may be an unsafe physical condition or unsafe acts of miners. Perception of a hazard is essential. If a hazard is not perceived, no action can be taken to remove it. A *hazard source* is the background condition which, while not posing a danger in itself, may give rise to a hazard. For example, methane is a source of hazard. An *accident* is the realization of a hazard. An accident of major proportions representing a substantial threat to human life may be said to have *disaster potential.* If a large number of people are in fact killed, it is deemed a *disaster.* Disasters command tremendous attention, due to their infrequent occurrence and the extent of human suffering involved, even though the number of deaths from non-disaster accidents may be many times higher.

Fortunately, many mine accidents that have disaster potential do not become disasters, either because the accidents occur at a time when no or few workers are in the mine, or because the response is effective and all the threatened miners are evacuated safely. For those involved in health and safety management, the

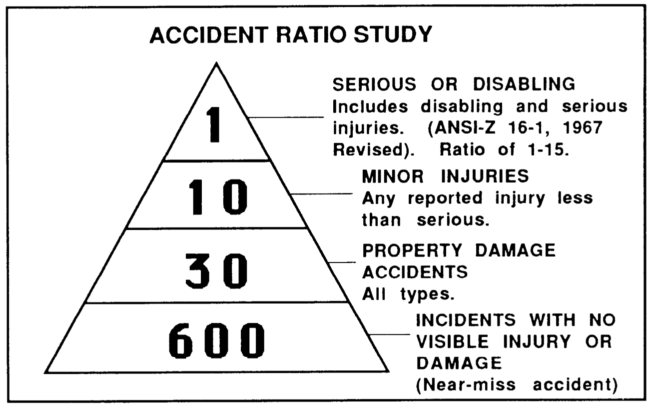


Fig. 11.1.1. Foundation of major accidents/injuries (Bird and Loftus, 1982).

distinction between accidents and disasters is not significant, and the aim is to identify, eliminate, or control the hazards.

11.1.3 HAZARD CONTROL REQUIREMENTS

The first requirement of *hazard control* is to recognize that any unintended occurrence is an accident whether such a happening results in injury or not. For example, if the objective of a study of mine explosions is to unearth the underlying causes of explosions and prevent their occurrence, the analysis of excessive gas accumulations, excessive frictional sparkings, and electrical arcings as well as any ignitions that did not result in injury or property damage can be just as significant as an incident (ignition or explosion) that resulted in injury. In fact, all these occur more frequently in a coal mine than explosions and provide a much larger basis for the analyses and determination of causative factors. This, in turn, can provide valuable information for developing more effective control strategies. According to a study by Heinrich (1959), the ratio of no injury to minor injury to major injury is 300:29:1. Underlying these minor injuries are numerous unsafe practices and unsafe conditions which fortunately may not result in any incident. Bird and Loftus (1982) have updated this ratio with further information on property damage accidents (Fig. 11.1.1). The moral of these ratio studies is that accident prevention must start with prevention of unsafe practices and unsafe conditions as well as of minor injuries.

A second requirement for an effective accident control program is a good index of safety performance. Many attempts have been made to develop comparative standards for safety measurement. Routine and surprise inspections, accident investigations, compensation costs, and injury ratings have all been used at one time or another. The number and severity of disabling injuries and the number of employee-hours lost can be compared with production and total employee-hours worked to judge safety performance between mines in the same company and between companies in the mining industry (see Chapter 11.0.4). The tremendous value of these statistics to focus attention on

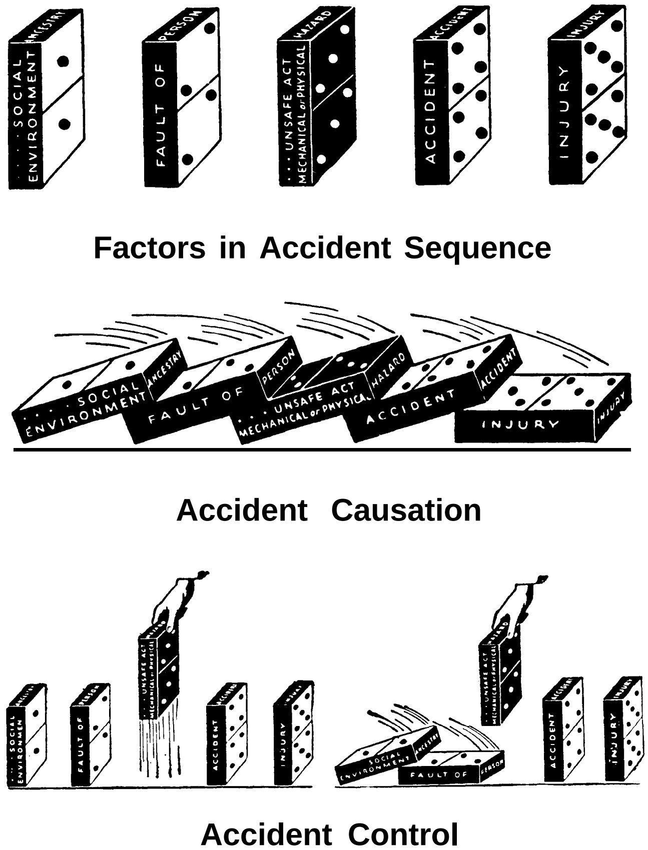


Fig. 11.1.2. Accident causation and control (Heinrich, 1959).

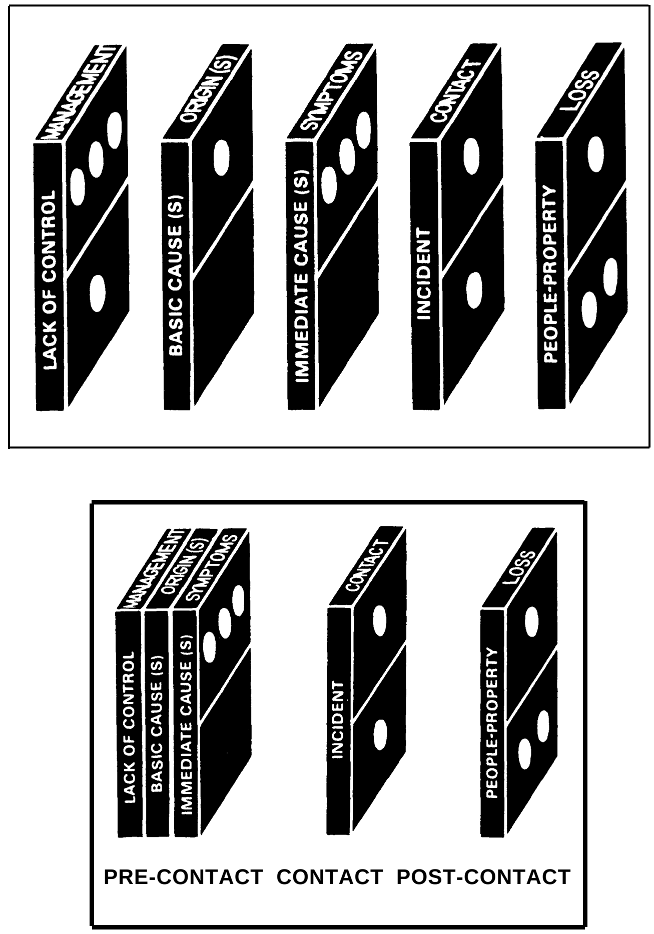


Fig. 11.1.3. Accident causation and control (Bird and Loftus, 1982).

health and safety issues must be recognized. But it is obvious that these performance ratings have a limited value in hazard control since they are still tied only to injury experience and lost employee hours. The ability to identify actual health and safety conditions by combining quantitative aspects of the existing indices with qualitative aspects is important.

A third area is the critical evaluation of the relative contributions made by the environment, human beings, and mechanical elements to an accident. The traditional method to assess their contribution has been through mandatory investigations and inspections. There are state and federal government inspection agencies with powers to ensure compliance with existing laws and for promulgating new regulations for safer performance. In many instances, mine personnel have to be certified by government regulatory agencies to be employed in certain categories (e.g., mine foreman) and these officials have a certain number of prescribed inspections to make. Inspections can be made to check operations against the prescribed standards, to detect deviations, and to suggest remedial actions. The chain of events leading to the causation of an accident or injury has been compared to a line of dominos, each domino in the sequence knocking down the one next to it (Heinrich, 1959). In this concept, the accident can be avoided by intervention to prevent an earlier domino from falling down such that the chain of events does not proceed to the last domino that represents losses to human resources and property (Fig. 11.1.2). This concept has been modified by Bird and Loftus (1982) to focus attention on the lack of control (Fig. 11.1.3) which is the first domino in the sequence. Also the highlighting of the key role of management in accident prevention by ensuring all aspects are under control at all times is to be noted. However, the causes of accidents are manifold and can result from unsafe acts of employees or unsafe mechanical and/or physical conditions, or both. It is comparatively easier to detect and spotlight physical hazards than unsafe acts of workers. However mechanically safe a system is designed to be, built-in safety can be jeopardized by an unsafe act. Maintenance and preservation of the safety features depend on compliance with many rules and regulations by the workers in the system. Miners must be well trained in these rules of safe operation and behavior. Further, during inspections, supervisors must spend more time studying the worker to discover possible acts of commission and omission beyond that required to ensure compliance.

In any case, compliance with laws and elimination of known unsafe practices are in mathematical terms "necessary" rather than "sufficient" conditions to consider a place safe. For instance, with the increasing ability to measure and characterize the elements of the mine atmospheric environment has come an awareness of factors, hitherto unknown, that may pose new threats to health and safety (Ramani, 1988). Recent examples here include, but are not limited to, the high concentrations of quartz in respirable airborne dust in coal mines, the large proportion of submicrometer particles in diesel engine exhaust and their impacts on mine air quality, the presence of asbestos-type fibers in mined products, and the adequacy of existing standards for radiation exposure in mines. These observations have enabled focusing attention to the need for increasing research on the health effects and developing more effective engineering controls. In other words, vigilance can never be relaxed in the search for new hazard sources and their control.

Finally, the collection, analysis, and use of accident statistics for developing hazard control programs must be a continuing endeavor. While controlled experimentation is difficult, the development of a theoretical framework for investigating hazard and accident phenomena is necessary. If accident statistics on an industry-wide basis are general and descriptive, they can result in conclusions and recommendations that are also of a broad and general nature. For developing specific programs for specific mines, the data collection activities and the analysis procedures must be focused to address the specific issues. For example, specific studies must be undertaken to achieve definite, realistic, and quantitative objectives, such as identifying the cause of specific accident types, initiating a new accident control program, or decreasing accident severity and injury rates.

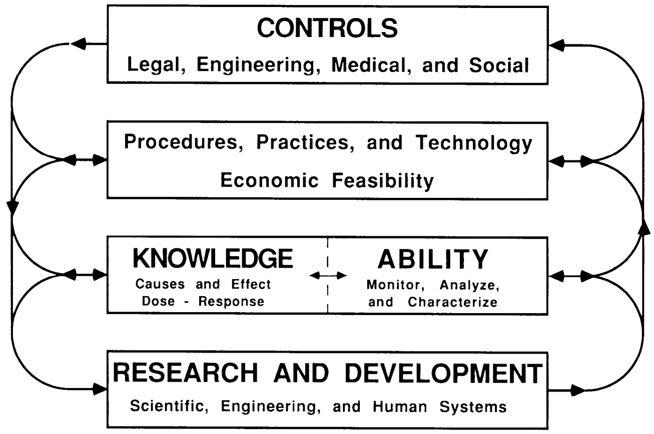


Fig. 11.1.4. Development paths for control mechanisms.

11.1.4 HAZARD CONTROL APPROACHES

Mining is one of the oldest of human occupations (Chapter 1.1). The awareness of occupational illness associated with mining is evident from the writings of Hippocrates (460–370 BC), Pliny the Elder (AD 23–79), and Galen (AD 131–210). Therefore, hazards associated with mining were known from ancient times. Georgius Agricola (1494–1535), in his *De Re Metallica,* refers to the diseases of miners and the need for their control through good planning and operating practices and responsible management. This early recognition of the occupational hazards has undoubtedly contributed to the development of occupational medicine and health care systems (Schwerha, 1989).

In assessing health and safety problems in the mining industry, there is no room for complacency or fatalistic attitudes. Yet one must recognize that mining is hazardous, and vigilance can never be relaxed. As distinct from practices in many other industries, the mine working environment cannot be precisely controlled. Also the environment is constantly changing. It is virtually impossible to foresee all the possible hazards and therefore, to take precautions against each of them. However, the control of the threats to health and safety of the miners has been a major concern for management, labor, and government alike. This concern has manifested itself in four primary mechanisms of control: (1) regulatory control through the passage of mine health and safety laws that set minimum standards of performance, (2) legal and social control through workmen's compensation laws for occupation-related injuries and health deterioration, (3) medical control through periodic physical examination, wearing of personnel protective devices, etc., and (4) engineering control through design and operation of mines according to the best recommended practices.

11.1.4.1 Health and Safety Regulations

It is only within the last 250 years that government regulation of mines and the promulgation of health and safety standards were initiated. France and Germany took a lead in this direction with the creation of inspectorates and schools of mines (Bryan, 1985). In England, statutes governing industrial working conditions were beginning to be debated by 1800. Initially, these efforts were spurred by the evils and excesses of the employment of women and children and increasing concerns to protect the community from the dangerous and unsanitary conditions in the mines. A major aspect of the early literature calling for legislation of mines was the recognition of the importance of adequate ventilation to health and safety and the need for better education and training of those responsible for the management of coal mines. Legislation specific to the mining industry was first enacted in England in 1842, and by 1850, a bill establishing a mine inspectorate was passed.

In the United States, Pennsylvania should be credited with the first formal move to legislate safety measures in its anthracite mines. In 1858, a bill was introduced in the Pennsylvania legislature authorizing state supervision of the mines in Schuylkill County. The bill failed that year and again in 1868, but in 1869 it became law, the first state mining statute in the United States (Core, Ramani, and Frantz, 1983). A bituminous mining law was not passed by Pennsylvania until 1877. As a matter of interest, the American Institute of Mining Engineers (AIME) was founded in Wilkes Barre, PA, in 1871, and one of its primary goals was the development of safer coal mines. Enactment of significant mine health and safety legislation in the United States has closely followed in the footsteps of major disasters (Table 11.1.1).

Prior to 1969, the federal government was extremely reluctant to intrude into the areas of health and safety, particularly the enforcement of standards, which was viewed as a state responsibility. The situation since 1969, as discussed in Chapter 11.0, has changed significantly. For a more detailed discussion of mine health and safety regulations, see Chapter 3.3 in this *Handbook.* It is worth emphasizing again that health and safety legislation, at any level, provides only a minimum framework for the engineering and operations of mines, and should rarely go beyond what is generally accepted as a good practice. Legal controls cannot solve problems of health and safety if they are not supported by sufficient technical clarity and feasibility. Otherwise, they will be no more than statements of good intent and may even frustrate professional progress towards safer work environments. The desirable process for developing effective control is better understanding of the cause and effect, dose-response relationships, feasible engineering designs and practices, and good professional judgment. These must be the foundations on which the legal control of health and safety is built (Fig. 11.1.4). However, advances in technology, good practices in other industries, and increases in knowledge provide sufficient grounds for the loop to run in the opposite direction. The danger to avoid is legislation that may restrict the alternatives available to the industry or does not fully consider the uniqueness of the mining operations and environment.

Table 11.1.1. Development of Federal and State Mine Safety Laws in the United States

Date	Disaster	Significant State and Federal Legislation	Proposals for or Enactment of Other Legislation
1842			Pennsylvania Mine Safety Act *Proposed*
1865			Federal Bureau of Mines *Proposed*
3.14.1866			Schuylkill County, Pennsylvania (Anthracite) Mine Inspection Act *Proposed*
4.12.1869			Schuylkill County, Pennsylvania (Anthracite) Mine Inspection Act
9.6.1869	Avondale, PA (108 killed)		
3.3.1870		Pennsylvania (Anthracite) Mine Inspection Act	
4.22.1870			Mercer County, Pennsylvania (Bituminous) Mine Inspection Act
3.27.1872			Illinois Mine Inspection Act
3.21.1874			Ohio Mine Inspection Act
4.18.1877		Pennsylvania (Bituminous) Mine Inspection Act	
3.11.1879			West Virginia Mine Inspection Act
5.10.1884			Kentucky Mine Inspection Act
2.25.1886			Federal Inspection of all mines where more than 10 men work—Wyoming Territory
3.3.1891			Federal Inspection of all mines in Territories with greater than 1000 tons production—New Mexico, Oklahoma, and Utah
7.1907			Federal Inspection of mines in the Territories transferred to the U.S. Geological Survey
12.6.1907	Monongah, WV (361 killed)		
12.19.1907	Darr, PA (239 killed)		
5.1908		US Geological Survey funded to investigate mine accidents and explosions	
11.28.1908	Marianna, PA (154 killed)		
11.13.1909	Cherry, IL (259 killed)		
7.10.1910		US Bureau of Mines established, no mine inspection rights	
3.13.1912			Virginia Mine Inspection Act
5.7.1941		PL 49: Federal Coal Mine Inspection Act	
7.24.1946		Federal Mine Safety Code	
3.25.1947	Centralia, IL (111 killed)		
8.4.1947		Mandatory compliance with the Federal Mine Safety Code for 12 months	
8.4.1947		PL 328: Survey of mines to determine compliance with the FMSC of 1946 (33% compliance found)	
12.21.1951	West Frankfort, IL (119 killed)		
7.16.1952		PL 552: Federal Coal Mine Safety Act	
1960			Senate Bill: S. 743 Eliminate Small Mines Exemption of PL 552 *Proposed*
3.26.1966		PL 376: Elimination of the small mines exemption of PL 552	
9.16.1966		PL 89-577 Federal Metal and Non-Metal Safety Act	
9.1968			Federal Coal Mine Health and Safety Act *Proposed*
11.20.1968	Farmington, WV (78 killed)		
12.30.1969		PL 91-173: Federal Coal Mine Health and Safety Act of 1969	
11.9.1977		PL 95-164: Federal Mine Safety and Health Act of 1977	

Source: Kinsey and Ramani, 1980.

11.1.4.2 Workmen's Compensation Laws

The use of penalties to control occupational illnesses and injuries has been in practice from the earliest of times. However, specific approaches to indemnifying workers through worker's compensation laws began in Germany in 1885, Great Britain in 1897, and the United States in 1902 (Grimaldi and Simonds, 1975). These laws required the employer to compensate the injured employee whether or not negligence could be proved. The cost incurred due to accident insurance and compensation is considered, in the framework of control, as a part of operating the business. Workmen's compensation cost is viewed as a deterrent since it is, in effect, an after-the-fact penalty whose magnitude can be so high that it encourages management to take many positive actions to avoid it. Workmen's compensation insurance is required in each of the 50 states. This insurance is carried through an insurance firm or a state-operated insurance fund. In the event of a health or an injury problem, compensation is paid to the worker. For example, in 1985 the average annual compensation cost per covered employee (benefits paid ÷ covered workers) in the 50 states varied from a low of $90 (Indiana) to a high of $618 (Alaska). The national average for all 50 states was approximately $290.

The 1969 Coal Mine Act considered the plight of miners suffering from coal worker's pneumoconiosis (CWP) arising out of employment in underground coal mines and provided for the payment of black lung benefits. This compensation program is funded by a charge on every ton of coal mined. At the present time, the charge is $1.10/ton ($1.22/t) of underground mined coal and $0.55/ton ($0.62/t) of surface mined coal. The cumulative annual benefits paid out of this fund, thus far, total over $25 billion. The number of claims processed in 1987 totalled nearly 300,000 and the amount expended nearly $1.8 billion. The number of new claims per year in the late 1980s averaged between 400 to 500. For professionals engaged in health and safety of mines, no compensation should be enough to equalize the dangers to health and safety, particularly if these can be avoided through design and operational procedures.

11.1.4.3 Medical Examinations

In recent years, increasing financial responsibility for occupation-related health problems and accident-injuries cases has been placed on employers. Pre-employment physical examinations and periodic continuing examinations are required to assure that employees' health and physical conditions are routinely monitored and documented. These examinations may reveal the on-set of physical problems such as hearing loss, loss of vision, heart problems, arthritic conditions, lung impairment, etc. Whether required by law or not, examinations provide medical evidence for job switching and settlement of claims.

Under the 1969 Coal Mine Act, there are two activities focused on the prevention of CWP: (1) the National Coal Workers' Health Surveillance Program, and (2) the Coal Mine Respirable Dust Sampling Program. Both these activities have been pursued since 1970 by federal government agencies. The two together combine both primary and secondary methods for CWP prevention: the dust control program by environmental control and the surveillance program by protecting the health of the coal miner (Attfield, 1984). The health surveillance program has several significant purposes: (1) the protection of health of the individual miner; (2) the monitoring of prevalence and progression of CWP; and (3) the identification of mines with high incidences of CWP for closer dust control monitoring. If a miner shows signs of CWP on a chest X-ray, then the miner has the option to transfer to a less dusty work area. In addition, the surveillance program provides a means to monitor the incidence and prevalence of CWP under the dust control program. The medical control provides a useful evaluation of the individual miner's health status through early detection of abnormal conditions and prescription of preventative action.

11.1.4.4 Engineering Control

Engineering control encompasses design of mines, selection of appropriate equipment, selection and training of the human resources into a knowledgeable work force, and management of these resources to achieve the goals of health, safety, and productivity. The specific aspects of designing mining engineering systems with effective control over hazards from mine atmospheric contaminants such as gases and dust, and environmental aspects such as radiation, heat, humidity, noise, and illumination, are the topics of coverage in the subsequent chapters of this section and therefore will not be amplified here. It is, however, important to stress that the greatest positive effect on the health and safety of miners can be achieved through proper application of engineering controls during the planning, designing, and operating phases of the mine.

11.1.4.5 Human Factors Engineering

During the last two decades, the application of human factors to mining engineering problems has been growing to enhance operational efficiency and health and safety of the miners. *Human factors engineering* is the systematic application of relevant information about human characteristics, abilities, expectations, and behaviors to the design of machines, tools, facilities, procedures, and environments that miners use (Sanders and Peay, 1988). There is a large body of human factor data, principles, and methods developed outside the mining industry that can be brought to bear on problems encountered within the industry. Increased activity in the human factors area has resulted in the documentation of human-factor-related mining problems and their solutions. The objective of all these efforts is to ensure that human factors engineering is an inherent component of the mining engineering design and that a mining system of greater safety and reliability is designed.

11.1.4.6 Systems Safety Analysis

Systems safety analysis methods provide a proactive approach to analyze systems for potential hazards that may threaten the health and safety of miners. This approach, developed specifically for the space program in the early sixties, made it possible to generate safety statistics *before* the deployment of new products and systems—that is, to ensure safety on the basis of analysis of the system rather than on the basis of past history. In the space program, this involved (1) making products of unexcelled quality and reliability, (2) identifying the failure characteristics of each unit and the impact of the unit's failure on the total system, (3) increasing system reliability by providing alternatives to permit safe functioning of the system in the face of failures of individual units, and (4) developing emergency procedures for contingencies. While the objectives of the space program permitted incurring larger costs in the interest of safety than may be practical in other programs, much of the systems safety approach is applicable to other industries, mining among them.

The *systems approach* to the safety problem focuses on the system taken as a whole, and not on its parts separately. It involves the interaction of people, machines, and environment within procedural constraints. It does not imply that the system must be risk-free, but rather that risk can be identified, quantified, managed, and controlled (Hammer, 1972). This systematic approach as applied to mine safety should include all phases from conceptual formulation of the system through analysis, synthesis, design, testing, evaluation, construction, training, approval, operation, and maintenance of the system. It requires: (1) a logical examination of all the elements of the system and their interactions; (2) identification of all sources of hazards; (3) calculation of the probability of hazard occurrence; (4) a search for the available options for hazard elimination or minimization, and in the extreme case, provisions for evacuation, escape, survival, and rescue; and (5) an analysis of costs and of problems associated with implementation and other procedural aspects of the various alternatives.

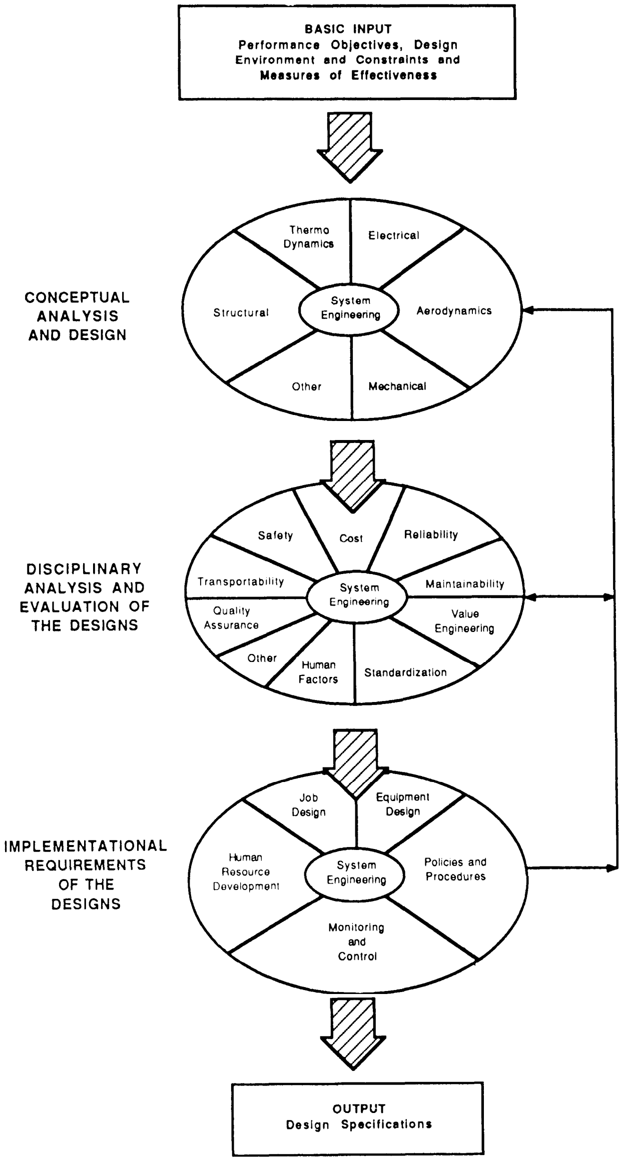
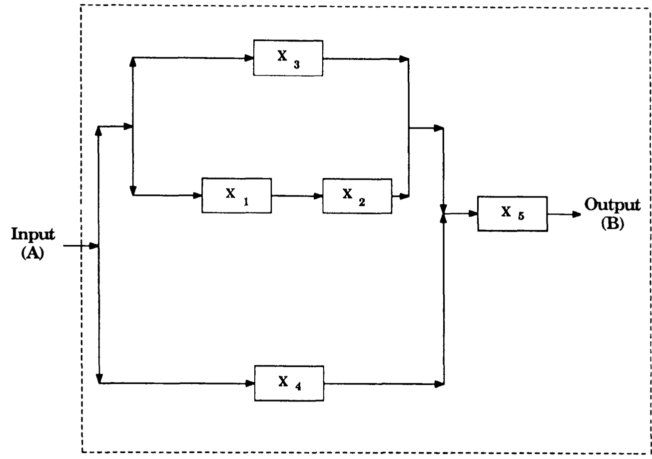
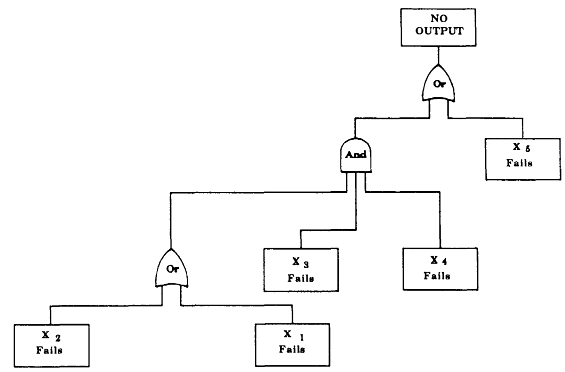


Fig. 11.1.5. System engineering process (after Anon., 1973).



(a) System Schematic



(b) Fault Tree of the System for the Top Event "No Output"

Fig. 11.1.6. Illustration of the fault tree analysis technique (after Anon., 1973).

The system engineering process is outlined in Fig. 11.1.5 as a three-step sequential process with iterative feedback loops to earlier steps from the later ones. The first step is conceptual analysis and design, which is often subject to constraints on input resources and to performance requirements of the final design. In the second step, these designs are evaluated for compatibility and technical and cost effectiveness through a number of speciality disciplines. In the third step, the implementational aspects are analyzed and developed. In this three-step process, the system designer moves from analysis to synthesis, pulling together parts into a system (Anon., 1973).

The systems safety approach is a composite of elements from a number of diverse disciplines including systems engineering, statistics, reliability theory, information theory, control theory, management, and behavioral psychology. It uses a number of techniques (Hammer, 1972). For example, the *technique of operations review* (TOR) is a tracing technique for identifying and defining a health and safety problem by searching for the root causes. In the *failure mode and effects analysis technique* (FME), the possible modes of failure of each component in a system are listed, and the effects of this failure on other components and the total system are analyzed. The outcome of the failure may be stated in qualitative terms for hazard potential as safe, marginal, critical, or catastrophic; and for frequency of occurrence, as probable, reasonably probable, remote, or extremely remote. In the *fault tree analysis technique* (FTA), all events and combinations of events that can lead to an undesirable event are arranged in a logic flow chart (or diagram) with the undesirable event on the top. A simple application of the technique is shown in Fig. 11.1.6 where for the system schematic shown, the undesired event is the loss of output B. The bottom level of a fault tree represents failures that cannot be further broken down meaningfully, that is, these failures are basic. Probabilities can be attached to these basic failures, and the probability of the top event occurring can be calculated as a function of these basic probabilities. The use of fault tree analysis technique to evaluate the reliability of escapeways in the event of a mine fire is provided by Goodman and Kissell (1989). More importantly, the very intensive nature of the inquiry required to apply TOR, FME, and FTA techniques to complex systems meaningfully will focus attention on potential weak links in the design and operations and provide the opportunity to solve them prior to implementation.

There are several aspects of mining in which the systems safety approach can be applied. It can be used by regulatory agencies to evaluate mine plans and procedures before approval and to conduct *post-audits* of disasters and accidents. It can be used by research organizations to unearth new sources of hazards in existing systems, to conduct *pre-design* audits for developing specifications and standards, and through *post-design* audits, to evaluate the developed product or system for effectiveness. It can be used by mine operators to evaluate existing systems, suggest modifications to operations or equipment or both, and develop training programs.

11.1.4.7 Miner Training

Manpower is the most precious investment made in a mine. The right person for the job is not just "born" but created by meticulous selection and adequate training and job orientation (Craig and Bittel, 1967). The past two decades have seen considerable progress in miner training. The fuel crisis of the early seventies, which highlighted a need for increased coal production, also focused attention on the alarming shortage of trained workers to achieve the new production goals. This in turn led to a reappraisal of the role of training in the health and safety of all personnel associated with mines and culminated in two documents that have had far-reaching effects on miner training. The first of these was the 1974 contract between the United Mine Workers of America and the Bituminous Coal Operators Association (UMWA-BCOA), in which an agreement for the development of health and safety training and retraining programs for all union bituminous coal mines was outlined. The second document was the 1977 Federal Mine Safety and Health Amendments Act which mandated training for the entire mineral industry and also required that several of the training programs be conducted by instructors certified by the Mine Safety and Health Administration (MSHA).

The federal regulations for new underground miners set forth a minimum of 40 hours of training, of which 32 hours is classroom training and eight hours is at the jobsite. Eight hours of training must be received before a new miner can go underground. For new surface miners, the regulations specify only a minimum of 32 hours of training, of which eight hours occurs before miners begin their actual work assignments. In addition, eight hours of annual refresher training is required of all miners. The topics required for training new miners, newly employed experienced miners, and annual refresher training under the federal regulations are listed in Table 11.1.2 for both underground and surface coal mining. Several coal producing states have additional training requirements, and in some states, these exceed the federal provisions. Included among the latter states are Illinois, Indiana, Kentucky, Pennsylvania, and West Virginia (Sanders and Peay, 1988). Finally, several mining companies have training programs for their personnel which go beyond the mandated state and federal levels (Masutomi, 1990).

The National Mine Health and Safety Academy, established in 1976 as an educational and training arm of MSHA, offers a wide variety of training products and services to the mining industry. Among these are films and videotapes, instructional programs, mine emergency simulation exercises, pre-shift inspection programs, self-study books on health and safety problems, safety manuals, and other important health and safety reference documents (Anon., 1990). As a result of these actions, there has been an accelerated development of mine health and safety training programs and increased emphasis on mine safety training and procedures by the industry. Mechanization, automation, and remote control technology are now changing the working environment and job functions within the mines. Determining how such innovative changes create hazards, affect accidents, and cause injuries for different occupations and how much the safety conditions will change is complicated. The need for new training and increased training to meet these technological changes poses a new challenge to the industry. A conceptual framework for the evaluation and development of effective training and job design strategies is presented in Fig. 11.1.7. The framework incorporates the tools and techniques from systems safety, and loss control procedures to identify and eliminate weak links in the human, machine, and environmental components of the system and their interactions.

Table 11.1.2. Elements of Underground and Surface Mine Health and Safety Training Programs.

	Introductory for		
	New miners	Newly employed experienced miners	Annual refresher for working miners
Statutory rights and responsibilities of supervisors	A	A	
Self-rescuer and respiratory devices	A		A
Introduction to the work environment	A	A	
Health	A		A
Hazard recognition	A	A	
Electrical hazards	A		A
First aid	A		A
Health and safety aspects of assigned tasks	A		
Mandatory health and safety standards		A	A
Prevention of accidents			A
Explosives	S		A
Entering and leaving the mine, transportation, communication	U	U	U
Mine map, escapeways, emergency evacuation, barricading	U	U	U
Roof and ground control, ventilation plans	U	U	
Cleanup, rock dusting	U		
Mine gases	U		U
Transportation and communication	S	S	S
Escape and emergency evacuation, firewarning and firefighting	S	S	S
Ground control, highwalls, water hazards, pits and spoil banks, illumination and night work	S	S	S

Legend: A = All miners; U = Underground miners only; S = Surface miners only.

Source: Modified from Sanders and Peay (1988); Digman and Grosso (1982).

11.1.5 ROLE OF MANAGEMENT

Management's role in health and safety is ubiquitous. It is reflected in the development of well-engineered systems, the choice of personnel for specific jobs, the provisions for education

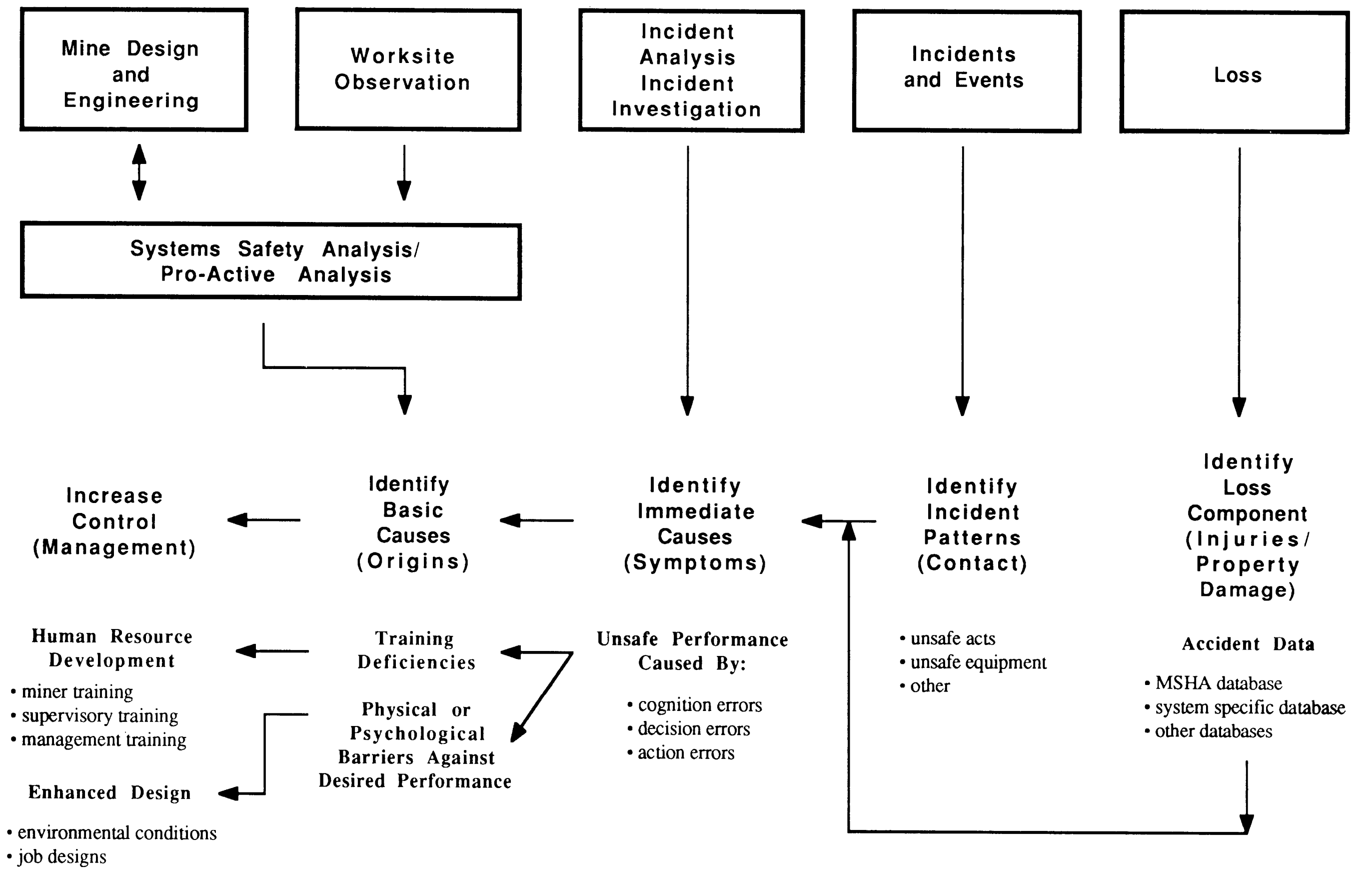


Fig. 11.1.7. Conceptual framework for evaluation and development of effective training and job-design strategies (Masutomi, 1990; Anon., 1981).

and training, and the continuing search for improvement in health and safety. Management has the sole authority to establish policies and priorities, to initiate and implement safety programs, to commit resources, and to reward managers and employees for achieving goals within the larger framework of its overall responsibilities (see Chapter 8.6).

The importance of management's commitment to safety was underscored in a National Academy of Sciences' mine safety study (Anon., 1982), which found that it was not possible to explain through statistical analyses of accident data the large and persistent differences between the injury rates of several mining companies as a function of physical, technological, or geographical factors. Instead, the differences were attributed to factors internal to the companies, particularly management's commitment, cooperation between management and labor, and quality of training of employees and manager.

It may even be worthwhile to side-step traditional concepts in mine management in favor of new ideas and principles for improving health, safety, and productivity. This was the case in a central Pennsylvania coal mine where a one-year test of the autonomous work group concept was conducted (Goodman, 1979; Trist, Susman, and Brown, 1977). In simplistic terms, the workers were responsible for planning and scheduling their own work, while the section foreman was responsible for ensuring that work practices adhered to applicable health and safety rules and regulations. An independent evaluation team analyzed the data and found a definite improvement with regard to safety. Mine safety violations had decreased; the miners felt that safety had improved; onsite observers noted better practices and habits in the autonomous work group than in the control group; and the responsibility given to the miners had the effect of reinforcing and creating better safety attitudes. Positive changes in attitudes and increased competency in job performance were also noticed. Although the study cannot be conclusive due to the small sample and short experimental period, it does point to the importance of teamwork, and suggests that closer cooperation between workers and management may be a way to achieve improved health and safety in mines.

The objectives of management, from the top executive to the first-line supervisor, must be to

1. Provide leadership in safety with clear definition of the goals and means to achieve them.
2. Ensure effective management through a safety organization that has clear lines of responsibility and support for managerial decision-making.
3. Promote and seek highest standards of safety performance at work through consistent and persistent development and use of knowledge and skill.
4. Provide a working environment for the miners in which the equipment, processes, and procedures are so reliable, well-defined, and understood as to eliminate hazards to the miners in the face of system failures.
5. Exemplify through actions at every available opportunity its deepest commitment to safety.

The major success of the 1969 Coal Mine Act rests on such strong foundations as methane concentration control, dust control, intrinsic safety and explosion-proof enclosures, minimum air quantity and quality standards, and escapeway provisions. These requirements drastically impacted mine ventilation planning, engineering, and practice, leading to both greater expectations and fulfillment of safe working conditions. The Act recognized the need for increased scientific, engineering, biomedical, and medical research studies to support not only the new legislation but also the development of new equipment and methods. A systems approach to the eradication of the problems of health and safety was prescribed through increased health and safety standards, inspections, personal protective devices, miner training, medical examinations, black lung benefits, and development of more effective engineering controls through increased research. Since the passage of the 1969 Coal Mine Act and the 1977 Mine Act, significant progress has been made in all aspects of health and safety. The vital role played by mine management and mine workers in achieving this improved record should not be overlooked or underestimated. The cooperative and coordinated efforts of the mining companies and miners, along with significant commitment of corporate financial resources, were among the major reasons for the rapid development and absorption of the newly developed technology. Much remains to be accomplished in achieving a hazard-free health and safety environment for the miners. There is little doubt, however, that mining in the United States has become less hazardous.

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Chapter 11.10
COMPUTER APPLICATIONS IN MINE VENTILATION AND THE ENVIRONMENT

SUKUMAR BANDOPADHYAY

11.10.1 INTRODUCTION

Over the past three decades, the role of the digital computer has expanded in an unprecedented manner. As mainframes and personal computers became cheaper and grew to astounding size, and the FORTRAN programming language became more sophisticated and less core bound, the general use of computers spread to more mining companies and, more importantly, to more engineers. Present-day utilization of computers in the mining industry varies across a wide spectrum of activities: from operations research techniques to aid mine management through payroll processing to mine design and monitoring and control.

During the last years, efforts have been made with encouraging results towards planning of ventilation systems with computers. Computers have played a greater role in evaluating ventilation parameters and processes, largely because they can be used conveniently to relieve the ventilation engineer of repetitive calculations and to reduce voluminous ventilation data often collected for analysis of trends and their statistical significance. While these are basically data-reduction applications, the tremendous advantage of the computer for more sophisticated utilization is not fully realized. The great power of a computer lies in its ability to store, recall, manipulate, and modify data for analysis as well as generate copious information records, all in a fraction of a second. The application of energy and mass transport phenomena to the quantification of ventilation engineering parameters for quality and quantity control is not new. However, the solution of complex nonlinear systems of equations has taken a great leap forward since the advent of digital computers that permit complex ventilation problems to be solved numerically (Stefanko and Ramani, 1977).

General mine ventilation theory and practice are discussed in Chapters 11.6 and 11.7.

11.10.2 COMPUTER APPLICATIONS IN VENTILATION PLANNING

Most ventilation problems associated with the mine planning environment are essentially engineering problems and can be scientifically analyzed. Ventilation requirements and system structure can be defined only on the basis of a production plan. However, the iterative and feedback nature of this process must be noted since production planning must consider the services required to support an operation. Thus all ventilation problems are associated with the mine planning environment, and this in turn is usually related to the determination of the ability of existing ventilation plant to maintain certain legal standards. Legal standards defined in the Federal Coal Mine Health and Safety Act of 1969, which dictated major changes in mine ventilation with regards to both air quantity and quality requirements, have also been placed on the ventilation of belt lines, gob, and face areas. These and other similar requirements have not only precipitated several modifications to existing ventilation systems but also have greatly increased the complexities of ventilation planning in new mines. It is apparent that the use of computers in ventilation system analysis and design is a necessity.

A computer-oriented approach to mine ventilation planning began essentially in the late 1950s with the development of network simulation programs. For this reason, major developments in computer programming have taken place in the field of mine ventilation planning.

In designing the future ventilation system of a mine that is already in existence, two procedures are involved: (1) obtaining the physical details and specifications of the existing layout by means of ventilation surveys, and (2) the reduction and utilization of these data to study the effect of changes in some parameter on the characteristics of airflow in the existing layout, such as ventilation network analysis. The computer can provide powerful assistance in both these stages.

11.10.2.1 Data Reduction In Pressure Survey

An altimeter measures the instantaneous static pressure at a point underground in terms of the height of a theoretical air column of 50°F (10°C) dry air at the existing station atmosphere pressure. Assuming measurements were made simultaneously at two locations or corrected to the same time frame through the use of a stationary base instrument, corrections can be made for difference in elevation, air density, and airstream velocity to give the absolute pressure difference between two points. Calculation methods used have been described in the literature by several authors (Williamson, 1932; Krickovic, 1945–46; Mancha, 1946; Hall, 1951; McElroy and Kingery, 1957; and Hartman, 1961). The method developed at the US Bureau of Mines and described by McElroy and Kingery (1957) is probably the most commonly used method and has been adapted for use on a programable desk calculator by Harris et al. (1973) and for a hand-held calculator by Anderson and Nugent (1977). Several computer programs for the treatment of pressure survey data using digital computers have been described by Luxbacher et al. (1977). A sample data input for a pressure survey calculation identical to that used in McElroy and Kingery (1957) is listed in Table 11.10.1. The output of a program using this sample data is reproduced in Table 11.10.2.

11.10.2.2 Mine Ventilation System Network Analysis

A *mine ventilation system* is essentially a set of interconnected airways. The system includes the fan(s), airways, and regulatory devices. With added legal requirements placed on the ventilation of belt lines, gob and face areas, and air velocities on trolley haulage roads, the ventilation system can be quite complex, and all but simple problems require computational aids to ease and speed their solution.

The structural flow model constructed from a physical flow system such as the mine ventilation system can be conceptualized as a connected-directed graph. The term network is commonly used instead of graph, especially when quantitative characteristics such as flow, capacity, and resistances are assigned to the points and interconnections. A *ventilation network* consists of an interconnected group of branches and nodes. Branches or airways are the roadways, shafts, etc., through which the air flows

Table 11.10.1. Data Input for Pressure Survey Program

0	1000	2000	3000	4000				
0	1000	2000	3000	4000				
1								
14								
'Test deck—altimeter survey without altimeter temperature correction'								
1	BAS 8:22	1391	1391	53.	49.	482	0	53.
2	INT 8:35	1373	1387	53.	50.	496	0	50.
3	INT 8:59	1355	1010	64.	57.	120	475	50.
4	INT 9:10	1355	998	65.	57.	104	450	50.
5	INT 9:18	1350	952	65.	60.	57	390	50.
6	INT 9:45	1325	859	65.	62.	−28	360	50.
7	INT 10:15	1311	848	64.	62.	−27	230	50.
8	INT 10:30	1291	847	65.	63.	−17	130	50.
9	RTN10:50	1291	889	72.	70.	−16	475	50.
10	RTN11:37	1276	894	70.	69.	−24	350	50.
11	RTN 1:30	1274	1203	70.	70.	114	220	50.
12	RTN 1:53	1264	1410	70.	70.	317	880	50.
13	RTN 2:45	1241	1610	68.	68.	482	1590	50.
14	BAS 2:52	1239	1238	50.	43.	482	0	50.
'ALTIM'								

Source: McElroy and Kingery (1957).

and in which the state of the air can be measured. A branch begins and ends at a node, which is defined as a position at which two or more branches meet. A mesh is defined as two or more branches connected in series and forming a closed loop.

Basically, the solution of mine ventilation network problems satisfies the well-known Kirchhoff's laws. The techniques for numerically calculating the conditions laid down by these laws were established by Hardy Cross (1936). The procedure to be followed for the manual application of the Hardy Cross (1936) method can be summarized as follows:

1. Estimate the air quantity flowing through each branch of the network and the pressures developed by the fans. The estimated airflows should normally satisfy Kirchhoff's first law at each junction.

2. Examine the network and decide upon a pattern of closed meshes. The minimum number of meshes is given by

$$\text{number of branches} - \text{number of junctions} + 1 \qquad (11.10.1)$$

3. For each mesh, evaluate the mesh correction factor from the equation given below:

$$\Delta Q_m = \frac{-Fm}{2\Sigma\ R_i \mid Q_i \mid} \qquad (11.10.2)$$

where F_m is pressure drop around the mth mesh, ΔQ_m is the correction for the mth mesh branches, and R_i and Q_i are the resistances and the quantities, respectively, in branches constituting the mth mesh.

4. Correct the flow in each branch.

Repeat steps 4 and 5 until all values of ΔQ_m are below a specified level. A satisfactory flow balance will then be reached.

Consider a simple example of a mine ventilation system, as shown in Fig. 11.10.1, where the total quantity flowing and the resistances in the branches are given. There are two meshes, BCFEB and CDGFC, in the network. Fig. 11.10.2 represents a simplified sketch of the meshes. Initially, let the quantities flowing through the branches be estimated to satisfy Kirchhoff's first law, as shown in the Fig. 11.10.2. Kirchhoff's second law now can be applied to the two meshes, BCFB and CDFC. Identify mesh BCFB as mesh I and CDFC as mesh II. In all the calculations, the clockwise flow has been considered as positive. Calculation is terminated whenever the value of corrections for all the meshes is 100 cfm (0.047 m^3/s) or lower. Calculations for the first iteration are presented in Table 11.10.3, and the flow distribution at the end of the iteration is shown in Fig. 11.10.3. Note that the junction laws still remain satisfied (Ramani, 1980).

Because the correction quantities are greater than 100 cfm (0.047 m^3/s), the calculations are continued. Iteration 2 with its associated correction quantities and flows is presented both in Table 11.10.4 and Fig. 11.10.4. At this stage, both ΔQ_{I} and ΔQ_{II} are greater than 100 cfm (0.047 m^3/s) and therefore the iterations are continued. The calculations of the third iteration are shown in Table 11.10.5, and the flows are presented in Fig. 11.10.5. Now the correction quantities ΔQ_{I} and ΔQ_{II} are smaller than 100 cfm (0.047 m^3/s) and therefore the iterations are terminated since the desired accuracy has been reached. Note that in these calculations, it has been assumed that the fan pressure remains the same during the variations in Q_i. Such an assumption will hold only if the fan is operating on the flat portion of its characteristics. However, Scott and Hinsley (1951) have considered the changes in fan pressure due to quantity changes and have modified the iterative equation:

$$\Delta Q_m = \frac{-Fm}{2\Sigma\ R_i \mid Q_i \mid} - dF\,(Q_i)/dQ_i \qquad (11.10.3)$$

where $dF\,(Q_i)/dQ_i$ is the slope of the fan characteristic at the operating point $(F(Q_i),\ Q_i)$. With knowledge of iterative solutions, attention can now be directed towards the solution of multi-mesh networks.

Solutions to network problems required extensive time on mechanical calculators, and even with modern electronic digital calculators, they still can be very tedious and time consuming. As a result, in the 1950s, there was a move to electrical analogue computers. The normal square-law relationship between flow and pressure was initially accommodated in the variable resistance of electric light bulbs (Maas, 1950). The method has a major advantage of onsite availability that was emphasized by Kline and Suboleski (1972), but it is not very flexible when several networks are to be solved simultaneously.

The situation has changed again with the increased use of digital computers. Modern machines have high calculating speed and adequate storage capacity for solving ventilation network problems economically. The digital computer offers the best available method for solution of mine ventilation networks, and it can easily provide a high degree of accuracy for network solutions.

11.10.2.3 Computer Solution for Mine Ventilation Problems

The Hardy Cross method of network solution, explained in detail in the earlier section, lends itself particularly well to programming for digital computers. To illustrate the use of the digital computer for mine ventilation analysis, a network of moderate complexity solved previously by Trafton and Hartman (1964) is used. The ventilation system shown in Figs. 11.10.6 and 11.10.7 is representative of a coal mine being worked from two shafts and employing bleeders for methane drainage through the gob. Although the network may appear to be an uncomplicated one, it consists of 42 airways, 32 junctions, and 11 loops. Each junction in the network is numbered, and airways are designated by these numbers. The loops comprised by the airways are given letter-number combinations. The directions of

Table 11.10.2. Test Deck—Altimeter Survey Without Altimeter Temperature Correction

Data Input Station	Type	Time	Base Altimeter	Temp.	Traverse Altimeter	Dry-bulb Temp.	Wet-bulb Temp.	Elev.	Velocity
1	BAS	8:22	1391.	53.00	1391.	53.00	49.00	482.	0.
2	INT	8:35	1373.	50.00	1387.	53.00	50.00	496.	0.
3	INT	8:59	1355.	50.00	1010.	64.00	57.00	120.	475.
4	INT	9:10	1355.	50.00	998.	65.00	57.00	104.	450.
5	INT	9:18	1350.	50.00	952.	65.00	60.00	57.	390.
6	INT	9:45	1325.	50.00	859.	65.00	62.00	−28.	360.
7	INT	10:15	1311.	50.00	848.	64.00	62.00	−27.	230.
8	INT	10:30	1291.	50.00	847.	65.00	63.00	−17.	130.
9	RTN	10:50	1291.	50.00	889.	72.00	70.00	−16.	475.
10	RTN	11:37	1276.	50.00	894.	70.00	69.00	−24.	350.
11	RTN	1:30	1274.	50.00	1203.	70.00	70.00	114.	220.
12	RTN	1:53	1264.	50.00	1410.	70.00	70.00	317.	880.
13	RTN	2:45	1241.	50.00	1610.	68.00	68.00	482.	1590.
14	BAS	2:52	1239.	50.00	1238.	50.00	43.00	482.	0.

Instrument Temperature Corrections

Base Altimeter		Traverse Altimeter	
Reading	Correction	Reading	Correction
0.	0.000	0.	0.000
1000.	0.000	1000.	0.000
2000.	0.000	2000.	0.000
3000.	0.000	3000.	0.000
4000.	0.000	4000.	0.000
0.	0.000	0.	0.000

Instrument Readings (Corrected to Calibration Temperature) Converted to Barometric Pressure

Station	Base Altimeter	Base Barometer	Traverse Altimeter	Traverse Barometer	Corrected Traverse Barometer
1	391.	29.47285	391.	29.47285	29.47285
2	373.	29.49237	387.	29.47719	29.45767
3	355.	29.51192	10.	29.88890	29.86909
4	355.	29.51192	−2.	29.90210	29.90210
5	350.	29.51733	−48.	29.95276	29.94725
6	325.	29.54451	−141.	30.05542	30.02777
7	311.	29.55972	−152.	30.06758	30.05209
8	291.	29.58148	−153.	30.06870	30.04657
9	291.	29.58148	−111.	30.02226	30.02226
10	276.	29.59781	−106.	30.01674	30.00017
11	274.	29.59999	203.	29.67741	29.67522
12	264.	29.61089	410.	29.45226	29.44141
13	241.	29.63596	610.	29.23633	29.21159
14	239.	29.63814	238.	29.63922	29.63704

Station	Type	Time	Traverse Altimeter	Base Corr.	Altim. Diff.	Elev. Corr.	Density Ratio	Relative Humidity	Density	Static Press.	Total Press.	Sum Total
1	BAS	8:22	391.00	0.00	0.00	0.00	0.0000	75.54	0.075916	−0.0000	0.0000	0.0000
2	INT	8:35	387.00	17.98	13.98	−13.86	1.0101	81.49	0.075852	−0.0018	−0.0018	−0.0018
3	INT	8:59	10.00	18.01	−358.99	368.11	1.0214	65.20	0.75240	−0.1353	−0.1229	−0.1247
4	INT	9:10	−2.00	0.00	−12.00	15.48	1.0336	61.20	0.075191	−0.0520	−0.0554	−0.1802
5	INT	9:18	−48.00	4.98	−41.02	45.41	1.0351	75.02	0.075220	−0.0656	−0.0703	−0.2504
6	INT	9:45	−141.00	25.00	−68.00	82.04	1.0361	84.70	0.075365	−0.2107	−0.2123	−0.4638
7	INT	10:15	−152.00	13.99	2.99	−0.97	1.0356	89.54	0.075560	−0.0305	−0.0360	−0.4997
8	INT	10:30	−153.00	20.00	19.00	−9.65	1.0358	89.69	0.075382	−0.1406	−0.1431	−0.6428
9	RTN	10:50	−111.00	0.00	42.00	−0.96	1.0448	90.65	0.074175	−0.6169	−0.6055	−1.2483
10	RTN	11:37	−106.00	15.00	20.00	7.61	1.0508	95.15	0.74413	−0.4146	−0.4225	−1.6708
11	RTN	1:30	203.00	2.00	311.00	−131.55	1.0491	100.00	0.73565	−2.6793	−2.6846	−4.3554
12	RTN	1:53	410.00	10.00	217.00	−193.44	1.0494	100.00	0.072980	−0.3484	−0.3095	−4.6649
13	RTN	2:45	610.00	22.98	222.98	−157.57	1.0472	100.00	0.072727	−0.9599	−0.8730	−5.5379
14	BAS	2:52	238.00	2.00	−370.00	−0.00	1.0232	55.75	0.076894	5.4487	5.2893	−0.2486

Table 11.10.2. Test Deck—Altimeter Survey Without Altimeter Temperature Correction (cont.)

	Mine ATM CNDX				Standard CNDX			
Station	Static	Sum	Total	Sum	Static	Sum	Total	Sum
1	−0.00000	0.00000	0.00000	0.00000	−0.00000	0.00000	0.00000	0.00000
2	−0.00183	−0.00183	−0.00183	−0.00183	−0.00185	−0.00185	−0.00185	−0.00185
3	−0.13824	−0.14008	−0.12416	−0.12599	−0.13930	−0.14115	−0.12526	−0.12711
4	−0.05379	−0.19386	−0.05525	−0.18124	−0.05396	−0.19511	−0.05542	−0.18253
5	−0.06791	−0.26177	−0.07105	−0.25229	−0.06809	−0.26321	−0.07122	−0.25375
6	−0.21828	−0.48005	−0.21967	−0.47196	−0.21915	−0.48235	−0.22052	−0.47428
7	−0.03157	−0.51162	−0.03637	−0.50832	−0.03178	−0.51413	−0.03654	−0.51081
8	−0.14560	−0.65722	−0.14786	−0.65618	−0.14655	−0.66068	−0.14880	−0.65961
9	−0.64456	−1.30178	−0.63174	−1.28792	−0.64290	−1.30358	−0.62993	−1.28954
10	−0.43571	−1.73748	−0.44203	−1.72994	−0.43160	−1.73518	−0.43797	−1.72750
11	−2.81081	−4.54829	−2.81542	−4.54536	−2.77369	−4.50887	−2.77839	−4.50589
12	−0.36562	−4.91391	−0.32170	−4.86706	−0.35772	−4.86609	−0.31207	−4.81796
13	−1.00514	−5.91906	−0.89953	5.76659	−0.97655	−5.84264	−0.86760	−5.68556
14	5.57503	−0.34402	5.42240	−0.34419	−5.56325	−0.27939	5.41438	−0.27118

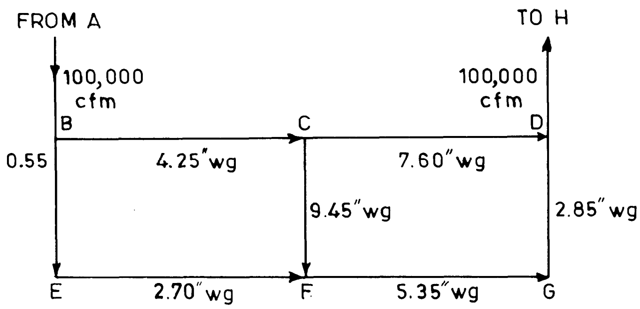

Fig. 11.10.1. Ventilation network of mine A without fan. Conversion factors: 1 cfm = 0.472 × 10^{-3} m^3/s, 1 in. water = 249 Pa.

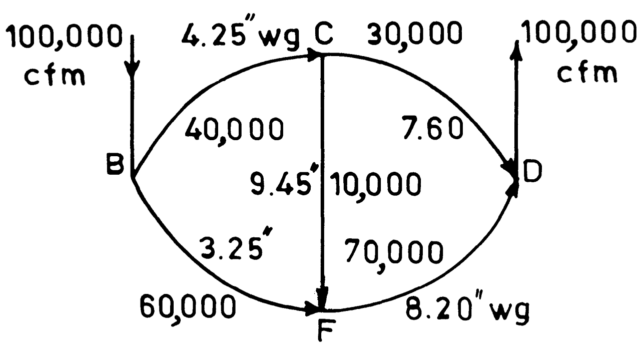

Fig. 11.10.2. Simplified network with initial quantities. Conversion factors: 1 cfm = 0.472 × 10^{-3} m^3/s, 1 in. water = 249 Pa.

flow indicated by arrows were assumed initially and are not necessarily the correct directions. The input data necessary in the problem, in the form in which they are read to the computer, appear in Tables 11.10.6 and 11.10.7.

The assumed flows that appear in Table 11.10.8 are those flows in each airway if 100,000 cfm (47.2 m^3/s) quantity of air is assumed to be entering the mine. This is demonstrated in Table 11.10.8 by the flow in airway 1-2. This assumed air input could be any, but some assumed amount of air must be going into the mine in order to solve the network problem. A quantity of 100,000 cfm (47.2 m^3/s) was chosen since this shows what percentage of the total amount of air input is distributed to the

Table 11.10.3. First Iteration (11.10.2.2)

Mesh I	Branch	*Q*	*R*	*R/Q/*	*R/Q/Q*
	bc	40,000	4.25	1.70×10^5	6.80×10^9
	cf	10,000	9.45	0.95×10^5	0.95×10^9
	fb	−60,000	3.25	1.95×10^5	-11.70×10^9
				4.60×10^5	-3.95×10^9

$$\Delta Q\ \text{I} = \frac{-(-3.95 \times 10^9)}{2\,(4.60 \times 10^5)} = 4.30 \times 10^3$$

Mesh II	Branch	*Q*	*R*	*R/Q/*	*R/Q/Q*
	cd	30,000	7.60	2.28×10^5	6.84×10^9
	df	−70,000	8.20	5.74×10^5	-40.10×10^9
	fc	−10,000	9.45	0.95×10^5	-0.95×10^9
				8.97×10^5	-34.21×10^9

$$\Delta Q\ \text{II} = \frac{-(-34.21 \times 10^9)}{2\,(8.97 \times 10^5)} = 1.91 \times 10^4$$

Balance the network:

$Q_{bc} = 40000 + 4300 = 44300$
$Q_{cf} = 10000 + 4300 - 19100 = -4800$
$Q_{fb} = 60000 - 4300 = 55700$
$Q_{cd} = 30000 + 19100 = 49100$
$Q_{df} = 70000 - 19100 = 50900$

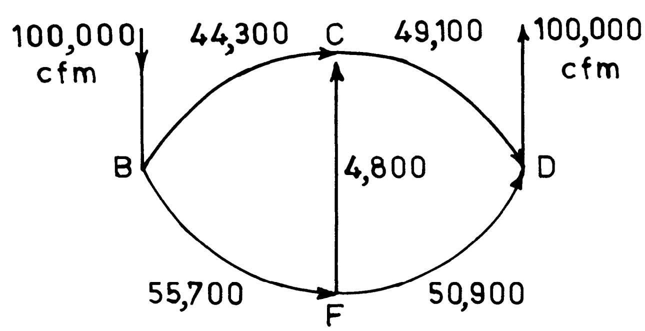

Fig. 11.10.3. Flows at the end of the first iteration. Conversion factor: 1 cfm = 0.472 × 10^{-3} m^3/s.

Table 11.10.4. Second Iteration (11.10.2.2)

Mesh I	Branch	Q	R	$R/Q/$	$R/Q/Q$
	bc	44,300	4.25	1.88×10^5	8.30×10^9
	cf	−4,800	9.45	0.45×10^5	-0.22×10^9
	fb	−55,700	3.25	1.81×10^5	-10.1×10^9
				4.14×10^5	-2.02×10^9

$$\Delta Q\ \text{I} = \frac{-(-2.02 \times 10^9)}{2\ (4.14 \times 10^5)} = 2.45 \times 10^3$$

Mesh II	Branch	Q	R	$R/Q/$	$R/Q/Q$
	cd	49,100	7.60	3.73×10^5	18.3×10^9
	df	−50,900	8.20	4.16×10^5	-21.2×10^9
	cc	4,800	9.45	0.45×10^5	0.22×10^9
				8.34×10^5	-2.68×10^9

$$\Delta Q\ \text{II} = \frac{-(-2.68 \times 10^9)}{2\ (8.34 \times 10^5)} = 1.59 \times 10^3$$

Balance the network:

$Q_{bc} = 44300 + 2450 = 46750$
$Q_{cf} = 4800 - 2450 + 1590 = 3940$
$Q_{fb} = 55700 - 2450 = 53250$
$Q_{cd} = 49100 + 1590 = 50690$
$Q_{df} = 50900 - 1590 = 49310$

Table 11.10.5. Third Iteration (11.10.2.2)

Mesh I	Branch	Q	R	$R/Q/$	$R/Q/Q$
	bc	46,750	4.25	1.99×10^5	9.30×10^9
	cf	−3,940	9.45	0.372×10^5	-0.147×10^9
	fb	−53,250	3.25	1.730×10^5	-9.20×10^9
				4.09×10^5	-0.05×10^9

$$\Delta Q\ \text{I} = \frac{-(-0.05 \times 10^9)}{2\ (4.09 \times 10^5)} = 60$$

Mesh II	Branch	Q	R	$R/Q/$	$R/Q/Q$
	cd	50,690	7.60	3.86×10^5	19.60×10^9
	df	−49,310	8.20	4.30×10^5	-19.80×10^9
	fc	3,940	9.45	0.37×10^5	0.15×10^9
				8.53×10^5	-0.05×10^9

$$\Delta Q\ \text{II} = \frac{-(-0.05 \times 10^9)}{2\ (8.53 \times 10^5)} = 30$$

Balance the network:

$Q_{bc} = 46750 + 60 = 46810$
$Q_{cf} = 3940 - 60 + 30 = 3910$
$Q_{fb} = 53250 - 60 = 53190$
$Q_{cd} = 50690 + 30 = 50720$
$Q_{df} = 49310 - 30 = 49280$

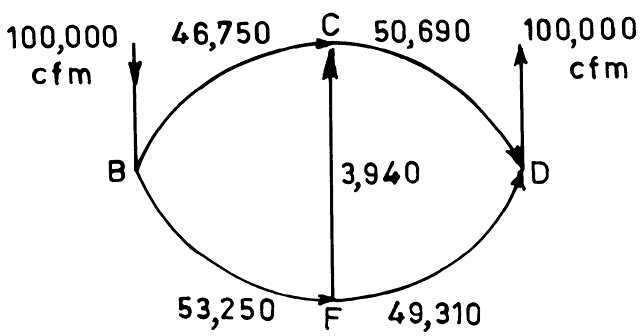

Fig. 11.10.4. Flows at the end of the second iteration. Conversion factor: 1 cfm = 0.472 × 10^{-3} m^3/s.

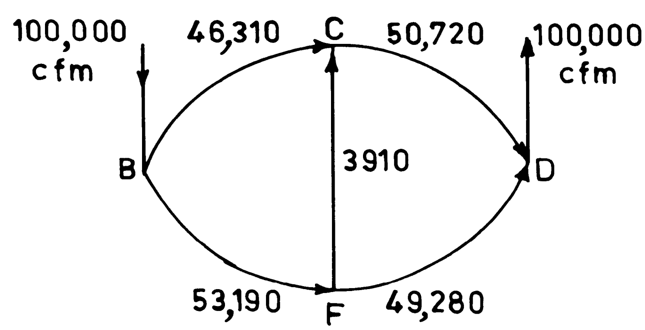

Fig. 11.10.5. Final flows in the network. Conversion factor: 1 cfm = 0.472 × 10^{-3} m^3/s.

various airways within the mine system. As an example, 40.3% of the total air input passes through airway 3–6, while 19.8% passes through airway 6–7. Of course, each mine air input varies, with the l00,000-cfm (47.2-m^3/s) value an input parameter, and a computer program is usually designed to allow any value to be used.

The adjusted values are those values of airflow through each airway that have been computed if a fan with the given characteristics had been installed externally to the mine ventilation network. In order to clarify this, airway 1–2 in Table 11.10.8, which as shown in Fig. 11.10.7, has 100,000 cfm (47.2 m^3/s) flowing within it if 100,000 cfm (47.2 m^3/s) is assumed to be entering the mine. However, if a fan of those given characteristics (Table 11.10.9) was installed, the fan H–Q characteristic curve and the mine characteristic curve is found by the computer to intersect at 265,608 cfm (125.4 m^3/s). This means that this fan is generating 265,608 cfm (125.4 m^3/s), which would be the mine air intake. Therefore, airway 1–2 has 265,608 cfm (125.4 m^3/s) of air flowing through it since 100% of the air input has to pass through this airway. Thus the value 265,608 cfm (125.4 m^3/s) is given as the adjusted flow with the fan in Table 11.10.8. Since airway 3–6 only has 40.3% of the air passing through it, then with a mine input of 265,608 cfm (125.4 m^3/s), only 106,995 cfm (50.5 m^3/s) will pass though airway 3–6.

From an original assumed mine quantity of 100,000 cfm (47.2 m^3/s) and a flow route 1–2–3–4–15–17–21–22–28–29–30–31–32, iterations were performed to correct the airway quantities until the cumulative error was reduced to 210 cfm (0.099 m^3/s). The total number of iterations required were 107. Then pressure losses were computed; values of H and Q, based on the assumed mine quantity, are given in Table 11.10.8. Direction of flow is indicated by the sequence of numbers designating each airway. The corresponding overall mine pressure loss is 0.37493 in. water (93.3 Pa). At the intersection point of the system curves, the

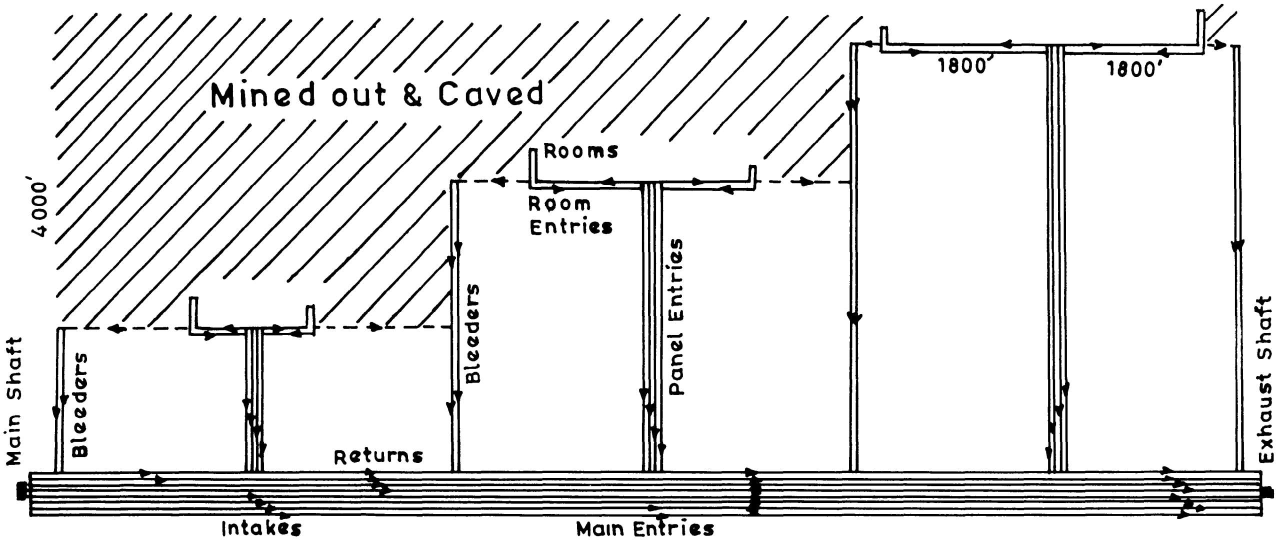

Fig. 11.10.6. Plan of room and pillar coal mine being worked with continuous equipment (Trafton and Hartman, 1964).

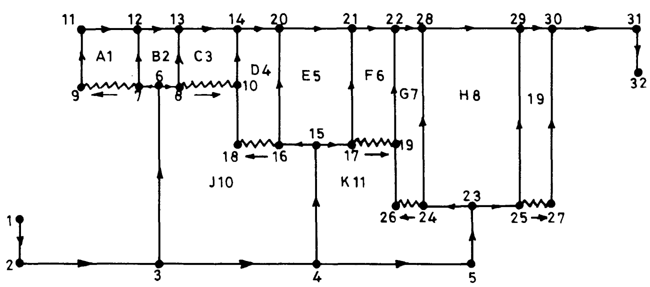

Fig. 11.10.7. Ventilation schematic of coal mine in Fig. 11.10.6. Complex network employing natural splitting with loops, airways, and junctions labeled and directions of flow indicated (Trafton and Hartman, 1964).

Table 11.10.6. Characteristics of Airways

Airway	Size, ft	Friction Factor $K \times 10^{10}$ lb-min^2/ft^4
main shaft	12 × 24	40
entries, rooms, crosscuts	6 × 15	60
caved ground	1 × 10	150
air shaft (31-32)	10 × 10	25

Conversion factor: 1 ft = 0.3048 m, 1 lb-min^2/ft^4 = 10^6 kg/m^3.

actual mine quantity was read as 265,608 cfm (125.4 m^3/s) and the mine pressure loss as 2.645 in. water (658 Pa). Using the proportion 265,608/100,000, new values of airway flows were computed. From the square of this proportion (since $H = RQ^2$), new values of airway head losses were likewise computed. These also appear in Table 11.10.8 and are the actual airway quantities and pressure losses that result when the given fan is installed in this mine. The computer program can be adapted to perform all calculations without intervention.

As expected, leakage through the gob is small (e.g., from 2081 cfm [0.98 m^3/s] in 7–9 to 6536 cfm [3.08 m^3/s] in 25–27). To increase the exhaust flow through the bleeders, room entries could be maintained open rather than caved; this may be desirable and would also serve to reduce the mine head somewhat.

The practical application of such a solution to the problem of natural splitting in a network may not be great, since practically every coal mine today employs controlled splitting. However, a rapid, simple means of obtaining accurate solutions by digital computer means that limits can be determined easily and a wide range of design variables can be explored readily.

11.10.2.4 Computer Programs For Solution Of Mine Ventilation Networks

Several computer programs have been developed in the United States (Hartman and Trafton, 1963; Wang and Hartman, 1967; Greurer, 1977; Hall et al., 1977; Stefanko and Ramani, 1972) and abroad (Hashimoto, 1961; Hitchcock and Hoover, 1976; McPherson, 1966) with the capabilities to solve complex ventilation networks and have been successfully used by many mining companies (Tien and Bjork, 1976; Mishra, 1973; Press and Johnston, 1976) and government organizations for analysis of complex ventilation systems. The use of ventilation network programs enables many more future situations to be investigated than would otherwise be practicable. It should perhaps be emphasized that the utilization of these programs does not require knowledge of their mode of operation. The ventilation engineer is intimately concerned with the preparation of his/her input data and analysis of the simulated results, but not with the mathematical and programming aspects of the network program. With these simulators, the mine ventilation engineer is provided with maximum information retrieval from the simulated network. In short, each possible ventilation alternative or situation can be easily tried on paper and evaluated before a final decision is taken. The general aspects of the information flow and logical structure of the mine ventilation simulator are shown in Fig. 11.10.8. For a detailed discussion of network solutions, see Wang (1982).

Table 11.10.7. Input Data (11.10.2.3)

Airway	Size, ft.	No.	Length, ft.	K_x 10^{10}	Airway	Size, ft.	No.	Length, ft.	K_x 10^{10}
1– 2	12×24	1	460	40	20–21	6×15	4	200	60
2– 3	6×15	6	2000	60	21–22	6×15	6	1800	60
3– 6	6×15	2	1400	60	15–17	6×15	1	1300	60
6– 7	6×15	1	750	60	17–21	6×15	1	3700	60
7–12	6×15	1	1950	60	17–19	1×10	1	650	150
7– 9	1×10	1	1200	150	19–22	6×15	2	2750	60
9–11	6×15	2	1400	60	4– 5	6×15	2	3600	60
11–12	6×15	2	1800	60	5–23	6×15	2	3800	60
12–13	6×15	2	200	60	23–24	6×15	1	1850	60
13–14	6×15	4	1800	60	24–28	6×15	1	5300	60
6– 8	6×15	1	700	60	24–26	1×10	1	100	150
8–13	6×15	1	1900	60	26–19	6×15	2	1050	60
8–10	1×10	1	1250	150	22–28	6×15	6	1800	60
10–14	6×15	2	1400	60	28–29	6×15	6	200	60
3– 4	6×15	4	3600	60	29–30	6×15	8	1800	60
4–15	6×15	2	2750	60	23–25	6×15	1	1800	60
15–16	6×15	1	1350	60	25–29	6×15	1	5250	60
16–20	6×15	1	3750	60	25–27	1×10	1	150	150
16–18	1×10	1	600	150	27–30	6×15	2	3800	60
18–10	6×15	2	1350	60	31–32	10×10	1	620	25
14–20	6×15	4	1800	60	30–31	6×15	8	200	60

Source: Trafton and Hartman (1964). Conversion factors: 1 ft = 0.3048 m, 1 lb-min²/ft⁴ = 1.855 × 10⁶ kg/m³.

Table 11.10.8. Corrected Quantities and Head Losses for Airways Assumed and Adjusted Values (11.10.2.3)

Number of Iterations = 107 **Cumulative Error = 210 cfm (for Assumed Flow Values)**

Assumed Original Flow			Adjusted Flow with Fan		Assumed Original Flow			Adjusted Flow with Fan	
Airway	*Q*, cfm	*H*, in. water	*Q*, cfm	*H*, in. water	Airway	*Q*, cfm	*H*, in. water	*Q*, cfm	*H*, in. water
1– 2	100,000	.01067	265,608	.07524	20–21	57,044	.00270	151,513	.01908
2– 3	100,000	.03693	265,608	.26054	21–22	73,089	.01776	194,130	.12526
3– 6	40,283	.03776	106,995	.26636	15–17	17,464	.02636	46,386	.18595
6– 7	19,800	.01955	52,591	.13789	17–21	16,045	.06332	42,617	.44672
7–12	19,016	.04688	50,509	.33071	17–19	1,419	.08305	3,769	.58593
7– 9	784.	.04677	2,081	.32992	19–22	3,152	.00045	8,371	.00320
9–11	784.	.00001	2,081	.00010	4– 5	25,492	.03888	67,709	.27429
11–12	784.	.00002	2,081	.00013	5–23	25,492	.04104	67,709	.28952
12–13	19,800	.00130	52,591	.00919	23–24	12,304	.01862	32,681	.13135
13–14	39,382	.01160	104,602	.08183	24–28	10,571	.03937	28,079	.27778
6– 8	20,483	.01952	54,404	.13776	24–26	1,733	.01905	4,602	.13442
8–13	19,582	.04843	52,012	.34168	26–19	1,733	.00005	4,602	.00037
8–10	901	.06436	2,392	.45404	22–28	76,241	.01932	202,501	.13630
10–14	1,994	.00009	5,298	.00065	28–29	86,812	.00278	230,580	.01964
3– 4	59,717	.05334	158,613	.37630	29–30	97,539	.01779	259,072	.12549
4–15	34,225	.00535	90,904	.03777	23–25	13,188	.02081	35,028	.14682
15–16	16,761	.02521	44,518	.17786	25–29	10,727	.04016	28,492	.28332
16–20	15,667	.06119	41,613	.43167	25–27	2,461	.07564	6,536	.40661
18–10	1,094	.00002	2,905	.00019	27–30	100,000	.00208	265,608	.01466
14–20	41,377	.01280	109,900	.09033	30–31	100,000	.11923	265,608	.84114
16–18	1,094	.04555	2,905	.32136	31–32	2,461	.00038	6,536	.00270
					TOTAL	100,000	0.37493	265,608	2.64500

Conversion factors: 1 cfm = 0.472 × 10^{-3} m³/s, 1 in. water = 249 Pa.

11.10.3 COMPUTER APPLICATION TO FLOW PROBLEMS IN MINE VENTILATION

Most mine air pollutants, such as respirable dust, methane, diesel exhaust, and blasting fumes, are produced in face areas and haulageways. Their dispersal in mine airways contribute to interesting flow or transfer problems. Knowledge about sources, emission, and transport of these pollutants is essential for ventilation planning and design to ensure good engineering control over air quality, quantity, and environmental conditions. These physical phenomena in mine ventilation involve transient heat

Table 11.10.9. Fan Characteristics (11.10.2.3)

Q, cfm	*H*, in. water
100,000	7.6
150,000	7.3
200,000	6.3
250,000	3.6
300,000	0

Conversion factors: 1 cfm = 0.472×10^{-3} m^3/s, 1 in. water = 249 Pa.

and mass transfer and can be described by parabolic differential equations (PDEs). Solution of these equations in the past was normally attempted using analytic approaches such as Laplace transforms. Even for simple problems, when account is taken of only one of the variable factors such as harmonic oscillations, stepwise changes, etc., calculations become complex. The advent of digital computers and advances in applied mathematics has made it possible to describe emission, dilution, and dispersion of contaminants into the mine atmosphere. The theoretical investigations into pollutant flow and diesel exhaust to be presented here are examples of ventilation problems that are completely computer-dependent. The computer is programmed to solve mathematical models, thereby enabling the flow phenomena to be simulated by varying parameters such as the characteristics of the gas source, boundary conditions, and air velocities (see Chapters 11.2, 11.3, and 11.5 for general coverage of contaminants).

11.10.3.1 Simulation Of Gas Emission And Transport

The major classifications of gases that enter a mine ventilation system are (1) flow through airways (herein, fully developed laminar or turbulent transfer of air, gas, and dust takes place in the airways, the geometry and surface properties of which are well defined); (2) flow through caved workings (flow through caved working can be laminar or turbulent, depending on the degree of consolidation of the broken strata and the pressure difference across the area); and (3) flow through porous media (an example is the seepage of methane from coal beds and adjacent strata in situ). The transport of dust, methane, and diesel exhaust in mine airways is but a variant of a diffusion-dilution problems relevant to underground mining practice.

Modeling of mass transfer phenomena in mine ventilation is largely based on extensive studies of turbulent dispersion in wind tunnels and the lower atmosphere of the earth. The general approach in the formulation of models lies in the recognition that these mass transfer phenomena can be conceptualized as a flow problem that can be solved by the general application of Taylor's theory (1954) of turbulent flow and diffusion with modification appropriate to the relevant static and dynamic parameters. If the face zone is initially contaminated by a pollutant over a long section (e.g., instantaneous exposure of large gas accumulations, rapid methane release, etc.), the process of decontaminating the mine workings can be regarded as displacement and diffusion of the trailing edge of the cloud of contami-

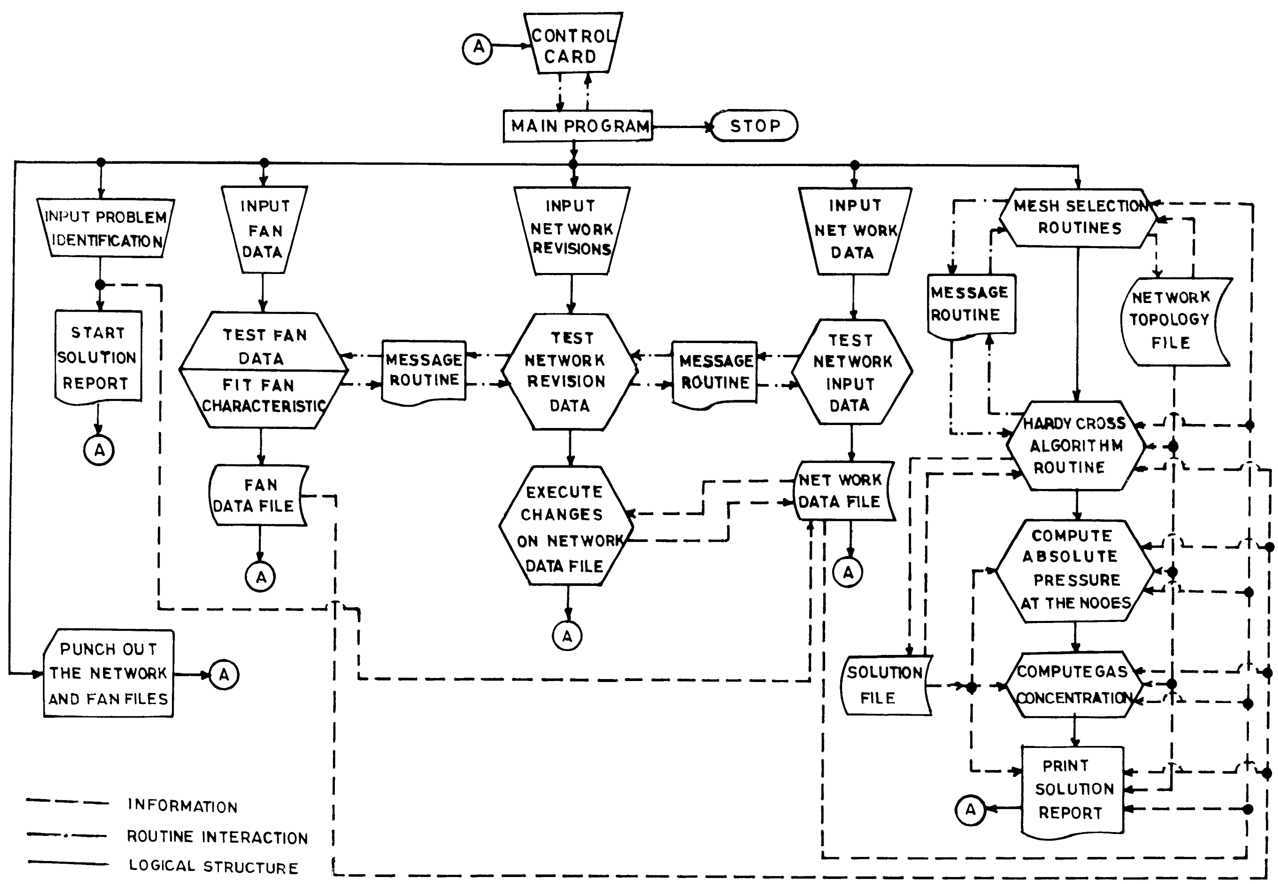

Fig. 11.10.8. Information and logical structure of the mine ventilation simulator (Didyk et al., 1979).

nants in the form of a semi-infinite gas body, for which a one-dimensional mass transfer equation is represented by:

$$\frac{\delta C}{\delta t} + \frac{u\delta C}{\delta x} = Ex\,\frac{\delta^2 C}{\delta x^2} + f(x,t) \qquad (11.10.4)$$

In Eq. 11.10.4, the term $\delta C/\delta t$ is the rate of growth of the concentration in the differential element, while $u(\delta C/\delta x)$ is the net gain of material due to convective transfer. The two terms balance the total loss of material due to turbulent dispersion that is represented by $Ex\,\frac{\delta^2 C}{\delta x^2}$, and $f(x,t)$, the source term for the pollutant in the roadway.

The approach in modeling is to use the above partial differential equation to predict the concentration growth in an airway.

Equations containing first-order spatial derivatives, such as the equation shown above, are known as "convection-diffusion" equations because of the physical processes that they describe. Typically, a suspended material (such as diesel exhaust pollutant) is carried along (convected) by a fluid at the same time as its concentration is being attenuated (diffused) within the flow. Numerically, more interesting problems arise when the first-order (convection) spatial derivatives are large in relation to the second-order (diffusion) derivative; that is, the solution behaves like the solution to the limiting hyperbolic (purely convective) case. In the converse situation, when diffusion is dominant, ordinary methods of parabolic equations can be used.

By employing numerical methods of solving partial differential equations for turbulent dispersal, the concentration of pollutants can be determined for any given source configuration, mine geometry, and air distribution (Bandopadhyay and Ramani, 1988). The option of various numerical solution techniques that are available to an analyst to the convection-diffusion equation for cases in which convection is dominant include

space discretization	finite difference finite element (consistent) finite element (lumped)
time discretization	implicit explicit
"tricks"	slope continuity elements upwinding penalty functions dispersion correction higher-order time scheme

11.10.3.2 Simulation of Diesel Exhaust Dispersion and Transport

The application of Taylor's theory (1954) to the study of diesel exhaust dispersion in mine airways appears to be an extension of the Russian work on dispersal of methane, blasting fumes, and dust. Skobunov (1970) recognized that ventilation calculations based on traditional dilution formulas did not adequately explain observed conditions. One reason was that multiple contaminations of the mine air could conceivably occur as a result of several vehicles operating in the same or parallel-connected entries. In addition, Holtz and Dalzell (l968) developed an effective ventilation concept for explaining the changes in concentration due to differential vehicle and air movement in mine entries. Models developed by Skobunov (1970) calculated the growth of contaminants in mine airways for single or multiple diesel vehicles. These models allowed consideration of the absorption of gases and air losses due to leakage. In later work, Skobunov (1974) examined the concept of turbulent flow across a mine entry cross section and developed a series of models based on transverse turbulent diffusion. These later models are applicable to moving axisymetric sources such as diesel vehicles, and linear sources such as a longwall plough.

Other Russian researchers (Osipov and Grekov, 1968) suggested the use of a convection-diffusion transport model to describe the transport of a gaseous pollutant in confined ventilation currents. Later, Grekov and Kalyusskii (1972) examined the concept of convection-diffusion transport of pollutants in which the current velocity varies with distance owing to air leakage.

The interest in increasing use of diesel-powered equipment in US coal mines has led to the development of mathematical models for the emission, dilution, and dispersion of gaseous pollutants from diesels. Stefanko et al. (1974, 1977) developed and evaluated diesel dispersion models through a grant from the US Bureau of Mines.

Diesel Exhaust Flow Models. When a diesel engine is traveling in an airway, load and speed change with time as a function of the engine duty cycle. The engine exhaust volume and composition are a function of engine speed and power, making it necessary to estimate the change in the pollutant volumes with time and location as a function of engine speed and load. To estimate the source function, it is necessary to know the engine speed, engine load, and vehicle speed at frequent intervals to account for variation in the pollutant flow rate. It would not be practical, however, to instrument every teletram or similar diesel-powered vehicle to acquire those data. It is possible, however, to generate the engine duty cycle and exhaust volumes and analyses on the basis of the simulation of a given production system (Ramani and Kenzy, 1978). One advantage of this approach is that several diesel-powered equipment deployments can be analyzed to generate values for the parameter of interest.

Generally, diesel engines move faster than air currents in the face areas and, consequently, the air that flows through the haulage road is contaminated several times before it is finally discharged into the return airways. This leads to a progressive rise in concentration in both the spatial and time axes owing to the superposition of contaminants. The concentration profile does not reach a steady-state situation because the velocity vector is not uniform. The concentration profile will not be fully developed in the face area because, when a moving plane source is used, the distance required to obtain adequate mixing to reach a steady-state situation is often greater than that found in the face area. The concentration gradient in the return airway, however, reaches steady-state. Air velocity affects the concentration growth in the return. The total contamination in the air at any point is the sum of the incremental contaminants from diesel engine(s) passed by the air. When the engine and the air are traveling in opposite directions, the predominant feature in the mass flow is the convective transfer, which causes deformation of both the front and rear profiles of the contaminant clouds. As the source moves further away from the face, the role of convective transfer gradually decreases along the length of the airway. In effect, the contaminant concentration front is an increasing function of distance and time from the face. Utilizing the conservative volume element methods, convection-diffusion mathematical models were developed for analyzing the growth of pollutant concentration for each of the following situations:

1. A single engine moving in an airway with or without considerable leakage.
2. Multiple engines in a single airway.
3. Multiple engines in a network of roadways.

The physical situation for the model in case 1 is operation of a diesel engine for face haulage. Operation of diesel engines for secondary and primary track haulage can be studied with the models for the other cases. Bandopadhyay and Ramani (1983, 1984, 1985) have detailed the finite difference approximation of the convective-diffusion equations and the boundary conditions used for the above model solutions.

11.10.3.3 Simulation of Methane Flow in Mine Airways

Much of the early work on the use of computers to simulate methane flow rates in mine airways was carried out during the 1960s, and only two are claimed to have been used with any success (Lidin, 1964; Airey, 1968).

Lidin's model is an empirical equation based on extensive research and compilation of actual gas emission data. Airey's model, on the other hand, was developed on the basis of the Darcian equation of gas migration through homogenous, permeable solids and can be solved numerically or analytically.

The quantity of methane emitted into the mine atmosphere and the movement of gas through solid coal and the adjoining country rock are dependent on the physical properties of the medium, boundary conditions, the initial gas pressure distribution, and the combination of natural and mining factors. Functional relationships among the factors are not yet known (Ramani and Owili-Eger, 1974). Consequently, the development of rigorous mathematical models to predict methane flow into mine air is difficult, and to date no model has been reported that is capable of such a function. The mathematical model that Ramani and Owili-Eger used is given in Eqs. 11.10.5 and 11.10.6, which are further attempts in presenting practical models for predicting the quantity of methane emission into the mine environment. Eq. 11.10.5 is the steady-state model and Eq. 11.10.6 is the unsteady state model:

$$\delta/\delta x \left\{K_x \frac{\delta P^2}{\delta x}\right\} + \delta/\delta y \left\{K_y \frac{\delta P^2}{\delta y}\right\} + W(x,y) = 0 \qquad (11.10.5)$$

$$\frac{\delta}{\delta x}\left\{KxA_x \frac{\delta P^2}{\delta x}\right\}\Delta x + \delta/\delta y\left\{K_yA_y \frac{\delta P^2}{\delta y}\right\}\Delta y + 0.1\ MZTq = \frac{M\phi CV_b}{5.615}\frac{\delta P^2}{\delta t} \qquad (11.10.6)$$

where P is gas pressure; $W(X,Y)$ is the source term for the steady-state model; x and y are the coordinate axes; K_x and K_Y are the directional permeabilites; A_x and A_y are the cross-sectional areas of the medium or block in the x direction and y direction, respectively; x and y are the dimensions of the block in the x direction and y direction, respectively; M is fluid viscosity; Z is gas deviation factor; q is gas flow rate; T is absolute temperature (in °R); ϕ is medium porosity; C is the gas compressibility coefficient; and V_b is volume of the block or medium. The variable t is the time. The source term is expressed as

$$W(X,Y) = \frac{0.1MZT}{V_b}q \qquad (11.10.7)$$

A finite difference analysis is used for solution of the above equations, each node being allocated initial rock properties and gas pressure. The model reacts to the changes that take place with respect to time. The flow of methane from block to block towards the ventilated airways is thus simulated. The simulation program provides output information on pressure distribution and methane flow rate at each block. The flow diagram for the methane generator program is shown in Fig. 11.10.9.

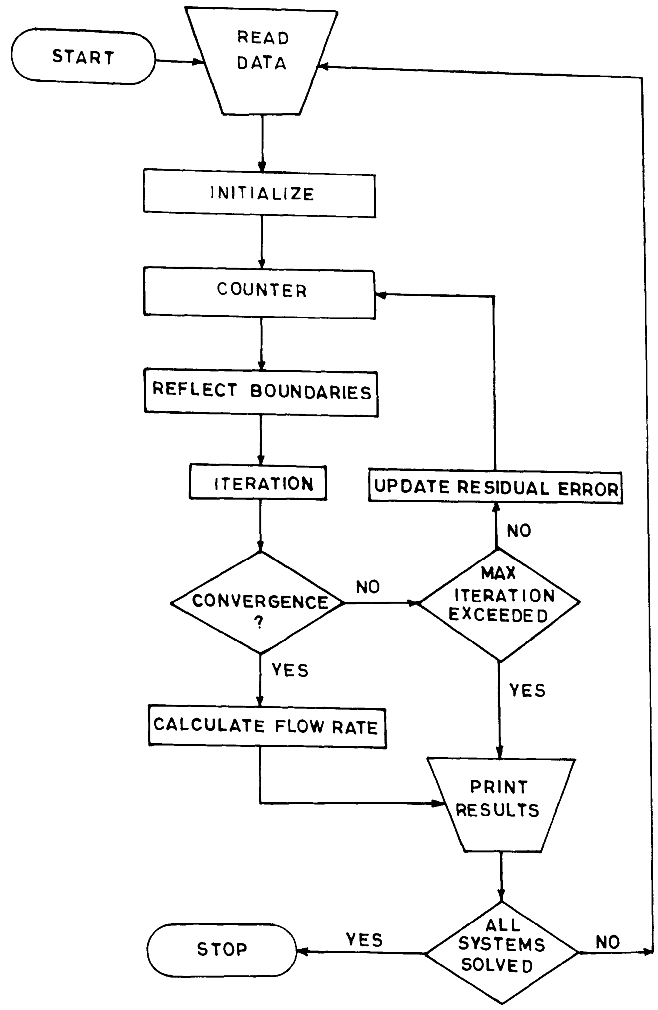

Fig. 11.10.9. Methane simulator flow diagram (Ramani and Owili-Eger, 1974).

In the United Kingdom, a method is being developed for prediction of methane emission into longwall workings. The method is based on the theoretical treatment of gas emission from coal seams (Airey, 1968; Dunmore, 1980). The method permits calculation of the emission rate from knowledge of the depth of the working, the sequence of gas-bearing coal seams in the disturbed strata, the initial gas content of these strata, and the proposed dimensions and the advance of the face. The National Coal Board's Mining Research and Development Establishment (MRDE) has developed a computer program to enable the emission rate to be calculated as a function of time (Curl, 1978). The program considers the shift-by-shift changes in the face advance, gives the required airflow to dilute methane concentrations to acceptable levels, and allows for the peaking of methane emission during the production periods. The program also enables the amount of gas liberated to be calculated at various positions in the gate roadways.

11.10.4 COMPUTER-AIDED MONITORING AND CONTROL SYSTEM

Recent advances and changes in requirements and philosophy have led to added emphasis being placed on remote control and monitoring in mines, particularly in relation to the underground environmental quality in mines worldwide. Hitherto, methods of monitoring the mine environment have been based on periodic spot observations by hand-held instruments. Such methods are labor-intensive, are expensive, provide only limited amounts of information, and sometimes suffer from the disadvantage that a comparatively long time may elapse between measurements and availability of results. The changing and varying characteristics of some hazards and the recent successful development of suitable sensors and transducers and monitoring systems have emphasized the need for, and benefits to be derived from, monitoring environments mine-wide.

Computer-based mine monitoring and control systems are the mining industry's state-of-the art use of specialized computer technology—both hardware and software—to "automate" a mine. It is a computerized data acquisition and analysis facility with the capability of controlling underground electrically powered equipment. These systems lend themselves readily to remote monitoring of the mine environment for continuous indication of state and early warning of abnormal conditions and adverse environmental trends. In addition, such systems can be applied to monitor and control machinery forming part of a mine environmental system, such as booster, auxiliary, and surface mine fans. Characteristics that can be sensed by the system include temperature, pressure, velocity flow rate, vibrations, and essentially any other parameters that can be measured and presented as electrical signals to the monitoring and control system. Environmental factors such as methane concentrations, carbon monoxide, diesel exhaust, oxygen, smoke, air velocity, and dust are sensed using available instruments.

For a general discussion of monitoring and control, see Chapter 12.6.

11.10.4.1 Monitoring System

Remote environmental monitoring enables continuous interrogation of selected parameters at one or more strategic positions to be made so that current values can be determined, variations detected and evaluated, and early warning given of abnormalities and adverse trends. Action can then be considered and, if necessary, taken more quickly than at present to correct the effects of an abnormal condition. Situations prior to, during and after abnormal events such as spontaneous methane emission, can be evaluated and factual information provided on trends, analysis, and potential remedies. Continuous monitoring of the underground environment involves interrogation and overall surveillance of certain parameters at strategic environmental positions at predetermined time intervals. A large number of on-off (digital or binary) signals are now available (Table 11.10.10) through two-state switches, relay instruments, instrument alarms, and the like. Analog functions, on the other hand, vary continuously to provide precise measurements of gas concentrations, pressures, velocities, flows, temperatures, etc.

Available environmental monitoring equipment includes

1. Methane monitors.
2. Smoke monitors.
3. Air velocity monitors.
4. Carbon monoxide monitors.
5. Carbon dioxide monitors.
6. Oxides of nitrogen monitors.
7. Relative humidity monitors.
8. Pressure monitors.
9. Temperature monitors.
10. Vibration monitors.

These digital and/or analog sensors are connected to "central station" input circuits. The central station contains electronics to perform processing of sensor signals, and reduce the data to digital form. The total number of sensors employed underground can range from one to hundreds, depending on the size of the system and layout philosophy of the mining operation. Various aspects of data collection and processing information are discussed with reference to Fig. 11.10.10, shows the physical configuration of the Conspec mine-wide monitoring and control system.

11.10.4.2 Conspec Senturion System

Specially designed accessor cards are the backbone of the Conspec system. The accessor is a single-point data transmittor/receiver processor and "intelligent," allowing the central computer to monitor and control all the connected equipment. The major advantage of the accessor is that it reduces the installation wiring cost and increases system reliability. The central microcomputer station communicates over four accessor trunks. The accessors have specially designed chips that multiplex the data for communication to the central station. The accessors are in the form of printed circuit cards upon which all the necessary logic circuitry is mounted. Reprogrammable rocker switches on the accessor allow address selection and point-data transmission.

Accessors are available that control equipment from simple start/stop functions to linear devices requiring proper feedback for control.

The primary means of displaying information is a CRT screen. The screen and its keyboard are the link from the system to the operator. Through this, the operator programs the system, receives a visual description of device status, and transmits signals to required accessors.

To set up the system, it is a simple matter of knowing what code each individual accessor was assigned and on which trunkline it is communicating. From here, the processor prompts the operator into entering information such as identification number, type of accessor and device being monitored, limits of operation and scaling factors, and a description of the device. Once a device is coded and entered in the system, it is ready to transmit information.

The central station can also be equipped with a variety of CRTs and printers. The use of two printers allows regular alarm and status change events to be printed simultaneously.

The CRT and printers can interact in a variety of ways. When a point is assigned to the system, it is given a code which dictates which way it appears on the screen and printers, and whether or not an alarm is sounded. In this way, repetitious events such as opening and closing of a ventilation door can be coded not to appear on the CRT or printer but to sound an alarm if opened too long. In all cases, an alarm will sound if a problem arises in the hardware of the system.

Environmental monitoring with a mine-wide monitoring and control system has provided specific benefits in US mines in relation to existing and potential hazards. Specific problems, for example, floor and roof methane emissions, high emission rates, etc., have been assessed and countered.

Mine-wide monitoring systems offer considerable potential and scope for continuous, essential, and factual information on particular aspects of the environment in relation to other parameters such as the incidence of methane peak flows or significant changes in methane emission rates. Computer programs are be-

Table 11.10.10. Selected Remote Environmental Monitors Used In British Mines

Parameter	Monitor and/or Switch	States
General body firedamp concentration in airways	BM.1 Single-head methane	Analog and/or ON/OFF
General body mine gases	Tube bundle	Analog and/or ON/OFF
Carbon monoxide	Tube bundle	Analog and/or ON/OFF
Air velocity in roadways	BA.1 Single-headed air velocity monitor	Analog and/or ON/OFF
Smoke	Trolex mine smoke detector	ON/OFF
Vacuum in firedamp drainage pipes	Vacuum switch and indicating unit	Analog and/or ON/OFF
Combined vacuum and flow in firedamp drainage pipes	Combined vacuum differential switch and indicating units	Analog and/or ON/OFF
Purity of drained firedamp	Acoustic methanometer Tube bundle	Analog and/or ON/OFF Analog and/or ON/OFF
Vibration levels in operating fans and firedamp extractors	Vibration monitor	Analog and/or ON/OFF
Ventilation system pressures	Pressure switch and/or recorder	Analog and/or ON/OFF
Temperature of fan or extractor motor carcases or water	Surface or probe type temperature monitors	ON/OFF
Auxiliary fan or multi-unit booster fan clusters ducts air velocities	IS or flameproof air flow switch	Analog and/or ON/OFF
Multi-unit booster fan clusters isolation doors' position	Proximity switches	ON/OFF
Auxiliary ventilating duct air pressures (suction or pressure)	Pressure or vacuum switches or recorder	Analog and/or ON/OFF
Firedamp concentration in auxiliary fan ducts (exhaust systems), or at auxiliary fan site	BM.1 Single-head methane monitor	Analog and/or ON/OFF
Auxiliary fan running condition	Vibration monitor	ON/OFF

Source: Dunn and Swift, 1977.

ing developed to analyze such situations and determine interrelationships of variables. The results of such analysis are bound to lead to improved and safer working conditions.

11.10.5 EXPERT SYSTEMS IN MINE VENTILATION

Much has been discussed on the suggested use of on-line computer systems for monitoring and control of the underground environment (Chapter 22.2). The next logical step in the development of these systems is to add predictive capabilities to simple reporting functions, such as alerting the operator that signs of spontaneous combustion are beginning to show in a district. Also in the sensor technology, development of "intelligent" sensors capable of acting independently is also a possible step in the future. In an emergency, intelligent sensors would not wait for a command from the central computer but would act independently to shut down electrical power, sound alarm, etc., thus speeding up response time.

Work is in progress in several countries on software systems, commonly known as "expert systems," that would combine in a real-time basis (1) the expert knowledge and response in a decision situation; (2) the existing knowledge based on laws and regulations, mine layout, etc.; and (3) real-time information gathered by the mine monitoring systems. Ramani and Prasad (1987) reported on the essential elements of such a system (Fig. 11.10.11) that can be used for mine ventilation system design. Three major elements identified in the system are

1. The knowledge base and its input-output interfaces to the design and analysis program (DAP).

2. The design and analysis programs, consisting of a ventilation database and ventilation programs.

3. The actual mine system which is operated on by natural factors and from which data can be collected on a real-time basis.

Much of the internal knowledge in the knowledge base is experience-dependent and can be acquired from experienced mine ventilation practitioners and from public domain sources. An integration of the expert system and traditional algorithmic programs would yield a superior decision support system for mine ventilation design. Development of such system is possible with the current state of system technology.

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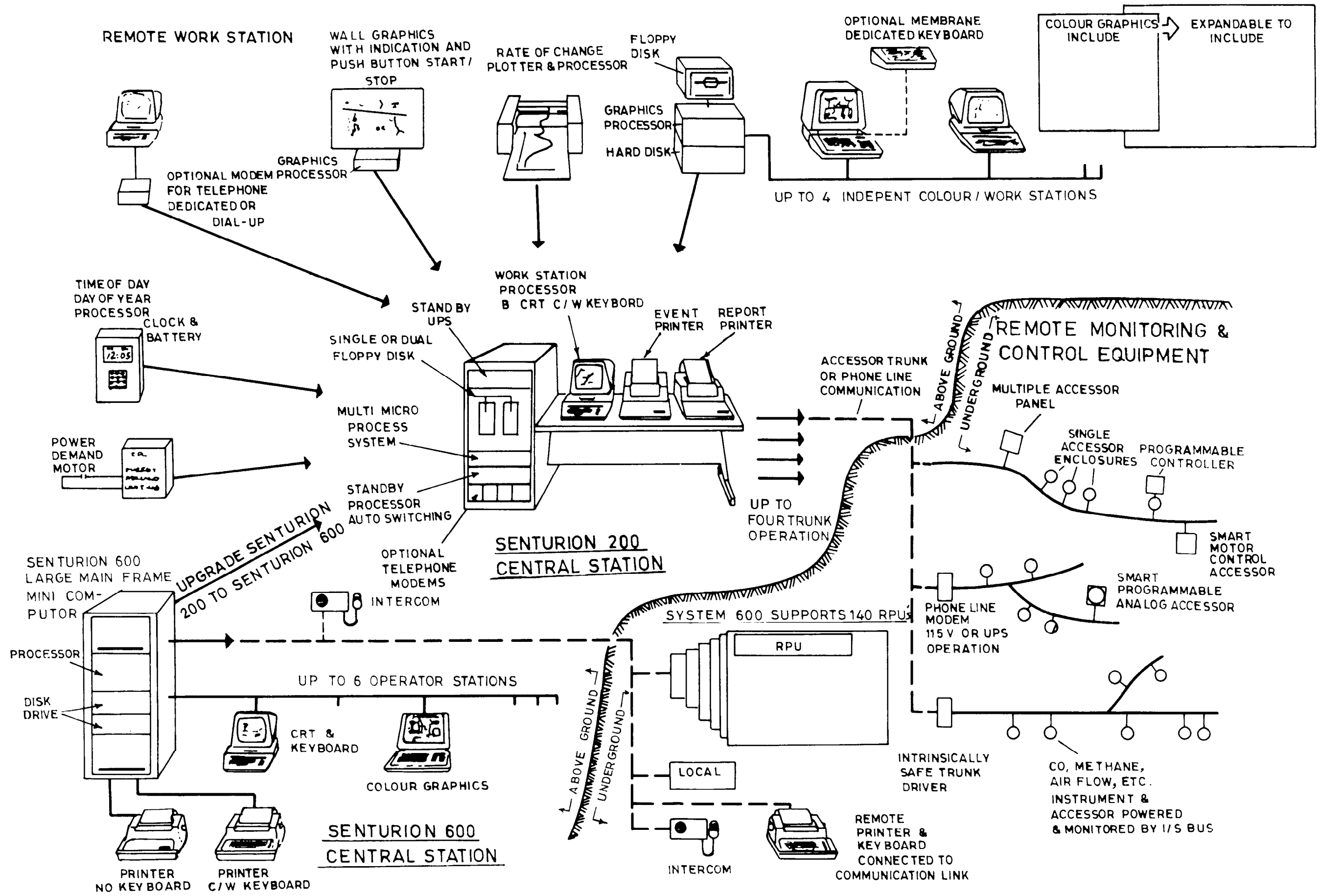

Fig. 11.10.10. Mine monitoring system (by permission from Senturion Mine Management System).

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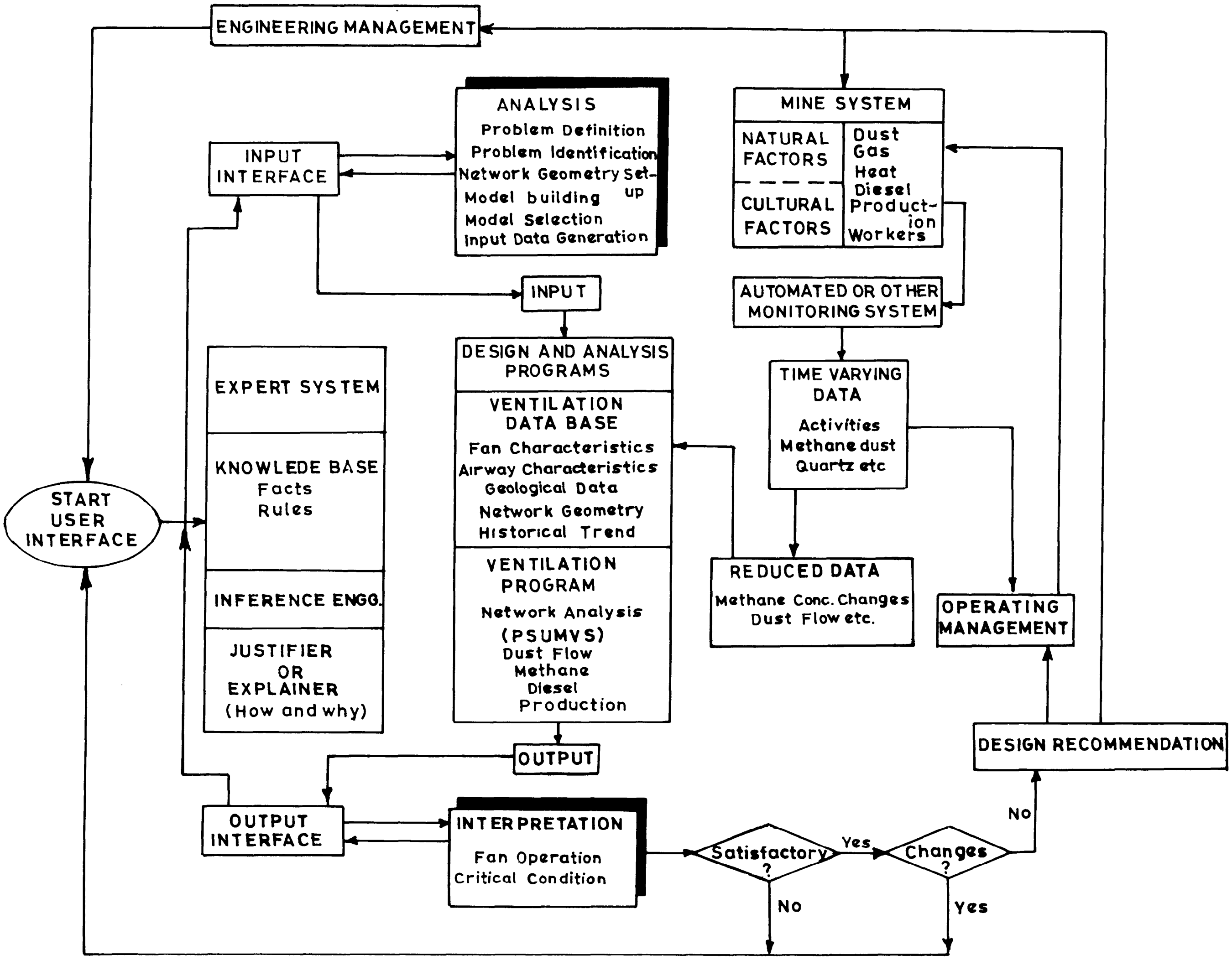

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Chapter 11.2
GAS AND DUST CONTROL

FRED N. KISSELL

11.2.1 METHANE CONTROL IN UNDERGROUND COAL MINES

The most common control method for methane gas and respirable coal mine dust is dilution ventilation. However, this chapter considers only face ventilation measures, which in practice must be integrated with overall mine ventilation planning to ensure delivery of an adequate amount of air (see Chapters 11.6 and 11.7).

All coalbeds contain methane, the amount of which can vary from 0.01 to more than 600 ft^3 /ton (0.0003 to 19 mL/g). Usually, the amount of methane in coal increases with higher rank and greater depth. The gas is both adsorbed on the micropore structure of the coal matrix and compressed in the fracture system of the coalbed. Wherever coal is exposed by erosion or mining, the equilibrium that exists in the coalbed under confining pressure is disturbed, and methane is emitted.

The methane emission from a prospective coal mine can be forecasted using the "direct method" test. This test employs coal cores taken during exploration drilling. Gas seeping from the core is measured and used to forecast the methane emission from the mine (Diamond, LaScola, and Hyman, 1986).

US underground bituminous coal mines emit more than 256 million ft^3 of methane (7.25 M) daily. Two hundred mines account for 98% of this, but they also account for 60% of underground coal production. In the early years of coal mining, inadequate ventilation, failure to test for gas (methane), use of open lights, smoking, failure to remove dust accumulations, and the improper use of black powder for blasting were the most frequently cited causes in explosions. Although both the number of explosions and the number of fatalities per million tons of coal produced has declined, approximately 10% of all underground coal mine fatalities are still due to explosions. Deul and Kim (1988) have provided a good general reference on methane control research up to about 1980. An expert system computer program to give advice on controlling methane is available from the US Bureau of Mines (USBM) (Kissell and King, 1988).

Noncoal mines, especially those located near carbonaceous deposits, may also emit methane.

11.2.1.1 Methane Control in Advance of Mining

Control in advance of mining is accomplished with vertical boreholes, horizontal boreholes, and directional slant holes. The common characteristic of these is that they are all implemented remote from mining operations.

VERTICAL BOREHOLES. Vertical boreholes are used to drain a large block of virgin coal. Essentially, they are gas wells that require continual removal of water from the gas-producing horizon. By lowering the coalbed's water saturation, its permeability to gas increases, thus enhancing methane flow rates.

Because of the inherently low permeability of many coalbeds, vertical boreholes also use hydraulic fracturing or stimulation treatments to create artificial permeability. This is accomplished by pumping a volume of fluid (usually water or foam) under high pressure down the vertical borehole to open up a fracture. A sand proppant is usually pumped along with the fluid to keep the fracture propped open after the injection pressure is released. The penetration of stimulation fluids into the mine roof, causing roof damage, has been a concern of the mining industry. However, 22 case studies of mine-throughs of stimulation treatments in coalbeds showed no evidence of adverse roof conditions that could be attributed to the well treatment (Diamond and Oyler, 1987).

Vertical wells drilled in advance of mining have proved to be an effective technique for removal of methane from coalbeds. Oyler and Stubbs (1985) examined a producing pattern of 23 wells after four years of production and found a 50% reduction in the in-place gas content within the pattern and a 29% reduction 500 ft (150 m) outside the pattern. The commercial use of vertical coalbed wells has been discussed by Dunn (1984).

HORIZONTAL BOREHOLES. Horizontal boreholes have proven to be a very effective technique for removing methane from a coalbed. One technique, horizontal boreholes from shaft bottoms, has also been shown to be an effective method for removal of methane from the coalbed in advance of mining. The biggest drawback to this technique is that it requires the sinking of a ventilation shaft or large-diameter borehole (a large capital expenditure) three to five years in advance of their use. The commercial use of horizontal boreholes has been discussed by Dixon (1987).

DIRECTIONAL SLANT HOLES. A directionally drilled slant hole is a small-diameter borehole that is drilled from the surface and intentionally deflected to intercept the coalbed horizon horizontally. After the coalbed is intercepted, the horizontal drilling is continued. This technique was developed to take advantage of the gas drainage efficiency of horizontal boreholes and the central collection of gas and use of nonpermissible drilling equipment common to vertical boreholes.

Two such holes have been drilled successfully; however, neither hole produced much gas from the coalbed. The problems were difficulty in dewatering the horizontal portion of the borehole and failure to case the borehole far enough into the coalbed.

11.2.1.2 Room and Pillar/Longwall Development

The control of methane during mining on continuous miner sections is accomplished with one or more of the following techniques: ventilation, horizontal boreholes, and water infusion. The common characteristic of these is that they are all implemented from within the mine.

VENTILATION. Ventilation is the universal method of controlling mine gases (Chapters 11.6 and 11.7). In US coal mines, the working place must have a minimum quantity of 3000 cfm (1.42 m^3/s) of air ventilating the face and 9000 cfm (4.25 m^3/s) at the last open crosscut. The purpose of mine ventilation air is to dilute and render harmless the methane emitted from the working face. Therefore, to stay within the statutory limits (1%) established by Mine Safety and Health Administration (MSHA), the mine operator must provide at least 100 cfm (0.047 m^3/s) of air for every 1 cfm (0.00047 m^3/s) of methane. The principles of

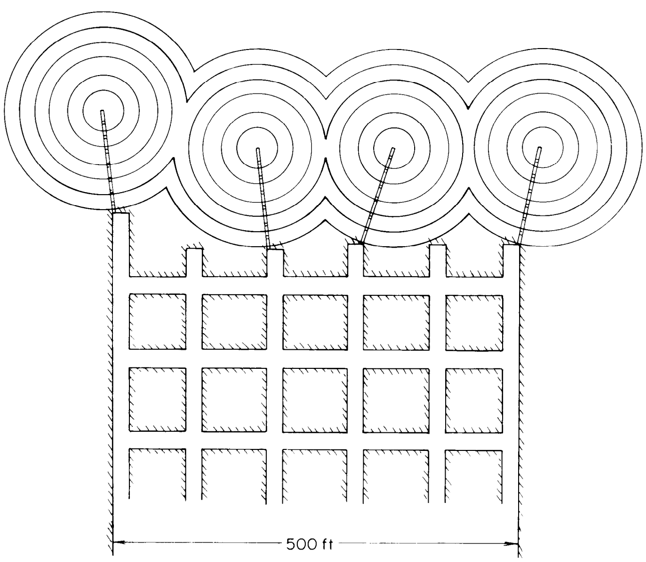


Fig. 11.2.1. Plan view of water infusing a development section.

ventilation are well understood and documented, and computer programs are available for analysis of mine ventilation systems.

HORIZONTAL BOREHOLES FOR METHANE DRAINAGE. The idea of drilling a horizontal borehole into a coalbed to remove methane has been known for at least 45 years; however, much of the technology to effectively drill these holes has only recently been developed (Kravits, Sainato, and Finfinger, 1985). For the mine ventilation engineer, the practical effects of horizontal methane drainage are a reduction in methane emissions during mining and a potential for selling the captured gas at a profit.

The use of horizontal boreholes has consistently reduced methane emissions by at least 50% (Prosser et al., 1981). Also on a longwall development section in an Alabama mine, a 50% increase in productivity was achieved in addition to a 54% decrease in methane emissions (Pate et al., 1983).

WATER INFUSION. Water infusion of coalbeds is a technique that can be used for methane control on working faces but also is beneficial for dust control. Fig. 11.2.1 illustrates a water infusion operation on a room and pillar section. The water that is infused into the coalbed flows through the fracture system displacing the methane that normally resides there. Infusing from the face area forces the methane from the face area and into the returns where the gas is safely diluted and carried from the mine. The result is a reduction in face emissions by as much as 70% (Deul and Kim, 1988).

Water infusion does not capture the methane displaced from the coalbed fracture system (like a drainage system), but just forces it out the ribline away from the working face. The net effect is to reduce methane emissions in the face area and increase methane emissions in the returns. Therefore, water infusion can only be considered as a possible control strategy for face emission problems.

11.2.1.3 Retreat Longwall

Methane problems on retreat longwall sections normally occur in one of five areas: the working place, returns, fresh air entry, bleeder system, or gob. When the immediate roof collapses and falls into the space created by the removal of the coal, fractures propagate up into the overlying strata. If there are methane-bearing rocks above the longwall panel, then methane gas can migrate from the roof and into the collapsed material comprising the gob. Control techniques to be discussed are gob holes and cross-measure boreholes (Baker, Garcia, and Cervik, 1988).

GOB HOLES. An effective method of removing methane from the gob is to drill a borehole from the surface to a point in the strata overlying the longwall panel. Then when mining passes beneath the gob hole, the methane can be drawn to the surface with the use of a surface exhauster installed on the borehole. Typically, three or four gob holes, drilled to within about 150 ft (45 m) of the top of the coalbed, are required on a longwall panel to provide effective control for the total length of the panel.

CROSS-MEASURE BOREHOLES. Although gob holes are normally used to control excess methane in bleeders and gobs, there are times when irregular surface topography, unavailability of surface right-of-ways, or costs prohibit their use. For these reasons, an alternative was developed, cross-measure boreholes.

In the cross-measure borehole system, small-diameter boreholes are drilled from the tailgate and out over the longwall panel. Each borehole is then connected to an underground pipeline that transports the captured gob gas to a vertical borehole for removal from the mine. The gas is captured because a partial vacuum is applied to the exhausting system and as the longwall retreats, the cross-measure boreholes intercept the fracture system over the caved gob and draw the gob gas into the system (Cervik and King, 1983).

11.2.1.4 Special Methane Situations

There are two situations that may arise for the mine ventilation engineer that may be considered special or extraordinary: frictional ignitions and gas outbursts.

FRICTIONAL IGNITIONS. When the bits on continuous miners or longwall shearers move across a hard surface such as sandstone in the mine floor, a streak of hot material will be left as a trail behind the bit. This streak is the source of frictional ignitions. Poor bit maintenance with tungsten carbide inserts lost or broken will aggravate the situation due to the higher incendivity of the steel shank (Courtney, 1981).

The first line of defense against frictional ignitions, as always, is good ventilation. Cecala (1985) has been able to show improved ventilation around a shearer with water sprays. One special area for emphasis that has been identified on longwalls is the headgate cutout area. In this area, several ideas for improving airflow have been identified for reducing frictional ignitions (Cecala et al., 1986). When improved ventilation is not able to solve a frictional ignition problem, then one of two machine-mounted techniques must be used: sprays directed to the back of the bit or modified bit design.

For a longwall shearer, the best machine-mounted solution is water sprays directed to the back of the bit to quench the hot streak. Field and laboratory tests have shown dramatic decreases in the number of frictional ignitions on shearer drums with the use of rear-directed water sprays (Agbede et al., 1982; Cecala et al., 1985). However, due to difficulties in developing a large-diameter water seal, this technique is not yet viable for continuous miners.

Frictional-ignition control technology is now available for longwalls and continuous miners in the form of a modified bit design (Cheng et al., 1983). Research has developed a "mushroom bit" that replaces the conventional plumb bob bit and

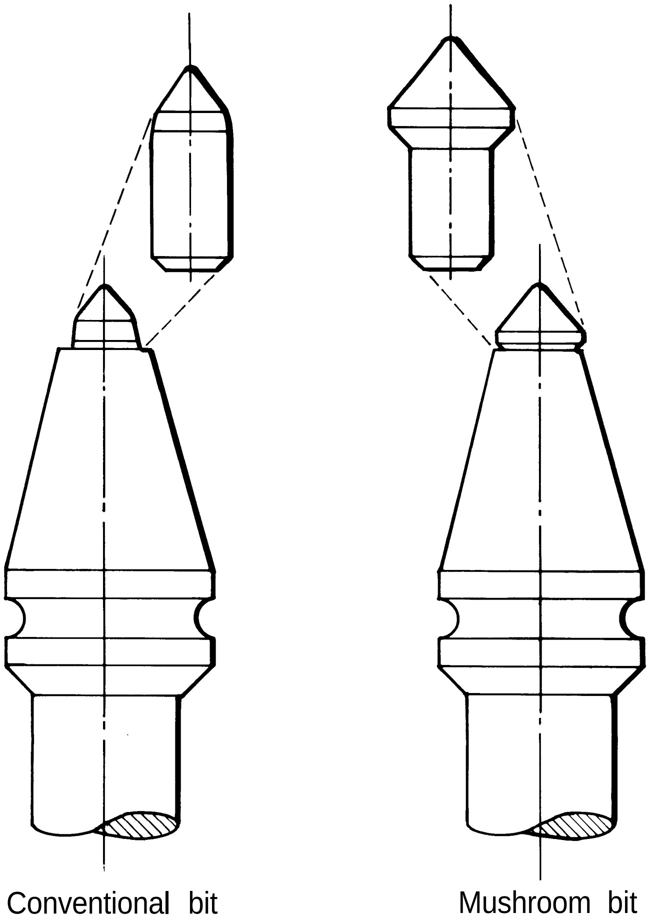


Fig. 11.2.2. Conventional and mushroom-type bits.

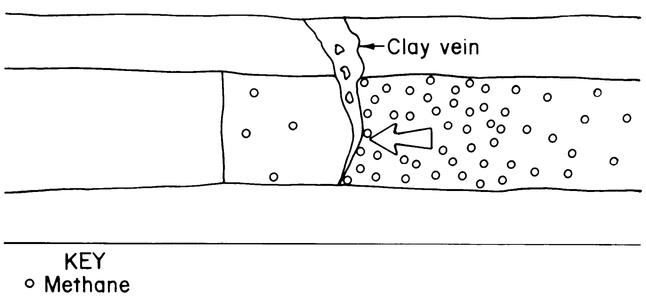


Fig. 11.2.3. Clay vein damming up gas flow.

provides added bit life and a reduction in the probability of frictional ignitions (Fig. 11.2.2).

GAS OUTBURSTS. The uncontrolled release of large volumes of methane into the working face is commonly referred to as a *gas outburst* and can be manifested in US coal mines in two manners. The first has been described by Campoli et al. (1985) as due to the inherent characteristics of the mined coalbed (e.g., gas content greater than 300 ft^3/ton or 9.4 mL/g), depth greater than 1300 ft (400 m), coalbed discontinuity present, etc. To safely mine coal prone to inherent gas outbursting, the mine operator must drain the methane from the coalbed to lower the gas content and reservoir pressure.

The other manifestation of gas outbursts has been described as being due to the formation of gas cells by intercepting clay veins. Fig. 11.2.3 illustrates how clay veins can form these cells from which methane is released into the working face when intercepted by mining. The clay veins are impermeable to the flow of methane and keep the gas confined within the cell at normal coalbed reservoir pressure, 0.44 psi/ft (9.95 kPa/m) of overburden. The method of controlling this problem is to drill a horizontal borehole in advance of mining to intercept the gas cell and reduce the pressure.

11.2.2 DUST CONTROL IN ROOM AND PILLAR COAL MINING

A good general reference to dust control for room and pillar coal mining has been written by Divers et al. (1987). A wealth of information on the health impacts of coal mine dust has been given by Lainhart et al. (1969).

Expert system computer programs that give advice on controlling dust are available from the USBM (Kissell and King, 1988).

11.2.2.1 Ventilation: Face Ventilation Equipment

The quantity of air and the distance from the end of the line brattice or tubing to the face are critical. Studies have shown that dust reductions are greater when the brattice or tubing is moved closer to the face; for this reason, the end of exhaust brattice or tubing should be maintained within 10 ft (3 m) of the face. Also mean entry air velocities above 60 fpm (0.3 m/s) will prevent face-dust rollback.

Brattice efficiency rapidly increases as the area between the tight rib and the brattice increases. The brattice should always be placed at a maximum distance from the rib, while allowing for machinery passage and placement of electrical cables and water hoses.

Because of leakage through and around permanent stoppings, more air must be forced into a mine than would otherwise be required for ventilation. As power costs increase, air leakage adds increasingly to the mine's operating costs. Most mines dry-stack the stopping blocks and trowel on a mortar coating to act as a sealant. Recent studies have shown that leakage can be significantly reduced if a modified mortar, supplemented with steel or fiberglass fibers, is used instead of a conventional troweling. Brushing with a plastic-bristle whitewash brush results in an even sealant coating and requires much less skill and experience than troweling.

As a general rule, especially in older mines, reducing stopping leakage is the most cost-effective technique to get more air to the face. Varying degrees of airtightness can be realized with different stoppings and the quality of construction used to erect the stoppings. Step-by-step instructions for building three types of stoppings, incorporating many of the procedures discussed, are available in a USBM handbook (Timko, 1983).

11.2.2.2 Water Sprays: Sizing Waterlines and Hoses

Consider a supply line for a continuous miner water-spray system that requires 40 gpm (2.5 L/s) at 100 psi (690 kPa). If the distance from the main tapping point is 500 ft (150 m) and a 1.25-in. (32-mm) inside diameter hose is used, the pressure loss is about 75 psi (518 kPa) for the entire length. Allowing an additional 10-psi (69-kPa) loss for fittings and the miner water circuit, a minimum of 185 psi (1280 kPa) should be available at the main tapping point. Selection of a hose only 0.25 in. (6.4 mm) wider (i.e., 1.5 in. or 38 mm) would reduce the pressure loss through the hose by approximately 67% (from 75 to 25 psi, or from 518 to 173 kPa).

WATER FILTRATION. A major problem associated with water-spray systems is the frequent clogging of spray nozzles caused mainly by particulate matter in the water line, including pipe scale, rock, and coal particles. Nozzle blockage can be minimized by using fewer nozzles, but with larger orifice diameters of at least 1/16 in. (1.6 mm).

The USBM developed a simple, nonclogging, water-filtration system to replace conventional spray filters (Divers et al., 1987). The system consists of an in-line Y-strainer to remove the plus 1/8-in. (3.2-mm) material, a hydrocyclone to remove virtually all of the remaining particulates, and a polishing filter to remove traces of particulates that are carried over the hydrocyclone overflow during the startup and shutdown of the spray system.

WATER SPRAY SYSTEMS FOR CONTINUOUS MINERS. Continuous miners are equipped with water systems designed to suppress dust, cool motors and bits, and serve as emergency fire-suppression systems. Spray nozzles mounted on the machine body deliver water to strategic areas for dust control. The purpose of these sprays is to wet the coal as it is cut. Once the liberated dust has been entrained in the ventilating air, the dust capture or "knockdown" afforded by these sprays is moderate. All spray systems should be turned on before cutting and should be left on for a short period after cutting. Tests have shown that the best spray systems can reduce respirable dust in the immediate return by 60%; typical reductions average 30 to 50%.

When mean entry air velocities are too low, an improper spray system can cause face-dust rollback to the machine operator's position. This can usually be determined by temporarily shutting off the sprays. If the rollback is reduced or eliminated, try the easiest of the following: (1) increase face airflow; (2) extend and tighten up the brattice or tubing; (3) if spray pressure is too high, reduce it; (4) if nozzle orientation is wrong, remove all nozzles pointed outby; or (5) install an anti-rollback spray system if mine is not gassy.

There are three types of spray systems recommended for continuous miners. The anti-rollback water spray system (Anon., 1985a) is recommended for nongassy sections with low airflow to the face. Spraying water sometimes can be self-defeating because the turbulence created by the sprays rolls the dust back toward the operator. High spray pressure (over 100 psi or 692 kPa) and the use of wide-angle top and side sprays that overspray the drum or are set too far from the drum promote this condition.

The anti-rollback spray system has alleviated this problem. With the new system, a moderate spray pressure of 100 psi (692 kPa), measured at the nozzle, is a practical maximum. Although higher pressure sprays have the potential to knock down more dust, they can increase the dust blown back to the operator. However, with severe dust problems, especially when cutting rock, water-flow rates should be as high as possible within a 25- to 35-gpm (1.6- to 2.2-L/s) range. As shown in Fig. 11.2.4, the top and side nozzles are arranged for "low" reach (about 12 in. or 300 mm) from the cutting head and no overspray that would increase rollback. Flat spray patterns, as opposed to cone spray patterns, are suitable because the entire flow from the nozzle can be directed to the cutting head. On the boom top, horizontal flat spray patterns near the cutting head cause the least air disturbance; on the sides, vertical flat spray patterns are best.

Efficiency of the anti-rollback system can be greatly improved through the use of underboom sprays. These are located on the rear corners of the shovel at the sides of the machine. They are aimed towards the front of the gathering arms (Fig. 11.2.5). Pressures can be as high as 200 psi (1380 kPa) and flow rates 4 to 5 gpm (0.25 to 0.31 L/s) (Anon., 1989).

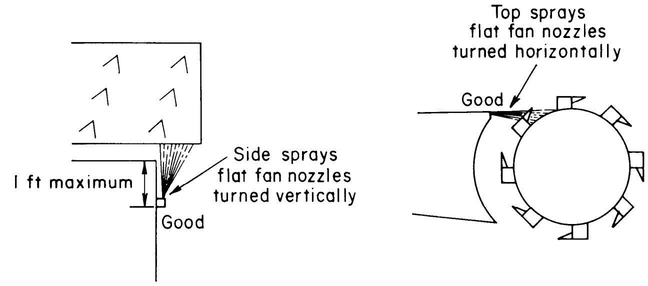


Fig. 11.2.4. Top and side nozzles arranged for low reach. Overspray causes rollback. Conversion factor: 1 ft = 0.3048 m.

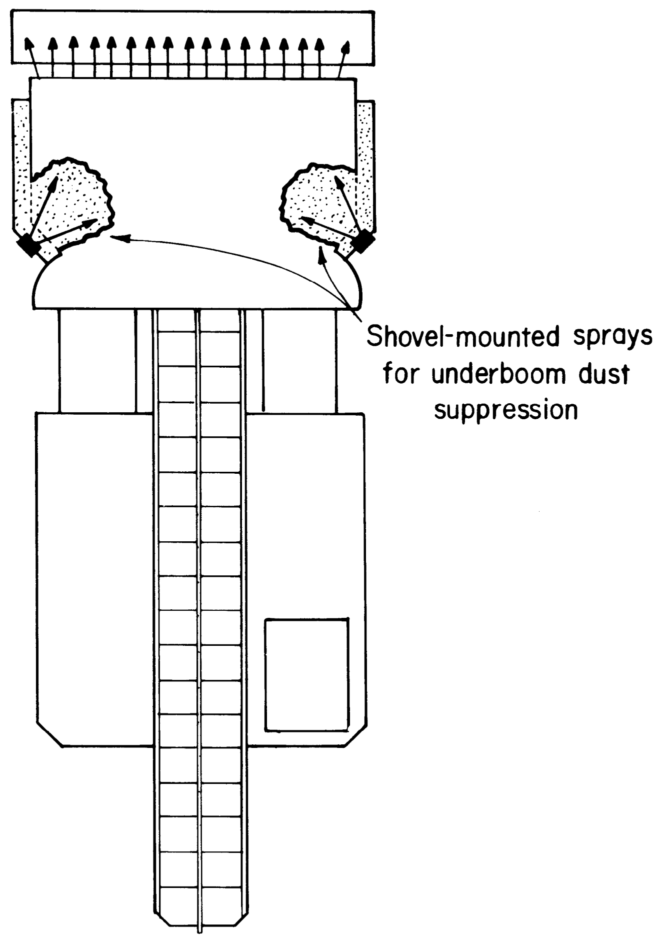


Fig. 11.2.5. Underboom sprays.

An anti-rollback system is particularly suited to faces without a gas problem and where dust standards are more stringent because of quartz. The conventional external water spray system is recommended for sections with good ventilation. The quantity of water needed depends upon the operating conditions of the individual section. Flowrates usually range from 20 to 30 gpm (1.3 to 1.9 L/s). Nozzles are typically located on top of the boom (directed at the top of the drum), beneath the boom (directed at the bottom of the drum and at the gathering arms), and in the conveyor throat. Sprays can also be located on both sides of the boom. Either full- or hollow-cone nozzles are used on the top to provide adequate coverage across the total width of the drum. These types of nozzles should also be used below the boom and should be uniformly spaced to maximize coverage of the area between the sprays and the bottom of the drum. Venturi sprays are suggested for the conveyor throat to prevent dust dispersion to the operator.

The spray fan system (Ruggieri, Babbitt, and Burnett, 1985) is primarily designed to reduce face methane concentrations, without stirring up more dust. It also provides water for bit lubrication and cooling and limited dust control. It is an auxiliary ventilation system consisting of several spray manifolds strategically placed on continuous miners to effectively ventilate the immediate face area. A series of water sprays direct the main ventilation flow to the face and sweep contaminated air and gas across the face toward the return. The spray fan system should only be used with good face ventilation (mean entry air velocities above 60 fpm or 0.3 m/s). It is also effective with curtain setback beyond 10 ft (3 m).

11.2.2.3 Mining Practices: Modified Cutting Cycle

Previous studies have shown that cutting rock can contribute five times the respirable dust compared with cutting coal. Rock cutting should be avoided if possible and should not be prolonged. The usual cutting pattern is to sump into the face at the roof and to shear down to the floor. An alternative is to sump into the face 1 to 2 ft (0.3 to 0.6 m) below the roof and to shear down to the floor. This should be continued for at least two sump-shear sequences (more if roof conditions and seam and/or miner height requirements allow). Then the machine is pulled back to cut the remaining top coal and roof rock. This procedure reduces respirable dust levels by allowing the roof rock to be cut to a free face.

BIT WEAR AND REPLACEMENT. Routine inspection of the cutting drum and replacement of dull, broken, or missing bits improves cutting efficiency and helps to minimize dust.

11.2.2.4 Roof Bolting Operations

The roof-bolter operator's dust exposure frequently comes from upwind dust sources, particularly the continuous miner. The use of a double-split ventilation system to provide the bolter operator with a clean split of air is the most effective way to combat this problem. In single-split sections, the mining-bolting cycle must be carefully designed to keep the roof bolter upwind of the continuous miner whenever possible. The roof-bolting machine can also contribute significantly to the respirable-dust problem, especially when the roof rock contains a high quartz content.

DUST COLLECTOR AND MAINTENANCE. Dust produced by roof-bolting operations can be easily detected and remedied with a few simple maintenance procedures. Dust in the collection system's blower exhaust is the most common problem encountered, a sign that dust is by passing the filters. Common causes of this are damaged or improperly seated filters. Cloth-bag-type filters are less efficient dust collectors than the pleated-paper cartridge-type and allow dust to bleed through the system and escape through the exhaust. Accumulations of dust between the filters and the blower (clean side) are a result of filter leaks. This dust can be removed by back-flushing the system with compressed air or by running the blower for several minutes. In either case, the filters should be removed and the access door opened. Finally, a visible dust plume from the collar of the drillhole is a sign of inadequate airflow to the chuck or bit and is often a result of air leaks within the system. These occur primarily through loose connections (especially at the chuck hose), air-pressure relief valves, poorly fitting dust-collector access doors, and worn and damaged hoses. Replacement of overloaded and clogged filters will generally increase airflow at the bit by more than 30%.

ROOF DRILLBITS. Studies have shown that a roof bolter's dust exposure can be substantially reduced by changing from a shank-type bit to a "dust hog" bit (air inlet port located on the bit instead of on the drill steel). Generally, this difference can be attributed to the initial few inches (millimeters) of bit penetration where the shank-type bit allows far more dust to become airborne. Underground evaluations of the two bit types showed that the dust hog bit reduced bolter dust exposure by over 80%, and it also drilled faster and cooler.

11.2.2.5 Auger Mining Operations

Face ventilation techniques applied on auger sections are generally limited to the push-pull type. Push-pull ventilation is more readily applied on auger mining sections because the wide entries permit construction of brattice lines on both sides of the entry without restricting machinery. The push-pull system has the advantage of delivering large quantities of ventilating air to the face while confining the dust cloud to the inby portion of the cut. When using the push-pull method, the return brattice line should always be maintained inby the intake line. Maintaining the exhaust brattice close to the face is particularly important.

Laboratory-model studies indicate that dust exposures for the jacksetters and machine operator may be reduced by as much as 95% if the exhaust-brattice setback distance is decreased from 20 to 10 ft (6 to 3 m). Air quantity should be maintained at about 4000 cfm (1.9 m^3/s) in the face return to reduce face dust concentrations. The distance of the return line curtain from the rib should be at least 5 ft (1.5 m) to obtain a large cross-sectional area for the face air. The blowing side can be closer to help ensure airflow towards the face. The airflow direction between the two curtains must be toward the face to keep the face workers in compliance. This is done by maintaining the return brattice tight against the roof and floor and keeping it at the maximum possible distance from the rib.

Wet augers that supply water to nozzles located along the auger shafts, in conjunction with external sprays mounted on the front of the machine frame, help to control dust in auger sections. If coal is not sufficiently wetted at the face, a significant amount of dust can be generated during transport by the machine conveyor and mobile bridge conveyors. An additional advantage of the wet auger system is the increased amount of water applied to the cutting and loading area, which provides more coal wetting and consequently lowers dust levels during transport.

11.2.2.6 Outby Dust Control: Conveyor Belts

The first step in minimizing the amount of dust generated during coal conveyance is to ensure that the coal is wetted adequately at the face. Rewetting the coal at intervals along the belt may also be necessary. This is best accomplished by uniformly wetting the coal stream with flat-fan sprays operating at 40 to 50 psi (280 to 350 kPa). The accumulation of coal and dust particles on the top and bottom sides of the returning belt can be controlled by mounting belt scrapers or wipers near the drive, or by spraying the belt with a low-quantity water nozzle. Maintaining the conveyor system (alignment, rollers, splices, etc.) in good working condition will also help reduce dust.

TRANSFER POINTS. Transfer points are the greatest source of dust in outby areas; these include shuttle-car-to-belt, belt-to-belt, and belt-to-mine-car transfer points. A major cause of dust at transfer points can be the dislodgement of dust adhering to the underside of the belt. Nozzles projecting a flat, fan-type spray directly at the underside of the belt have reduced dust levels in this area by as much as 60%; water quantities of 0.3 to 0.5 gpm (0.019 to 0.032 L/s) are typical for this application. It is also important that the coal be wet prior to reaching the transfer point. If rewetting of the coal is required, it should occur at a point far enough upstream of the transfer point to allow time for the coal and water to mix. Hoods and chutes may also be used to prevent the ventilating air from agitating dust and to reduce the amount of coal fragmentation and breakage associated with excessive free-fall distances.

HAUL ROADS. Good housekeeping (i.e., scooping up excess coal when a cut is finished, shoveling coal along the ribs to the middle of the entry where the machine can reach it, and minimizing shuttle car spillage) is a basic approach to this problem that can be very effective. Another method is to wet the roadway with water; however, this is usually a temporary measure because the water can rapidly evaporate, depending on air velocity and mine humidity. In order to keep the moisture content of the roadway dust at a desired 10% level, the use of a hygroscopic salt, like calcium chloride, is frequently required. It should be spread in two applications: three-quarters applied one hour after wetting the dust with water, and the remaining one-quarter applied about one week later. Retreatment of the roadway should not be necessary for about six months, but spraying the roadway with water after three months is recommended.

11.2.2.7 Techniques Requiring Machine Modification

Machine-mounted dust collectors consist of a fan to direct the dust-laden air into the scrubber, a wet dust-removal system to separate the dust particles from the airstream, and a de-mister unit to remove the water from the airstream. Scrubbers are primarily used with blowing face ventilation. Where seam height allows, the flooded-bed-type scrubber is becoming increasingly popular for use on continuous miners in room and pillar operations. Scrubbers are the only device that can remove most of the respirable dust from the airstream.

Remote control units allow the continuous miner operator to move farther back from the face area than the conventional cab location, thus reducing exposure when rollback of the dust cloud occurs. A growing application of these devices is in sections employing blowing ventilation, which utilize a scrubber-equipped miner to take deeper cuts (> 25 ft or 7.6 m) with extended face-brattice setback distances (> 25 ft or 7.6 m). Such cut depths and setback distances must first be approved by MSHA, and a reasonably good roof is essential.

Dust levels can be reduced through selection of proper cutting parameters, such as depth of cut, cutting speed, and bit spacing. Some options currently available for controlling primary-dust generation include: (1) bit spacing equal to two to three times two-thirds of the cut depth, (2) changing head gearing to reduce drum rpm, (3) increasing the advance (sump) rate, (4) using sharp bits, and (5) using a minimum number of gage cutters on the end ring.

Low-pressure wet drilling for roof bolting operations utilizes hollow drill steels to deliver water (5 to 6 gpm or 0.32 to 0.38 L/s per chuck) to the bits and offers the advantages of improved bit life, faster drilling, and excellent dust control. However, wet drilling can create problems in sections that cannot tolerate additional water accumulation on the mine floor.

11.2.3 DUST CONTROL IN LONGWALL COAL MINING

Longwall dust control techniques can be divided into the following:

1. Basic dust control techniques.
2. Additional necessary controls.
3. Optional supplementary controls.

A good source of information is the USBM *Dust Control Handbook for Longwall Mining Operations* (Shirey, Colinet, and Kost, 1985).

11.2.3.1 Basic Dust Control Techniques

There are three basic dust control techniques that should be implemented on all longwall mining operations:

1. A passive barrier or external water-spray system design that confines shearer-generated dust near the face and away from the operators.

2. Large quantities of water through the sprays, particularly the drum sprays, to aid in dust knockdown and prevent dust from becoming airborne.

3. A cutting sequence that allows the shearer operators to work primarily on the intake-air side of the lead cutting drum.

SHEARER CLEARER. There are some poorly designed shearer-mounted spray systems with nozzles directed upwind at the cutting and loading zone of the intake-side drum. These actually carry dust away from the face and upstream of the drum. Here it mixes with the clean intake air and is carried out into the walkway over the shearer operators.

The USBM has developed a novel shearer spray system, called the *shearer clearer*. It takes advantage of the air-moving capabilities of water sprays. The system consists of several shearer-mounted water sprays, oriented downwind, and one or more passive barriers, that split the airflow around the shearer into clean and contaminated air (Fig. 11.2.6). The air split is initiated by a splitter arm; conveyor belting hangs from the splitter arm down to the panline. This extends from the top gob side corner of the shearer body to 18 in. (460 mm) beyond the cutting edge of the upwind drum.

Spray manifolds mounted on the splitter arm confine the dust cloud generated by the cutting drum, further enhancing the air split. The dust-laden air is drawn over the shearer body and held against the face by two spray manifolds positioned between the drums, on the face side of the machine.

The air is then redirected around the downwind drum by a set of sprays located on the downwind splitter arm. Operating pressure must be about 150 psi (1034 kPa), measured at the nozzle, to assure effective air movement. Total water flow rate with all sprays operating is about 12 gpm (0.76 L/s).

In underground tests, the shearer clearer reduced operator exposure from shearer-generated dust by at least 50% when cutting against the ventilation, and 30% when cutting with the ventilation (Shirey, Colinet, and Kost, 1985).

COOLING WATER DISCHARGE. Shearer cooling water is often discharged through spray nozzles oriented against the primary airflow or directed into the face. This causes dust to be carried back into the walkway. An alternative for discharging cooling water is panline spray manifolds, mounted on both ends of the shearer, aimed down onto the panline. This minimizes the turbulence caused by face side sprays, and respirable dust reductions of up to 35% at the shearer operator's location can result.

DRUM SPRAY-WATER SUPPLY SYSTEMS. Dust generated by the shearer can also be reduced by increasing the quantity of

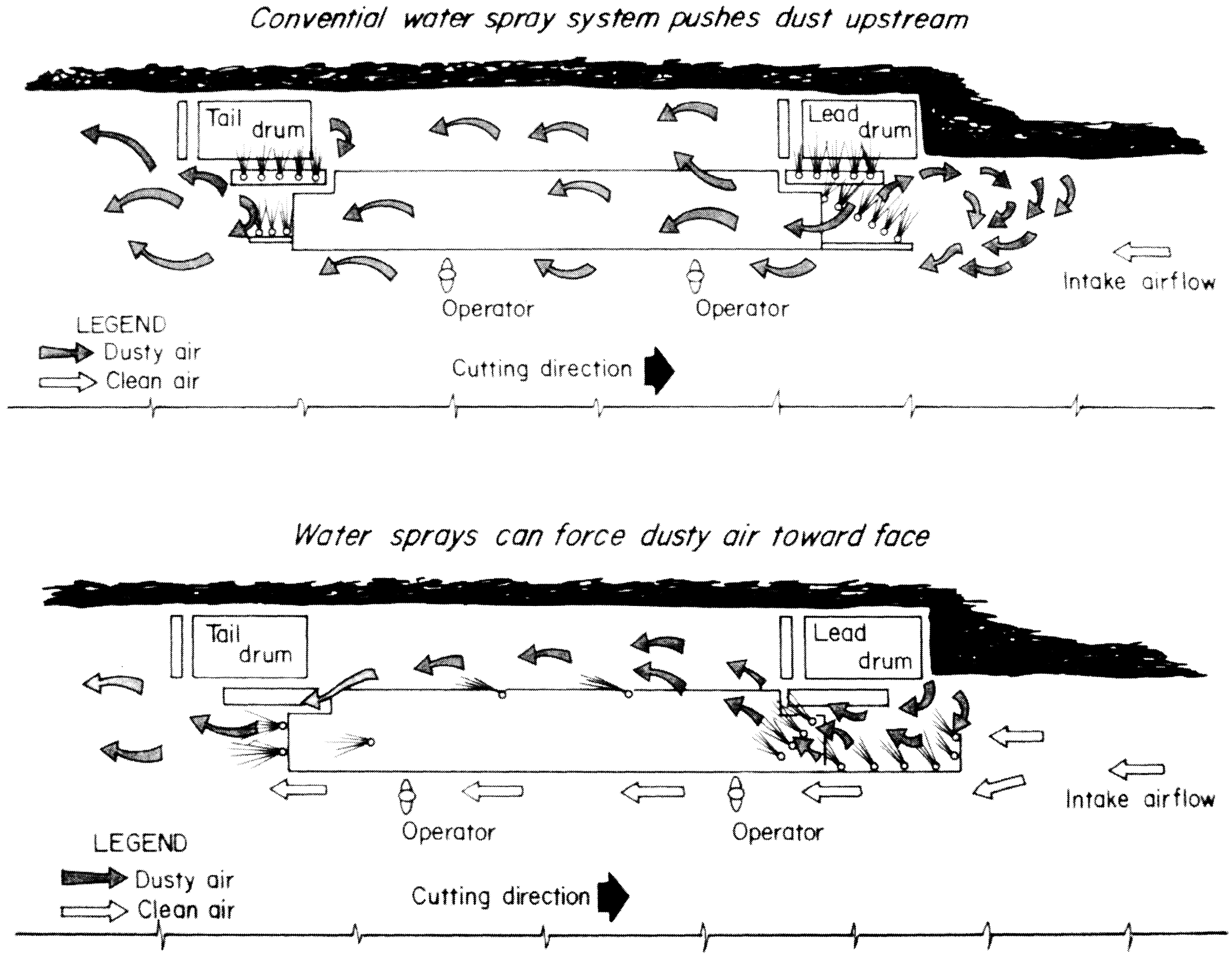


Fig. 11.2.6. Air currents with conventional sprays (top) and shearer-clearer system (bottom).

water supplied to the shearer. In two separate studies (Kost, Colinet, and Shirey, 1985; Ruggieri and Babbitt, 1986), water flow to the shearer was increased about 40 to 62 gpm and 45 to 65 gpm (2.5 to 3.9 and 2.8 to 4.1 L/s), respectively. In both cases, dust levels at the shearer were reduced 40%.

Dust exposure of the shearer operators may depend on the operating pressure of the water supplied to drum sprays. In two separate studies (Pimental, Adam, and Jankowski, 1984; Kok and Adam, 1986), water pressure to the drum sprays was increased about 50%, 75 to 150 psi and 80 to 150 psi, (517 to 1034 kPa and 552 to 1034 kPa), respectively. In both instances, dust exposure of the shearer operators increased 25%. Thus the maximum drum spray pressure appears to be from 70 to 100 psi (483 to 690 kPa). Water flow rate should be increased by increasing the nozzle orifice size, rather than the operating spray pressure.

All US longwall shearers use some type of internal water spray system. The pick-point flushing system uses nozzles mounted in the bit blocks or special blocks immediately in front of the bits. Cavity-filling systems use sprays installed in a pipe manifold mounted on the drum hub between the scrolls, or along the edge of the vanes. The water-through-the-bit concept is a new development. A channel through the bit directs water behind where the bit strikes the coal.

These three commonly used drum water-spray systems were compared on two longwall mining sections with very diverse geologic and mining conditions (Kost, Colinet, and Shirey, 1985). The pick-point flushing system with solid-stream (jet) nozzles was the most effective in suppressing respirable dust near the shearer operator's position. The pick-point system with cone-type sprays was only 70% as effective. The through-the-bit system with flat-fan sprays was only 60% as effective. The cavity-filling system was only 47% as effective as the pick-point-jet-spray system at reducing operators dust exposure. Down-wind concentrations were essentially the same for all systems.

Improved Cutting Sequence Designs. A primary means of controlling dust exposures on double-drum shearer longwall faces is to modify the cutting sequence to move workers upwind of dust sources. Currently, 75% of operating sections use a sequence where the shearer cuts coal in only one direction. In a bidirectional sequence, full cuts are taken in both directions. Typically, the lead drum takes a full cut in the raised position during the main cutting pass. The trailing drum cuts bottom

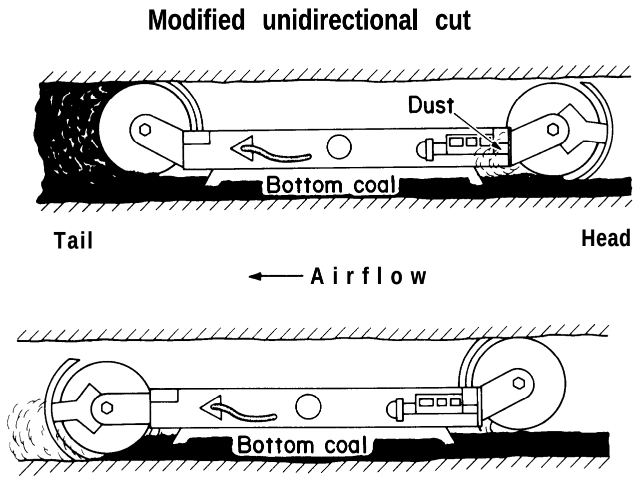


Fig. 11.2.7. Modified cutting sequence. Both directions shown.

coal. A minimal amount of coal is cut and loaded during the return cleanup pass.

On faces where the main cutting pass is taken upwind (typically from tail-to-head), both operators must remain at their controls. They are, therefore, positioned on the return-air side of the lead drum. An alternative method to reduce their dust exposure is to take the primary cut downwind (typically from head-to-tail), with the operators positioned upwind of the lead cutting drum.

Comparison of the instantaneous dust profiles for head-to-tail and tail-to-head cutting shows the large dust concentration gradient that can exist between the intake and return-air side of the lead cutting drum. Dust surveys conducted by the Bureau of Mines on 14 longwall faces (Anon., 1984) showed that the average shearer operator's exposure when cutting tail-to-head was 44% greater than when cutting head-to-tail.

One disadvantage of head-to-tail cutting is possible blockage beneath the machine underframe as material cut by the lead (tailgate) drum is loaded onto the face conveyor.

During a typical head-to-tail cut, the lead drum takes a full cut while the trailing drum cuts the remaining bottom coal. However, the dust generated by the trailing drum can spread into the walkway and increase the dust exposure of both operators. With a modified cutting sequence, the lead drum continues to take a full cut during the head-to-tail pass while the trailing drum is "free-wheeling" or cutting a minimal amount of coal (Fig. 11.2.7). During the tail-to-head cleanup pass, the trailing (return-side) drum cuts the remaining bottom coal. This enables both operators to remain on the intake-air side of the primary dust generating source, except when cutting out at the headgate.

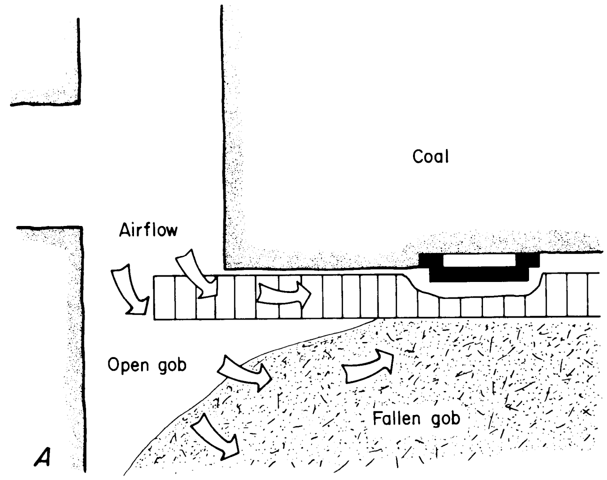
11.2.3.2 Additional Necessary Controls

The above control methods have proved to be effective on all longwall faces. Other techniques, however, are often required to bring a section into compliance with the federal dust standard.

VENTILATION. The primary function of any mine ventilation system is to dilute liberated methane to safe concentrations. However, it is also one of the principal methods used to control dust on longwalls. Face air velocities of 400 to 450 fpm (2.0 to 2.3 m/s) appear to be the most appropriate for longwall dust control.

Adequate longwall panel ventilation involves more than supplying the required volume of air to the headgate entry. Maintaining that airflow along the entire face is just as critical. Air leakage is greatest in the headgate area because there is often a large gap between the first shield and adjacent rib. Also the gob behind the first few shields remains open because the headgate entry is supported with roof bolts. This air loss prevents maximum use of the air available to ventilate the face. In addition, dust generated during gob falls may be entrained by this airflow and carried back into the face area. A gob curtain, installed from the roof to the floor between the first support and adjacent rib in the headgate entry, forces the ventilation airflow to make a 90º turn and stay on the face side of the supports (Fig. 11.2.8). During underground trials, the average face air velocity with the curtain installed was 35% greater than that without the curtain. The most significant improvement was seen for the first 25 to 30 supports.

However, misapplication of the primary ventilation airflow can increase dust exposure. Shearer operators are often exposed to very high concentrations as the headgate drum cuts into the headgate entry. The high-velocity primary airstream passing over and through the drum entrains and carries large quantities of dust out into the walkway and over both operators. An effective solution is to install a wing curtain between the panel-side rib and the stage loader (Fig. 11.2.9). This curtain shields the



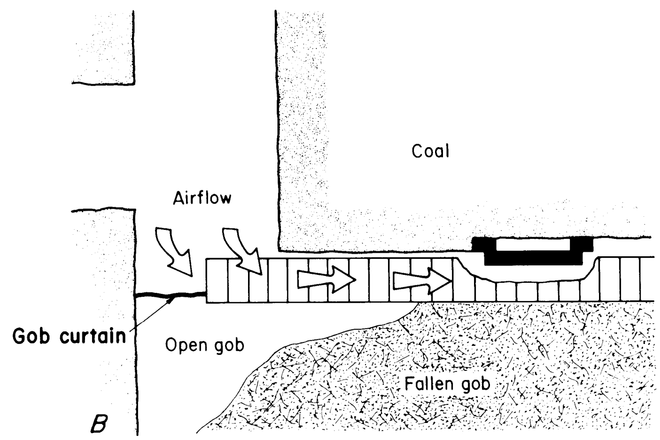


Fig. 11.2.8. Airflow at longwall headgate with a gob curtain (B) and without a gob curtain (A).

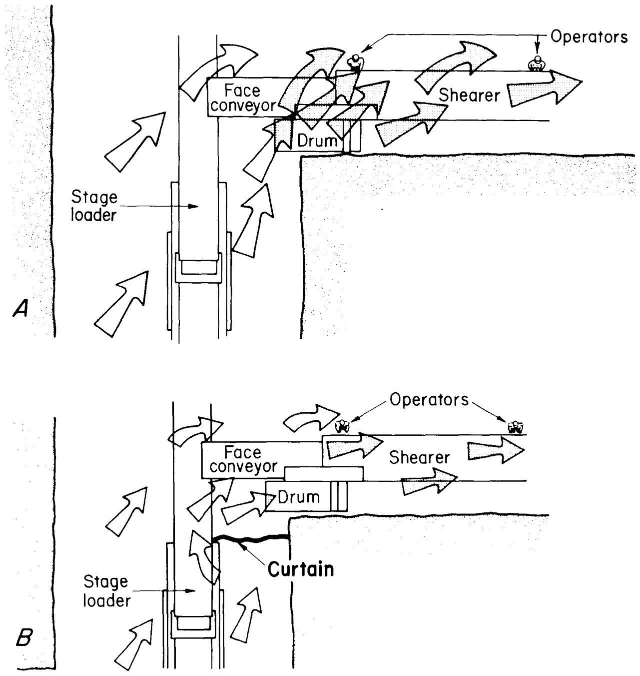


Fig. 11.2.9. Airflow at longwall headgate with (B) and without (A) a wing curtain.

headgate drum from the airstream as it cuts out into the headgate entry. It is typically located 4 to 6 ft (1.2 to 1.8 m) back from the corner of the face to provide maximum shielding without interfering with the drum. The curtain is only in place during the cutout operation, and is generally advanced every other pass. Dust concentrations monitored at the shearer operator positions indicate that the curtain can reduce their dust exposure by 50 to 60% during headgate cutout.

DEEP CUTTING. Reducing drum speed is one of only a few changes a longwall operator can make to increase output, reduce respirable dust, and decrease machine power consumption (Ludlow and Jankowski, 1984). Deep cutting is a function of drum speed and machine advance rate.

Pick spacing must be increased and gage length must be adjusted to take full advantage of deep cutting. The rotational speed of the drum is reduced (typically to 30 to 40 rpm). The depth of cut is increased by using large bits with wider spacing of the bit lines while maintaining the same advance rate used at higher rpm.

Field tests have confirmed the benefits of slow-speed deep cutting (Ludlow and Wilson, 1982). For these tests, the depth of cut was altered from a minimum of 1.7 in. (43 mm) to a maximum of 5.3 in. (135 mm) by varying drum speed and pick spacing while keeping the advance rate of the shearer constant. A 60% reduction in dust generation was achieved by reducing the drum speed from 70 to 35 rpm. This effectively increased bit penetration from 1.7 to 3.4 in. (43 to 86 mm).

11.2.3.3 Optional Supplementary Controls

There are several control techniques that are site specific, and therefore, cannot be successfully applied at every longwall installation.

HEADGATE STAGE LOADER AND CRUSHER. Dust generated at the stage loader is often overlooked, but warrants greater attention. This dust remains airborne across the entire face. It can have a significant impact on the full-shift dust exposure of all face personnel. The major source of dust in the headgate entry is the stage loader/crusher. A basic approach is to mount water sprays along the stage loader. Several sprays are mounted in spray bars, which usually span the width of the conveyor to ensure uniform spray coverage of the coal stream. Recommended spray bar locations inside the crusher include the area just inby and at the point where the stage loader dumps on the belt. The dust capture efficiency of these sprays may be enhanced by enclosing the stage loader, either with steel plates or strips of conveyor belting. The enclosure also isolates the conveyed material from the airstream, thus reducing dust entrainment. During underground trials with a fully covered stage loader and additional water-spray manifolds, improvements at the headgate operator and at support 20 locations were 80 and 45% respectively (Anon., 1985b).

WATER PROPORTIONING. Increasing the water flow to the drums usually reduces airborne dust levels produced by the shearer. Some operations, though, cannot tolerate an increase in water flow because of clay floors. Excess water in the coal reduces the run-of-mine Btu value. It can also cause problems in the coal transport system and in the coal preparation plant. For these operations, supplying larger quantities of water only to the upwind drum can have a significant impact on the amount of water used while still reducing the operator dust exposure.

A study was conducted to assess the effect of proportioning the water flows to the drum (Kok and Adam, 1986). Exposure of the downwind shearer operator and the total amount of respirable dust generated (as indicated by a tailgate measurement) were measured with the shearer cutting downwind. When water was shifted from the downwind drum to the upwind drum, both the operator dust exposure and total dust generated were reduced. The switch from a low flow on the upwind drum and a high flow on the downwind drum to the reverse situation resulted in an average 63% reduction in the operator dust exposure and a 32% reduction in the tailgate concentration.

SUPPORT MOVEMENT PRACTICES. Cutting and support movement practices are designed to minimize the time that jacksetters work downwind from the shearer and to maintain shearer operator dust exposure as low as possible. One such practice involves support advance on the downwind side of the shearer, keeping the shearer operators in uncontaminated air. When the pass cycle against the airflow is used as a clean-up pass, the dust levels generated by the shearer are typically low. The dust exposure of the jacksetters downwind of the shearer remains relatively low as long as no significant amount of rock is cut during the cleanup pass.

When the primary cut is taken against the airflow, a properly designed external shearer water-spray system can adequately confine shearer-generated dust against the face for about 40 ft (12 m) downwind from the shearer. This provides an acceptable environment for jacksetters working immediately downwind.

It is sometimes not feasible to move the supports on the downwind side of the shearer during the cleanup pass. Roof conditions require support advance immediately after the primary cut. In this case, supports are moved on the intake-air side of the shearer as it cuts head-to-tail (downwind). Shearer operators are then exposed to any dust generated by support movement. Under these circumstances, some mine operators find that support-generated dust can be effectively diluted before it reaches the shearer operators by increasing the distance between support advance and the shearer from 20 to 50 ft (6 to 15 m).

Water application on the immediate roof may also help to suppress some of the support dust generated during lowering, advancing, and resetting the roof supports. The immediate roof can be wetted adequately by

1. Maintaining enough water pressure at the drum sprays to wet the roof while the face is being cut.
2. Free wheeling the lead drum near the roof during the clean-up pass so that the water sprays wet the roof.
3. Spraying the roof with one or more water sprays mounted on top of the shearer body, directing water downwind, at an upward 45º angle.

Roof support automation will greatly help to reduce dust exposure, depending on the degree to which jacksetters are moved upwind of dust sources.

SHEARER REMOTE CONTROL. Longwall automation, specifically the remote control of the shearer, can significantly reduce dust exposure of the machine operators. Operators control the machine from positions along the face less contaminated than their normal work positions. A survey shows that exposure was reduced 68% by moving the operator (20 ft or 6 m) upwind of the normal work position (Anon., 1984).

Most manufacturers now offer either radio or umbilical (hard-wire) remote control (see Chapter 22.2). The greatest potential for remote control may be in high coal, where face spalls can seriously injure the operators. Umbilical remote controls are easy to retrofit to modern shearers. They are more common in the United States than radio controls.

11.2.4 PERSONAL PROTECTIVE EQUIPMENT

The Federal Coal Mine Health and Safety Act of 1969 mandated that approved respiratory equipment be made available to personnel when exposed to respirable dust concentrations greater than 2.0 mg/m^3. However, this equipment cannot be used in lieu of achieving the 2.0 mg/m^3 standard.

11.2.4.1 Replaceable Filter Respirators

The replaceable filter respirator consists of a filter-holding unit, typically fabricated from plastic, metal, or hard rubber, that also contains intake and exhaust valves. The filtering efficiency of approved filters is 98% or more. Soft rubber or cloth is used to form a face piece around the filter-holding unit that forms a seal against the wearer's face in an attempt to prevent dust-laden air from bypassing the filter. With a reasonably leak-tight face piece fit, the respirator should remove approximately 95% of the respirable dust.

During one respirator evaluation program (Cole, 1984), two models of face-mask respirators were tested on four longwall sections to determine the protection afforded to the wearer when worn in the tailgate area for an entire shift. The dust exposure of the tailgate worker was 80 to 92% less when wearing either one of the respirators.

Although the respirator does an excellent job of dust removal when properly fitted, some personal discomforts may arise, including increased breathing resistance, aggravated by dust loading on the filter, facial irritation caused by the face seal, interference with normal voice communication, and interference with eye glasses or goggles.

SINGLE-USE RESPIRATORS. The single-use respirator is a much lighter and simpler design. The entire mask is fabricated from filter material and covers the mouth and nose, similar to a surgical mask. The single-use respirator offers the following advantages when compared to the replaceable-filter respirator:

1. Lighter and thus more comfortable to the wearer.
2. Provides greater filtering area and, consequently, has lower breathing resistance.
3. Filter material contacts face and generally is less irritating than rubber or cloth.
4. Requires no maintenance.

However, single-use respirators usually do not form as tight a seal against the wearer's face as replaceable-filter types, thus allowing more leakage, Also single-use respirators provide no protection for the filter material, and, as a result, are not as durable as the replaceable filter type.

AIR HELMET. The air helmet is a redesigned hard hat equipped with a battery-powered fan, filtering system, and face visor, thus providing protection for the head, lungs, and eyes in one unit. Although the air helmet is slightly larger and heavier than conventional hard hats, weighing approximately 3 lb (1.4 kg), wearer acceptance has been quite favorable.

A small fan is mounted in the rear of the helmet to draw dust-laden air through a filtering system; the resulting cleaned air is directed behind a full-face visor and over the wearer's face. Seals are provided along both sides so that exhaled air and excess clean air are allowed to exit the helmet at the bottom of the visor. Also these face seals and additional seals inside the helmet limit contamination from unfiltered air. The fan is externally powered by a rechargeable battery smaller than a conventional cap-lamp battery to be worn on the miner's belt.

Studies were conducted to test the effect of air velocity and helmet position in the airflow on the effectiveness of the air helmet (Cecala et al., 1981). They showed that air velocity and position can have a major impact on the effectiveness of the helmet.

Underground tests on four longwall double-drum shearer faces were also conducted. For three of the four mines, the air velocity in the face walkway was less than 400 fpm (2.0 m/s), and the air helmets reduced respirable dust in the wearers' (shearer operator, jacksetter, and two researchers) breathing zones by an average of 84%. At the fourth mine, the air velocity was approximately 1200 fpm (6.0 m/s), and as a result, the air helmet was not as effective, with an average reduction of 49%. The underground sampling included periods when the face visor was lowered and periods when it was raised, according to normal underground use; this would tend to minimize differences between inside and outside samples, thus reducing the apparent effectiveness of the helmet.

11.2.5 UNDERGROUND METAL MINE DUST CONTROL

The exposure of workers to respirable dust in hard-rock mines, including both metal and nonmetal mines, may be reduced by a systematic approach that includes all or some of the following:

1. Prevention of formation at its source.
2. Preventing dispersal of the dust cloud.
3. Providing dilution ventilation.
4. Avoiding the dust.

11.2.5.1 Preventing Formation

Every effort should be made to prevent both the formation of dust at its source and its liberation into the atmosphere.

WATER. To improve the effectiveness of water when wetting down, a spray nozzle ensures an adequate spread of water over a greater area and prevents the settled dust from being stirred up. A moisture content of the rock of only 1% produces a very significant reduction in dust production when compared with dry rock. As it is difficult to maintain a moisture content of

1% under conditions encountered underground, the optimum moisture content should be maintained at about 5%.

The water used for dust suppression, particularly in drilling and in water-blasts, should be as clean as possible as the evaporation of dirty water can release considerable quantities of dust.

DRILLING. Adequate water suppresses drilling dust. Provide enough to keep the rock surface wet all the time, so that the rock is actually broken under a film of water.

This does not, however, prevent dust from entering the air during the initial collaring period. Various means have been tried to prevent the escape of dust during collaring, ranging from simple hand-held sprays to elaborate types of suction traps around the end of the drill steel, but no single method has been found to be very efficient.

If some of the compressed air operating the drill leaks into the front head of the drill and escapes down the drill steel, it will cause dry drilling and carry dust out of the hole. Also some of this compressed air will escape through the front head release ports and atomize some of the water in the front head. This atomized water, which forms a fog at the front head release ports of many rock drills, evaporates rapidly; and if the water is dirty, as it often is, many dust particles will remain in the air.

For dry drilling methods, the generated dust is collected. The air and dust drawn off must be exhausted into unoccupied return airways or filtered.

BLASTING. Blasting is unlike other mining operations, since it produces not only dust but other contaminants, such as carbon monoxide and nitrogen dioxide fumes. Since little can be done to prevent the production of dust during a blast, the emphasis should be on means to control the resultant contaminants.

The first step in controlling dust produced by blasting is to ensure that the area surrounding the blast (walls, floor, and back) is thoroughly wetted beforehand. This precaution will prevent dust settled out during previous operations from becoming airborne. Furthermore, some of the dust created by the blast will adhere to the wet surfaces in the area, thereby reducing the concentration in the airstream. The second step is to use a two-phase fog spray nozzle that employs both water and compressed air. These nozzles are effective in reducing dust and the amount of nitrous fumes (because of their solubility in water).

Proper ventilation is critical since water alone is inadequate. Blasting dust and fumes should be diluted and exhausted to surface via an untraveled route, preferably an upcast raise designed for that purpose. If this is not feasible, the blasting schedule should be arranged so that the contaminated air will pass through working places when the miners are absent.

11.2.5.2 Preventing Dispersal

The dispersal of respirable dust from crushers, conveyors, and similar equipment can be eliminated in most instances by confining the dust-producing operation within an enclosure and controlling the air contained therein. The air from within the enclosure can be exhausted directly to the upcast airway or, if this is not feasible, it can be filtered. Segment 11.2.8 on minerals processing dust control gives guidelines.

ORE AND WASTE PASSES. Ore and waste passes produce large quantities of airborne dust. The broken rock delivered to the passes contains a considerable amount of inherent dust as a result of preceding operations, such as blasting and loading. Furthermore, the autogenous grinding action of the rock as it is dumped and falls down the pass produces more dust. The first line of defense is to ensure that the rock is thoroughly wetted before delivery to the dump. More wetting can be obtained at the dump site by installing a mist-type atomizer to spray the rock as it falls into the pass. Excessive use of water at the orepass can be objectionable for many reasons: (1) adverse impact on crushing and milling, (2) a large quantity of water may accumulate on top of the material in the chute, creating a hazard for workers on the lower levels, and (3) plugging of clay minerals.

The second step in providing a good environment at ore and waste passes is to prevent the escape and dispersal of dust into working areas by confining it within the passes. This can be accomplished by a system of stoppings and airtight doors over the dumps or "tipping" points. The maintenance of these doors is of prime importance, since their main purpose is to prevent the escape of dust-laden air.

A third step, difficult to accomplish in practice, is providing means to keep the confined ore or waste pass under negative pressure to ensure that all leakage paths are indraft, and to capture the air displaced when rock enters the raise. A suitable fan exhausts from a convenient point in the raise. The contaminated air is filtered or sent via a direct untraveled route to the return air raise.

11.2.5.3 Providing Dilution Ventilation

Ventilation is the best method of controlling contaminants at underground operations. This is carried out in producing areas, such as stopes, scraper drifts, etc. by directing an air split from the main ventilating stream through the workings. The design criterion is 30 to 50 fpm (0.15 to 0.25 m/s), depending upon the type of operation and other local conditions. In some instances, volume may have to be increased greatly—for example, high-speed drives or scraper drifts, where the severe dust-producing operations may require as much as 150 fpm (0.75 m/s).

In headings and raises, the design volume also is based on providing an air velocity of 30 to 50 fpm (0.15 to 0.25 m/s). The type of auxiliary ventilating system employed will be governed by the local conditions at the mine. In most cases, the push-pull (or overlap system) will provide the most satisfactory environment although it is difficult to control in practice. This system consists of a main ventilating duct which exhausts dusty air, and a small blowing line kept to within 20 to 30 ft (6 to 9 m) of the face. The length of the overlap of the exhaust and blowing lines depends upon the size of the drift and, in any case, should be not less than 30 ft (9 m). The blowing volume should not exceed approximately 60% of the exhaust volume in order to avoid recirculation.

The overlap system should not be adopted for those working areas subject to a high virgin rock temperature. Under these conditions, the cool fresh air is heated and becomes saturated with moisture during the time it takes to reach the working face. To avoid this, the forcing or blowing system of auxiliary ventilation should be installed so that the fresh air can be kept dry and delivered to the face with a reserve of cooling power.

The exclusive use of exhaust systems for auxiliary ventilation should be discouraged, since it is impractical to keep the end of the duct near the dust-producing operation, particularly when blasting.

11.2.5.4 Avoiding Dust

This is an effective way of preventing exposure to respirable dust. It is applied mainly after blasting by the insistence on a minimum re-entry period, by arranging a fixed blasting time for each working place so that other workers are not exposed to the blasting dust and fumes, and by ensuring that blasting takes place only at the end of the shift when most other workers have already been withdrawn.

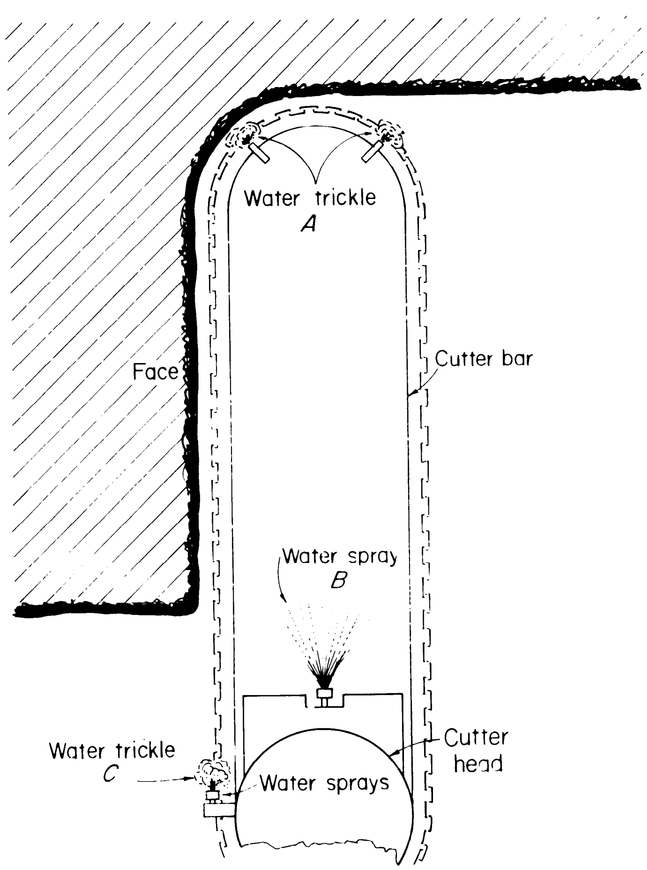


Fig. 11.2.10. Wet-bar water spray systems.

Other ways in which workers are kept out of dusty air are by arranging that men travel in downcast shafts and by having all underground waiting places in fresh air. Work may be scheduled in such a way as to reduce dusty operations upstream of a designated location.

11.2.6 DUST CONTROL IN WATER-SOLUBLE ORES

This segment reviews strategies to suppress and collect dust on cutter machines and face drills where water usage must be restricted to very low flow rates because the ore is water-soluble. Typical flow rates are 1 gpm (0.063 L/s).

11.2.6.1 Cutting Machine Dust Control

Achieving effective dust control on cutter machines can be difficult. Complications arise from the five major dust sources on a cutter machine: (1) the cutter chain during cutting, (2) the star wheel, (3) the cutter chain reentering the kerf, (4) the bugduster, and (5) a recirculation loop that develops within the kerf. The cutter chain during cutting and the recirculation loop are probably the most severe.

Wet bar type cutting techniques are used to help control mine dust by keeping the cutting chain wet. As the chain cuts the kerf, it dampens the dust produced by the cutting action at the point of generation.

Three basic wet-bar techniques, are used (Fig. 11.2.10). Water-trickle system A uses a gravity-feed or low-pressure (less than 50 psi or 345 kPa) pump to send water through the bar and to discharge it at the bar tip. Water-spray system B uses a low-pressure spray nozzle located at the front of the cutter head. For protection, the nozzle can be located within the cutter head. Water-trickle system C uses a gravity-feed or low-pressure pump to discharge water onto the cutter chain on either or both sides of the cutter head.

Tests on all three designs conducted in two salt mines showed there was no significant difference in the dust control performance. While there are many variations of the three designs, it is not expected that any one type will be more effective in controlling dust. Also the performance of the three systems is independent of water flow rate when operated between 0.25 and 1.0 gpm (0.016 and 0.063 L/s). At these water flow rates, the average total dust reduction of the three systems is 60 to 70%.

Dust control efficiency deteriorates significantly when water flow is cut to less than 0.25 gpm (0.016 L/s). Higher water flow rates (more than 0.5 to 1.0 gpm, or 0.032 to 0.063 L/s) are not likely to improve the performance of the wet bar system significantly. Also, excess water causes the dust to cake and solidify on the bit blocks, increasing bit-changing times from 10 to 15 minutes to 45 minutes or more. However, cutting the last few feet (meters) of the kerf dry will help clean the bit blocks without producing a significant amount of dust.

Several dry collection systems on cutter machines have been tried but have been generally unsuccessful. The reasons are (1) the large number of nonlocalized dust sources, (2) the tremendous amounts of dust produced are more than a machine-mounted collector can handle, and (3) the size of a suitable collector is not practical for machine mounting. More information on cutter machine dry collection systems is available in the literature (e.g., Page, 1983).

11.2.6.2 Face-Drill Dust Control

Dust control on face drills used in water-soluble ores is generally easier to accomplish than on cutter machines because there are fewer sources. Techniques tried by the mining industry to control the dust produced by face drilling includes both wet and dry systems.

Wet dust suppression systems include external water sprays, water injection through the drill, and foam injection through the drill. External boom mounted systems are typically homemade and reduce dust about 50%. Water-mist injection through the drill is available from several manufacturers. Dust reduction is much better, typically 90% or more. A homemade foam system has been used successfully (Anon., 1982). It achieved 95% dust reduction at lower water flow rates.

Dry collection systems are generally inferior to wet suppression systems. This is due primarily to the inability of dry collection systems to maintain effective intake dust capture because of face irregularities. The overall dust reduction is a product of the inlet capture efficiency and the filtration efficiency of the collector. Therefore, high filtration efficiency is pointless unless the inlet dust capture efficiency is also very high. An additional reason why dry systems are generally inferior to wet systems is maintenance. Dry systems require that the dust collected be disposed of in some convenient manner. Wet systems require no material handling; the wetted material merely dribbles down the face.

One dry collector tested has shown 76% respirable dust efficiency (Page, 1983).

11.2.7 SURFACE MINE DUST CONTROL

Personal exposure to respirable dust at surface mines primarily results from overburden drilling. Both wet and dry methods are available to suppress dust from this source. Haul road dust control is also important.

11.2.7.1 Overburden Drilling: Wet Suppression

Wet drilling systems consist of a water tank mounted on the drill from which water is pumped into the downhole airline. The water droplets in the bail air conglomerate dust particles as they travel up the annular space of the drilled hole, thus controlling dust as the air bails the cuttings from the hole.

Typical water flow rates are 0.1 to 2.0 gpm (0.0063 to 0.126 L/s) depending upon the size and type of drill, as well as the condition of the material being drilled. Flow rate is controlled manually by the drill operator by means of a control valve located in the cab. Some drills may also be equipped with a flow meter to give the operator visual indication of the flow rate. The operator simply watches the cuttings as they are bailed from the hole and adjusts the flow rate according to how moist the cuttings appear to be. This technique can be effective; however, the delay between the time the valve is opened and the time the cuttings are expelled from the hole can be several seconds. This makes it difficult for the operator to find the proper flow setting. This is especially true if the drill is drilling through alternating dry and wet strata.

Too much water pumped into the bail air wets the drill cuttings to a point where they are too heavy to be bailed up the hole. This results in the drillbit and lower part of the drill stem being seized in the hole. Freeing the drill stem is a time-consuming and costly procedure. Excessive water in the hole may also result in plugging the air orifices of the bit. The most obvious drawback to a wet system occurs when outside temperatures drop below freezing. The entire system must then be heated while the drill is in operation, and during downtime the system must be drained. The use of antifreeze compounds is prohibitively expensive because of the amount of water used.

Tests show that wet suppression systems can control respirable dust with up to 97% efficiency (Anon., 1987). Control efficiencies for 8-in. (200-mm) holes varied from a low of 9.1% at a flow of 0.2 gpm (0.013 L/s) to a high of 96.3% at a flow of 1.2 gpm (0.076 L/s). The control-efficiency vs. flow-rate curve shows a significant increase in efficiency generally between 0.2 and 0.6 gpm (0.013 to 0.038 L/s). The rate of increase of efficiency then decreases until the drill's upper flow limit is reached. In the case of the drills tested, a flow rate approaching 1.0 gpm (0.063 L/s) began to cause operational problems.

A practical and simple operational guideline can be provided to all mine operators who perform wet drilling. The guide also automatically accounts for various operating conditions such as different drills, changes in bit size, and different strata. In order to operate at close to the optimum water flow rate, the operator should slowly increase the amount of water just to the point where visible dust emissions abate. Due to the initial sharp increase of dust control effectiveness, the visible dust abatement point will be easy to identify. Increasing water flow beyond this point will not yield any significant improvements in dust control but will most likely cause seizing of the drill stem. It is important that the water flow be increased slowly to account for lag time as the air/water/dust mixture travels from the bottom to the top of the hole.

11.2.7.2 Overburden Drilling: Dry Collection

The use of a dry system involves enclosing the area where the drill stem enters the ground. This enclosure is usually accomplished by hanging a rubber or cloth "shroud" from the underside of the drill deck. This enclosure is ducted to a filtering device, the clean side of which is equipped with a fan. The fan creates a negative pressure inside the entire system, thus capturing dust as it exits the hole during drilling. The dust is removed in the filtering device and clean air is exhausted through the fan.

The integrity of the drill stem shroud, including how well it seals to the ground, is probably the single most important factor contributing to the effectiveness of a dry collection system. The shroud volume should be 1.8 times the volume of the hole and should be at a negative pressure of at least 0.2 in. of water (50 Pa). The length and width of the shroud should be 2.5 times its height. The air is ducted out of the drill stem shroud either from the top of the shroud near the outside edge or from the side of the shroud near the top. Varying the open area of the shroud will change the shroud's dust capture efficiency. As the open area is reduced, the velocity in the open area will increase. The most common open area is the gap between the bottom of the shroud and the ground, which is called the shroud height. With a shroud height of 6 to 9 in. (150 to 225 mm) or lower, it is apparent that the control system works well. However, as the height increases, the control efficiencies decrease.

During drilling, it is sometimes necessary to raise the drill shroud. This is done for two reasons: (1) to prevent large cuttings from falling back into the hole, and (2) so the operator can observe when the coal seam has been reached and stop drilling. As a result, there are times when a broken seal between the shroud and the ground or cutting cannot be avoided. Therefore, it is important for the driller to keep the open area to a minimum. This will involve raising the drill shroud more frequently.

11.2.7.3 Haul Road Dust Control

Many methods are available for haul road dust control. Water is the most obvious, but there are many others, including:

Salts—Hygroscopic compounds, i.e., calcium chloride, magnesium chloride, hydrated lime, sodium silicates, etc.

Surfactants—Substances capable of reducing the surface tension of the transport liquid, i.e., soaps, detergents, dust-set monawet, etc.

Soil cements—Compounds that are mixed with the native soils to form a new surface, i.e., calcium lignon sulphonate, sodium lignon sulfonate, ammonium lignon sulphonate, portland cement, etc.

Bitumens—Compounds derived from coal or petroleum, i.e., coherex peneprime, asphalt, oils, etc.

Films—Polymers which form discrete tissues, layers or membranes, i.e., latexes, acrylics, vinyls, fabrics, etc.

Salts increase roadway surface moisture by hygroscopically extracting moisture from the atmosphere. Surfactants decrease the surface tension of water, which allows the available moisture to wet more particles per unit volume. Soil cements, bitumens, and films generally form coherent surface layers that seal the road surface and thereby reduce the quantity of dust generated.

A recent study (Rosbury and Zimmer, 1983) showed that the highest control efficiency measured for a chemical dust suppressant, 82%, was for calcium chloride two weeks after application. Generally, however, the control efficiencies hovered in the 40 to 60% range over the first two weeks after application, and then decreased with time. After the fifth week, the limited number of data points suggest a control efficiency of less than 20%. Composite watering data were fairly uniform. Watering once per hour resulted in a control efficiency of approximately 40%. Doubling the application rate increased the control effectiveness by about 15 to 55%. Chemical dust suppressants (primarily salts and lignons) can be shown to be more cost-effective than watering under some conditions.

Table 11.2.1. Control Type vs. Proper Road Aggregate Size Gradation

Road Surface	Suppressant With Best Potential for Control
Gravel	Water
Sand	Bitumens
Good gradation	Any
Silt	Rebuild road

The cost of using a dust suppressant is very site specific. Shipping costs to. the mine are a major part of the product cost and may exceed product cost. Although sufficient test data is not yet available to conclusively prove the factor, it is apparent that certain types of dust suppressants work better in certain types of road aggregate. Recommendations appear in Table 11.2.1. They are based on the following:

1. In road surfaces with too much gravel, only watering will be effective. Chemical dust suppressants can neither compact the surface because of the poor size gradation, nor form a new surface, and water-soluble suppressants will leach.

2. In compact sandy soils, bitumens, which are not water-soluble, will be the most effective. Water-soluble suppressants such as salts, lignons, and acrylics will leach from the upper road surface. However, in loose, medium, and fine sands, bearing capacity will not be adequate for the bitumen to maintain a new surface.

3. In road surfaces with a good surface gradation, all chemical suppressant types offer potential for equal control.

4. In road surfaces with too much silt (greater than about 20 to 25% as determined from a scoop sample, not a vacuum or swept sample), no dust suppression program will be effective, and the road should be rebuilt. In high silt locations, the chemical suppressants will tend to make the road slippery, will not be able to compact the surface, or maintain a new road surface because of poor bearing capacity. Further, rutting under high moisture conditions will require that the road be regraded, which almost completely destroys chemical dust suppressant effectiveness. If the road cannot be rebuilt, watering is the best program.

All chemical dust suppressants (with infrequent watering) share one common failing as compared with frequent watering. Material spillage on roadways is extremely common in mines, particularly if ash haulers are also using haul roads. Material spilled is subject to reentrainment. With frequent watering, the spilled material is moistened at approximately hourly intervals. With chemicals, the spilled material could go 8 to 24 hours before being moistened, and the material would again redry. Therefore, in mines where spillage cannot be effectively controlled, watering will probably be more effective for dust control. In locations where trackout from an unpaved road to a paved road is a problem, chemical suppressants will generally be a better choice. Watering aggravates the trackout problem with moisture and mud, whereas a chemical suppressant, particularly bitumens and adhesives, leaves the road dry. Some mines have a dust problem in winter when temperatures are subfreezing but little moisture is present. The case for chemical suppressants over water in this case is clear.

11.2.8 MINERALS PROCESSING DUST CONTROL

A good general reference for minerals processing dust control is the *Dust Control Handbook for Minerals Processing* (Mody and Jakhete, 1987). For a discussion of the processes involved, see Chapter 25.3 in this *Handbook.*

11.2.8.1 Preventing Dust Formation

BELT CONVEYORS. The material should be loaded onto the center of the belt. Also the material and the belt should travel in the same direction and at the same speed whenever possible. The belt conveyor should be designed to operate at 75% of its full rated capacity.

LOADING POINTS. Closely spaced impact idlers (1-ft or 0.3-m centers) should be located at transfer points. These will absorb the force of impact and prevent deflection of the belt between the idlers, thus preventing dust leakage under the skirting rubber seal.

Skirtboards are used to keep the material on the belt after it leaves the loading chute. They are equipped with flat rubber strips that provide a dust seal between the skirtboards and the moving belt. Improved skirtboard designs are available (Mody and Jakhete, 1987). Muckshelves can be installed in the belt conveyor's material impact zone to load the material centrally on the belt.

A belt scraper should be installed at the head pulley to dislodge fine dust particles that may adhere to the belt surface. A scrapings chute should also be provided to redirect the material removed by the belt scraper into the process stream or container. A V-plow installed on the noncarrying side of the belt will clean the belt and prevent buildup of material and dust on the tail pulley, thus keeping the belt properly aligned.

TRANSFER CHUTES. Transfer chutes transport ore from one piece of equipment to another. When designing a transfer chute: (1) the chute depth should be at least three times the maximum lump size to avoid jamming; (2) the chute should be designed so the material falls on the sloping bottom of the chute and not on the succeeding equipment; (3) wherever possible, the material should fall on a local rockbox or stonebox rather than on the metal surfaces; (4) abrupt changes of direction must be avoided to reduce the possibility of material buildup, material jamming, and dust generation; (5) curved, perforated, or grizzly chute bottoms should be used when the product stream consists of fines and lumps; placing a layer of fines ahead of the lumps on the belt helps prevent heavy impact of material on the belt, which reduces belt wear and dust generation; (6) spiral chutes are used to prevent breakage of fragile or soft material; and (7) bin-lowering chutes are used to feed bins and hoppers without generating large amounts of dust.

ENCLOSURES. Enclosures are used to contain dust emissions around a dust source. When designing an enclosure for a dust source: (1) enclosures should be spacious enough to permit internal circulation of the dust-laden air; (2) enclosures should be arranged in removable sections for easy maintenance; (3) a hinged access door should be provided to aid routine inspection and maintenance; and (4) dust curtains should be installed at the open ends of the enclosures to contain dust and reduce airflow.

CRUSHERS. Crushers emit dust primarily from two points, the discharge and the feed. Dust control measures are not usually considered in the design of a crusher. However, the use of shrouds or enclosures for crushers can contain the dust so that a dust control system can operate more efficiently. The following measures are recommended: (1) a crusher feedbox with a minimum number of openings should be installed, and rubber curtains should be used to minimize dust escape and airflow, and (2) the crusher should be choke fed to reduce air entrainment and dust emission. Dust escape at the crusher discharge end can be minimized by properly designed and installed transfer chutes.

SCREENS. The rate of dust generated by screens cannot be altered. However, properly enclosing the screen can reduce dust emissions. A complete enclosure that can be easily removed for maintenance and inspection should be used. Some screen manufacturers provide sheet-metal covers to enclose the top of the screen. These covers are effective when properly maintained. However, they do not provide a dust seal between the moving screen surfaces and the stationary chutes.

STORAGE BINS AND HOPPERS. Dust emissions during feeding operations can be minimized by installing a bin-lowering chute and by completely enclosing the bin or hopper. Dust emissions can be minimized by installing a telescopic chute or by installing a loading spout. Such spouts are sophisticated versions of the telescopic chute and are used to load and stack ore into barges, trucks, and railroad cars. The falling material is enclosed by a flexible duct, acting as a chute, which retracts as the height of the material pile increases. The duct also prevents airflow during free fall of material between the chute and stockpile. The generated dust is captured by the same flexible duct and is conveyed, countercurrent to the material flow, to a dust collector.

BUCKET ELEVATORS. Bucket elevators emit dust from two points, the boot where material is fed and the head wheel where material is discharged. The steel casing that enclose the buckets and chain assembly contains dust effectively unless there are holes or openings in the casing. Emissions at the boot of the bucket elevator can be reduced by proper design of a transfer chute between the feeding equipment and the elevator. Dust production can be reduced significantly by keeping the height of material fall to a minimum and by gently loading material into the boot of the elevator. Proper venting to a dust collector will control dust emission at the discharge end of the bucket elevator.

SCREW CONVEYORS. Normally, screw conveyors are totally enclosed except at the ends, where emissions can be controlled by proper transfer chute design. To maintain a proper dust seal, a neoprene rubber gasket should be installed on the trough cover. Many manufacturers provide two-bar flanges and formed-channel cross members that make a continuous pocket around the trough. The flange-cover sections are set in this channel. Once the channel section is filled with dust, an effective dust seal is created.

STOCKPILES. All types of stockpiles can be a significant dust source. Generation of dust emissions from stockpiles is due to the formation of new stockpiles and wind erosion of previously formed piles. During formation of stockpiles by conveyors, dust is generated by wind blowing across the stream of falling material and separating fine from course particles. Additional dust is generated when the material hits the stockpile.

Dust from stockpiles can be reduced through the following measures:

1. Minimizing height of free fall of material and providing wind protection using

   a. Stone ladders, which consist of a section of vertical pipe into which stone is discharged from the conveyor. At different levels, the pipe has square or rectangular openings through which the material flows to form the stockpile. In addition to reducing the height of free fall of material, stone ladders also provide protection against wind.

   b. Telescopic chutes, in which the material is discharged to a retractable chute. As the height of the stockpile increases or decreases, the chute is raised or lowered accordingly. Proper design of the chute can keep the drop to a minimum.

   c. Stacker conveyors, which operate on the same principle as telescopic chutes.

2. Minimizing wind erosion of the stockpile by locating stockpiles behind natural or manufactured windbreaks, locating the working area on the leeward side of the active piles, and covering inactive piles with tarps or other inexpensive materials.

3. Minimizing vehicle traffic on or around the stockpile.

4. Using specialized equipment such as a reclaimer to minimize the disturbance of the stockpile or providing a tunnel underneath to reclaim the material.

11.2.8.2 Dust Control Systems

A dust collection system, also known as the local exhaust ventilation system, is one of the most effective ways to reduce dust emissions.

EXHAUST HOODS. The rate of airflow through the exhaust hood (i.e., the exhaust volume rate) is the most important factor for all types of hoods. For local, side, downdraft, and canopy hoods, the location is also important because the rate of airflow is based on the relative distance between the hood and the source. The shape of the exhaust hood is another design consideration. If the hood shape is not selected properly, considerable static pressure losses may result.

DUCTWORK. Ductwork design includes the selection of duct sizes based on the velocity necessary to carry the dust to the collector without settling in the duct. To prevent dust from settling and blocking the ductwork, transport velocities should range from 3500 to 4000 fpm (17.5 to 20 m/s) for most industrial dust (such as granite, silica flour, limestone, coal, asbestos, and clay) and from 4000 to 5000 fpm (20 to 25 m/s) for heavy or moist dust, such as lead, cement, and quick lime. Following are recommended minimum transport velocities for different characteristics of dust:

Material	Minimum Design Velocity, fpm (m/s)	
Very fine, light dusts	2000	(10)
Fine, dry dusts and powders	3000	(15)
Average industrial dusts	3500	(17.5)
Coarse dusts	4000-4500	(20–22.5)
Heavy or moist dust loading	4500 and up	(22.5 and up)

11.2.8.3 Wet Dust Suppression Systems

Wet suppression systems fall into three categories: (1) Plain water sprays. This method uses plain water to wet the material. However, it is sometimes difficult to wet surfaces with plain water due to its high surface tension. (2) Water sprays with surfactant. This method uses surfactants to lower the surface tension of water. The droplets spread further and penetrate deeper into the material pile. (3) Foam. Water and a special blend of surfactant make the foam. The foam increases the surface area per unit volume, which increases wetting efficiency.

The advantages and disadvantages of each are shown in Table 11.2.2.

11.2.8.4 Background Dust Sources

Background dust sources can expose workers at mineral processing facilities to more significant dust concentrations than from their normal job functions. These background dust sources include such things as soiled work clothes, blowing clothes off with compressed air, broken bags of product material both at the fill station and during the conveying process, bag hoppers overflowing with product, improper housekeeping techniques such as dry sweeping of floors, and dusty makeup air which many times flows into the mill buildings from outside sources.

The best way to truly detect and measure these background dust sources is through the use of an instantaneous dust monitor,

Table 11.2.2.
Comparison of Water Sprays

Advantages	Disadvantages
Plain Water Sprays	
It is probably the least expensive method of dust control.	Water sprays cannot be used for products that cannot tolerate excessive moisture.
The system is simple to design and operate.	Water sprays cannot be used when temperatures fall below freezing.
A limited carryover effect at subsequent transfer points is possible.	Usually, dust control efficiency is low, unless large quantities of water are used.
When good mixing of water and material can be achieved, dust generation can be reduced effectively.	Freeze protection of all hardware is necessary.
Enclosure tightness is not essential.	Careful application at transfer points that precede a screen is required to prevent blinding.
Water Sprays With Surfactants	
This method is used when surfactants are tolerated but excessive moisture is not acceptable.	Capital and operating costs are higher than water-spray systems.
In some cases, dust control efficiency is higher than with plain water sprays.	Careful application at transfer points that precede a screen is required to prevent blinding.
Equivalent efficiency is possible with less water.	Equipment such as the pump and the proportioning equipment used to meter the flow of surfactant requires maintenance.
	Freeze protection of all hardware is necessary.
Foam	
When good mixing of foam and product stream can be achieved, dust control efficiency is greater than water with surfactants.	Operating costs are higher than with finely atomized water-spray systems.
	The product is contaminated with surfactants.
Moisture addition is usually less than 0.1% of the material weight.	Careful application at transfer points that precede a screen is required to prevent blinding.

such as the MIE Model RAM-1 unit. By monitoring the worker's exposure throughout the entire workday, significant dust producing events can be identified and controlled.

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Chapter 11.3
RADIATION CONTROL

ROBERT C. BATES

Providing a mining environment that is as safe and healthful as possible concerns everyone in the mining industry. Radon, thoron, and their respective daughter products create airborne radiation problems, but with proper safety precautions, these hazards can be minimized.

Although always associated with uranium mining, radioactive contaminants are not confined to uranium mines alone, and significant radon daughter concentrations have been measured in copper, gold, lead/zinc, phosphate, limestone, coal, and over 20 other types of mines (Anon., 1987; Rock et al., 1975). A review of Mine Safety and Health Administration (MSHA) data for 1985 through 1987 revealed that 139 mines in the United States, extracting 18 commodities, had radon daughter concentrations at a working level of 0.1 or more (Anon., 1988). (See 11.3.5 for a more detailed definition of working level.) Table 11.3.1 summarizes information on five commodities.

11.3.1 RADIATION SOURCES

Ionizing radiation damages any material it passes through, and any excessive exposure is a hazard. Three types of ionizing radiation occur naturally: alpha, beta, and gamma. Alpha particles consist of two protons and two neutrons and can travel only 1 to 3 in. (25 to 75 mm) in air. Although they cannot penetrate a piece of paper or a layer of skin, they cause the greatest amount of damage per unit length of path, and internally they can be quite damaging.

Beta particles are negatively charged electrons and are much smaller than alpha particles. They can travel several yards (meters) in air and about 0.4 in. (10 mm) in water or tissue. Actual travel distances depend on the energy of the particle and density of the absorbing material.

Gamma photons have no mass or charge and do not have a definite range. They interact directly with orbital electrons to ionize atoms, and all or part of their energy is transferred during an interaction.

Two radioactive decay chains, that is, uranium and thorium, are of concern in mining. Uranium is found throughout the earth's crust; its decay chain includes radium (^{226}Ra), which is the parent of radon (^{222}Rn) (Table 11.3.2). Radon is a chemically inert gas that easily escapes from the material in which it is formed and decays into the particulate daughter products RaA, RaB, RaC, and RaC′ (Fig. 11.3.1). Complete descriptions of radon and radon daughter build-up and decay are available in Evans (1969). When radon daughters are inhaled, they may be deposited on exposed tissues of the tracheobronchial airways, where subsequent decay of RaA and RaC′ (alpha radiation) poses a cancer risk (Anon., 1987).

The thorium decay chain (Table 11.3.3) involves another isotope of radon (^{220}Rn) called thoron. It also is chemically inert and behaves in the same manner as radon. Thoron is common in some uranium mines in Canada (Bigu, 1981) and is a factor in UK mines (Dixon et al., 1984), but it is infrequently found in US mines (Rock, 1975). It can become a problem in any mine where thorium minerals exist because alpha radiation from thoron daughters ThA, ThC, and ThC′ (Fig. 11.3.2) poses the same cancer risk as radon daughters.

Probably the best-known fact about radon and thoron concentrations in mine atmospheres is their variability in space and time. Many parameters, including ore particle size and radium content, influence the escape of radon and thoron atoms from their points of origin. The smallest grains and largest concentrations of radium release the most radon atoms (Pogorski, Busigin, and Phillips, 1981). Only a fraction of the radon escapes the mineral grain in which it is formed, however; this fraction is defined by the coefficient of emanation, which might range from 0% to near 100% (Austin and Droullard, 1978). Once in interstitial spaces, movement of radon is controlled by porosity, permeability, pressure changes, interstitial fluid, and fractures (Tanner, 1964). An excellent summary of the radon literature, including a review of steady-state diffusion and convection equations, is given in Tanner (1964).

However, conditions in mines are far from steady state; mining activities, such as blasting and slushing, influence concentrations (Fig. 11.3.3) (Franklin, 1981). Even if all mining ceases and other factors remain stable, large changes in radon concentrations can be caused by changes in barometric pressure (Fig. 11.3.4) (Franklin, 1981). Water has also been shown to be a carrier of radon in Newfoundland fluorspar mines (Morrison et al., 1984).

Table 11.3.1. Number of Mines Having Radon Daughter Concentrations in Excess of 0.1 WL

Commodity	≥ 0.1 WL	≥ 0.3 WL[a]
Uranium	50	45
Gold (lode/placer)	41	29
Uranium/Vanadium	13	13
Silver	12	6
Lead/Zinc	5	4
Total No. of Mines	121	97

Source: Anon., 1988.
[a]Includes mines in which radon daughter concentrations measured 0.1 WL or greater.

11.3.2 TECHNICAL CONTROL

Control of airborne radiation starts with preventing or retarding the flow of radon and thoron into a mine ventilating system. Since a radioactive decay process is involved, time is a factor in ventilation design. Methods that confine radon or thoron, such as bulkheading or sealing, provide the time necessary to allow radioactive particles to decay through their short half-life isotopes to less hazardous lead isotopes. However, once radon, thoron, and their daughters reach a mine atmosphere, then dilution and removal are the only control measures.

11.3.2.1 Confinement

The purpose behind confinement is to prevent radon and thoron from reaching the atmosphere of a mine. Even if the flow

Table 11.3.2. Uranium Disintegration Series

Isotope	Symbol	Historical Name	Half-Life	Radiation	α Energy, Mev
Uranium 238	^{238}U	Uranium I	4.5×10^9 yr	α	
Thorium 234	^{234}Th	Uranium X_1	24.1 days	β, γ	
Protactinium 234	^{234}Pa	Uranium X_2	1.18 min	β, γ	
Uranium 234	^{234}U	Uranium II	2.50×10^5 yr	α, γ	
Thorium 230	^{230}Th	Ionium	7.6×10^4 yr	α	
Radium 226	^{226}Ra	Radium	1620 yr	α, γ	
Radon 222	^{222}Rn	Radon	3.82 days	α	5.49
Polonium 218	^{218}Po	Radium A	3.05 min	α	6.00
Lead 214	^{214}Pb	Radium B	26.8 min	β, γ	
Bismuth 214	^{214}Bi	Radium C	19.7 min	β, γ	
Polonium 214	^{214}Po	Radium C′	164×10^{-6} sec	α	7.69
Lead 210	^{210}Pb	Radium D	22.0 yr	β, γ	
Bismuth 210	^{210}Bi	Radium E	5.0 days	β	
Polonium 210	^{210}Po	Radium F	138.4 days	α	5.30
Lead 206	^{206}Pb	Radium G	Stable		

Source: Anon., 1967.

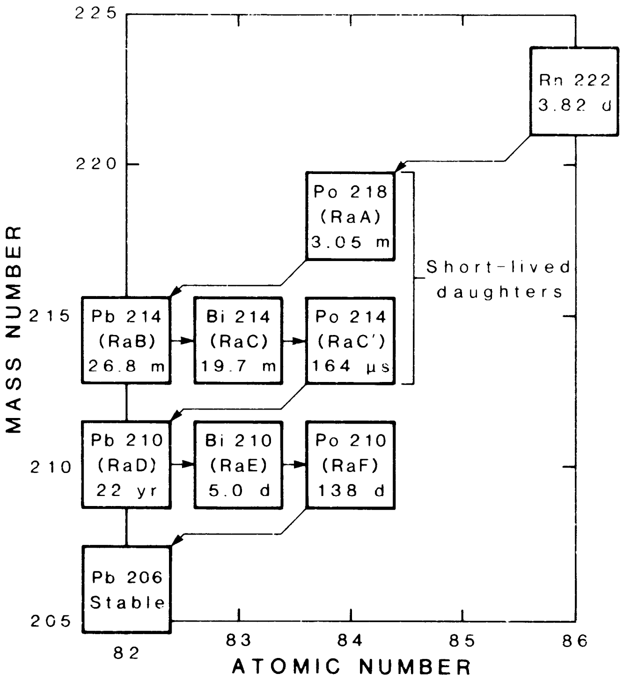

Fig. 11.3.1. Radioactive decay of radon through short-lived daughters to stable lead (Anon., 1979).

cannot be totally stopped, there is benefit in slowing it to allow some radioactive decay to take place.

Newly produced RaA and ThA are free (unattached) atoms, and they are attracted very quickly to rock surfaces, mine aerosols, or other materials. If trapped on surfaces such as mine walls or grain boundaries within the rock, they are no longer a hazard unless they are redistributed by blasting. Confinement methods include the use of bulkheads, backfill, overpressure, and sealants.

BULKHEADS. The most important method of confining radon, thoron, and their daughters is to seal mined-out stopes or other inactive areas (raises, shafts, and drifts) with good ventilation bulkheads. Many tests have demonstrated that effective bulkheads must be either leak-proof (which is almost impossible in a mine environment) or designed so that any leakage is from an active part of a mine into the bulkhead area. Even a small amount of highly contaminated air leaking from behind a faulty bulkhead can make otherwise good ventilating air unusable. Leakage can be prevented, even against barometric pressure changes, by establishing a negative pressure behind the bulkhead. Such pressure can be developed as part of the main ventilation system if the area behind the bulkhead is connected to exhaust airways. For a deadend area, negative pressure can be assured by running bleeder pipes through the bulkhead and placing a fan at the exhaust end to maintain a flow out of the bulkhead area into an exhaust airway (Franklin, 1981).

BACKFILL. The use of mill tailings and other materials in metal mines to backfill for ground support has been practiced for many years. The trend seems to be increasing, and more mines, including uranium mines, will be using backfill in the future. If use of uranium mill tailings is planned, then, in addition to standard properties determinations, some evaluation of radiation properties is needed (Nantel and Archibald, 1981). Radium content, emanating power at different grain-size fractions, and effects of cements can be tested in a laboratory before fill is placed underground.

Backfilling mined-out stopes can be of use in radiation control as well as ground control. The sheer bulk of backfill material inhibits radon and thoron flow from stope surfaces. Also the physical barrier prevents loss of ventilating air through inactive areas and recirculation of contaminated air. During a field test in a uranium mine in Grants, NM, there was an 87% reduction in radon flow from a stope after the stope was filled with plus 200–mesh uranium mill tailings. A temporary increase in downstream radon concentrations occurred while placing sand slurry, but concentrations quickly returned to normal after pouring stopped (Franklin, 1981).

Some questions have been raised concerning possible environmental problems associated with small quantities of radium in uranium mill tailings. In the Franklin study, measurements taken from a localized drainage ditch at the point where slurry water drained from the stope indicated increased radium concentrations, but measurements taken farther downstream (that is, where the water left the mine) showed no detectable increases in radium.

Table 11.3.3. Thorium Disintegration Series

Isotope	Symbol	Historical Name	Half-Life	Radiation	α Energy, Mev
Thorium 232	^{232}Th		1.41×10^{10} yr	α	4.01 (76%) 3.95 (24%)
Radium 228	^{228}Ra	MesoThorium I	6.7 yr	β	
Actinium 228	^{228}Ac	MesoThorium II	6.13 yr	β, γ	
Thorium 228	^{228}Th	RadioThorium	1.910 yr	α, γ	5.43 (71%) 5.34 (28%)
Radium 224	^{224}Ra	Thorium X	3.64 days	α, γ	5.68
Radon 220	^{220}Rn	Thoron	54.5 sec	α	6.29
Polonium 216	^{216}Po	Thorium A	0.145 sec	α	6.78
Lead 212	^{212}Pb	Thorium B	10.64 hr	β, γ	
Bismuth 212	^{212}Bi	Thorium C	60.6 min	α, γ (36%) β, γ (64%)	6.051 (69.9%) 6.090 (27.2%)
Polonium 212[a]	^{212}Po	Thorium C′	3.04×10^{-7} sec	α	8.78
Thallium 208[b]	^{208}Tl	Thorium C″	3.10 min	β, γ	
Lead 208	^{208}Pb	Thorium D	Stable		

Source: Anon., 1979.
[a] Product of Bismuth 212 beta decay (64%).
[b] Product of Bismuth 212 alpha decay (36%).

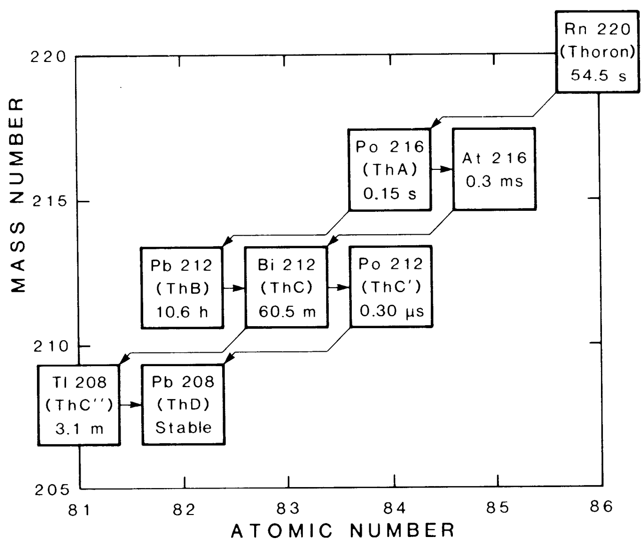

Fig. 11.3.2. Radioactive decay of thoron to stable lead (Anon., 1979).

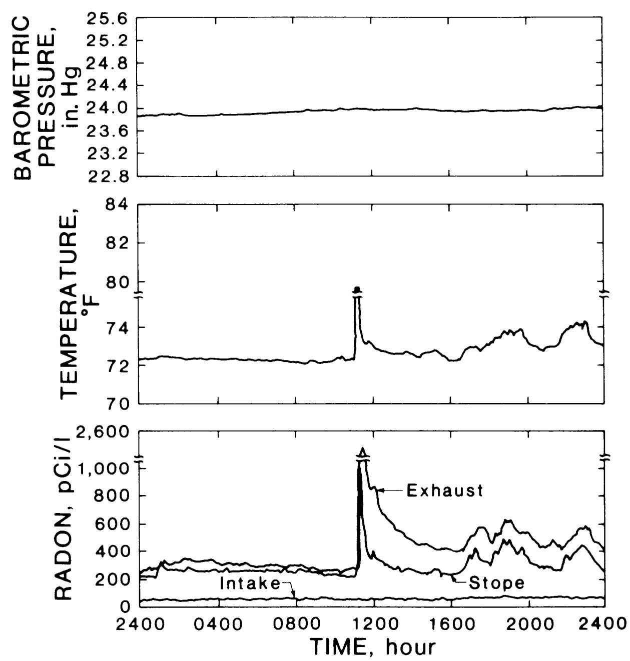

Fig. 11.3.3. Effects of blasting (highest peak) and slushing on radon concentrations.

OVERPRESSURE. Several research projects have examined the potential of reducing radon concentrations through the use of pressurization. In the earliest overpressure experiment, radon concentrations in a small underground room were lowered significantly by increasing pressure inside the room (Schroeder, Evans, and Kramer, 1966). A more recent test of blowing vs. exhaust ventilation was performed in a small, shallow uranium mine. Total radon production from the mine was reduced 20% by changing the ventilation pressure from –2.1 to +1.6 in. water (–523 to 398 Pa) (Franklin, 1981).

Three conditions are needed to make pressurization feasible: porous rock, permeable rock, and a sink to receive the radon. A mathematical study (Edwards and Bates, 1980) indicated that a large, permeable mass is not an effective sink, because eventually the sink is pressurized and diffusion rebuilds the flux to original levels. However, another study indicated that a mine opening less than 330 ft (100 m) away could serve as a sink (Bates and Edwards, 1981). Greater degrees of porosity and permeability allow more flow of interstitial gases toward a sink, and thereby permit larger reductions in radon flux for a given pressure. The calculations also project a large radon flux increase in the sink; therefore, personnel should be kept out of the low-pressure, exhaust side of a mine.

SEALANT. It would appear that an ideal way to prevent radon from escaping from the rock would be to seal the surface with a barrier coating. Laboratory tests on over 100 potential sealants demonstrated that some materials could stop 85 to 100% of the radon loss (Franklin, Nuzum, and Hill, 1975), but

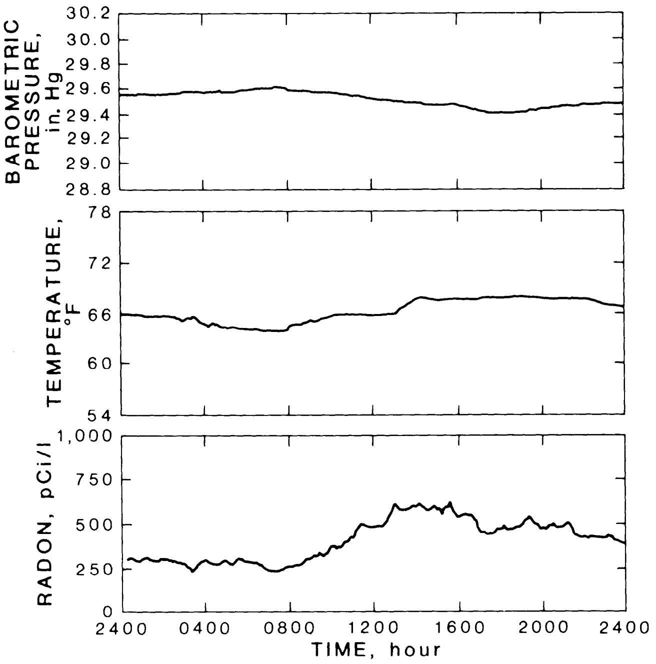

Fig. 11.3.4. Effects of diurnal changes on radon concentration in a shallow mine.

many sealants were rejected because they were poor barriers or were not safe to use underground. Water-based epoxy and acrylic latex sealants were safe for underground use. In one underground uranium mine test, radon concentrations were reduced from 80 to 100 picocuries per liter (pCi/L) to 25 to 30 pCi/L after the walls had been coated, which is about a 70% reduction (Lindsay, Schroeder, and Summers, 1981). Laboratory tests in West Germany found several materials that stopped over 93% of the radon exhalation from drill cores (Reim, Thome, and Haupt, 1984). Even though sealants can reduce the amount of radon escaping from rocks, there are two factors that limit their use: (1) ease of application and (2) cost. If the rock surface is rough or fractured, a smoothing layer of shotcrete is needed before two coats of the sealant are applied. Such multistep process is usually not compatible with mining activities in stopes. Material costs in 1981 ranged from \$0.30 to \$1.10/ft^2 (\$3 to \$12/m^2 (Franklin, 1981), and labor costs to place and maintain sealants raise the costs even more. Total application costs limit the use of sealants to special cases, such as permanent airways.

11.3.2.2 Dilution and Removal

By far the most common and important means of controlling radon and thoron is mechanical ventilation (see Chapters 11.6 and 11.7). Radon and thoron daughter concentrations start increasing as soon as fresh air is contaminated with the parent gas. In about 20 minutes, radon daughters have reached a little over 30% of their final equilibrium concentrations with the parent radon. In many cases, this is enough to make the air unusable. Therefore, moving the ventilating air through the work areas efficiently and into the exhaust airways is important.

General Ventilation. The general concepts for ventilating mines where radon and thoron are present are the same as for any mine (Rock and Walker, 1970). Air enters the mine through one or more fresh air openings, circulates through active workings, and returns to the surface through exhaust airways. Ideally, mining starts near return airways and progresses toward intake airways. Fans for a blowing system are located in the fresh air openings and keep the entire mine under positive pressure. For an exhaust system, fans are located in the return airways; consequently, the mine is under a negative pressure.

In a shallow underground mine, radon and thoron control might benefit from a blowing system whereas an exhaust system might exacerbate the problem (Franklin, 1981). In deep mines, however, other factors become more important. Many ventilation engineers prefer a combination push-pull system. Fans are installed in both intake and return airways. Fresh air is delivered to active areas under positive pressures. After the air is used, it is exhausted through inactive areas under negative pressures. Pressures across air doors and stoppings can be lower with these combination systems than with either blowing or exhaust systems.

To prevent contamination from radioactive products, openings for fresh air should be in nonradioactive waste rock. This is a major difference between ventilating a typical metal or nonmetal mine and any mine containing radon or thoron. Low-grade lead mineralization does not affect ventilating air passing through it; however, air passing through a zone of low-grade uranium mineralization can be so extensively contaminated that the only solution is to increase the airflow to dilute the contamination. In mines with radon and thoron, it is important to deliver fresh air to workers as quickly as possible and then quickly direct it away from them. If the air must be used in more than one work area, then the velocities must be maintained at high levels to keep the travel times and daughter growths low. Additional references on the use of ventilation to control radon and thoron are found in Dory (1981) and Schroeder and Evans (1969).

Daughter products continue to be generated as long as there is radon or thoron available, and any recirculation of ventilating air results in unwanted residence times. During such delays, the daughters can increase to unacceptable levels. Causes of recirculation include (1) fans or fan inlet pipes that are installed in the wrong locations, (2) flow through interconnecting openings that permits exhaust from one part of a mine to mix with fresh air from another, and (3) higher pressures on the exhaust side of an air door or stopping that forces contaminated air through any leakage points into fresh air. Less obvious are situations where high-velocity air passes the entrance of a dead-end side drift and sets up an eddy. Eddies are also sometimes created as air enters a stope or by ventilation tubing directed to sweep only part of a stope.

In the writer's opinion, the most insidious recirculation problem stems from temperature differentials. Natural drafts and convection currents can complicate an otherwise well-designed system. Airflow on the floor of a drift or stope may be opposite to flow along the back. In an unreported field test, warm air from a stope moved more than 20 ft (6 m) upstream along the back of a colder intake airway and was recirculated by an auxiliary ventilation fan.

There is at least one computer program available to identify recirculation loops (Notely, Wheeler, and Fytas, 1984) but the best tool an engineer can have to identify most recirculation problems is a smoke tube. With smoke, directions of low-velocity airflow, differences in direction from floor to back, and eddies can be seen. Careful measurements of airflow in all branches of a ventilation system will help the engineer identify where there are gains and losses in air quality.

Auxiliary Ventilation. All development openings and some stopes must be ventilated with auxiliary fans, pipes, and/or flexible tubing. Blowing systems are the most effective in supplying clean air to miners, provided fans and ducts are installed properly and sufficient air is supplied to dilute and move away any airborne contaminants. High-velocity air at the discharge end of a duct or tube can be directed to sweep a work area clean, and as with any system, contaminated air should be directed toward exhaust airways as efficiently as possible. Auxiliary ventilation systems are always in active workings where they are exposed to moving equipment and possibly flyrock; therefore, these systems should be inspected frequently and repaired.

ESTIMATING VENTILATION REQUIREMENTS. Experience and information collected from other mines in the same area are usually the best approach to developing ventilation requirements for a new mine. If practical information is not available, a more theoretical approach might be useful (Chakravatti, 1981). However, it is much easier to estimate the type and degree of ventilation needed to reduce existing radon daughter levels in a stope or small mine. In 1964, Harris and Bales published their derivation of a ventilation equation,

$$V_2 = V_1\ (WL_1/WL_2)^{0.56} \qquad (11.3.1)$$

where V_2 is new ventilation quantity, V_1 is initial ventilation quantity, WL_1 is initial radon daughter working level, and WL_2 is desired radon daughter working level.

Successful application of Eq. 11.3.1 requires that (1) the intake air is not contaminated, and (2) the air change per minute is between 0.03 and 0.30. (Air change per minute is stope or mine volume divided by air quantity.) If these two conditions are met, the predicted volume, V_2, should be within 5%. However, if these conditions are not met, the equation underestimates the amount of required ventilation.

Using contaminated intake air drastically increases the quantity needed. Obviously, if contamination of intake air is equal to the desired level, then effective ventilation is impossible. In a study of uranium-mine ventilation costs for both current and low-exposure limits, there were some mines that could not meet the low-exposure standards because of contaminated intake air (Bloomster et al., 1984).

11.3.2.3 Air Cleaning

Removing daughter products from mine air for reuse in other areas is one way of reducing overall ventilation costs. A study of possible systems established an effectiveness criterion of 90% reduction with throughput capacity between 4500 and 60,000 cfm (2.1 to 28.3 m^3/s) (Lawter, Lindsay, and Jashani, 1981). Of the many industrial cleaning systems examined, only filtration and electrostatic precipitation were judged to have the necessary capabilities for mine use. Any filtration system must have efficient filters, and the system must be cleaned frequently to maintain a reasonable pressure drop across the filter. Electrostatic precipitators also need to be maintained, but the main caution is to keep airflow within design limits. Forcing too much air through a precipitator severely impairs collection efficiency.

A pulse-cleaning filter system was designed as part of a US Bureau of Mines contract research effort (Schroeder, Muldoon, and Babbitt, 1984). During tests of the system, radon daughter and dust concentration reductions averaged 95% with an airflow of 11,000 to 12,000 cfm (5.2 to 5.7 m^3/s). Filter cartridge life was greater than six months, and pulse cleaning effectively removed dust and diesel contaminants. During a six-month follow-up test in an active mine (Holub, 1987), radon daughters were reduced 98% and condensation nuclei were reduced only 60%. This is an important result, as there are many nuclei left in the filtered air for attachment for newly formed RaA.

Radon is not removed by these systems; therefore, radon daughters will be created at predictable rates. Depending on incoming radon concentrations, air might be unusable in 1 to 10 minutes. Thoron, with a 55-second half-life, does not present the problems of radon, with a 3.825-day half-life. About 97% of the thoron released into a mine atmosphere decays within 4.6 minutes.

11.3.3 MONITORING

Ventilation patterns can be changed as result of relatively minor damage to a bulkhead. Even without mining-related activities, concentrations can change in response to barometric pressure (Fig. 11.3.4). Therefore, for any control program to be effective, there must be an effective monitoring program.

11.3.3.1 Grab Sampling

At a minimum, radon and thoron daughters must be sampled. While there are several methods for measuring airborne radon daughters (Borak et al., 1981), the method most frequently used in the United States is the Kusnetz method (Kusnetz, 1956), in which a known volume of mine air is filtered for a 5-minute period and the alpha radioactivity from the filter is measured 40 to 90 minutes later. If information on individual radon daughters (RaA, RaB, and RaC) is needed, alpha measurements must be taken on each filter at several designated times, for example, at 5, 15, and 30 minutes after sampling. More convenient is the Thomas (1972) method, which involves scalar measurements taken 2 to 5, 6 to 20, and 21 to 30 minutes after the end of sampling.

Thoron daughters can be measured on the same filter used for the radon daughter measurement. However, at least 50 L of air should be filtered, and in contrast to the five-minute sampling period needed for radon daughters, thoron daughter samples can be collected during a one-hour sampling period (Rock, 1975). The filter is counted five hours or more after sampling and the daughter concentration calculated using the appropriate instrument calibration and tables of decay factors. If both radon and thoron daughters are present, then a correction is needed to calculate radon daughters.

Information on radon concentrations is sometimes needed. Point sources, such as fractures or water, can be identified more precisely with radon measurements than they can with radon daughter measurements because radon daughters require time to build. Radon information is also useful for age-of-air calculations, because older-than-expected age-of-air might indicate recirculation in a ventilation system. The easiest method uses a phosphor-lined flask as first described in Lucas (1957). A description of flask calibration procedures is given in Beckman (1975).

11.3.3.2 Continuous Monitors

Continuous monitors provide the ventilation engineer with timely information on conditions underground and may include sensors to measure air velocity, temperature, relative humidity, barometric pressure, fan status, and air-door status, in addition to measuring radon and radon daughters. Several continuous monitoring systems have been described in the literature (Briggs, King, and Franklin, 1984; Franklin and Droullard, 1983; Kawaji, Pai, and Phillips, 1981). Some continuous radon daughter monitors measure the beta and gamma radiation from filtered RaB and RaC, while others use alpha counts to calculate thoron daughter concentrations in addition to radon daughter concentrations. None of the continuous monitors give an instantaneous reading of true concentrations because there is always some buildup and decay of radon daughters within the device to consider. Various schemes, such as polynomial and binomial filtering (Roze, Raz, and Bigu, 1984), have been proposed to show short-term fluctuations in radon daughter concentrations and to correct for the time lags inherent in the instruments. Even without special data-handling techniques, continuous monitors have a place in radiation control. They can warn mine management of control problems much earlier than can a human sampler, thereby decreasing exposure time or preventing any overexposure at all.

11.3.4 BETA AND GAMMA RADIATION EXPOSURES

In past years, uranium ore grades in the United States and Canada were between 0.1 to 1.0% U_3O_8. At these grades, the Federal Radiation Council (Anon., 1967) noted that it was unlikely that external whole-body radiation doses would exceed the 5 rem annual exposure level. However, the council expressed some concern with ore grades over 1 to 5% U_3O_8. A reevaluation of beta radiation exposure during the mining and milling of high-grade uranium ore suggested that such exposures might become important (Mernagh and Chambers, 1984). In four Canadian mines, the exposure limit for gamma radiation and radon daughters individually was not exceeded (Ching, Ho, and Bayne, 1984); however, it was likely that some miners would exceed a proposed standard for a combination of gamma and radon daughters. If beta and gamma radiation becomes a concern, as indicated by these papers, then moving individuals away from high-exposure areas is probably the main control option.

11.3.5 REGULATIONS

Permissible radon daughter concentrations and exposures are expressed in terms of *working levels*, or WL. One WL is defined as 1.3×10^5 mev of potential alpha energy per liter of air resulting from the decay of short-lived radon daughters (RaA through RaC). (Radon daughters in equilibrium with 100 pCi of radon will yield approximately 1 WL.) Originally, exposure to 1 WL of radon daughters was thought to be safe. However, with additional epidemiological data available, exposure limits were lowered and redefined to 4 working-level months (WLM) of exposure per year (Anon., 1989). WLM is calculated by adding each individual WL exposure $\times$ hours exposed $\div$ 173 hours. An average radon daughter concentration of 0.33 WL will yield about 4 WLM of exposure per year. Presently, recommendations are being considered to make the regulations more stringent by decreasing the permitted exposure levels (Anon., 1987).

If any individual is accidentally exposed to higher concentrations that will cause the cumulative radon or thoron daughter exposure to approach the limit, then that individual should be moved to a location where concentrations are lower. This is administrative control of total exposure, and it does not permit deliberate overexposure.

Personal respirators, properly fitted, are required in environments exceeding 1.0 WL (Anon., 1989, 30 CFR 57.5044). This is a temporary measure to protect individuals while establishing controls or to allow occasional entry into hazardous areas. Where radon daughter concentrations exceed 10 WL, then a respirator that removes radon as well as its daughter products or that supplies fresh air must be worn (Anon., 1989, 30 CFR 57.5046).

Finally, at least one radon daughter sample must be taken in the exhaust airway of each mine in the United States. If the concentration is over 0.1 WL, then additional samples must be collected. Additional studies and sampling are needed if active working places have high concentrations. The regulations require records to be kept where radon daughter concentrations are above 0.3 WL.

11.3.6 COORDINATING PRODUCTION AND RADIATION CONTROL

Control of airborne contaminants is an important factor in supplying underground workers with an environment that facilitates productivity. Production decisions, both short- (daily) and long-range, should involve ventilation and radiation control personnel in addition to management people, planning engineers, and foremen. Continuing education of everyone at each mine is needed to keep all aware of the importance of maintaining the integrity of the radiation control system and reporting any observed problems. Removing a bulkhead on the night shift to get extra room for broken ore can ruin a manager's day, especially if an inspector arrives.

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Chapter 11.4
HEAT, HUMIDITY, AND AIR CONDITIONING

BRUCE JOHNSON

11.4.1 SYMBOL CONVENTION

AHP Air horsepower, hp (kW)
BHP Brake horsepower, bhp (kW), of a fan motor
c Humid specific heat of humid air, Btu/lb-ºF (kJ/kg-°C)
d_a Density of dry air, lb/ft^3 (Kg/m^3)
d_h Density of humid air, lb/ft^3 (Kg/m^3)
h_a Enthalpy of dry air, Btu/lb (kJ/kg)
h_h Enthalpy of water vapor, Btu/lb (kJ/kg)
h_v Enthalpy of humid air, Btu/lb (kJ/kg)
h_w Enthalpy of liquid water present as vapor, Btu/lb (kJ/kg)
P Barometric pressure, in. Hg (kPa)
p_s Saturation vapor pressure at the dry-bulb temperature, in. Hg (Pa)
p_s' Saturation vapor pressure at the wet-bulb temperature, in. Hg (Pa)
p_v Vapor pressure of the humid air, in. Hg (Pa)
Δp Fan static pressure rise, in. water (Pa)
Q Quantity flow of humid air, cfm (m^3/s)
q Heat transfer, Btu/hr (kJ/h)
RH Relative humidity, %
S Sigma heat of humid air, Btu/lb (kJ/kg)
Sl Slope of the wet-bulb temperature change
T Absolute temperature, °F (°C)
t Dry-bulb (DB) temperature, °F (°C)
t' Wet-bulb (WB) temperature, °F (°C)
t'' Dew-point temperature, °F (°C)
t_{isen} Isentropic temperature, °F (°C)
V Air velocity, fpm (m/s)
v Specific volume of humid air, ft^3/lb (m^3/kg)
W Specific humidity, lb water vapor/lb dry air (g water vapor/g dry air)
η Fan efficiency, decimal or percentage

11.4.2 PSYCHROMETRY

Psychrometry is the study of air-water vapor mixtures under normal atmospheric conditions. The equations given in this chapter for calculation of psychrometric properties are only valid when English system units are used; in the SI system different coefficients are needed. For brevity and to minimize confusion, only the equations using English system units will be given. To work in SI units, simply make the calculations in English system units, then convert the solution to SI units.

11.4.2.1 Temperatures

There are four temperatures that characterize any air-water vapor mixture. These are the dry-bulb temperature, dew-point temperature, temperature of adiabatic saturation, and the wet-bulb temperature.

The *dry-bulb temperature*, normally understood by the term "temperature," is characteristic of the kinetic energy of the gas molecules; it is measured with a thermometer in the usual manner.

The *dew-point temperature* is the temperature at which the water vapor present in any given mixture of air and water vapor will condense. It is only dependent on the gas pressure of the water vapor. Addition of more water vapor will cause the dew-point temperature to rise. There would also be a dew point for, say, ethyl alcohol in a gas mixture containing this substance. However, since water is the only gas normally present that condenses in the normal temperature range, dew point always refers to water vapor.

The *temperature of adiabatic saturation* is the temperature at which an air-water vapor mixture will be saturated if water is added adiabatically (i.e., without extraneous heat) to the mixture and allowed to evaporate. The evaporating water has to get from somewhere the heat needed for evaporation, and the only available source is the mixture. There will be no addition of heat, but the temperature of the mixture will drop until saturation is reached.

The *wet-bulb temperature* is the temperature at which water evaporates adiabatically into an air-vapor mixture. Since the rate at which evaporating water removes heat from a mass of water is practically the same as the rate at which the air-vapor mixture adds heat to the mass of water, the wet-bulb temperature and temperature of adiabatic saturation of water into air are, for all practical purposes, the same.

It is important to remember that the wet-bulb temperature, or more correctly, temperature of adiabatic saturation, of any air-vapor mixture is dependent only on the heat content of the mixture and not on the water vapor present in the air. Perfectly dry air has a wet-bulb temperature, and the heat contents of two mixtures of dry air and vapor with the same wet-bulb temperature but with different dry-bulb temperatures are nearly identical, as is explained in the following.

11.4.2.2 Dry Air

In actuality, the substance called air consists mostly of empty space, with a gas molecule here and there. All the gaseous constituents of air except one are of such uniform properties at normal conditions and are of such constant proportion that they can be lumped together as if they were a single gas, and this uniform mixture of gases is called *air*. The constituent that does not follow the properties of the others, as its concentration varies widely and because it can condense at normal temperatures, is gaseous water, here referred to as "water vapor," although "steam" is more correct, as other gases can also be called vapor.

Dry air is air in the absence of vapor, and *saturated air* is air containing so much vapor that no more can exist; an attempt to add any more will result in condensation of the added vapor to liquid water. *Saturated air* is also a confusing term, as there is a tendency to think of dry air as soaking up water vapor, like a sponge. This is absurd—the vapor and other gases are as independent from each other as the oxygen is from the nitrogen. Condensation would occur at the same temperature for a given quantity of vapor even if no other gases at all were present. When air is at saturation, its temperature is the dew-point temperature. *Humid air* is air containing water vapor up to the saturation point.

11.4.2.3 Humid Air

Now any gas follows a temperature-pressure relationship to saturation; that is, for every temperature, there is a corresponding pressure of the gas at which condensation will occur. All the other gases in the air condense at such high pressures and at such low temperatures that there is no use to worry about them under normal conditions. Water vapor is the exception. When the temperature is reached that corresponds to the dew-point temperature for the pressure of water vapor present, condensation will occur. Actually, condensation can begin before this temperature is reached because of the effect of water condensing onto dust particles, but this effect is difficult to analyze and is normally ignored in psychrometry.

The only sure way of determining saturation vapor pressures at various temperatures is experimentally, and tables of saturation vapor pressures exist (Goff and Gratch, 1945). However, an empirical equation exists for calculating *saturation vapor pressure* at any temperature *t*. The formula is

$$p_s = 0.18079 \exp \frac{17.27t - 552.64}{t + 395.14} \tag{11.4.1}$$

which yields results correct to within less than 1/2% error of the values given in standard tables.

All the psychrometric properties of air can be calculated from wet- and dry-bulb temperatures, measured with a sling psychrometer or other instruments, and from the barometric pressure, which is normally measured with a portable aneroid barometer. The first property calculated is the saturation vapor pressure (p'_s) at the wet-bulb temperature (t') by substituting t' for t in Eq. 11.4.1. From this, the *actual vapor pressure* p_v of the air is calculated by

$$p_v = p'_s - \frac{(p - p'_s)(t - t')}{2800 - 1.3t'} \tag{11.4.2}$$

The density of the *dry* air, ignoring the vapor present, is

$$d_a = \frac{P - p_v}{0.754\,(t + 460)} \tag{11.4.3}$$

where d_a is in lb/ft^3 (kg/m^3).

The *density of the humid air,* including the vapor present, is

$$d_h = \frac{1.325\,(P - 0.378p_v)}{t + 460} \tag{11.4.4}$$

where d_h is also in lb/ft^3 (kg/m^3).

Sometimes one density is used, sometimes the other, depending on the process. In air conditioning problems, the dry-air density is normally used, as the humidity is included in the sigma heat, as shown below.

Specific humidity is the weight of vapor per weight of dry air and is given by

$$W = 0.622 \frac{p_v}{p - p_v} \tag{11.4.5}$$

It is measured in units of lb vapor/lb dry air (g vapor/g dry air).

Relative humidity is the ratio between actual vapor pressure and what the vapor pressure would be if the air were saturated at the dry-bulb temperature, or

$$RH = \frac{p_v}{p_s} \times 100 \tag{11.4.6}$$

where p_s is calculated from Eq. 11.4.1 but with the dry-bulb rather than wet-bulb temperature. Relative humidity is normally not used in calculations, but is the way of expressing humidity most understood by lay persons. It is expressed as a percentage.

Specific volume is the reciprocal of humid air density:

$$v = 1/d_h \tag{11.4.7}$$

where v is in units of ft^3/lb (m^3/kg).

Dew-point temperature can be measured or calculated from the vapor pressure in a rearrangement of Eq. 11.4.1

$$t'' = \frac{-395.14 \ln p_v - 1228.495}{\ln p_v - 15.5596} \tag{11.4.8}$$

11.4.2.4 Heat Content of Air

Enthalpy is the heat content of the air expressed in Btu/lb (kJ/kg) dry air. In air conditioning practice, it is common to use 0°F (also taken as 0°C; see below) as the base starting point for enthalpy of air and vapor, since air is usually above this temperature. Of course, there is actually always heat present above absolute zero.

Some authorities arbitrarily either add or subtract 10 Btu/lb to the dry-air enthalpy in English units, so tables and charts do not always agree on enthalpy values. In the SI system, it is common to use 0°C as the starting point to simplify the equations, so conversion of enthalpy values from one system to the other will not agree, either. However, the important quantity is the change in enthalpy in a heating or cooling process, not the actual enthalpy value at any one point.

When calculating the enthalpy of humid air, it is necessary to consider three components:

1. Since the specific heat of dry air is 0.24 at normal temperatures, the *enthalpy of dry air* is given by

$$h_a = 0.24t \tag{11.4.9}$$

2. *Enthalpy of water vapor* is a little more complicated, since water condenses in the normal temperature range, and many processes involve condensation and evaporation, so the latent heat of vaporization must be taken into account. The formula for the enthalpy of water vapor is

$$h_v = 0.45t + 1061 \tag{11.4.10}$$

in which 0.45 is the specific heat of vapor and 1061 the latent heat of vaporization.

3. The third component of enthalpy is the *heat of liquid water* present in the air as vapor between 0°F and the wet-bulb temperature of the air:

$$h_w = Wt' \tag{11.4.11}$$

Finally, since the usual manner of handling air conditioning calculations is to express everything in terms of pounds (grams) dry air, and the term W is specific humidity in terms of pounds vapor/pound dry air (grams/gram), the formula for the *enthalpy of humid air* (h_h) *is*

$$h_h = 0.24t + W(0.45t + 1061 + t') \qquad (11.4.12)$$

The enthalpy of the liquid water is frequently disregarded as an unnecessary complication; in heating, the term drops out anyway, and in refrigeration, the liquid water is usually removed as it condenses, so it is not necessary to cool the liquid water below its condensation temperature. When the enthalpy of the liquid water is ignored, Eq. 11.4.12 simplifies to

$$S = 0.24t + W(0.45t + 1061) \qquad (11.4.13)$$

From Eq. 11.4.13 the following,

$$c = 0.24 + 0.45W \qquad (11.4.14)$$

can be extracted. This value c is the *humid specific heat* of air, and may be used in heat-exchange calculations when no evaporation or condensation takes place.

The value S calculated by Eq. 11.4.13 is called the *sigma heat* of the air. There is a complication here in that many writers, especially in this country, use the term enthalpy when they really mean sigma heat. That is, they use Eq. 11.4.13 but call it enthalpy anyway. In Commonwealth countries, the practice of using the correct form of sigma heat is more judiciously followed. In this chapter, sigma heat is used.

One thing that is very important is that sigma heat depends only upon the wet-bulb of the air, not the dry-bulb temperature. The reason is that as water evaporates adiabatically into dry air, it draws its latent heat of vaporization from the air-vapor mixture, whose temperature, of course, drops. However, the sensible heat of the air has been replaced by an equal amount of latent heat. While the heat is still present as latent heat, the latent heat often does not have the same physiological effect as does the sensible heat.

Therefore, if Eq. 11.4.13 is applied to an air-vapor mixture into which water is evaporating at successive times as the water evaporates, the dry-bulb temperature will drop as the specific humidity W increases. It is this increase of W with decrease of t that causes the sigma heat to remain constant.

If the psychrometric formulas are applied to various dry-bulb temperatures at the same wet-bulb and barometric pressure, and the resultant W values from Eq. 11.4.5 used in Eq. 11.4.13 at different dry-bulb temperatures, the resulting sigma heat values should be the same.

However, solving Eq. 11.4.13 for various dry-bulb temperatures in this manner will not yield exactly the same sigma heats; the reason is that the values given here for specific heats of both dry air and vapor are not exact, as they vary with temperature. Substitution of the exact specific heats in Eq. 11.4.13 would yield identical sigma heats.

Another problem frequently encountered in understanding this concept of sigma heat varying only with the wet-bulb temperature is the confusion of wet-bulb temperature and dew-point temperature. For instance, if a given air-vapor mixture is heated, both the dry-bulb and the wet-bulb temperatures will increase, although not at the same rate; the dew-point will remain the same.

For derivations of the foregoing equations, see Carrier (1904), Goodman (1941), Goff and Gratch (1945), and Hartman (1982).

Example 11.4.1. Calculate all properties of an air-vapor mixture whose temperatures are 60°F (16°C) WB, 80°F (27°C) DB, at a barometric pressure of 30.0 in. Hg (101.6 kPa).

Solution. First, calculate the saturation vapor pressure at the wet-bulb temperature from Eq. 11.4.1:

$$p_s' = 0.18079 \exp \frac{17.27(60) - 552.64}{60 + 395.14}$$

$$p_s' = 0.5212 \text{ in. Hg } (1.765 \text{ kPa})$$

and from this value, calculate the vapor pressure from Eq. 11.4.2:

$$p_v = 0.5212 - \frac{(30 - 0.5212)(80 - 60)}{2800 - 1.3(60)}$$

$$p_v = 0.3046 \text{ in. Hg } (1.031 \text{ kPa})$$

The density of the dry air, from Eq. 11.4.3 is

$$d_a = \frac{30.0 - 0.3046}{0.754(80 + 460)}$$

$$d_a = 0.0729 \text{ lb/ft}^3 \ (1.16 \text{ kg/m}^3)$$

The density of the humid air, from Eq. 11.4.4 is

$$d_h = \frac{1.325\ (30 - 0.378)\ (.3046)}{80 + 460}$$

$$d_h = 0.0733 \text{ lb/ft}^3 \ (1.17 \text{ kg/m}_3)$$

From Eq. 11.4.5, specific humidity is

$$W = 0.622 \frac{0.3046}{30 - 0.3046}$$

$$W = 0.0064 \text{ lb vapor/lb dry air (g vapor/g dry air).}$$

To calculate relative humidity, Eq. 11.4.1 must first again be used to calculate saturation vapor pressure at the dry-bulb temperature:

$$p_s = 0.18079 \exp \frac{17.27(80) - 552.64}{80 + 395.14}$$

$$p_s = 1.0348 \text{ in. Hg } (3.505 \text{ kPa})$$

Then Eq. 11.4.6 is used:

$$RH = \frac{0.3046}{1.0348} \times 100$$

$$RH = 29.6\%$$

Specific volume from Eq. 11.4.7 is

$$v = \frac{1}{0.0733}$$

$$v = 13.6 \text{ ft}^3\text{/lb } (0.852 \text{ m}^3\text{/kg})$$

Dew-point temperature from Eq. 11.4.8 is

$$t'' = \frac{-395.14\ ln\ 0.3065 - 1228.495}{ln\ 0.3065 - 15.5596}$$

$$t'' = 45°F\ (8°C)$$

Enthalpy, from Eq. 11.4.12 is

$$h_h = (0.24 \times 80) + 0.0064\ [(0.45 \times 80) + 1061 + 60]$$

$$h_h = 26.60\ \text{Btu/lb}\ (6.38\ \text{kJ/kg})\ \text{dry air}$$

From Eq. 11.4.13, sigma heat is

$$S = (0.24 \times 80) + 0.0064[(0.45 \times 80) + 1061]$$

$$S = 26.22\ \text{Btu/lb}\ (6.29\ \text{kJ/kg})\ \text{dry air}$$

and from Eq. 11.4.14 the humid specific heat is

$$c = 0.24 + 0.45(0.0064)$$

$$c = 0.2429\ \text{Btu/lb dry air-°F}\ (0.082\ \text{kJ/kg-°C}).$$

11.4.2.5 Heat Transfer Equations

Heat transfer between two media cannot be calculated by simple energy balance equations, as various coefficients of transfer are also involved. However, simplified equations can be used for the illustrative purposes here, as a complete treatment of the subject is beyond the scope of this chapter. See a standard handbook for details (e.g., Kern, 1950; Whillier, 1982).

If initial and final sigma heats are known, the heat transferred to moving air is given by

$$q = 60Qd_a\,(S_2 - S_1) \qquad (11.4.15)$$

where q is in Btu/h (kJ/hr), the normal manner of expressing heat flow in the English system, Q is airflow in cfm (m^3/s), d_a is dry air density in lb/ft^3 (kg/m^3), and S_2 and S_1 are final and initial sigma heats, respectively.

If the initial and final air dry-bulb temperatures are known, a formula that can be used is

$$q = 60Qd_ac(t_2 - t_1) \qquad (11.4.16)$$

where t_2 and t_1 are final and initial dry-bulbs. While Eq. 11.4.15 can also be used for calculating cooling by reversal of the sigma heat terms, Eq. 11.4.16 can not because it does not take into account condensation of vapor.

11.4.2.6 Psychrometric Chart

Prior to the advent of computers and electronic calculators, the psychrometric chart (Fig. 11.4.1) was indispensable as an aid to psychrometric calculations, as the formulas were too awkward for mechanical calculators, and slide rules were not accurate enough. Presently, direct calculation with electronic calculators and computers has obviated the need of the psychrometric chart for making calculations, but it is still invaluable as a visual aid to understanding the processes of changes of state, as the processes can be plotted on the chart. A psychrometric chart is accurate at only one barometric pressure, and correction factors (or a sheaf of charts) must be used for applying the chart at other pressures.

11.4.3 HEAT IN MINING AND THE NEED FOR COOLING

The cooling mechanism used by the bodies of all mammals, humans included, is that of evaporation of sweat from the skin. Therefore, wet-bulb temperature is an important factor as it limits the temperature to which the body can be lowered. A relatively high dry-bulb temperature can be tolerated as long as the wet-bulb temperature is sufficiently low. Air velocity is also a factor as it aids in evaporation by carrying away the evaporating moisture. The "cooling breeze" felt on a hot summer day is this effect of the evaporating moisture being dissipated; the air temperature is usually the same as before the breeze came up.

11.4.3.1 Human Comfort Indices

Several different indices for determining human comfort under hot conditions have been proposed over the years. Probably the oldest in this country, and one still widely used, is the *effective temperature* (ET). A nomograph for determining ET is given in Fig. 11.4.2. This nomograph relates wet-bulb temperature, dry-bulb temperature, and air velocity. A graph showing the effect of increasing ET on worker productivity is given in Fig. 11.4.3. This graph has been criticized as having been determined with experiments using unacclimatized men and therefore being overly conservative.

The *wet-kata thermometer* index (Stewart, 1982) has been used in South African mines for many years, although it has been criticized because the kata thermometer is so small compared to the surface of the human body. South African authorities have developed the *specific coolingpower* index in recent years (Mitchell and Whillier, 1972), which is more scientific and which employs graphs developed from experiments with acclimatized miners.

The *wet-bulb globe thermometer* (WBGT) index (Brief, 1973) is widely used by government agencies in this country, but because of its taking into account radiant heat, normally not a significant heat source in mines, and because of the time and the difficulty in measurement of WBGT, this index has not been well accepted in the mining industry.

A good standard to use is to try to maintain effective temperature between 70°F (21°C) and 80°F (26.5°C). Between 80°F and 85°F (26.5°C and 30°C), some means should be taken to provide cooling, and 85°F (30°C) should be the maximum ET allowed before cooling is provided. Effective temperatures lower than 70°F (21°C) are of little additional benefit, and air below 60°F (16°C) ET actually becomes uncomfortably cold. In mine cooling situations, 70°F (21°C) ET is a good goal temperature to use for the air that is leaving cooling apparatuses.

11.4.3.2 Heat Sources in Mining

The following are common sources of heat in mining:

1. Initially hot surface air.
2. Adiabatic compression of downcast air.
3. Heat added to air by fans; this is both from compression and from bearing friction in the motor and fan bearings.
4. Heat from other machinery, especially diesel engines.
5. Heat from water; this is both sensible (if the water is hotter than the ventilation air) and latent, from evaporation of the water.
6. Heat from blasting.
7. Heat generated in the friction of falling rock.
8. Wall rock heat.
9. Heat from human metabolism.

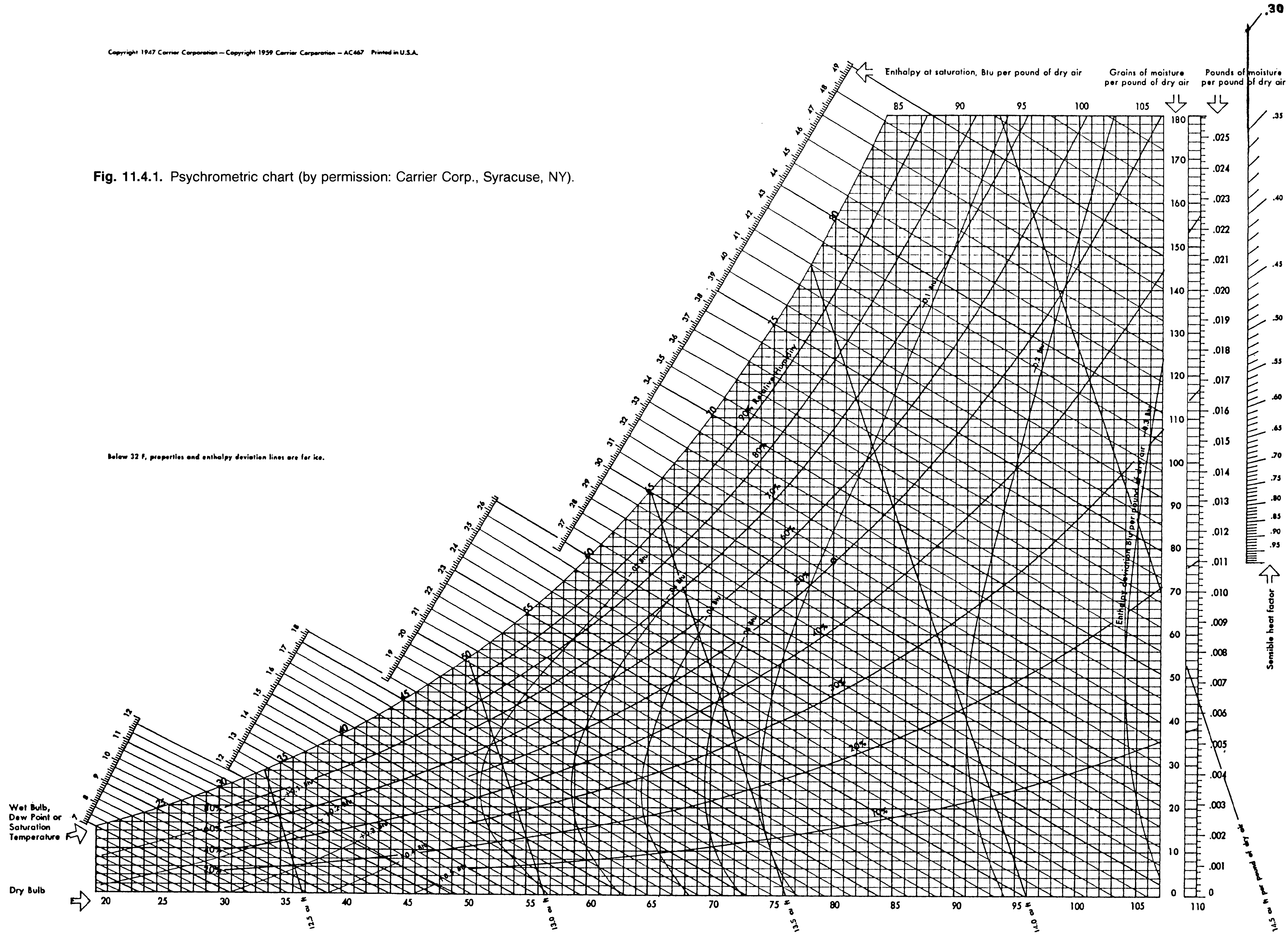

Fig. 11.4.1. Psychrometric chart (by permission: Carrier Corp., Syracuse, NY).

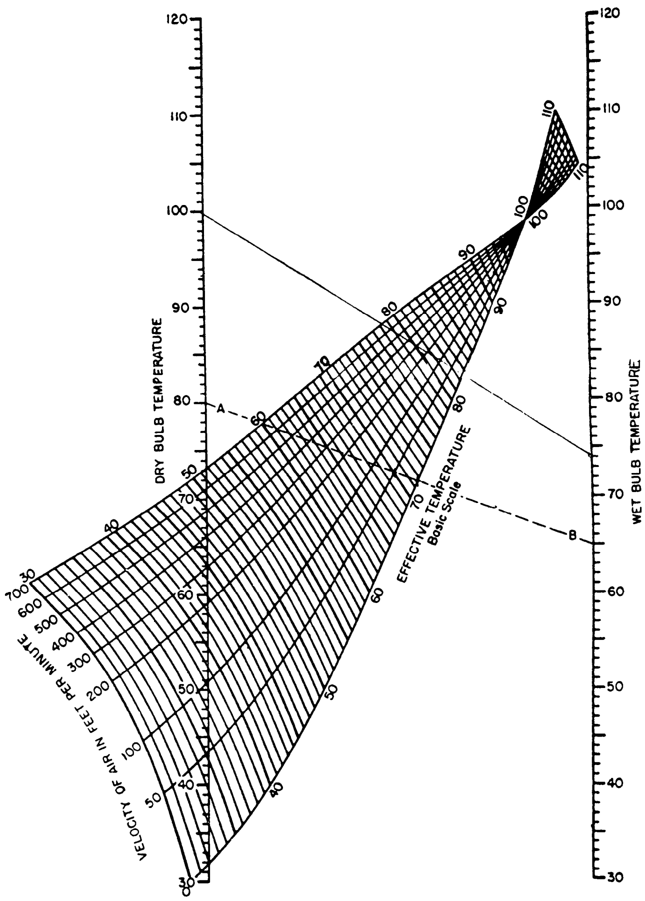

Fig. 11.4.2. Effective temperature chart at rest or doing light physical work in rooms heated by convection methods (Anon., 1988, Fig. 3.5; permission: ASHRAE).

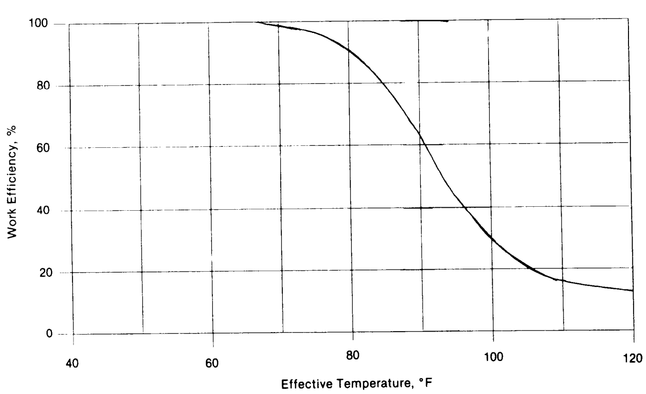

Fig. 11.4.3. Effect of effective temperature on work efficiency. (Carrier, 1950, p. 108; by permission: Penton Publishing Co., a division of Pittway Corp.)

10. Chemical heat, from oxidation of ore, curing of cement, etc.

Initially hot surface air: If this is the only significant heat problem, cooling is often a simple matter, as it can be done either as the air enters the shaft or on each level right off the shaft. However, if ground heat is present as well, the cooled air will absorb more heat than it would if it were carried nearer to the workings before being cooled.

Adiabatic compression of downcast air: This can be precisely calculated by the temperature-rise equation for adiabatic compression, but is usually estimated at 5.3°F/1000 ft (1°C/100 m) of shaft depth for the dry-bulb increase. Wet-bulb increase is taken to be half the dry-bulb increase. However, these figures are not completely accurate in practical situations due to the effect of shaft walls and steel absorbing heat during the hotter part of the day and then emanating heat during the cooler hours. No complete study of this effect is known to have been made; however, some empirical studies have been published (Vost, 1975).

Heat added by fans. Temperature rise in the air as it passes through a fan is considered to be an adiabatic process. Although there will be some heat added due to motor inefficiency and bearing friction, these sources are generally negligible. The procedure given here for estimating air temperature rise across a fan is from a paper by McPherson (1971).

1. Calculate the air horsepower of the fan at the anticipated quantity and fan pressure:

$$AHP = \frac{(Q)(\Delta p)}{6350} \tag{11.4.17}$$

2. From the power curve of the fan, determine the input brake horsepower at this operating point, and calculate the fan efficiency (or read the efficiency directly if the fan curve chart being used shows plotted efficiencies):

$$\eta = \frac{AHP}{BHP} \tag{11.4.18}$$

3. Calculate the isentropic temperature rise Δt_{isen} across the fan:

$$\Delta t_{isen} = 0.286 T_1 \times \frac{\Delta P}{13.6 P} \tag{11.4.19}$$

where T_1 is absolute temperature at fan inlet and P is barometric pressure of the air entering the fan in in. Hg (kPa).

4. Calculate the actual dry-bulb temperature rise from the isentropic temperature rise and fan efficiency:

$$\Delta t = \frac{\Delta t_{isen}}{\eta} \tag{11.4.20}$$

Wet-bulb temperature rise is estimated at half that of the dry-bulb temperature rise.

Heat from other machinery, especially diesel engines. All the energy used by machinery can be assumed to enter the ventilation air passing over the machinery. For diesel engines, 90% of the heat content of the fuel is assumed to enter the air as heat. For other machinery, such as electric motors, use the kW value of the machinery at full load.

Heat from water. Water evaporating can be assumed for estimation purposes to add 0.2°F dry-bulb/100 ft of drift length (0.4°C/100 m) in addition to the temperature rise caused by wall rock heat that is described below. If the water is very hot, an additional increase of 2°F/ 100 ft (3.6°C/100 m) up to a maximum temperature of 110°F (43°C) may be used. The wet-bulb temperature will be within 2°F (1.1°C) of the dry-bulb temperature at all times in very wet drifts (Johnson, 1988).

Heat from blasting. Blasting is generally done off shift, so much of the heat generated by the blast will have dissipated by the next shift, but a considerable amount will be retained in the muck pile and will be emanated during the mucking operation. Increase the predicted temperature in a working face by 2°F wet-bulb, 3°F dry-bulb (1°C wet-bulb, 1.7°C dry-bulb) temperature to allow for blasting heat.

Heat generated by the friction of falling rock. This is of some importance, especially in block cave mines. While no studies are known to have been made on this heat source, a rise of 1°F wet-bulb, 2°F dry-bulb temperature (0.5°C wet-bulb, 1°C dry-bulb) for a quantity of 6000 cfm (2.8 m^3/s) of ventilating air in an area where there is a good deal of falling rock can be used for a fair estimate.

Wall rock heat. The effect of this source is difficult to estimate accurately, as it varies not only with virgin rock temperature and rock type, but also with the time the area has been opened, size and shape of the opening, whether the area is wet or dry, the speed of the ventilating air, etc. There is far from universal agreement on the best way to estimate this effect, and the subject is currently being studied by various researchers. The method outlined by Johnson (1982) is fairly accurate although some of the assumptions are somewhat dated. Several computer programs are now commercially available that have been found to yield satisfactory results. This heat source is often the most important of all. For rough estimation purposes in hot rock mines, an increase of 0.5°F wet-bulb, 0.5°F dry-bulb per 100 ft of drift length (0.8°C wet-bulb, 1.7°C dry-bulb/ 100 m) is acceptable.

Heat from human metabolism. Unlike the air conditioning of buildings, this heat source is minor in mines due to the relatively large quantities of air being circulated, and can normally be ignored.

Chemical heat. This would have to be considered on an individual basis, but could be significant in mines where, for example, there is extensive oxidation of pyrite.

There are two errors frequently made in the estimation of mine heat loads for cooling system design:

1. The error of calculating the heat load as it would be done in a building, where air movement is very slight and most of the air is recirculated. In a mine, the air passes rapidly through the workings and heat added at, say, the end of the stope by an exhaust fan need not be added to the heat load of the stope, as the miners in the stope will not be affected by it. (Exception: If the air is being used in another stope, it is, of course, necessary to add it to the heat load of the other stope).

2. The error of grossly overestimating the cooling-system requirements by using intake air at the conditions of maximum heat load, say, at midsummer, for a condition that may only exist for a couple of weeks. Usually it is better to "tough it out" for the hotter season if this season does not last too long.

Mines contemplating the installation of major cooling facilities are advised to consult with mines that have such systems, or with knowledgeable consultants if there are no personnel on their own staffs with adequate background, as the planning of this type of installation requires a good deal of the "feel" of experience, at least at the present state of development of the art. In reality, most mines employing cooling systems use at least some empirical methods gained from experience for estimating their cooling needs (e.g., Lambrechts, 1950; Johnson, 1988).

Example 11.4.2. An air quantity of 100,000 cfm (47.2 m^3/s) at 60°F WB, 65°F DB (11°C WB, 18°C DB) downcasts a 1000-ft (300-m) deep shaft, and passes through a fan which develops 5 in. water (1250 Pa) of static pressure; fan inlet barometric pressure is 28 in. Hg (94.8 kPa), and efficiency of the fan is 75%. There is a 125-hp (93-kW) haulage motor operating on the level; this is a diesel motor that uses 8 gal (30.3 L) of fuel per hour; heat value of the fuel is 140,000 Btu/gal (38,780 kJ/L).

Air travels 600 ft (185 m) from the shaft and enters a stoping area, where the air is split among five stopes, each of which has a cross-sectional area of 12 × 15 ft (3.5 × 4.5 m). Heat emanating from each stope is 200,000 Btu/hr (210,800 kW). Calculate cooling requirements of cooling apparatus to be installed at the stoping area outlet so that the dry-bulb temperature of the air leaving the area will not exceed 80°F (21.5°C), as the air will be reused several times and it is desired to maintain it fairly cool.

Solution.

1. Adiabatic compression in the shaft:

$$\Delta t = 1000 \text{ ft} \times \frac{5°\text{F}}{1000 \text{ ft}} = 5°\text{F } (3°\text{C})$$

$$\Delta t' = \frac{5°\text{F}}{2} \simeq 2°\text{F } (1°\text{C})$$

Therefore, temperatures at the shaft station are

$$t = 65° + 5° = 70°\text{F } (21°\text{C})$$

$$t = 60° + 2° = 62°\text{F } (16.5°\text{C})$$

2. Fan heat.

$$AHP = \frac{(100{,}000)\ (5)}{6350} = 78 \text{ Hp } (58 \text{ kW})$$

Based on fan, h = 0.75,

$$t_{isen} = 0.286\ (70° + 460°) \times \frac{5}{(13.6)\ (28)}$$

$$\Delta t_{isen} = 2°\text{F } (1°\text{C})$$

$$\Delta t = \frac{2°}{0.75} = 2.7°\text{F, say } 3°\text{F } (1.5°\text{C})$$

$$\Delta t' = \frac{\Delta t}{2} = \frac{3°\text{F}}{2} = 1.5°\text{F, say } 2°\text{F } (1°\text{C})$$

Therefore, temperatures leaving the fan are

$$t = 70° + 3° = 73°\text{F } (23°\text{C})$$

$$t' = 62° + 2° = 64°\text{F } (18°\text{C})$$

3. Diesel haulage motor heat:

$$q = (0.9)\ (8)\ (140{,}000)$$
$$= 1{,}008{,}000 \text{ Btu/hr } (1{,}061{,}052 \text{ kJ/h}).$$

By rearrangement of Eq. 11.4.16,

$$\Delta t = \frac{q}{60\ Qd_a c}$$

$$\Delta t = \frac{1{,}008{,}000}{(60)\ (100{,}000)(0.075)(0.24)}$$

$$\Delta t = 9°\text{F } (5°\text{C})$$

$$\Delta t = \frac{9°}{2}\ 5°\text{F } (3°\text{C})$$

Therefore, temperatures after passing over the haulage motor are

$$t = 73° + 9° = 82°\text{F } (28°\text{C})$$
$$t' = 64° + 5° = 69°\text{F } (20.5°\text{C})$$

4. Wall rock heat transfer along the drift:

$$t = 600 \text{ ft} \times \frac{0.5°\text{F}}{100 \text{ ft}} = 3°\text{F } (2°\text{C})$$
$$t' = 600 \text{ ft} \times \frac{0.5°\text{F}}{100 \text{ ft}} = 3°\text{F } (2°\text{C})$$

Therefore, temperatures at the end of the drift entering the stoping area are

$$t = 80° + 3° = 85°\text{F } (29.5°\text{C})$$
$$t' = 69° + 3° = 72°\text{F } (22°\text{C})$$

5. Each stope uses 20,000 cfm (9.4 m^3/s). Barometric pressure can be taken as 28 in. Hg (94.8 kPa). From the psychrometric formulas above, d_a = 0.0667 lb/ft^3 (1.07 kg/m^3), c = 0.2465 Btu/lb-°F (1.85 kJ/kg-°C), S_1 = 35.96 Btu/lb (149.8 kJ/kg), q is 200,000 Btu/hr (210,800 kW), and by rearrangement of Eq. 11.4.16,

$$t_2 = t_1 + \frac{q}{60Qd_ac}$$
$$t_2 = 85° + \frac{200{,}000}{(60)\ (20{,}000)(0.0667)(0.2465)}$$
$$t_2 = 95.1°, \text{ say } 95°\text{F } (35°\text{C})$$

By rearrangement of Eq. 11.4.15,

$$S = \frac{q}{60Qd_a}$$
$$S = \frac{200{,}000}{(60)(20{,}000)(0.0667)}$$
$$S = 2.50 \text{ Btu/lb dry air } (10.41 \text{ kJ/kg})$$

At this point a method of estimating the final wet-bulb temperature will be given, as the actual wet-bulb cannot be calculated directly. However, McPherson (1983) has developed a method for calculation of the wet-bulb increase that utilizes the slope of the curve of the psychrometric chart. Since the slope varies over the chart, the slope must first be approximated at the original wet-bulb:

$$\frac{Sl}{\Delta t'} = 0.322 \exp(0.0237t_1' - 0.755)$$
$$= 0.322 \exp(0.0237\{72\} - 0.755) \qquad (11.4.21)$$
$$= 0.83$$

Rearrangement of Eq. 11.4.17 gives

$$\Delta t' = \frac{Sl}{1.03}$$
$$\Delta t' = \frac{2.50}{0.83} = 3.0°\text{F } (1.7°\text{C})$$
$$t'_2 = t'_1 + \Delta t'$$
$$t'_2 = 72° + 3° = 75°\text{F } (24°\text{C})$$

For large heat increases, the initial and final wet-bulb temperatures should be averaged and a second slope at this averaged wet-bulb calculated by Eq. 11.4.17. Then this slope should be used to compute a final t_2', which is more accurate. However, in this case the value of 75°F (24°C) is accurate enough. So the air leaving the stope will be 75°F WB (24°C WB), 95°F DB (35°C DB).

It is necessary to cool this air to 80°F DB (26.5°C). Normally in cooling, wet-bulb will decrease at about half the rate of dry-bulb, so calculate the needed drop in dry-bulb, and estimate the corresponding drop in wet-bulb:

$$\Delta t = 95°\text{F} - 80°\text{F} = 15°\text{F } (8.5°\text{C})$$
$$\Delta t' = \frac{15°\text{F}}{2} = 7.5°\text{F, say } 8°\text{F } (4.5°\text{C})$$
$$t' = 75° - 8°\text{F} = 67°\text{F } (19.5°\text{C})$$

Therefore, the cooling apparatus must cool 100,000 cfm (47.2 m^3/s) of air from 75°F wet-bulb, 95°F dry-bulb (24°C wet-bulb, 35°C dry-bulb) to 67°F wet-bulb, 80°F dry-bulb (19.5°C wet-bulb, 26.5°C dry-bulb). Assume barometric pressure to still be 28 in. Hg (94.8 kPa).

From the psychrometric formulas, calculate the entering and leaving sigma heat S_1 and S_2 at the cooling apparatus, and the dry air density entering the apparatus:

S_1 = 39.70 Btu/lb (165.4 kJ/kg), d_a = 0.0653 lb/ft^3 (1.06 kg/m^3) entering;

S_2 = 32.49 Btu/lb (135.4 kJ/kg) leaving.

From Eq. 11.4.15, rearranged for cooling calculations, calculate the cooling required at the cooling apparatus:

$$q = 60Qd_a(S_2 - S_1)$$
$$q = (60)(100{,}000)(0.0653)(39.70 - 32.49)$$
$$q = 2{,}824{,}878 \text{ Btu/hr } (823.9 \text{ kW})$$

or, since 1 ton refrigeration = 12,000 Btu/hr (3.5 kW),

q = 235.4 tons refrigeration

11.4.3.3 Mine Cooling Systems

The three techniques used to cool mine workings are

1. Increased air quantities.
2. Cooling of working areas with chilled service water.
3. Cooling of the ventilation air going to the workings.

Although alternative 1 is the first to be considered, it is frequently impractical. The air might already be at the maximum desired velocity anyway, and even if not, increase of air velocity might raise the air temperature because of additional fan heat.

Alternative 2 has met with a great deal of success in the South African mines, but it is not the panacea claimed by some authorities. It is necessary to maintain the cooling water below 60°F (15°C) as it enters the workings, and this means insulating the service water lines and frequently installing double lines so that the water can be kept circulating, in order that it not warm up while sitting in the pipes. The drawdown of service water is frequently so low that hours elapse between the time the water leaves the shaft and the time it reaches the face. Some mining systems consume so little service water that using very cold water does little good. However, where applicable, this approach can often be used to the exclusion of the much more complicated remaining alternative.

Alternative 3 is the most costly solution but is often the only practical one. Sometimes, cooling of the entire air mass is unnecessary; cooling of a portion of it and remixing with the uncooled air is best, as one of the principal problems in mine cooling is the passing of large amounts of air through cooling apparatuses.

Rejection of Mine Heat: It is of course necessary to reject the heat from the mine air to a heat sump somewhere. This can be done by

1. Rejection into mine drain water.
2. Rejection into upcast air.
3. Rejection into water circulating to the surface and back to underground.

Method 1 is the easiest to design, but often puts an intolerable additional burden on the mine dewatering system, and is costly because of the pumping power needed to lift the water out of the mine. It is only practical for small amounts of cooling, or for systems where drain water can be used as condenser water in cooling units located underground. Drain water is often so dirty that its use in condensers is impractical.

Method 2 is normally implemented through cooling towers built at the intakes to upcast shafts; the cooled water is then used in the condensers of underground refrigeration machinery. This system is still extensively used in the South African gold mines; it requires the availability of large amounts of unsaturated upcast air. No mine in this country has ever used the system on a large scale.

Method 3 is the most common method in most places, but is complicated by the water-handling systems needed, as is discussed below.

Ventilation air can be cooled at the surface or underground near the workings. Cooling at the surface, although the oldest system, was not commonly used for many years because of the poor positional efficiency, that is, the rewarming of air before it reached the areas where it was needed. However, this method is now finding new acceptance, as sometimes the rewarming effect is not significant. This is most true in areas where the main heat problem is caused by hot intake air.

Cooling of the Air Mass: This can be done in four ways:

1. By evaporative cooling of the air, if it is relatively dry.
2. By fin-and-tube cooling coils.
3. By spray chambers.
4. By direct expansion cooling coils working as part of a refrigeration apparatus.

Alternative 1 is restricted to very dry inlet air. Alternatives 2 and 3 are the most commonly used; alternative 4 is restricted to small local cooling units, commonly used in development headings.

Handling of Underground Cooling Water: There is a problem of high static head on water brought from the surface. This problem can be overcome by

1. Dumping the water into underground sumps that are then used as distribution points.
2. Reduction of the static head in turbines, which are then in turn used to generate part of the power needed to lift the water back out of the mine.
3. Using the water in the condensers of underground water cooling units, the chilled water from which is then circulated to air-cooling stations.
4. Using the water in the tubes of underground water-water heat exchangers; water circulating around the tubes of these exchangers, normally also in closed circuit, is then used in the air-cooling stations.
5. Using the water from the surface directly in air-cooling stations which are designed to handle high pressures.
6. Breaking and then recovering the head with a three-chamber pipe feeder system.

Alternative 1 is commonly used only for chilled service-water systems, where the water will be consumed anyway. It is uneconomical in loop-type systems because of the high cost of overcoming the static head to pump the water out of the mine.

Alternative 2 has been used in some South African mines, although there are often problems with balancing the flows into and out of the mine, as the system is normally used where the water to the air coolers is in open circuit.

Alternative 3 has been used in some German mines successfully. It, of course, necessitates ordering special machines equipped with condensers to handle the higher-pressure water. It also obviates the possibility of using evaporatively cooled water, as discussed in alternative (4) below.

Alternative 4 has been used in mines in the American Southwest, in the Soviet Union, and in South Africa. The problems here are the thermal limitations on heat transfer across the exchangers and the high pumping heads needed for the water circulating around the outside of the tubes (the “shell-side”) water. Water arriving at the surface can be cooled in cooling towers if conditions permit, in water chillers, or in towers first and then in water chillers.

Alternative 5 is known to have been used only in the Butte mines, and necessitates very high-pressure piping throughout the mine. It would be practical only where very few air-cooling stations near the shaft were needed.

Alternative (6) appears to be the overall best method (see Walker, 1986, for a description). This is a newly developed system, however, and operational data were not available at the time of this writing.

Use of Ice in Mine Cooling Systems: A recent South African development has been to transport ice underground in water slurries and then use it to cool water that is circulating in a closed loop underground to cooling stations. Heat exchange is initially being carried out by trickling the underground circuit water directly across an ice bed (Sheer, 1985).

If such a system proves practical, it would be a more propitious use of water, as it takes 6 Btu to melt a pound (14 kJ/kg) of ice, whereas only 1 Btu is absorbed in raising the temperature of a pound of water 1°F (1.3 kJ/kg-°C).

Other methods can sometimes be used in special instances, such as the utilization of cold stream water (where available) or an underground ice field in the summer that has been frozen the previous winter with cold winter downcast air (see Bossard and Stout, 1973, for a description of these and other possible systems).

11.4.4 CASE STUDIES

The cooling systems mentioned here are examples of various manners of effecting cooling; they are not necessarily “ideal” systems but have been proven in practice in the conditions for

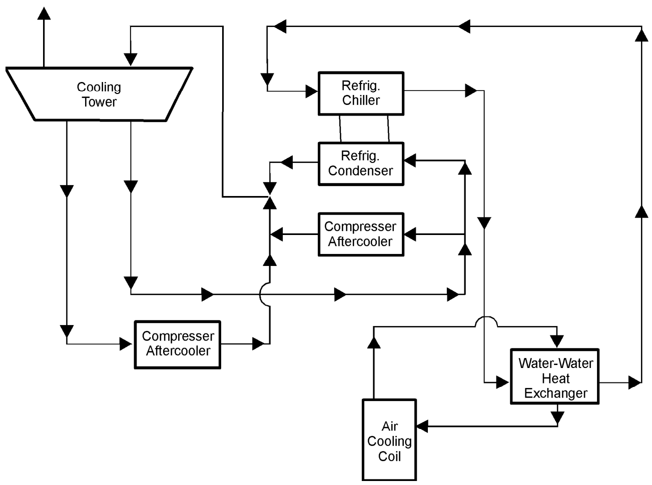

Fig. 11.4.4. San Manuel mine cooling system schematic.

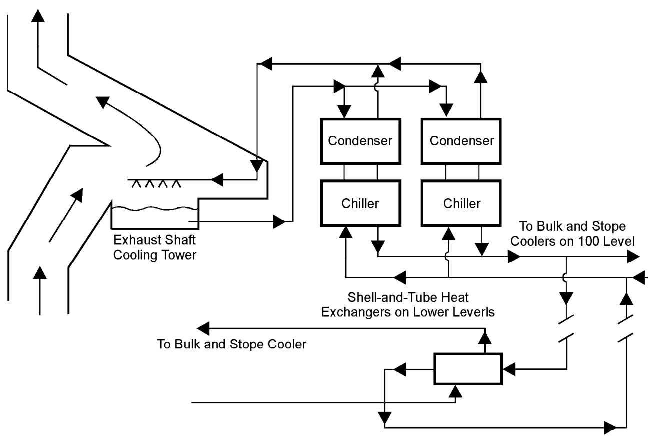

Fig. 11.4.5. Schematic of Western deep levels cooling system (Stroh, 1974).

their respective mines. Only general layouts are given with no specific figures on temperatures, cooling loads, etc.

11.4.4.1 San Manuel Mine, San Manuel, AZ

Main cooling is needed only during the summer months, although some development cooling with small air cooling units that have condensers utilizing service water is done year-round. The main cooling system consists of a cooling tower, surface water chillers, and underground water-water heat exchangers (Fig. 11.4.4). The heat exchangers can use water from the cooling tower directly, water from the surface water chillers that are in turn cooled by the cooling tower, or water cooled first in the tower, then in the water chillers.

Water circulating in the underground circuit through the heat exchangers is utilized principally in spray chambers; some is utilized in cooling coils near the working areas. Spray chambers are run as evaporative air-coolers, using only service water, in the early, dry part of the hot season.

11.4.2 Western Deep Levels, Johannesburg, South Africa

Water chilling units are located underground on several levels near exhaust shafts (Fig. 11.4.5). Condensers of the units are cooled by water circulating in closed circuit to cooling towers through which exhaust shaft air is routed. Chilled water is utilized in cooling coil stations and smaller, semiportable units at stope raises. Water for the deeper levels is run from a level where there are refrigeration units down through the tubes of water-water heat exchangers; shell water circulating through these exchangers is in turn circulated to air cooling stations.

The reference providing a description of this system is fairly old, and the system has since been somewhat modified.

11.4.3 Homestake Mine, Lead, SD

For a description of the mine's ventilation system, see Chapter 11.7. Cooling water is processed in water chillers on the 5000-ft (1.5-km) level for service in the deepest parts of the mine; the condensers are cooled by water that is dumped into the mine dewatering system. The chilled water is utilized in spray-chamber type coolers. Water from the spray chambers is rejected to the dewatering system. Some direct-expansion coil air chillers are also used; the condensers serving these units are cooled by service water that is rejected to the dewatering system.

11.4.4 Prosper Haniel, Germany

Air is cooled in fin-and-tube coils; the water for these coils is cooled in a series arrangement underground, first in water-water heat exchangers, then in evaporators of water chillers (Fig. 11.4.6). The water from the surface is run first through the tubes of the heat exchangers, then through high-pressure condensers serving the water chillers. Heat rejection on the surface occurs first through a cooling tower, then through the evaporators of water chillers; the condensers of these water chillers are cooled directly by surface air. However, in the winter, the tower and water chillers are unnecessary, as the water from underground can be cooled directly in fin-and-tube coils on the surface that utilize the cold air.

11.4.5 HEATING OF MINE AIR

In cold winter conditions, it is sometimes necessary to heat air entering the mine to avoid the problems caused by water freezing. This is a very different situation than cooling, as the problem is always caused only by cold surface air; underground conditions do not exacerbate the situation as they do in cooling situations.

Heating is generally best accomplished as the air enters the ventilation intakes from the surface. Usually the air is only heated to about 34°F (1°C).

11.4.5.1 Estimation of Heating Requirements

Eq. 11.4.15 or 11.4.16 can be used to calculate overall heating requirements, but manufacturers of the heating equipment being considered must be consulted for design of the actual system.

The system should be designed for average cold weather conditions, not for the maximum condition. For instance, if the average temperature is 0°F (– 18°C), but sometimes snaps down to – 40°F (– 40°C), a system designed for – 40°F (– 40°C) would be too large most of the time, and the needed equipment would be much more expensive than that needed for the average situation. In extreme cold conditions, it is better to reduce air quantities, or try to get along with the colder air for a short time.

11.4.5.2 Heating Systems

Cooling Tower. Sometimes, a cooling tower can be constructed over the exhaust shaft or portal; the tower extracts heat

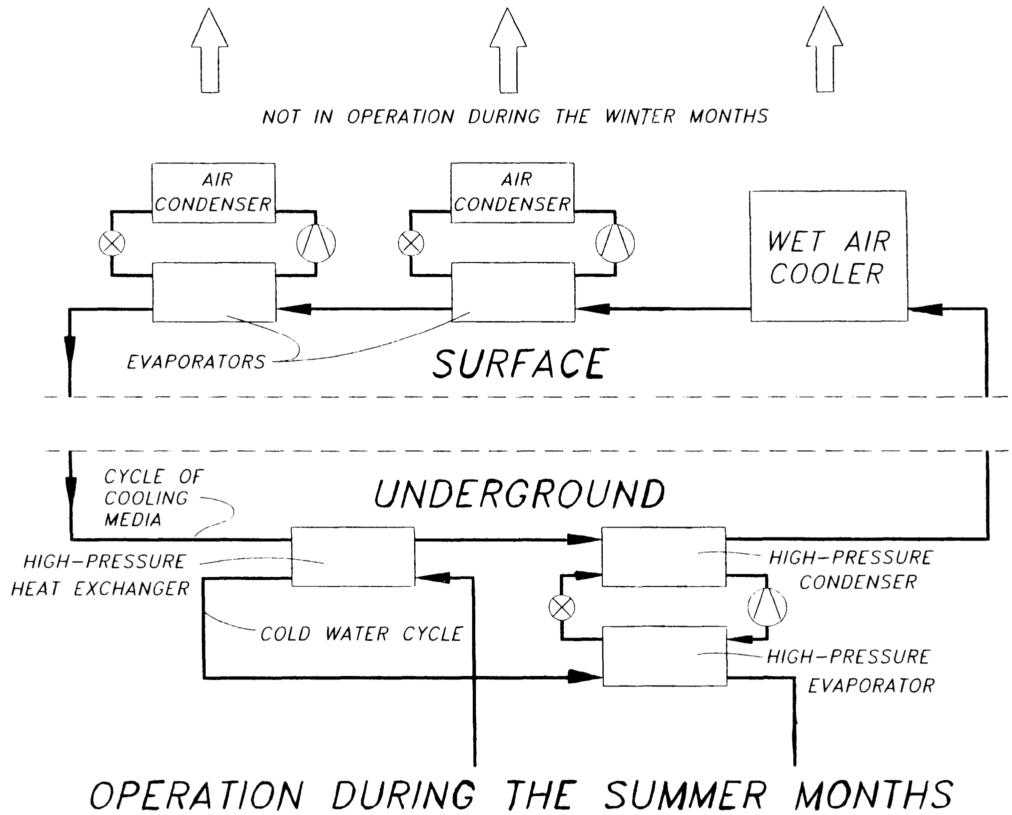

Fig. 11.4.6. Schematic of Prosper Hamel Colliery cooling system, German Federal Republic (Hamm, 1980, Fig. 10).

from the upcast air, and the warmed water is then pumped to heat exchanger coils at the fresh air inlet. This, of course, will only work if the upcast air is relatively warm to start with, about 70°F (21°C) dry-bulb. Since the water into the tower will be colder than the wet-bulb of the exhaust air, no unwanted cooling of the water will take place.

Waste Heat: If boilers are used in the area, the hot water from them may be circulated to heating coils at the mine inlet. Another possible heat source might be cooling water from compressors or generators that utilize diesel or gasoline engines. Sometimes, even geothermal energy may be used if the water from the mine is hot.

Indirect Heating with Furnaces: The furnaces used may be oil, gas, coal, or electric, depending on which is the cheapest power source in the area. If used in the indirect manner, whatever type furnace is utilized would heat water, or a glycol solution, and the solution would then be pumped in closed circuit through heating coils at the air inlet.

Direct Heating with Furnaces: In this system, the air is run directly through the furnace, as is the case in most buildings. Sometimes, the exhaust is even allowed to enter the airstream; this utilizes all the fuel consumed, as no heat is lost in exhaust gas. If the furnace is well maintained, no contamination problems should exist, as the amount of exhaust gas will be small compared to the amount of inlet air. However, this is not recommended, as potential disasters could occur if the furnace were to explode, or the air quantity suddenly drop drastically for some reason.

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Chapter 11.5
DIESEL EXHAUST CONTROL

ROBERT W. WAYTULONIS

11.5.1 INTRODUCTION

This chapter deals with the control of exhaust emissions from diesel-powered equipment operated in confined spaces, such as mines.

The operation of diesel engines releases both gaseous and particulate pollutants into the atmosphere. The control of these emissions is necessary to ensure a healthful work environment. Several components of the gaseous fraction of the emissions can be toxic, asphyxiating, or strongly irritating at low concentrations. The particulate matter in the exhaust is entirely respirable and contains adsorbed substances, some of which are known carcinogens. The concentration and composition of the pollutant fraction of diesel exhaust depends on many interrelated factors that include fuel composition, engine type and its state-of-maintenance, duty cycle, and the operating environment.

Statutory safety and health requirements for diesel-powered equipment in the United States are found in Title 30, Mineral Resources, of the *Code of Federal Regulations* (CFR) (Anon., 1988). The Mine Safety and Health Administration (MSHA) of the US Department of Labor has the responsibility for establishing and enforcing standards concerning basic machinery design, safety features, ventilation requirements, and other criteria intended to minimize health and safety risks associated with use of diesel equipment. The exhaust pollutants upon which limits are placed are carbon dioxide (CO_2), carbon monoxide (CO), oxides of nitrogen (NO_x), sulfur dioxide (SO_2), sulfuric acid (H_2SO_4), and aldehydes. Hydrocarbons (HC), oxides of sulfur (SO_x), and odor are not specifically regulated. Diesel particulate matter (DPM) is currently regulated indirectly by limiting worker exposure to respirable dust. Diesel legislation worldwide is summarized elsewhere (Waytulonis and Johnson, 1987).

Diesel exhaust emissions are controlled by proper selection of fuel and engine, good maintenance and work practices, and exhaust aftertreatment. Controls are discussed in 11.5.3.

11.5.2 BACKGROUND

11.5.2.1 Engine Operation and Emissions

OPERATION. In a diesel engine, fuel is sprayed under high pressure into air heated by compression; typical compression ratios range between 16 and 22:1. The fuel ignites spontaneously, further heats the compressed air, and forces the piston downwards doing work. The fuel injection process is critical to ignition, efficiency, and emissions characteristics. Good mixing between fuel and air is essential and must occur over a period of a few milliseconds. The air induction and fuel injection systems, therefore, must be designed and maintained to achieve optimum interaction. Injection timing, pressure, spray geometry, and delivery rate are critical to good performance (Anon., 1982; Taylor, 1979). Engine speed and power are controlled by varying the amount of fuel injected into the combustion chamber. The intake airflow is unthrottled, allowing the engine to operate over a wide range of fuel-air ratios.

EMISSIONS. Emission of exhaust pollutants is due to incomplete combustion of fuel. Their concentrations in the exhaust are primarily related to fuel composition, combustion efficiency, and the fuel-air (F:A) ratio.

Diesel fuel (by weight) consists of about 85 to 86% carbon, 13 to 14% hydrogen, 0.00 to 0.9% sulfur, and traces of other elements. Air consists of essentially oxygen and nitrogen. If complete combustion occurred, all the fuel carbon would combust to CO_2, all the hydrogen to water vapor (H_2O), and all the sulfur to SO_2. However, because combustion is never complete, the exhaust also contains CO, HC, and particulate matter. Because of high temperatures during combustion, a small portion of the excess oxygen and nitrogen from the air reacts to form nitric oxide (NO) and nitrogen dioxide (NO_2). Essentially all of the fuel sulfur can be expected to be present as oxides (SO_x in the exhaust. If the engine is in proper mechanical adjustment, these pollutants are contained in diesel exhaust only in low concentrations over the normal operating range. The number of possible reactions in a combustion chamber is very large, and therefore, the exhaust contains thousands of different compounds. For practical reasons, a limited number of species are measured in emissions studies (Springer and Patterson, 1973).

The ratio of the mass of fuel to the mass of air in the combustion chamber is expressed as the *F:A ratio.* The value of this ratio dominates combustion and emissions characteristics. At the stoichiometric F:A ratio, the mixture contains the amount of air required to burn the fuel to CO_2 and water, with neither an excess nor deficiency of oxygen (Taylor, 1979).

For engineering purposes, the stoichiometric F:A ratio for typical diesel fuels is 0.067 (indicated by the arrow on Fig. 11.5.1). Ratios less than 0.067 identify lean mixtures containing more air than required for complete combustion. Ratios greater than 0.067 are rich and contain less air than is needed. In the normal operating range where there is excess air for combustion, useful power increases nearly in direct proportion to the F:A ratio. This relation ends rather abruptly near stoichiometry. In the rich range of F:A ratios, power is nearly constant despite greatly increased fuel consumption. Thus, in diesel engines overall, F:A ratios are lean. This results in good fuel economy and performance, particularly during low load operation (Anon., 1982).

Typical relationships between F:A ratio and changes in exhaust composition are shown in Fig. 11.5.1. Note that the vertical scale is logarithmic. This figure shows that diesel engines operated in the abnormally rich range can produce unacceptably high concentrations of certain exhaust pollutants. F:A ratios greater than about 0.058 indicate improper engine operation. Engines are operated in this abnormally rich region only for the purpose of obtaining laboratory data. It is particularly important that diesel engines used in mines be adjusted to prevent operation near stochiometry.

About 1.2 lb (0.54 kg) of water are produced from each pound of fuel burned. The H_2O curve in Fig. 11.5.1 is calculated from hydrogen balance during combustion. The nitrogen (N_2) curve represents not only N_2 in the intake air, but also argon and other inert constituents. Nitrogen is not toxic and is always

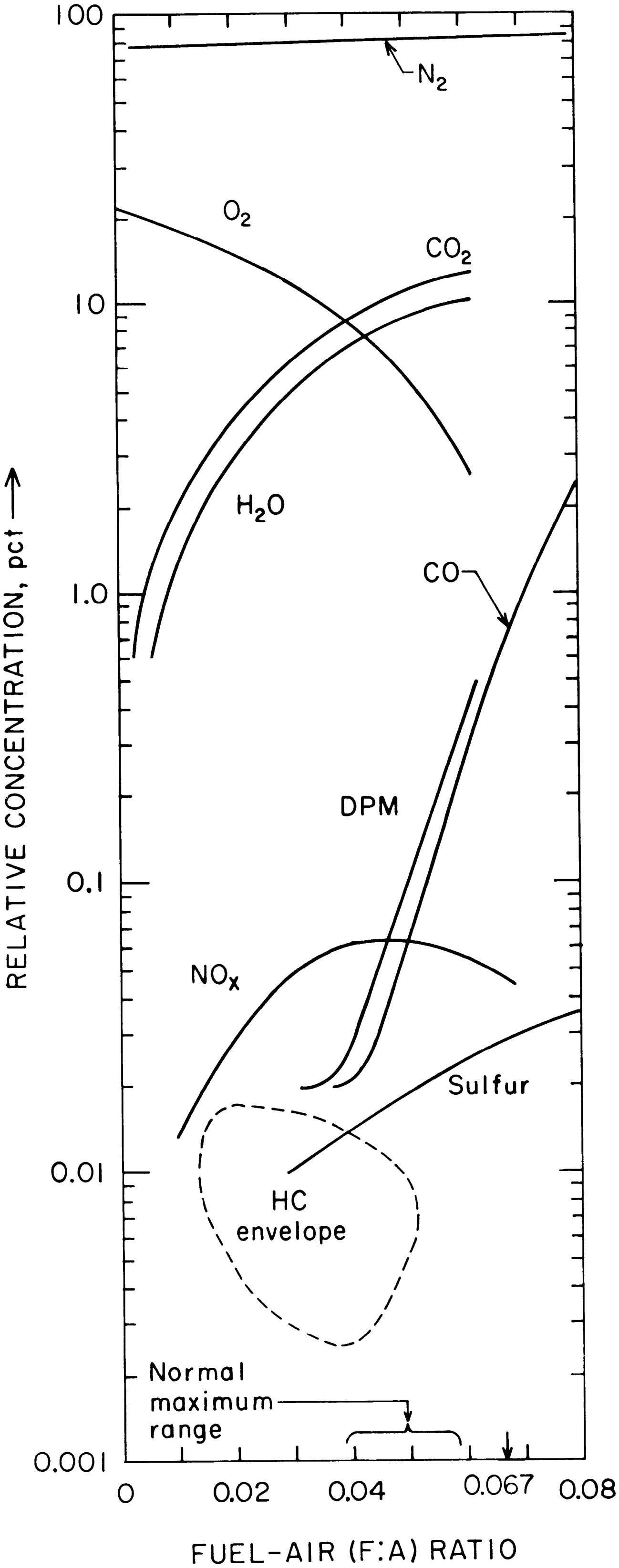

Fig. 11.5.1. Typical relationships of fuel-air ratio and relative concentration of diesel exhaust components.

present in exhaust gas in high concentrations. Although not shown, representative values of exhaust temperature are 260°F (127°C) for low F:A ratios to 1160°F (627°C) at stoichiometry.

Carbon Dioxide—Carbon dioxide is a non-poisonous, asphyxiating gas regarded as a hazard because of the large volumes produced and its tendency to layer at floor level. The emitted volume of CO_2 is directly related to the amount of fuel burned. The concentration of CO_2 increases almost directly with F:A ratio to a maximum at the stoichiometric F:A ratio.

Carbon Monoxide—Carbon monoxide results from incomplete combustion of carbon in the fuel. If sufficient oxygen is available, oxidation from CO to CO_2 will occur, but the reaction may be incomplete due to short residence time or low gas temperature (Watson and Janota, 1982).

Oxides of Nitrogen—Oxides of nitrogen (NO and NO_2) are always present in diesel exhaust gas, and concentrations of NO_x are generally greater in direct-injection (DI) engines than in equivalent indirect-injection (IDI) or divided-chamber engines (Holtz, 1960). The majority of NO_x formed in the combustion process is NO. Nitrogen dioxide is primarily a post-combustion product formed by oxidation of NO after the exhaust has been discharged (Marshall, 1975). As engine load and exhaust temperature increase, an IDI engine will typically emit up to 700 ppm NO_x. The concentration of NO_2 in raw exhaust is typically less than 10% of the total NO_x.

Oxides of Sulfur—Sulfur-containing species are emitted in diesel exhaust because sulfur is an impurity in the fuel. There are two sulfur oxides, SO_2 and sulfur trioxide (SO_3). The latter is normally collected as sulfuric acid since it reacts rapidly with water vapor in the exhaust. The gases SO_2 and SO_3 are strongly irritating at nontoxic concentrations and thus give considerable warning of their presence. Most fuel-bound sulfur is emitted as SO_2, less than 5% as sulfates (sulfuric acid and sulfate salts), and even smaller amounts as other organic sulfides (Anon., 1981). The sulfur curve in Fig. 11.5.1 is calculated for a fuel sulfur content of 0.5% by weight. It represents the total relative concentration of sulfur in the exhaust regardless of form: SO_2, sulfates, or other substances in gas, liquid, or solid phase.

Hydrocarbons— Some unburned and partially burned HCs are always present in diesel engine exhaust, both in the gas phase and adsorbed on DPM. The amount is usually small, but their presence is objectionable due to the odor of some species and their potential toxic or carcinogenic effects. They consist of some partially decomposed fuel molecules, some recombined intermediate compounds, and some HCs from the lubricating oil (Anon., 1981). Hydrocarbons in Fig. 11.5.1 are shown as carbon equivalent, measured by a flame ionization detector. Due to the great variability and absence of any trend, an envelope of realistic HC values is shown.

The major source of HC emissions in well-maintained engines results from fuel expansion and dripping from the injector sac volume (between the needle seat and injector holes) after the injector has closed (Watson and Janota, 1982). The sources of excessive HC emissions in poorly maintained engines includes misfires, poor injection, or raw fuel deposited on combustion chamber walls.

Aldehydes— Aldehydes are part of a broad group of partially oxidized HCs. Their concentrations in the exhaust are directly related to the unburned HC fraction.

Odor—Production of odorants is inherent in the diesel combustion process. There are two sources: (1) unburned fuel and its thermal breakdown products and (2) nitrogen compounds and other products of incomplete combustion (Marshall and Seizinger, 1978; Anon., 1982). Aldehydes contribute to odor, but there appears to be no single group of compounds that accounts for odor (Stewart, Dainty, and Mogan, 1976; Lilly, 1984). Since odor is not clearly defined, methods for reducing it are also not well defined. However, for a given engine, measures to reduce HC emissions also tend to reduce odor.

Diesel Particulate Matter—DPM has lately come under close scrutiny from a health effect viewpoint. For this reason, it will be discussed in more detail than the other diesel exhaust pollutants. Currently, there is no occupational exposure limit for DPM in the United States, but it may be regulated in the future due to the nature of the compounds it contains (Anon., 1988a, 1988b). Because DPM particles are predominantly less than 1 micrometer (μm) size, DPM is a component of respirable dust and thus indirectly regulated via compliance sampling for respirable dust (Cantrell, 1987).

Most particles result from the incomplete combustion of fuel and lubricating oil HCs. Engine oil contributes from 50 to 280 times as much material to the particulate emissions as does an equal amount of fuel consumed. So even though oil consumption is relatively small compared to that of the fuel, the oil can still have a significant effect on particulate emissions (Mayer, Lechman, and Hilden, 1980). Also leakage of fuel, hydraulic oil, etc., onto hot surfaces can add substantially to DPM in the atmosphere. Minor leakage rates on the order of 0.5 oz/hr (3.9 mg/s) of oil have been found to contribute about 0.5 to 2.5 mg/m^3 to the HCs in the air (Carlson, et al., 1986). Engine blowby (vapors that escape the combustion chamber into the crankcase) is a small source of DPM since the flow rate is typically about 1% of the air consumption rate of the engine. However, the concentration of HCs in crankcase vapors can be several times that found in the engine exhaust gases. Control systems are available to feed these gases to the intake system of the engine, where they are drawn into the combustion chamber (Adler et al., 1986). Typically in mining engines, these gases are released untreated into the atmosphere.

DPM is defined as the material, excluding water, that is collected on a filter after dilution of the exhaust with ambient air. Other terms related to DPM include smoke, particulates, and soot. For purposes of this discussion, *smoke* is defined as particles, either solid or liquid (aerosols), suspended in the exhaust gases, that obstruct, reflect, or scatter light. The term *particulates* refers to particles collected by filtration. *Soot* is defined as deposited DPM. The individual particles are carbonaceous solid chain aggregates with adsorbed or condensed organic compounds. These compounds include high-molecular-weight organic compounds such as unburned and oxygenated HCs, polyaromatic HCs (PAH), and inorganic species such as SO_2, NO_2, and sulfates (Khatri, Johnson, and Leddy, 1978; Vuk, Jones, and Johnson, 1976).

For discussion purposes, DPM can be subdivided into (1) non-gaseous oxides, (2) solid non-extractable carbonaceous material, and (3) a soluble organic fraction (SOF) (carbon-containing solvent-extractable material).

Non-gaseous Oxides—The oxide portion of DPM includes SO_4 and trace metal oxides (from lubrication oil and wear metals from engine components). In humid atmospheres, sulfates hydrate to form sulfuric acid and acid mists that can cause health and environmental hazards. Sulfates appear in diesel particulates as either acid droplets or more complex forms (Khatri, Johnson, and Leddy, 1978).

Carbonaceous Material—The carbonaceous portion of DPM results from incomplete combustion of fuel and lubrication oil, and is commonly called smoke. There are always some carbon particles present in the diesel exhaust under any operating conditions, and zero smoke emission is impossible (Taylor, 1979). Visible smoke is an indication of high DPM emissions. Diesel engines generate the highest DPM concentrations and visible smoke under high load and during cold-starting. Generally, smoke values above 4% opacity are visible to the human eye.

Smoke can be discussed in terms of observable colors. Diesel smoke is usually classified as either blue/white or grey/black in appearance. The blue component originates from an excess of lubricating oil in the combustion chamber. This results from deterioration of piston ring sealing, or valve-guide wear, and indicates a need for mechanical overhaul. However, unburned fuel can also appear as blue smoke if the droplet size is about 0.5 μm. The white component has a droplet size of about 1.3 μm and results from too low a temperature in the combustion chamber during fuel injection. White smoke can occur upon cold starting, at idle, and at low-loads. These conditions are made more obvious by low ambient temperatures and high altitude operation. Also white smoke can result from late fuel injection or low compression.

Grey/black smoke consists of solid particles of carbon from incomplete combustion. These particles are produced at or near full load if excess fuel is injected, or if the air intake is restricted. Main causes are poor maintenance of air filters and fuel injectors, or incorrect setting of the fuel injection pump (Lilly, 1984).

Soluble Organic Fraction—The *SOF* is the portion of the DPM that is extractable using a dichloromethane solution. The SOF originates from the fuel and lubrication oil. The soluble material is a complex mixture of raw fuel, cracked and partially oxidized fuel, and PAH. Because of their complexity, these mixtures must be fractionated before analysis (Anon., 1982).

The SOF can range from 2 to 70% by weight of the total DPM. Lubrication oil is the dominant source of the SOF. The remaining SOF is derived from fuel and typically makes up 10 to 30% of the total SOF on today's engines (Springer, 1988). The SOF portion of DPM contains mutagenic and carcinogenic substances, and it is important that any emission control technology under consideration be assessed with respect to its effects on the nature of the SOF (Anon., 1988b). Determination of the PAH gives an indication of the levels of known carcinogens. The Ames salmonella/microsomal test is the most popular short-term bioassay for measuring the activity of the SOF and its subfractions (Ames, McCann, and Yamasaki, 1975).

REGULATORY SCHEME. MSHA has the responsibility for establishing and enforcing mining regulations. CFR Title 30 describes requirements applicable to the emissions from diesel engines used in underground noncoal mines (Anon., 1988). Diesel-powered machines in gassy noncoal mines and tunnels are subject to several "parts" that are periodically revised. A part specifically for diesels in coal mines has not yet been established (adaptations of existing parts are used), but promulgation of such regulations have been recommended by an independent committee (Anon., 1988a).

MSHA generally uses the approved ventilation, methane, and respirable dust-control plans of individual mines to address some hazards of diesel use not specifically regulated. Some mine ventilation plans require minimum ventilation (volume flow) rates, ventilation requirements when multiple diesel units operate on the same ventilation current, environmental sampling, and specific work practices associated with the operation of diesel equipment. Air quality standards for some components of diesel exhaust are adopted from the American Conference of Governmental Industrial Hygienists (ACGIH) threshold limit values (TLV) (Anon., 1988c). These values are reviewed annually by ACGIH and revised as new research data warrant. MSHA regulations for coal and noncoal mines have cited different yearly issues, and thus compliance limits for substances may vary with mine type.

Dilution Ventilation—Ventilation is the principal means to control exhaust emissions. MSHA has established regulations limiting the concentrations of certain components in the raw exhaust and in mine air, and defining calculation methods to determine the "assigned" ventilating airflow which must pass over operating diesel machines. The rate of ventilation air is calculated from a combination of engine tests to determine the flow rate of exhaust gas, concentrations of pollutants, and their hygienically safe levels. From test data, the volume of fresh air to dilute the exhaust gas to a safe condition for the exposure of workers is calculated. When ventilation is calculated using a safety factor, concentrations of the exhaust pollutants will not exceed their limits (Holtz, 1960).

Currently, MSHA determines the maximum engine generation rate of the worst single gaseous pollutant, and the volume flow (quantity) of air to dilute that component to a safe level is calculated. Frequently, safe ventilation is based on the concentrations of NO_x in the exhaust; however, the specific gas varies for different engines. Sometimes maximum ventilation is based on either CO_2 or CO. DPM is not presently considered in measurements and calculation of safe ventilation.

Emission levels differ among engines and with different conditions of operation. The maximum exhaust dilution requirement does not always occur at maximum engine speed and load (Marshall and Hurn, 1968). Some time ago, 75 cfm/max hp (0.047 m^3/s/max kW) was suggested as enough ventilation for any diesel engine used underground. Experience has since indicated that safe dilution may range from about 90 to 250 cfm/max hp (0.057 to 0.158 m^3/s/max kW), provided the engine is adjusted to minimize CO (Holtz, 1960; Anon., 1985, 1985a). Because of these factors, no accurate empirical formula has been developed for estimating safe dilution ventilation for different types of diesel engines. For these reasons, no diesel engine should be placed in underground service until tests have been made to determine the safe ventilation rate. Both the maximum fuel injection rates at various altitudes and a safe ventilation rate are given in the approval or certification letter for each MSHA-approved diesel machine.

11.5.3 CONTROL OF EXHAUST EMISSIONS

11.5.3.1 Work Practices

Diesel engines were first used underground in the United States around 1937, mostly to drive locomotives. Guidelines for the safe use of diesel-powered equipment were subsequently developed. Results of research and practical experience have identified important work practices.

FUEL SELECTION. The American Society for Testing and Materials specifications for diesel fuels, ASTM D-975, is widely accepted. Number 1 diesel fuel is primarily produced for circumstances in which No. 2 causes cold-weather handling and engine starting problems. Some burner fuels meeting ASTM D-396 are also used in diesel operation (Anon., 1988d).

Fuel composition affects diesel emissions, and fuels with different properties do not necessarily respond the same way in engines with different combustion systems. Using fuel composition as an emission control technique will not significantly improve all diesel engine exhaust characteristics. Reasonable control of diesel fuel properties will, however, minimize operational, cold-start, and durability problems, along with some emission problems (primarily SO_x, HC, and DPM) (Ryan et al., 1981; Baranescu, 1988; Weidmann et al., 1988). Properties of diesel fuel of particular interest are cetane number, volatility, sulfur content, and hydrocarbon types.

The cetane number, sometimes estimated by the cetane index, describes the ignitability (ignition delay) of diesel fuel. Fuels with high cetane numbers have low self-ignition temperatures. Fuels with low cetane numbers cause engine roughness, but cetane numbers higher than the minimum requirement for a particular engine will not measurably improve engine performance. Cetane numbers range from 0 to 100; typical cetane numbers of diesel fuels used in the United States range from 40 to 57. Although the terms cetane number and cetane index both describe ignitability, they are sometimes erroneously used interchangeably. A cetane index can be calculated by a four-variable equation with density and volatility data from the particular fuel analysis (Dickson and Woodward, 1988).

Table 11.5.1. Optimum Property Limits for Diesel Fuel

Property	Limit
Cetane number	> 48
Aromatics	< 20%
90% dist. temp.	< 600°F (316°C)
Sulfur	< 0.10% by mass

Increased cetane number and volatility reduces the tendency to produce white smoke. Using a fuel with a cetane number greater than 48 and a seasonably adjusted cloud point reduces cold-start HC emissions, noise, odor, irritant, and fuel system wax separation problems.

Volatility is determined by distillation of a fuel at a prescribed rate, with observations of temperature vs. volume of distillate and amount of residue. The 10 and 90% distillation temperatures are frequently cited to give a concise indication of fuel volatility.

The most important emission-related fuel properties for controlling DPM are sulfur content and aromatic HC content. The ASTM standard for No. 2 diesel fuel sets a 0.5% by weight limit on sulfur content; many states have more restrictive standards. Reducing fuel sulfur reduces SO_2 emissions and the sulfate fraction of DPM. Reducing sulfur content of diesel fuels has the added benefit of reducing corrosive wear and oil fouling and extending engine life.

The HCs in diesel fuel are numerous, but mostly fall into three families: paraffins, naphthenes, and aromatics. Each series has its own molecular characteristics, and this has a strong influence on diesel fuel quality (Marshall and Hurn, 1968; Ryan et al., 1981; Weidmann et al., 1988). An important but usually unspecified property of diesel fuel is aromatic HC content. Typical commercial No. 2 diesel fuel has an aromatic HC content of 20 to 40% by weight. Reducing fuel aromatic content reduces HC and the carbonaceous fraction of DPM (Weaver et al., 1986).

Increasing aromatic content and the 90% distillation temperature will increase DPM and odor emissions (Lilly, 1984). However, some data indicate that changes in the distillation temperature have little potential for DPM emissions reduction (Weaver et al., 1986). These conflicting findings highlight the difficulties of isolating fuel effects on emissions when a number of fuel properties are interrelated.

High-quality fuels with low sulfur levels, low aromatic content, increased volatility, and cetane numbers greater than 48 have generally resulted in lower particulate emission levels. Optimum property limits for diesel fuel are shown in Table 11.5.1.

Fuel Additives—Fuel additives for smoke control modify ignition delay, catalyze the combustion of carbon, or improve the fuel-air mixing processes by assisting fuel atomization. Small quantities of additives, typically compounds of the alkaline earths, barium, and calcium, can reduce smoke opacity. These additives not only catalyze the conversion of carbon particles to CO_2, they also aid in the suppression of free carbon formation during the early stages of the combustion process. As a result, carbonaceous particles are replaced with emissions of oxides and sulfates of barium and calcium, and although smoke opacity is reduced, total particulate emissions are frequently increased (Kittleson et al., 1979).

The effectiveness of additives is load-dependent and variable. Using additives resulted in changes ranging from a 25% reduction to a 48% increase in DPM mass at steady-state operation (Roessler et al., 1980). A barium additive tested at the manufacturer's recommended concentration and at light engine loads, elevated DPM emissions by about 30%. The additive also produced statistically significant increases in both exhaust mutagenicity and PAH concentrations. In contrast, the same additive used at one-fourth the recommended level decreased DPM emissions at full load up to 74%. These results suggest that the barium additive should not be used for smoke suppression under light-duty operation, if at all (Zeller, 1987; Draper, Phillips, and Zeller, 1988).

Overall, experience with fuel additives for combustion modification and emission control is limited. The cost of additives as well as the fate and toxicity of their exhaust end products must be considered.

Commercial cetane improvers are available for diesel fuels and are mostly composed of mixtures of primary amyl nitrates or primary hexyl nitrates, which in concentrations near 1000 ppm in the fuel provide an increase of about four cetane numbers. Organic peroxides are also known to act as cetane improvers (Lilly, 1984). However, sensitivity to a cetane improver is fuel-dependent and generally causes particulate emissions to increase (Burley and Rosebrock, 1979).

Diesel fuel is sometimes modified with kerosene or gasoline to improve engine starting in cold weather. This modification can possibly lower the fuel flash point. Fuel flash point is an important parameter for evaluating fire hazards. Number 1 and 2 diesel fuels have minimum permissible flash points of 100°F (37.8°C) and 125°F (51.7°C), respectively. Users are advised that fuel additives to improve starting may result in lowering the flash point to less than 100°F (37.8°C). A gasoline content of only a few percent, for example, can reduce the flash point of diesel fuel to room temperature range, thus allowing the formation of explosive mixtures above the fuel in the tank (Adler et al., 1986). As a result, the fuel must be reclassified from a combustible liquid to a flammable liquid (Anon., 1988e).

ENGINE SELECTION. There are, as identified in 11.5.2.1, two types of diesel engine combustion systems, DI and IDI. In the DI system, the fuel is injected directly into a combustion chamber. The DI engine has about 10% lower fuel consumption, is less expensive, and offers a slightly better power-to-weight ratio than the IDI engines (Anon., 1982). In the IDI diesel engine, there is an additional chamber into which the fuel is injected. This prechamber is designed so that the intake air swirls as the piston rises. The air motion in the IDI combustion chamber is more violent than the movement in the DI type and promotes rapid fuel-air mixing for more complete combustion.

Comparison of emissions between DI and IDI diesel engines shows that the IDI engine produces the lowest CO, NO, HC, and aldehydes, but odor levels are not greatly different (Marshall and Fleming, 1971). The IDI diesel engine is one of the lowest emission internal-combustion engines available, and this has long been recognized in the underground mining industry.

ENGINE DERATING. Limiting engine power output by adjusting the maximum fuel rate, *derating,* reduces emission of pollutants. Reducing the maximum volume of fuel that can be injected makes excess oxygen available for combustion, limiting products of incomplete combustion. Combustion products are a sensitive function of the amount of excess oxygen, as shown in Fig. 11.5.1. *Derating is not a method to reduce excessive exhaust pollutants caused by deficient engine maintenance.* Derating requires the maximum fuel injection rate to be correctly adjusted and sealed. Consequently, *tampering with the fuel adjustment must not be tolerated.*

EFFECTS OF INTAKE AIR COMPOSITION. In mines using diesels, the atmosphere always will contain some exhaust gas. Blasting gases are similar in composition and at times may occur in relatively high concentrations in some locations. Naturally occurring methane may also be present. The presence of such contaminants, in addition to barometric conditions, will change the characteristics of the exhaust.

Operation in Gassy Atmospheres—Addition of methane to the engine intake air has the effect of supplying extra fuel. Thus, even with the fuel pump adjusted to operate in the normal range of F:A ratios, the natural gas can increase the F:A ratios into the rich range, increasing some pollutants in the exhaust. For example, 0.5% by volume (0.28% by mass) of methane in the intake air would have the effect of changing a F:A of 0.048 to 0.051.

Operation Without Ventilation—Diesel machines should never operate in areas with little or no ventilation. For example, operating a 50-hp (37.3-kW) diesel engine at full load in a 10 by 10 by 50-ft (3 by 3 by 15-m) entry without ventilation produces dangerous levels of CO_2 and depletes oxygen in a matter of minutes (Holtz, 1960). If such operation were continued, the concentration of CO would rapidly accumulate to dangerous levels. *All diesel equipment should be shut down immediately when positive ventilation is interrupted.*

When an engine rebreathes its own exhaust, the F:A ratio increases, which increases the CO and DPM emissions. Rebreathing can occur where ventilation is locally inadequate. Prolonged operation of diesel equipment in such an area causes hazardous conditions that deteriorate rapidly, particularly if the engine is operating at a high load. The effect of an engine rebreathing its exhaust on the composition of exhaust can be nearly the same as when a diesel engine is operated in normal air at F:A ratios in the rich range. The effect is countered by passing additional ventilation air over the diesel at a rate adequate to maintain the atmosphere hygienically safe (Berger et al., 1940).

Operation at Altitude—The effect of altitude on emissions is important when engines are operated in mountainous regions. The fuel rate must be decreased, as air density decreases, to maintain the maximum allowable F:A ratio. In naturally aspirated engines, a fixed maximum fuel rate setting is valid only over a moderate range of altitude if excessive emissions of CO and DPM are to be avoided.

The safe maximum fuel rate can be calculated for higher altitudes using CO emission data taken at any other barometric pressure. For example, the nominal fuel rate at full throttle for an engine at 1000 rpm is 12 lb/hr (5.4 kg/h) at 30 in. Hg (101.6 kPa) barometric pressure. For these conditions, the F:A ratio is about 0.054, and the concentration of CO in the exhaust is 0.09%. At a barometric pressure of 25 in. Hg (84.7 kPa), the maximum fuel rate should be reduced to 12(25/30) = 10 lb/hr (4.5 kg/h). This reduction would maintain the maximum F:A ratio at 0.054 and the concentration of CO at 0.09% in the exhaust. However, the maximum power output of the engine is reduced from 28 to 22 hp (20.9 to 16.4 kW) because of the reduced maximum fuel injection rate. When the engine is adjusted for the effects of barometric pressure in this way, the same volume of dilution air will provide safe ventilation at any altitude (Elliott, 1948; Holtz, 1960).

Fig. 11.5.2 shows how F:A ratio would change if an engine were not derated for altitude. For example, an engine adjusted for a F:A of 0.05 at sea level would operate with a F:A ratio of

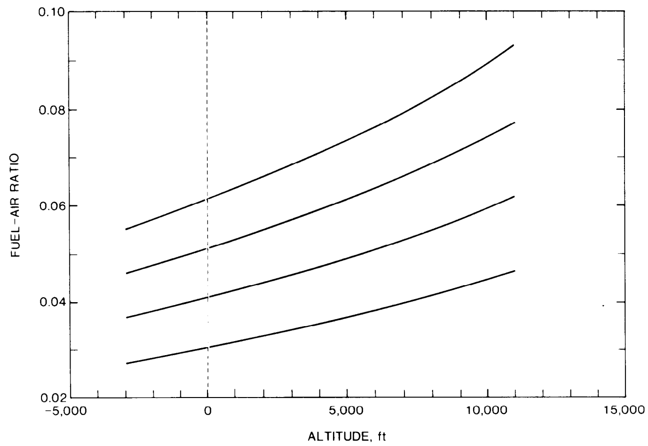

Fig. 11.5.2. Influence of altitude on fuel-air ratio for several constant fuel rate settings. Conversion factor: 1 ft = 0.3048 m.

about 0.07 at 9000 ft (2743 m) altitude. Typical exhaust emission levels for the uncompensated fueling condition from Fig. 11.5.1 indicate a significant increase in CO and DPM.

Supercharging (pressurizing the intake air) can compensate for loss of power at high altitude, and thus the maximum setting of the fuel injection pump can be increased (Watson and Janota, 1982). This increase in fuel adjustment results in more power while combustion conditions remain within the normal range of F:A ratios. Increasing the pressure of the engine intake air increases the oxygen available for combustion. The maximum fuel rates can be increased accordingly to yield increased maximum power at lower altitudes, or recovery of power lost from derating, at higher altitudes.

ENGINE MAINTENANCE. The objective of diesel engine maintenance is to keep engines in an optimum operating condition and extend useful engine life. Preventive maintenance, periodic repairs, and adjustments are all part of a basic maintenance program. Poor maintenance will result in increases in exhaust pollutants.

Table 11.5.2 shows the effects of faults or maladjustments on HC, CO, NO_x, and DPM emissions of an IDI engine, with the increase above baseline given in percent. For convenience, the percentage increases are given in groups (i.e., < 50, 50 to 200, and ≥ 200). These faults change the engine's F:A ratio and may result in increased pollutants in the exhaust. Also the level of emission caused by a pair of faults occurring individually is not as severe as the level when the same faults are combined (Branstetter, Burrahm, and Dietzmann, 1983; Waytulonis, 1985).

Of all the potential causes of excessive exhaust pollutants, overfueling is the most critical. Overfueling can result from improper injection-pump setting or improper altitude adjustment, worn injection system components, or improper injectors. For example, an engine adjusted for sea level, but operating at 4000 ft (12 19 m) is overfueled by 10%, and if operating at 7000 ft (2134 m), it is overfueled by 20%. Overfueling by 10% can increase CO by 50 to 200%, and 20% overfueling increases CO by more than 200%. Once set correctly, the fuel-injection setting does not require frequent adjustment.

Intake restriction, caused by a plugged intake air filter, will cause dramatic increases in CO and DPM. For example, 20% overfueling combined with 50 in. water (12.4 kPa) pressure drop across the air cleaner results in a ≥ 200% increase in CO and DPM. Proper and timely air cleaner maintenance can prevent such severe conditions.

Fuel injection timing increases CO at both an advanced or retarded position from the correct setting, and has mixed effects on HC and DPM. Timing advance is the primary engine fault that can lead to increased NO_x, while retarded timing will decrease it. Once properly set, injection timing does not require frequent adjustment.

Many premature engine failures are a result of ingestion of dust, fuel contamination, and cooling system irregularities. Internal engine parts must be kept free of dust to prevent accelerated wear of moving parts that have very close tolerances. Dual, paper-element air filters provide good protection. The inner (backup) filter element protects the engine against failure of the outer cartridge. Air filter cleaning intervals are difficult to predict, but a vacuum gage can effectively indicate excessive restriction. Reuse of filter cartridges is not recommended, nor is the practice of backflushing them with compressed air.

Using clean diesel fuel will contribute toward long engine life. Both dust and water in fuel will lead indirectly to higher pollutants in the exhaust by accelerating wear. Water collects in storage tanks and if allowed to enter the engine, will damage fuel injectors and pumps. Water and suspended matter can be removed from the fuel by filters and water separators. Minimizing fuel transfers and keeping storage tanks full helps to prevent contamination.

Overheating is a common cause of engine failure. A combination of several engine faults can result in overburdening the cooling system and cause permanent damage to the engine. Typically, overfueling and retarded timing will increase the exhaust-gas temperature, thus increasing the amount of heat rejected into the cooling system. Minerals in mine water form deposits on cooling surfaces and act as heat insulators. Damaged fans and shrouds reduce airflow through the radiator and over the engine, and slipping belts cause fans or coolant pumps to perform poorly. Proper and timely preventative maintenance can minimize these problems.

11.5.3.2 Exhaust Aftertreatment

The most important pollutants to control in confined spaces are probably CO, NO_2, and DPM. Exposure to high concentrations of CO and NO_2 eventually are fatal. DPM is important because of the known carcinogens adsorbed on exhaust particles. Discussion of exhaust aftertreatment therefore centers on these pollutants.

WATERBATH EXHAUST CONDITIONERS. These are also referred to as *water scrubbers* and are designed to cool the exhaust and to suppress flames and sparks in the exhaust. These safety functions are mandatory on diesel equipment operated in certain areas of gassy mines (Anon., 1988). A properly designed and maintained waterbath conditioner will not allow the exhaust temperature to exceed 170°F (77°C). Waterbath conditioners use the principle of adiabatic saturation of the hot exhaust by evaporation of water. Approximately 1100 Btu (1161 kJ) are required to vaporize 1 lb (0.45 kg) of water. The hot exhaust gas is bubbled through water to obtain the contact area necessary for rapid evaporation of the water and cooling of the exhaust. Under some conditions, a waterbath exhaust conditioner may create fog that can reduce visibility. About 3 lb (1.36 kg) of water are typically emitted/lb (0.45 kg) of fuel burned.

Waterbath exhaust conditioners may be batch- or a constant-level type. Batch-type conditioners contain all their operating water in one "batch" and operate with a variable level. Constant-level conditioners maintain a set water level controlled by a float valve or some other device. Water is supplied from an external

Table 11.5.2. Effects of Engine Faults on HC, CO, NO_x, and DPM Emissions, Increase Above Baseline in Percent

Fault description	HC	CO	NO_x	DPM[a]
Intake restriction:				
25 in. H_2O	<50	<50	<50	<50
50 in. H_2O	<50	<50	<50	50–200
Exhaust restriction:				
3 in. Hg.	<50	<50	<50	<50
6 in. Hg.	<50	<50	<50	<50
Timing change:[b]				
Retard 4°	> 200	50–200	<50	> 200
Advance 4°	50–200	<50	<50	<50
Advance 8°	50–200	<50	50–200	<50
Overfueling:				
10%	<50	50–200	<50	50–200
20%	<50	> 200	<50	50–200
Combined faults:				
25 in. H_2O intake restriction, retard 4° timing	50–200	<50	<50	50–200
50 in. H_2O intake restriction, retard 4° timing	> 200	50–200	<50	> 200
3 in. Hg exhaust restriction, advance 4° timing	50–200	<50	<50	<50
6 in. Hg exhaust restriction, advance 8° timing	<50	<50	50–200	50–200
25 in. H_2O intake restriction, 10% overfueling	<50	50–200	<50	50–200
50 in. H_2O intake restriction, 20% overfueling	<50	> 200	<50	> 200
10% overfueling and advance 4° timing	<50	50–200	<50	50–200
20% overfueling and advance 8° timing	<50	50–200	<50	> 200
25 in. H_2O intake restriction and 3 in. Hg exhaust restriction	<50	<50	<50	<50
50 in. H_2O intake restriction and 6 in. Hg exhaust restriction	<50	50–200	<50	50–200
3 in. Hg exhaust restriction and 10% overfueling	<50	50–200	<50	> 200
6 in. Hg exhaust restriction and 20% overfueling	<50	> 200	<50	> 200
25 in. H_2O intake restriction, 3 in. Hg exhaust restriction, 10% overfueling, and advance 4° timing	<50	50–200	<50	> 200
50 in. H_2O intake restriction, 6 in. Hg exhaust restriction, 20% overfueling, and advance 8° timing	<50	> 200	<50	> 200

[a] DPM production at most severe engine operating mode.
[b] Deviation from manufacturer's specification.

makeup tank. Fig. 11.5.3 shows a typical installation of a constant-level exhaust conditioner.

A waterbath exhaust conditioner can also act as a scrubber to remove some exhaust pollutants. The gases CO_2, SO_2, SO_3, and NO_2 are water soluble and are partially trapped, and some DPM is captured. Rapid corrosion from acid gases can occur unless the conditioner is constructed of stainless steel and flushed with fresh water regularly (Waytulonis, Smith, and Mejia, 1982). The gases CO and NO are practically insoluble in water and are not absorbed in any significant quantities. Also, because the conditioners cool the exhaust, some HCs are condensed and odor is reduced (Mogan, Dainty, and Lawson, 1986).

Most of the solid particles in the exhaust pass through the waterbath because a large proportion of the exhaust gas has no contact with the water. Internal baffles to breakup and distribute exhaust gas bubbles are thought to increase DPM and SO_2, removal. Total DPM is reduced 20 to 30% (Eccleston, Seizinger and Clingenpeel, 1981). Because of their design diversity, the effectiveness of waterbath exhaust conditioners to remove exhaust pollutants vary widely.

OXIDATION CATALYTIC CONVERTERS. These are sometimes referred to as *purifiers* and have been routinely used for many years for emission control on diesel engines used underground. During the early years of dieselization of underground mines, the functions performed by catalytic converters were considered more significant than they are today. Early work reported CO levels in the range of 800 to 2600 ppm and 100 ppm of aldehydes in some IDI engines used underground (Elliott, 1948; Holtz, 1960; Marshall and Hurn, 1968). Modern derated IDI engines in good mechanical condition typically have CO emissions less than 600 ppm and total HC emissions less than 200 ppm and, consequently, reduce the potential benefits of

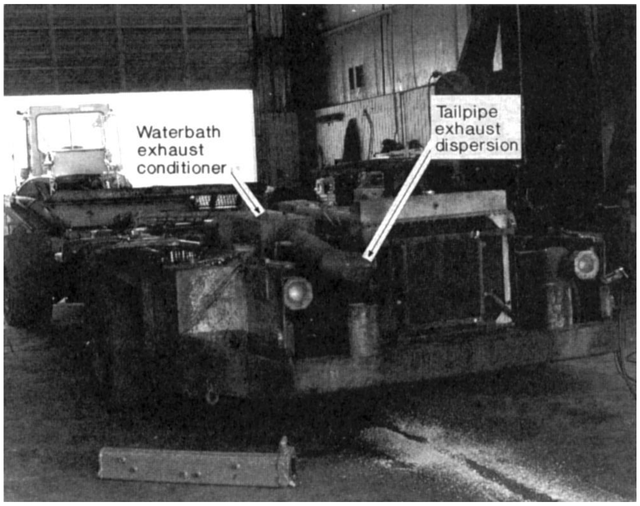

Fig. 11.5.3. Typical installation of a waterbath exhaust conditioner and tailpipe exhaust dispersion.

Fig. 11.5.4. Typical installation of an oxidation catalytic converter.

catalytic converters. In most instances, modem, well-maintained diesel engines operating in adequately ventilated areas do not need oxidation catalytic converters for emission control.

A *catalyst* is a substance that alters the rate of a chemical reaction without being consumed. In diesel exhaust, gas-phase reactions normally proceed at a very slow rate. On the catalyst, however, the rate is fast enough to be of practical value for controlling emissions. Noble metals (platinum and palladium) are typically used as the active components of the catalyst. The catalyst is deposited on a carrier or substrate and housed in a stainless steel container fitted into the exhaust system as shown in Fig. 11.5.4. There are two substrate configurations: pellets (alumina balls) and monoliths (ceramic or metallic honeycomb structure coated with alumina). Some configurations integrate a catalyst into a noise-reduction device. The monolith, shown schematically in Fig. 11.5.5, is the dominant type because it has better performance and stability than the pelleted alternatives and is generally smaller (Steel, 1988).

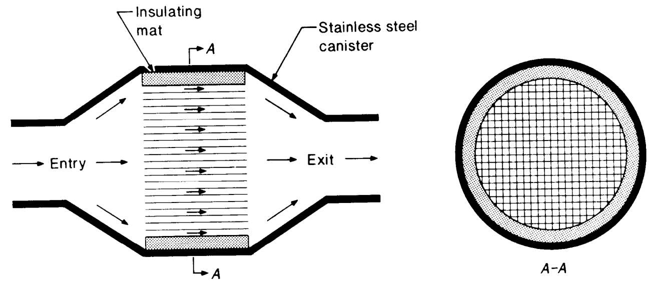

Fig. 11.5.5. Schematic of a monolithic oxidation catalytic converter.

Table 11.5.3. Effects of Oxidation Catalytic Converters on Some Diesel Exhaust Pollutants

Constituent	% Change
CO	−39 to −90%
HC	−2 to −68%
NO_2	0 to +25%
SO_4	+10 to + > 1000%
Aldehydes	up to −55%

The function of catalytic converters is to oxidize exhaust pollutants. This renders *some* of them less toxic. As hot exhaust gas passes over the catalyst, CO and HC (including aldehydes) are oxidized to CO_2 and water vapor, reducing the amount of CO, HC, and offensive odors in the exhaust (Marshall and Seizinger, 1978; Eccleston, Seizinger, and Clingenpeel, 1981). The catalyst formulation and its operating temperature are critical factors in converter performance (Rehnberg, 1981). The converter should be located as close as possible to the exhaust manifold to ensure maximum exhaust gas temperature. The temperatures required for 50% conversion of CO and HC are typically about 370°F (188°C) and 500°F (260°C), respectively. As higher exhaust-gas temperatures are attained, the conversion efficiency increases (McClure, Baumgard, and Watts, 1988).

Because specific catalyst formulations are proprietary, systematic studies of their effects on exhaust constituents cannot be made. However, the effects of current commercial catalytic converters on emissions are well documented. In addition to changing some exhaust pollutants to less toxic forms, properly functioning catalysts also have disadvantages dependent on fuel quality, catalyst formulation, and configuration. The disadvantages are conversion of SO_2 to SO_3 and H_2SO_4, conversion of NO to NO_2, and increasing mutagenicity of DPM. These effects are summarized in Table 11.5.3.

Catalysts are effective in reducing the SOF portion of DPM, but they increase the non-gaseous oxide portion (specifically sulfates). Almost all fuel sulfur emerges as SO_2 in raw exhaust. Upon passing over the catalyst, 5 to 90% of the fuel sulfur is oxidized to sulfate, depending on engine load and speed (Lawson et al., 1979; Hunter et al., 1981). However, a catalytic converter in service is likely to be coated and partially plugged with DPM so that the sulfur conversion to sulfate over a load cycle is probably in the 10 to 40% range (Johnson and Carlson, 1984).

Downstream in the exhaust, further reactions with water vapor form sulfuric acid mist. The lifetime of the acid mist in the mine environment is not known. Measurements in mine air have failed to identify any traces of SO_3 or H_2SO_4. One study reported measurements downstream of an operating vehicle and suggest that the acid mist quickly becomes an ore-related sulfate

(Dainty et al., 1987). Also at high exhaust temperatures, catalytic converters tend to oxidize NO to more toxic NO_2 (Lawson and Vergeer, 1977; Rhenberg, 1981).

Catalytic converters accumulate particulate material to varying degrees. This permits the DPM to be exposed to undiluted exhaust with subsequent nitration of organic compounds that can result in an increase in the concentrations of the biologically active SOF (Hunter et al., 1981). This enhanced mutagenic activity is probably due to nitration of the pyrene in the exhaust (Mogan et al., 1986a). Monolithic-type catalysts have little effect on PAH levels, while pelleted types cause a considerable increase. It has not been determined whether support configuration or catalyst formulation, both, or some other factor produces the increase in biological activity. These results suggest that any underground emission control device that permits the soot to be exposed to undiluted exhaust gas for significant periods of time should be carefully evaluated under actual working conditions before its use is considered (Mogan et al., 1983).

There may be special circumstances where catalytic converters could be beneficially used, for example, engines in good mechanical condition operated under moderate to heavy duty cycles and using fuel with sulfur less than 0.1% by weight. Also converters may reduce the incidence of false alarms where CO-based sensors are used for fire detection. *It might appear to be beneficial to use catalysts on diesel engines that are operated in inadequately ventilated spaces, or that are rebreathing their own exhaust, or on engines that do not receive regular preventive maintenance. However, these conditions violate basic safety principles in the use of diesel-powered equipment and must not be tolerated.*

In mine service, it must be remembered that catalytic devices deteriorate with use. Exhaust backpressure must be monitored and the converter replaced or cleaned at regular intervals to remain effective. Catalytic converters can maintain their performance up to 1000 hr with proper engine maintenance. Overall, current commercial oxidation catalytic converters appear to be counter-productive, but may have application in special circumstances. Developments by manufacturers are in progress to increase the range of applications where benefits outweigh the disadvantages.

DIESEL PARTICULATE FILTERS. It is difficult to trap the DPM contained in the exhaust stream because of its small aerodynamic diameter and tendency to plug filters. A number of filter materials have been tested including ceramic monoliths, wire mesh, ceramic foam, mat-like ceramic fibers, and woven silica fiber coils. To a lesser degree, paper and other fiber materials are under investigation, but the ceramic monolith filter or trap is thus far most promising for underground mining applications. The concept of a disposable trap is also possible, but has not yet been demonstrated for heavy-duty mining equipment. A typical installation of a ceramic monolith diesel particulate filter (DPF) is shown in Fig. 11.5.6.

Fig. 11.5.6. Typical installation of a diesel particulate filter.

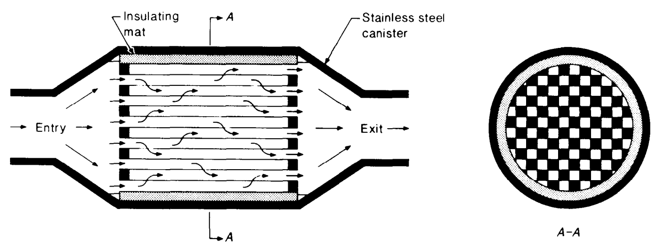

Fig. 11.5.7. Schematic of a ceramic monolith diesel particulate filter.

The ceramic monolith trap consists of a cellular substrate with square cells running the length of the filter, similar to monolithic catalyst supports used in most catalytic converters. On the inlet end, every other cell is plugged with ceramic material, with the adjacent cell plugged on the outlet end. This causes the exhaust to enter one cell, pass through the ceramic wall, and leave by an adjacent cell as shown in Fig. 11.5.7. This trap has a mass collection efficiency of 80 to 95%; however, it is less efficient at removing the SOF. This is because the carbonaceous portion of DPM is collected before the SOF has completely sorbed or condensed onto the surfaces of entrained particles and thus passes through as a gas or vapor (Baumgard and Kittleson, 1985). The SOF sampled downstream of the trap has a higher specific mutagenic activity than upstream samples, although the increased activity is largely offset by a decrease in the mass of SOF emitted (Draper et al., 1987). Only temporary reductions of SO_4 emissions are expected since much of the trapped SO_4 is released during regeneration (self-cleaning by combustion or evaporation of accumulated soot).

The most promising means of disposal of the collected soot is to burn or oxidize it in the trap, thus regenerating the filter. Regeneration depends upon the interaction between the deposited particulate material with oxygen, and varies with the temperature of the exhaust gases. About 5% of the sea level concentration of oxygen and exhaust temperature excursions in excess of 932°F (500°C) are necessary for regeneration. Soot regeneration (ignition) temperature can be lowered to about 750°F (399°C) by coating the substrate with noble metal catalysts and to 830°F (443°C) by coating with base metal catalysts. These catalytic traps also oxidize some exhaust components and are therefore termed trap oxidizers.

Regeneration of particulate traps is difficult to control on a diesel engine for a variety of reasons. One significant factor is that the diesel engine operating cycle can include periods with exhaust temperatures too low to start regeneration. Uncontrolled regeneration can occur when excessive amounts of soot have accumulated in the trap, and it is subsequently exposed to sufficient oxygen and heat to provide intense combustion. The result can damage the ceramic element, and also decrease efficiency. This type of failure is a potential safety hazard. Uncontrolled regeneration has resulted in melting of the ceramic element, CO excursions beyond 5000 ppm for several minutes, and surface temperatures in excess of 600°F (316°C) (Baumgard and Bickel, 1987).

The ability of particulate traps to store DPM over a useful operating period and to safely regenerate is necessary for their success. Methods to regenerate without creating safety hazards are being pursued. The first applications of particulate filters have been on mining machines that consistently generate high exhaust temperatures (those with moderate to heavy duty cycles) and operated at altitudes less than 2000 ft (610 m). Well-engineered systems have been used successfully in nongassy mine service (Dainty and Enga, 1988).

The key to safe operation is limiting the amount of soot in the filter. The amount of soot can be indicated by a pressure sensor mounted upstream of the trap and used to alert the operator to excess exhaust backpressure and the need for remedial action. Applications of traps in gassy mines are constrained by equipment-surface and exhaust-temperature limitations (Anon., 1988). Developments are underway to solve these problems (Waytulonis and Dvorznak, 1987).

11.5.3.3 Exhaust Dispersion Methods and Devices

After diesel exhaust leaves the tailpipe, it is diluted by the surrounding air and carried in the mine ventilation system. The gases CO and CO_2 are relatively stable, but some pollutants can undergo changes that may alter their chemical composition or their toxicological effects. In particular, water vapor and carbonaceous material react with gas-phase NO_2 (e.g., oxidation of NO to NO_2, adsorption of HC onto ambient particles). Conversion of NO to NO_2 can occur, especially in mines with large stopes, slow moving ventilation, and where no respirable dust standard is in force. Immediate dilution of NO from the tailpipe is effective in reducing its conversion to NO_2.

TAILPIPE LOCATION. To reduce exposure to raw exhaust, the exhaust should be dispersed immediately upon exiting the tailpipe. One effective means is to release the exhaust gas into the moving air from the radiator fan. This reduces the danger from local areas becoming contaminated and from the engine rebreathing exhaust. For liquid-cooled engines (with pusher fans), the exhaust flow can be directed into the airstream in front of the radiator, if the operator's position is favorable. See Fig. 11.5.3.

Another dispersion technique useful in headings with adequate roof clearance takes advantage of the lower density of hot exhaust, which tends to rise. Concentrations of exhaust often are higher near the roof, and cooler fresh air tends to enter deadened headings along the lower half of the drift (Janelid, 1976). The natural buoyancy effect can be enhanced by directing the exhaust upward and to the rear at about a 20 to 30° angle to horizontal. Machine operator's exposures to diesel exhaust were reduced up to 30 to 40% in a 150-ft (46-m) drift by this method (Johnson and Carlson, 1984).

FUME DILUTER. Jet-type exhaust dispersion devices entrain surrounding air and direct the diluted exhaust away from the machine operator's position. The exhaust gas is piped into the fume diluter manifold and released through a pre-set annular gap before it passes over an aerofoil surface. The high velocity of the jet creates an area of low pressure that inducts the surrounding air into the throat of the fume diluter. The secondary flow (which is about 10 to 12 times the volume of the primary flow) mixes with the exhaust causing rapid cooling and dilution. A jet of cool diluted exhaust is emitted from the fume diluter outlet at high velocity and mixes with surrounding air. A typical installation of a diesel fume diluter is shown in Fig. 11.5.8, with a schematic in Fig. 11.5.9.

In contrast with thermal convection, fume diluters do not rely on exhaust temperature but will dilute and cool under most engine conditions, low speed or idle being exceptions. Smoke opacity is reduced, and rapid dilution reduces the risk of NO to NO_2 conversion. The fume diluter has no moving parts. Periodic cleaning of the aerofoils and checking gap clearance is important to prevent plugging. The disadvantages of fume diluters are that there is no removal of exhaust pollutants, and they increase backpressure on the engine. Particularly appropriate applications are on diesel machines operated in tunnel development or dead-end headings.

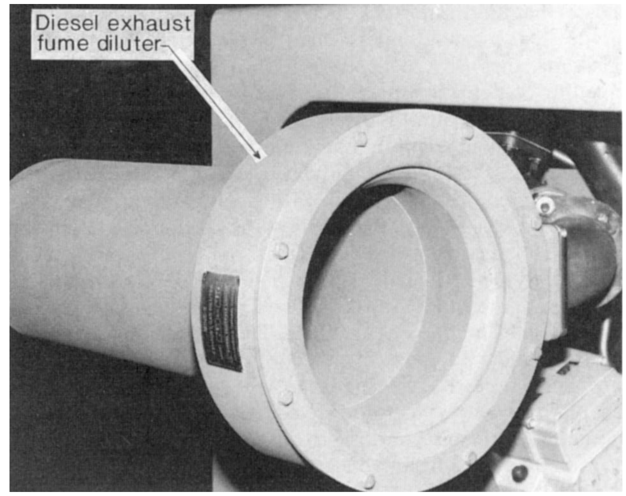

Fig. 11.5.8. Typical installation of diesel exhaust fume diluter.

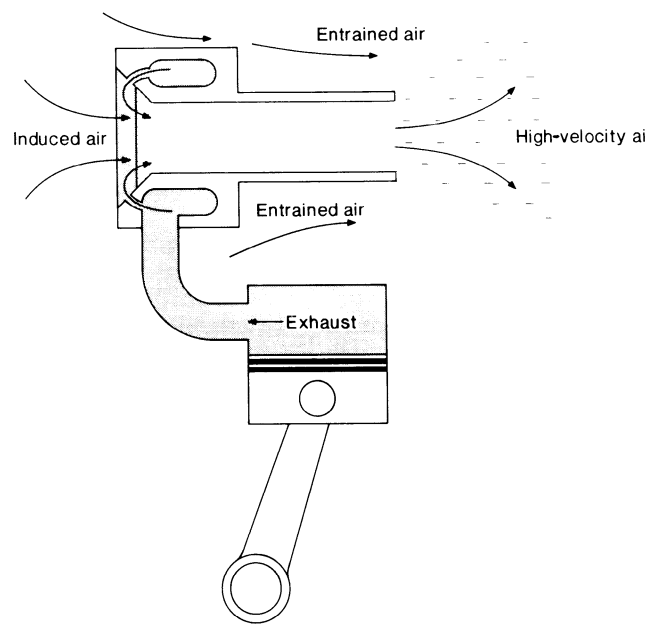

Fig. 11.5.9. Schematic of diesel exhaust fume diluter.

11.5.3.4 Future Exhaust Control Technology

The development of more effective emission controls will be driven by a lowering fuel quality and more stringent air quality standards. With conventional diesel fuels, the formation of particulates is fundamental to the combustion process. Projections indicate that the important fuel properties for emissions will degrade in the future unless there is regulatory intervention. Legislation to control fuel specifications (such as aromatic HCs and sulfur content) is already in effect in some states. Further, if a DPM exposure limit is established, MSHA will likely factor DPM emissions into dilution ventilation requirements.

Currently, the leading candidates for DPM aftertreatment are the trap oxidizer and the flow-through catalytic converter. The trap oxidizer is very effective in trapping the carbon portion of DPM and oxidizing the SOF. Trap material improvements could greatly benefit particulate trap system development. Applications of particulate traps could be increased if a safe, reliable regeneration technique was available. The performance of particulate traps and catalytic converters would improve with the use of low-sulfur diesel fuel. Catalysts, whether they are used on particulate traps or converters, are effective in reducing the SOF, and manufacturers are working to develop catalyst formulations that minimize their negative attributes.

DRY-TYPE EXHAUST CONDITIONING SYSTEMS. An alternative for waterbath exhaust conditioners is a dry-type exhaust conditioning system under development. This design does not require direct contact between the exhaust gas and water. Instead, the exhaust from the engine passes through a heat exchanger. The waste heat from the exhaust is dissipated by either the engine radiator or a separate cooling circuit. This heat load requires about 70% supplemental cooling capacity. A mechanical flame arrester and spark trap are used downstream of the heat exchanger. The dry system has the potential for requiring less maintenance than waterbath conditioners. Although this system does not alter pollutant concentrations, it can be used in tandem with a DPF, such as a trap oxidizer (Waytulonis and Dvorznak, 1987a).

INTEGRATED DPF AND DRY CONDITIONING SYSTEM. This is a combination of a DPF and the dry-type exhaust conditioner into a single, integrated control system for applications where permissible equipment is legislated. The integrated system simultaneously controls fire hazards and reduces DPM. As more efficient diesel exhaust control becomes necessary, integration of various devices into a system will become common.

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Chapter 11.6
MINE VENTILATION

RAJA V. RAMANI

11.6.1 INTRODUCTION

Mine ventilation is essentially the application of the principles of fluid dynamics to the flow of air in mine openings. While air is a compressible fluid, airflow in mines is generally treated as steady state, turbulent, and incompressible. Airflow is induced from the atmosphere, through the mine, and back to the atmosphere by creating a differential pressure between the intake and return openings of the mine. This differential pressure is very small, of the order of 2 to 3% in most cases, when compared to the absolute pressure of the system. Therefore, volume and density changes are neglected without serious loss of accuracy or validity. However, in situations where the pressure, temperature, and humidity changes are large and air conditioning processes are involved, calculations must incorporate compressibility considerations. Standard references for mine ventilation include books by Hartman, Mutmansky, and Wang (1982) and Burrows et al. (1982).

The *mine ventilation* system consists of fans, airways (openings to surface and interconnections in the mine between the openings through the working areas), and control devices for air coursing. Quantity control in this system means achieving desired airflows through the optimal selection of openings to surface; the shape, size, and number of airways; location of control devices; and selection and location of fans. The term "optimal" as used here is global and not restricted to ventilation only. Very often, the airflow logistics problem is intertwined with miner and materials transport as well as with many other safety and productivity considerations. The mine airflow distribution is completely defined by (1) the physical parameters of the airways—shape, area, length, and characteristics of the airway surface; (2) the layout of the mine openings; (3) the pressure sources (e.g., fans) in the system, their location and characteristics; and (4) the interconnections between the airways, mine openings, and pressure sources.

This chapter addresses mine ventilation fundamentals while those following (11.7.1 and 11.7.2) are concerned with ventilation applications to metal and coal mining. Mine air conditioning is covered in Chapter 11.4 and computer applications in 11.10.

11.6.2 DENSITY OF MINE AIR

Mine air is a mixture of gases and water vapor. The density or the specific weight of the air is the weight per unit of volume of the air-water vapor mixture. It is a function of the barometric pressure and dry- and wet-bulb temperatures.

An approximate formula (the symbol "a" in the equation number indicates SI units) for calculating density for dry air conditions is (also see Eq. 11.4.3)

$$w = \frac{1.327B}{(460 + t_d)} \tag{11.6.1}$$

$$w = \frac{B}{0.287\ (273 + t_d)} \tag{11.6.1a}$$

where w is density of dry air in lb/ft^3 (kg/m^3), B is barometric pressure in in. mercury (Pa), and t_d is dry-bulb temperature in °F (°C). Table 11.6.1 provides values for w for t_d from 0 to 90°F (– 17.8 to 32.2°C) and B from 22.0 to 31.0 in. mercury (74.29 to 104.68 kPa).

Incorporating a correction for water vapor in the air, the general formula becomes (see Eq. 11.4.4)

$$w = \frac{1.327}{(460 + t_d)} (B - 0.378f) \tag{11.6.2}$$

$$w = \frac{1}{0.287\ (273 + t_d)} (B - 0.378f) \tag{11.6.2a}$$

where f is the vapor pressure at the dew-point temperature in in. mercury (kPa). Dew-point temperatures and vapor pressures can be obtained from Table 11.6.2. A convenient chart for determining air density is shown in Fig. 11.6.1.

Example 11.6.1. What is the air density under the following conditions: wet-bulb 60°F (15.56°C), dry-bulb 70°F (21.11°C), barometric pressure 28.75 in. Hg (97.13 kPa)?

Solution.

(a) Neglecting vapor considerations, the density can be read from Table 11.6.1:

$$w = 0.0740\ \text{lb/ft}^3\ (1.185\ \text{kg/m}^2)$$

(b) Using Eq. 11.6.1,

$$w = \frac{1.327}{460 + 70} (28.75) = 0.0720\ \text{lb/ft}^3\ (1.153\ \text{kg/m}^3)$$

(c) Using Table 11.6.2, determine the following dew-point temperature at 70°F dry-bulb (21.11°C), 60°F (15.56°C) wet-bulb (i.e., 10°F (5.55°C) wet-bulb depression), and the vapor pressure at the temperature.

Dew-point temperature = 53°F (11.66°C)

Vapor pressure at 53°F = 0.402 in. mercury (1.36 kPa)

$$\begin{aligned} w &= \frac{1.327}{460 + 70} (28.75 - 0.402) \\ &= 0.0710\ \text{lb/ft}^3\ (1.137\ \text{kg/m}^3) \end{aligned}$$

Method (c) is most accurate. The accompanying chart (Fig. 11.6.1) can also be used to estimate the densities of air/water vapor mixtures.

The density of dry air at sea level and 70°F (21.11°C) is the reference standard. This standard density value is 0.0750 lb/ft^3 (1.20 kg/m^3). The unit used in mine ventilation for pressure is in. water (Pa or kPa). The relationship between this unit and other units of pressure is

1 in. water (248.84 Pa) = 5.2 lb/ft^2 ≅ 70 ft air pressure at standard density

Table 11.6.1. Approximate Air Densities for Dry Air at Different Temperatures

Barometer, in. mercury	Dry-bulb Temperature 0° F	5° F	10° F	15° F	20° F	25° F	30° F	35° F	40° F	45° F	50° F	55° F	60° F	65° F	70° F	75° F	80° F	85° F	90° F
31.0	0.0894	0.0885	0.0875	0.0866	0.0857	0.0848	0.0839	0.0831	0.0823	0.0815	0.0807	0.0799	0.0791	0.0783	0.0776	0.0769	0.0762	0.0755	0.0748
30.5	.0880	.0870	.0861	.0852	.0843	.0834	.0826	.0818	.0809	.0801	.0794	.0786	.0778	.0771	.0764	.0756	.0749	.0723	.0736
30.0	.0865	.0856	.0847	.0838	.0829	.0821	.0812	.0804	.0796	.0788	.0781	.0773	.0766	.0758	.0751	.0744	.0737	.0730	.0724
29.5	.0851	.0842	.0833	.0824	.0816	.0807	.0799	.0791	.0783	.0775	.0768	.0760	.0753	.0746	.0739	.0732	.0725	.0718	.0712
29.0	.0837	.0828	.0819	.0810	.0802	.0793	.0785	.0777	.0770	.0762	.0755	.0747	.0740	.0733	.0726	.0719	.0712	.0706	.0700
28.5	.0822	.0813	.0805	.0796	.0788	.0780	.0772	.0764	.0756	.0749	.0742	.0734	.0727	.0720	.0714	.0707	.0700	.0694	.0688
28.0	.0808	.0799	.0790	.0782	.0774	.0766	.0758	.0751	.0743	.0736	.0729	.0721	.0715	.0708	.0701	.0694	.0688	.0682	.0676
27.5	.0793	.0785	.0776	.0768	.0760	.0752	.0745	.0737	.0730	.0723	.0715	.0709	.0702	.0695	.0689	.0682	.0676	.0700	.0664
27.0	.0779	.0770	.0762	.0754	.0746	.0739	.0731	.0724	.0717	.0710	.0702	.0696	.0689	.0683	.0676	.0670	.0663	.0657	.0651
26.5	.0764	.0756	.0748	.0740	.0733	.0725	.0718	.0710	.0703	.0696	.0689	.0683	.0676	.0670	.0664	.0657	.0651	.0645	.0639
26.0	.0750	.0742	.0734	.0726	.0719	.0711	.0704	.0697	.0690	.0683	.0676	.0670	.0663	.0657	.0651	.0645	.0639	.0633	.0627
25.5	.0736	.0728	.0720	.0712	.0705	.0698	.0690	.0684	.0677	.0670	.0663	.0657	.0651	.0645	.0638	.0632	.0626	.0621	.0615
25.0	.0721	.0713	.0706	.0698	.0691	.0684	.0677	.0670	.0663	.0657	.0650	.0644	.0638	.0632	.0626	.0620	.0614	.0609	.0603
24.5	.0707	.0699	.0692	.0684	.0677	.0670	.0663	.0657	.0650	.0644	.0637	.0631	.0625	.0619	.0613	.0608	.0602	.0597	.0591
24.0	.0692	.0685	.0677	.0671	.0664	.0657	.0650	.0643	.0637	.0631	.0624	.0618	.0612	.0607	.0601	.0595	.0590	.0584	.0579
23.5	.0678	.0671	.0663	.0657	.0650	.0643	.0636	.0630	.0624	.0618	.0611	.0606	.0600	.0594	.0588	.0583	.0577	.0572	.0567
23.0	.0663	.0656	.0649	.0643	.0636	.0629	.0623	.0617	.0610	.0604	.0598	.0593	.0587	.0581	.0576	.0570	.0565	.0560	.0555
22.5	.0649	.0642	.0635	.0629	.0622	.0616	.0609	.0603	.0597	.0591	.0585	.0580	.0574	.0569	.0563	.0558	.0553	.0548	.0543
22.0	.0635	.0628	.0621	.0615	.0608	.0602	.0596	.0590	.0584	.0578	.0572	.0567	.0561	.0556	.0551	.0546	.0540	.0536	.0531

Calculated from formula $d = \frac{1.327}{(460+T)} \times B$, lb/ft³

Source: Kingery, 1960.
Conversion Factors:
1 in. Hg = 3.3768 kPa
°C = 5/9 (°F – 32)
1 lb/ft^3 = 16.018 kg/m^3

Table 11.6.2. Dew-point Temperature, °F (Barometric Pressure = 29.0 in. Mercury)

Air temperature t, °F	Vapor pressure e, in. Hg	Depression of wet-bulb thermometer $t - t'$													
		1.0	2.0	3.0	4.0	5.0	6.0	7.0	8.0	9.0	10.0	11.0	12.0	13.0	14.0
20	0.103	17	13	8	2	−5	−18								
21	.108	18	14	10	4	−3	−14	−42							
22	.113	19	15	11	6	−1	−10	−29							
23	.118	20	16	12	8	+1	−7	−22							
24	.124	21	18	14	9	3	−4	−17							
25	.130	22	19	15	11	5	−2	−12	−36						
26	.136	23	20	16	12	7	±0	−9	−26						
27	.143	24	21	18	14	9	+3	−5	−19						
28	.150	25	22	19	15	11	5	−2	−14	−45					
29	.157	26	24	20	17	12	7	±0	−9	−29					
30	.164	27	25	22	18	14	9	+3	−5	−20					
31	.172	29	26	23	20	16	11	5	−2	−14	−50				
32	.180	30	27	24	21	17	13	8	+1	−9	−29				
33	.187	31	28	25	22	19	15	10	3	−5	−20				
34	.195	32	29	27	24	20	16	12	6	−2	−14	−50			
35	.203	33	30	28	25	22	18	14	8	+1	−8	−28			
36	.211	34	31	29	26	23	20	15	11	4	−4	−19			
37	.219	35	32	30	27	24	21	17	13	7	−1	−12	−44		
38	.228	36	33	31	28	26	23	19	14	9	+3	−7	−25		
39	.237	37	34	32	29	27	24	21	16	12	6	−3	−16		
40	.247	38	35	33	31	28	25	22	18	14	8	+1	−10	−35	
41	.256	39	37	34	32	29	26	23	20	16	11	4	−5	−21	
42	.266	40	38	35	33	30	28	25	21	17	13	7	−1	−13	−59
43	.277	41	39	36	34	31	29	26	23	19	15	10	+3	−7	−28
44	.287	42	40	38	35	32	30	27	24	21	17	12	6	−2	−17
45	.298	43	41	39	36	34	31	29	26	22	19	14	8	+2	−9
46	.310	44	42	40	37	35	32	30	27	24	20	16	11	5	−4
47	.322	45	43	41	39	36	34	31	28	25	22	18	13	8	±0
48	.334	46	44	42	40	37	35	32	30	27	23	20	15	10	+4
49	.347	47	45	43	41	39	36	34	31	28	25	21	17	13	7
50	.360	48	46	44	42	40	37	35	32	29	27	23	19	15	9
51	.373	49	47	45	43	41	39	36	34	31	28	25	21	17	12
52	.387	50	48	46	44	42	40	37	35	32	29	26	23	19	14
53	.402	51	49	47	45	43	41	39	36	34	31	28	24	21	16
54	.417	52	50	49	47	44	42	40	38	35	32	29	26	23	19
55	.432	53	52	50	48	46	43	41	39	36	34	31	28	24	21
56	.448	54	53	51	49	47	45	43	40	38	35	32	29	26	23
57	.465	55	54	52	50	48	46	44	42	39	36	34	31	28	24
58	.482	56	55	53	51	49	47	45	43	40	38	35	32	29	26
59	.499	57	56	54	52	50	48	46	44	42	39	37	34	31	28
60	.517	58	57	55	53	51	49	47	45	43	41	38	35	32	29
61	.536	59	58	56	54	52	51	49	46	44	42	39	37	34	31
62	.555	60	59	57	55	54	52	50	48	46	43	41	38	35	32
63	.575	61	60	58	56	55	53	51	49	47	45	42	40	37	34
64	.595	62	61	59	58	56	54	52	50	48	46	44	41	38	36
65	.616	63	62	60	59	57	55	53	51	49	47	45	43	40	37
66	.638	64	63	61	60	58	56	54	53	51	48	46	44	42	39
67	.661	65	64	62	61	59	57	56	54	52	50	48	45	43	40
68	.684	67	65	63	62	60	58	57	55	53	51	49	47	44	42
69	.707	68	66	64	63	61	60	58	56	54	52	50	48	46	43
70	.732	69	67	66	64	62	61	59	57	55	53	51	49	47	45
71	.757	70	68	67	65	63	62	60	58	57	55	53	51	49	46
72	.783	71	69	68	66	65	63	61	60	58	56	54	52	50	48
73	.810	72	70	69	67	66	64	62	61	59	57	55	53	51	49
74	.838	73	71	70	68	67	65	64	62	60	58	56	54	53	50
75	.866	74	72	71	69	68	66	65	63	61	60	58	56	54	52
76	.896	75	73	72	70	69	67	66	64	62	61	59	57	55	53
77	.926	76	74	73	71	70	68	67	65	64	62	60	58	56	54
78	.957	77	75	74	72	71	69	68	66	65	63	61	59	58	56
79	.989	78	76	75	73	72	70	69	67	66	64	62	61	59	57
80	1.022	79	77	76	75	73	72	70	69	67	65	64	62	60	58

Source: Marvin, 1915. Conversion Factors: 1 in. Hg = 3.3768 kPa, °C = 5/9 (°F – 32).

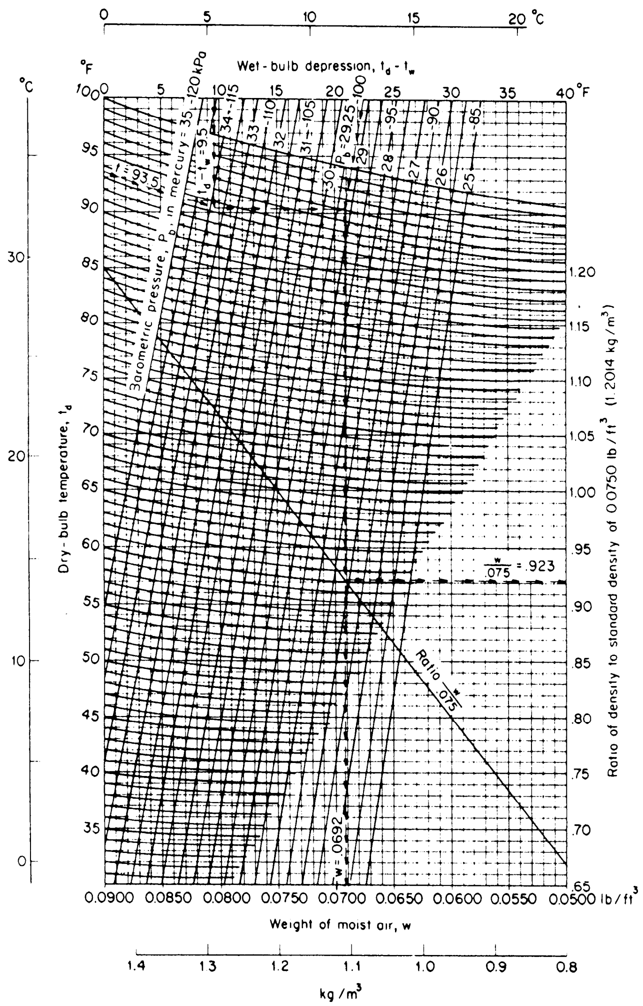


Fig. 11.6.1. Chart for determining air density and ratio to standard density (McElroy, 1935; Hartman, Mutmansky, and Wang, 1982).

11.6.3 AIRFLOW FUNDAMENTALS

11.6.3.1 General Equation for Pressure Difference

The relationship of the total pressures (or heads) at two points between which airflow occurs is given by

$$H_{T_1} = H_{T_2} + H_{\ell_{1-2}} \tag{11.6.3}$$

where H_{T_1} and H_{T_2} are the total pressures at points 1 and 2, respectively, and $H_{\ell_{1-2}}$ is the head loss due to flow between 1 and 2.

The total head at a point is the sum of the absolute static head (atmospheric pressure H_{a_1} + the gage static pressure H_{s_1}), the velocity head H_{v_1}, and the elevation head at the point H_{z_1}:

$$H_T = (H_a + H_s) + H_v + H_z \tag{11.6.4}$$

Velocity Head—The velocity head at a point is given by

$$H_v = w\left(\frac{V}{1098}\right)^2 \tag{11.6.5}$$

$$H_v = \frac{wV^2}{2g} \tag{11.6.5a}$$

where H_v is velocity head in in. water (Pa) and V is air velocity in fpm (m/s). In Eq. 11.6.5a, w/g is the mass density in kg/m^3.

Under standard density conditions (w = 0.0750 lb/ft^3), Eq. 11.6.5 reduces to

$$H_v \cong \left(\frac{V}{4000}\right)^2 \tag{11.6.6}$$

Similarly, velocity,

$$V = 4000\sqrt{H_v} \tag{11.6.7}$$

Simplified Gage Pressure Equation—In mine ventilation, calculations are generally performed in gage pressures, in which case, the total head is defined as

$$H_T = H_s + H_v \tag{11.6.8}$$

and the head loss between two points 1 and 2, as

$$\begin{aligned} H_{\ell_{1-2}} &= H_{T_1} - H_{T_2} \\ &= (H_{s_1} - H_{s_2}) + (H_{v_1} - H_{v_2}) \end{aligned} \tag{11.6.9}$$

Air always flows from a point with higher total head to a point with lower total head. Because velocity heads and velocity head differences are very small, only static pressures are used in many mine ventilation calculations. However, where velocities are high and where there are major changes in cross-sectional areas (e.g., shafts, fan drifts, etc.), velocity pressures must be considered as well.

11.6.3.2 Resistance to Airflow

The resistance to the flow of a fluid in a pipe arises from several sources: (1) viscosity of the fluid (internal friction), (2) friction between the fluid and the pipe internal surface, (3) changes in area and direction of flow, and (4) obstructions in the path of flow. The pressure loss incurred in overcoming the resistance from the first two sources is *friction loss* and that in overcoming the last two is *shock loss.* Friction losses account for over 70% of the pressure loss in mine ventilation and is, by far, the most important. Friction loss due to viscosity of the fluid is generally neglected as the airflow in mine airways is virtually always turbulent. Shock losses, which may account for as much as 30% of the pressure losses, must be properly considered in mine ventilation design.

Friction Loss—The most widely used formula for friction pressure loss in a mine airway is the Atkinson formula:

$$H_f = \frac{KLOV^2}{5.2A} \tag{11.6.10}$$

$$H_f = \frac{KLOV^2}{A} \tag{11.6.10a}$$

where H_f is pressure (or head) loss in in. water (Pa), L is length of the airway in ft (m), O is perimeter of the airway in ft (m), A is a cross-sectional area of the airway in ft^2(m^2), and K is friction factor for the airway in lb-min^2/ft^4 (kg/m^3).

Since L, O, A, and K are constant for a given airway,

$$H_f \propto V^2$$

Atkinson's equation for head loss is similar to the Darcy-Weisbach formula for calculating the head loss for fluid flow in pipes and conduits.

Friction Factors—The coefficient K in the Atkinson equation is an empirical factor. Its value is determined by measuring the values for H_f, O, L, A, and V in an established airway:

English units

$$K = \frac{5.2AH_f}{LOV^2} \tag{11.6.11}$$

SI units

$$K = \frac{AH_f}{LOV^2} \tag{11.6.11a}$$

The value for K is a function of the type of flow (laminar or turbulent) in an airway. Therefore, it is a function of Reynold's number and, in turn, of the velocity V. The friction head loss varies with the velocity raised to a power between 1.75 and 2.0. In a mine airway, however, there are many minor variations in area and shape and in the velocity of the air. In view of this fact and that the flow in most cases is turbulent, no major error is committed by using Atkinson's equation for estimating head loss. Where more accurate calculations are needed or where the flow is not turbulent, the influence of Reynold's number on friction factors may be taken into consideration.

US Bureau of Mines Schedule of Friction Factors—McElroy and Richardson (1927) have developed a table of friction factors that can be applied to mine airways in both coal and metal mines (Table 11.6.3). The tabulated values apply to air of standard density. When working with air densities other than standard, the value of K should be corrected as follows:

$$K_{\text{actual}} = K_{\text{table}} \times \frac{w}{0.0750} \tag{11.6.12}$$

$$K_{\text{actual}} = K_{\text{table}} \times \frac{w}{1.201} \tag{11.6.12a}$$

Kharkar, Ramani, and Stefanko (1974) surveyed five coal mines in Pennsylvania, West Virginia, and Virginia and 108 roadway segments to find friction factor values for coal mine airways. The results of their survey are shown in Table 11.6.4.

Whenever possible, the friction factor of an airway should be determined from actual measurements. Such determinations may not be possible where the mine is in the planning stages or where new areas are being opened up. In these cases, it is a good idea to get an approximately correct value for the friction factor from measurements under mine airway conditions that are approximately the same.

Example 11.6.2. Consider a concrete-lined shaft 300 ft (9 1.44 m) deep, 18 ft (5.49 m) in diameter, and moderately obstructed. Calculate the head loss if a quantity of 400,000 cfm (188.78 m^3/s) is flowing in the shaft.

Solution.

$$L = 300 \text{ ft } (91.44 \text{ m})$$

$$O = \pi D = \pi 18 = 56.52 \text{ ft } (17.23 \text{ m})$$

$$A = \frac{\pi D^2}{4} = \frac{\pi\,(18)^2}{4} = 254.34 \text{ ft}^2\ (23.63 \text{ m}^2)$$

Air velocity can be found from this relation

$$V = \frac{Q}{A} \tag{11.6.13}$$

$$= \frac{400{,}000}{254.34} = 1572.70 \text{ fpm } (7.99 \text{ m/s})$$

The maximum value for K (from Table 11.6.3) for a smooth-lined, straight, moderately obstructed airway is 35×10^{-10} lb-min^2/ft^4 (0.0065 kg/m^3). Using Eq. 11.6.10,

$$H_f = \frac{KLOV^2}{5.2A}$$

$$H_f = \frac{35 \times 10^{-10} \times 300 \times 56.52 \times (1572.70)^2}{5.2 \times 254.34}$$

$$= 0.111 \text{ in. water } (27.67 \text{ Pa})$$

Using Eq. 11.6.10a, the head loss due to friction is H_f = 27.67 Pa.

Example 11.6.3. Consider a roadway 2400 ft (731.52 m) long with an area of 90 ft^2 (8.36 m^2) and a perimeter of 40 ft (12.19 m). A head loss of 1.5 in. water (373.26 Pa) is measured when the airflow is 54,000 cfm (25.29 m^3/s). Calculate the friction factor for the roadway.

Solution. Use Eq. 11.6.11, first finding V (Eq. 11.6.13):

$$V = \frac{Q}{A} = \frac{54{,}000}{90} = 600 \text{ fpm } (3.048 \text{ m/s})$$

$$K = \frac{5.2 \times 1.5 \times 90}{2400 \times 40 \times (600)^2 \times 600}$$

$$= 203 \times 10^{-10} \text{ lb-min}^2/\text{ft}^4\ (0.03767 \text{ kg/m}^3)$$

Head Loss Equations—Since $V = \dfrac{Q}{A}$, it is readily seen that

$$\text{H}_f = \frac{KLO(Q/A)^2}{5.2A} = \frac{KLOQ^2}{5.2A^3} \tag{11.6.14}$$

$$H_f = \frac{KLOQ^2}{A^3} \tag{11.6.14a}$$

It is a common practice in ventilation calculations to express the quantity in units of 100,000 cfm (i.e., $Q = Q \times 10^{-5}$), and use for the value of K the whole number given in Tables 11.6.3 and 11.6.4. Using this procedure, the 10^{-10} associated with K is compensated for by the 10^{10} associated with Q^2 :

$$H_f = \frac{[K]\ 10^{-10}\ L\ O\ [Q \times 10^5]^2}{5.2A^3} = \frac{KLOQ^2}{5.2A^3} \tag{11.6.15}$$

where K is the whole-number friction factor termed henceforth the airway friction factor, and Q is quantity flowing in units of 100,000 cfm (47.195 m^3/s).

For a given airway, K, L, O, and A do not vary. As such, the friction loss is proportional to the square of the quantity. Further, the *resistance* of the airway, defined below, is a constant:

Table 11.6.3. US Bureau of Mines Schedule of Friction Factors for Mine Airways

		Values of $K \times 10^{10}$, lb-min^2/ft^4											
		Straight			Sinuous or curved								
					Slightly			Moderately			High degree		
Type of airway	Irregularities of surfaces, areas, and alignment	Clean (basic values)	Slightly ob-structed	Moder-ately ob-structed	Clean	Slightly ob-structed	Moder-ately ob-structed	Clean	Slightly ob-structed	Moder-ately ob-structed	Clean	Slightly ob-structed	Moder-ately ob-structed
Smooth lined	Minimum	10	15	25	20	25	35	25	30	40	35	40	50
	Average	15	20	30	25	30	40	30	35	45	40	45	55
	Maximum	20	25	35	30	35	45	35	40	50	45	50	60
Sedimentary rock (or coal)	Minimum	30	35	45	40	45	55	45	50	60	55	60	70
	Average	55	60	70	65	70	80	70	75	85	80	85	95
	Maximum	70	75	85	80	85	95	85	95	100	95	100	110
Timbered (5-foot centers)	Minimum	80	85	95	90	95	105	95	100	110	105	110	120
	Average	95	100	110	105	110	120	110	115	125	120	125	135
	Maximum	105	110	120	115	120	130	120	125	135	130	135	145
Igneous rock	Minimum	90	95	105	100	105	115	105	110	120	115	120	130
	Average	145	150	160	155	160	165	160	165	175	170	175	195
	Maximum	195	200	210	205	210	220	210	215	225	220	225	235

Source: McElroy, 1935. Note: All values of K are for air weighing 0.0750 lb/ft^3. Values in the table are expressed in whole numbers but must be multiplied by 10^{-10} to obtain the proper K value.

Conversion Factors: 1 lb-min^2/ft^4 = 1.855×10^6 kg/m^3, 1 lb/ft^3 = 16.018 kg/m^3.

Table 11.6.4. Friction Factors for Coal Mine Airways. Values of $K \times 10^{10}$, lb . min^2/ft^4

Type of airways	Straight: Clean	Straight: Slightly obstructed	Straight: Moderately obstructed	Curved: Clean	Curved: Slightly obstructed	Curved: Moderately obstructed
Smooth lined	25 10, 15, 20	28 15, 20, 25	34 25, 30, 35	31 25, 30, 25	30 30, 35, 40	43 40, 45, 50
Unlined or roof bolted	43 30, 55, 70	49 35, 60, 75	61 45, 70, 85	62 45, 70, 85	68 50, 75, 95	74 60, 85, 110
Timbered	67 80, 95, 105	75 85, 110, 110	82 95, 110, 120	85 95, 110, 120	87 110, 115, 125	90 110, 125, 135

Source: Kharkar, Ramani, and Stefanko, 1974.

Notes: All values of K are for air weighing 0.0750 lb/ft^3. Values in the table are expressed in whole numbers but must be multiplied by 10^{-10} to obtain the proper K value. Units: lb-min^2/ft^4. Figures underlined are McElroy's min., avg., and mas values of friction factors from Table 11.6.3. Conversion Factors: 1 lb-min^2/ft^4 = 1.855×10^6 kg/m^3, 1 lb/ft^3 = 16.018 kg/m^3.

$$R = \frac{KLO}{5.2A^3} \quad (11.6.16)$$

$$R = \frac{KLO}{A^3} \quad (11.6.16a)$$

where R is resistance of the airway in in.-min^2/ft^6 (N-s^2/m^8). Similarly,

$$H_f = RQ^2 \quad (11.6.17)$$

Example 11.6.4. It is desired to pass 20,000 cfm (9.44 m^3/s) of air through a straight timbered drift measuring $4 \times 5 \times 2000$ ft ($1.22 \times 1.52 \times 609.6$ m). The density of air is 0.0500 lb/ft^3 (0.8000 kg/m^3). Calculate (a) the pressure difference that must be maintained between the ends of the drift, and (b) the resistance of the drift.

Solution .

(a) Find the air velocity by Eq. 11.6.13:

$$V = \frac{20{,}000}{20} = 1000 \text{ fpm } (5.08 \text{ m/s})$$

From Table 11.6.3, the value for K, assuming a clean airway, is 95×10^{-10} lb-min^2/ft^4 (0.0176 kg/m^3). Using Eq. 11.6.10,

$$H_f = \frac{95 \times 10^{-10} \times 2000 \times 18 \times (1000)^2}{5.2 \times 20}$$

= 3.29 in. water at w = 0.075 lb/ft^3 (818.68 Pa at w = 1.20 kg/m^3)

H_f at 0.0500 lb/ft^3 (0.8000 kg/m^3) density $= 3.29 \times \dfrac{0.0500}{0.0750} =$ 2.19 in. water (544.96 Pa)

(b) $H_f = RQ^2$ where Q is in units of 100,000 cfm and therefore 2.19 in. water = $R(0.2)^2$ or

R = 54.75 in.-min^2/ft^6(6.12 N-s^2/m^8) at w = 0.0500 lb/ft^3 (0.8000 kg/m^3)

$R = 54.75 \times \dfrac{0.0750}{0.0500} =$ 82.13 in.-min^2/ft^6 (9.1739 N-s^2/m^8) at w = 0.0750 lb/ft^3 (1.20 kg/m^3)

SHOCK LOSSES. *Shock losses* arise from changes in direction (e.g., bends), changes in cross-sectional areas (e.g., obstructions), or changes in both (e.g., junctions and splits). Shock losses are independent of the roughness of walls and therefore cannot be computed directly as friction losses. However, shock losses bear a constant ratio to the velocity pressure corresponding to the mean velocity of flow. This ratio is absolutely constant for most conditions of area changes and virtually constant for bends (McElroy, 1935). There are three methods for estimating shock losses.

1. *Calculate the shock loss as a function of the velocity head:*

$$H_x = XH_v \quad (11.6.18)$$

where H_x is head loss due to shock, and X is an empirical shock loss factor found by experiment.

2. *Account for shock losses by increasing the value of the friction factor K for that section of the airway where shock losses occur.* The US Bureau of Mines' friction factor table (Table 11.6.3) accounts for obstructions and sinuosity of airways.

3. *Account for shock loss by expressing a shock loss condition as an additional length of a straight airway to be added to the given length of the airway.* This length is known as the equivalent length, L_e. With the equivalent length method, the head loss in an airway H_L is obtained by including in the Atkinson equation the equivalent length:

$$H_L = H_f + H_x = \frac{K(L + L_e)\,O\,V^2}{5.2A} \quad (11.6.19)$$

$$H_L = H_f + H_x = \frac{K(L + L_e)\,OV^2}{A} \quad (11.6.19a)$$

where L_e is equivalent length in ft (m).

This method is recommended for all routine mine ventilation calculations. Hartman and Mutmansky (1982) provide values for equivalent length for various shock conditions based on an airway with $K = 100 \times 10^{-10}$ lb-min^2/ft^4 (0.01855 kg/m^3) and a hydraulic radius (area/perimeter) R_H = 2 ft (0.61 m) (Table 11.6.5). They also recommend the following procedure for its use:

1. Values of L_e from Table 11.6.5 need not be corrected for K or R_H.

Table 11.6.5. Equivalent Lengths for Various Sources of Shock Loss

Shock Loss Condition	Equivalent Length, ft
Bend, acute, round	3
Bend, acute, sharp	150
Bend, right, round	1
Bend, right, sharp	70
Bend, obtuse, round	0.5
Bend, obtuse, sharp	15
Doorway	70
Overcast	65
Entrance	3
Discharge	65
Contraction, gradual	1
Contraction, abrupt	10
Expansion, gradual	1
Expansion, abrupt	20
Splitting, straight branch	30
Splitting, deflected branch (90°)	200
Junction, straight branch	60
Junction, deflected branch (90°)	30
Mine car or skip (20% roadway)	100
Mine car or skip (40% roadway)	500

Source: Hartman and Mutmansky, 1982. Conversion Factor: 1 ft = 0.3048 m.

2. With a change in area (splitting not involved), shock loss is included in the airway section following the change. This also applies to a bend in conjunction with an area change. Separate values are provided for shock losses at entrance and discharge.

3. At splits and junctions in airways, only the portion of the total flow involved in a change of direction or area is used. Values from Table 11.6.5 assume an even division of flow and allow for bend and area change. Include the loss at a split or junction in the pressure drop for the particular branch.

4. Judgment must be exercised in making proper allowance for unusual sources of shock loss such as obstructions.

Values from Table 11.6.5 are sufficiently accurate for all routine work. For more precise calculations, such as would be required for research, the following formulas should be used (McElroy, 1935):

$$L_e = \frac{3240\ R_H X}{10^{10} K} \tag{11.6.20}$$

$$L_e = \frac{w R_H X}{2gK} \tag{11.6.20a}$$

where K is coefficient of friction in lb-min^2/ft^4 (kg/m^3) for a straight section of the airway with the same internal surface characteristics as that of the section for which shock loss is being determined. Values for the shock loss factor X (dimensionless) may be calculated using the methods suggested by McElroy (1935) or Hartman and Mutmansky (1982). Shock loss can be a significant factor in the total losses sustained in an airway. Irrespective of the method used to calculate shock loss, the underlying relationship is that shock loss is proportional to the square of the air velocity and therefore of the air quantity.

11.6.3.3 Power Considerations

The total head loss (H_L) is the sum of both frictional and shock losses in flow from one point to another, as determined

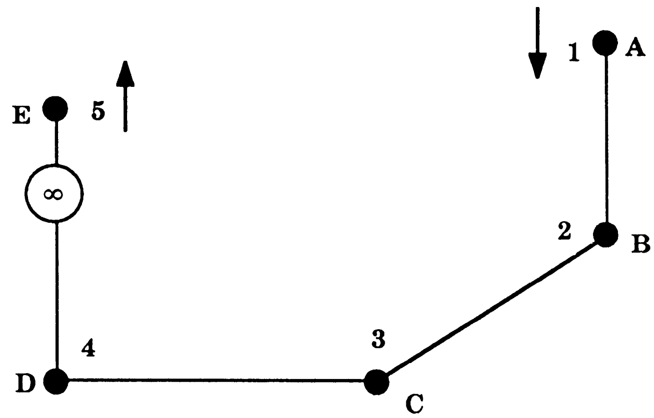


Fig. 11.6.2. Line diagram of the ventilation system (Ex. 11.6.5).

by Eq. 11.6.19. For flow to be continuous in the airway, the head H_L lost by the air must be compensated for by a continuous input of head H_L to the air. In other words, the loss of power due to friction and shock, must be compensated for by input of equivalent power. This power is known as the air horsepower and is given by

$$AHP = \frac{H_L Q}{6350} \tag{11.6.21}$$

$$AHP = \frac{H_L Q}{1000} \tag{11.6.21a}$$

where AHP is air horsepower (kW). Since H_L varies as Q^2, AHP varies as Q^3. The actual power input required to create the flow in the airway will be higher and will be determined by the efficiencies of the pressure generating source and its drive.

Example 11.6.5. Given the following data (see Fig. 11.6.2):

Airway	Width, W ft (m)	Height, H ft (m)	Length, L ft (m)	Friction Factor, k lb-min^2/ft^4 $\times$ 10^{10} (kg/m^3)
AB (shaft)	12 diameter (3.66)		600 (182.88)	25 (0.0046)
BC	20 (6.10)	6 (1.83)	2000 (609.60)	50 (0.0093)
CD	20 (6.10)	6 (1.83)	3000 (914.40)	50 (0.0093)
DE (shaft)	12 diameter (3.66)		600 (182.88)	25 (0.0046)

Calculate the friction and shock losses if the quantity through the fan is 40,000 cfm (18.88 m^2/s).

Solution. To be precise, it is necessary to calculate for each of the airways the value of hydraulic radius R_H and X, the shock loss factor, and then calculate the equivalent length L_e for each shock loss condition. However, values from Table 11.6.5 are acceptable:

Shock Loss Condition	Equivalent Length L_e, ft (m)
1. Entrance	3 (0.9144)
2. Gradual expansion	1 (0.3048)
3. Obtuse bend, sharp	15 (4.5720)
4. Sharp bend, right	70 (21.3360)
5. Discharge (fan)	0 (0.0000)

Using Eq. 11.6.15, and $Q = Q \times 10^{-5} = 40{,}000 \times 10^{-5} = 0.4$ cfm units (18.87 m^3/s) calculate the head loss for each airway:

Airway	L ft	Le ft	A ft^2	O ft	K lb-min^2/ft^6	H_L in. water (Pa)
AB	600	3	113.10	37.7	25	0.0121 (3.011)
BC	2000	1	120	52	50	0.0931 (23.167)
CD	3000	15	120	52	50	0.1396 (34.738)
DE	600	70	113.10	37.7	25	0.0134 (3.335)
					ΣH_L =	0.2582 (64.251)

let resistance of mine = R_m

$$H = R_m Q^2$$

$$(0.258)^2 = R_m (0.4)^2$$

$$R_m = 1.6138 \text{ in.-min}^2/\text{ft}^6 \ (0.1803\ N\text{-}s^2/\text{m}^8)$$

$$AHP = \frac{0.2582 \times 40{,}000}{6350}$$

$$= 1.6265 \text{ hp } (1.21 \text{ kW})$$

11.6.4 CONTROL DEVICES

Two distinct airstreams—intake and return air—are generally recognized in mine ventilation, particularly in coal mines. The fresh airstream or *intake air* is the air from the surface that is conducted directly to the workings and other desired locations to provide fresh air to the miners and to dilute and carry away gases and dust from the workings. After this air has served this purpose, it becomes *return air* to be conducted back to the surface. In general, intake and return airstreams must not be allowed to mix. This is always true for coal mines.

Control devices in mine ventilation serve to (1) separate the intake and return airstreams in adjacent airways (stoppings), (2) allow the crossing of the intake and return streams without mixing (overcasts), and (3) regulate the flow of air through the various airways in the desired manner when the quantity has to be split between the airways (regulators).

11.6.4.1 Stoppings

Stoppings are physical barriers erected between intake and return airways to prevent the air flowing in them from mixing with each other. Stoppings can be temporary or permanent.

Temporary stoppings are often constructed of fire-resistant jute fabric, plastic, rough lumber covered with plastic, or even various types of sheet metal sections. These are extensively used in areas where frequent adjustments to air directions are necessary, such as in the working panels. Where the stoppings are constructed of concrete, gravel, cinder, or slag blocks, the blocks are usually wedged in place with dry joints. This permits fast erection and even allows for the recovery of the blocks.

Permanent stoppings are installed in places where a permanent or a long-term control of flow is needed, such as between the main intakes and returns. They should be substantially built so that they are airtight and, if they are functioning as seals or bulkheads, they must resist the disruptive forces of explosions. For the latter purpose, the stoppings must be built according to specifications of the Coal Mine Health and Safety Act of 1969 as given in Title 30, *Code of Federal Regulations* (Anon., 1988).

Fig. 11.6.3. Well-constructed explosion-proof stopping.

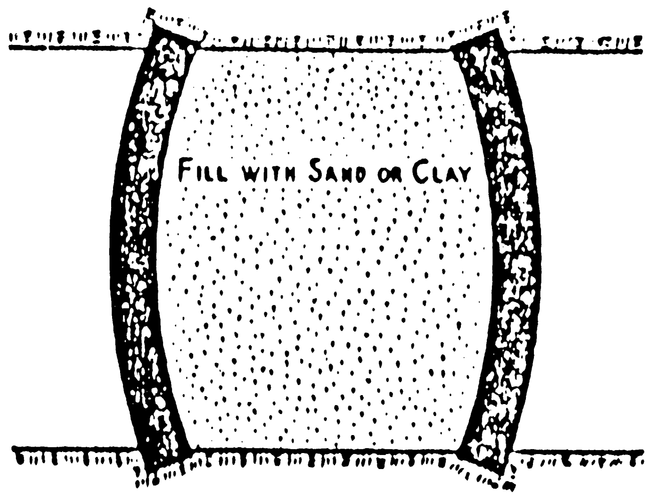


Fig. 11.6.4. Excellent construction for explosion-proof stopping.

Shown in Figs. 11.6.3 and 11.6.4 are two such constructions, the one in Fig. 11.6.4 being able to withstand greater pressures than the stopping in Fig. 11.6.3. The greater the distance between the two masonry walls, the more effective will be the stopping.

Stoppings utilizing such construction are rarely seen in mines. Most permanent stoppings generally consist of 8 by 8 by 16 in. (≅ 200 × 200 × 400 mm) solid or hollow blocks, laid with mortar joints. The entire block sides are then coated with cement or some type of block coating. The stopping is usually keyed into the roof, floor, and sides. It is common practice to provide a fire-resistant stringer or cushion across the top to prevent cracking due to strata pressure. Since stoppings must be accessible for inspection and repair, it is advisable that they be built where the roof and sides are secured. Additionally, gobbing rock against the stopping (Fig. 11.6.5) must be avoided as this makes both inspection and maintenance difficult.

A *door* is simply a hinged or movable partition within a stopping designed to permit the passage of men and equipment. Mine doors may be constructed of metal (required for doors between the intake and return) or of lumber covered with tarpaper, plastic, or other sealant material. They serve the same

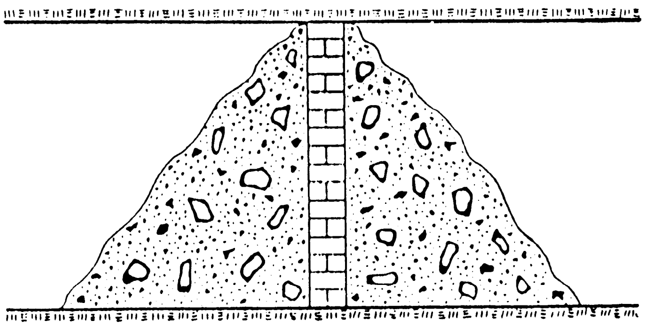


Fig. 11.6.5. Gobbing of rock against stopping.

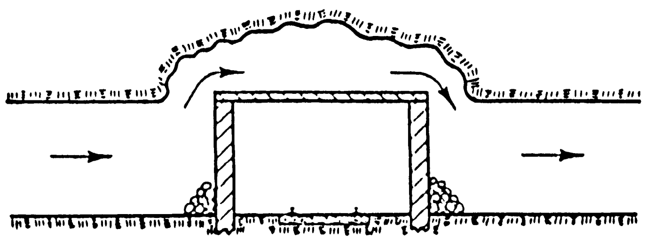


Fig. 11.6.6. Poor construction of overcast.

function as stoppings and are frequently used in haulageways. To avoid short circuiting between the return and intakes, doors should always be arranged in pairs to provide an *airlock*—one door will always be closed while the other is open. Automatic self-opening doors are especially useful along haulageways and equipment travelways. However, doors along the haulageways do not have sills, so that conveyor belting or brattice cloth should be attached to the bottom of the door.

Because the health and safety of underground employees may depend upon the ventilation doors, personnel should be trained in their proper use. They should be marked in bold letters, KEEP CLOSED or KEEP OPEN, so that they serve their ventilation purpose.

11.6.4.2 Overcasts

Overcasts or crossings are air bridges that allow the intake and return airways to cross one another without mixing. Undercasts are less frequently used since these are below the level of the surrounding openings, and water tends to accumulate in them. Shown in Fig. 11.6.6 is a very poorly constructed overcast. Shown in Fig. 11.6.7 is an ideal overcast with long sweeping curves at both the approach and discharge sides. The cross-sectional area of the overcast should be one-half the area of the approaching airway, if the approach is gradually contracted and the discharge is gradually expanded, as shown in Fig. 11.6.7.

The overcast should be properly built with high-quality material. In the area where an overcast is required, the roof is raised to the necessary height. Common practice where continuous miners are used is to ramp up at the spot where the overcast is to be located before returning to bench the coal (Stefanko, 1983). In constructing the overcast, solid blocks set in mortar should be used. The two wing walls must be anchored firmly in the ribs. Steel beams are then placed on the wing-walls over which the roof is built. Two walls are then built from this roof to the heightened roof line. An inexpensive but good-quality overcast,

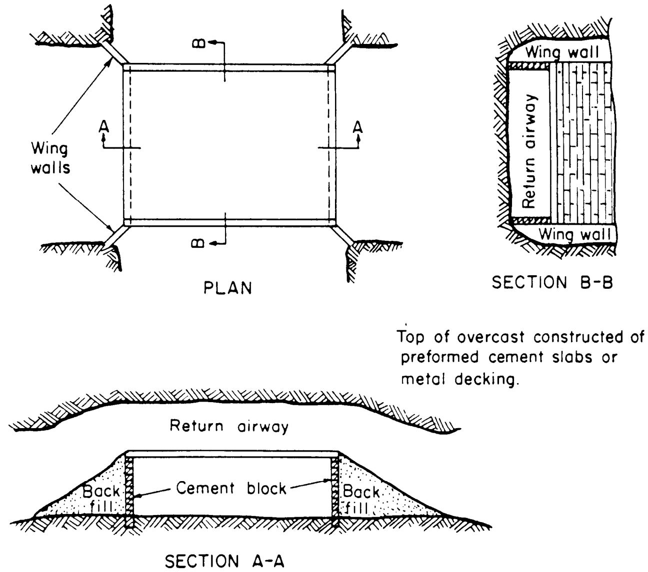


Fig. 11.6.7. Excellent construction for overcast.

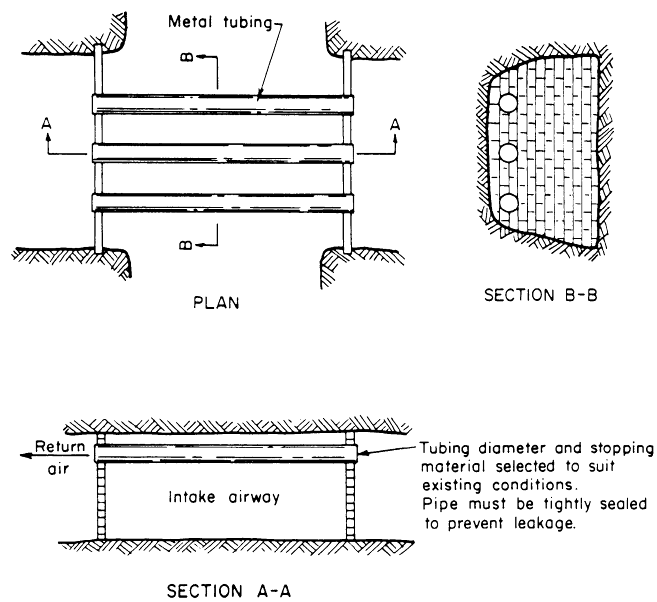


Fig. 11.6.8. Inexpensive but efficient overcast (Rock, Dalzell, and Harris, 1971).

most suitable for temporary use, is illustrated in Fig. 11.6.8. Here a solid (stopping) wall is built on either side of the airway, and a sufficient number of pipes of large diameter are laid on the top of the walls to carry the air across the main airway. It is possible to transport up to 25,000 cfm (11.8 m^3/s) with two 36-in. (914-mm) or three 30-in. (762-mm) pipes. The ends of the pipes should be embedded in the stopping and made air-tight.

11.6.4.3 Regulators

Regulators are used to control and redistribute the quantity of flow in each split of air. The regulator is an opening in a stopping in an airway and may be equipped with an adjustable

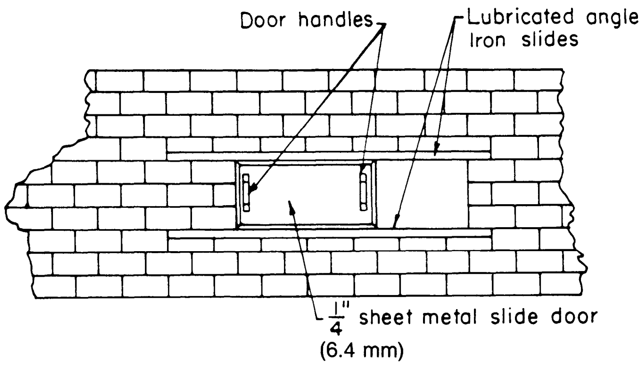


Fig. 11.6.9. Box-type regulator (Rock, Dalzell, and Harris, 1971).

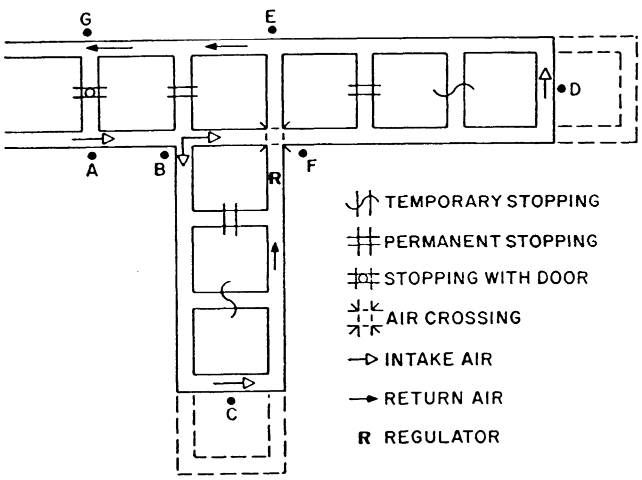


Fig. 11.6.10. Ventilation circuit with leakage sources.

or sliding door (Fig. 11.6.9). These are known as box regulators and are used almost exclusively in coal mines. Regulators should be easily accessible and located in places where roof and rib conditions are good. They should be kept clear of debris and obstructions. The ideal location for regulators in coal mines is in the return near the beginning of a ventilation split so as not to interfere with haulage and materials transport. In this location, it may not have to be moved for the life of the working section and it may be possible to use it even after the section is mined out or idled.

11.6.5 LEAKAGE CONSIDERATIONS

Consider a somewhat oversimplified diagram of a part of a mine ventilation system (Fig. 11.6.10). Fresh air to the workings is split at B. The roles of the stoppings, door, overcast, and regulators are evident. Stoppings are also built in the connecting entries between ABFD and DEG and in the connecting entries between BC and CF. The dotted lines show the mine advance projection; as the mine advances, the temporary stoppings will be replaced by permanent stoppings. The return from the workings at C must cross the intake in ABFD at F; an overcast is built at F to course the return air from C directly to E over the intake air to D. The two splits of air reunite at E. Since access to the return air may be needed, a door is built into the stopping in the connecting airway AG. A regulator R is placed in the return airway CF to regulate the flow into the workings at C. Since air always flows from a point of higher total head to one of lower total head, the head at every point on the intake side is higher than that at the return side. Consider the head across the stopping in the airway AG. On the side of the stopping towards A, the pressure head is higher than it is on the side of the stopping towards G. Because of the stopping, free flow of air from A to G is not possible. In practice, however, some air will flow through the cracks and crevices in the stopping due to the pressure difference. Such *unintended* losses of air directly to the return from the intake are known as *leakage.* Leakages occur, as explained before, through stoppings, overcasts, and doors. Leakages have also been known to occur through crushed pillars and improperly packed gob.

Leakage phenomena are complex, and the amount of leakage that occurs is a function of the pressure differential across the control devices, the construction and condition of the devices, and the area exposed. In the past, on the average, not more than 20% of the air quantity measured at the fans reached the working places. Conditions today are slightly better in that this figure is anywhere from 30 to 50%. Leakage through stoppings, doors, and regulators depends not only upon the pressure across the control device but also on the condition of the device itself. Ground pressures destroy control devices, and fugitive air losses become so great that it is almost impossible to provide adequate air at the face. Generally, leakage is most severe through old stoppings in the outby portion of the circuit. These are also subjected to higher pressure differences than the newer inby stoppings. Therefore, the circuit air quantity diminishes at a decreasing rate as the air progresses from outby to inby in the circuit. Painting and plastering a stopping decreases leakage, on the average, to around 2 or 3% of the leakage through a mortar-laid, unplastered, and unpainted stopping. The exact amount of leakage, however, is determined by how well the plastering and painting is done.

Kharkar, Ramani, and Stefanko (1974) carried out studies to determine leakage across stoppings under several different airflow conditions. Graphs were plotted for cumulative leakage against the number of stoppings (Fig. 11.6.11). The graphs show that the rate of air loss is variable over the length of the airway, the largest values being furthest from the working face with three-quarters of the total loss occurring in the first one-half of the air intake.

In ventilation planning, leakages have been handled by a system of rough allowances since they often cannot be measured accurately. There is very little information on leakage through overcasts, doors, and air locks. Table 11.6.6 lists some recommended figures for use in planning (Roberts, 1960). Where volume measurements can be made on either side of the leakage source, these measurements are not only more reliable than visual inspection, but may also serve as a base for estimating leakages. The leaking air does not help in the ventilation of the workings. Thus leakage is doubly disadvantageous—it does not improve health and safety conditions, and it increases the cost of getting the quantity of air needed into the main split. Leakage should be kept to a practical minimum.

11.6.6 AIRFLOW IN MINES

Two basic circuits or combinations of airways—series or parallel—are used to distribute air through the mine. A *mine ventilation network* is a combination of several series and parallel airways.

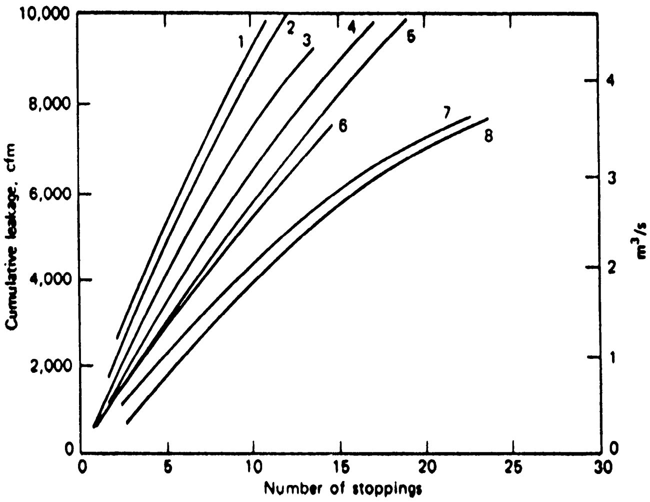


Fig. 11.6.11. Results of study to determine leakage vs. number of stoppings in several coal mines. (1) Mine 4, Section A: H = 1.7; L = 1530. (2) Mine 2, Section B: H = 2.8; L = 1520. (3) Mine 2, Section B: H = 2.3; L = 630. (4) Mine 3, Section A: H = 1.9; L = 1200. (5) Mine 2, Section C: H = .51; L = 1050. (6) Mine 4, Section B: H = .71; L = 990. (7) Mine 1, Section B: H = .78; L = 1360. (8) Mine 1, Section A: H = 1.32; L = 2125. L: Length in ft of roadway considered. H: Pressure drop in in. water across stoppings (*Source:* Kharkar, Ramani and Stefanko, 1974; Hartman, Mutmansky and Wang, 1982).

Table 11.6.6. Leakage Estimate in Coal Mines

1. Leakage across newly formed gobs (longwall faces)

Distance Between Intake and Return, ft	Leakage Across Gob as Percentage of the Air on the Face, %
150	20
300	10
600	5

No allowance made where solid stowing is done.

2. Separation doors: 3000 cfm
3. Overcasts: 3000 cfm
4. Surface Leakage: This is the air that leaks through the casing at the top of the upcast shaft. It is a function of fan head.

Fan Head in. water	Leakage cfm
5	25,000
10	35,000
15	45,000
20	50,000

5. Leakage through stoppings

   a. Explosion proof—None
   b. Temporary stoppings: 100–200 cfm dependent on the age, physical construction, maintenance, and head across the stopping.

Source: Roberts, 1960. Conversions Factors: 1 ft = 0.3048 m, 1 cfm = 0.47195 x 10^{-3} m^3/s, 1 in. water = 248.84 Pa.

11.6.6.1 Series Flow

In *series* combinations, the airways are connected end to end, and the same quantity flows through each of the airways. For a system of N airways in series, the following relationships hold:

$$Q = Q_1 = Q_2 = Q_3 = Q_4 = \cdots = Q_N \tag{11.6.22}$$

$$H_L = H_1 + H_2 + H_3 + H_4 + \cdots + H_N \tag{11.6.23}$$

$$R = R_1 + R_2 + R_3 + \cdots + R_N \tag{11.6.24}$$

where Q_i, H_i, and R_i are quantity, head, and resistance of the ith airway, and Q, H_L, and R are the quantity, head, and resistance of the system, respectively.

11.6.6.2 Parallel Flow

In *parallel* flow, all airways start at the same point and end at the same point. Therefore, the pressure difference between the ends of each airway is the same. For a system of N airways in parallel, the following relationships hold:

$$Q = Q_1 + Q_2 + Q_3 + \cdots + Q_N \tag{11.6.25}$$

$$H_L = H_1 = H_2 = H_3 = \cdots = H_N \tag{11.6.26}$$

$$\frac{1}{\sqrt{R}} = \frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + \frac{1}{\sqrt{R_3}} + \cdots + \frac{1}{\sqrt{R_N}} \tag{11.6.27}$$

Also the following relationships for quantity distribution in the parallel airways can be developed. This quantity distribution is called *natural splitting:*

$$Q_1 = Q\left(\frac{R}{R_i}\right)^{\frac{1}{2}}$$

or

$$\frac{Q_i}{Q_j} = \left(\frac{R_j}{R_i}\right)^{\frac{1}{2}} \tag{11.6.28}$$

In natural splitting, the highest quantity flows in the lowest resistance split; quantity flow in a split with higher resistance is lower than that in a lower-resistance split.

Example 11.6.6. Given three airways, 1, 2, and 3, in series, Q = 20,000 cfm (9.44 m^3/s), with head losses H_{L_1} = 2 in. water (497.68 Pa), H_{L_2} = 1 in. water (248.84 Pa), and H_{L_3} = 3 in. water (746.52 Pa). Find Q, *H,* and *R.*

Solution.

$$Q = Q_1 = Q_2 = Q_3 = 20{,}000 \text{ cfm } (9.44 \text{ m}^3/\text{s})$$

$$H = H_1 + H_2 + H_3 = 2 + 1 + 3 = 6 \text{ in. water } (1493.04 \text{ Pa})$$

$$R = H/Q^2 = 6/(0.2)^2 = 150 \times 10^{-10} \text{ in.-min}^2/\text{ft}^6 \quad (16.755\ N\text{-}s^2/\text{m}^8)$$

$$\text{[Check: } R_1 = \frac{2}{(0.02)^2} = 50 \times 10^{-10} \text{ in.-min}^2/\text{ft}^6;$$

$$R_2 = \frac{1}{(0.02)^2} = 25 \times 10^{-10} \text{ in.-min}^2/\text{ft}^6;$$

$$R_3 = \frac{3}{(0.2)^2} = 75 \text{ in.-min}^2/\text{ft}^6;$$

$$R = R_1 + R_2 + R_3 = 150 \times 10^{-10} \text{ in.-min}^2/\text{ft}^6\text{]}$$

Example 11.6.7. Given the three airways in the previous example with resistances $R_1 = 50 \times 10^{-10}$ (5.585), $R_2 = 25 \times 10^{-10}$ (2.793), and $R_3 = 75 \times 10^{-10}$ (8.378) in in.-min^2/ft^6 (N-s^2/m^8), arranged in parallel, with the total Q = 100,000 cfm (47.195 m^3/s); find system H and R and the quantities Q_1, Q_2, and Q_3.

Solution.

$$\frac{1}{\sqrt{R}} = \frac{1}{\sqrt{50}} + \frac{1}{\sqrt{25}} + \frac{1}{\sqrt{75}} = 0.141 + 0.200 + 0.116$$

$$= 0.457$$

$$R = 4.79 \times 10^{-10} \text{ in.-min}^2/\text{ft}^6 \ (0.535\ Ns^2/\text{m}^8)$$

$$H = RQ^2 = (4.79)(1.00)^2 = 4.79 \text{ in. water (1.192 kPa)}$$

$$Q_1 = Q\left(\frac{R}{R_1}\right)^{\frac{1}{2}} = 100{,}000\left(\frac{4.79}{50}\right)^{\frac{1}{2}} = 30{,}952 \text{ cfm } (14.61 \text{ m}^3/\text{s})$$

$$Q_2 = 100{,}000\left(\frac{4.79}{25}\right)^{\frac{1}{2}} = 43{,}772 \text{ cfm } (20.66 \text{ m}^3/\text{s})$$

$$Q_3 = 100{,}000\left(\frac{4.79}{75}\right)^{\frac{1}{2}} = 25{,}272 \text{ cfm } (11.93 \text{ m}^3/\text{s})$$

[Check: $Q = Q_1 + Q_2 + Q_3 \cong 100{,}000$ cfm (47.2 m^3/s)]

11.6.6.3 Controlled Splitting

When airways are connected in parallel and the quantity required in each airway is specified, natural splitting generally will not result in the required distribution.

Example 11.6.8. Suppose for the airways in Ex. 11.6.7 that the required air distribution was Q_1 = 25,000 cfm (11.80 m^3/s), Q_2 = 35,000 cfm (16.52 m^3/s), and Q_3 = 40,000 cfm (18.88 m^3/s). Calculate the head loss and determine the amount of regulation in each airway.

Solution.

(1) Head losses in the airways.

$$H_1 = R_1Q_1^2 = 50(0.25)^2 = 3.13 \text{ in. water (778.87 Pa)}$$

$$H_2 = R_2Q_2^2 = 25(0.35)^2 = 3.06 \text{ in. water (761.45 Pa)}$$

$$H_3 = R_3Q_3^2 = 75(0.40)^2 = 12.00 \text{ in. water (2986.08 Pa)}$$

To obtain 75,000 cfm (35.4 m^3/s) in airway 3, 12 in. water (2986.08 Pa) pressure difference between the ends of airway 3 is needed. In *controlled splitting,* the airway with the highest head loss (in this case, airway 3), is known as the *free split.* In the other two airways, since only 3.13 in. (778.87 Pa) and 3.06 in. (761.45 Pa) are needed for the desired flows, when a head of 12 in. water (2986.08 Pa) is developed across airway 3, regulators are needed to restrict the flow to 25,000 (11.80 m^3/s) and 35,000 cfm (16.52 m^3/s), respectively.

(2) Regulator shock loss. The regulators are, in effect, resistances added in series to the airway to increase the head loss from 3.13 in. water (778.87 Pa) to 12.00 in. water (2986.08 Pa) in airway 1, and from 3.06 in. water (761.45 Pa) to 12.00 in. water (2986.08 Pa) in airway 2. The resistances to be added in series must create shock losses that are calculated as follows:

Airway 1: 12.00 – 3.13 = 8.87 in. water (2207.21 Pa)
Airway 2: 12.00 – 3.06 = 8.94 in. water (2224.63 Pa)

Regulators create shock loss conditions under which 8.87 in. water (2207.21 Pa) and 8.94 in. water (224.63 Pa) are dissipated as shock losses in the respective airways.

A number of formulas are available to calculate the size of regulators. A simple but approximate formula, known as the *equivalent orifice* formula, is as follows (Kingery, 1960):

$$A = \frac{0.0004Q}{\sqrt{H_x}} \tag{11.6.29}$$

$$A = \frac{1.2Q}{\sqrt{H_x}} \tag{11.6.29a}$$

where A is the area of the regulator in ft^2 (m^2), Q is quantity in the airway in cfm (m^3/s), and H_x is shock loss to be dissipated in the regulator in in. water (Pa).

Example 11.6.9. Calculate the areas of the regulators for airways 1 and 2 of Ex. 11.6.8.

Solution. The areas A_1 and A_2, respectively, of the regulators in airways 1 and 2 are calculated as follows:

$$A_1 = \frac{0.0004 \times 25{,}000}{\sqrt{8.87}} = 3.36 \text{ ft}^2 \ (0.312 \text{ m}^2)$$

$$A_2 = \frac{0.0004 \times 35{,}000}{\sqrt{8.94}} = 4.68 \text{ ft}^2 \ (0.435 \text{ m}^2)$$

In practice, a box-type regulator is placed in a stopping in the airway and the opening is adjusted until the desired quantity flow is obtained. For example, in airway 1, the regulator may be a 2 by 2 ft (0.186 by 0.186 m) opening, and in airway 2, 2.5 by 2 ft (0.232 by 0.186 m) opening. Based on flow conditions, the regulator size can be adjusted. There are alternative methods of ensuring the required distribution, including decreasing the resistance of airway 3 or by incorporating pressure sources (booster fans) in each airway to generate the required heads.

11.6.6.4 Mine Ventilation Networks

A mine ventilation network is defined as a *simple network* if the series and parallel airways can be combined through the equivalent resistance formulas into one airway with a resistance equal to the network resistance. Otherwise, the network is *complex.* Solutions for the flow and head for large ventilation networks, whether simple or complex, can be obtained through the use of computer programs that apply the mathematical theory of networks and physical laws of mass and energy conservation to solve the head-quantity distribution problem (Chapter 11.10; Wang, 1982; Ramani, 1988).

Example 11.6.10. Consider the ventilation of a mine with three production units (Fig. 11.6.12). AB is the intake shaft and is used as a man-materials shaft. EF is a return shaft used exclusively for ventilation purposes.

Characteristics of AB: $K = 35 \times 10^{-10}$ lb-min^2/ft^4(0.0065 kg/m^3); L = 1500 ft (457.2 m); cross section: rectangular, 16 × 6 ft (4.88 × 1.83 m)

Characteristics of EF: $K = 21 \times 10^{-10}$ lb-min^2/ft^4(0.0039 kg/m^3); L = 1000 ft (304.8 m); cross section: rectangular, 20 × 5 ft (6.1 m × 1.52 m)

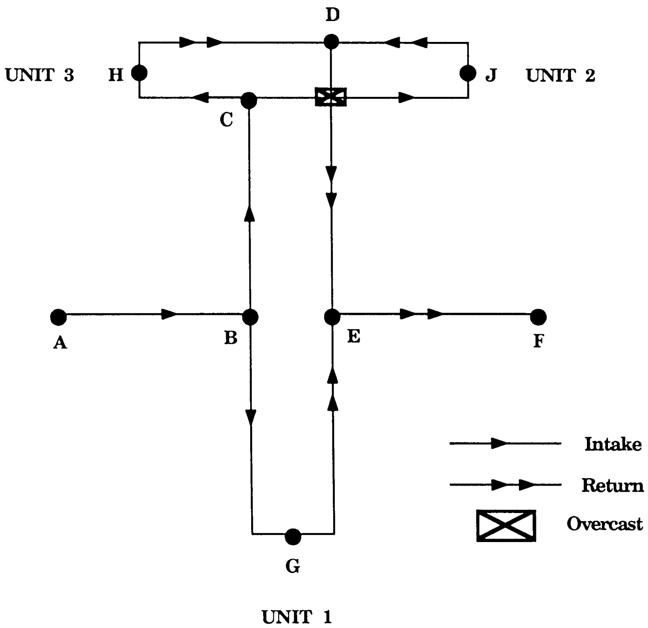


Fig. 11.6.12. Line diagram of mine ventilation system (Ex. 11.6.10).

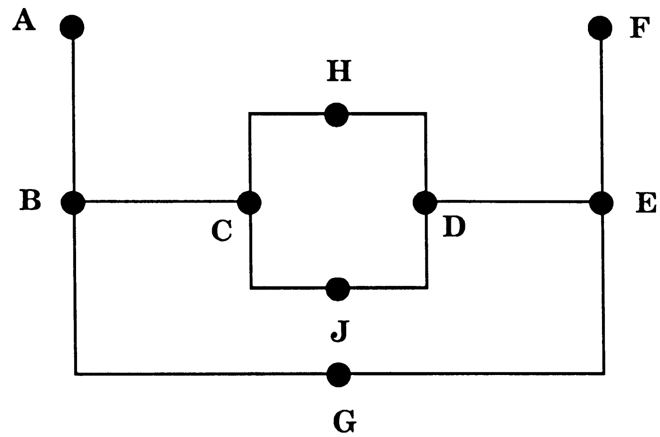


Fig. 11.6.13. Network schematic of mine ventilation system (Ex. 11.6.10).

The resistances in in.-min^2/ft^6 (N-s^2/m^8) of the other airways are as follows:

$$BGE = 2.0 \times 10^{-10}\ (0.2234) \quad CHD = 1.0 \times 10^{-10}\ (0.1117)$$
$$BC = 0.3 \times 10^{-10}\ (0.0335^8) \quad CJD = 2.0 \times 10^{-10}\ (0.2234)$$
$$DE = 0.36 \times 10^{-10}\ (0.0402)$$

Exactly 100,000 cfm (47.195 m^3/s) of air is flowing down the shaft AB.

(a) Calculate the following: the quantity flowing in airways *CHD* and *CJD* and head loss from A to F under natural splitting conditions.

(b) If the quantity required in the airways *CHD* and *CJD* is 40,000 cfm (18.878 m^3/s) each, and the total quantity entering the mine is 100,000 cfm (47.195 m^3/s) as before, calculate the head loss from A to F and the air horsepower. Also identify the free split and the required regulation airways.

Solution. First, draw a network schematic of the ventilation system (Fig. 11.6.13), and then calculate the resistances and proceed with reducing the network to a single airway.

$$R_{AB} = \frac{KLO}{5.2A^3} = \frac{35 \times 1500 \times 2(16 + 6)}{5.2 \times (96)^3}$$
$$= 0.5 \text{ in.-min}^2/\text{ft}^6\ (0.0559\ N\text{-}s^2/\text{m}^8)$$
$$R_{EF} = \frac{21 \times 1000 \times 2(20 + 5)}{5.2 \times (100)^3}$$
$$= 0.25 \text{ in.-min}^2/\text{ft}^6\ (0.0279\ N\text{-}s^2/\text{m}^8)$$
$$\frac{1}{\sqrt{R_{CD}}} = \frac{1}{\sqrt{R_{CHD}}} + \frac{1}{\sqrt{R_{CJD}}} = \frac{1}{\sqrt{1}} + \frac{1}{\sqrt{2}} = 1.71$$
$$R_{CD} = 0.34 \text{ in.-min}^2/\text{ft}^6\ (0.038\ N/s^2/\text{m}^8)$$
$$R_{BCDE} = R_{BC} + R_{CD} + R_{DE} = 0.3 + 0.34 + 0.36$$
$$= 1.0 \text{ in.-min}^2/\text{ft}^6\ (0.1117\ N\text{-}s^2/\text{m}^8)$$
$$\frac{1}{\sqrt{R_{BE}}} = \frac{1}{\sqrt{\mathrm{R}_{BCDE}}} + \frac{1}{\sqrt{R_{BGE}}} = \frac{1}{\sqrt{1}} + \frac{1}{\sqrt{2}} = 1.71$$
$$R_{BE} = 0.34 \text{ in.-min}^2/\text{ft}^6\ (0.038\ N\text{-}s^2/\text{m}^8)$$
$$R = R_M = R_{AB} + R_{BE} + R_{EF} = 0.5 + 0.34 + 0.25$$
$$= 1.09 \text{ in.-min}^2/\text{ft}^6\ (0.1218\ N\text{-}s^2/\text{m}^8)$$
$$H_S = R_M Q^2 = 1.09(1.0)^2 = 1.09 \text{ in. water (271.24 Pa)}$$
$$Q_{BCDE} = Q\left(\frac{R_{BE}}{R_{BCDE}}\right)^{\frac{1}{2}} = 100{,}000\left(\frac{0.34}{1.00}\right)^{\frac{1}{2}}$$
$$= 58{,}310 \text{ cfm } (27.52 \text{ m}^3/\text{s})$$
$$Q_{BGE} = \left(\frac{R_{BE}}{R_{BGE}}\right)^{\frac{1}{2}} = 100{,}000\left(\frac{0.34}{2.0}\right)^{\frac{1}{2}}$$
$$= 41{,}230 \text{ cfm } (19.46 \text{ m}^3/\text{s})$$

Check: total $Q \cong 100{,}000$ cfm (47.2 m^3/s)

$$Q_{CHD} = Q_{BCDE}\left(\frac{R_{CD}}{R_{CHD}}\right)^{\frac{1}{2}} = 58{,}310\left(\frac{0.34}{1.0}\right)^{\frac{1}{2}}$$
$$= 34{,}000 \text{ cfm } (16.05 \text{ m}^3/\text{s})$$
$$Q_{CJD} = Q_{BCDE}\left(\frac{R_{CD}}{R_{CJD}}\right)^{\frac{1}{2}} = 58{,}310\left(\frac{0.34}{2.0}\right)^{\frac{1}{2}}$$
$$= 24{,}042 \text{ cfm } (11.35 \text{ m}^3/\text{s})$$

Check: $Q_{BCDE} = 34{,}000 + 24{,}042 \cong 58{,}310$ (27.52 m^3/s)

The air distribution is summarized in Fig. 11.6.14.

CONTROLLED SPLITTING. In the network of the mine, first the quantities of flow in the various airways are calculated on the basis of the given data and the requirement that the quantity entering a junction must leave the junction (also known as Kirchhoff's first law). Secondly, the head loss for each airway is calculated using the airway resistance and the quantity flowing in it. The network is shown with the quantity and head loss in each airway (Fig. 11.6.15). The regulators required in branches that constitute parallel paths between two junctions is calculated

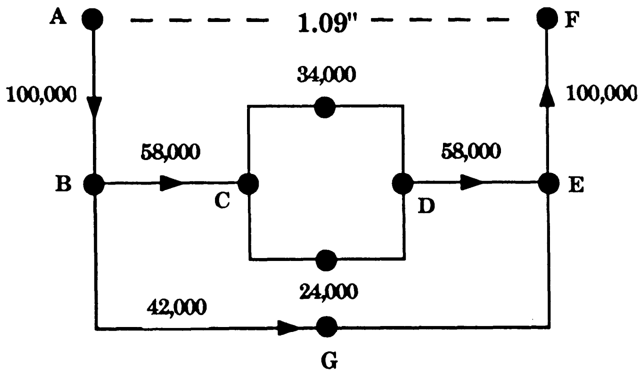


Fig. 11.6.14. Summary of air distribution under natural splitting conditions (Ex. 11.6.10). Conversion factors: 1 cfm = 0.47195 × 10^{-3} m^3/s, 1 in. water = 248.84 Pa.

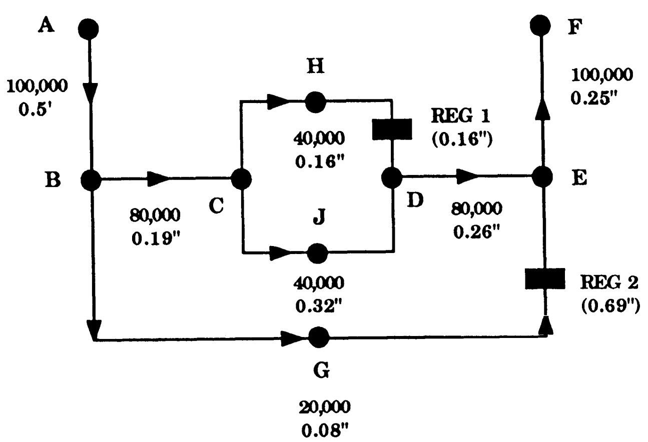


Fig. 11.6.15. Controlled splitting flow conditions (Ex. 11.6.10). Conversion factors: 1 cfm = 0.47195 × 10^{-3} m^3/s, 1 in. water = 248.84 Pa.

applying the requirement that the head loss across parallel splits must be the same (also known as Kirchhoff's second law). The splits are balanced starting from the innermost parallel splits and working to outer parallel splits.

Note that *CHD* and *CJD* are parallel airways and *CJD* is the airway with the higher pressure drop. *CHD* is therefore regulated (shown as REG1). Also *BCDE* and *BGE* are parallel airways. The pressure drop across *BCDE* is 0.19 + 0.32 + 0.26 = 0.77 in. water (191.61 Pa), whereas across *BGE* it is only 0.08 in. water (19.91 Pa). Therefore, *BGE* is regulated (shown as REG2) to lose (0.77 – 0.08) = 0.69 in. water (171.70 Pa). The only unregulated split from A to F consists of the airways AB, BC, CJD, DE, and EF. This is the free split, and the head loss in the mine is the sum of the head losses in the airways constituting the free split, which is 1.52 in. water (378.24 Pa). Finally,

$$AHP = \frac{HQ}{6350} = \frac{(1.52)(100{,}000)}{6350} \cong 23.94 \text{ hp } (17.85 \text{ kW})$$

11.6.7 MINE HEADS

By definition, three mine pressure heads are recognized: the static head H_{S_M}, the velocity head H_{V_D}, and the total head H_{T_M}. The *static head* is the sum of the friction and shock losses in the free split of the mine. The *velocity head* of the mine is the head due to the velocity of the air at the discharge of a mine. The *total head* is the sum of the static and velocity heads of the mine:

$$H_{S_M} = \sum_{i=1}^{N} (H_f + H_x)_i \tag{11.6.30}$$

$$H_{V_M} = w\left(\frac{V_D}{1098}\right)^2 \tag{11.6.31}$$

$$H_{V_M} = \frac{wV_D{}^2}{2g} \tag{11.6.31a}$$

$$H_{T_M} = H_{S_M} + H_{V_M} \tag{11.6.32}$$

where N is number of sections (or airways) in the free split, $(H_f + H_x)_i$ is friction and shock loss in section (or airway) i in in. water (Pa), and V_D is velocity at the discharge of the mine in fpm (m/s).

Example 11.6.11. In Ex. 11.6.10, consider that the return shaft is open to the atmosphere. Calculate the mine heads.

Solution. The velocity of the air at discharge

$$V_D = \frac{\text{Quantity discharged (cfm)}}{\text{Area at discharge (ft}^2\text{)}} = \frac{100{,}000}{100}$$
$$= 1000 \text{ fpm } (5.08 \text{ m/s})$$

Assuming standard air density,

$$H_V = w\left(\frac{V_D}{1098}\right)^2 = 0.0750\left(\frac{1000}{1098}\right)^2$$
$$= 0.0620 \text{ in. water } (15.43 \text{ Pa})$$

The static head of the mine (H_{S_M}) for the controlled splitting case has already been calculated as 1.520 in. water (378.24 Pa). Hence, the total mine head,

$$H_{T_M} = H_{S_M} + H_{V_M} = 1.520 + 0.062$$
$$= 1.582 \text{ in. water } (393.66 \text{ Pa})$$

11.6.7.1 Mine Characteristic Curves

The plotting of the mine head (static, total, or both) vs. the quantity results in curves called *mine characteristic curves.* Given

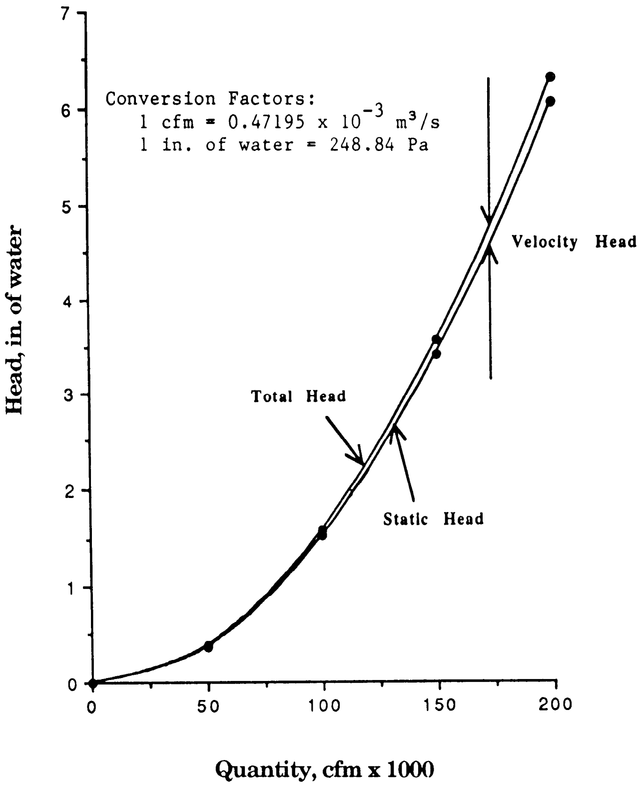


Fig. 11.6.16. Mine static and total head characteristics (Ex. 11.6.11).

one point on the characteristic (say, H_1, Q_1), the points on the characteristic can be calculated for any other quantity (say, Q_2) by repeated application of the head loss formula, $H = RQ^2$:

$$H_1 = RQ_1^2$$

$$H_2 = RQ_2^2$$

$$\frac{H_1}{H_2} = \left(\frac{Q_1}{Q_2}\right)^2$$

$$H_2 = H_1\left(\frac{Q_2}{Q_1}\right)^2 \tag{11.6.33}$$

Consider Ex. 11.6.11. The static and total heads for 100,000 cfm (47.12 m^3/s) are 1.52 (378.24 Pa) and 1.58 in. water (393.66 Pa). The mine characteristic curves for this mine can be calculated as follows:

Quantity, cfm (m^3/s)	Static Head, in. water (Pa)	Total Head, in. water (Pa)
0	0	0
50,000 (23.600)	0.38 (94.56)	0.40 (99.54)
100,000 (47.195)	1.52 (378.24)	1.58 (393.67)
150,000 (70.793)	3.42 (859.03)	3.56 (885.87)
200,000 (94.390)	6.08 (1512.95)	6.33 (1575.16)

Mine characteristics are useful to visualize the solutions to ventilation problems (Fig. 11.6.16) and to facilitate the selection of fans for mine ventilation systems.

11.6.8 MINE FANS

Fans provide the energy required to overcome the resistance to flow. They are mechanical devices which convert mechanical energy to fluid energy. By increasing the pressure of the air flowing through it, a fan creates a pressure difference in the system that causes the air to flow and the flowing air to overcome the pressure losses in the system. There are two main kinds of fans: *centrifugal* and *axial* flow. While centrifugal fans are still used in some mines and in other countries, the majority of modern mine fans in the United States are axial flow fans. Airflow established by fans is termed *mechanical ventilation.*

Measures of fan performance include the range of head and quantity delivered, power requirements, efficiency, noise characteristics, ease of changing operating points, etc. While most important among these is the head-quantity relationship, efficiency and noise characteristics are gaining importance from the economic and environmental points of view (Wang, 1982). Fan characteristics can be developed from the fundamentals using the theories of fluid dynamics. However, the practical or the actual characteristics—the useful output of fans—cannot be developed from theory. Instead, the actual head-quantity performance of a fan is developed empirically from fan test data of similar fans through the application of fan laws.

As with mine heads, three kinds of fan heads are defined. The *total head* H_{T_F} of a fan is the difference between the total head at fan outlet and the total head at the fan inlet. The *velocity head* H_{V_F} of the fan is the head due to the velocity at the discharge of the fan (fan outlet). The *static head* H_{S_F} of the fan is the difference between the fan total head and fan velocity head.

Mathematically,

$$H_{T_F} = \text{fan outlet total head} - \text{fan inlet total head} \tag{11.6.34}$$

$$H_{V_F} = \text{velocity head at the fan outlet} \tag{11.6.35}$$

$$H_{S_F} = H_{T_F} - H_{V_F} \tag{11.6.36}$$

Fan efficiency can be defined in several ways. The *static* η_s and *total efficiency* η_t of the fan are based on the static P_{as} and total power output P_{at} of the fan, respectively, and the power input to the fan P_F. P_{as} and P_{at} can be calculated using Eqs. 11.6.2 or 11.6.21a. P_F is measured during a fan test.

$$\eta_s = P_{as}/P_F \tag{11.6.37}$$

$$\eta_t = P_{at}/P_F \tag{11.6.38}$$

The *overall efficiency* η_o is the ratio of the total power output of the fan to the power input P_I supplied to the motor driving the fan:

$$\eta_o = (P_{at}/P_I) \tag{11.6.39}$$

The graphical method is the most useful way of presenting the performance curves of a fan. These curves are provided by the manufacturers and are for standard density conditions. The design type, diameter, and speed of the fan for which the characteristics are valid are specified. It is customary to plot the quantity as the independent variable with head (usually static), power, and efficiency as the dependent variables (Fig. 11.6.17). In most

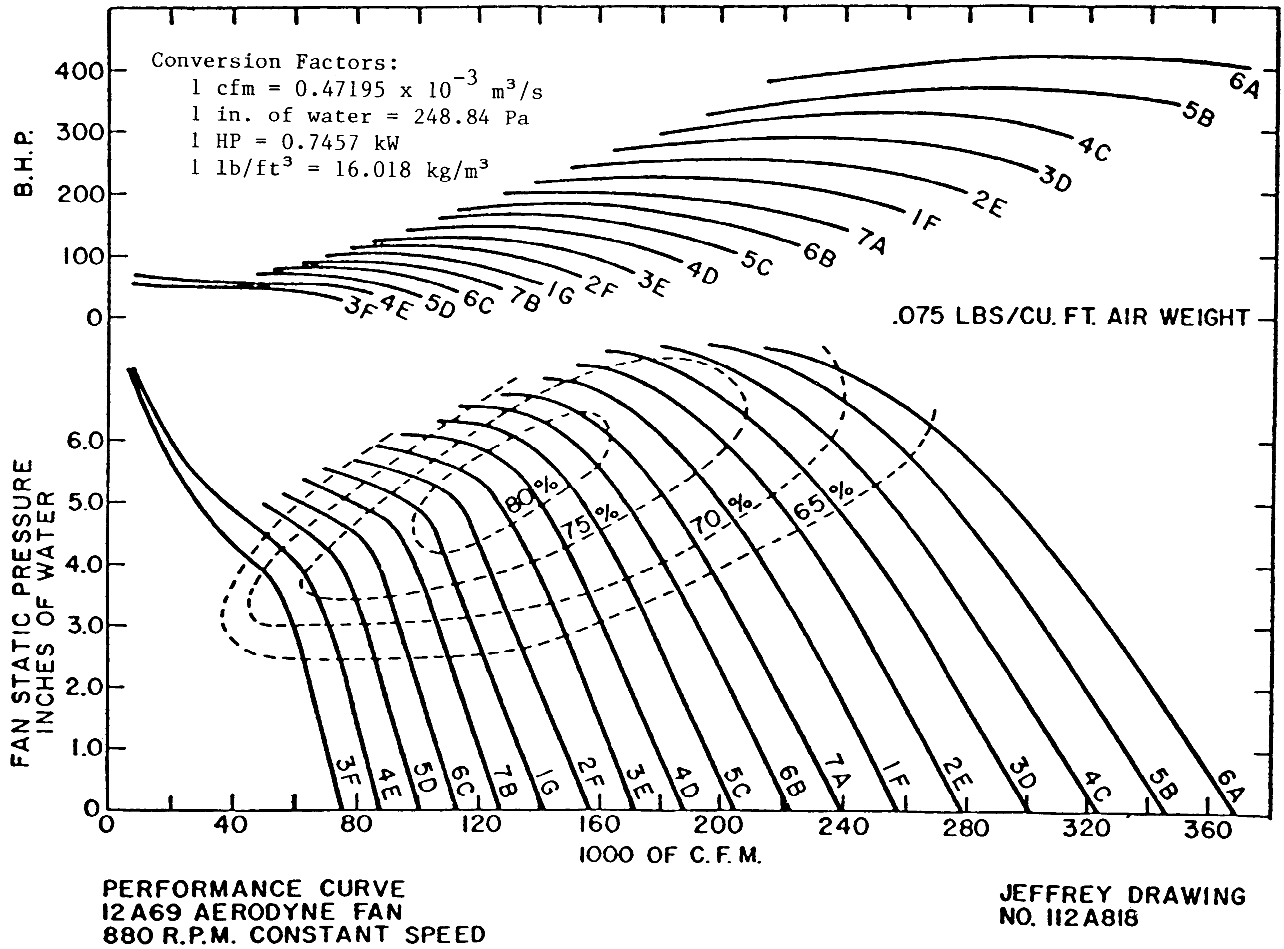


Fig. 11.6.17. Axial flow fan characteristics for various blade settings (*courtesy:* Jeffrey Mining Machine Division, Dresser Industries, Inc., Cumberland, OH).

mine ventilation calculations, the static head characteristics of the fans are used. As long as the fan velocity head is small or the difference between fan and mine velocity heads is small, this is acceptable. However, for precise calculations, total head characteristics should be used.

11.6.8.1 Fan Laws

Generalized. Fan rotational speed n, fan size (diameter) D, and air density w affect the performance of a fan. For geometrically similar fans and for a particular point of operation on the head quantity characteristic, the following relationships are valid:

$$\text{Quantity, } Q \propto nD^3 \tag{11.6.40}$$

$$\text{Head, } H \propto n^2D^2w \tag{11.6.41}$$

$$\text{Power, } P_m \propto n^3D^5w \tag{11.6.42}$$

The *fan laws* are derived from the above equations and are useful to predict the performance of the fans under new conditions of speed, size, and density.

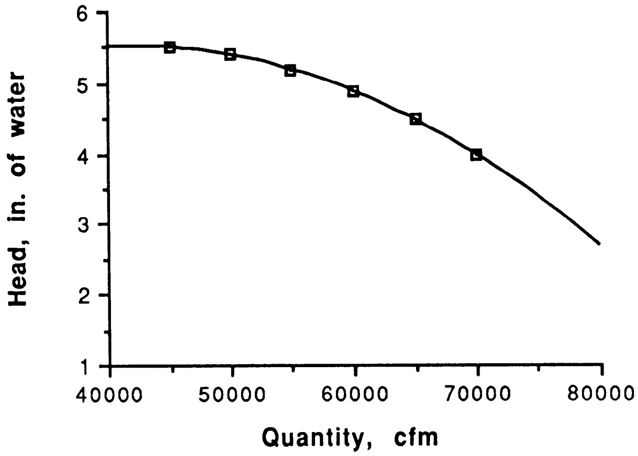
Fan Performance Variable	Law 1 Speed (n) changes w, D constant	Law 2 Size (D) changes w, nD constant	Law 3 Density (w) changes nD constant
Quantity, Q	directly	as square	no change
Head, H_{S_F}, H_{T_F}	as square	no change	directly
Power, P_a, P_F	as cube	as square	directly
Efficiency, η	no change	no change	no change

With axial flow fans, changes in the performance characteristics are obtained by varying the pitch of the blades over a limited range.

Example 11.6.12. The head-quantity characteristic of a 48-in. (1.22-m) fan operating at 1200 rpm and density of 0.0750 lb/ft^3 (1.2000 kg/m^3) is given below and in Fig. 11.6.18:

Point	1	2	3	4	5	6
H (in. water)	5.5	5.4	5.2	4.9	4.5	4.0
(kPa)	1.369	1.344	1.294	1.219	1.120	0.995
Q (cfm)	45,000	50,000	55,000	60,000	65,000	70,000
(m^3/s)	21.24	23.60	25.96	28.32	30.68	33.04

Calculate and plot the new head-quantity characteristic for each of the following conditions:



Conversion Factors:
1 cfm = 0.47195 x 10^{-3} m^3/s
1 in. of water = 248.84 Pa

Fig. 11.6.18. Head-quantity characteristic for fan (Ex. 11.6.12).

a. Speed is reduced to 800 rpm; *w* and *D* constant.

b. Diameter is increased to 60 in. (1.524 m); *nD* and *w* constant. What will be the new speed of the fan?

c. Density decreases to 0.0700 lb/ft^3 (1.12 kg/m^3); *nD* constant.

d. Speed, diameter and density are changed to 800, 60 in. (1.524 m) and 0.0700 lb/ft^3 (1.12 kg/m^3) simultaneously.

Solution. Using the subscript *O* for given conditions and *N* for new conditions, the following relationships are established.

a. *Speed change:* $Q_N = Q_O\left(\frac{N_N}{N_O}\right)$; $H_N = H_O\left(\frac{N_N}{N_O}\right)^2$

$$\frac{N_N}{N_O} = \frac{800}{1200} = 0.67; \left(\frac{N_N}{N_O}\right)^2 = 0.44$$

New Points	1	2	3	4	5	6
H_N (in. water)	2.42	2.38	2.29	2.16	1.98	1.76
(kPa)	0.602	0.592	0.570	0.537	0.493	0.438
Q_N (cfm)	30,150	33,500	36,850	40,200	43,550	46,900
(m^3/s)	14.23	15.81	17.39	18.97	20.55	22.13

b. *Diameter change:* $Q_N = Q_O\left(\frac{D_N}{D_O}\right)^2$, $H_N = H_O$

Constant tip speed: $D_N \times N_N = D_O \times N_O$

$$N_N = \frac{D_O \times N_O}{D_N}$$

$$\frac{D_N}{D_O} = \frac{60}{48} = 1.25; \left(\frac{D_N}{D_O}\right)^2 = 1.56; N_N = \frac{48 \times 1200}{60}$$
$$= 960 \text{ rpm}$$

New Points	1	2	3	4	5	6
H_N (in. water)	5.5	5.4	5.2	4.9	4.5	4.0
(kPa)	1.369	1.344	1.294	1.219	1.120	0.995
Q_N (cfm)	70,200	78,000	85,800	93,600	101,400	109,200
(m^3/s)	33.13	36.81	40.49	44.17	47.86	51.54

c. *Density change:* $Q_N = Q_O$, $H_N = H_O\left(\frac{w_N}{w_O}\right)$

$$\frac{w_N}{w_O} = \frac{0.0700}{0.0750} = 0.93$$

New Points	1	2	3	4	5	6
H_N (in. water)	5.12	5.02	4.84	4.56	4.19	3.72
(kPa)	1.274	1.249	1.204	1.135	1.043	0.926
Q_N (cfm)	45,000	50,000	55,000	60,000	65,000	70,000
(m^3/s)	21.24	23.60	25.96	28.32	30.68	33.04

d. *All changes occur simultaneously:*

$$Q_N = Q_O\left(\frac{N_N}{N_O}\right)\left(\frac{D_N}{D_O}\right)^3 = Q_O\left(\frac{800}{1200}\right)\left(\frac{60}{48}\right)^3 = 1.30\,Q_O$$

$$H_N = H_O\left(\frac{N_N}{N_O}\right)^2\left(\frac{D_N}{D_O}\right)^2\left(\frac{w_N}{w_O}\right) = H_O\left(\frac{800}{1200}\right)^2\left(\frac{60}{48}\right)^2\left(\frac{0.0700}{0.0750}\right)$$
$$= 0.65\,H_O$$

New Points	1	2	3	4	5	6
H_N (in. water)	3.58	3.51	3.38	3.19	2.93	2.60
(kPa)	0.891	0.873	0.841	0.794	0.729	0.647
Q_N (cfm)	58,500	65,000	71,500	78,000	84,500	91,000
(m^3/s)	27.61	30.68	33.74	36.81	39.88	42.95

The original characteristic and characteristics for the four cases are shown in Fig. 11.6.19.

Specialized.

In addition to the conventional fan characteristic curves, fan manufacturers provide special characteristic curves to aid in the initial selection of fans. In these curves, the performances of an entire series of fans is summarized using special parameters. For example, two such parameters are *specific speed* n_s and *specific volume* Q_s, defined as

$$n_s = \frac{Dn}{\sqrt{H}} \qquad (11.6.43)$$

$$Q_s = \frac{Q}{D^2\sqrt{H}} \qquad (11.6.44)$$

For constant density conditions, *H* and *Q* are the head and quantity developed by a fan with diameter *D* and rotational speed *n*. In Fig. 11.6.20 and Fig. 11.6.21 are shown typical specific volume-specific speed curves from two fan manufacturers.

11.6.8.2 Simplified Fan Selection Charts

Over the life of the mine, the demands for head and quantity will vary. When the mine is fully developed and all the units are in full operation, the demand for head and quantity may stabilize somewhat with only small variations due to mine extensions and

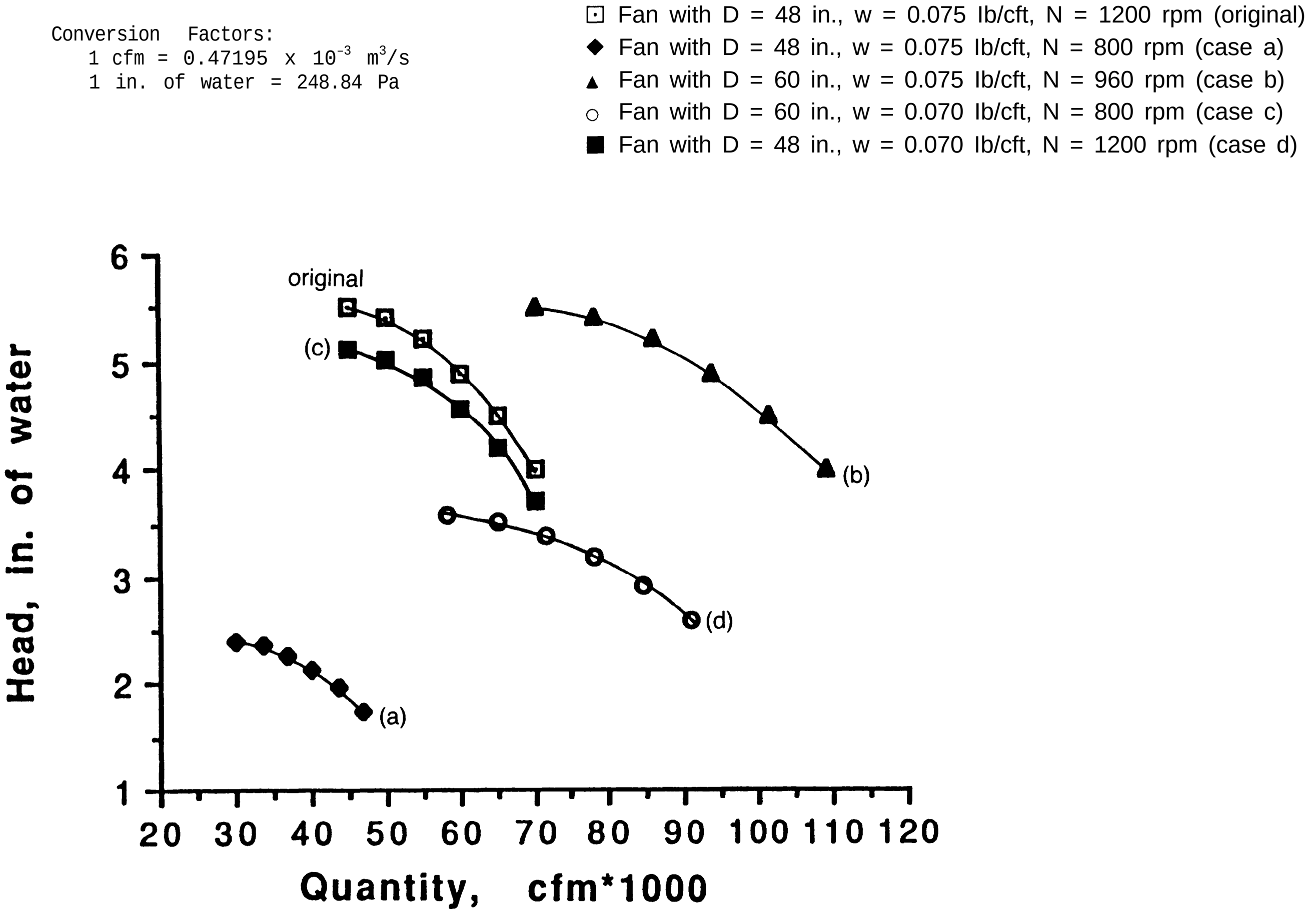


Fig. 11.6.19. Head-quantity characteristics for changing conditions (Ex. 11.6.12).

opening and closing of units. As the workings move away from the main fan shafts, as happens typically in coal mines, the duty on the fan may dramatically increase due to new operating sections or decrease due to addition of new shafts and fans. Fan selection procedure must consider these varying duties so that a fan with a range of heads and quantities at high efficiencies is selected. Further, multiple fans at different locations influence each other's performance and require careful selection to act with each other rather than act against each other. Some manufacturers provide simplified fan selection charts to aid in the selection of fans for given or anticipated mine conditions. These charts are useful for the initial selection of fans and for initial selection of fan curves for computer program inputs. Two such charts are shown in Figs. 11.6.22 and 11.6.23.

11.6.8.3 Fan Rating

An infinite number of combinations of pressure and volume can be obtained for a fan. *Fan rating,* on the other hand, is a statement of fan performance at one point of operation. Specifically, it is the head, quantity, power, and efficiency to be expected (fan diameter, speed, and air density remaining constant) when a fan is operating at peak mechanical efficiency. Manufacturers may provide a rating table with a fan that shows the complete range of capacities such as pressures, speeds, quantities, and horsepowers for a particular fan.

Since the static head developed by a fan cannot be mathematically calculated, the static head-quantity curve for a fan must be developed by running fan tests. A complete fan test provides the data for plotting its total and static head-quantity curves, and mechanical efficiency and horsepower curves (Mancha, 1940, 1942).

To test a fan, it must be operated at constant speed under varying conditions of resistance. Mancha (1940) described the procedures used by Jeffrey to rate their mine fans at the time that axial flow fans (as opposed to centrifugal fans) were coming into wide use in the United States. Most of the Jeffrey fans in use today were designed and their curves generated some 30 years ago, using a 48-in. (1.22-m) fan of each type. Characteristics for other sizes of fans were then developed through the application of fan laws. If at all possible, test arrangements and conditions, as well as all the curves developed, including total head-quantity characteristics, should be available to the ventilation engineer.

11.6.8.4 Fan and Mine Characteristics

When fan and mine characteristics are plotted on the same graph, they intersect, and their point of intersection is known as the *operating point.* Simply stated, when a fan is introduced into a mine circuit, the air quantity flowing through the mine and the head generated by the fan are completely determined by the fan and mine characteristics.

Consider the mine characteristic (B) shown in Fig. 11.6.24, and the fan characteristic (C) superimposed on it. These two characteristics intersect at the operating point (O). A fan whose

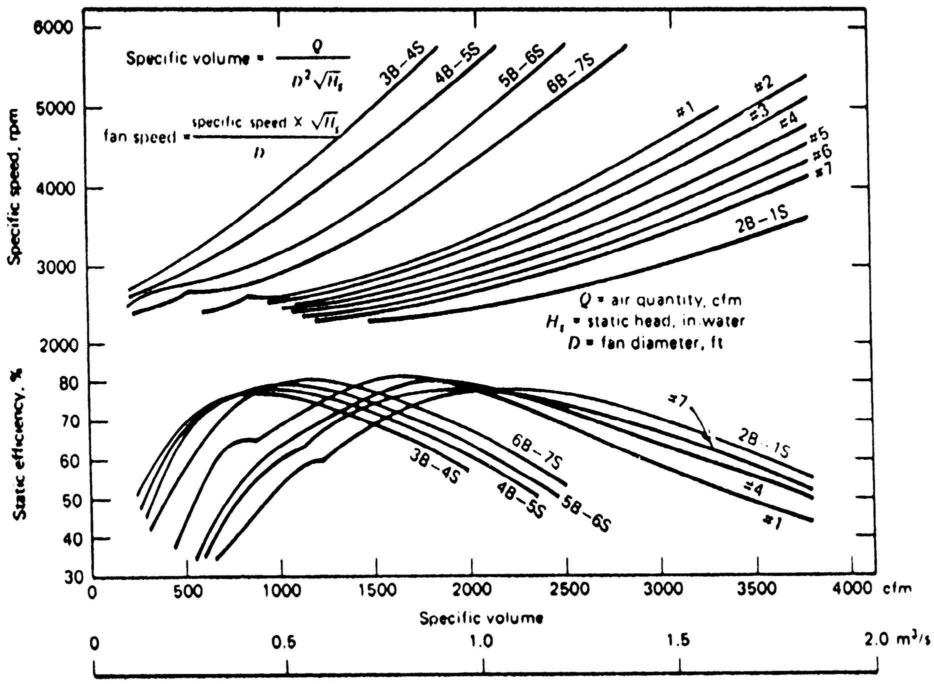


Fig. 11.6.20. Typical specific-volume vs. specific-speed plot for type 8H axial flow fans (*courtesy:* Jeffrey Mining Machine Division, Dresser Industries, Inc., Columbus, OH; Hartman, Mutmansky, and Wang, 1982).

head-quantity characteristic is defined by curve A will force through a mine for which the head-quantity characteristic is defined by curve B: 100,000 cfm (47.2 m^2/s) at 5 in. water (1.244 kPa). It should be noted that the true operating point is the point of intersection of the total head characteristics of the fan and the mine. The intersection of the two static head curves gives the operating point only if the mine and fan velocity heads are the same or if their difference is very small.

To avoid low efficiencies and/or unstable performance, the intersection of the mine and fan characteristics curves should lie along the upper part of the fan characteristic to the right of the fan stall point where unstable operation can occur, especially with axial flow fans. If the measured operating point of a fan does not fall on the fan characteristic curve supplied by the manufacturer, and the fan performance is stable, the fan is operating on the descending, low-efficiency portion of the curve. The importance of correctly matching the fan and mine characteristics is illustrated with an example in Fig. 11.6.25 (Rock, Dalzell, and Harris, 1971). Here, the head-quantity and power curves for an axial flow fan are shown for two blade positions. The dashed line indicates the portion of the curve that was not supplied by the manufacturer but was extrapolated.

The fan is presently operating in a mine in the No. 6 blade position supplying 40,000 cfm (18.88 m^3/s) of air at a pressure of 7.8 in. water (1.941 kPa) corrected to 0.0750 lb/ft^3 (1.2000 kg/m^3) density. The power consumed is 136 bhp (101.42 kW). To evaluate fan performance, the operating point and the mine characteristic are drawn on the same graph to the same scale. Since it is known that when H = 7.8 in. water (1.941 kPa), Q = 40,000 cfm (18.88 m^3/s) and $H = RQ^2$, the specific relationship between H and Q for this mine is

$$H = 48.8Q^2$$

Values for H are calculated for assumed values of Q (Table 11.6.7) leading to the mine characteristic shown in Fig. 11.6.25.

The operating point in blade position 6 is to the left of the stall point on the unstable and inefficient portion of the fan characteristic. On the other hand, if the fan were operated in blade position 12, the operating point would be to the right of the stall point in the high-efficiency portion of the curve. In fact, the power consumed is only 64 bhp (47.72 kW), and the fan will generate 39,000 cfm (18.41 m^3/s) at 7.4 in. water (1.841 kPa). This example clearly points out the need to match the fan and mine characteristic for efficient and economic operation.

11.6.8.5 Fan Applications

There are three distinct applications of fans in mine ventilation: main fans, booster fans, and auxiliary fans (Turcic and Banfield, 1982). *Main fans,* as the name implies, are the principal fans used to produce the general ventilation air current in mines. They are normally permanent in nature and large in capacity. According to the 1969 Federal Coal Mine Health and Safety Act, main fans must be located on the surface. *Booster fans* are usually installed to ventilate parts of the mine where the main fans may, by themselves, be inadequate. Booster fans are used extensively in metal mines but presently are not allowed in US coal mines. *Auxiliary fans* are usually of small size, portable, and temporarily installed to ventilate working places through which the general ventilating air may not otherwise pass (dead-end openings). It is a common practice to conduct the air from the auxiliary fan to the working place by a ventilation tube or duct.

MAIN FAN INSTALLATION. Main fans in coal mines must be placed on the surface. The primary advantages of locating a fan on the surface are

1. Accessibility—the fan is accessible in the event of underground disasters such as fires and flooding.

2. Safety—the fan is unlikely to be damaged in the event of underground disasters such as explosions and it is not subjected to ground movement or roof falls.

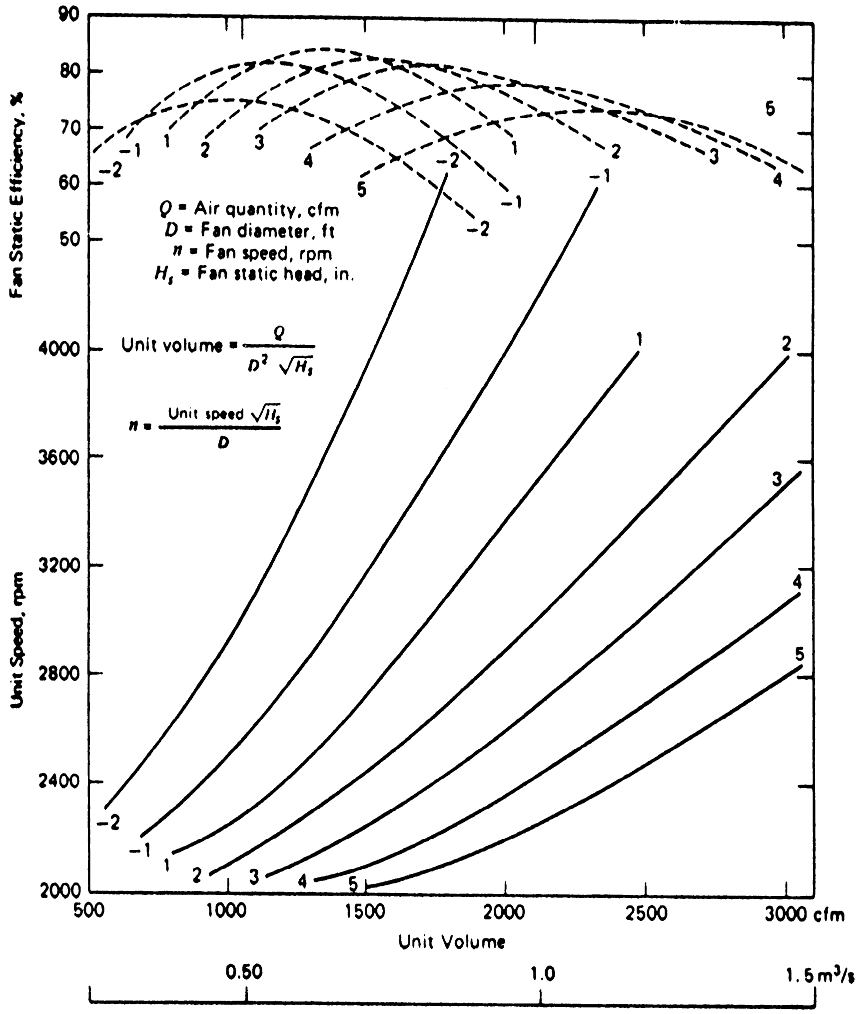


Fig. 11.6.21. Typical unit-volume, unit-speed plot for axial flow fans (by permission: Joy Technologies, Inc., Joy/Green Fan Division, New Philadelphia, OH; Hartman, Mutmansky, and Wang, 1982).

3. Ease of installation—installation of a fan underground requires that the workings be advanced; also the size of the fan to be used underground is constrained by space limitations, and a complicated network of airways is necessary to channel the air into the fan drift.

When installed on the surface, there can be a considerable leakage of air from the outlet side of the fan to the inlet side of the fan unless care is taken. As much as 20% of the air quantity handled by the fan may be air leakage through the surface air lock, fan housing, etc., in which case, only 80% circulates through the mine.

The placing of main fans underground is more common in metal mines. In underground metal mines, it is common to have separate fans for different underground levels (Chapter 11.7.1). In coal mines, when the workings are too far from the fan for efficient circulation, it is common practice to sink additional shafts and install additional surface fans (Chapter 11.7.2).

EXHAUSTING vs. FORCING FANS. Depending on the location of the fan in the mine circuit, the fan static and total pressures generated by it relative to the atmosphere can be positive or negative or both. When the fan is placed at the top of the intake shaft, it is known as a *forcing (blowing) fan.* Pressures in the mine are above atmospheric. When it is placed on the top of the return shaft, it is known as an *exhausting fan.* Pressures in the mine are below atmospheric. When it is placed in an underground location, it is known as a *booster fan.* In the booster location the mine pressure is negative from the intake shaft to the fan and is positive from the fan to the exhaust shaft.

REVERSAL OF AIR CURRENT. In the event of a stoppage of the primary fan or one fan in a multi-fan system, the 1969 Coal Mine Health and Safety Act requires that provisions be taken to prevent adverse reversals of air currents. In some countries, it is required by law to provide arrangements for the reversal of air current if such a need arises. With centrifugal fans, reversal of the air current requires a complicated arrangement of doors and ducts. With axial flow fans, reversing the direction of the rotor movement ensure that air will flow in the opposite direction, though the air quantity is reduced by 50 to 60%.

11.6.8.6 Pressure Gradients for Fan Systems

Important points to remember in plotting the *pressure gradients,* the relationships of system pressures to system profile, for mine/fan systems are

1. The total pressure (H_s + H_v) is always zero at the entrance (atmospheric) to a system.

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Conversion Factors:
  1 cfm = 0.47195 x 10^-3 m^3/s
  1 in. of water = 248.84 Pa
  1 HP = 0.7457 kw
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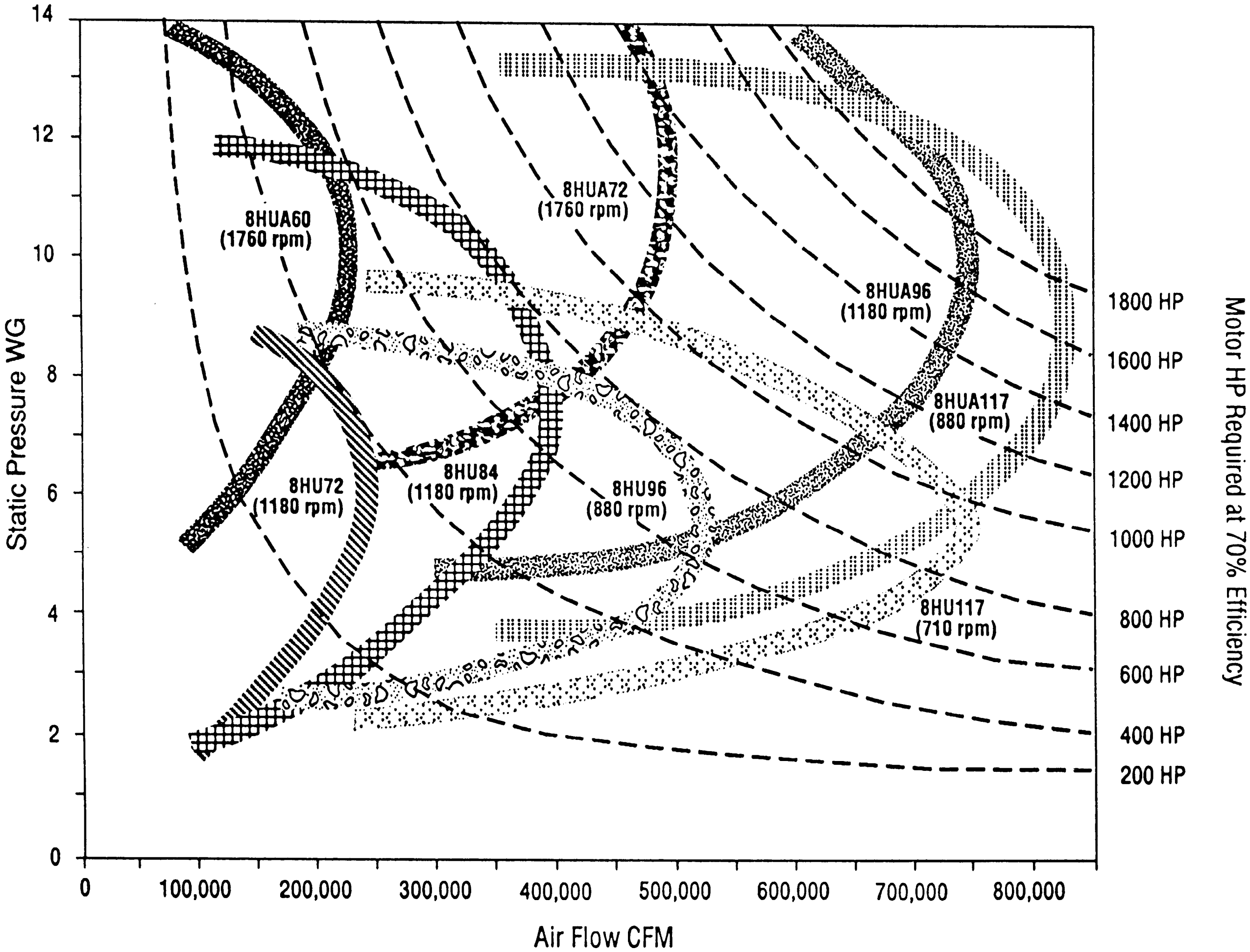


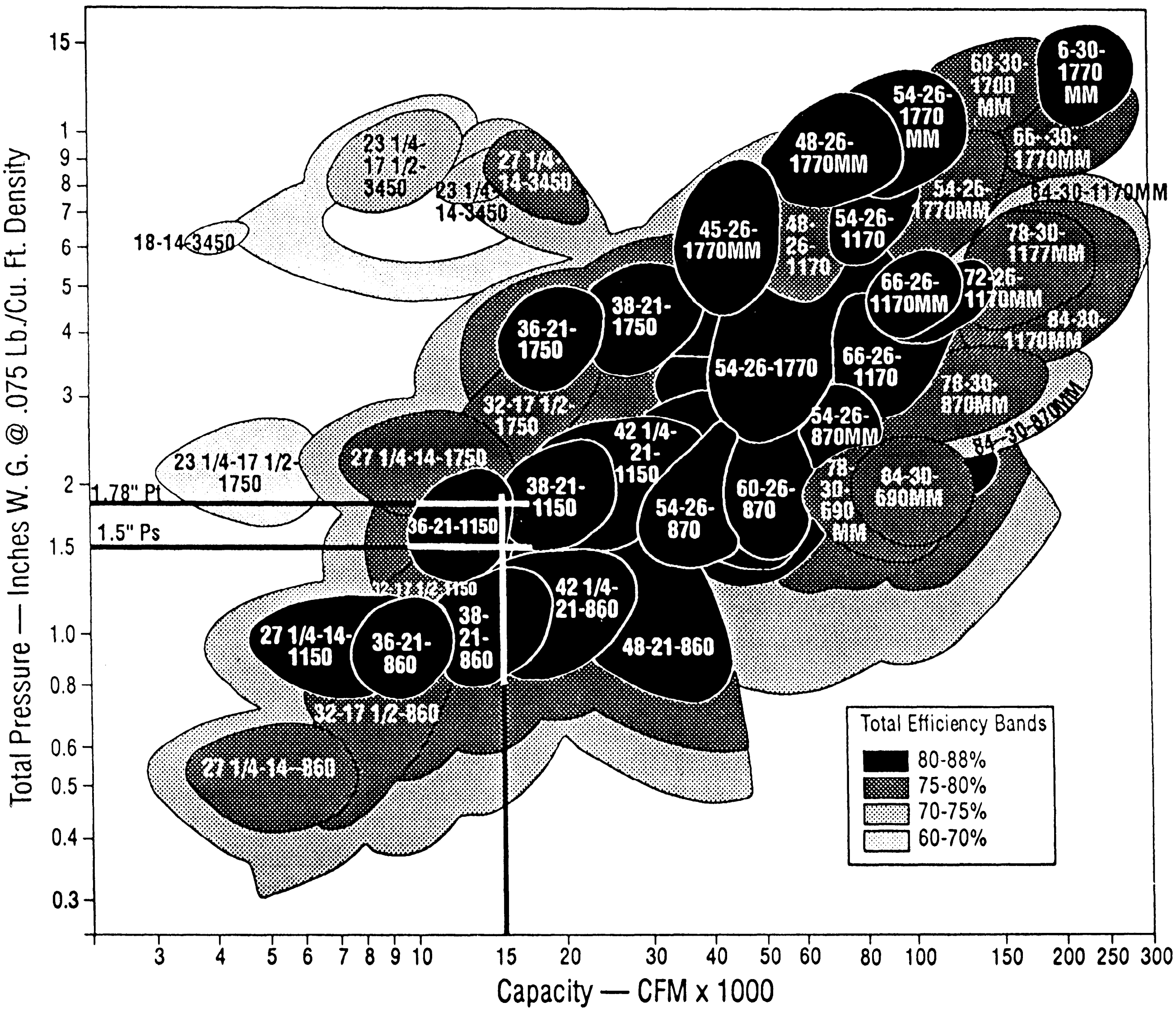
The curves show the approximate operating range of the Jeffrey eight-bladed fans. The area of maximum efficiency (over 80%) occurs at about 1/2 the maximum flow shown on the curves and over approximately the middle two thirds of the pressure range shown at the volume level. The horsepower lines correspond to a fan efficiency of approximately 70%.

Fig. 11.6.22. Jeffrey fan selection chart (by permission: Jeffrey Mining Machine Division, Dresser Industries, Inc., Columbus, OH).

2. The total pressure (H_s + H_v) is always positive and is equal to the velocity pressure at discharge to the atmosphere.

Since H_t = H_s + H_v at any point in the system and, by definition, at entrance, H_t = 0, then H_s at entrance must equal ($-H_v$). Similarly, H_t = H_v, at discharge, and H_s = 0. In effect, the static head is always zero at discharge, and is always negative and equals the velocity head at the entrance to the system. In drawing pressure gradients, a useful procedure is to start plotting the total pressure gradient from the point(s) away from the fan and work towards the fan. After plotting the total pressure gradient, mark off the velocity heads in straight sections of the airway, and draw the static pressure gradients parallel to the total gradient.

Forcing Systems. In the forcing system, the fan inlet is open to the atmosphere and the fan outlet to the mine intake shaft. The top of the intake shaft is equipped with an airlock. The air increases in velocity and static head as it flows through the fan. After flowing through the mine workings, it eventually returns to the surface through the return shaft. The absolute pressure measured at any point in the mine is higher than that of the atmosphere. The pressure gradient for a simple layout is shown in Fig. 11.6.26. The difference between the total pressure and static pressure gradients is the velocity pressure. Note that where there is an increase in area, there is an increase in static pressure at the expense of velocity pressure; but where the area decreases, there is drop in static pressure and velocity pressure



Conversion Factors:

1 cfm = 0.47195 x 10^{-3} m^3/s

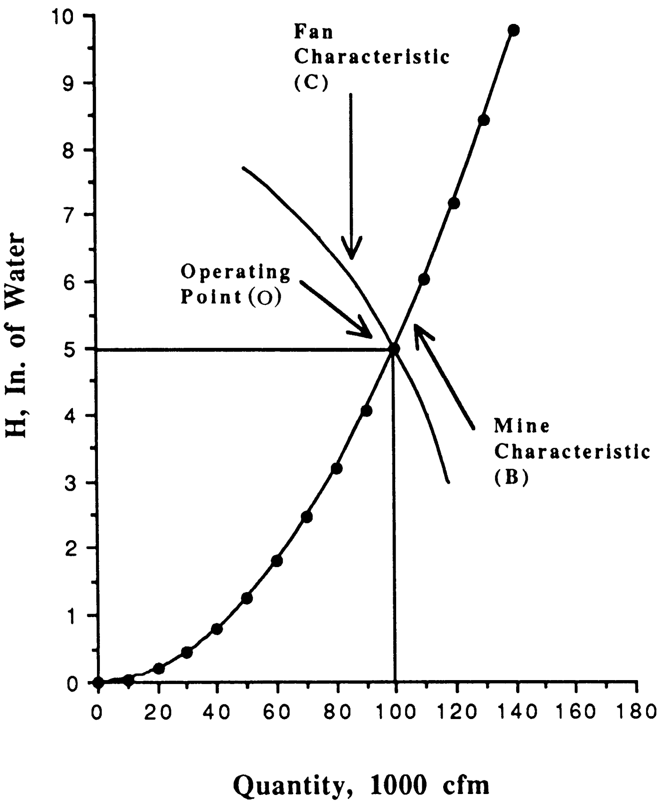
1 in. of water = 248.84 Pa

Fig. 11.6.23. Joy optimum efficiency fan selection chart (by permission: Joy Technologies, Inc., Joy/Green Fan Division, New Philadelphia, OH).

increases. In forcing systems, there is no shock loss at the entrance to the system as the fan absorbs this loss. The total pressure curve is above the static pressure curve and also above the atmospheric datum at all points except at the entrance.

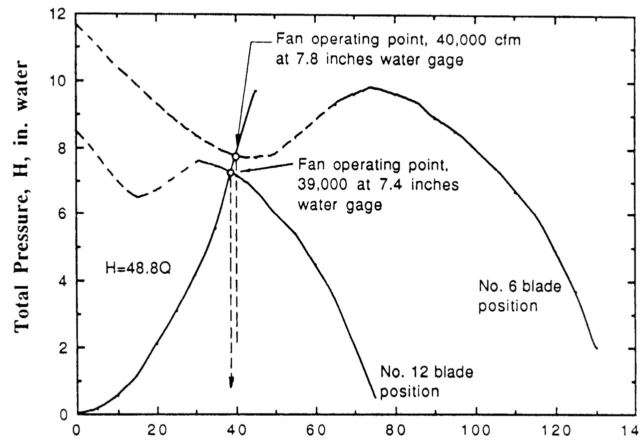
Exhausting Systems. In an exhausting system, the fan inlet is open to the mine through the return shaft, and the fan outlet is open to the atmosphere, usually through an *evase,* a gradually expanding exhaust duct that recovers velocity head. The top of the return shaft has a weakwall or an airlock. When the fan is started, the air on the inlet side is drawn into it, increases in velocity and static heads, and flows through the evase into the atmosphere. The absolute pressure on the inlet side of the fan is lower than that of the atmosphere. Also there is reduced shock loss at the discharge of the system.

On theoretical grounds, there is very little difference between blowing and exhausting systems. Forcing fans handle clean air from the atmosphere that is denser than the hot return air. In a blowing system, gases are held back in the gobs. While this is an advantage when the fan is operating, if the fan is stopped due to any reason, the workings may be flooded by gases from the gob. On practical grounds, exhausting fans are preferred for coal mine ventilation. It is best not to locate the fan at the top of a shaft that is frequently used for personnel and material transport. This way, the surface airlock to the fan need not be disturbed at any time. Also the main access and exit to the mine must be through the fresh intake side. More importantly, the workers can escape through the intake escapeway. To protect the main fan on the surface from the force of an explosion, it should be offset from

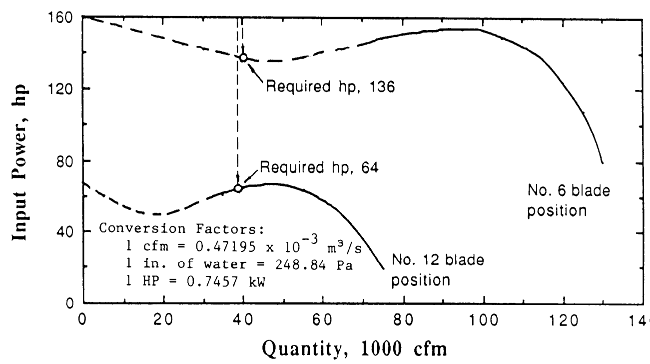


Conversion factors: 1 cfm = 0.47195 × 10^{-3}/s, 1 in. of water = 248.84 Pa.

Fig. 11.6.24. Mine and fan characteristics.



(a) Fan Head-Quantity Characteristics for No. 6 and No. 12 Blade Positions Showing the Operating Points



(b) Fan Input Power for the Operating Points for No. 6 and No. 12 Blade Positions

Fig. 11.6.25. Fan characteristics at two blade settings for the same mine characteristics (Rock, Dalzell, and Harris, 1971).

the nearest side of the mine opening with explosion doors (or a weak wall) installed in the direct line of the possible explosion force (Fig. 11.6.27). The fan must be equipped with a pressure-recording gage and an automatic signal device (audio and/or visual) to give an alarm should it slow down or stop. In the event of a fan stoppage, electrical power is disconnected from the mine and personnel are evacuated.

BOOSTER SYSTEM. The pressure gradient for a booster system is a combination of the gradients of the exhausting and forcing systems. The pressure gradient on the inlet side of the fan is similar to that of the exhausting system. After fan discharge, the pressure gradient is similar to that of the forcing system. The total pressure curve, as before, is always above the static pressure curve. However, the total and static pressures on the discharge side of the fan are greater than those on the inlet side. Because of this, unless the two sides of the fan are well separated, leakage can be severe, and air can recirculate from the outlet side of the fan to the inlet side or to the workings. Also there are shock losses at both entrance and discharge of the system.

Example 11.6.13. Draw the pressure gradients (total and static) and label all the relevant heads on the figure for the mine-fan system shown in Fig. 11.6.28 for (a) a forcing fan, (b) an exhausting fan, and (c) a booster fan located at the middle of section A.

Table 11.6.7. Calculation of the Mine Characteristic

Q cfm	H in. water $H = 48.8\left(\frac{Q}{100{,}000}\right)^2$
10,000	0.49
20,000	1.95
30,000	4.39
40,000	7.80
50,000	12.20

Conversion Factors:
1 cfm = 0.47195 x 10^{-3} m³/s
1 in. water = 248.84 Pa

Section A:	Friction loss (H_{F_A}): 1.0 in. water (248.84 Pa)
	Velocity head (H_{V_A}):0.5 in. water (124.42 Pa)
Section B:	Friction loss (H_{F_B}): 1.5 in. water (373.26 Pa)
	Velocity head (H_{V_B}): 1.0 in. water (248.84 Pa)
Shock Losses:	Entrance (H_{X_E}): 0.5 in. water (124.42 Pa)
	Discharge (H_{X_D}): 0.5 in. water (124.42 Pa)
	Contraction (H_{X_C}): 0.5 in. water (124.42 Pa)

Assume $H_{A_{\text{inlet}}} = H_{V_A}$; $H_{V_{\text{disch}}} = H_{V_B}$

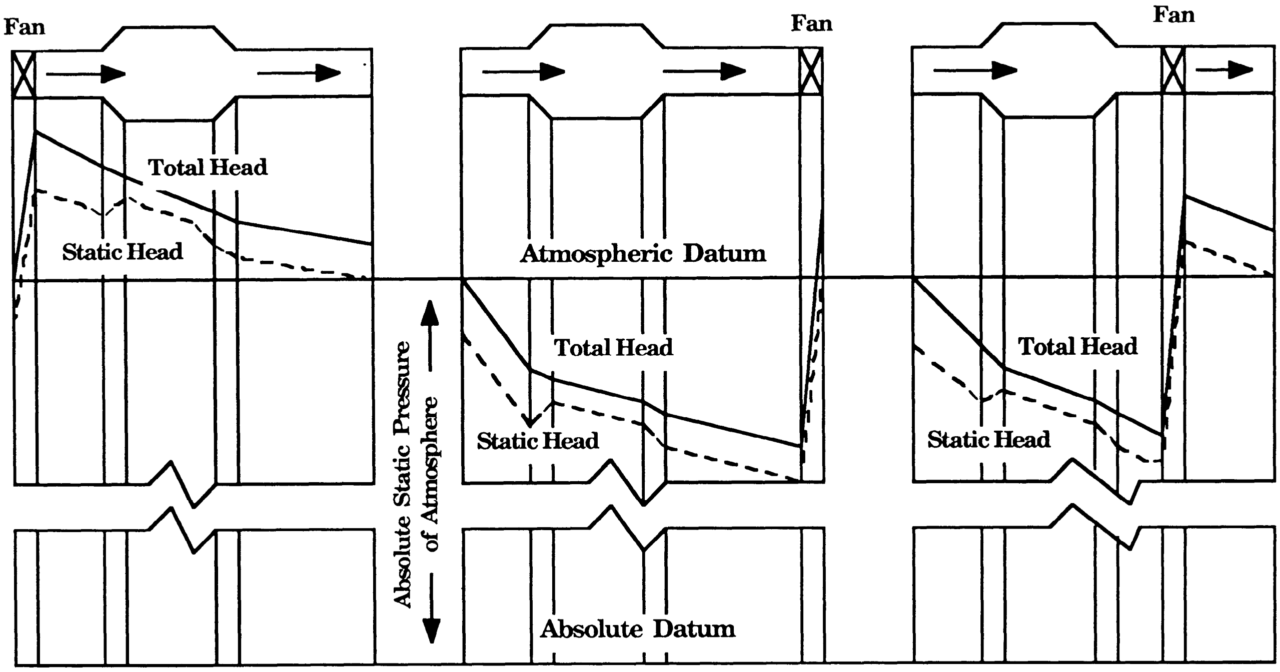


Fig. 11.6.26. Pressure gradients for fan-mine systems (McElroy, 1935).

Solution. Following the procedures outlined earlier, the pressure gradients are drawn for the three positions of the fan in the mine. See Figs. 11.6.28(a), (b), (c) for the forcing, exhausting, and booster fan locations.

FAN EVASE. In practice, all the theoretical head developed by a fan is not converted to static head. For example, in the radial-bladed fan, only half of the theoretical head is actually available as static head to generate the flow of air through the fan. The other half is in the form of kinetic energy in the air leaving the periphery of the fan. If this energy is to be recovered as static head, then the velocity of the air must be decreased. It is for this purpose that the fan is placed in a casing, and the air is then allowed to flow into a duct (Fig. 11.6.29) whose cross section is gradually increased from the fan outlet (DD) to the point of discharge. This duct is known as an evase. The gain in static head due to the evase can be mathematically stated as

$$H_g = \frac{w}{5.2}\left(\frac{V_1^2}{2g} - \frac{V_2^2}{2g}\right)\left(\frac{1}{3600}\right) \qquad (11.6.45)$$

$$H_g = \frac{w}{2g}(V_1^2 - V_2^2) \qquad (11.6.45a)$$

where H_g is static head gain due to evase in in. water (Pa), V_1 is velocity at the small end of the evase $= \frac{Q}{A_1}$ in fpm (m/s), V is velocity at the large end of the evase $= \frac{Q}{A_2}$ in fpm (m/s), Q is quantity flowing in cfm (m^3/s), A_1 and A_2 are area at the small and large ends of the evase, respectively, in ft^2 (m^2), w is density of air in lb/ft^3 (kg/m^3), and g is acceleration due to gravity = 32.2 ft/sec^2 (9.80665 m/s^2).

The evase must be properly shaped; if the expansion is abrupt, that is, the increase in area from the foot to the top is sudden, then eddy currents are set up, and the recovery is poor. A gentle slope of the evase sides allows for efficient conversion of the kinetic energy to static energy. According to Weeks (1926), the slopes should not be much over 6°. According to Briggs and Williamson (Penman, 1927), the ideal shape for vertical evases is one in which the walls nearest the fan and farthest from it slope at an angle of 7°, the side walls have an angle of 3.5°, and the area at discharge is about four times the area at the foot. Evases shaped as an expanding cone are commonly employed in mines. With forcing or booster fans, the intake shaft and the mine itself may serve the purpose of an evase. However, with exhaust fans, an evase is necessary to conserve as much energy as possible by decreasing the velocity at the discharge of the mine.

11.6.8.7 Multi-fan Arrangements

Increases in head, quantity, or both can be obtained by operating more than one fan. Multiple fans can be operated in series, parallel, or in combination as shown in Fig. 11.6.30. The analysis here neglects the velocity head of the fan; only static heads are considered.

SERIES ARRANGEMENT. Fans operating in series are on the same flow circuit, each handling the same airflow, and each generating a part of the total pressure required. Series operation of fans may be necessary when there is a need for increasing the pressure considerably without increasing the air quantity by the same proportion. Such conditions arise in high-resistance mines.

The combined fan characteristic for series operation is determined by plotting the sum of the pressures generated by each fan at a given quantity against that quantity. For example, consider fans A and B with the following characteristics (Table 11.6.8 and Fig. 11.6.31).

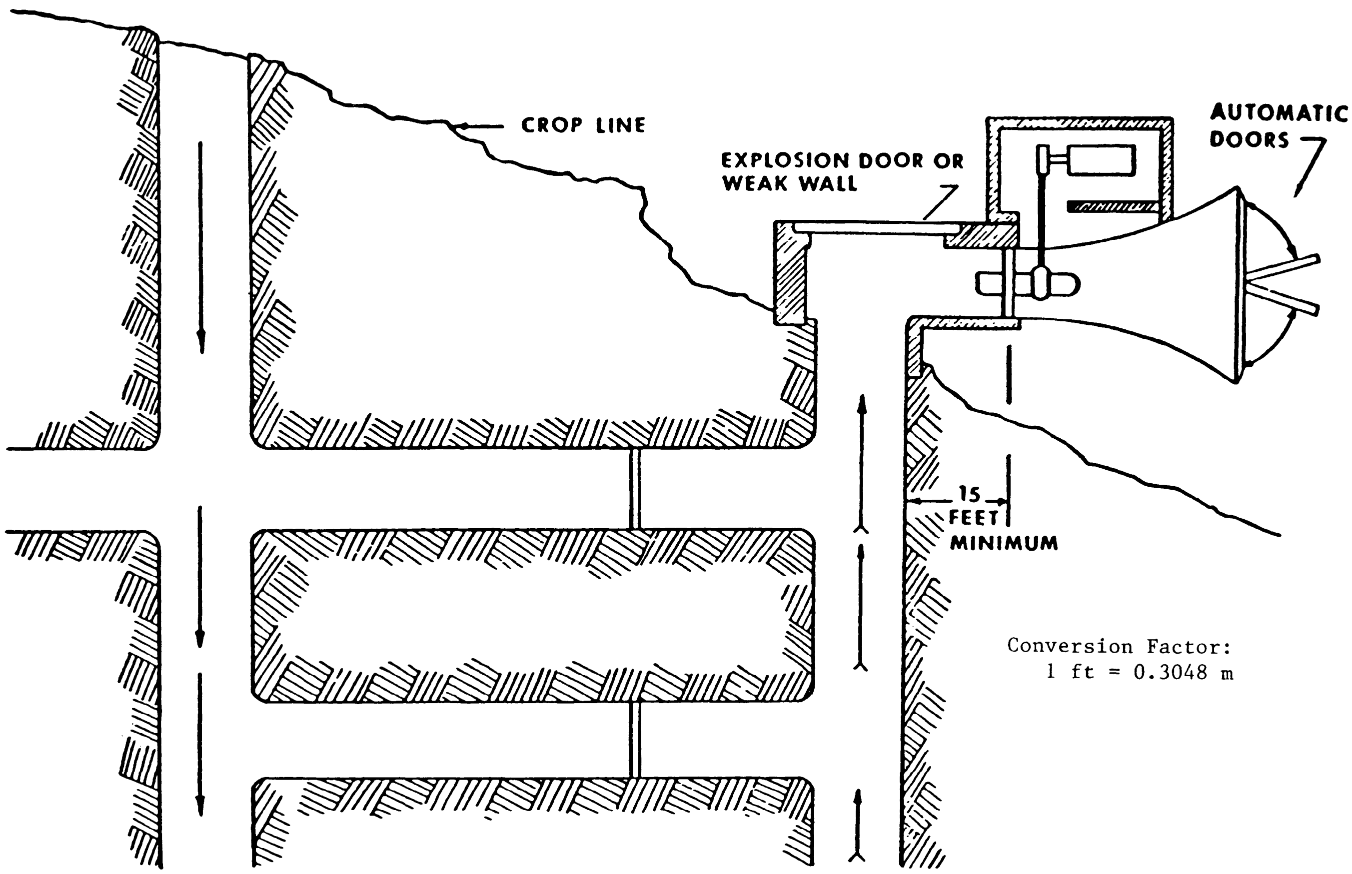


Fig. 11.6.27a. Diagram of main fan at drift or slope.

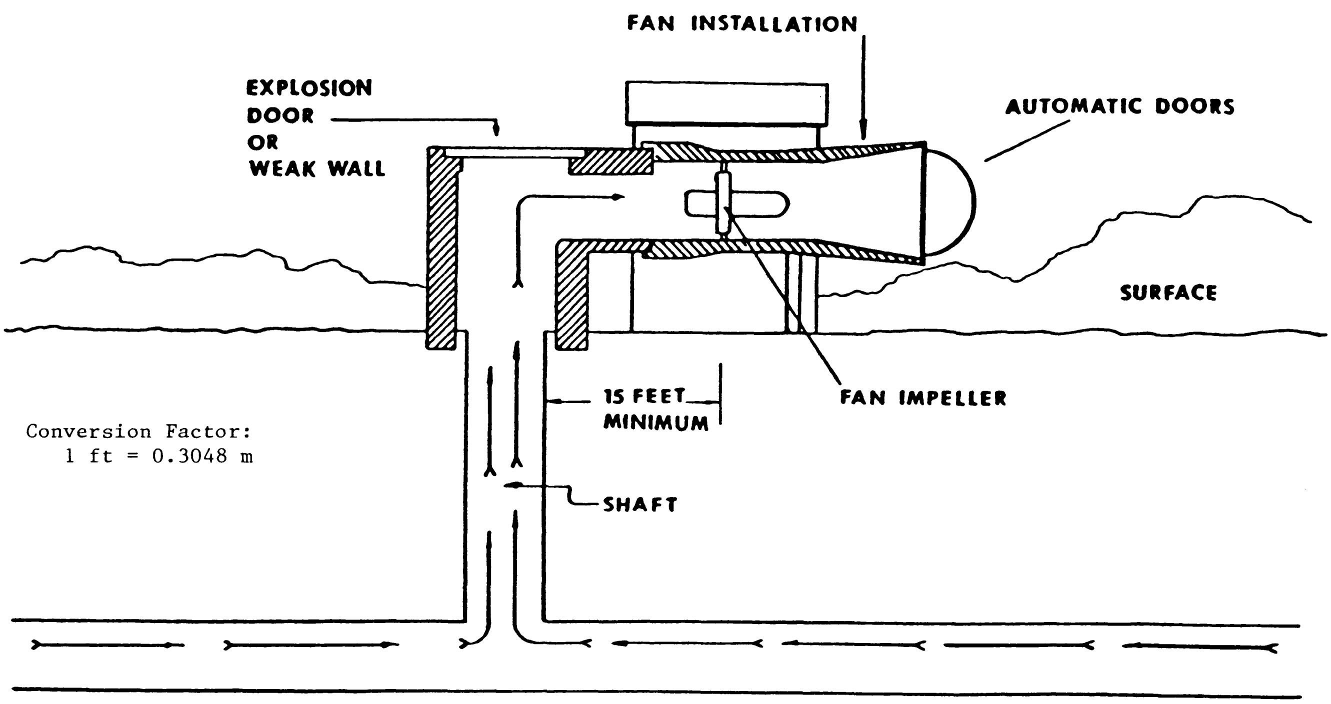


Fig. 11.6.27b. Diagram of main fan at shaft.

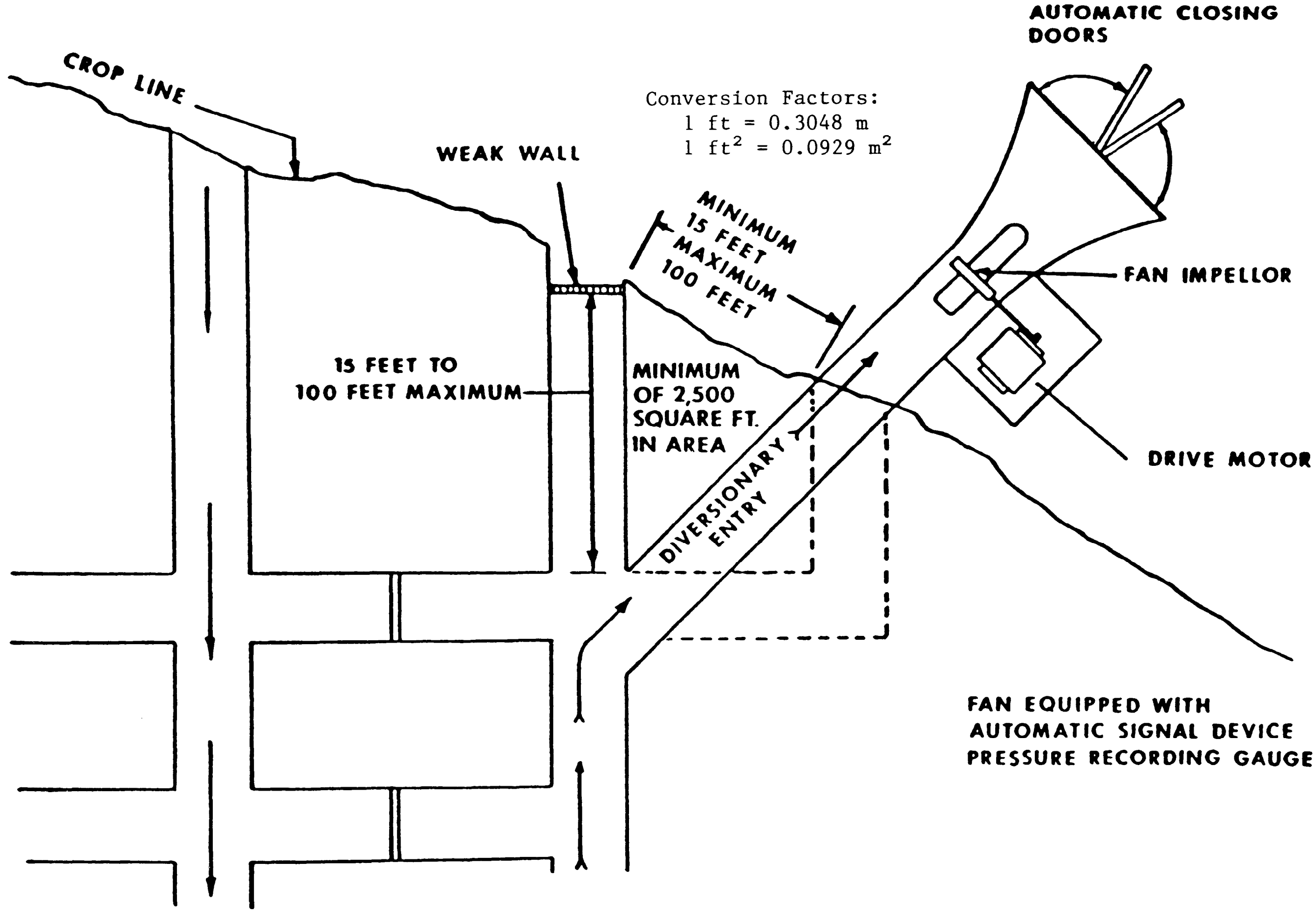


Fig. 11.6.27c. Diagram of main fan and diversionary entry.

When fan A and fan B are arranged in series, the combined characteristic is obtained as below:

Quantity, cfm (m^3/s)	Head, in. water (kPa)
0 (0)	3.0 + 1.75 = 4.75 (1.182)
10,000 (4.72)	2.5 + 1.5 = 4.0 (0.995)
20,000 (9.44)	0.5 + 1.0 = 1.5 (0.373)
22,500 (10.62)	0 + 0.8 = 0.8 (0.199)
30,000 (14.16)	0 + 0.25 = 0.25 (0.062)

Consider this arrangement in a mine whose characteristic is given by the curve C in Fig. 11.6.31. The operating point (O) is the intersection of the mine characteristic and the series characteristic. The quantity of air flowing through the mine and through each fan is approximately 17,000 cfm (8.02 m^3/s) at a head of approximately 2.63 in. water (654.45 Pa). A vertical line from O to the x axis intersects the fan curves for fans A and B at A_1 and B_1, respectively. Reading the pressure heads at A_1 and B_1, it is readily seen that fan A develops 1.5 in. water (373.26 Pa) and fan B, 1.13 in. water (281.20 Pa). Fan A does not generate any head when the quantity flowing through the fans exceeds 22,500 cfm (10.62 m^3/s), but will add a small amount of resistance to flow.

PARALLEL ARRANGEMENT. When two fans are installed in separate openings in such a way that they draw air from the same common point and deliver it to the same destination, they are said to be operating in parallel. Mine fans may be operated in parallel to increase the quantity of air flowing through the system when the pressure capacity of each fan is adequate. These conditions may arise in low-resistance mines. The combined fan characteristic for parallel operation is determined by plotting the sum of the quantities generated by each fan at a pressure against that pressure. For example, when fans A and B above are arranged in parallel, the combined characteristic is obtained as follows (Fig. 11.6.32). At 1.75 in. water (435.47 Pa), fan A delivers 15,000 cfm (7.08 m^3/s) and fan B delivers 0 cfm. At 1.5 in. water (373.26 Pa), fan A delivers 16,300 cfm (7.69 m^3/s) and fan B delivers 10,000 cfm (4.72 m^3/s). At 0.25 in. water (62.21 Pa), fan A handles 21,250 cfm (10.03 m^3/s), and fan B handles 30,000 cfm (14.16 m^3/s). The combined characteristic is calculated as follows:

Head, in. water (Pa)	Quantity, cfm (m^3/s)
1.75 (435.47)	15,000 + 0 = 15,000 (7.08)
1.5 (373.26)	16,300 + 10,000 = 26,300 (12.41)
0.25 (62.21)	21,250 + 30,000 = 51,250 (24.19)

If the mine characteristic is given by the curve C in Fig. 11.6.32, the head and quantity flowing are 1 in. water (248.84 Pa) and 38,000 cfm (17.93 m^3/s), respectively. Fan A develops 1 in. water (248.84 Pa) and 18,000 cfm (8.50 m^3/s). Fan B develops 1 in. water (248.48 Pa) and 20,000 cfm (9.43 m^3/s). If

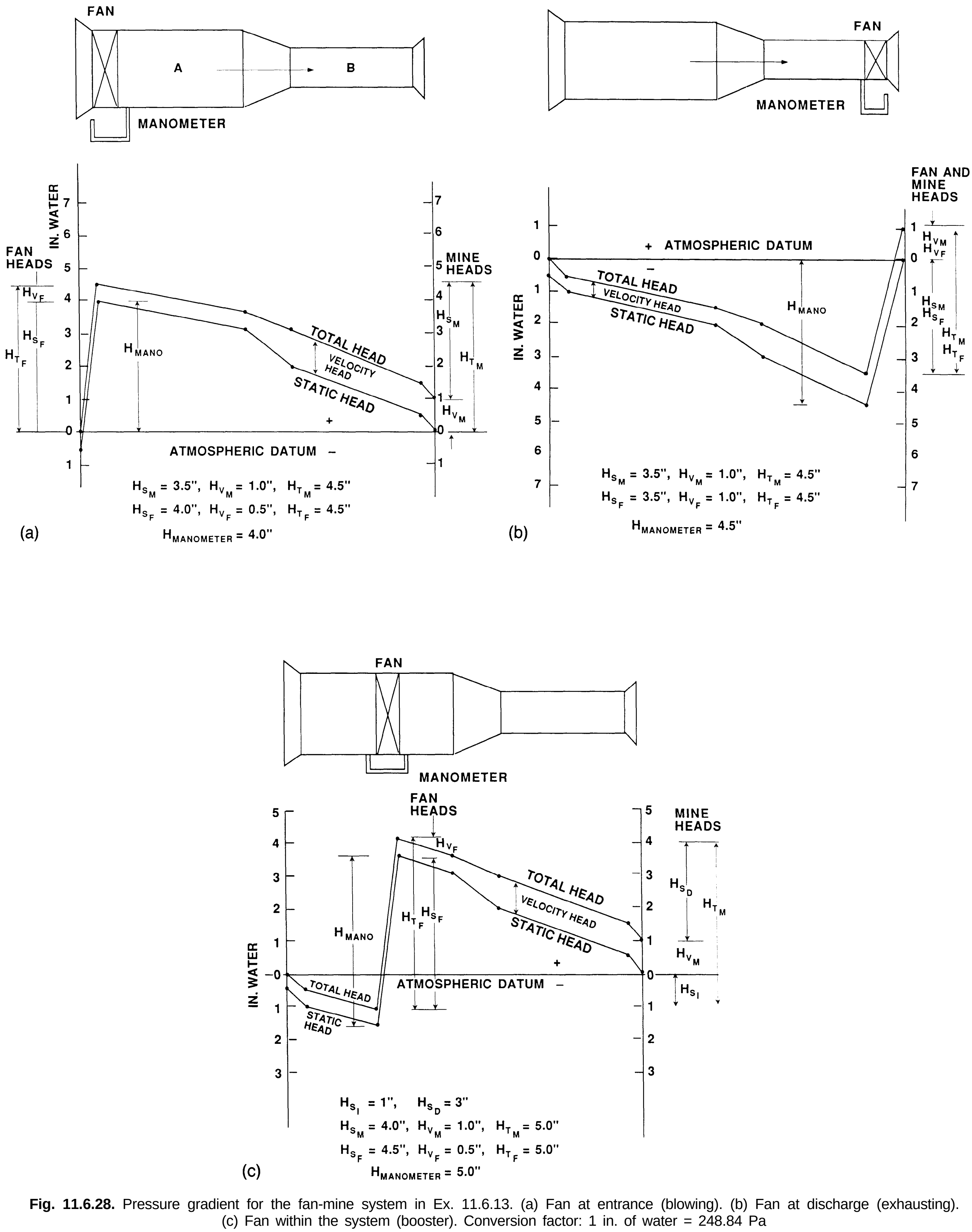


Fig. 11.6.28. Pressure gradient for the fan-mine system in Ex. 11.6.13. (a) Fan at entrance (blowing). (b) Fan at discharge (exhausting). (c) Fan within the system (booster). Conversion factor: 1 in. of water = 248.84 Pa

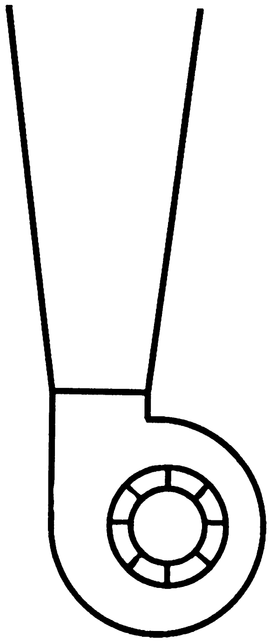


Fig. 11.6.29. Fan with an evase.

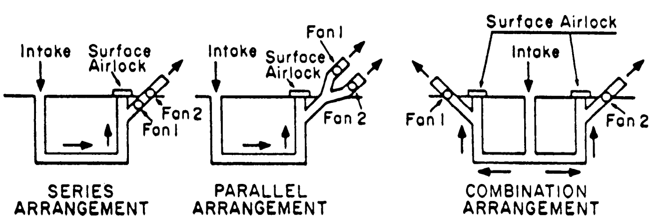


Fig. 11.6.30. Various arrangements for multiple fans.

Table 11.6.8. Head-Quantity Characteristics of Fan A and Fan B

Fan A		Fan B	
Head, in. water	Quantity, cfm	Head, in. water	Quantity, cfm
3	0	1.75	0
2.5	10,000	1.5	10,000
0.5	20,000	1.0	20,000
0	22,500	0.8	22,500
		0.25	30,000

Conversion factors: 1 in. water = 248.84 Pa, 1 cfm = 0.47195 × 10^{-3} m^3/s.

the pressure required is above 1.75 in. water (435.47 Pa), fan B will not only be of no assistance, but will hinder the flow of air from the mine by recirculating some air through it back to fan A.

FANS IN COMBINATION. Many fans are operated in combination rather than in true series or parallel operation. Two possible applications are fans in semi-parallel and fans in semi-series (Fig. 11.6.33). In these arrangement, each fan shares a portion of the total mine resistance load, but retains its own specific zone

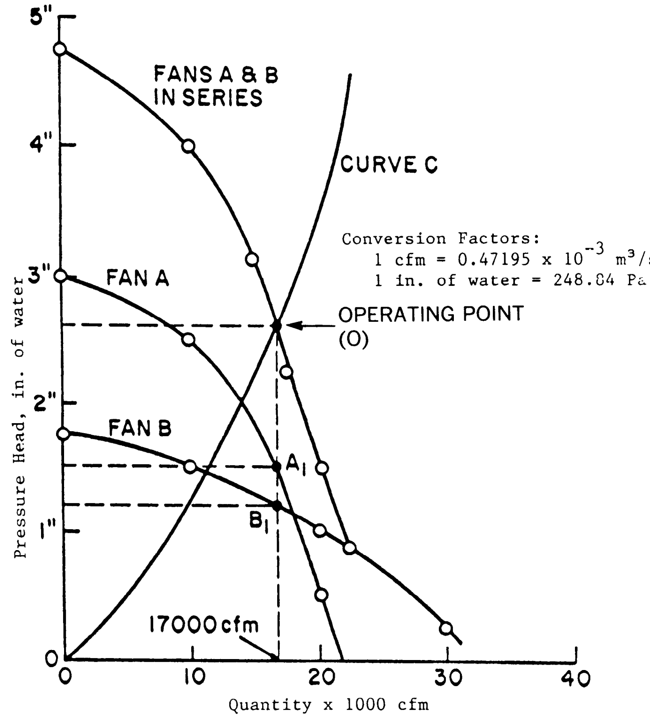


Fig. 11.6.31. Combined characteristics for series operation.

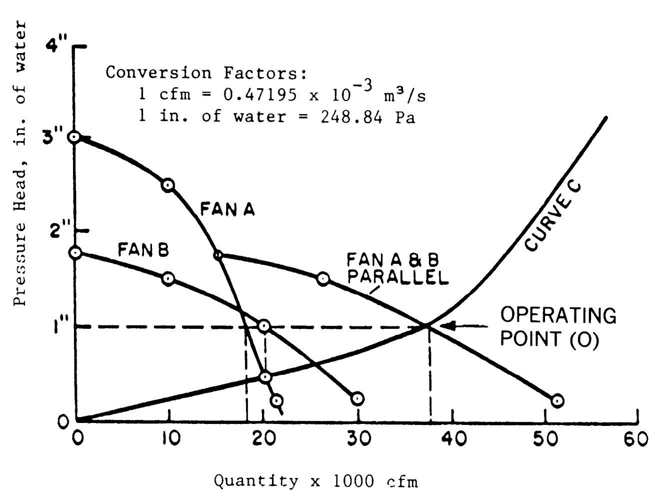


Fig. 11.6.32. Combined characteristics for parallel operation.

of influence. Thus the operation of each fan is influenced by the others in the system. The mathematical analysis of multiple-fan systems can become complex and is usually performed more rapidly and accurately with the use of computer-oriented mine ventilation network solution programs (Chapter 11.10). However, graphical techniques are available to solve simple problems (Rock, Dalzell, and Harris, 1971).

11.6.9 NATURAL VENTILATION

Natural ventilation is a result of the differences in elevation between mine openings and the addition of heat energy to the

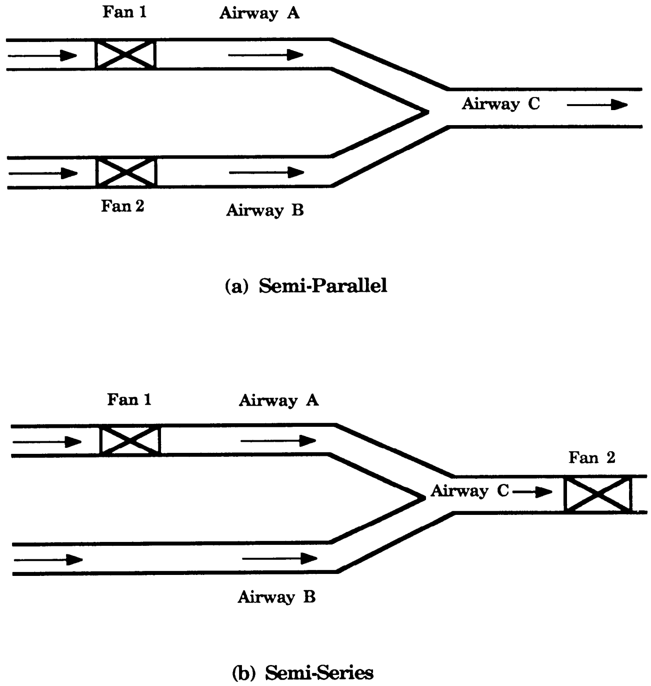


Fig. 11.6.33. Fans in combination. (a) Semi-parallel. (b) Semi-series.

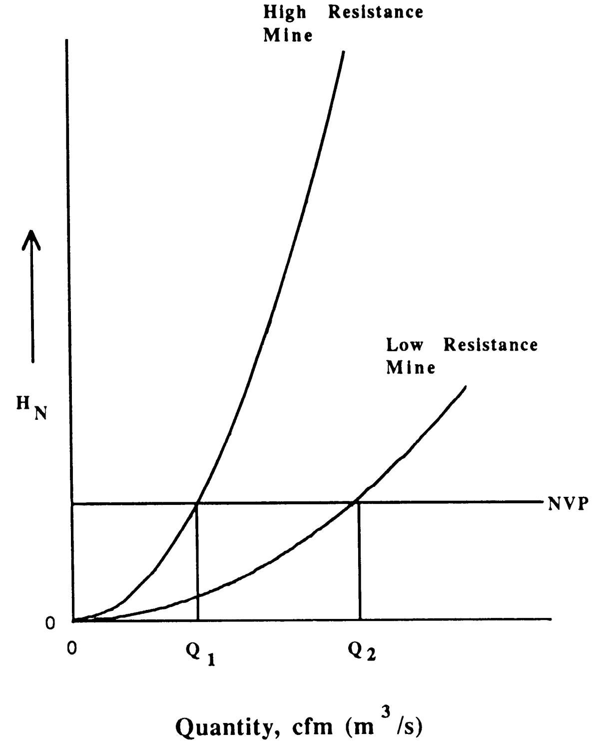


Fig. 11.6.34. Quantity of flow due to natural ventilation pressure.

air as it passes through the workings. Air, as it flows through a mine, gains in heat due to reasons such as the geothermal gradient, autocompression, and oxidation. Air may also be cooled due to contact with flowing water in the shafts or due to rock temperatures being colder than outside air. As a result, the displacement of a hot air column by a colder air columns ensues. Where the difference in the elevations of the openings is very small so that the columns of air do not have significant differences in density, flow will not occur naturally. If, on the other hand, flow is induced by some mechanical means such as a fan, the flow may continue as long as heat is added to the flowing air and the temperature difference between the two air columns exists. Even if the fan stops, this difference in temperature between the two air columns will continue to aid in the movement of air (assuming natural ventilation pressure was exerted in the same direction as the airflow) from the intake to the return although the quantity handled may fall off rapidly. Even if the elevations of the openings are the same, if local weather conditions cause a significant difference in the surface air temperatures above the openings, a flow of air can ensue.

11.6.9.1. Natural Ventilation Pressure

The rate of displacement of one column of air by another is proportional to the difference in the temperatures of the two columns. The differences in temperatures account for the density difference and the pressure head generated. This pressure head, because it arises due to natural conditions, is known as *natural ventilation pressure* (NVP). However, because the ventilation pressure provided by natural means is often inadequate, subject to variations, and difficult to control, ventilation by mechanical means is absolutely essential for most mines. On the other hand, natural ventilation affects the performance of the mechanical ventilator, so it is necessary to understand its effects on mechanical ventilation.

The magnitude and direction of NVP are not controllable. NVP is independent of the quantity flowing, depending only on the temperature difference and the height of the air column. NVP can be viewed as a fan with a characteristic parallel to the quantity axis (i.e., a constant head fan). Thus the amount of air flowing through a mine due to NVP is a function of the mine characteristic only. In a high-resistance mine, a lower quantity will flow than in a low-resistance mine (Fig. 11.6.34).

NVP ESTIMATION. The correct method of calculating the pressure generated by this addition of heat energy is the thermodynamic approach. The difference in temperature of the air in the intake and return air columns leads to a difference in the densities of the air columns. Calculations of the natural ventilating pressures using the densities of two columns of equal heights lead to results that are reasonably accurate for routine mine ventilation calculations.

Natural ventilation pressure changes frequently in magnitude. Its direction also changes, though this change can be seasonal or daily depending on the frequency and amount of temperature variations on the surface. In fact, due to the continual changes in temperature during a day, the NVP also continually changes. Fortunately, NVP tends to be small and, therefore, its calculation using one of several approximate formulas is adequate for routine mine ventilation calculations. On the other hand, where a detailed ventilation survey is being conducted, it is necessary to calculate NVP more accurately from detailed temperature and pressure surveys. For rapid estimates of NVP, the following approximate formulas can be employed (McElroy, 1935).

1.
$$H_n = \frac{1.325\ BL}{5.2}\left(\frac{1}{T_1} - \frac{1}{T_2}\right) = 0.255\ BL\left(\frac{1}{T_1} - \frac{1}{T_2}\right) \quad (11.6.46)$$

$$H_n = \frac{BL}{0.287}\left(\frac{1}{T_1} - \frac{1}{T_2}\right) \quad (11.6.46a)$$

where H_n is the natural ventilation pressure in in. water (Pa); B is the average absolute pressure in in. of mercury (Pa); L is the vertical height of the air columns in ft (m), and T_1 and T_2 are the average absolute temperatures of the columns in °F (°C).

The approximate average absolute pressure is obtained by direct measurement near the center of the columns or from a measurement at any elevation. In the latter case, the measurement must be corrected for difference in elevation from the center at the rate of 1 in. water change/1000 ft (816 Pa/1000 m), or more closely if the actual rate of change is known. The average absolute temperatures are obtained as weighted means according to the column lengths in the case of variable temperatures.

2. The intensity of natural-draft pressures may also be roughly estimated as about 0.03 in. water for each 10°F difference in average temperature for each 100 ft (100 m) difference in vertical elevation at standard air density (44 Pa/10°C/100 m), i.e.,

$$H_n = 0.03\left(\frac{T_2 - T_1}{10}\right)\left(\frac{L}{100}\right) \quad (11.6.47)$$

$$H_n = 44\left(\frac{T_2 - T_1}{10}\right)\left(\frac{L}{100}\right) \quad (11.6.47a)$$

3. A third approximation formula for calculating NVP is (Rees, 1950)

$$H_n = \left(\frac{T_2 - T_1}{5.2\ T_m}\right)(wL) \quad (11.6.48)$$

$$H_n = \left(\frac{T_2 - T_1}{T_m}\right)(wL) \quad (11.6.48a)$$

where T_2, T_1, and L are the same as above, w is density of the air, and $T_m = ½(T_2 + T_1)$.

4. Another approximate formula using the weights of the air columns is derived as follows. Let

$\overline{w}_d$ = average density of the air column in the downcast or intake shaft, lb/ft^3 (kg/m^3)

$\overline{w}_u$ = average density of an equivalent air column in the upcast or return shaft, lb/ft^3 (kg/m^3)

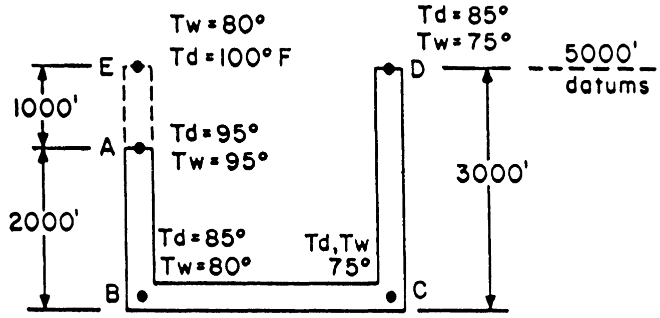
L = length of the air column, ft (m)

A column of air 1 ft^3 (0.02832 m^3) in volume and 1 ft (0.3048 m) in height in the intake shaft will exert a pressure of $(\overline{w}_d - \overline{w}_u)$ lb/ft^2 (kPa) on a similar volume of air in the return shaft. Therefore, the pressure exerted by the entire air column equals:

$$H_n = \frac{1}{5.2}(\overline{w}_d - \overline{w}_u)\,L \quad (11.6.49)$$

$$H_n = g\,(\overline{w}_d - \overline{w}_u)\,L \quad (11.6.49a)$$

in which $\overline{w}_d$ and $\overline{w}_u$ can be calculated by measuring the wet- and dry-bulb temperatures and the barometric pressure at different points in the intake and return shafts, and by averaging the calculated densities.



Conversion Factors:
1 ft = 0.3048 m
°C = 5/9 (°F-32)

Fig. 11.6.35. Results of a temperature survey (Ex. 11.6.14).

11.6.9.2. Quantity and Air Horsepower Due to NVP

Natural ventilation pressure is independent of the quantity of circulating air, and the quantity that will circulate is determined by the mine resistance only. However, Atkinson's equation still holds, i.e.,

$$H_n = \frac{KLOQ^2}{5.2A^3}$$

$$AHP = \frac{H_n Q}{6350}$$

Example 11.6.14. The application of the various formulas is illustrated with the following example. The results of a temperature survey are shown in the mine schematic (Fig. 11.6.35). The distance from B to C is 5 miles (8 km). All airways are 10 by 10 ft (3.048 × 3.048 m), and the K factor is 100×10^{-10} lb-min^2/ft^4 (0.0186 kg/m^3) at standard density. The elevation of the collar of the deeper shaft is 5000 ft (1524 m) above sea level.

Solution.

1. Since in all NVP calculations, columns of equal height have to be considered, an imaginary column AE is added to the shallower shaft.

2. The temperatures of the outside air at A are considered as the same at E.

3. Determine the barometric pressures at E, A, B, C, and D.

Point	Elevation, ft (m)	Barometric Pressure in. Hg (kPa)	Barometric Pressure psi
E	5000 (1524)	24.90 (84.6681)	12.28
A	4000 (1219)	25.84 (87.8398)	12.74
B	2000 (609.6)	27.82 (94.5277)	13.71
C	2000 (609.6)	27.82 (94.5277)	13.71
D	5000 (1524)	24.90 (84.6681)	12.28

4. Determine the density of air at E, A, B, C, and D using the chart (Fig. 11.6.1) and the given wet- and dry-bulb temperatures and the barometric pressure.

Point	Barometric Pressure in. Hg	t_d°F(°C)	t_w°F(°C)	Density Ratio* d_r	Density $0.0750 \times d_r$ (lb/ft^3) (kg/m^3)
E	24.90	100° (37.8)	80° (26.7)	0.77	0.0578 (0.925)
A (outside air)	25.84	100° (37.8)	80° (26.7)	0.805	0.0604 (0.966)
A (mine air)	25.84	95° (35)	95° (35.0)	0.800	0.0600 (0.960)
B	27.82	85° (29.4)	80° (26.7)	0.89	0.0668 (1.069)
C	27.82	75° (23.9)	75° (29.4)	0.91	0.0683 (1.093)
D	24.90	85° (29.4)	75° (29.4)	0.795	0.0596 (0.954)

*From Fig. 11.6.1.

5. Average the density, temperature, barometric pressure, taking into consideration the lengths of the air columns.

a. *Density*

Average density of the outside air in column EA =

$$\frac{0.0578 + 0.0604}{2} = 0.0591 \text{ lb/ft}^3 \ (0.946 \text{ kg/m}^3)$$

Average density of the air column AB in the mine =

$$\frac{0.0600 + 0.0668}{2} = 0.0634 \text{ lb/ft}^3 \ (1.010 \text{ kg/m}^3)$$

Average density of air column EB (weighted by length) =

$$\frac{1000 \times 0.0591 + 2000 \times 0.0634}{3000} = 0.0620 \text{ lb/ft}^3 \ (0.992 \text{ kg/m}^3)$$

Average density of air column CD =

$$\frac{0.0683 + 0.0596}{2} = 0.064 \text{ lb/ft}^3 \ (1.023 \text{ kg/m}^3)$$

Average density of the air in the two columns =

$$\frac{0.062 + 0.064}{2} = 0.063 \text{ lb/ft}^3 \ (1.008 \text{ kg/m}^3)$$

b. *Temperature*

Average dry-bulb temperature, column (outside) EA = 100°F (37.8°C)

Average dry-bulb temperature in mine, AB $= \dfrac{95 + 85}{2} =$ 90°F (32.2°C)

Average dry-bulb temperature of column EB =

$$\frac{1000 \times 100 + 2000 \times 90}{3000} = 93.3\text{°F} \ (54.1\text{°C})$$

Average dry-bulb temperature of column CD $= \dfrac{75 + 85}{2} =$ 80°F (26.7°C)

c. *Average Barometric Pressure*

Top of the air column = 5000 ft (1524 m) above sea level

Bottom of the air column = 2000 ft (609.6 m) above sea level

Average height of column $= \dfrac{5000 + 2000}{2} =$ 3500 ft (1066.8 m) above sea level

Barometric pressure at 3500 ft = 26.33 in. Hg = 12.94 psi (89.2187 kPa)

6. Natural ventilation pressure by various formulas:

a. $$H_n = 0.255\, BL \left(\frac{1}{T_1} - \frac{1}{T_2}\right)$$

$$= 0.255 \times 26.33 \times 3000 \left(\frac{1}{460 + 80} = \frac{1}{460 + 93.3}\right)$$

$$= 0.90 \text{ in. water (223.21 Pa)}$$

b. $$H_n = 0.03\, \frac{(T_2 - T_1)}{10} \times \frac{L}{100} \text{ at standard density}$$

$$H_n = 0.03\, \frac{(T_2 - T_1)}{10} \times \frac{L}{100} \times \frac{w}{0.075} \text{ at a density } w.$$

$$w = 0.03 \left(\frac{553.3 - 540}{10}\right) \times \frac{3000}{100} \times \frac{0.063}{0.075}$$

$$= 1.01 \text{ in. water (251.33 Pa)}$$

c. $$H = \frac{T_2 - T_1}{5.2 T_m}\, w\, L \quad \text{where } T_m = \frac{T_2 + T_1}{2};$$

$$= \frac{(553.3 - 540)}{5.2 \times 546.65} \times 0.063 \times 3000$$

$$= 0.88 \text{ in. water (218.98 Pa)}$$

d. $$H = \frac{1}{5.2}\,(\overline{w}_d - \overline{w}_u)\, L = \frac{1}{5.2}\,(0.064 - 0.062) \times 3000$$

$$= 1.15 \text{ in. water (287.11 Pa)}$$

The air will flow from C to B.

7. Calculate the airflow quantity and air horsepower.

$$H_n = \frac{KLOQ^2}{5.2A^3}$$

where $K = 100 \times 10$ in.-min^2/ft^4 (0.0186 kg/m^3) for standard air

The value of K must be corrected for the density of the air because the average density of the air varies during its flow through the mine.

average density of air in column AB = 0.0634 lb/ft^3 (1.010 kg/m^3)

average density of air in column CB = 0.0640 lb/ft^3 (1.023 kg/m^3)

average density of air in column BC = 0.0676 lb/ft^3 (1.082 kg/m^3)

(average of density at B and C)

The weighted average density of air

$$w = \frac{[0.0634 \times 2000] + [0.0676 \times 5 \times 5280] + [0.064 \times 3000]}{2000 + 5 \times 5280 + 3000}$$

$$= \frac{2103.44}{31400} = 0.0669 \text{ lb/ft}^3 \text{ (1.070 kg/m}^3\text{)}$$

Value of K to be used $= 100 \times 10^{-10} \times \dfrac{0.0669}{0.0750} = 89 \times 10^{-10}$ lb-min^2/ft^4 (0.0167 kg/m^3)

$$L = 31{,}400 \text{ ft (9571 m)}$$

$$A = 10 \times 10 = 100 \text{ ft}^2 \text{ (9.29 m}^2\text{)}$$

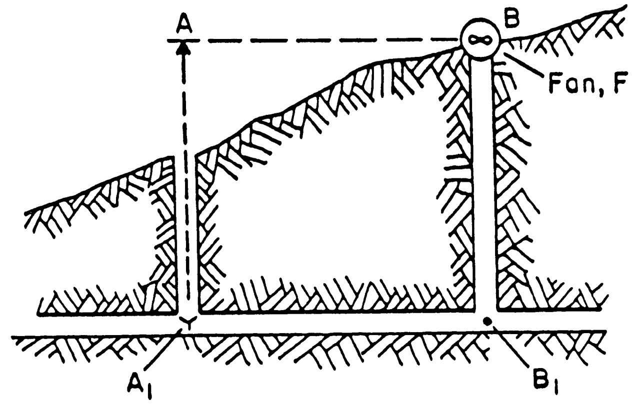
$$O = 2(10 + 10) = 40 \text{ ft (12.19 m)}$$

Using a value H = 1.1538 in. water (287.11 Pa) (calculation 2d) and $H = \dfrac{KLOQ^2}{5.2A^3}$,

$$1.1538 = \frac{89 \times 10^{-10} \times 31{,}400 \times 40 \times Q^2}{5.2 \times 100 \times 100 \times 100}$$

$$Q = 23{,}200 \text{ cfm (10.95 m}^3\text{/s)}$$

$$AHP = \frac{HQ}{6350} = \frac{1.1538 \times 23{,}200}{6350} = 4.22 \text{ hp (3.15 kW)}$$



11.6.9.3. Natural Ventilation Pressure and Fans

Since NVP acts on the same circuit as the fan, it can be treated as acting in series with the fan. However, it can act in the same direction as the fan and aid the fan or act in the opposite direction and hinder its performance. These effects can be demonstrated with the aid of mine, fan, and NVP characteristics.

Shown in Fig. 11.6.36 are the fan and mine characteristics without any NVP acting in the circuit. The head developed by the fan is H_1, and the quantity of air circulating is Q_1.

Fig. 11.6.37 shows the fan, NVP, and mine characteristics. When NVP is acting in the *same* direction as the fan, the combined characteristic is obtained by *adding* the NVP to the fan pressure at each quantity. The quantity circulating in the system is Q_2 with a head of H_2. However, the head developed by the fan is only H_f, which is lower than H_1.

Shown in Fig. 11.6.38 are the fan, NVP, and mine characteristics when NVP is acting in the *opposite* direction to the fan. In this case, the combined characteristic is obtained by *subtracting* the NVP from the fan pressure at each quantity. The quantity circulating is now Q_3 with a head of H_3. However, the head developed by the fan is H_f, which is higher than H_1. If NVP can significantly affect the operation of the fan, it is necessary to know its magnitude.

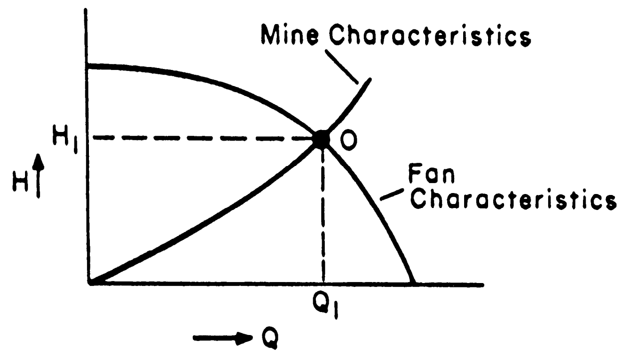


Fig. 11.6.36. Mine and fan characteristics for simple mine ventilation system without NVP.

11.6.9.4 Practical Application of Fan Laws and Air Densities

The application of the fan laws and the influence of air densities and NVP on fan performance is illustrated in the following example from Kingery (1960).

Example 11.6.15. The following data are available on the performance of two different fans:

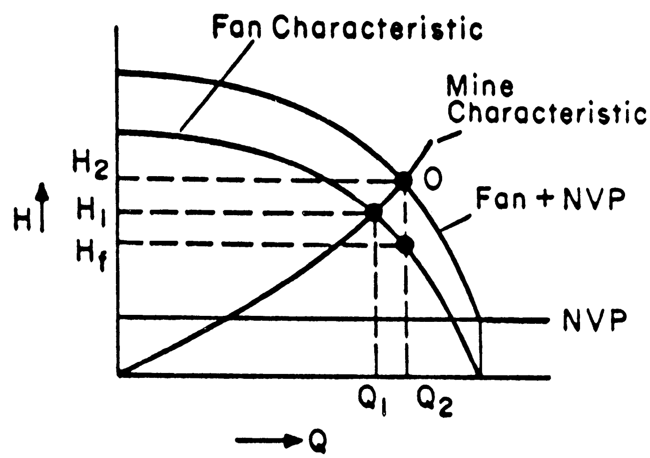


Fig. 11.6.37. Mine and fan characteristics with NVP acting in favor of the fan.

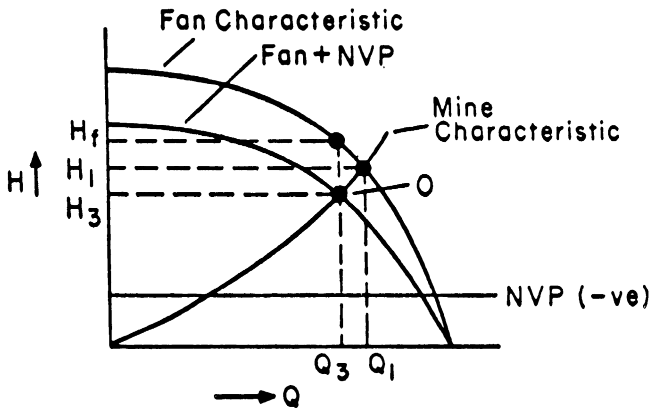


Fig. 11.6.38. Mine and fan characteristics with NVP acting against the fan.

	Fan A	Fan B
Speed, rpm	700	800
Density of air, lb/ft^3 (kg/m^3)	0.0700 (1.12)	0.0750 (1.20)
Head *H*, in. water (kPa) at following air quantities, cfm (m^3/s):		
150,000 (70.793)	4.90 (1.219)	6.55 (1.630)
160,000 (75.512)	4.62 (1.150)	6.05 (1.505)
170,000 (80.232)	4.16 (1.035)	5.45 (1.356)
180,000 (84.951)	3.64 (0.906)	4.70 (1.170)
190,000 (89.671)	3.04 (0.756)	3.85 (0.958)
200,000 (94.390)	2.29 (0.570)	2.80 (0.697)

These two fans are to be installed in series in a mine at an air density of 0.0700 lb/ft^3 (1.12 kg/m^3); both fans operate at speeds of 800 rpm. Mine resistance has been established at 3 in. water pressure (746.52 Pa) for 100,000 cfm (47.195 m^3/s) at a density of 0.0750 lb/ft^3 (1.2 kg/m^3). The natural ventilation pressure is 0.5 in. water (124.42 Pa) assisting fans.

What quantity will fans deliver? At what pressure will each fan operate?

Solution. The first step is to correct the pressure-quantity data for each fan to 800 rpm and a density of 0.0700 lb/ft^3 (1.12 kg/m^3). Fan A: No correction needed for density. The head-quantity characteristic for 800 rpm is developed using fan law 1.

$$Q_n = Q_o\left(\frac{800}{700}\right)$$

$$H_n = H_o\left(\frac{800}{700}\right)^2$$

Fan A: [Air density, 0.0700 lb/ft^3 (1.12 kg/m^3)].

Speed, 700 rpm		Speed, 800 rpm	
Quantity cfm (m^3/s)	Head in. water (kPa)	Quantity, cfm (m^3/s)	Head in. water (kPa)
150,000 (70.793)	4.90 (1.219)	171,500 (80.939)	6.40 (1.593)
160,000 (75.512)	4.62 (1.150)	183,000 (86,367)	6.03 (1.501)
170,000 (80.232)	4.16 (1.035)	194,300 (91,700)	5.43 (1.351)
180,000 (84.951)	3.64 (0.906)	206,000 (97.222)	4.75 (1.182)
190,000 (89.671)	3.04 (0.756)	217,000 (102.413)	3.97 (0.988)
200,000 (94.390)	2.29 (0.570)	229,000 (108.077)	2.99 (0.744)

Fan B: No correction necessary for speed. The head-quantity characteristic for w = 0.0700 lb/ft^3 (1.12 kg/m^3) is developed using fan law 3.

$$H_n = H_o\left(\frac{0.070}{0.075}\right)$$

Speed, 800 rpm; quantity, cfm (m^3/s)	Fan B Head, in. water (kPa) at density (kg/m^3) 0.0750 (1.20)	0.0700 (1.12)
		$\left(\frac{0.0700}{0.0750}\right) \times H_1$
150,000 (70.793)	6.55 (1.630)	6.11 (1.521)
160,000 (75.512)	6.05 (1.505)	5.65 (1.405)
170,000 (80.232)	5.45 (1.356)	5.09 (1.266)
180,000 (84.951)	4.70 (1.170)	4.39 (1.092)
190,000 (89.671)	3.85 (0.958)	3.59 (0.894)
200,000 (94.390)	2.80 (0.697)	2.61 (0.651)

Speed, 800 rpm; quantity, cfm (m^3/s)	Combined Series Curve Data, Fans A and B Density, 0.0700 lb/ft^3 — *H*, fan A* (in in. water (kPa))	*H*, fan B	*H*, combined
150,000 (70.793)	—	6.11 (1.520)	—
160,000 (75.512)	6.71 (1.670)	5.65 (1.406)	12.36 (3.076)
170,000 (80.232)	6.45 (1.605)	5.09 (1.267)	11.54 (2.072)
180,000 (84.951)	6.12 (1.523)	4.39 (1.092)	10.51 (2.615)
190,000 (89.671)	5.67 (1.411)	3.59 (0.893)	9.26 (2.304)
200,000 (94.390)	5.12 (1.274)	—	—

* Pressure-quantity data, fan A, taken from curve of fan A plotted from calculated data.

Influence of natural draft at 0.5 in. water (124.42 Pa)

Quantity, cfm (m^3/s)	*H*, combined	*H*, NVP in in. water (kPa)	*H*, total
160,000 (75.5 12)	12.36 (3.076)	0.50 (0.124)	12.86 (3.200)
170,000 (80.232)	11.54 (2.072)	.50 (0.124)	12.04 (2.996)
180,000 (84.951)	10.51 (2.615)	.50 (0.124)	11.01 (2.740)
190,000 (89.671)	9.26 (2.304)	.50 (0.124)	9.76 (2.429)
200,000 (94.390)	—	—	—

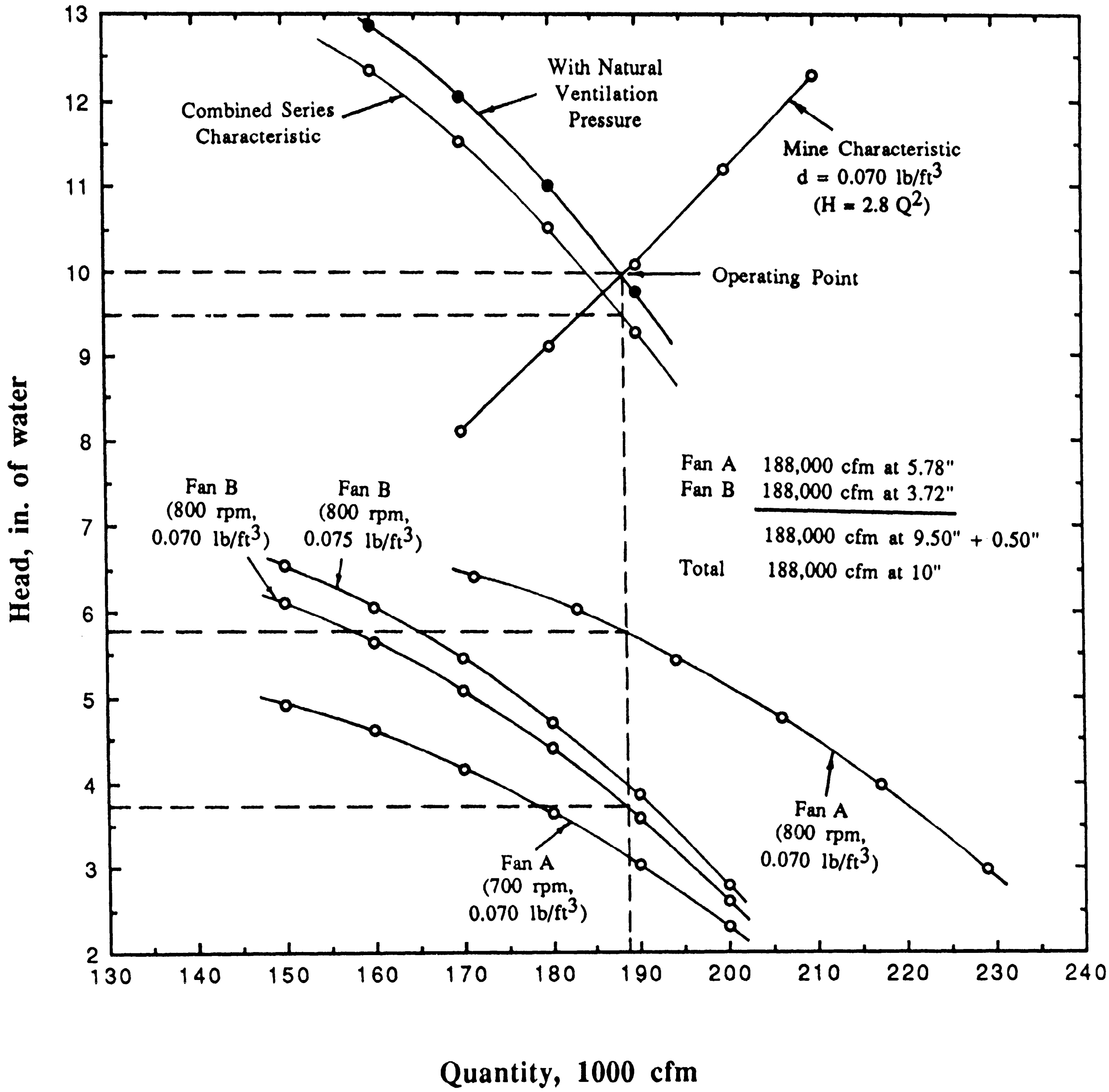
Plotting these data forms the total pressure-quantity relationship of all fans, plus natural draft (Fig. 11.6.39). The mine pressure *H* is 3 in. water (746.52 Pa) at 100,000 cfm (47.195 m^3/s) at standard density. Therefore, the pressure at a density of 0.0700 lb/ft^3 (1.12 kg/m^3) is $3 \times \dfrac{0.0700}{0.0750} = 2.8$ in. water (696.75 Pa).

$$R = \frac{H}{Q^2} = \frac{2.8}{(1.0)^2} = 2.8 \text{ in.-min}^2/\text{ft}^6 \ (0.313\ N\text{-}s^2/\text{m}^8)$$

The mine characteristic is calculated from the relationship $H = 2.8Q^2$.

Air quantity, cfm (m^3/s)	Head, in. water (kPa)
170,000 (80.232)	8.1 (2.016)
180,000 (84.951)	9.1 (2.264)
190,000 (89.671)	10.1 (2.513)
200,000 (94.390)	11.2 (2.787)
210,000 (99.110)	12.3 (3.061)

Plotting these data on Fig. 11.6.39, it is seen that the total air quantity of 188,000 cfm (88.73 m^3/s) at a combined pressure



Conversion factors: 1 cfm = 0.47195 × 10^{-3} m^3/s, 1 in. of water = 248.84 Pa, 1 lb/ft^3 = 16.018 kg/m^3.

Fig. 11.6.39. Fan, NVP, and mine characteristics for Ex. 11.6.15.

of 10.0 in. water (2.488 kPa) is flowing through the mine. The total pressure is distributed as follows:

	Head in. water (kPa)
Fan A	5.8 (1.443)
Fan B	3.7 (0.921)
Combined	9.5 (2.364)
Natural draft	0.5 (0.124)
Total head	10.0 (2.488)

11.6.10 VENTILATION SURVEYS

Comprehensive ventilation surveys are necessary to determine if the mine ventilation system meets statutory requirements, to decide what improvements in the current ventilation system are needed, and to enable planning for future expansion. Routine measurements made to check on the air quantity in a split or the amount of methane in the workings do not qualify as comprising a ventilation survey. Four major areas are included under the general heading of ventilation surveys: (1) air quantity survey, (2) pressure survey, (3) temperature survey, and (4) air quality survey (Luxbacher, 1977).

In the temperature survey, the measurements may include virgin rock temperatures and the dry- and wet-bulb temperatures. Often the operation of the fan must also be examined. In fact, the common practice is to conduct air quantity, pressure, and temperature surveys concurrently. Air quality and fan surveys are conducted as and when necessary along with these other surveys.

11.6.10.1. Air Quantity Surveys

In most cases in an air quantity survey, the volume flow rate of air is not measured directly but is calculated from velocity and area measurements at selected cross sections of the mine airway, using the relation (see Eq. 11.6.13),

$$Q = V \times A \tag{11.6.50}$$

where Q is quantity of air flowing in cfm (m^3/s), V is velocity of airflow in fpm (m/s), and A is area of the airway section in ft^2 (m^2).

To measure velocity, the physical effects of air in motion are used. Among these are (1) mechanical effects, (2) variations in air pressure, and (3) the cooling power of the air. Instruments that use mechanical effects to determine velocity include (1) the smoke tube, (2) the velometer, and (3) the anemometer, which is the principal instrument used since the early 1900s. Among the instruments that use pressure variations to determine velocity, the most common one is the pitot tube. Instruments such as kata thermometers and hot wire anemometers that use the cooling power of the air in motion to determine velocity are not generally used for that purpose in mines. The characteristics of several air-measuring devices are summarized in Table 11.6.9. The particular instrument chosen for velocity measurement is usually dictated by the range of air velocities anticipated [low < 150 fpm (0.76 m/s), medium 150 to 1000 fpm (0.76 to 5.08 m/s), or high > 1000 fpm (5.08 m/s)] and the accuracy desired. The accuracy of a quantity measurement is a function of the accuracy of the velocity and area measurements. Errors in velocity and area measurements can, of course, be compensating or compounding.

SINGLE- AND MULTI-POINT TRAVERSES. Whatever the instrument used, there are several methods by which the velocity of the air in an airway can be measured: (1) single-point centerline measurements, (2) multi-point measurements at predetermined points (fixed traversing), and (3) continuous traversing.

Because of the variation in velocity over the cross section of an airway, single-point measurements must be corrected to determine the average velocity of the air in an airway. This is usually found to be 80 to 90% of the measurement taken at the centerline. For day-to-day comparisons of velocity, single-point measurements taken at the same place (with no change in cross section) are satisfactory. For more accurate determinations, traversing methods, either multi-point (fixed) or continuous, must be employed.

Fixed-point traversing is warranted only when very precise ventilation surveys are needed such as for fan tests and shaft measurements. The total area of airflow is broken down into small equal areas for measurement purpose. Precise positioning of the instrument is then required at each of these measurement points. Fig. 11.6.40a shows the arrangement for fixed-point traversing in rectangular and circular airways. The rectangular airway is divided into 16 equal areas with more measurement points in the peripheral areas than in the areas in the center. Average velocity for each area is calculated and then the overall average velocity in the airway is taken as the average of the average velocities for all the areas. The number of areas should not be less than 24 for a section in which one dimension is twice the other.

For a circular airway or duct, the area is divided into a number of concentric but equal areas, and readings are taken at the center of each area along the horizontal and vertical diameters (i.e., four readings in an area). In Fig. 11.6.40b, the area is divided into five concentric areas with 20 measurement points. The velocity in the airway is the average of the 20 velocities.

INSTRUMENTS. Two of the most commonly used velocity instruments in mines are now described.

Vane Anemometer—The vane anemometer is a small windmill geared to a mechanical counter. The gearing converts the spindle revolution into a measure of lineal distance. The windmill is rotated by the moving air, and the air velocity is calculated by taking an observation over a measured period of time. To make a measurement, the gearing is engaged to the counter by a clutch. In some recent models, in place of a conventional clockwork mechanism, a low-friction digital counter is used. The common commercial anemometer is about 4 in. (100 mm) in diameter.

Anemometer readings are virtually unaffected by the air densities and air temperatures encountered in underground work. For routine measurements, the instrument may be handheld. However, for precise work, it should be mounted on an extension rod to reduce the effect of the hand, arm, and body of the observer on the airstream.

The anemometer registers a velocity somewhere within ± 10% of the true velocity. Calibrations furnished by the manufacturers are usually in the form of tables of plus or minus corrections to the observed velocity. Kingery (1960) recommends fitting a linear equation to the manufacturer's correction factors and using the equation to correct the observed velocity to true velocity. If the anemometer is to be used for very precise measurements, it is recommended that it be sent to the manufacturer for calibration prior to use.

The auxiliary equipment necessary to determine air quantity with the anemometer includes anemometer extension rods (if required), a stopwatch, and a tape measure. Measurements should be made at cross sections that are in straight lengths of airway. They should not be taken near sections where there are bends or changes in area.

Pitot Tube—The pitot tube is a primary standard instrument for determining velocities. It is made up of two coaxial tubes, one arranged to measure the total pressure and the other to measure the static pressure through small holes at right angles to the flow (Fig. 11.6.41). The head containing the holes for detecting total and static pressures is carried on a stem of suitable length set at right angles to the head. When the two tubes are connected across a manometer, the difference between the static and total pressure (i.e., the velocity pressure) is measured. The velocity of the air can then be found from solving Eq. 11.6.5 and 11.6.5a for:

$$V = 1098\sqrt{H_v/w}$$

$$V = \sqrt{2gH_v/w}$$

where V is air velocity in fpm (m/s), H_v is velocity pressure in in. water (Pa), and w is density of air in lb/ft^3. In Eq. 11.6.5a w/g is in kg/m^3.

While the facing holes reflect accurately the total pressure, the holes for static pressure are subject to some interference by the facing tube and stem, and have to be sized and located correctly (Drummond, 1974). Commercial types are generally accurate to within 1%, and specially made types can be accurate

Table 11.6.9. Characteristics of Air Velocity Measurement Instruments

Instruments	Range, fpm	Hole Size (for ducts)	Range, Temp. *	Dust, Fume Difficulty	Calibration Requirements	Rugged-ness	General Usefulness and Comments
PITOT TUBES with inclined manometer	600–up	3/8″	Wide	Some	None	Good	
Standard	600–up	3/8″	Wide	Some	None	Good	Good except at low velocities
Small Size	600–up	3/16″	Wide	Yes	Once	Good	Good except at low velocities
Double	500–up	3/4″	Wide	Small	Once	Good	Special
SWINGING VANE ANEMOMETERS							
Alnor Velometer	25–10,000	1/2″–1″	300°F	Some	Frequent	Fair	Good
ROTATING VANE ANEMOMETERS							
Conventional	30–10,000	Not for duct use	Narrow	Yes	Frequent	Poor	Special; limited use
Electronic (Airflow Development)	25–200 25–500 25–2000 25–5000	Not for duct use	Narrow	Yes	Frequent	Poor	Special; can record; direct reading
Digital (Airflow Development)	200–5000	Not for duct use	Narrow	Yes	Frequent	Poor	Special
BRIDLED VANE ANEMOMETERS							
Florite Air Velocity Meter	50–2500	Not for duct use	Narrow	Yes	Frequent	Fair	Special
HEATED WIRE ANEMOMETERS							
Anemotherm Model 60	10–8000	3/8″	Medium	Some	Frequent	Poor	Good
Anemotherm Gas Flow Meter	10–5000	1″	300°F	No	Frequent	Good	Not portable; for permanent station air flow
Flowtronic Air Meter 55A	0–1000 1000–2000 2000–4000	1/2″	Medium	Yes	Frequent	Poor	Good
HEATED THERMOCOUPLE ANEMOMETERS							
Alnor Thermo-anemometer Model 8500	10–2000 2 scales	5/16″	Narrow	Yes	Frequent	Poor	Good
Hasting Precision Air Meter B-22	10–500 500–10,000	5/16″	Narrow	Yes	Frequent	Poor	Good
Flow Corporation Series 800	10–4000	1/2″	Narrow	Yes		Poor	Good
Alnor Air Velocity Transducer System (AVT)	20–500 50–1000 100–2000	5/16″	Narrow	Yes	Frequent	Poor	Special; for permanent station use
VARIABLE AREA METER							
Airmeter F. W. Dwyer Co.	200–1200 1000–4000	Not for duct use	Narrow	Yes	Occasional (needs cleaning)	Good	Satisfactory for estimates of flow
VORTEX SHEDDING INSTRUMENT							
J-Tec (Mine Applications)	60–3000	Can be miniaturized	Wide	Occasional cleaning of the post	Twice a year or less	Good	Fixed-point automatic mine monitoring

* Range of Temperatures: Narrow - 20–150°F
Medium - 20–300°F
Wide - 0–800°F

Source: Anon., 1976.

Conversion factors: 1 fpm = 0.005080 m/s, 1 in. = 25.4 mm, °C = 5/9 (°F – 32).

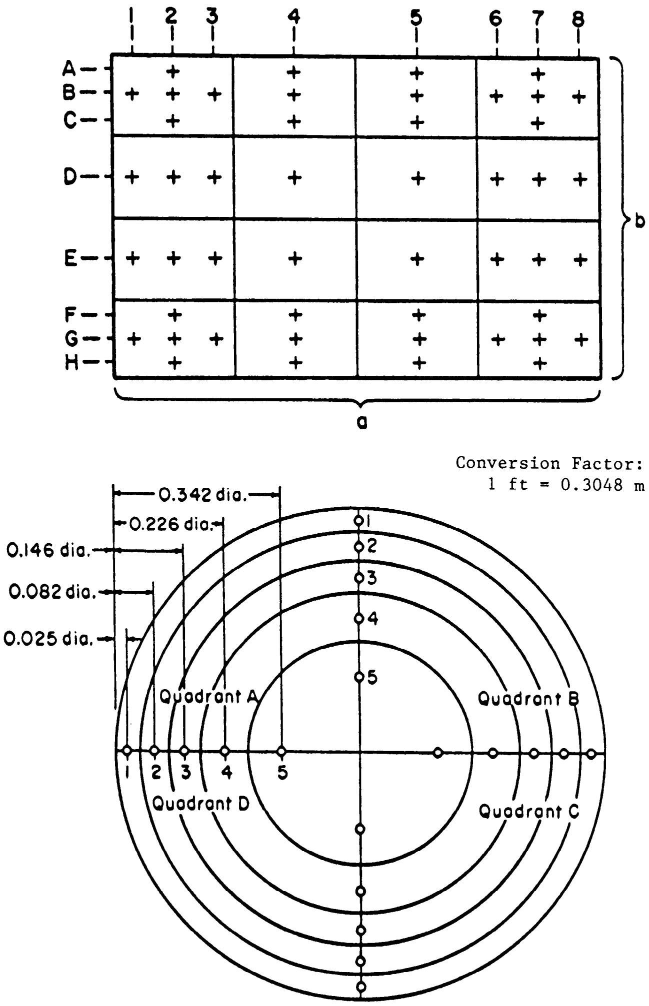


Fig. 11.6.40. Measurement points for fixed traversing in (a) rectangular and (b) circular airways.

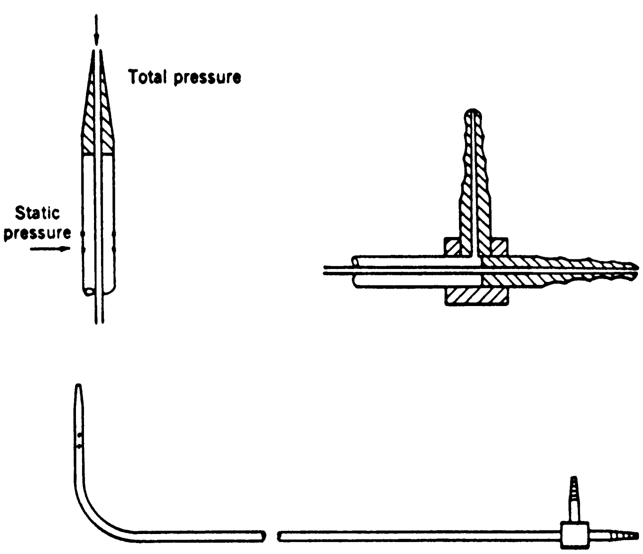


Fig. 11.6.41. Illustration of a pitot tube.

to within 0.01%. Since velocity pressures in mines are ordinarily small, the use of the pitot tube is limited to high-velocity determinations at fans and in ducts and tubing. At velocities below 1500 fpm (2.5 m/s), the accuracy of pitot-tube measurements is not very high. However, anemometers can be used in this velocity range with satisfactory results.

11.6.10.2 Quantity Measurement by the Tracer Gas Technique

Using the tracer gas technique, the quantity of air flowing in an airway is calculated directly. The method is advantageous where areas are irregular and the flow is not steady. The test gas is usually selected on the basis of (1) the ease with which it can be detected and analyzed, (2) its not being present in the normal mine air, (3) its not being absorbed by chemical or physical means in the airway, (4) non-reactivity with other gases in mine air, and (5) non-toxicity or explosiveness. Gases which have been considered suitable include ozone, hydrogen, carbon dioxide, and sulfur hexafluoride (SF_6).

There are two methods of using the tracer gas technique. In the first method, the tracer gas is continuously metered into an airway. After thorough mixing has occurred and equilibrium has been established, air samples are taken at a point downstream. The concentration of the tracer gas is determined in the sample. The rate of airflow is calculated as

$$Q = \frac{Q_g}{C} \qquad (11.6.51)$$

where Q is quantity of air flowing in cfm (m^3/s), Q_g is feed rate of the tracer gas in cfm (m^3/s), and C is concentration of the tracer gas in $\left(\frac{\%}{100}\right)$.

In the second method, a known mass (or volume) of the gas is injected into the airstream, and its concentration at a downstream point is sampled either continuously or as often as possible until the concentration can no longer be measured. The quantity of tracer flowing through the sampling point at any time interval $(t,\ t + dt)$ is given by $QC_t \times d_t$, where C_t is concentration at time t, and Q is the airflow. Thus, over the time interval $(t_o,\ t_f)$,

$$Q_g = \int_{t_o}^{t_f} Q\, C_t\, dt \qquad (11.6.52)$$

where t_o is arrival time of first measurable concentration, and t_f is time after which the concentration is not measureable.

If C_t were plotted against t, the curve would look like that shown in Fig. 11.6.42 where

$$\int_{t_o}^{t_f} C_t\, dt$$

is the area under the curve and

$$Q = \frac{Q_g}{\int_{t_o}^{t_f} C_t\, dt}$$

On the other hand, the integration of $\int_{t_o}^{t_f} C_t\, dt$ can be done by simply taking the average of the measured concentrations (C_{ave}),

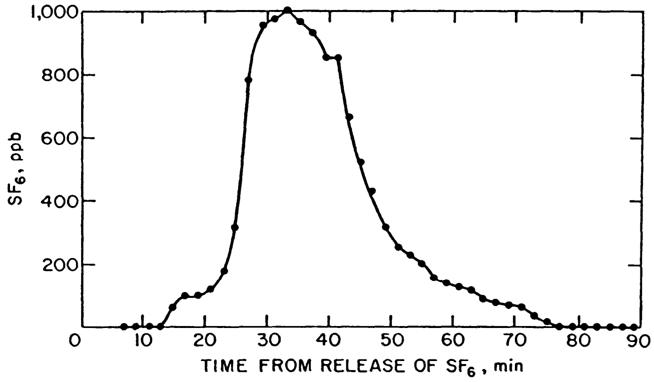


Fig. 11.6.42. Concentration of SF_6 at two-minute intervals (Ex. 11.6.16)

and multiplying the average by the total time $(T = t_f - t_o)$ which measurable amounts of concentration are found at the sampling point.
Then

$$Q_g = Q \int_{t_o}^{t_f} C_t \, dt = Q \times C_{ave} \times [t_f - t_o] = Q \times C_{ave} \times T$$

from which

$$Q = \frac{Q_g}{C_{ave} T} \qquad (11.6.53)$$

Example 11.6.16. Thimons and Kissell (1974) report that, when they released 0.371 ft^3 (0.011 m^3) of sulfur hexafluoride in a mine and monitored its concentration in the return, of the 45 samples taken at 2-min intervals, only 31 samples contained measurable quantities of the gas (Fig. 11.6.42). Average concentration in the 31 samples of the gas is calculated as 3.77×10^{-7} ft^3/ft^3 (m^3/m^3) of air. The total sampling time was 62 min. Therefore, the quantity of air flowing at the measurement point is calculated as

$$Q = \frac{0.371}{3.77 \times 10^{-7} \times 62} = 16{,}000 \text{ cfm } (7.55 \text{ m}^3/\text{s})$$

The tracer gas technique has potential applications in measuring air quantities in problem areas such as recirculation of return into intake air, leakage from adjacent mines, "lost" intake air, and unknown transit flow times through stoped or gobbed areas.

11.6.10.3 Pressure Surveys

Analysis of an existing mine ventilation system, including the evaluation of modifications to the system, requires accurate input data that can be developed *only* by a detailed pressure-quantity survey in the mine (McElroy and Kingery, 1957). The purposes of an accurate underground pressure survey are to: (1) obtain a pressure gradient along the circuit or circuits under investigation, and (2) determine the values of friction factor for various types of airways.

The pressure gradient aids in determining regions of excessive resistance and the feasibility of correcting the conditions causing them by cleaning airways, driving additional ones, or modifying existing ones. The pressure gradient is also useful for predicting the effect on other parts of the system when adjusting air regulation on any one split or when planning a new air shaft, slope, or drift. When projecting new mine development, high-precision work is required to establish these vital data for predicting the characteristics of mine pressure and air quantity.

Pressure measurements in underground mines can be made on either an absolute or a differential basis. Measurements made on an absolute basis at each station are subtracted, one from the other, to find the head loss between stations.

PRESSURE SURVEY INSTRUMENTS. There are several types of instruments available for measuring both absolute and differential pressures. Absolute pressures are generally measured by the use of altimeters or aneroid barometers while U-tubes or magnehelics are used to measure differential pressure.

Absolute Pressure Measurement—In making absolute pressure measurements with a barometer, the pressure reading obtained at a particular point, called the barometric pressure, is dependent upon the altitude and the gravitational force at that point and the prevailing atmospheric conditions. There are two common types of barometers, mercury and aneroid. Aneroid barometers are commonly used underground because mercury barometers are too bulky and fragile. An aneroid barometer calibrated in terms of elevation is called an altimeter. Since the response of each capsule to atmospheric pressure variations is slightly different from that of any other capsule, the reading scale for each instrument is unique to it. As a result, recalibration costs for aneroids tend to be high. Two types of aneroid barometers are widely used for mine pressure survey work, a direct-reading type manufactured by Wallace and Tiernan and an indirect-reading type manufactured by American Paulin System.

The equation used in all survey altimeter calibrations in the United States is taken from the Smithsonian Meteorological Table Number 51 (Anon., 1931):

$$H = 62583.6 \log (29.9/B) \qquad (11.6.54)$$

$$H = 19075 \log (101.3/B) \qquad (11.6.54a)$$

where H is elevation above sea level in ft (m), and B is barometric pressure in in. mercury (kPa). The formula assumes an air temperature of 50°F (10°C), no humidity (dry air), and a sea level atmospheric pressure of 29.92 in. mercury (101.325 kPa). Therefore, when readings are taken with an altimeter, corrections must be made for deviations from the assumption of dry air at 50°F (10°C). American Paulin System altimeters incorporate a correction for humidity in the dial of the instrument and use of volumetric gas expansion formula to correct for changes in temperature. Wallace and Tiernan altimeters utilize the density ratio of 50°F (10°C) dry air to the actual air density for this correction. The scale on the Wallace and Tiernan altimeter is offset by 1000 ft (304.8 m) to permit use below sea level. Charts for conversion of altimeter readings to direct units of pressure in in. mercury, and conversion factors for converting pressure differences due to flow from ft (m) of air column of altimeter scale density to in. water (pascals) are available.

Differential Pressure Measurement—Precision differential pressure measurements are usually made with a manometer. In this instrument, the difference between the pressure applied to two different surfaces of a liquid is balanced by the displacement of the liquid. Correct choice of the manometer liquid will avoid problems that may be encountered due to (1) a poorly formed meniscus, (2) poor surface tension, (3) poor visibility of the fluid level, and (4) changes in the density of the fluid.

The most important of the several types of manometers are the vertical U-tube manometer, the reservoir-type vertical manometer, and the inclined-tube manometer. The modern inclined-tube manometer comes with an adjustable limb that can be set at two or three different inclinations with alternative measuring scales for each inclination.

Among the devices available for measuring differential pressure underground, the magnehelic manometer is the most useful for underground spot surveys. The instrument is, strictly speaking, an aneroid barometer. It consists of a diaphragm across which the pressure difference is applied. This causes the diaphragm to move against a spring. At the end of the diaphragm is a magnet adjacent to which is a helix that is free to turn so as to maintain the minimum air gap between the magnet and helix. In effect, movement of the diaphragm through this magnetic linkage causes a rotation of the helix which can be read on a dial. Magnehelics are available with several different pressure ranges.

PRESSURE SURVEY METHODS. There are two methods of conducting pressure surveys. In the first method, the manometer survey, the differential pressure between two points is directly measured with a manometer. The basic requirements are two hoses connecting the two points to the limbs of a manometer. When the object is to measure the static pressure drop between the two points, a static-pressure measuring connection should be used at the extremities of the hoses. Manometer surveys are recommended where extreme precision is required such as for determining friction factor values and calculating pressure drops across doors or for surveys where handling of the hose would not be a problem. An essential precaution is guarding against the formation of condensate in the hose connections when they are moved from warm humid air to cold air. For friction factor determination, a sling psychrometer and a barometer should also be used to determine wet-bulb and dry-bulb temperatures and the barometric pressures. Additionally, velocity and area measurements are required for calculating the flow volume of air. Manometer surveys have found limited application in the United States due to the extensive size of most mines. In other countries, the use of the manometer is preferred over that of the altimeter for pressure surveys.

Altimeter Survey—Absolute pressure measurements made with an altimeter or barometer at the various stations in the mine must be compared to find the head loss or pressure differential between the stations. The instruments could be used to directly determine the pressure difference, in ft (m) of a theoretical 50°F (10°C) dry air column, in. mercury (pascals), or any other scale such as in. water (pascals) between any two points in a ventilation system if the two points were at the same elevation, had the same air density and airflow velocities (or velocity head), and the measurements were made simultaneously. Unfortunately, these conditions rarely exist during underground pressure surveys, and corrections must be applied. Two altimeter survey methods are used, differing only in the treatment of the time frame between measurements. These are the simultaneous measurement method and the fixed-base method, both of which require two altimeters.

In the simultaneous method, measurements are made simultaneously at two adjacent stations (stations 1 and 2). Immediately after the measurement, the observer at the rear instrument (station 1) moves past the other observer to the next forward station (station 3). The procedure is repeated for each successive station, and the technique has come to be known as "leap frogging" (Fig. 11.6.43).

The simultaneous method is more accurate than the fixed-base method as it avoids possible changes in air conditions and obviates the necessity of correcting readings for changes in atmospheric pressure. However, it is slow because two readings must be taken at each station. The logistics of alternate station moves

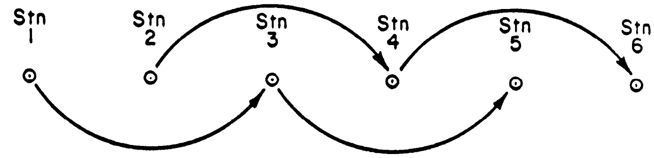


Fig. 11.6.43. Leap-frogging method of pressure surveying.

for the observer add difficulty to its use. It is still a useful procedure for small mines or portions of mines such as shafts.

In the fixed-base method, a base is established at which an altimeter (base instrument) is read at short intervals of time while readings are being taken with the other instrument (roving instrument) at all traverse stations in succession. The difference in the readings of the two instruments at adjacent stations is corrected for temperature, elevation, and atmospheric pressure differences that exist at the time of the measurements.

When the survey involves a closed pressure loop, starting and finishing at the same point, then in the absence of NVP and errors in the survey, the final pressure should be zero. However, a final instrument reading of 3 to 4 ft (0.9 to 1.2 m), representing the error of closure, can be expected. A small error of closure can result due to compensating errors in elevation measurements, altimeter readings, and velocity measurements. Therefore, accuracy of a survey can be established only by independent surveys and spot checks.

PRESSURE SURVEY CALCULATIONS. The method of calculating the survey results for the fixed-base method is explained in a US Bureau of Mines publication (McElroy and Kingery, 1957). With the results of the pressure survey, the following formula is used for calculating the difference in total pressure (head loss or head gain) between two stations:

$$DTP = \frac{-\left[(A_2 - A_1) - (B_2 - B_1) - \dfrac{(E_2 - E_1)}{DR}\right]}{CONFAC} + \frac{V_2^2 - V_1^2}{(4008)^2} \tag{11.6.55}$$

$$DTP = \frac{-\left[(A_2 - A_1) - (B_2 - B_1) - \dfrac{(E_2 - E_1)}{DR}\right]}{CONFAC} + \frac{w(V_2{}^2 - V_1{}^2)}{2g} \tag{11.6.55a}$$

where DTP is difference in total pressure between station 2 and station 1 in in. water (Pa); A_2, A_1 are roving altimeter readings, in ft (m) of air; B_2, B_1 are base altimeter readings in ft (m) of air; E_2, E_1 are elevations of stations in ft (m); V_2, V_1 are velocities of air in fpm (m/s); DR is average density ratio between stations 1 and 2; and $CONFAC$ is ft of air per in. water (m of air per Pa) pressure at the average altimeter reading between the two stations. Charts are available for the calculation of DR and $CONFAC$.

COMPUTER PROGRAMS. The altimeter survey calculations, though simple, are tedious and subject to arithmetical errors, particularly when extensive altimeter surveys are involved. Because of the sequential and repetitive nature of the calculations, the method can be easily programmed for a digital computer or a desk calculator and, for small surveys, for a hand-held calculator with small memory (Chapter 11.10). In view of the computer's ability to perform calculations at a very rapid rate and store information accurate to more than three or four decimal places, exact equations for conversion of velocity to velocity heads, for density determinations, and for conversion of ft (m) of air to in. water (Pa) can be used.

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Chapter 11.7
MINE VENTILATION DESIGN

BRUCE JOHNSON AND RAJA V. RAMANI

11.7.1 METAL MINE VENTILATION

BRUCE JOHNSON

Metal mine (or noncoal mine) as used here refers to both metal and nonmetallic mines, including uranium mines. In most cases, these mines are not classified as gassy, that is, containing methane, although there may be other gases present such as radon or sulfur dioxide. The biggest differences in ventilation techniques between coal and nongassy metal mines in the United States are the permitted use of underground fans and recirculation or series utilization of air in metal mines.

Much of the discussion that follows is based on O'Neil and Johnson (1982). For coverage of ventilation principles, see Chapter 11.6; for air conditioning see Chapter 11.4.

11.7.1.1 General Considerations

AIR QUANTITIES. In most cases, air quantities are not determined by statute for metal mines, the exception being mines where radioactivity is present. Air quantities where diesels are used are determined from manufacturers' specifications or by testing and certification done on individual units by the US Bureau of Mines (USBM). For older uncertified units, it is generally satisfactory to allow 150 ft^3/min/bhp or cfm/bhp (1.0 m^3/s/kW) for each unit. For several units on the same split of air, the USBM recommends the full quantity requirement for the first unit, 75% of full quantity for the second unit, and 50% of the full quantity for each additional unit.

Otherwise, air quantities are usually determined by practical considerations and experience. Velocities even in main airways should not exceed 1500 ft/min or fpm (7.6 m/s) wherever possible, both for comfort and to reduce head losses due to friction, but this is not always possible where a large quantity of air may be needed to service an area.

Air velocities at a minimum always should be kept above 50 fpm (0.25 m/s), as this is about the lowest velocity at which air movement can be sensed; however, this requirement can be abrogated in large service bays.

In working places, air velocities should be 200 to 400 fpm (1.0 to 2.0 m/s) in stopes and 400 to 600 fpm (2.0 to 3.0 m/s) in drifts. These velocities are based on the author's experience in warm mines and may have to be reduced in cold mines.

OVERALL MINE LAYOUTS. In a shaft mine, intake air should be directed down the main man-materials shaft(s) and exhausted up some other shaft or shafts, preferably shafts sunk exclusively for exhaust. In an adit mine, the intake would normally be the adit(s) where people enter. Sometimes, a separate adit for exhaust would be used, which would be connected to the workings by raises and winzes. Some mines are close to the surface, and multiple boreholes can be used for ventilation.

AIR MOVING DEVICES. It is best, when possible, to have main fans located on the surface. This way, they can be accessed easily in emergencies, and routine maintenance or major repairs are facilitated. If the mine is hot, exhaust fans are preferred in order that the fans will not themselves add heat to the air. Where the mine is cold, the reverse would be true.

Generally, surface exhaust fans are preferred. However, where the mine contains numerous old open workings, exhaust fans will draw excessive air through areas where it is not needed. Sometimes the best solution is to have main surface exhaust fans supplemented by underground booster fans. If the exhaust shafts are also the hoisting shafts, surface location of the fans is impractical because of the design problem of moving skips through fan plenums. All things considered, though, the location of main fans on the surface is to be preferred.

BOOSTER FANS. Unlike coal mines, metal mines can usually use booster fans underground to either divert airflow or to give additional air moving power where needed. Booster fans are often needed in a situation where there are widely scattered areas being served by main fans. To facilitate distribution of air to remote areas, either a booster fan to move air to these areas is needed, or a larger main fan will have to be used to pressurize all areas. If a larger main fan is used, restrictions will then be needed in non-remote areas to waste the unneeded pressure to these areas and balance the system. Clearly, the second solution may be an unnecessary expense when boosters are applicable, although all possible solutions should be considered.

AUXILIARY FANS. These are relatively small fans used in directing air to individual faces or small areas. Where dead-end faces are to be ventilated, the air is delivered to (or exhausted from) the faces with duct connected to the fan.

In the case of large openings, auxiliary fans are often hung in main airstreams to divert the air into these openings with little or no duct connected to the fan. This is a common technique in room (stope) and pillar systems. A fan used in this manner is termed a "jet fan," as it not only moves the air that is passing through the fan but causes a vena contracta that sucks in additional air and moves it as well.

COMPRESSED AIR MOVERS. These work in the same manner as jet fans, but use a stream of compressed air emanating from an eductor to form the vena contracta. Compressed air movers are expensive from a power consumption point as it takes quite a bit of power to compress the air that they consume, but they are often applicable where only small amounts of air are needed and where fans may be impractical. At least one large American mine uses them extensively for secondary development and production ventilation.

REUSE OF VENTILATION AIR. Reuse of air is allowable in most metal mine situations, especially where the used air is mixed with fresh air, and this is normally preferred for economic reasons to using air only once. In most cases, there will not be much gas contamination, and oxygen depletion will be minimal. However, where dust or heat is a problem, filtration or cooling may first be needed. South African mines commonly use fume filters to extract nitrous oxide fumes from development blasting when blasting is done on shift, and also use baghouse filters at ore transfers located at intake shafts. However, these techniques are seldom employed in this country.

COURSING OF AIR. The normal way of coursing air is to direct the air to a stope raise and upcast it to the level above (a technique termed *ascentional ventilation).* This technique is used for two reasons: (1) most mines practice some sort of overhead mining method in which ore extraction is commenced at or just above the sill and proceeds upwards with successive cuts; and (2) in most mines, the rock is warmer than the ventilation air, so air is warmed as it rises, and this "chimney effect" aids airflow.

The design of ventilation for an individual stope is usually simple; the problem lies in the handling of air in a multilevel mine. An old practice, formerly used in the Coeur d'Alene, was to direct all air down the intake shaft to the lowest level of the mine, course it to the raises, and let it rise up throughout all the workings to the top active level, then direct it to the exhaust shaft(s).

This is an easy ventilation layout, but it has the obvious disadvantage of the upper stopes getting nothing but exhaust air from the lower stopes. While reuse of air is usually desirable for maximum utilization, there is also generally enough contamination with dust, gas, and heat that some addition of fresh air is necessary. Now if this addition is carried out on each level, with successive additions of fresh air, the upper stopes will have much more air than they need; besides, there are usually fewer stopes on the upper levels. This problem can be alleviated by directing some of the air up special exhaust raises or unused old stope raises.

Another solution is to have only alternate levels as intake levels, with the in-between levels serving as exhaust levels. This is a better method of handling air but necessitates special exhaust raises at the ends of the exhaust levels if the exhaust levels do not connect directly to the exhaust shaft(s). The disadvantage of the fresh air moving downward rather than upward in some areas is relatively slight and can easily be overcome with small exhaust fans.

11.7.1.2 Design Procedure

Generally, the design procedure for a metal mine is similar to that of a coal mine, except that devices such as booster fans are not presently allowed in American coal mines. For a design example, see 11.7.1.7.

1. Select main ventilation inlets and outlets.
2. From mining plans, determine the airflow requirements, and design tentative ventilation schematics. This must be done for various stages in the life of the mine.
3. Tentatively specify main and booster fan locations.
4. Determine airflow resistances for all branches, and build a computer model of the mine ventilation system.
5. Make simplified head loss calculations to initially determine fan characteristics.
6. Run a computer simulation of the ventilation system.
7. Adjust resistances, fans, etc., where needed, and rerun the model until computer output shows the model to be viable.
8. Consider alternate setups, and run alternate computer simulations on these alternates.
9. Review the planned setup with design personnel, and look over the plan from the standpoint of safety, for example, in case of fire in various areas, the time that would be needed for personnel working in remote parts of the mine to reach fresh air.

11.7.1.3 Ventilation Layouts for Specific Mining Systems

For details on the following mining methods, see Sections 18, 19, and 20.

OPEN STOPE: ROOM AND PILLAR MINING. Ventilation of this type of method is practically identical with that of coal mining. The main difference is the normally permitted use of underground fans in metal mining. Usually, diesels are employed and air quantities are predicated on the diesel units' requirements. Jet fans are frequently used to good advantage to direct air into rooms.

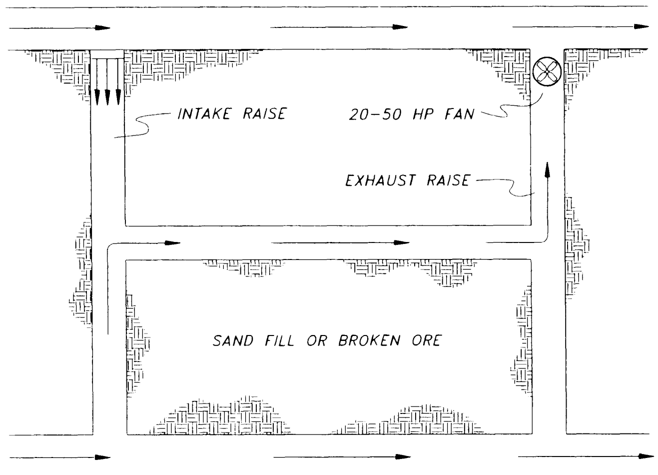

Fig. 11.7.1.1. Ventilation layout for horizontal cut-and-fill and shrinkage stopes. Conversion factor: 1 hp = 0.746 kW.

HORIZONTAL CUT AND FILL AND SHRINKAGE STOPES. This type of stope, at least theoretically, is the easiest to ventilate if developed in the normal manner (see Fig. 11.7.1.1) because there are raises on each end of the stope that are connected after the initial sill cut is made. The normal ventilation pattern is to bring air up one raise to the mining level of the current cut, direct it through the stope to the other raise, then up to the next level. Motive force for air movement between levels is best supplied by small exhaust fans, 20 to 50 hp (15 to 37 kW), at the tops of exhaust raises. Relying on brattices on the levels to maintain the desired pattern of airflow often results in poor distribution.

The initial sill cut will, of course, be dead-end until breakthrough into the next raise and must be ventilated as a breast stope, although if there are frequent open chutes that have already been driven up to the sill level, this will not be necessary.

BREAST STOPING METHODS: SQUARE SET, UNDERCUT AND FILL, SUBLEVEL CAVING. These methods are here classified as "breast stoping" because they all involve dead-end areas to be ventilated, as do drift headings, and these dead-end areas must be ventilated with auxiliary fans and ducts. Usually flexible rather than rigid duct is used because of the greater ease of handling of flexible dust and because of the brevity of life of the duct installation. If only one raise between levels is available, the air is directed up to the raise to an auxiliary fan at the sublevel being driven, coursed to the face, then exhausted back to the raise and up to the level above. Exhaust is preferably effected with an exhaust fan at the top of the raise.

Where multiple headings must be ventilated off the same raise, problems inevitably occur, especially if dust, heat, or exhaust gases are present (Fig. 11.7.1.2). Sometimes two compartments in the raise can be used, one dedicated to intake, the other to exhaust.

SUBLEVEL STOPES AND VERTICAL CRATER RETREAT (VCR) STOPES. These stopes are similar from a ventilation standpoint as they require sublevels. Air for the sublevels should be brought up the access raise, across the sublevel, and in the case of a sublevel stope, exhausted up through the open area. In a VCR stope, a second raise should be provided.

Draw in these systems is either from slusher trenches cut above the main level, or from extraction chambers crosscutting

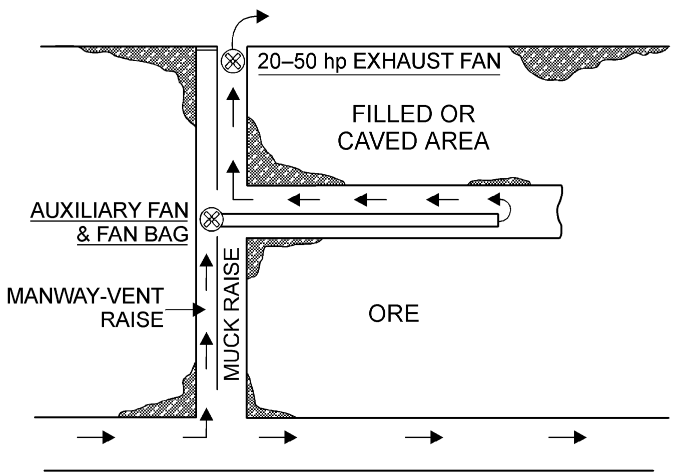

Fig. 11.7.1.2. Ventilation layout for breast stoping methods. Conversion factor: 1 hp = 0.746 kW.

into the ore body. The extraction chambers are located either on the main level or just above the main level. If the extraction area is above the main level, air should be brought up from the main level, directed across the trench or load-haul-dump (LHD) run, and then preferably up to the next level. In the case of sublevel stopes, exhaust can be through the open area. If extraction chambers are located on a main level, it is usually sufficient just to direct the air through the extraction area and back into the main drift.

BLOCK CAVING MINES. As far as ventilation is concerned, block cave mines are of two types: (1) both the ore control (grizzly) level and the haulage level are connected to the main shaft (or are separate adits); and (2) the haulage level is the only main level, and the control level is developed above the main level off raises from the main level.

In the first case, the air can either be brought across the control level into the blocks, then downcast to the haulage, or the two levels can be ventilated separately with an additional level or with special drifts serving for exhaust.

In the case where the haulage is the only main level, the air is brought across the haulage, up to the control level in each area by the mine, and is exhausted, preferably to a separate exhaust drift on the haulage level.

11.7.1.4 Control Devices

CONTROL DOORS. Control doors are installed in drifts and raises to divert air while permitting passage. These are best kept to a minimum as they are expensive to construct and maintain and tend to leak. Fire control doors are required by federal law to separate main intake shafts from possible contamination from mine fires. These must be fire-resistant and often require spray nozzles nearby. It is best to consult with Mine Safety and Health Administration (MSHA) inspectors before constructing them, as MSHA regulations governing these doors change frequently.

Where the pressure is more than about 1 in. water (250 Pa) across a door, an airlock composed of two doors in the drift is best, as opening a door against a lot of pressure is difficult, even with mechanical devices, and eventually results in damage to the door. The airlock should be long enough to accommodate a train of the length normally used at the particular mine. An airlock has the added advantage of always having one door in service if the other is damaged, and thus minimizes sometimes intolerable reversal of air currents.

Doors in small drifts can be made of double layers of ¾-in. (19-mm) plywood suspended on long steel hinges. The cracks around the door can be sealed with brattice cloth.

Wooden doors have the advantages of being cheap, easy to build, and fitting perfectly to the frame so that there is practically no leakage. But they have the disadvantages of soon wearing out and of usually being damaged beyond repair if struck by a locomotive.

Where drifts are wider, 12 ft (3.6 m) or greater, steel double doors or sometimes roll-up doors are better. Steel double doors can be fabricated from ⅛-in. (3-mm), mild steel plate with pipe or angle iron braces on one side. They are commercially available. Steel doors usually have to be opened with mechanical devices; these are often compressed air cylinders, although electric motors can also be used.

A compressed air cylinder is often mounted at the top of the door; the cylinder arm pushes the door closed and pulls it open. Another method is to mount the cylinder upright and connect the arm to a cable that goes through a pulley and then to the top of the door. The door is pulled open by the cable when the arm is retracted. When the arm is extended, the door is pulled closed by a counterweight attached via a cable and pulley to the other side of the door. This latter arrangement is sometimes better, as it is often difficult to properly maintain alignment when the door is pushed directly by the cylinder arm, and the cylinder will be badly damaged if the door is struck severely by a locomotive.

Roll-up doors can sometimes be used, but they are impractical for differential pressures greater than 1 in. water (250 Pa). Steel sectional roll-up doors can be used for fire control doors only if the anticipated differential pressure is sufficiently low. But steel roll-up doors will be completely ruined if struck by a motor (and, inevitably, they are), so are often impractical for normal air control doors.

Rubber roll-up doors are available, and these will not be damaged if struck; they will simply be pushed out of their guides and will have to be snapped back into the guides. But again, these are impractical for differential pressures greater than 1 in. water (250 Pa).

DOOR ACTUATING DEVICES. Steel-hinged doors are commonly opened by compressed air cylinders as described above or by electric motors, although electric motors open the doors rather slowly. Activation of either is best done by a remote device as the motorman approaches the door. These devices should be manually operated pull cords or pushbuttons. While such things as photoelectric cells or switches in the track will work, they are often difficult to keep maintained properly in the adverse conditions of underground mines.

Usually the electrically actuated valve that controls the air cylinders at a door is activated by a pull cord in the back of the drift, or by a pushbutton mounted on the rib. The latter arrangement is preferable as motor crews are forced to stop the train to reach the button. If the door is opened by a pull cord, there is just too much temptation to yank the cord on the fly instead of stopping. A missed cord, or a malfunctioning switch, and there is much likelihood of a wreck.

The worst disadvantage of automatic devices is that they are not "miner-proof." A motorman approaching a door activated by a remote device will expect it to open, and if there is a delay, or the device is malfunctioning, or speed is misjudged, a smashed door and possible human injury might result. Even a manually activated device should have overrides at the door itself so the door can always be opened.

Manway doors at the sides of main doors are also desirable in case the main door cannot be opened for some reason, as well as to minimize wear and tear on the door by not having to open it for foot traffic.

BRATTICES AND STOPPINGS. Brattices or stoppings to divert airflow can be built in a variety of ways. Cement block construction as used in coal mines is one technique, and is often preferable if the stoppings are permanent and the spaces to be crossed are wide, but their construction is labor intensive.

Poured concrete bulkheads can also be constructed if the mine is one that uses concrete in its operations and is therefore set up to handle concrete. Mines that do not would have to bring in the dry cement and gravel and employ mixers to mix the cement on the spot, which would be labor intensive. Of course, forms must first be constructed.

The simplest brattice is one made by stretching brattice cloth across an opening. The cloth may be fastened to the rock by studs shot into the rock, by constructing a timber set around the periphery, or by rock-bolting wooden plates to the ground around the periphery and fastening the brattice cloth to the plates. Velcro-fastened brattice cloth is available and has the advantages of permitting passage of personnel and of the velcro fasteners opening in case of a nearby blast, rather than having the brattice cloth tear. Of course, this type of brattice is leaky and easily damaged and is not recommended for long-term applications.

Wooden brattices can be quickly constructed if the openings are not too high. Such brattices can be better sealed by tacking brattice cloth over them, or, if more permanent use is desired, by covering the brattice with metal lath and plastering or shotcreting the brattice. The lath can be run back along the rock for a few inches (tens of millimeters) and the plaster or shotcrete applied over onto the rock to make a good seal. However, these seals will crack with ground movement. Plasters specially developed for mines are available; normal gypsum plaster used in house interiors will not work, as it is not waterproof. The plaster can either be troweled on or applied with guns.

Covering a wooden brattice with polyurethane foam is another method of minimizing air leakage, although the foam itself does permit some air passage. However, regulatory agencies sometimes prohibit the use of polyurethane foam.

There are some commercially available temporary brattices that are made of light cloth that balloon out with the ventilation air pressure and fill the opening. These are limited to areas where there is a sufficient differential pressure to fill the brattice.

11.7.1.5 Development Ventilation

This segment describes ventilation methods for drift headings, raises, shaft sinking, and other areas where ventilation of a dead-end face is necessary. Some manner of directing air to the face, or exhausting it from the face, must be provided.

DRIFT HEADINGS—GENERAL CONSIDERATIONS. Drift faces are often far removed from fresh air sources and require several fans working in series to provide ventilation. The fans are best placed along the length of the duct at about the points where static pressures in the dust have dropped back to atmospheric (in the case of fans blowing to the face) or where the pressures at fan discharges will be just below atmospheric (in the case of suction systems).

Ventilation of headings is greatly facilitated if two parallel drifts are being driven. In this case, crosscuts can be cut every few hundred feet (about 30 meters) between the drifts; auxiliary fans will then only be needed from the last crosscut to the headings, as air can be coursed in through one drift and out through the other. Of course, this type of ventilation is feasible only if it

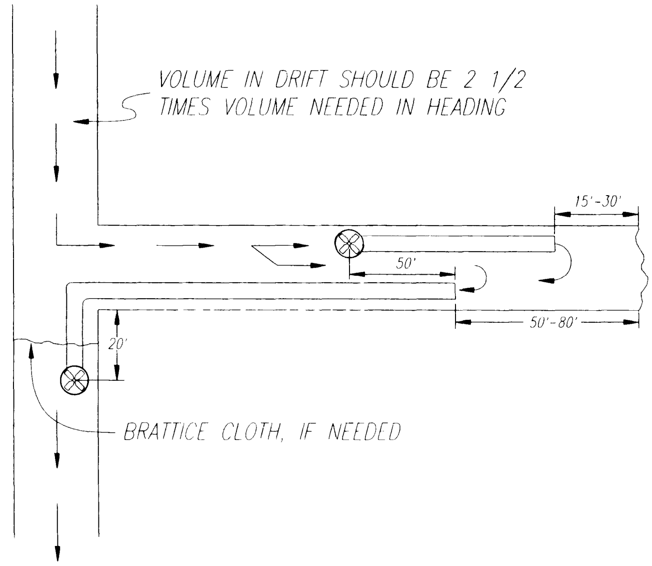

Fig.11.7.1.3. Exhaust overlap ventilation for development headings. Conversion factor: 1 ft = 0.3048 m.

is necessary to drive the two drifts anyway. It is usually not economical to drive a second drift just to facilitate development ventilation.

A similar plan can be used if two levels are parallel and are being advanced simultaneously; raises can be cut between the two levels at frequent intervals.

It is usually impossible to maintain the optimum air velocities in development that are given at the beginning of this chapter; the ventilation engineer does well to have 50 fpm (0.25 m/s) moving through most drifts. The reason, of course, is the difficulty of moving the larger quantities of air through a duct. A compromise, when blowing ventilation is used, is to have the duct discharging near the face and directing fairly high-velocity air onto the worker near the face. Ventilation in this manner is optimum when the air velocity emanating from the duct is between 600 and 1000 fpm (3 and 5 m/s).

EXHAUST vs. BLOWING VENTILATION. Air can be drawn away from the face, in which case it travels through the drift and out through an exhaust duct carried about 15 ft (4.6 m) from the advancing face, or the air can be directed to the face with a duct, in which case the air leaves the face through the drift. The former is preferable when dust and/or gases at the face are problems, the latter when heat at the face is a problem. A combination of the two methods is the exhaust overlap system (Fig. 11.7.1.3) in which prime motivation is supplied through the exhaust duct, but a portion of the air moving along the drift toward the face is drawn out with a small fan and directed to the face. This is usually done about 50 ft (15 m) from the exhaust duct inlet.

In blowing (forcing) ventilation, the advantages are those of having a positive airblast on the miners at the face, the possibility of using flexible duct with its greater versatility and lower cost, and of being able to cool the inlet air if needed.

LOCATION OF AUXILIARY FANS. When auxiliary fans are utilized to divert air from a moving airstream to a dead-end face, the fan should be located about 20 ft (6 m) from the upstream side of the turnout. The air quantity moving in the main airstream should be 2.5 times that being diverted by the auxiliary fan to minimize the possibility of exhaust air from the face being recirculated back to the fan. This is not always possible, as sufficient air is often not available. If not, a brattice curtain across the main airstream between the turnout and auxiliary fan should be used (Fig. 11.7.1.3).

VENTILATION DUCT. Duct or tubing used in mines is commonly of fiberglass, corrugated iron, or plastic-impregnated cloth; the latter is usually referred to as "fan bag" or "vent bag." Fiberglass has the advantages of being light, having low resistance to airflow, and being easily repaired if damaged; it is flexible to some extent, and will maintain its shape under light to medium impact. The fiberglass duct is usually suspended from the back with light cables wrapped around the duct, which are attached to caps or roof bolts. The small rings attached to the duct by the manufacturer are to facilitate handling and are not for suspending the duct.

Fiberglass duct is either of the bell-and-spigot type or is straight; when straight, lengths are connected with rubber connectors. This duct is commonly supplied in 10-ft (3-m) lengths. The bell-and-spigot type is generally better, as it is quicker to install and the joints are tighter. Tees, laterals, and elbows are available for all sizes; tees and laterals can be purchased with contained dampers to facilitate directing the airflow. Oval duct is available as well as round; oval is more expensive than round, but is sometimes better where clearance is a problem.

Corrugated galvanized-iron duct is cheaper than fiberglass but is heavier and more difficult to install. It can either be fastened together with the metal sleeves that are supplied with the duct or with short sewn lengths of brattice cloth that are clamped onto each length of duct with metal bands. Corrugated iron is easily knocked out of shape when being moved and is therefore not as reusable as fiberglass; it is adversely affected by wet conditions.

Plastic-impregnated cloth fan bag is available from several manufacturers. This can, of course, only be used for blowing applications although spiral wire-reinforced tubing can be used at low exhaust pressures. This duct is easy to transport as it can be contracted for handling, is very easy to install, and is relatively cheap. It can be purchased with suspension hooks fastened to the duct. The material is fairly resistant to tearing and to damage from flying rock, but cannot be compared to the solid ducts for durability. However, this is sometimes an advantage, as drops can be sewed into already installed duct when it is desired to split some air off at a certain point.

Flexible duct is more resistant to airflow than is rigid duct, and often is not reusable, especially if it has been in place more than a few months, as it becomes hard and brittle. Another problem is that flexible duct needs a certain amount of retained static pressure all the way to its discharge point to remain distended, usually about ½ in. water (125 Pa). The retained static pressure is turned into a shock loss at the end of the duct. This disadvantage can be minimized by stretching the duct taut and by using flexible duct with a ring sewn in at the ends.

Flexible duct is fastened together with various devices available from the manufacturers. Tees and laterals are also available, although some mines using large amounts of this material find it more economical to sew their own. This sewing can be done with sailcloth needles and light hemp or nylon twine. The work is best done by a person specially assigned to this work, usually an older miner, as it requires some practice and patience to do a good job.

Another type of flexible duct has wire sewn into it in a spiral, and usually comes in 10-ft and 20-ft (3-m) lengths. This accordion-type duct is rather awkward to handle, is very resistant to airflow, and should only be used occasionally where a short suction duct is needed, to turn flexible duct around a corner, or to direct air over some obstruction.

The size of duct used, whether rigid or flexible, varies with the airflow desired and the clearance available. Charts giving resistance values of various duct sizes are available from manufacturers. In general, it is best to use the largest-diameter duct possible for the headroom available in order to maximize airflow and minimize resistance; however, mines should standardize on no more than four sizes for simplification of supply. When planning an installation, be sure to determine the pressure to which the duct will be subjected and whether it will be blowing or exhausting so that the manufacturer can select the material accordingly.

QUANTITY REQUIREMENTS IN DRIFT DEVELOPMENT. Quantity requirements vary, of course, with the situation. If diesels are being used, or if radon or other gases are being emitted from the face, minimum air quantities are determined by statute. However, where such situations do not exist, the following quantities may be used as guidelines.

In relatively small headings, say 10 ft (3 m) square, air quantities should be between 3000 and 10,000 cfm (1.4 and 4.7 m^3/s). In larger headings, say 15 ft (4.6 m) square, increase the quantity to 20,000 cfm (9.4 m^3/s). Very large headings may require up to 30,000 cfm (14.2 m^3/s), but quantities greater than this are seldom necessary.

RAISES AND SMALL HEADINGS. Raises are usually small in cross section, and ventilation is simple; just use flexible ducting from an auxiliary fan and carry the duct near the face; quantity should be about 6000 cfm (2.8 m^3/s). At blast time, draw the duct back from the face to avoid damage, but keep air blowing to facilitate smoke dispersal.

Other small development headings, like block cave undercuts, are ventilated in the same manner as raises.

SHAFTS. Shaft sinking operations are best ventilated by the exhaust overlap system, as for drifts. Normally, when only one duct is to be used, suction ventilation is preferable. However, where heat is a problem and only one duct is to be used, it is a good idea to install the fan into the duct at the collar as a blowing fan, with a reversal system so as to have the duct on suction at blast time, and to have it blowing when personnel are at the bottom. Air quantity in a large shaft should be 30,000 to 40,000 cfm (14.2 to 18.9 m^3/s).

11.7.1.6 Case Studies of Ventilation Systems

The ventilation systems described in this segment are typical of those used by modern mines. They are not necessarily to be construed as "ideal" systems, but usually represent the best layout obtainable within operating and cost constraints.

BLOCK CAVING MINES: *San Manuel Mine*—This mine is developed by driving pairs of grizzly and haulage levels 60 ft (18.3 m) apart; there is a 300-ft (91.5-m) elevation between level pairs (Fig. 11.7.1.4). Intake is by a pair of man-materials shafts on the east end of the mine; these intake shafts are connected to all levels. Exhaust is via the four hoisting shafts on the west end of the mine; the exhaust shafts are only connected to the haulage levels. Motivation for airflow is by main intake fans near the intake shafts on the grizzly levels.

Air leaving the intake fan on each grizzly level is coursed along the north periphery of the ore body and enters through panel drifts driven into the ore body and along the strike. The air then is directed through the grizzly lines to 20-hp (15-kW) exhaust fans which are located in raises connecting the grizzly level to the haulage level; these raises hole into intermediate drifts that are driven between panel drifts. Additional boosting for ventilation air through individual grizzly lines is provided by compressed air movers that are used only when muck is running; inactive grizzly lines are bratticed off. A quantity of 3000 to

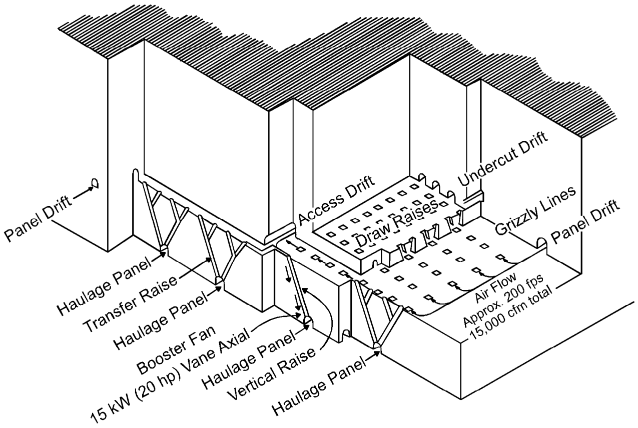

Fig. 11.7.1.4. Ventilation layout of San Manuel mine. Conversion factors: 1 fpm = 0.0051 m/s, 1 cfm = 0.472×10^{-3} m^3/s.

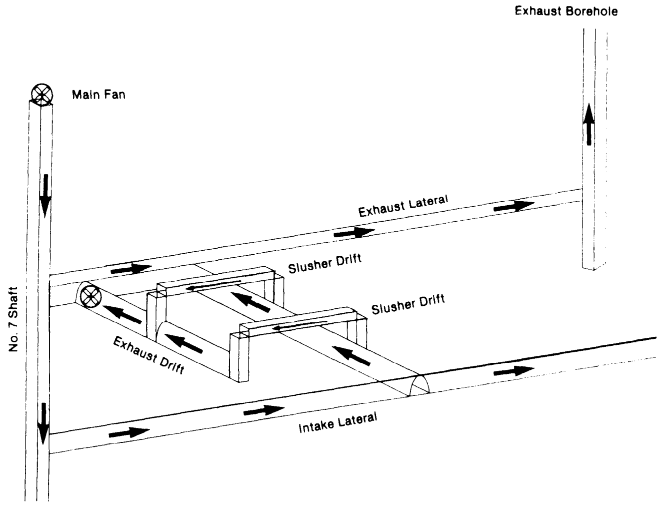

Fig. 11.7.1.6. Ventilation layout of Climax mine (after Smith, 1988).

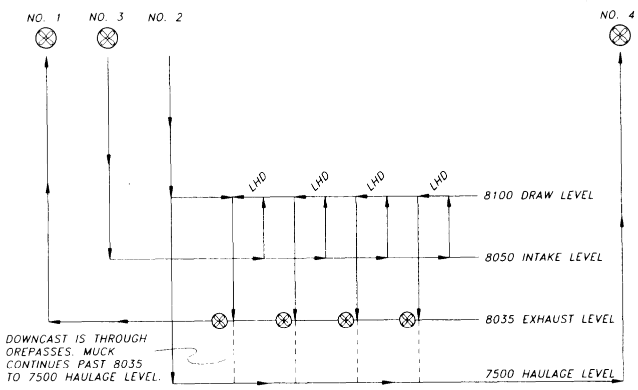

Fig. 11.7.1.5. Ventilation layout of Henderson mine (Knape, 1985).

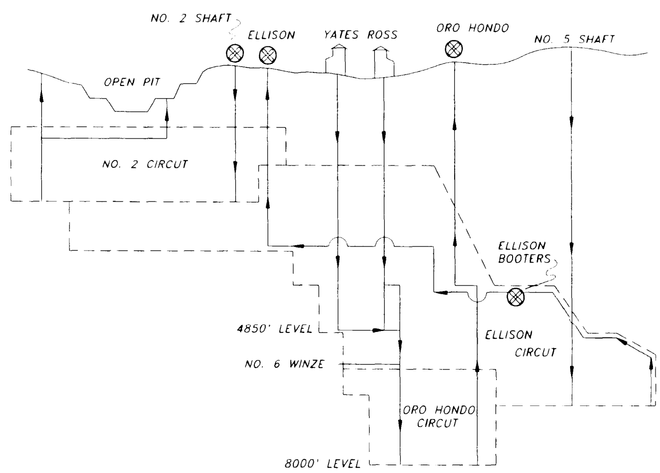

Fig. 11.7.1.7. Ventilation layout of Homestake mine (Marks, Struble and Brown, 1987, by permission: SME).

6000 cfm (1.4 to 2.8 m/s) is provided for each grizzly line being worked.

In areas where LHDs are used, grizzly lines are replaced by draw drifts that access the caving ore via chambers driven at the sides of the draw drifts. Here one exhaust raise fan to the haulage level is provided for each draw drift, and jet fans are used for moving the air rather than compressed air movers.

Henderson Mine—Air is downcast through No. 3 shaft and directed across the 8050 intake level to the production area (Fig. 11.7.1.5). Air is then upcast to individual draw drifts on the 8100 level where LHDs are in use. Fresh air travels past the LHD and downcasts the ore pass in which the LHD is dumping and is drawn off the orepass on the 8035 exhaust level by booster fans; the muck continues down the ore pass to the 7500 haulage level. Air travels across the 8035 level to exhaust at No. 1 shaft. Main intake fans are located at the collar of No. 3 shaft, and main exhaust fans at the collar of No. 1.

The 7500 haulage level is ventilated by a separate intake shaft, No. 2, and a separate exhaust shaft, No. 4. Motivation for this system is by exhaust fans at the collar of No. 4.

Climax Mine—This was the older type of block cave that did not have separate connections to the main openings for the draw-control and haulage levels (Fig. 11.7.1.6). In the latter days, production was from the 600 level, which was below surface level, and from the Storke level, which was accessed by an adit. Air was downcast through No. 7 shaft via intake fans located at the collar, then traveled through a ventilation intake lateral and into the haulage drifts.

Air upcast to each scraper drift from the haulage drift; scraper drifts were driven just above and at right angles to the haulage drifts. Air traveled to the end of the scraper drifts and downcast to exhaust drifts, one of which was driven between each pair of haulage drifts and parallel to them; motivation was provided by exhaust fans in the exhaust drifts. Air was then gathered into exhaust laterals and conducted to raises that exhausted up into the open pit. There were also some exhaust fans in the main exhaust laterals.

Underground operations at Climax ceased in 1987.

STOPE MINES: *Homestake Mine*—The mine is ventilated by three relatively independent circuits (Fig. 11.7.1.7). For the upper part, No. 2 shaft is the intake; there is an intake fan at the collar of this shaft, and exhaust is via raises that break into an open-cut mine. The mid-depth part of the mine intakes via No. 5 shaft with exhaust up Ellison Shaft; fans are underground boosters and an exhaust fan at the collar of the Ellison. The deepest part of the mine is designated the Oro Hondo Circuit; intake is via Ross Shaft, and exhaust is up Oro Hondo Shaft,

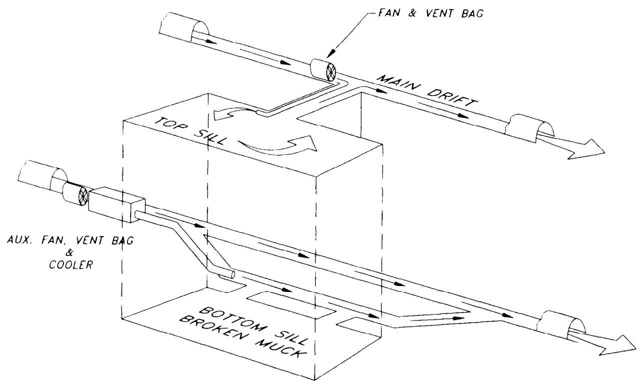

Fig. 11.7.1.8. VCR stope ventilation at Homestake mine (after Lappin-Randolph, 1987; Marks, 1988).

with main air motivation supplied by an exhaust fan at the collar of the Oro Hondo.

Most mining nowadays is with overhand cut and fill or vertical crater retreat (VCR) stoping systems. In the cut and fill system, air is directed into the stope with auxiliary fans from the drift on the bottom level of the stope; air is often blown through a cooler before entering the stope. Exhaust is through ventilation boreholes that are sunk from the upper main level of the stope along the strike of the stope in its early days of development. A quantity of 25,000 cfm (11.8 m^3/s) is provided for each cut and fill stope that uses extensive diesel equipment; smaller stopes that employ only compressed air machinery need 7000 cfm (3.3 m^3/s).

In the VCR method (see Fig. 11.7.1.8), it is necessary to provide ventilation for the top sill of the stope, from whence downholes are drilled to the bottom sill and loaded, and to the bottom sill of the stope where the muck is picked up; both these sills are driven by crosscutting from main level drifts. In the top sill, air is directed through an auxiliary fan to the face through vent bag. In the bottom sill, ventilation is flow-through since there are crosscuts in either end of the sill to the drift; air is directed through an auxiliary fan and ventbag to the muck bays that connect into the broken muck area; coolers are often used in line with the auxiliary fans. An airflow of 7000 cfm (3.3 m^3/s) is allowed for top sills, and 9000 to 15,000 cfm (9.9 to 7.1 m^3/s) for bottom sills.

Sunshine Mine—Air intakes the Jewell shaft and is divided into east and west ventilation districts. Initial motivation power is provided by underground boosters near the Jewell (Fig. 11.7.1.9). Air is then coursed through a series of winzes and raises to be distributed to the workings. Main exhaust is up the Bighole shaft, which is furnished with surface exhaust fans. Some supplementary exhaust is through the Sunshine tunnel and Silver Summit shaft. Details on stope ventilation were not available.

Mount Isa Mine—The overall ventilation layout of this mine is complex and beyond the scope of this article. However, stope ventilation can be described.

Footwall and hanging wall drifts are driven on each sublevel around the periphery of the ore body; these drifts either connect directly to shafts or to inclines between main sublevels. Stopes do not extend along the full width of the ore body, so several crosscuts connect the drifts, and air is supplied off these crosscuts to individual stopes (Fig. 11.7.1.10).

The stoping method used is sublevel stoping with ring drilling. Initial development is to bore a cutoff raise in the middle of the stope and slash this out to the stope width. Sublevels are driven along the stope length to intersect the cutoff raise. The raise connects to an exhaust horizon above the top of the stope, and to a fill conveyor level above the exhaust horizon. Air is supplied to the sublevels either directly via footwall pressure shafts, or via access ramps and fresh air raises. Air courses through sublevels and exhausts up the stope cutoff raise through an exhaust fan on the exhaust horizon. In stope development, auxiliary fans supply the faces of the sublevels before they connect to the cutoff raise; sometimes these fans continue to be used once hole-through to the cutoff raise is effected. About 15,000 cfm (7 m^3/s) of air per sublevel is needed.

Magma Mine—Ventilation is by downcasting No. 9 shaft, the main men-materials hoisting shaft, and upcasting No. 6 shaft, which is located west of the active workings (Fig. 11.7.1.11). Supplemental exhaust is supplied through a series of raises that connect to No. 4 shaft in the old western portion of the mine. Three of the shafts in the western portion supply intake ventilation for the older, inactive part of the mine. Motive power is provided by intake fans near No. 9 shaft that blow through cooling coils, and by main exhaust fans at the collars of both exhaust shafts. Some large underground exhaust booster fans are also used.

The ventilation pattern is to intake alternate levels from No. 9, upcast and downcast the air to the levels between the intake levels through mining raises, and then direct the air to the exhaust shafts. This is the ideal situation, although not always possible.

The mining method employed is undercut and fill stoping, principally in a replacement deposit that dips at a 30° angle. A raise was driven between levels, and two stopes started off each raise. One stope was just below the top sill, the other about halfway between levels. Stoping proceeds downward, with cuts on each sublevel being filled before the cut on the sublevel below is begun. Extraction of the ore on each sublevel is by an initial sublevel crosscut being driven along with the footwall, then by parallel cuts (termed panels) being driven across the dip of the ore from the crosscut to the hanging wall. About three panels are driven at a time; the panels are then sand-filled, and new panels driven. This sequence continues until the sublevel is mined out. Ventilation is supplied by drawing air off the raise and directing it to the faces of the various panels with auxiliary fans, then exhausting up another raise to the level above. The exhaust raise is often furnished with an exhaust fan at the exhaust level. A quantity of 6000 cfm (2.8 m^3/s) per active panel in each stope is allowed.

11.7.1.7 Design Calculations

Example 11.7.1.1. A gold mine is to be developed to an ultimate depth of 1500 ft (457 m). Levels are to be 100 ft (30 m) apart, starting with the uppermost level at a depth of 1000 ft (305 m). Two stopes per level pair are normally in production, with a total of five to six stopes at any one time.

The main hoisting and intake air shafts are circular, with a diameter of 15 ft (4.6 m). Exhaust is by an 8-ft (2.4-m) diameter bored shaft on the other end of the ore body from the intake shaft. Drifts are to be 10 by 10 ft (3 by 3 m) in cross section and timbered. Stoping is by the shrinkage method, with a raise at either end of each stope; ventilation of individual stopes is by upcasting one raise and exhausting up the raise at the other end. Raises have 3 by 3 ft (1 by 1 m) available for air passage. Stopes are 100 ft (30 m) long between raises, and have the same cross-sectional area as the drifts. Layout of the mine at its ultimate extent is as shown in Fig. 11.7.1.12.

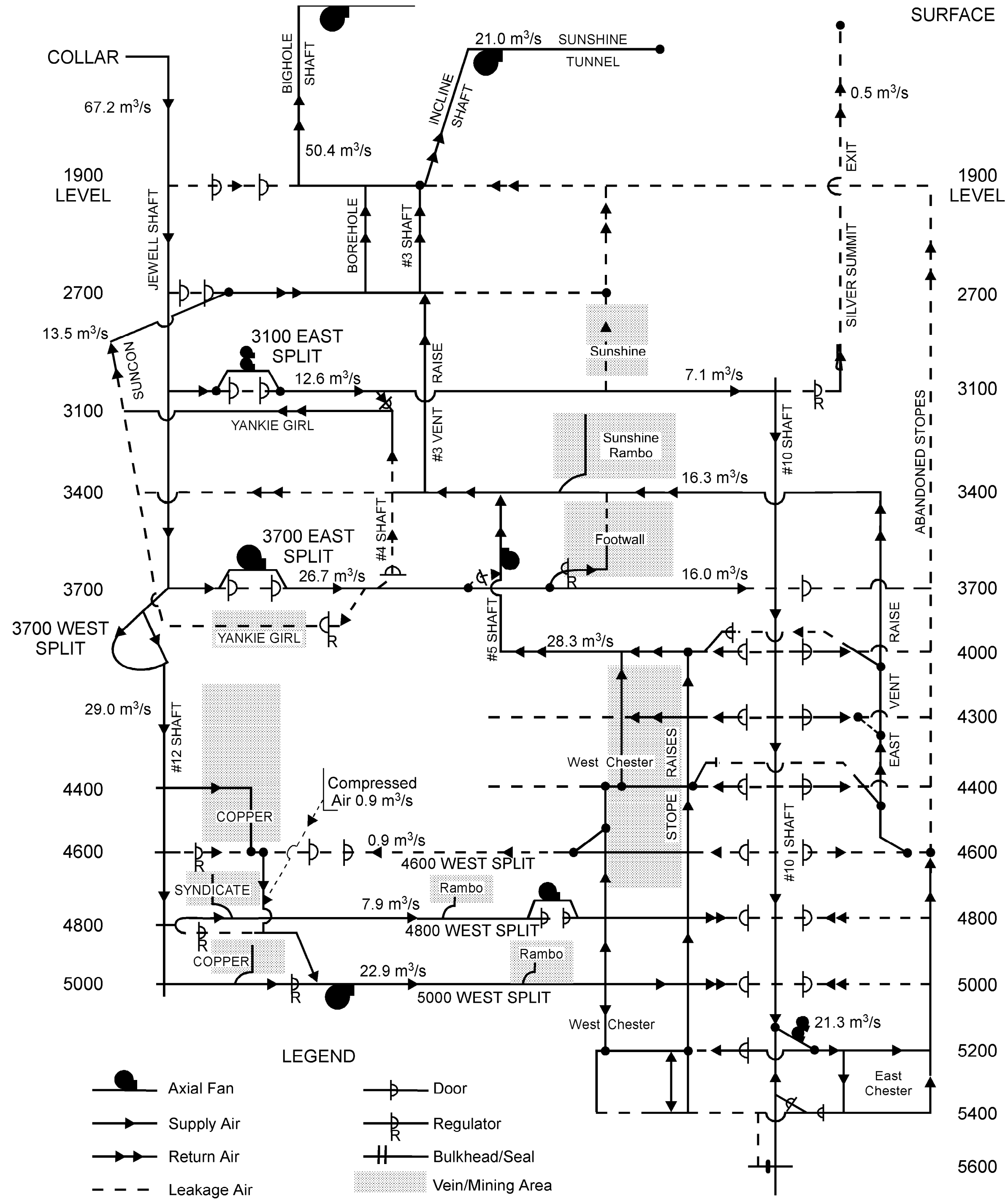

Fig. 11.7.1.9. Ventilation layout of the Sunshine mine (Jurani, 1977, by permission: SME). Conversion factor: 1 cfm = 0.472×10^{-3} m³/s.

Following the procedure given in 11.7.2, a ventilation system is designed as follows:

1. Select main ventilation inlets and outlets. This has already been done.

2. From mining plans, determine the airflow requirements, and design tentative ventilation schematics. An intake on alternate levels, with upcast to the level above, is to be used. Airflows are shown on the schematic, with no reuse of air. A velocity of 100 fpm (0.5 m/s) is used for airflow in the stopes and a minimum airflow of 200 fpm (1.0 m/s) on the levels.

3. Specify main and booster fan locations. A main exhaust fan at the top of the exhaust shaft is specified. Initially, booster fans are not being considered.

4. Determine airflow resistances for all branches, and build a computer model of the ventilation system. The resistances R in

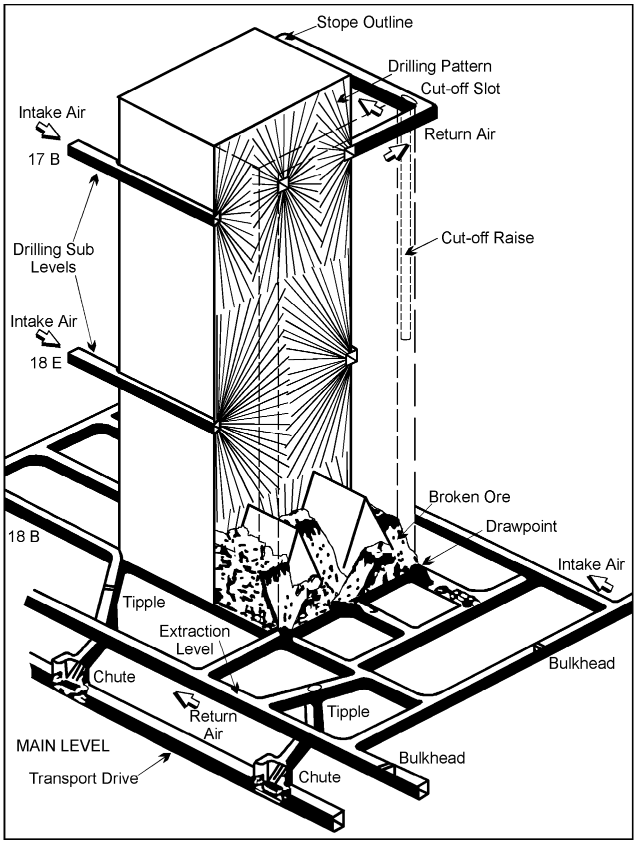

Fig. 11.7.1.10. Stope ventilation at Mt. Isa mine (Allen, 1976. By permission of the Mine Ventilation Society of South Africa, Johannesburg).

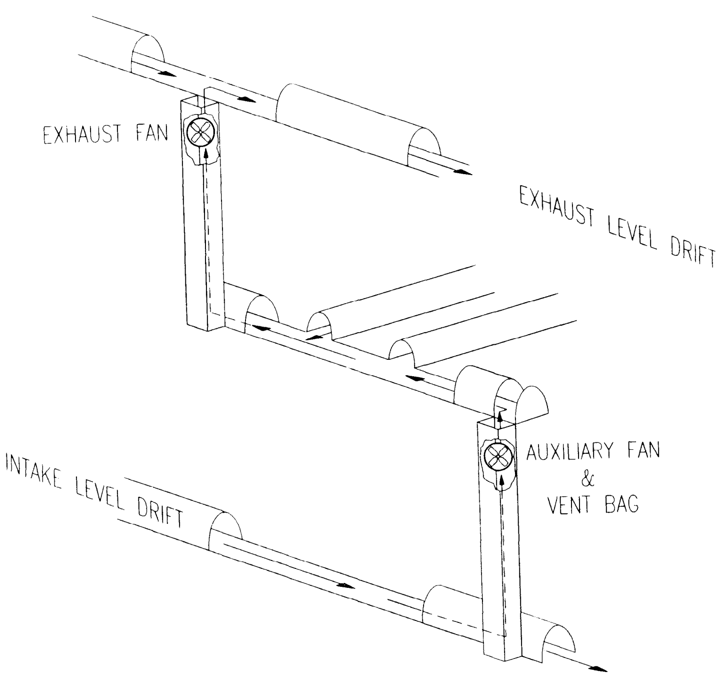

Fig. 11.7.1.11. Stope ventilation at Magma mine.

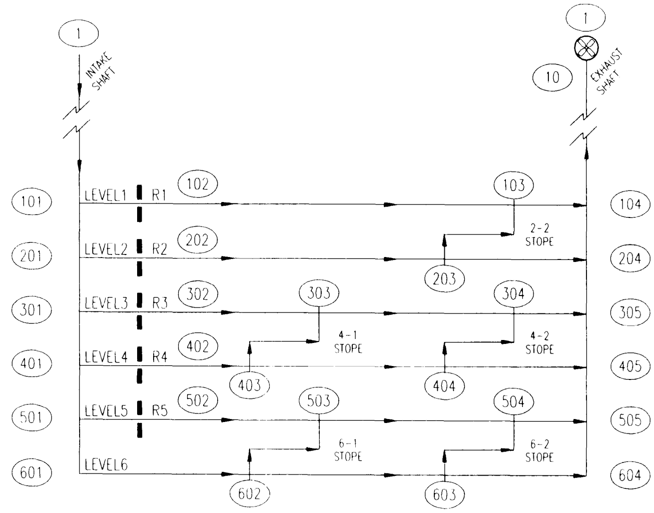

Fig. 11.7.1.12. Schematic of mine, Ex. 11.7.1.1.

in.-min^2/ft^6 (N-s^2/m^8) are calculated from the familiar Atkinson equation (see Eq. 11.6.15):

$$R = \frac{KLP}{5.2A^3} \tag{11.7.1.1}$$

in which K is friction factor in lb-min^2/ft^4 (kg/m^3), P is the airway perimeter in ft (m), L is airway length in ft (m), and A is airway area in ft^2 (m^2).

Resistances are calculated in terms of resistance/l00 ft (30 m), except for stopes; the stopes all have practically the same dimensions, so the same resistance for each suffices.

Equations employed in these calculations were derived and demonstrated in Chapter 11.6.

Solution.

A. Intake Shaft. Resistance calculation here is somewhat difficult, as the moving conveyances have some effect on the resistance, and the shaft fixtures must be taken into account. However, the procedure is simplified in this case by calculating the resistance as if the shaft were unobstructed and then increasing the resistance by 25% to allow for the shaft fixtures and conveyances. Use a K factor of 30×10^{-10} lb-min^2/ft^4 (5.6×10^{-3} kg/m^3) from Table 11.6.3 (see Chapter 11.6):

$$R = \frac{(30 \times 10^{-10})(15\pi)(100)}{5.2\left(\frac{15^2}{4} \times \pi\right)^3} \times 1.25$$

$$= 0.062 \times 10^{-10} \text{ in.-min}^2/\text{ft}^6/100 \text{ ft } (0.0069\ N\text{-}s^2/\text{m}^8)$$

B. Calculate the R factor of the timbered drifts. Use a K factor of 100×10^{-10} lb-min^2/ft^4 (18.6×10^{-3} kg/m^3):

$$R = \frac{(100 \times 10^{-10})(4 \times 10)(100)}{5.2(10 \times 10)^3}$$

$$= 0.0769 \times 10^{-10} \text{ in.-min}^2/\text{ft}^6/100 \text{ ft } (0.0086\ N\text{-}s^2/\text{m}^8)$$

C. Calculate the R factor of the exhaust shaft. Use a K factor of 15×10^{-10} lb-min^2/ft^4 (2.8×10^{-3} kg/m^3):

Table 11.7.1.1. Calculation of Head Loss for the Most Resistant Circuit of Ex. 11.7.1.1

Node 1	Node 2	Length, ft	$R \times 10^{-10}$/100 ft	Air Quant, kcfm	Hd. loss in. water
1	101	1000	0.0620	170	0.179
101	201	100	″	150	0.140
201	301	100	″	120	0.089
301	401	100	″	100	0.062
401	501	100	″	60	0.022
501	601	100	″	40	0.010
601	503	—	34.9749 (total)	1	0.350
503	504	300	0.0769	30	0.021
504	505	300	″	40	0.037
505	405	100	0.0571	60	0.021
405	305	100	″	80	0.036
305	204	100	″	120	0.082
204	104	100	″	140	0.112
104	10	1000	″	170	1.650
TOTAL					2.81

Conversion factors: 1 ft = 0.3048 m, 1 kcfm = 0.472 m^3/s, 1 in. water = 250 Pa, 1 in.-min^2/ft^6 = 11.17×10^{10} N-s^2/m^8.

$$R = \frac{(15 \times 10^{-10})(8\pi)(100)}{5.2\left(\frac{8^2}{4}\pi\right)^3}$$

$$= 0.0571 \times 10^{-10} \text{ in.-min}^2/\text{ft}^6/100 \text{ ft } (0.0064\ N\text{-}s^2/\text{m}^8)$$

D. Calculate the R factor of each slope. Use $K = 110 \times 10^{-10}$ lb-min^2/ft^4 (20.4×10^{-3} kg/m^3) for the raises and $K = 200 \times 10^{-10}$ lb-min^2/ft^4 (37.1×10^{-3} kg/ms^3) for the sublevel.

(1) Raises:

$$R = \frac{(110 \times 10^{-10})(4 \times 3)(100)}{5.2(3 \times 3)^3}$$

$$= 34.8211 \times 10^{-10} \text{ in.-min}^2/\text{ft}^6\ (3.890\ N\text{-}s^2/\text{m}^8)$$

(2) Sublevel:

$$R = \frac{(200 \times 10^{-10})(4 \times 10)(100)}{5.2\ (10 \times 10)^3}$$

$$= 0.1538 \times 10^{-10} \text{ in.-min}^2/\text{ft}^6\ (0.0172\ N\text{-}s^2/\text{m}^8)$$

(3) Total: $R = 34.9749 \times 10^{-10}$ in.-min^2/ft^6 (3.907 N-s^2/m^8)

E. Calculate the head loss of the most resistant circuit of the mine, using Eq. 11.6.16. This, from inspection, will probably be the airflow path of nodes 1-601-602-503-505-10. The calculation is shown in Table 11.7.1.1. Quantities are expressed in 1000 cfm (kcfm). One of several mine ventilation computer programs can be used for head-loss calculations and to balance the circuits.

Use, say, 3.0 in. water (750 Pa), as the total static head. Then a fan capable of supplying 170,000 cfm at 3.0 in. water (80.2 m^3/s at 750 Pa) will be needed. The fan selected will tentatively be one whose curves are shown in Fig. 11.7.1.13. Note that these are static pressure curves; were they total pressure curves, the fan's evase velocity pressure would have to be added to the static pressure to read the curves correctly. The estimated quantity and static pressure fall on the midpoint of the #2 setting of the fan, so the #2 setting will be that whose points will be input to the computer simulation.

An inspection of Fig. 11.7.1.12 leads to the conclusion that resistors (regulators) may be required on all levels except level 6; therefore, for the initial simulation run, resistors will be specified at the intake shaft, with arbitrary resistances of 0.5 in.-min^2/ft^6 (0.056 N-s^2/m^8) per resistor. The computer will calculate any increase or decrease in resistance needed.

Input quantities and output quantities are shown in Table 11.7.1.2, along with additional resistances calculated for the fixed branches. Resistances are calculated in the same manner as those shown in Table 11.7.1.1 (not shown). The fan is computed to be moving 178,900 cfm at 2.4 in. water (84.4 m^3/s at 600 Pa).

The 1-10 branch is necessary to close the mesh around the fan and is the fan brattice leakage. A very high resistance is input, as leakage should be minimal with a good installation. The resistance used in this run is 1000×10^{-10} in.-min^2/ft^6 (111.7 N-s^2/m^8).

The first computer run shows that expected outputs on the variable branches are about the same as input quantities, except for the stopes, where the quantities are too low. The simplest way to solve this problem in a metal mine is with the use of booster fans, at the exhaust raise of each stope, that will overcome the entire stope resistance. This is the solution that is used here.

For the fixed-quantity branches, an examination of the computer printout reveals that the only place where resistance really is needed is on level 4.

A second computer run is now made, with booster fans installed at each stope. Small fans like those needed at the stopes can be entered into a computer model, but the solution with them entered into the model is suspect due to the difficulty of realistically calculating the resistance of each stope. A more practical method is simply to eliminate the stope branches from the model and assume that the booster fans can take care of the stopes, with perhaps some field adjustments. This particular computer program does not require airflow branches, so airflow quantities can be added and subtracted at nodes without specifying the paths of those quantities.

The fixed branches are changed to variable branches, except for the required 401–402 resistor. These former resistors are assigned resistances of 0, as the branches no longer really exist; however, they remain in the model in case they are needed later, and to save the trouble of renumbering some nodes. Input quantities and output quantities for the second computer model

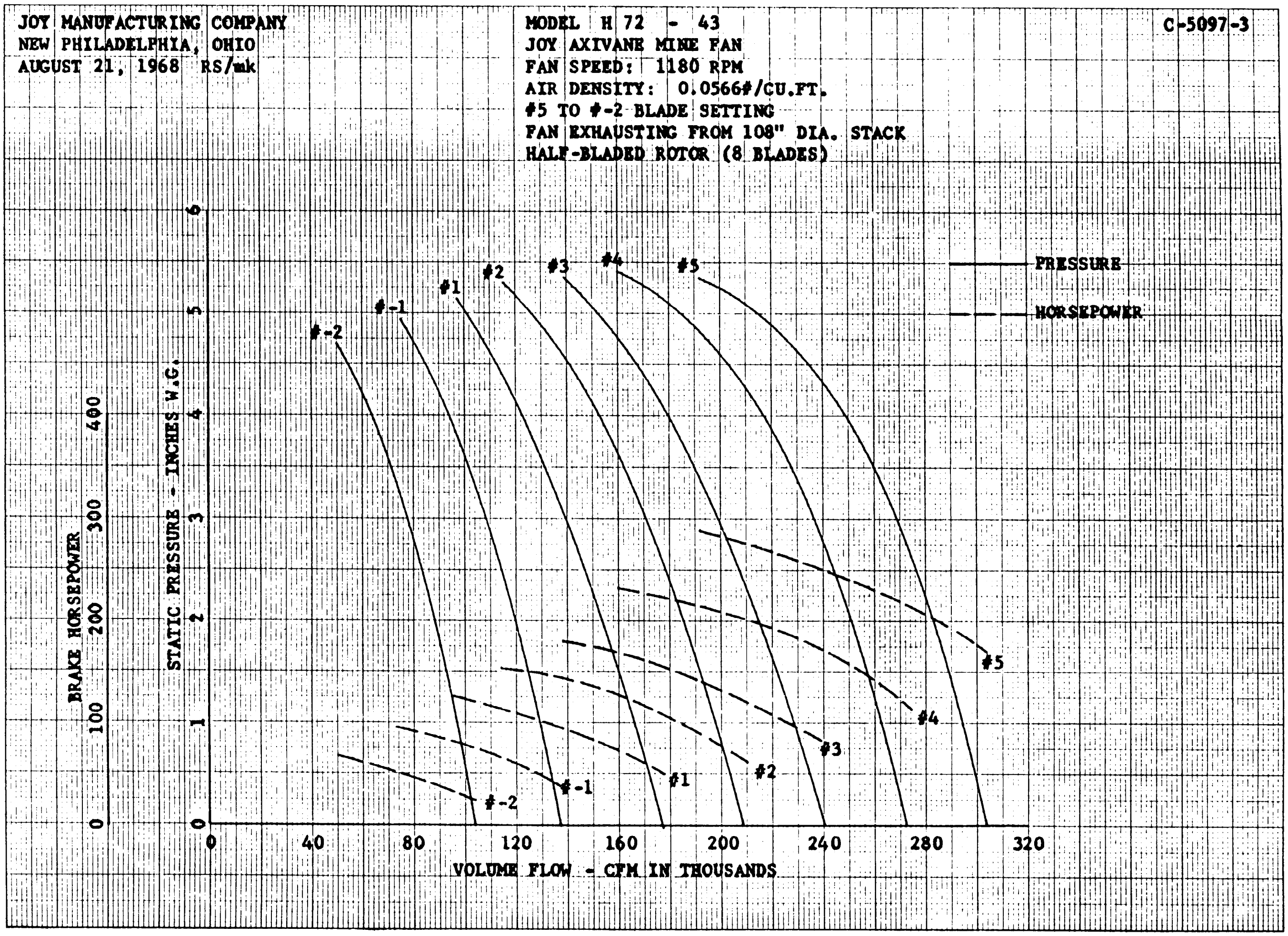

Fig. 11.7.1.13. Curve of fan for mine of Ex. 11.7.1.1 (by permission: Joy Mfg. Co., New Philadelphia, OH).

are shown in Table 11.7.1.3. The fan is computed to be moving 178,900 cfm at 2.4 in. water (84.4 m^3/s at 600 Pa).

The second computer model shows the airflows to be generally as desired, so no more modeling is needed for this example. Of course, in an actual problem, a good many more runs would be made with varying changes in order to obtain the optimum solution.

However, one point to be noted is that the fan develops 178,900 cfm at 2.4 in. water (84.4 m^3/s at 600 Pa). From this, the air horsepower and efficiency can be calculated (see Eqs. 11.6.17 and 11.6.31).

$$\text{Air power } AHP = \frac{H_L Q}{6350} = \frac{(178{,}900)(2.4)}{6350}$$
$$= 67.6 \text{ hp } (50.4 \text{ kW})$$

From the horsepower curve of the fan at a #2 setting and a quantity of 178,900 cfm (84.4 m^3/s), the fan develops 105 bhp (78.2 kW). The fan efficiency η therefore is

$$\eta = \frac{P_{as}}{P_f} = \frac{67.6}{105} \times 100 = 64\%$$

One notes that 64% is not a good efficiency; fans in their normal operating range should develop 70 to 75% efficiency. Therefore, further search should be made for a more suitable fan, even if the output quantities are considered adequate.

Some variation in method of solution may also be dictated according to the computer algorithm used. The algorithm used in this solution requires desired air quantities to be input, but does not require a balance in air quantities at any node; therefore, the stopes could simply be eliminated from the network as was done for simplification of the problem after the first run. Other algorithms would not permit this, and other solutions would have to be sought in the modeling that included the stopes.

11.7.1.8 Ventilation Department Organization

Personnel Requirements. At mines where there are many fans and much ventilation construction work is done, there should be at least one full-time engineer employed in ventilation. At mines where a full-time ventilation engineer cannot be justified, ventilation responsibilities should be assigned to one staff engineer— the old saw that ventilation is “everybody’s job and nobody’s responsibility” just cannot apply in this day and age.

Professional Development. When a person is assigned to ventilation, that person should at first train under someone who is familiar with ventilation. If nobody familiar with ventilation is available, the person assigned should be sent to one of the short courses in ventilation taught from time to time at the

Table 11.7.1.2. Input and Output Air Quantities for the Initial Computer Simulation of Ex. 11.7.1.1

Node 1	Node 2	Input kcfm	Output kcfm	Additional Resistance
1	101	170	167.9	
101	201	150	147.9	
201	301	120	117.9	
301	401	100	97.9	
401	501	60	57.9	
501	601	40	37.9	
101*	102	20	20.0	−7.25
102	103	20	20.0	
103	104	30	26.6	
201*	202	30	30.0	−1.11
202	203	30	30.0	
203	204	20	20.0	
301*	302	20	20.0	−1.75
302	303	20	20.0	
303	304	30	25.5	
304	305	40	31.2	
401*	402	40	40.0	+0.88
402	403	40	40.0	
403	404	30	34.4	
404	405	20	28.8	
501*	502	20	20.0	0.00
502	503	20	20.0	
503	504	30	25.0	
504	505	40	30.1	
601	602	40	37.9	
602	603	30	32.9	
603	604	20	27.8	
203	103	10	6.6	
403	303	10	5.5	
404	304	10	5.5	
602	503	10	5.0	
603	504	10	5.0	
604	505	20	27.8	
505	405	60	57.9	
405	305	80	86.7	
305	204	120	117.9	
204	104	140	141.3	
104	10	170	167.9	
1	10	0.1	1.1	
10	2	(Fan)171	178.9	

* = fixed quantity branch. Conversion factor: 1 kcfm = 0.472 m^3/s.

Table 11.7.1.3. Input and Output Air Quantities for the Second Computer Simulation of Ex. 11.7.1.1

Node 1	Node 2	Input kcfm	Output kcfm	Additional Resistance
1	101	170	167.9	
101	201	150	125.2	
201	301	120	90.3	
301	401	100	77.4	
401	501	60	37.4	
501	601	40	28.0	
101	102	20	42.7	
102	103	20	42.7	
103	104	30	52.7	
201	202	30	34.9	
202	203	30	34.9	
203	204	20	24.9	
301	302	20	12.9	
302	303	20	12.9	
303	304	30	22.9	
304	305	40	32.9	
401*	402	40	40.0	+1.06
402	403	40	40.0	
403	404	30	30.0	
404	405	20	20.0	
501	502	20	9.4	
502	503	20	9.3	
503	504	30	19.3	
504	505	40	29.4	
601	602	40	28.0	
602	603	30	18.0	
603	604	20	8.0	
604	505	20	8.0	
505	405	60	37.4	
405	305	80	57.4	
305	204	120	90.3	
204	104	140	115.2	
104	10	170	167.9	
1	10	0.1	1.1	
10	2	(Fan) 171	80.7	

* = fixed quantity branch. Conversion factor: 1 kcfm = 0.472 m^2/s.

various mining engineering institutions. Relying on knowledge acquired as an undergraduate in a university mine ventilation course is often insufficient, unless the course was just recently completed, as this type of subject matter is easily forgotten unless used.

Ventilation personnel should understand both the theoretical and practical sides of their field. At least one of the ventilation textbooks listed in the references of this chapter should be acquired. A subscription to the *Journal of the Mine Ventilation Society of South Africa* is recommended as this it the only English language journal devoted exclusively to mine ventilation.

The SME Underground Ventilation Committee holds biannual symposia (in odd-numbered years) and attendance is encouraged for all engaged in mine ventilation. This Committee sponsors technical sessions at the annual SME meeting as well. Proceedings of past symposia and preprints of papers from the annual meetings are available from SME.

There are also International Mine Ventilation Congresses held every four years (in even-numbered years, divisible by four) in different parts of the world. Information on these congresses is always provided in the various mining journals.

VENTILATION INSTRUMENTS. Every mine ventilation department should have at least one ball-bearing vane anemometer equipped with extension rods for velocity measurements. These are better for average flow measurements than thermal or vortex-shedding anemometers as they permit integration of air velocities over the cross section of an opening. The other instruments are accurate but are only good for instantaneous readings and spot checks.

A mine ventilation department should also have at least one aneroid barometer accurate to within 0.005 in. Hg (1.7 Pa) (two barometers are needed for traverses), chemical smoke tubes and aspirator bulbs, sling psychrometers (or equivalent electronic instruments), and gages for reading differential water gage pressures—dial-type indicators are handy here, although for reading very small differences such as for making trailing-tube pressure traverses, inclined oil-filled manometers are better.

Gas detection instruments are necessary in many mines, especially those using diesels. Portable electronic instruments for measuring many gases are available, and stain tube indicators are available for virtually all gases found in mining. Dust measuring instruments are also necessary to maintain air quality.

VENTILATION SURVEYS. Mine-wide surveys should be made as often as necessary to determine the overall air distribution

Table 11.7.1.4. Sample Computer-Generated Report on Weekly Face Ventilation Conditions at a Western Metal Mine

Level	Panel	Line	Air In °F	Air Out °F	Air Vel fpm	Eff. Temp °F	Air Quant cfm	Remarks
2315	24	3	68/78	72/81	150	70	4200	
2315	24	8	70/75	79/82	200	73	5600	
2615	3	7	81/85	85/92	200	83	2200	
2615	11	13	79/83	80/86	270	76	14000	
2615	13	18	73/83	79/85	260	75	7800	

Conversion factors: °C = 5/9 (°F − 32), 1 fpm = 0.005 m/s, 1 cfm = 0.472×10^{-3} m³/s.

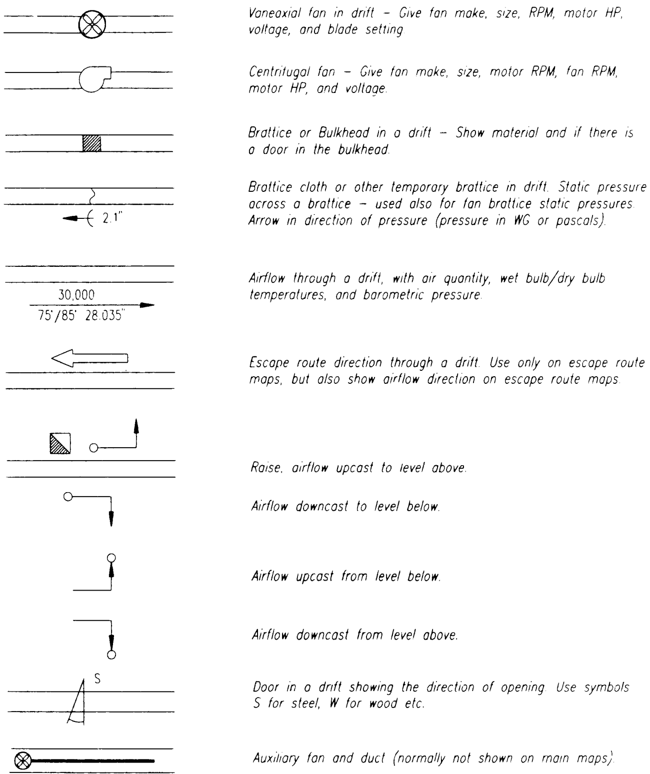

Fig. 11.7.1.14. Suggested ventilation map symbols.

pattern and to evaluate the performance of fans. The minimum frequency for doing this is annually, and some mines do it as often as monthly, although this is seldom justifiable unless ventilation changes radically from month to month. Once every three months is about right for most mines.

Results of overall surveys should be entered on maps and/or ventilation schematics, and calculated air quantities should be stored on computer printouts. There is really no standardized method of mapping ventilation beyond the obvious one of using arrows to indicate airflow direction. Some mines use solid arrows to indicate fresh air and dashed arrows to indicate exhaust air; some prefer to do this with colored arrows. Some places, where heat is a problem, use different colored arrows according to the temperature. Airflows, preferably corrected to standardized density of 0.075 lb/ft³ (1.20 kg/m³), should be logged on the maps or schematics along with wet- and dry-bulb temperatures and fan pressures. See Fig. 11.7.1.14 for a set of suggested ventilation mapping symbols.

Avoid putting too much detail on overall ventilation maps or schematics. If details of flow in a certain area must be shown, use a blown-up detail of the area.

In addition to overall surveys, checks should be frequently made of ventilation conditions at the working faces. This is really the most important type of measurement, as the "business end" of ventilation is where the rock is being broken. These measurements will vary according to the mining method and conditions. For instance, in diesel headings, measurements should be made to assure that adequate airflows are supplied and noxious gas levels are within legal limits. In mines with heat problems, temperature-velocity measurements are needed. Where dust is a problem, dust levels and air velocities are important.

Working face measurements should be made at least monthly; often semi-weekly or even weekly observations are better, especially if potentially hazardous conditions requiring constant surveillance exist. Results of these surveys should be logged on reports and distributed—the computer is a great aid in doing this type of work. Table 11.7.1.4 illustrates a computerized weekly report used at a western metal mine.

Airflow resistance factors of the mine for use in computer simulations should be kept either on schematics, maps, or in computer files. These factors should be updated from time to time with pressure surveys as new areas open up and old ones close down. For determination of mine friction and resistance factors, see Chapter 11.6.

CONSTRUCTION CREWS. Ventilation departments that are responsible for getting much construction work done are advised to have a few timbermen under their direct control. Larger operations may need a full crew with a full-time supervisor. General repair crews with other work to do often tend to leave ventilation work to the last.

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11.7.2 COAL MINE VENTILATION

RAJA V. RAMANI

Coal mine ventilation systems must be designed so as to maintain at all times healthy and safe atmospheric working conditions. The main purposes of a coal mine ventilation system are to provide adequate quantities of fresh air to the miners in the workings and to render harmless toxic, noxious, and explosive gases and dusts and carry them out of the mine through dilution by fresh air.

For coverage of ventilation principles, see Chapter 11.6.

11.7.2.1 General Considerations

Correct estimation of the required air quantity at each of the workings is vital. This quantity should be determined on the basis of number of workers in the workings, kinds of machinery used, and the make up of gases, dust, heat, and humidity. The required quantities of fresh air at the faces and other places must be increased in proportion to the amount of leakage anticipated in the airways. Leakage estimation may be problematical, but an allowance of 100% of the estimated quantities for faces is common. Quantity estimation is critical as head losses and power are proportional to the square and cube of quantity, respectively. Small errors in quantity estimates can therefore lead to large errors in pressure head and power estimates and affect the total mine ventilation, health, safety, and production performance.

The number of airways required to course the various quantities through the mine is important as this number influences the velocity of the air in the airway. Air velocities are critical from two points of view. From the safety viewpoint, inadequate air velocities can cause undesirable accumulations and layering of gases; very high velocities, on the other hand, can raise dust clouds. From the economic viewpoint, the head loss in a mine airway is proportional to the square of the velocity, and high velocities will entail large horsepower dissipation. Thus velocities must be carefully chosen so that the safety and economic factors are not compromised. Since velocity is a function of the area of an airway and of the number of airways in parallel, definition of the velocity and quantity requirements will automatically determine the number of airways once the shape and size of an airway are determined by mining machine and ground control conditions.

The head loss in individual splits can be calculated once the resistances of the airways and the volumes of air circulating in them are known. The head loss in the individual splits can be accumulated using series or parallel circuit laws as applicable until the total head loss in the system is calculated. Knowing the quantity and head required for a mine, selection of a suitable mine fan can be made. This is not to imply that mine ventilation system planning and designing is a simple straightforward process but to emphasize that it is an ordered process with many interactions with other aspects of mine planning and designing (Stefanko, 1983; Suboleski and Kalasky, 1982; Luxbacher and Ramani, 1980). Factors such as ground control, production requirements, and equipment limitations may often have great bearing on the mine design. In fact, mine ventilation planning is quite complicated when data on strata gases, mining methods, and mine openings are not readily available or estimates of these have large variances. Therefore, the final ventilation design selected must be flexible in order to adapt to changing conditions.

11.7.2.2 Coal Mine Ventilation Design

The flow diagram in Fig. 11.7.2.1a illustrates the planning and design of the ventilation system as it relates to the overall mine planning problem. At the exploration stage, data must be collected on those specific geological factors that may affect the ventilation system. Mine design also includes many other geological data and, as indicated, is affected by rules and regulations on health and safety. The end result is the development of a mining method and a mine infrastructure. Mine ventilation considerations play an important role at this initial design stage. However, a more detailed ventilation analysis is required as more definite plans are made to mine the coal seam (Fig. 11.7.2. lb). This analysis must be very specific with respect to quantity and quality control of the mine air, and should include a sensitivity analysis covering factors such as methane and leakage that cannot be accurately estimated. Once a suitable plan has been selected, it must be examined with relation to such factors as performance in case of emergencies (e.g., fires and explosions) and adaptability to changing mining plans (e.g., from room and pillar to longwall).

On the basis of such an analysis, the plan may be accepted as presented or modifications may be required in the ventilation plan or even in mine design. Early liaison between mine planning and ventilation staffs can prevent many problems and shorten the length of planning time by revealing the need for additional investigations or new calculations on engineering revisions. In effect, mine development schemes should incorporate full details of ventilation arrangements and the means of achieving effective ventilation at each stage of the mine life. The design of the ventilation system, therefore, should be considered in relation to the long-, medium-, and short-range plans of the mining plan (Anon., 1971). While safety is the prime consideration, to maintain or to increase production and productivity levels, good ventilation planning is essential.

LONG-RANGE PLAN. A long-range plan is defined 10 to 20 years in advance. The general layout of airways underground should be decided by ventilation considerations as well as by other considerations. The sizes of the underground entries must be sufficient for their varying duties as airways at different stages of development. Since production plans may change with changing geologic and/or market conditions, the ventilation plan must be based on a split system that will give some flexibility to blocking off old or opening new sections as the need arises. The evaluation of specific details related to ventilation should be made as early as possible. Though there may be uncertainty with regard to geologic and market data, decisions regarding development of access to the seams (e.g., slopes, shafts, adits, etc.) and method of mining, for instance, cannot be deferred indefinitely. Such decisions as those pertaining to main and sub-

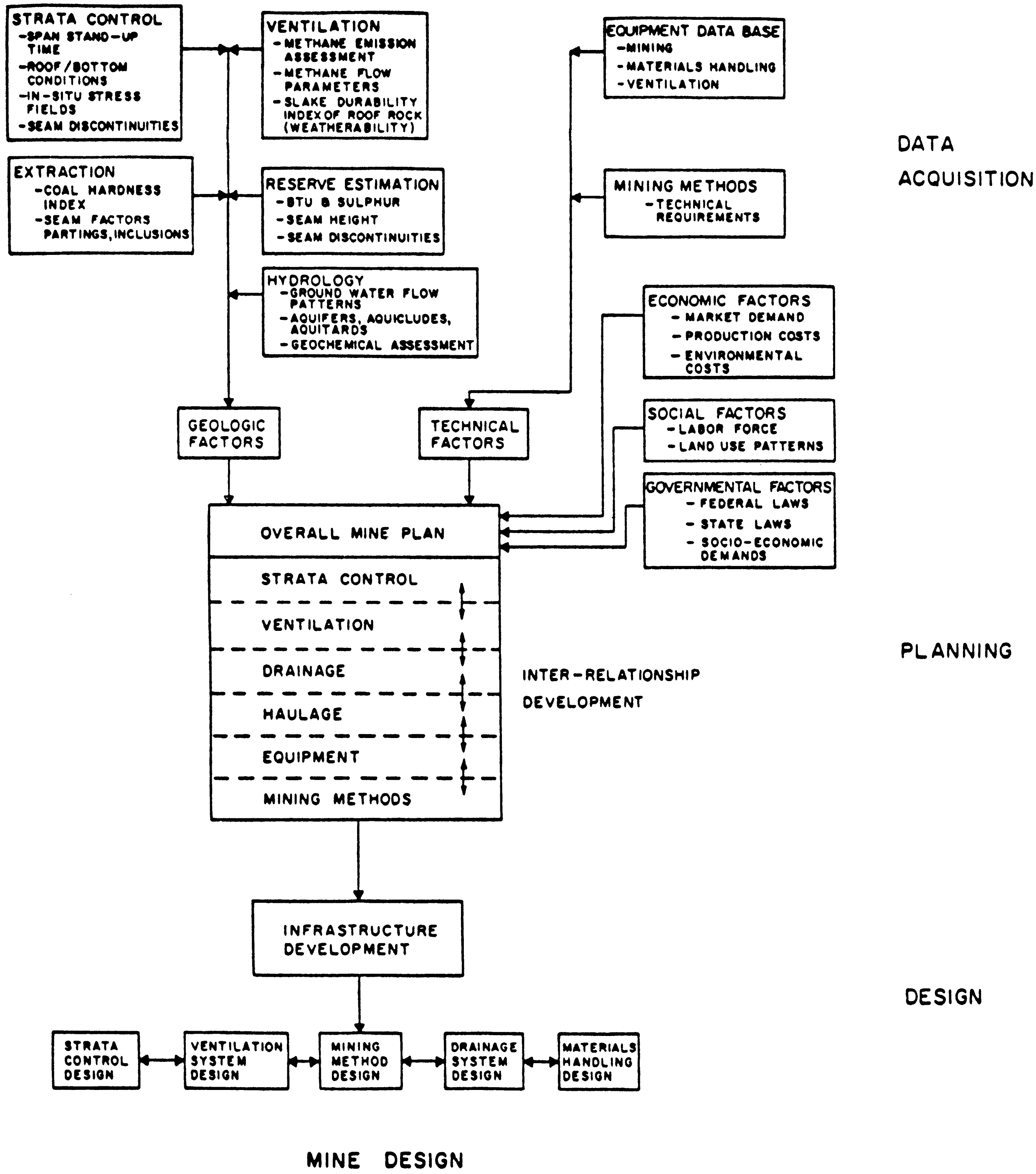
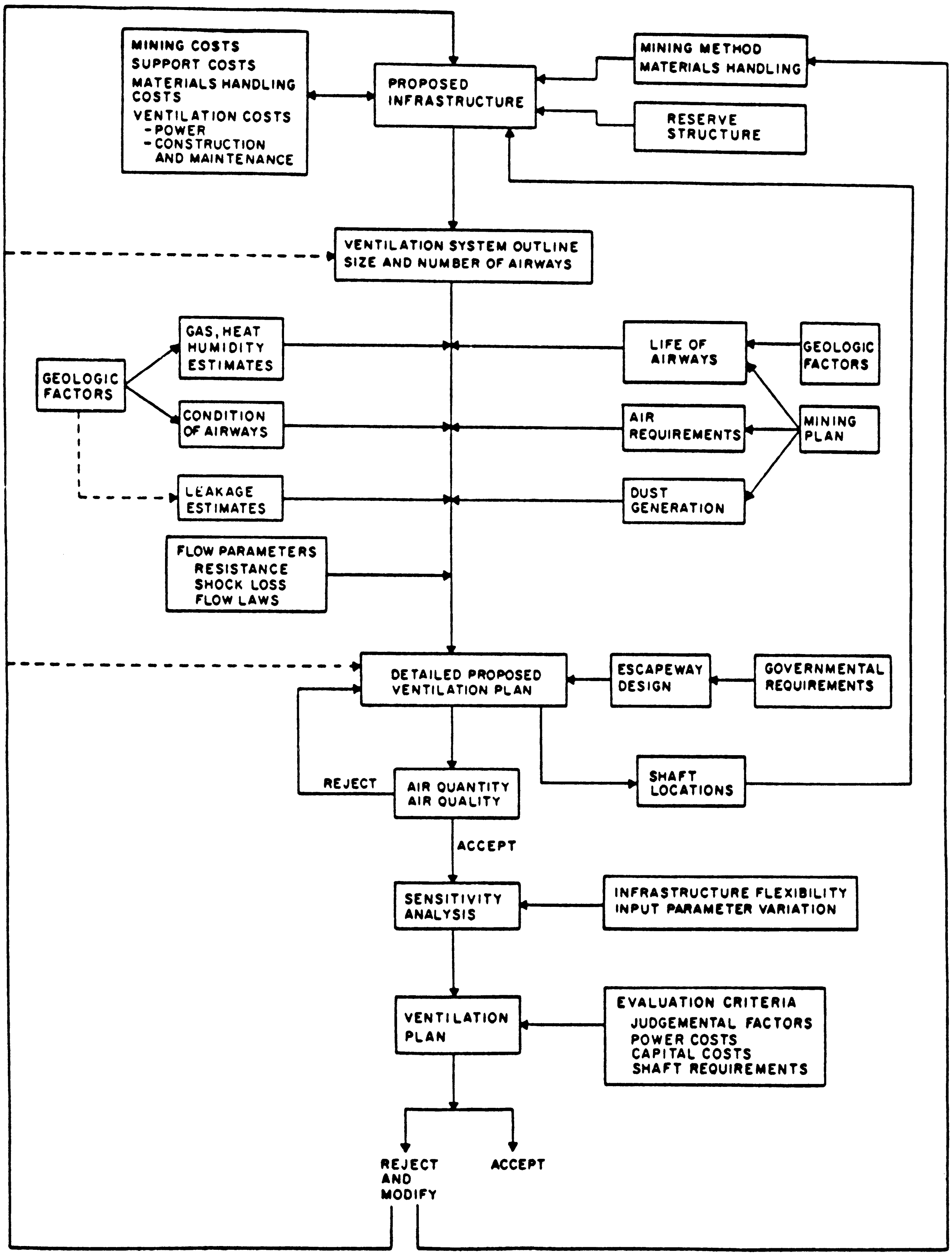


Fig. 11.7.2.1a. Mine design and ventilation system design (Luxbacher and Ramani, 1980).

main layouts, cross-sectional areas, and lining of the airways must be made. Since mine openings and fans normally function for the life of a mine, they should be selected carefully taking into account their life cycle duties. Such total system considerations can reveal the need for relocation of shafts or the opportunities for eliminating shafts in the preliminary plan. Further, they will amplify the sequencing needs for these critical ventilation-related activities.

MEDIUM-RANGE PLAN. Medium-range plans must show the projected method of working and the general method of ventilation to be used for at least three to five years into the future. These projections should be updated at least once a year to include changes and variations from the plan. These plans must be available to the operational staff to schedule ventilation-related actions in their operational plans. Medium-range plans should be based on underground pressure/quantity, methane,



VENTILATION SYSTEM DESIGN

Fig. 11.7.2.1b. Mine design and ventilation system design.

and temperature surveys except in cases where the workings are entering virgin areas. In the latter case, use of good estimation techniques for parameter values and of extensive sensitivity analysis cannot be overemphasized.

SHORT-TERM PLAN. The short-term plan, as the name implies, is for the immediate future, usually a year or so, and is based on the projected long- and medium-range plans. Short-range planning of ventilation is necessary to maintain control of the nature and timing of ventilation construction and activities. It must include at least the following: (1) layout of development work with a time sequence, (2) air quantities, (3) ventilation arrangement at the working faces, (4) schedule for ventilation changes, (5) schedule for the construction of air crossings, doors, etc., (6) special precautions for firefighting, and (7) emergency escapeways.

11.7.2.3 Quantity Distribution

Success in providing adequate ventilation to the active workings of a mine for any method of mining depends on adequate fan capacities and good air distribution (primary ventilation) and, when the air reaches the face area, good control and distribution of the air at that location (face ventilation). An acceptable system provides overcasts, airlocks, regulators, and bleeders arranged so that air is coursed in the desired manner in the desired quantities.

MAIN ENTRIES. The main entries that extend from the shafts and slopes carry the major volume of air to the splits and are usually required to last throughout the life of the mine. These two aspects make their design with regard to number, size, support, and lining material important. Since material (coal and supplies) is usually conveyed through one or more of the entries, limitations such as isolation of the belt and maximum velocity in trolley haulage airways must be considered in determining the number of airways. The air quantity flowing in the airway will be a function of the air velocity range desirable in the airway.

Example 11.7.2.1. Given these ventilation and haulage specifications, calculate the number of entries in the mains:

Mainline haulage:	belt
Supply haulage:	battery-powered scoops
Range of maximum velocity:	600 to 800 fpm (3.05 to 4.06 m/s)
Minimum quantity to be carried:	300,000 cfm (141.6 m^3/s)
Seam height:	6 ft (1.82 m)
Width of entry:	20 ft (6.1 m)

Solution. (a) In the above case, the area of an airway is 6 × 20 ft = 120 ft^2 (11.15 m^2). But it is possible that this entire area may not always be available for airflow since there may be some obstructions in the form of stationary equipment and supplies that are either required or left in the entries. Additionally, due to convergence, the actual height of the entry may be less than 6 ft (1.82 m). Assuming the actual area for airflow at 100 ft^2 (9.29 m^2) and an air velocity of 800 fpm (4.06 m/s), the number of airways needed to convey 300,000 cfm (141.6 m^3/s) will be

$$\frac{300{,}000}{800 \times 100} = 3.75 \simeq 4 \text{ entries}$$

$$\frac{141.6}{4.06 \times 9.29} = 3.75 \text{ entries}$$

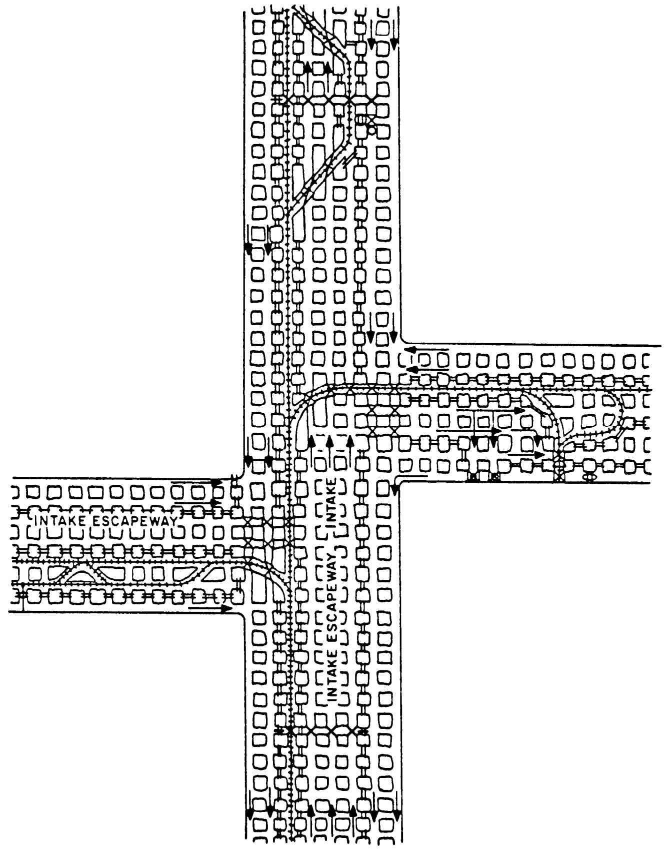


Fig. 11.7.2.2. Eight-entry main development with track haulage.

At a velocity of 600 fpm (3.05 m/s), the number of airways needed will be

$$\frac{300{,}000}{600 \times 100} = 5 \text{ entries}$$

$$\frac{141.6}{3.05 \times 9.29} = 5 \text{ entries}$$

(b) It can be assumed that the main entries will be carrying the entire quantity over only a short distance and that air from the main entries splits to the submains and panels at frequent intervals. Thus, from the velocity point of view, at least four intakes and four returns (i.e., a total of eight entries) are necessary.

(c) Because a belt entry must be isolated, an additional entry is needed to accommodate the belt, making nine the total number of entries required. If trolley haulage is used for men and material transport, then the intake escapeway must be isolated from the trolley entry. Additionally, the velocity in the trolley entry must not exceed 250 fpm (1.27 m/s). This may increase the number of entries in the mains by at least one.

If, instead of a belt, trolley haulage is used for men, supplies, and coal transport, then only the intake escapeway must be isolated from the entry in which the trolley operates. However, the velocity restriction still demands that the trolley be isolated and regulated. In this case, however, eight-entry mains may be sufficient.

Shown in Fig. 11.7.2.2 is an eight-entry mains development. In this example, the two outside entries on each side are the

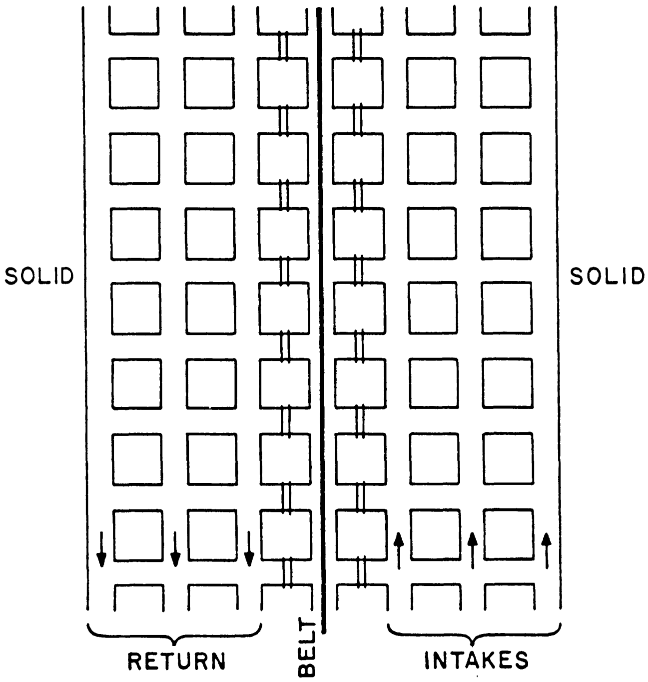


Fig. 11.7.2.3. Mains development with belt haulage to minimize number of stoppings.

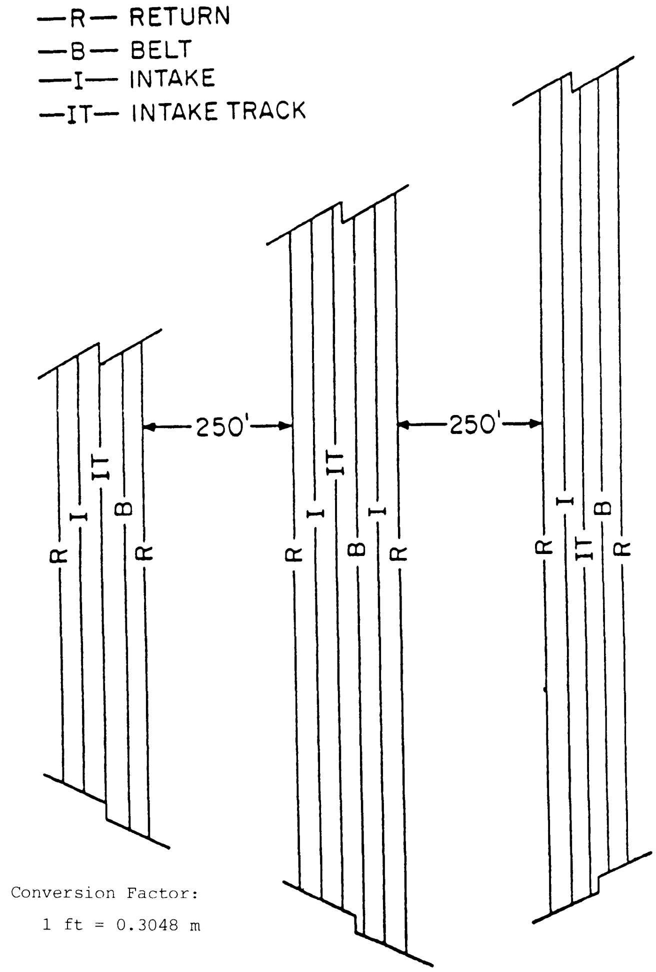


Fig. 11.7.2.4. Mains development for achieving high production and for reducing leakage. Conversion factor: 1 ft = 0.3048 m.

returns. The No. 3 entry (always counting from the left facing inby) contains the tracks for trolley haulage. The trolley entry is isolated and followed by the three intake entries. One of these is designated as the intake escapeway. Whenever the trolley haulage has to be carried into the splits, the trolley entry needs to be isolated from the other intakes and returns by a set of overcasts. It is also a good practice to connect the returns on either side by a set of overcasts every 2000 ft (600 m). These are known as "equalizers." Thus, if there is a fall on the returns blocking any set of overcasts completely, the air can cross over and back without unduly affecting the overall resistance of the returns. The idea of putting the returns adjacent to the solid rib of coal is advantageous because of any gas that bleeds off the solid coal will not be carried to the face. If symmetry with regard to the haulage is required, the trolley or belt can be moved to the fourth or fifth entry. In this case, however, an additional row of stoppings will be required (i.e., two rows of stoppings to isolate the intakes from the return on each side, and two rows to isolate the trolley or belt from the intakes on either side).

There are several other possible layouts for the mains. For example, to minimize the number of stoppings, one side could be made intake and the other return as shown in Fig. 11.7.2.3. The disadvantage of this layout is the contamination of the intake air from the solid rib bleedoff and worked-out panels. Because of ventilation economics, some mines with low methane emissions use such an arrangement (Stefanko, 1983).

Shown in Fig. 11.7.2.4 is a standard set of mains in a high-methane, high-production mine (Stefanko et al., 1977). Sixteen entries divided into three groups are laid out so that, during development, each group has its own belt (B), track (IT), intakes (I) and returns (R). Once development is complete, the outlying groups, which consist of five parallel entries, will be used as returns. The central group, which consists of six entries, will utilize five entries as intakes and an isolated belt will be placed in the No. 4 entry. The eventual isolation of the three groups into returns and intakes with 250-ft (75-m) barriers between them reduces both leakage and resistance. Furthermore, this type of mains development is suitable for use when large tonnages are being produced during first mining. Figure 11.7.2.5 shows a scheme employed at a mine employing eight entries per section with two sections paralleling each other. The two sets of entries are connected at convenient intervals and have the same advantages as listed above.

SUBMAINS. The submains or panel entries have a much shorter life than the mains and carry much less air. For example, according to the 1969 Coal Mine Health and Safety Act, the minimum volume of air required is only 9000 cfm (4.25 m^3/s) at the last open crosscut. In practice, however, the quantity of air required at the last open crosscut is a function of the methane liberation rate and the number of machine units and workers in the section. It is good ventilation practice to place the miner operator on one split of air that will be conveyed directly to the return, and the roof bolter operator (or other machine units) on another split of air. This type of face ventilation, called dual-split or fishtail arrangement, is very common in gassy mines as well. The air in the face areas is coursed by line brattice and check curtains. Here leakage losses will be severe. For example, the air quantity on the basis of the above considerations at the last open crosscut is, say, 40,000 cfm (18.88 m^3/s) (20,000 cfm, or 9.94 m^3/s, on each side). The air at the entrance to the panel

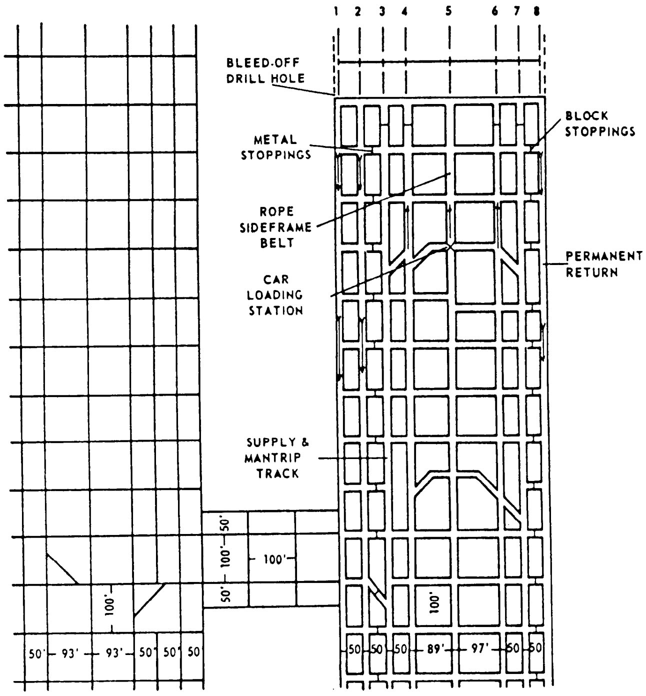


Fig. 11.7.2.5. Design of mine entries according to pressure-arch theory and reflecting future ventilation expansion (Belton, 1962). Conversion factor: 1 ft = 0.3048 m.

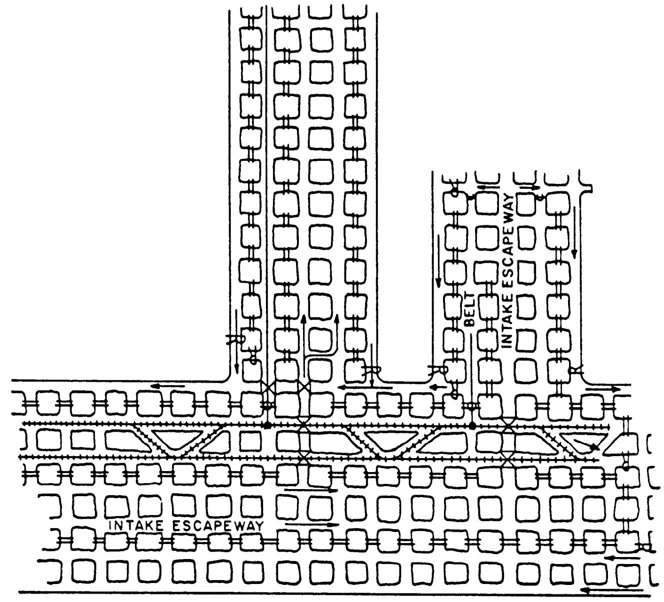


Fig. 11.7.2.6. Five-entry submains development.

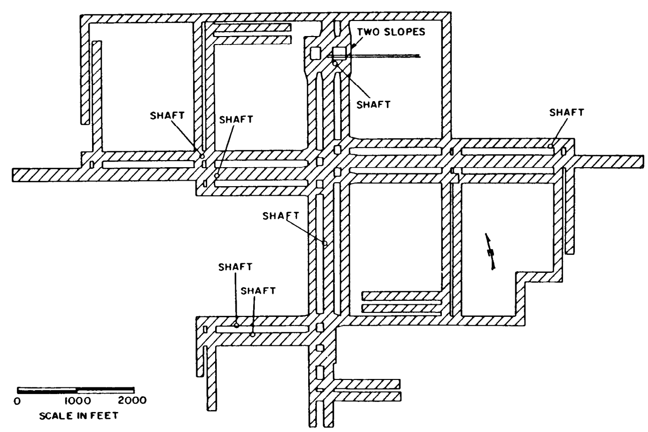


Fig. 11.7.2.7. Mine plan for high-development production and degasification—development after 36 months. Conversion factor: 1 ft = 0.3048 m.

will have to be much more than this quantity to account for the leakage from the intake into the returns all along the panel length. This can be estimated on the basis of the number of stoppings and the stopping construction. For the purposes of illustration, the air at the panel entrance is assumed to be 60,000 cfm (28.32 m^3/s). The air velocity in the submains (or panel entries) is considerably less than that in the mains.

Example 11.7.2. Given the following ventilation and haulage considerations, calculate the number of panel entries.

1. Maximum velocity: 300 fpm (1.52 m/s)
2. Width of entries: 20 ft (6.1 m)
3. Height of entries: 6 ft (1.8 m)
4. Belt for coal transportation
5. Battery haulage for men and materials
6. Minimum quantity to be carried: 50,000 cfm (23.6 m^3/s)

Solution. The cross-sectional area of an airway, although 120 ft^2 (11.15 m^2), is rounded off to 100 ft^2 (9.29 m^2) as before for the same reasons. Using the maximum velocity of 300 fpm (1.52 m/s), it is obvious that one airway can convey 30,000 cfm (14.16 m^3/s). Thus two intakes and two returns will be enough from the ventilation point of view. However, a separate entry must be driven for the isolated belt. In summary, a five-entry development is adequate.

Shown in Fig. 11.7.2.6 is an example of a five-entry development taking off from a seven-entry submain with trolley haulage. Notice that in the submains, two entries are isolated from the intakes and returns for the trolley.

Studies by the US Bureau of Mines have shown that isolating a large block of coal by developing sets of headings around it to allow bleedoff for at least one year prior to mining can provide safety and production advantages. An example of such a panel development is shown in Fig. 11.7.2.7 where several blocks have been isolated before extraction within any one block begins. From an economic point of view, this practice is not most desirable due to the high investment in the mine, the high cost of development coal and the delay caused in the start of second mining (i.e., pillaring or longwalling). Methane drainage ahead of mining through vertical boreholes from the surface or horizontal holes in the seam is a preferred method (Deul and Kim, 1986).

11.7.2.4 Longwall Ventilation Planning

The development for a longwall face consists of two sets of parallel entries with two, three, or four entries in each set. The most common in the eastern United States is the three- or four-entry development, accounting for over 80% of the longwalls in operation. In the western United States, two-entry developments are widely used. The distance between the two sets (i.e., the face length) is variable and can be anywhere from 450 to 1000 ft (137.2 to 304.8 m). These entries may be driven 6000 to 10,000 ft (1829 to 3048 m) in length.

The amount of air required in a longwall face is dictated by the methane emission rate during mining and conveying. The area available for the air to travel from the intake entries to the return entries is limited. The total cross-sectional area of the face is determined by the height of coal cut (usually the thickness of the coal seam), and the distance from the face to the gob line (usually, 12 ft (3.7 m)). However, the area occupied by the machine, roof support equipment, and conveyor must be subtracted from the total area to calculate the effective area available for ventilation. This area may be as little as 50% of the total face area. The net result of such a low area for ventilation is high air velocities at the face. While it is advantageous from the methane point of view to have high velocities, from the dust point of view, velocities over 500 fpm (2.54 m/s) are undesirable. Assuming a 5-ft (1.5 m) coal seam and a 12-ft (3.66 m) distance from the face to the gob line, the maximum cross-sectional area for airflow is 60 ft^2 (5.57 m^2). Assuming that only 75% of this area is available for ventilation, the remaining 25% being covered by obstructions such as the machine, conveyors, and support, the maximum velocity in the face for a flow of 22,500 cfm (10.62 m^3/s) will be about 500 fpm (2.54 m/s). A limit on the face velocity, somewhat on the high side, is 600 fpm (3.05 m/s). Thus, if the methane emission rate per ton of coal mined is high, then the emission of methane can be reduced only by mining at a lower rate, by degasifying the seam prior to longwalling by isolated panels and/or by drainage through vertical or horizontal holes, or by water infusion.

As an illustration, consider a gas emission rate of 10 ft^3/ton (0.31 m^3/t) of coal mined and a maximum production rate of 10 tpm (9 t/m). The amount of gas liberated is 100 cfm (0.047 m/s), requiring nearly 10,000 cfm (4.72 m^3/s) of fresh air for dilution at the face. An additional 5000 cfm (2.36 m^3/s) can be assumed to directly enter the gob from the face. Therefore, at the entrance to the face, there must be 15,000 cfm (7.08 m^3/s). Providing for leakage, say, 5000 cfm (2.36 m^3/s), in conveying the air from the main intakes to the face through the panel entries for ventilating the panel belt, say, 3000 cfm (1.42 m^3/s) is needed; and for other purposes (5000 cfm, or 2.36 m^3/s); a total of approximately 28,000 cfm (13.21 m^3/s) must be obtained from the main intake. This quantity can be easily handled by a single entry whose cross-sectional area is 100 ft^2 (9.29 m^2). However, requirements of a neutral belt, bleeding from the solid coal, and the entries to serve the adjacent longwall face, demand a two-, three- or even a four-entry longwall development. In modern high-production longwalls, the air quantity traversing the face can be as much as 30,000 to 50,000 cfm (14.16 to 23.60 m^3/s).

RETREATING LONGWALL VENTILATION. A four-entry layout for longwall mining is shown in Fig. 11.7.2.8. The ventilation system consists of the following: two butts, labeled l-Butt and 2-Butt in Fig. 11.7.2.8, each consisting of four entries, are driven the length of the panel; the butts are connected by four entries in the back (three rows of pillars). The three most inby entries form the bleeder (and the rib of the 4th entry forms the retreating longwall face). Entry No. 4 of 2-Butt is the isolated belt entry. Entries 2 and 3 of 2-Butt carry the major portion of the intake air (about 20,000 cfm or 9.45 m^3/s) of which about 18,000 cfm (8.50 m^3/s) will reach the headgate. At this point, about 15,000 cfm (7.08 m^3/s) will enter the face, and 3000 cfm (1.42 m^3/s) will be allowed to return through the belt into the main returns. A small quantity of air, say, 3000 cfm (1.42 m^3/s), will be allowed to enter entry 1 of 2-Butt and be directly conveyed to the bleeder. This will enable bleeding of the solid block to the left. A small amount of intake air (say, 5000 cfm, or 2.36 m^3/s), will enter through entry No. 1 of l-Butt that will be used to ventilate the A entries and the tail end of the face. Entries 2 and 3 of 1-Butt serve as the returns for the panel. Fig. 11.7.2.9 shows the ventilation plan for a three-entry longwall layout in a 5-ft (1.5-m) thick seam.

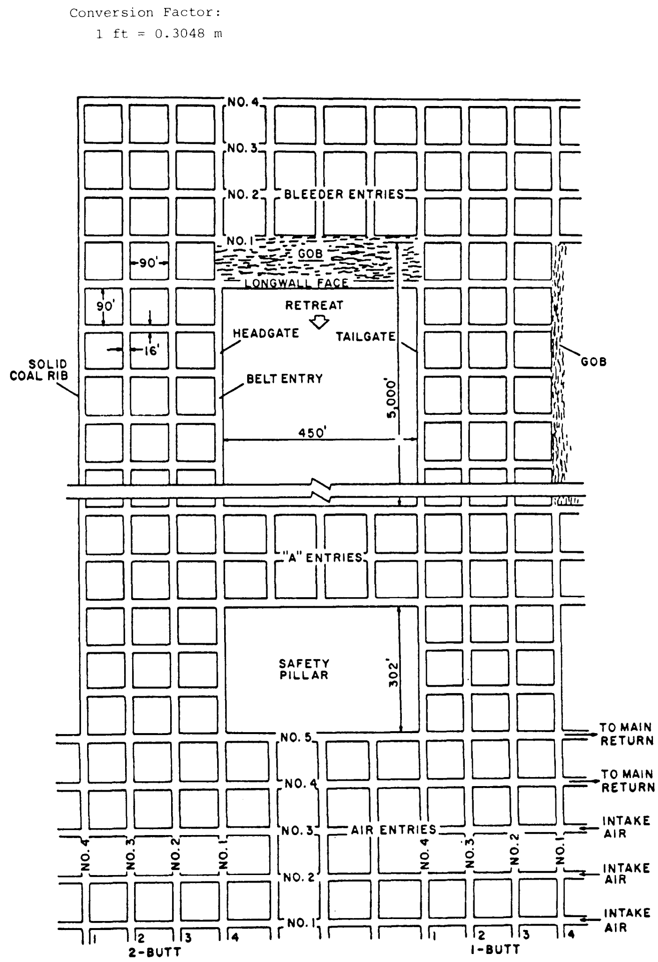


Fig. 11.7.2.8. Four-entry longwall ventilation scheme (Anon., 1976).

Dust control in longwalls is a problem, particularly when the face is mined in both directions (see Chapter 11.2). In bidirectional cutting, when cutting in the direction of the flow of air, the dust generated by the machine is blown away from the operator. However, when cutting in the reverse direction, the operator and other workers are in the downwind side.

ADVANCING LONGWALL VENTILATION. Fig. 11.7.2.10 illustrates the ventilation system of an advancing longwall face (Dalzell, 1972). In this system, the five-entry panel is advanced by a continuous miner ventilated by 17,000 cfm (8.02 m^3/s) of air while the longwall face is ventilated by a separate split of 18,000 cfm (8.50 m^3/s). The No. 4 entry in the panel serves as a common intake to the development section and the longwall face up to the crosscut inby the longwall face where the air is split to provide separate air currents to each unit. The longwall and the development utilize a common belt in the No. 3 entry of the panel. The belt entry is ventilated by low-velocity air traveling in by along it to a regulator installed on the third crosscut outby the No. 1 butt development faces where it enters the No. 1 and No. 2 entry return air courses.

The bleeder system for the advancing gob is developed by converting into bleeders the No. 5 entry, of the active panel (adjacent to the gob) and No. 1 and 2 entries of the previously mined panel which is the tailgate for the advancing longwall. Bleeder flow is controlled by checks and regulators. The return air from the longwall face is directed to the main returns through

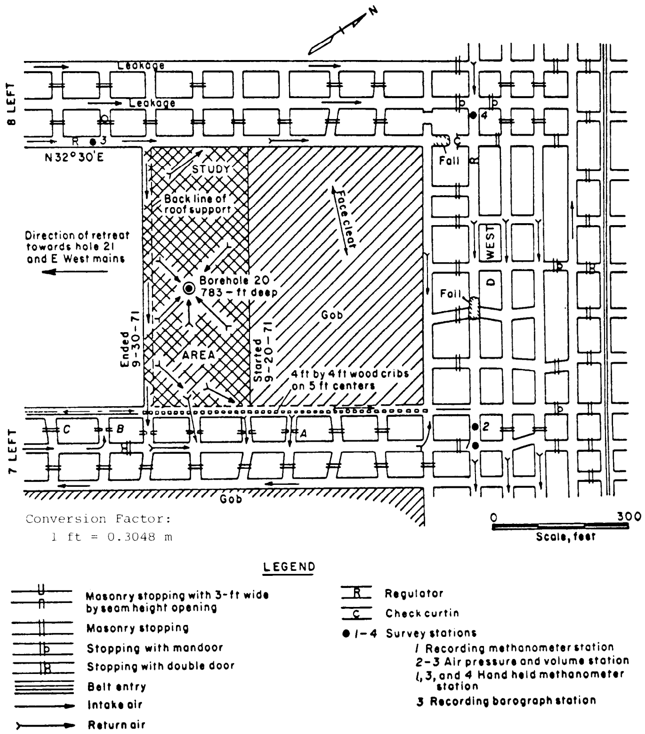


Fig. 11.7.2.9. Three-entry longwall ventilation scheme (Moore, Deul, and Kissell, 1976).

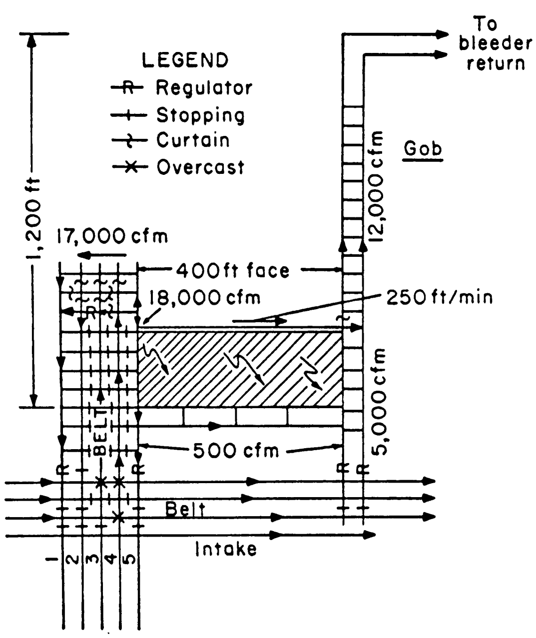


Fig. 11.7.2.10. Advancing longwall face ventilation (Dalzell, 1972). Conversion factors: 1 ft = 0.3048 m, 1 cfm = 0.47195 × 10^{-3} m^3/s.

No. 1 and No. 2 entries of the previously mined panel (5000 cfm or 2.36 m^3/s) with the remainder diverted to the bleeder entries (12,000 cfm or 5.66 m^3/s), allowing 1000 cfm (0.472 m^3/s) leakage through the active gob to the No. 5 entry to the active panel. The advancing longwall system is not commonly practiced in the United States.

11.7.2.5 Shortwall Ventilation

The shortwall layout, as shown in Fig. 11.7.2.11, is very similar to that of a longwall (Stefanko, 1983). The primary difference is that the shortwall face is generally 150 to 250 ft (45.7 to 76.2 m) wide. The web depth is about 10 ft (3.0 m), as opposed to a maximum of about 30 in. (0.76 m) in a longwall. The minimum operating height of a shortwall face is about 54 in. (1.37 m). Thus the face area open for ventilating air is much larger than that for a longwall.

The shortwall face shown in Fig. 11.7.2.11 was located in Eastern Kentucky in the Elkhorn No. 3 seam that averages 44 in. (1.12 m) in thickness, ranging from 42 to 48 in.(1.07 to 1.22 m). The main entries, which are not shown in the figure, consist of five intakes and four returns in a two-split arrangement with two returns on each side. The submains, represented by 10 Left in the figure, consist of three intakes, one controlled belt-track intake, and five returns in a two-split or fishtail arrangement. One Right Flats, off 10 Left Submain, has a similar configuration but consist of only seven entries. The panel entries perpendicular to One Right Flats consist of an intake on the right side (No. 3 entry) with a return in No. 1 entry and a neutral belt-track entry in No. 2. A bleeder system is shown in use with panel production. This ventilation system is very effective in maintaining the methane below the 1% level.

In shortwall, the continuous miner cuts only in one direction. Then it is backed up to the headgate and another cut is started. Therefore, by arranging the ventilation flow from headgate to tailgate, the miner operator and the jack setters are always in fresh air and are not exposed to the dust generated by the machine. Other than this, there is essentially little difference between the ventilation of longwall and shortwall faces.

11.7.2.6 Bleeders

Methane control during development, although requiring constant and careful attention, is easier than methane control during pillaring. During development, frequent inspections and regular monitoring are possible. However, as soon as conventional pillaring or retreat of longwall is started, it is impossible to inspect the gob areas. To keep the gob clear of gases, a controlled flow of air across it must be established. This is known as bleeding the gob (Kalasky and Krikovic, 1973). It is usually achieved by placing regulators at strategic locations at both panel ends and utilizing whatever permeability has been established in the gob. The location of the regulators should be chosen to provide as much flow across the gob as is needed to sweep away gases and avoid dangerous accumulations in the gob. These regulators should be accessible since, as retreat mining increases, adjustment regulator is needed to compensate for the increased natural resistance of the developing gob. To accomplish “bleeding,” an adequate pressure differential from the front to the back of the pillar line, as well as to both sides, is required.

Effective bleeding should remove the methane sufficiently far from the active pillar line or face so that a temporary ventilation interruption or sudden atmospheric pressure drop due to an approaching storm will not cause a large influx of gas from the gob.

Supplementary degasification through vertical boreholes drilled from the surface ahead of approaching pillaring lines or in the longwall gob is also practiced. However, effective bleeding systems should still be used to reduce the methane problem in the gob.

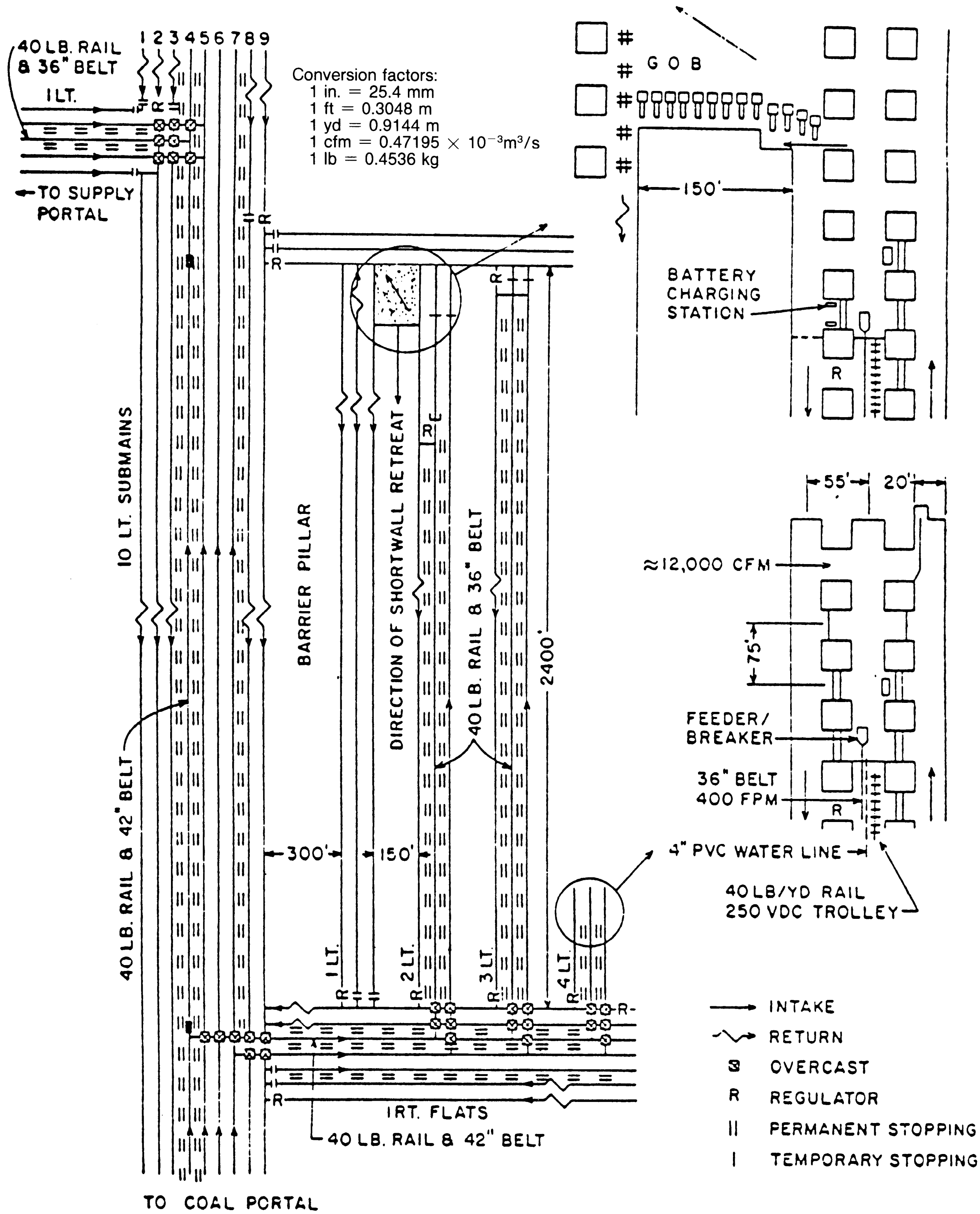


Fig. 11.7.2.11. Shortwall ventilation scheme (Stefanko, 1983).

Bleeders for Room and Pillar Systems. A simple bleeder entry system for room and pillar mining is shown in Fig. 11.7.2.12. Here the mining method involves mining panel headings usually advancing from one side of a set of submains to another set, that is, from 6 North to 7 North, and from 7 North to 8 North, as shown. When the panel headings approach the submains, 7 North and 8 North, they are reduced to a single heading that acts as the bleeder. Regulation is provided in the

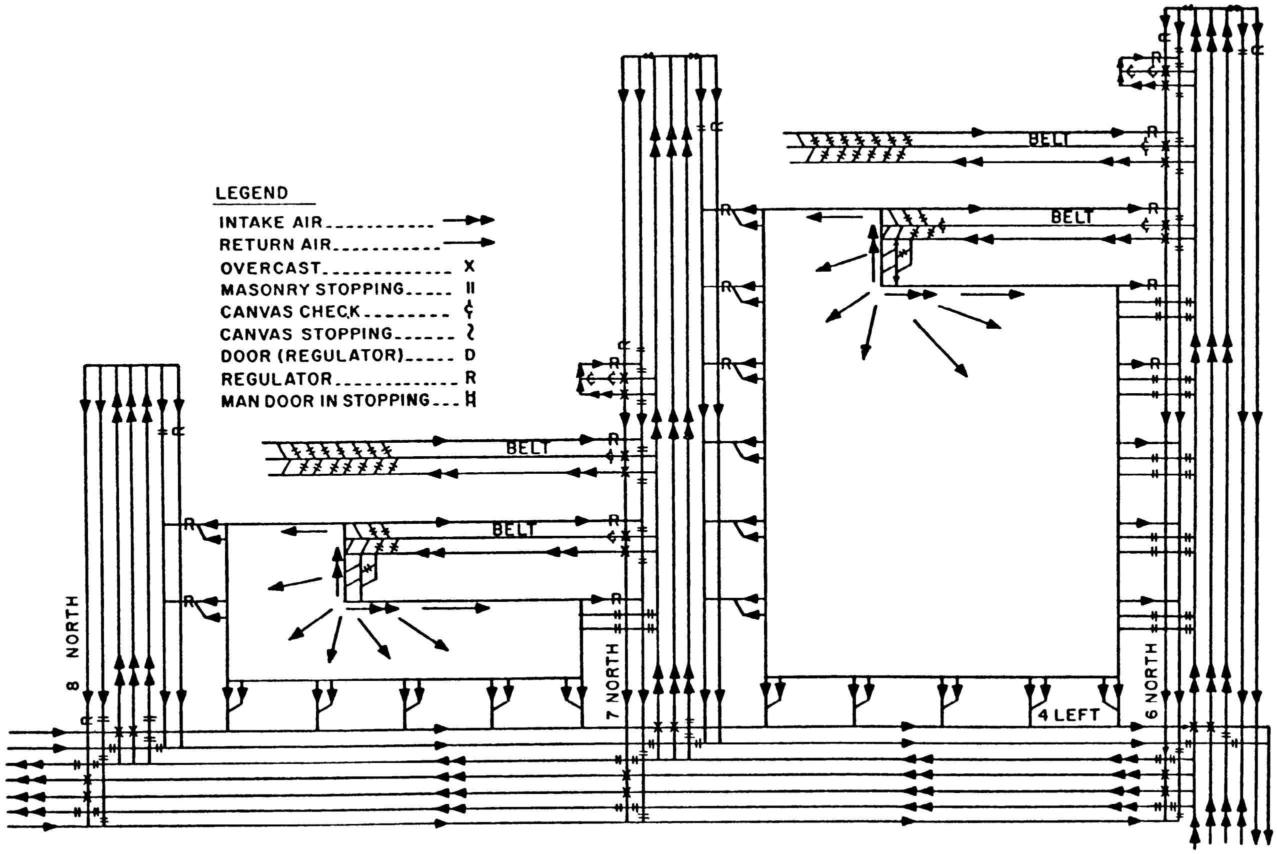


Fig. 11.7.2.12. Bleeder system for room and pillar workings (Kalasky and Krikovic, 1973).

bleeder system as the gob becomes larger in size; regulators in the bleeders are closed to provide greater resistance to flow through the more permeable areas of the gob, and opened to provide more flow through the less permeable areas. If the pressure differential across the gob is not adequate, the fan pressure may have to be increased, and, if that proves impractical, vertical boreholes must be drilled from the surface and a high-pressure exhauster installed on the top. One significant advantage of this arrangement is that the bleeders are accessible from the intake airway. This facilitates cleanup of falls and pumping of water accumulations. Otherwise, such conditions would cause restrictions to adequate bleeding.

In the plan of mining shown in Fig. 11.7.2.13, panel headings are driven to mine the block of coal on the left side of the 6 North submains, and retreated on an adjacent side (from 7 North submains) with intermediate bleeders. In this case, the distance between the submains is double the panel length, minimizing the number of submains required for development. In this method of extraction, the same bleeder headings utilized in mining the first group of panels are also utilized for the second group. This arrangement may be unsatisfactory, particularly in very gassy mines, since it is difficult, if not impossible, to maintain the bleeder headings between the two gobs, especially during the final retreating cycle. In any case, all bleeders in this arrangement must be adequately supported. For pools of water that might block the bleeders, one remedy is to drive panel headings in advance and provide pumping facilities to remove water and, of course, avoid placement of bleeders in the low spot of a seam.

An alternative method of providing bleeders for the mining plan is shown in Fig. 11.7.2.14. Here a four-entry system is developed in between the submains with two intakes in the center and a return on either side. The butt drivage at the end is reduced to a single heading (bleeder) and is connected to the nearest return with regulation in the bleeder. This plan provides an effective and flexible bleeder system, approachable from the intake side for cleaning the bleeders and adjusting the regulators.

BLEEDER ENTRIES FOR LONGWALL SYSTEMS. Bleeder entry development for longwall mining calls for careful planning. A good method consists of providing a set of four or five entries as main bleeder returns with solid barriers on each side (Fig. 11.7.2.15). The head and tailgate development for the longwall will be connected at the back end by four or five entries that form the backend bleeder.

Initially, at one mine, the longwall block was retreated from the outside heading of the backend bleeder. Present design (Fig. 11.7.2.16) provides for a protective barrier of 200 ft (61 m) where three rooms on narrow centers are provided when the longwall is started. Extremely gaseous conditions encountered when the immediate roof and overlying strata collapse here necessitated the use of vertical degasification boreholes equipped with exhausters, in some cases to supplement the underground bleeder system.

The problems involved in bleeding and bleeder system development are extremely difficult ones, and deserve critical attention in the planning stage.

11.7.2.7 Ventilation Planning Example

Example 11.7.2.1. To illustrate the complete procedure involved in designing a coal mine ventilation system, the mine

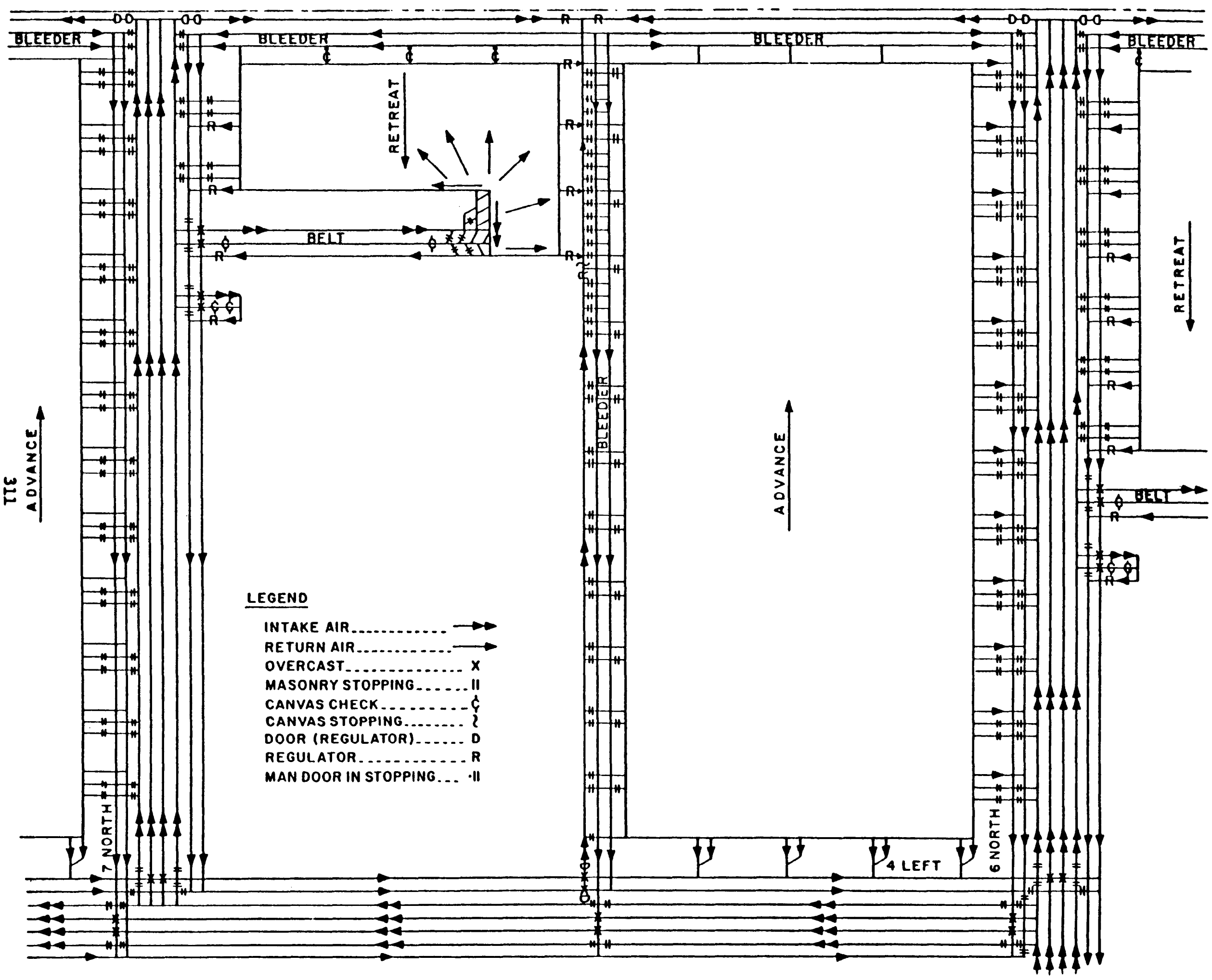


Fig. 11.7.2.13. Central bleeder system (Kalasky and Krikovic, 1973).

shown in Fig. 11.7.2.17 is used (Stefanko, 1983). It is located on a property 1 by 3 miles (1.6 by 4.8 km) in area at a depth of 1000 ft (305 m) in a highly gaseous seam. The mains are driven in the north-south direction (3 m or 4.8 km) along the center of the property. The property will be mined on a half-advance, half-retreat arrangement. Panels (2000 ft or 610 m long) will be driven and mined to the left on advance until the property line is reached; panels will then be driven on the right on retreat. Only the northern half of the mine on advance is shown as the southern half is a mirror image of the northern half. All entries are 16 ft (4.9 m) wide on 100-ft (30.5 m) centers. It is assumed that six machine units (four on development and two on pillaring) are required to meet the production tonnage of 2000 tons (1814 t)/ shift. Panel belts will transfer coal to a main belt.

Solution.

QUANTITY CALCULATIONS—MINING UNIT. Assuming a gas liberation rate of 40 ft^3/ton (1.25 m^3/t) of coal mined and a mining rate of 5 tpm (4.54 t/min), the amount of gas liberated per minute during cutting is 200 ft^3 (5.66 m^3). The amount of fresh air required to dilute the gas to a methane content below 1% is 20,000 cfm (9.44 m^3/s).

It is good practice from the health point of view to provide two splits of air in a panel, one for the continuous miner and another for the roof bolter. In this arrangement, the dust-laden air from the continuous miner is placed directly in the return. Further, there is no line brattice through which shuttle cars must tram, reducing leakage which otherwise adversely affects the air distribution in the face area. Thus two splits of air, 20,000 cfm (9.44 m^3/s) for the miner and another 20,000 cfm (9.44 m^3/s) for the other machine units, are utilized in a section.

BELT VENTILATION. Belts may not be placed in main intakes or returns but must be isolated in order to prevent the spreading of a fire, and only that amount of air is to circulate in them that is necessary to maintain methane concentrations below 1%. The air passing over the belt must be placed directly into the return rather than being transported to working faces. Since there is a certain amount of air leakage where the belt passes through the doors near the head end, belt air travel direction is usually into the panel to utilize the leakage. Normally, a temporary stopping will be placed across the belt entry just outby the belt-loading point with a regulator in the outby crosscut that regulates the belt air directly to the return.

In the sample calculation, a small quantity of air (5000 cfm or 2.36 m^3/s) will be split from the main intakes at the head of the belt and dumped directly into the returns near the tail of the belt.

LEAKAGE CONSIDERATIONS. There will be leakage through the stoppings in the panel from intake to return. The quantity

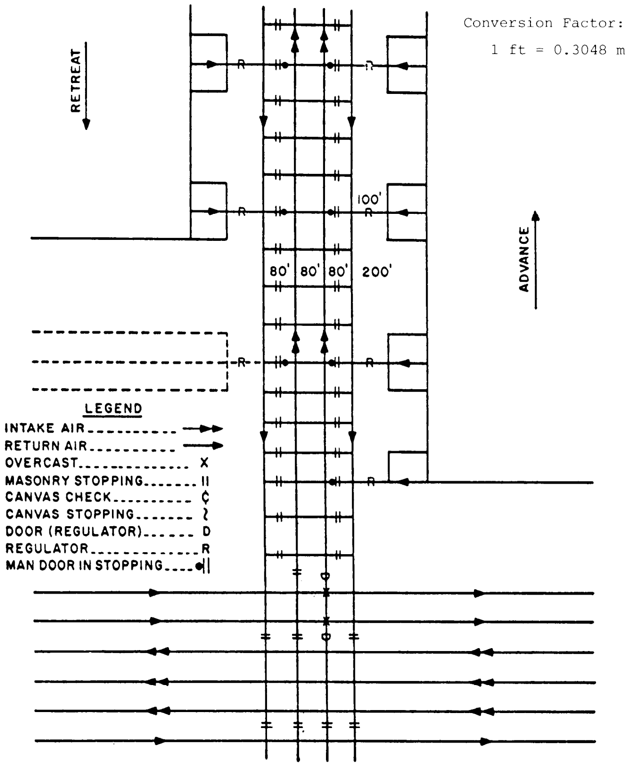


Fig. 11.7.2.14. Better design for a central bleeder (Kalasky and Krikovic, 1973).

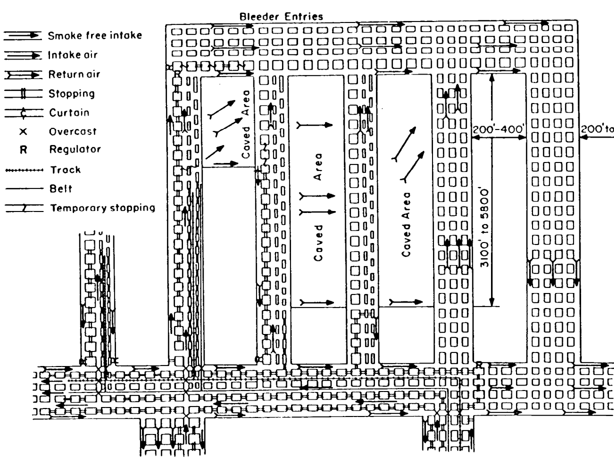


Fig. 11.7.2.15. Bleeder system for longwall workings—Case A (Kalasky and Krikovic, 1973). Conversion factor: 1 ft = 0.3048 m.

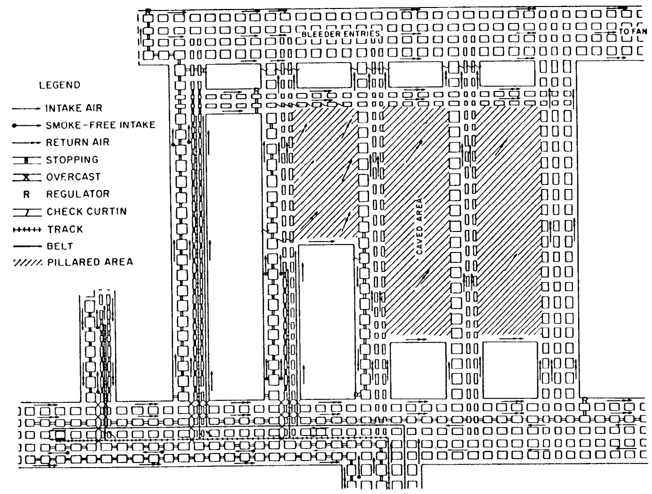


Fig. 11.7.2.16. Bleeder system for longwall workings—Case B (Kalasky and Krikovic, 1973).

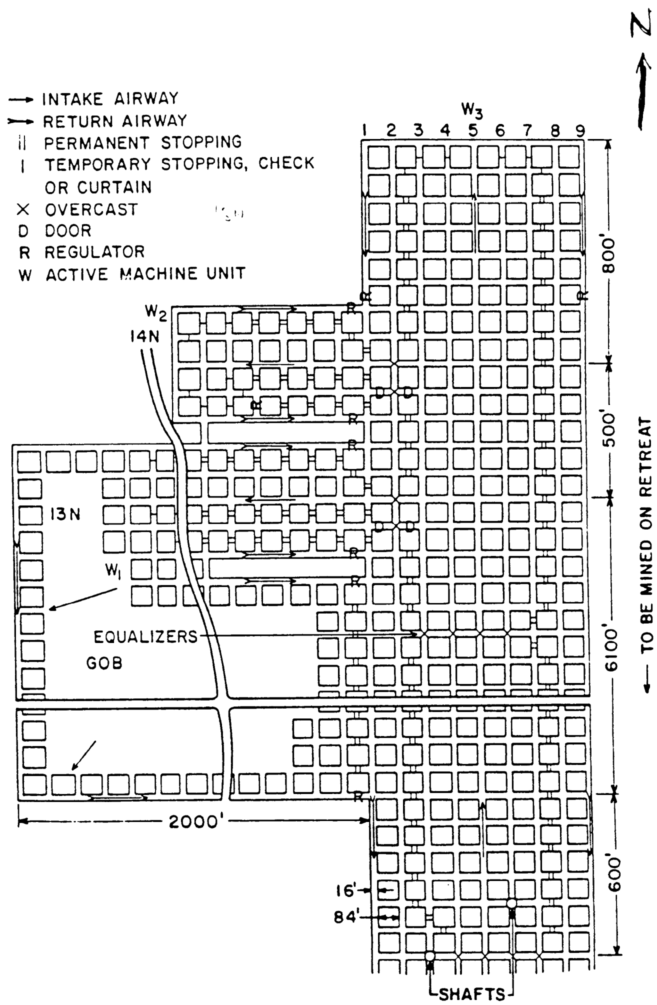


Fig. 11.7.2.17. Double-split system of mine ventilation. (Stefanko, 1983). Conversion factor: 1 ft = 0.3048 m.

of air required at the entrance to the panel will be much higher than the 40,000 cfm (18.88 m^3/s) calculated as required at the last open crosscut. When the panel is fully developed (i.e., 2000 ft or 610 m, with 20 crosscuts), there will be two rows of stoppings between the intakes and the returns on either side of the intakes, and one additional row of stoppings for the isolated belt. Assuming a maximum leakage of 1000 cfm (0.472 m^3/s) through the stoppings in each crosscut, a total leakage quantity of 20,000 cfm (9.44 m^3/s) is estimated, making the air required at the panel entrance 65,000 cfm (30.68 m^3/s) (60,000 cfm, or 28.32 m^3/s, for the unit + 5000 cfm, or 2.36 m^3/s, for the belt).

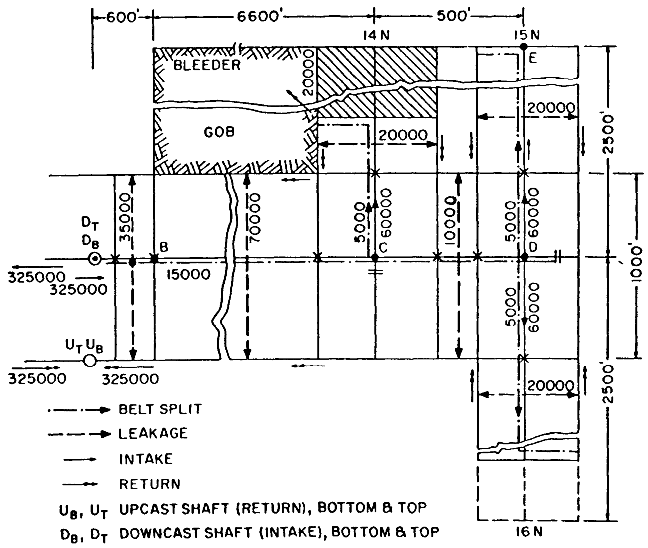


Fig. 11.7.2.18. Ventilation schematic for mine in Fig. 11.7.2.17. Conversion factors: 1 ft = 0.3048 m, 1 cfm = 0.47195 × 10^{-3} m^3/s.

The air quantity that must be apportioned at the fan for a panel must provide for air leakage between the main intakes and returns through the stoppings, overcasts and mandoors. The farthest panel (i.e., the one at the property line) will be approximately 8000 ft (2438 m) or 80 crosscuts away from the fan shaft. Fortunately, control devices in the mains are built more substantially than those in panels because they have to be in service through the life of the mine. Assuming 500 cfm (0.236 m^3/s) leakage of air per stopping through each crosscut, a total leakage of 40,000 cfm (18.88 m^3/s) is estimated, making the air required at the fan 105,000 cfm (49.55 m^3/s) for a panel. Thus, with three active panels in the north, the total air required for the panels will be 315,000 cfm (148.66 m^3/s). To this must be added the air required for the main belt, which is estimated at 10,000 to 15,000 cfm (4.72 to 7.078 m^3/s), making the total air required for the north side 325,000 cfm (153.38 m^3/s). The air required for the south side will be another 325,000 cfm (153.38 m^3/s), making a total fan quantity of 650,000 cfm (306.76 m^3/s).

HEAD LOSS CALCULATIONS—MAIN ENTRIES. The cross-sectional area of an entry is 16 by 6 ft (4.88 by 1.83 m) = 96 ft^2 (8.92 m^2). The maximum velocity permitted in the mains is, say, 800 fpm (4.06 m/s). Then one entry can carry 76,800 cfm (36.25 m^3/s) of air. The number of entries needed to carry 315,000 cfm (148.66 m^3/s) (10,000 cfm, or 4.72 m^3/s, will be carried by the belt entry) is just over four entries (315,000 ÷ 76,800). Because the air from the mains is split into the submains and panels, and leakages occur between the main intakes and returns, in practice, the mains handle progressively lower quantities. On the other hand, the entire 96-ft^2 (89.2-m^2) area of an entry may not be available for airflow. Therefore, for this example, the number of intake entries is set at five, the number of return entries at four and one isolated belt entry is provided, making a 10-entry main.

PANEL ENTRIES. Since the panel entries have to carry only 60,000 cfm (28.32 m^3/s), a three-entry development—one intake entry, one return and one isolated belt entry—can suffice. However, for providing a separate intake escapeway and greater production potential, a five-entry development is adequate.

BLEEDER. The quantity of air to be regulated into the bleeder from the active pillaring section is 20,000 cfm (9.44 m^3/s). The air flows through the gob and joins the main return at a distance of 600 ft (183 m) away from the return shaft.

LEAKAGE. For the purpose of head loss calculation, unless it is definitely known that an inordinate quantity of air is leaking through a particular airway, it is assumed that leakage air is uniformly distributed over the entire airway. Thus the quantity flowing in an airway is the average of the quantity flowing in the beginning and end of the roadway. In the ventilation schematic shown in Fig. 11.7.2.18, the North mains have been developed to the property limits. The most critical situation with regard to ventilation will arise when the unit in 14-N panel will be pillaring, 15-N panel is nearing completion of development to the boundary to the west prior to pillaring, and 16-N panel (the first panel to the east of the mains) will be on development. Examination of the ventilation schematic reveals that the free split is the path from the downcast shaft through the 15-N panel to the upcast shaft.

FRICTION FACTORS. The values for friction factors must always be chosen on the basis of field measurements under conditions that are similar to the mine conditions. For new mine projections in virgin areas, the following general values can be used.

Concrete-lined shaft:	15 × 10^{-10} lb-min^2/ft^4	(0.00278 kg/m^3)
Intake airways:	80 × 10^{-10} lb-min^2/ft^4	(0.0148 kg/m^3)
Return airways:	100 × 10^{-10} lb-min^2/ft^4	(0.0186 kg/m^3)

SHOCK LOSS CONSIDERATIONS. In the calculation of head loss in each segment of the airway, consideration must be made for shock losses that may be encountered due to such conditions as splitting, changes in areas, overcasts, and changes in direction. Shock losses can be minimized by gradual changes in area and direction, but can never be eliminated. The longest segment in the example mine is from the bleeder junction (B) to the 14N entrance (C). Considerations for shock loss on the basis of the number of overcasts, doorways, obstructions, and area changes must be made for this segment. The second largest segment is the panel mains in 15N from D to E. Here also, considerations for shock loss on the basis of the number of doorways and obstructions must be made. In the calculations in Table 11.7.2.1, therefore, equivalent lengths of 1000 ft (305 m) for the segment BC and 200 ft (61 m) for the segment DE have been assumed.

TOTAL HEAD LOSS. It is convenient to calculate the head loss in each segment of the airways and add the head losses in series to calculate the total head loss. It is also preferable to use a tabular format for the calculations. The following formulas are used for calculating head loss (see Chapter 11.6):

$$H = RQ^2 \tag{11.7.2.1}$$

where R is resistance of the airways in parallel

$$R = \frac{R_1}{n^2}, \tag{11.7.2.2}$$

where R_1 is resistance of one airway, n = number of identical airways in parallel.

$$R_1 = \frac{K(L + L_e)O}{5.2A^3} \tag{11.7.2.3}$$

$$R_1 = \frac{K(L + L_e)O}{A^3} \tag{11.7.2.3a}$$

Table 11.7.2.1. Resistance Calculations for the Mine Segments

Mine Segment	Description Figure	Code in Fig. 11.7.2.18	$K \times 10^{10+}$ lb-min²/ft⁴	Distance L ft	Equivalent Length L_e ft	Height H ft	Width W ft	Perimeter $O = 2(H + W)$ ft	Area $A = H \times W$ ft²	Resistance of one entry, R_1 $\frac{K(L + L_e)O}{5.2A^3}$ in.-min²/ft⁶	No. of Partial Entries	Total Resistance $R_T = R/n^2$
1	Intake shaft	D_T–D_B	15	1000	0	12	20	64	240	0.0134	1	0.0134
2	Intake shaft bleeder J	D_B–B	80	600	0	6	16	44	96	0.4591	5	0.0184
3	Bleeder J–14N	B–C	80	6600	1000	6	16	44	96	5.8153	5	0.2326
4	15N–15N	C–D	80	500	—	6	16	44	96	0.3826	5	0.0153
5	15N Panel Intake	D–E	80	2000	200	6	16	44	96	1.6834	2	0.4208
6	15N Panel Return	E–D	100	2000	200	6	16	44	96	2.1043	4	0.5261
7	15N–14N	D–C	100	500	—	6	16	44	96	0.4783	4	0.0299
8	14N-Bleeder J	C–B	100	6600	1000	6	16	44	96	7.2691	4	0.4543
9	Bleeder J	B–U_B	100	600	—	6	16	44	96	0.5739	4	0.0359
10	Upcast Shaft	U_B–U_C	15	1000	—	diameter 18		56.55	294.5	0.0100	1	0.0100

Conversion factors: 1 lb-min²/ft⁴ = 1.855 × 10⁶ kg/m³, 1 ft = 0.3048 m, 1 ft² = 0.0929 m², 1 in.-min²/ft⁶ = 1.117 × 10⁹ N-s²/m⁸.

Table 11.7.2.2. Head Loss Calculations for the Mine Segments

Mine Segment	Quantity in 100,000 cfm: Begin Q_B	Leakage Q_L	End $Q_E = Q_B - Q_L$	Average $Q_A = \frac{Q_B + Q_E}{2}$	Total Resistance R_T (in.-min²/ft⁶ × 10¹⁰)	Head Loss $H_L = R_T Q_A^2$ in. water
1. D_T–D_B	6.5	0	6.5	6.5	0.0314	0.5662
2. D_B–B	3.10[1]	0.35	2.75	2.925	0.0184	0.1574
3. B–C	2.75	0.70	2.05	2.40	0.2326	0.1574
4. C–D	1.40[2]	0.10	1.30	1.35	0.0153	0.0279
5. D–E	0.60	0.20	0.40	0.50	0.4208	0.1052
6. E–D	0.40	0.25[3]	0.65	0.525	0.5361	0.1450
7. D–C	1.45[4]	1.10	1.55	1.50	0.299	0.0673
8. C–B	2.00[5]	0.70	2.70	2.350	0.4543	2.5089
9. B–U_B	2.90[6]	0.35	3.25	3.075	0.0359	0.3395
10. U_B–U_C	6.5	0	6.5	6.5	0.0100	0.4225
						Σ = 5.6797

Notes:
1. A split of 15,000 cfm is taken for the main belt.
2. A split of 65,000 cfm is taken for the 14-N section and panel.
3. The leakage into return also includes the belt return.
4. This includes return from 15N, 16N and main belt.
5. Return from 14-N is 45,000 cfm.
6. Return from the bleeder 20,000 cfm.

Conversion factors: 1 cfm = 0.47195 × 10⁻³ m³/s, 1 in.-min²/ft⁶ = 1.117 × 10⁹ N-s²/m⁸, 1 in. water = 248.84 Pa.

where R is the resistance in in.-min²/ft⁶ (N-s^2/m⁸), K is friction factor in lb-min²/ft⁴ (kg/m³), L is length of airway in ft (m), L_e is equivalent length to account for shock conditions in the airway in ft (m), O is perimeter of the airway in ft (m), and A is cross sectional area of the airway in ft² (m²).

Table 11.7.2.1 shows the calculation of mine resistance for each segment, and Table 11.7.2.2 shows the calculation of head loss for each segment. The head loss in the circuit is calculated as 5.68 in. water (1.413 kPa). Adding about 0.32 in. water pressure (79.63 Pa) for losses across the face in 15-N on a line brattice, the fan must develop a static head of nearly 6 in. water (1.493 kPa).

FAN SELECTION. A fan that can develop static head of 6 in. (1.493 kPa) water and 650,000 cfm (306.77 m³/s) is needed, and it must be able to handle this quantity with high efficiency. The fan motor selected to provide adequate horsepower for the total mine/fan head, and the fan must operate in an adequate range of capacities since it must be operated during the life of the mine during which time the mine requirements for quantity and head will increase.

Using the Jeffrey fan selection chart (see Fig. 11.6.22, Chapter 11.6), an initial selection of a fan can be made. It can be seen that a 8HU117 fan operating at 710 to 880 rpm can provide the required head and quantity. However, the efficiency of this fan at this high quantity is rather low. To increase the efficiency, a possible combination may be two 8HU117 fans operating in parallel at 710 rpm, each producing 350,000 cfm (165.18 m³/s) at 6 in. (1.493 kPa) static head. The operating point of each fan is nearly at the middle of its quantity capacity (≅ 750,000 cfm, 353.96 m³/s), and the head is within the range of 2 to 10 in. water (0.498 to 2.488 kPa) . At this operating point, the fan efficiency is well above 80%. The horsepower of the motor (from the curve) at 70% efficiency is 500 (373 kW). At 80% efficiency, the fan motor will be sized at 440 hp (328 kW). The static air horsepower at the operating point is given by $\frac{6 \times 350{,}000}{6350} \cong 331$ (247 kW), resulting in an overall static efficiency of approximately 75% [= (331/440) × 100].

Yet another method for initial selection can be the use of charts of specific speeds and specific volumes. Assume a specific volume Qs = 2000 cfm (0.94 m^3/s); then from Eq. 11.6.44 (Chapter 11.6),

$$D^2 = \frac{Q}{Q_s\sqrt{H}} = \frac{650{,}000}{2000 \times \sqrt{6.0}} = 132.68 \text{ ft}^2 \ (12.33 \text{ m}^2)$$

$$D \cong 11.52 \text{ ft } (3.35 \text{ m})$$

Choose a fan 12 ft (3.66 m) diameter. The specific volume for a 12-ft (3.66-m) fan is

$$Q_S = \frac{650{,}000}{(12 \times 12)\sqrt{6.0}} = 1850 \text{ cfm } (0.87 \text{ m}^3/\text{s})$$

Enter chart (Fig. 11.6.20, Chapter 11.6) at 1850 cfm (0.87 m^3/s), and note that several blade settings (#1, #4, #7, and 2B-1S) have efficiencies approaching 80% at $Q_s \cong 1850$ cfm (0.87 m^3/s). The specific speed n_s for the (2B-1S) blade setting is approximately 2500, and for the #1 blade setting is approximately 3000. Choosing the lower specific speed, the fan speed can be calculated from Eq. 11.6.43 (Chapter 11.6) as

$$n = \frac{n_s\sqrt{H}}{D} = \frac{2500 \times \sqrt{6}}{12} = 510 \text{ rpm}$$

$$P_m = \frac{650{,}000 \times 6}{6350 \times 0.8} \cong 768 \text{ bhp } (573 \text{ kW})$$

A 12-ft (3.66-m) diameter fan connected to a 768-hp (573-kW) motor operating at 510 rpm may be an initial choice.

For a lower Q_s, say, Q_s = 1500 cfm (0.71 m^3/s),

$$D^2 = \frac{650{,}000}{1500 \times \sqrt{6.0}} = 176.91 \text{ ft}^2 \ (16.43 \text{ m}^2)$$

$$D = 13.50 \text{ ft} \cong 14 \text{ ft } (4.27 \text{ m})$$

This is a very large-diameter fan. Assuming that two smaller fans will be placed in parallel, to divide the quantity between them equally, the diameter of the fan for the same specific volume (i.e., Q_s = 1500, or 0.71 m^3/s),

$$D^2 = \frac{325{,}000}{1500 \times \sqrt{6.0}} = 88.45 \text{ ft}^2 \ (8.22 \text{ m}^2)$$

$$D = 9.3 \text{ ft} \cong 10 \text{ ft } (3.05 \text{ m})$$

Assume two 10-ft fans in parallel:

$$Q_s = \frac{325{,}000}{100 \times \sqrt{6.0}} = 1327 \cong 1400 \text{ cfm } (0.66 \text{ m}^3/\text{s})$$

Enter chart (Fig. 11.6.20) at 1400 cfm (0.66 m^3/s) and note that peak efficiency (80%) occurs at blade setting No. 1 and that specific speed is 2800.

$$n = \frac{2800 \times \sqrt{6}}{10} = 685.86 \cong 700 \text{ rpm}$$

$$P_m = \frac{325{,}000 \times 6.0}{6350 \times 0.80} = 384 \text{ bhp } (286.35 \text{ kW})$$

Thus two 10-ft (3.05-m) diameter fans in parallel, each operating at 700 rpm and connected to a 384-hp (286.35-kW) motor, may be another choice.

Each of these alternatives is associated with quantifiable factors such as capital and operating costs, and non-quantifiable but important qualitative factors such as flexibility, availability, standardization, and future mine extensions. The actual selection of a fan will be the result of the judgmental evaluation by the engineer of these factors. Economics in mine ventilation costs must be accomplished by reducing the static and total heads of the mine. The reductions are achieved by having straight, clean, and unobstructed airways and, wherever possible, a number of airways in parallel; by reducing the leakage through stoppings, overcasts, and airlocks; by selecting and operating fans with high efficiency for the desired mine quantity and head. In addition to the extreme care in the design of the system, frequent mine ventilation surveys and regular maintenance of the components of the ventilation system—airways, control devices, and fans—are required to assure continued safe performance of the ventilation system. The modern automatic monitoring techniques—for gases, quantities, pressures, temperatures, power, etc.—can be effectively applied to the ventilation system to aid managers and engineers in maintaining the integrity of the health and safety aspects of the system (Ramani, 1982) (also see Chapter 12.6).

The example just developed is for a relatively small mine. In large mines, it is impractical to take in all the air through one shaft and return it through another, and so a multiple-fan-and-shaft arrangement becomes necessary. In such a situation, each fan has a zone of influence, and all the zones are interconnected. Difficult ventilation problems may arise at zone interfaces where there may be no airflow (i.e., neutral zones) or air directions may be the reverse of those desired. The areas of influence of each fan vary as a complex function of such factors as the advance rates of the interconnected zones, distances from the main intakes, and the fans and their blade settings (Mishra, Ramani and Wang, 1978).

Mine ventilation planning must include provisions for future expansion and exigencies that must be incorporated into the initial design. When planning with computer-oriented mine ventilation network analysis programs, it is possible to try out various alternatives with regard to factors such as fans, shafts, and mains (Wang, 1982; Ramani, 1988). Stefanko and Ramani (1972) present the application of a mine ventilation network analysis program to the ventilation planning example discussed in this chapter. The complex calculations involved with multiple-fan, multiple-shaft ventilation systems are best carried out on a mine ventilation network analysis digital computer program (see Chapter 11.10).

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Chapter 11.8
NOISE ABATEMENT

LEONARD MARRACCINI

11.8.1 INTRODUCTION

In the United States, mining activities began in the 1700s. This was strictly manual, labor-intensive type of mining. By the late 1800s, mechanization began to replace manual labor. Today mechanization in the mining area involves state-of-the-art technology, including lasers, computer-operated equipment, etc. However, along with mechanization has come one byproduct—the generation of *noise.* Various types of mining equipment generate noise levels for different lengths of time, depending upon the mining operation. This results in employee noise exposure. Many times, this exposure is in excess of the federal mining noise regulations. Over exposure to excessive noise levels has resulted in hearing loss.

Depending on the situation, noise can be a significant problem, particularly in view of the large number of mines and miners. In the beginning of 1988, there were 17,778 mines in operation in the United States. Of this total, 6276 were coal mines, and 11,502 were noncoal (metal or nonmetal) mines. A total of 330,871 people were employed in the industry, 145,096 in coal mining, and 185,775 in noncoal mining.

Due to the high degree of mechanization involved in the industry, and the number of operations and people employed, noise is one problem that must be addressed. Throughout this chapter, various concepts will be presented regarding noise and noise abatement. Some of the key terminology and definitions that are used repeatedly are listed as follows:

1. Noise—unwanted sound.
2. dBA noise level—the overall noise level, measured in decibels, A-weighted.
3. Frequency distribution—the range of frequencies that compose the noise spectrum, typically in the audible range of 20 to 16,000 Hz.
4. Acoustical materials—any material specifically designed for or used to reduce the noise levels.
5. Retrofit—to modify existing equipment that is already in service.
6. Noise dose—the amount of noise exposure that an employee is exposed to for a given amount of time.

This chapter briefly summarizes mining noise regulations, noise measurement instrumentation, survey procedures, and noise abatement techniques.

11.8.2 NOISE REGULATIONS

Regulations dealing with noise exposure in the mining industry fall into two categories, state, and federal. Essentially, the various states that have mining noise regulations refer to or mirror the federal regulations. Thus this discussion focuses on the federal regulations. The Mine Safety and Health Administration (MSHA) has jurisdiction of the enforcement activities for the mining industry. The Federal Coal Mine Safety and Health Act of 1969 contained the first noise regulations pertaining to mining in the United States. This Act prescribed noise levels as stated in the Walsh-Healy Public Contracts Act. The Walsh-Healy criteria for noise are shown in Table 11.8.1.

Table 11.8.1. Walsh-Healy Noise Criteria

Maximum Noise Level, dBA	Maximum Exposure Time, min.
90	480
95	240
100	120
105	60
110	30
115	15

No noise level shall exceed 115 dBA for any period of time.
Source: Anon., 1989.

Basically, these criteria involve exposure to specific levels of noise for specific time intervals. For each 5-dBA increase in the noise level, the maximum exposure time is halved.

The Occupational Safety and Health Administration (OSHA) standards differ from MSHA limits by assigning 85 dBA for 8 hours for hearing conservation purposes. For enforcement purposes, OSHA also uses 90 dBA for 8 hours.

The Federal Metal and Nonmetallic Mine Safety Act of 1974 incorporated the same Walsh-Healy criteria. Several years later, the Federal Mine Safety and Health Act of 1977 amended the Federal Coal Mine Safety and Health Act of 1969 to include all mines, coal and metal and nonmetal. The 1977 Act also repealed the Federal Metal and Nonmetallic Mine Safety Act of 1974.

The current regulations concerning noise in the mining industry can be found in Title 30, *Code of Federal Regulations* (CFR), Chapter 1 (Anon., 1989). In the Code, regulations pertaining to noise in metal and nonmetal facilities can be found in Subchapter N, Part 56, Subpart D (surface operations), and Part 57, Subpart D (underground operations). Regulations pertaining to noise in coal facilities are found in Subchapter O, Part 70, Subpart F (underground operations), and Part 71, Subpart I (surface operations). These regulations still contain the same criteria as in the Walsh-Healy Act. The regulations also specify noise measurement instrumentation, instrument calibration, measurement procedure, etc.

11.8.3 NOISE INSTRUMENTATION/SURVEY PROCEDURES

Before undertaking any noise abatement work, the trained investigator should utilize appropriate instrumentation to make various noise measurements. A wide range of instrumentation is commercially available to measure noise. The specific type of noise instrumentation chosen depends upon what type of data is required. All instrumentation, regardless of its usage, does have to meet specific standards, primarily those of the Acoustical Society of America (ASA) or American National Standards Institute (ANSI). Additionally, when noise measurement instrumentation is used in an underground gassy mine, the instrumentation must also meet the MSHA permissibility requirements. The permissibility requirement essentially safeguards against the ignition of methane gas by electrical equipment.

Fig. 11.8.1. Sound level meter.

Fig. 11.8.2. Noise dosimeter.

Some of the more common types of noise instrumentation and their associated functions are described as follows. One of the most basic instruments is the *sound level meter* (SLM), which is used to measure the overall noise level. With various electronic weighting networks built into the instrument, the SLM can respond to noise in the same way the human ear responds (A-weighting), or it can have an almost flat or linear response (C-weighting). A typical sound level meter is shown in Fig. 11.8.1.[1]

Another instrument frequently used is the *personal noise dosimeter*. This instrument, worn by the employee, measures the noise exposure for an employee and cumulates this information for up to 8 hours. This noise dose is based upon either OSHA, MSHA, or other regulatory specifications. For an example of this device, see Fig. 11.8.2.

The sound level meter and personal noise dosimeter are instruments that yield information primarily used in noise regulation work. However, to obtain specific acoustical engineering data, other types of instrumentation are required. One is a *portable noise analyzer* that measures the frequency components of the noise in either octave or third-octave bands (Fig. 11.8.3).

Another instrumentation system is a *precision tape recorder and microphone system* (Fig. 11.8.4). This system is used to tape record the noise in field situations. The tape recorded data can then be returned to the laboratory where it is analyzed by means of an analyzer, either octave band, third-octave band, narrow band, or even fast fourier transform analyzer.

By using the data from the portable noise analyzer or the analyzed tape data, the investigator can develop a graph illustrating the frequency distribution of the noise levels. As stated before, this is an important piece of engineering data needed for noise control work. An example of a graph illustrating the frequency distribution of noise is shown in Fig. 11.8.5.

Thus, depending upon the type of noise data required, various types of instrumentation are available to conduct a noise survey.

Noise surveys can have a wide range of descriptors; however, they basically are considered as either a health survey or an engineering survey. Although the bottom line of the work is to protect the employees' hearing, there are distinct differences between the health and engineering survey. The health survey is conducted to determine if an employee receives an overexposure to noise. The survey involves the use of either a sound level meter or personal noise dosimeter. Regardless of the instrumentation used, this type of survey requires that the investigator follow the employee for the entire shift, using a stop watch to accurately note work duties and locations. These times are then correlated with the starting time of the employee and the instrumentation measurements. Thus, at the end of the shift, the investigator can determine the employee's noise exposure on an overall basis, and more importantly, on a work duty or work location basis.

The engineering survey is conducted to determine what mechanical component, or components, on a piece of noisy mine equipment is creating the problem. The investigator already knows that the mine equipment is the source of the employee's overexposure to noise. It is his job then to determine which component, or components, on the equipment need to be treated with acoustical materials, etc. To do this, the investigator obtains data using either a microphone-tape recorder system or portable noise analyzer. Diagnostic tests are conducted on the various machine components, or operating modes, of the machine. The resulting noise spectra (frequency distribution of the noise) can be used to pinpoint the machine component or operating mode that is causing the problem. Engineering noise control techniques can then be instituted.

[1] Reference to a product name or illustration of a product does not constitute endorsement by MSHA.

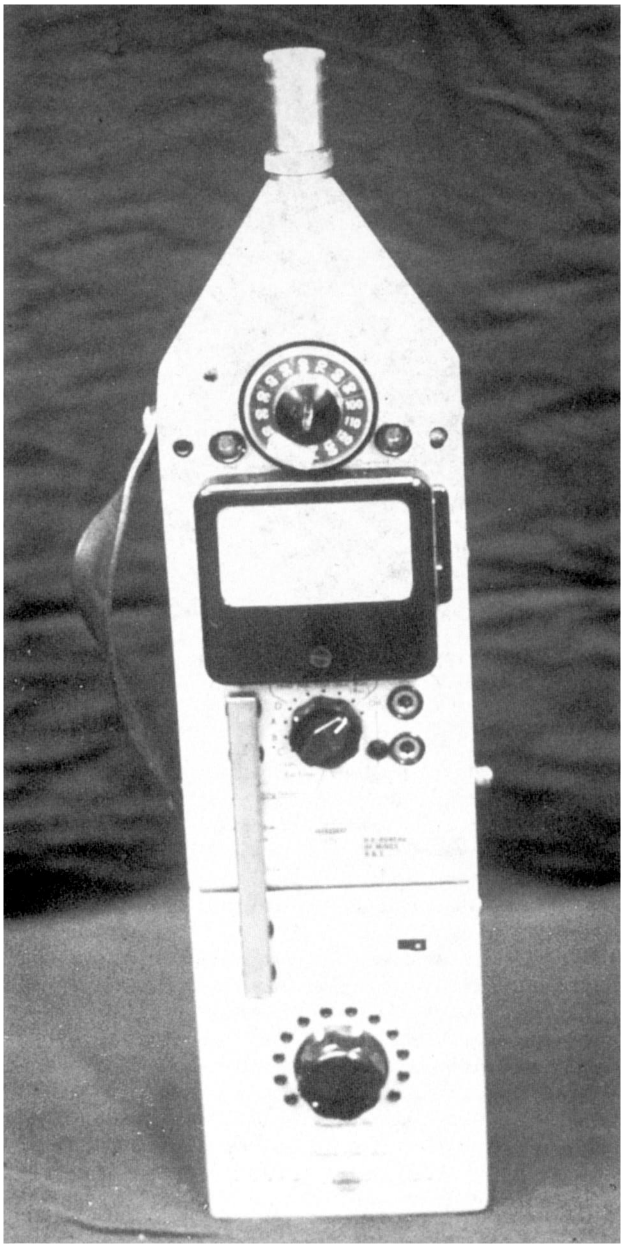

Fig. 11.8.3. Portable noise analyzer.

11.8.4 BASIC NOISE ABATEMENT

When noise abatement work is anticipated, the investigator has to decide the most appropriate approach. In classical noise abatement, this involves either the treatment of the noise *source,* interrupting the *path* or route the noise travels, or isolating the *receiver* or employee who is overexposed to the noise. With regard to each of these approaches, various questions must be answered such as the anticipated length of time the mine equipment will be out of production, the economics of the approach, and whether the equipment will be retained or shortly phased out.

In deciding the most appropriate approach to noise abatement, the mine operator may be confronted with certain situations that may involve a newly purchased or soon-to-be purchased piece of equipment. In the best interest of the mining company, it has been shown that noise controls designed and installed at the manufacturing site are more effective than noise controls installed at the mine site as a retrofit approach after the equipment has been purchased. Also many mining companies have begun to specify in the purchase agreements that the new equipment meet current noise regulations. This puts the burden of effective noise control on the manufacturer as opposed to the mine operator.

If the mine operator decides that the noise abatement work will be done at the mine site as a retrofit program, he must realize that the most important factors in achieving maximum noise reduction are the selection of the *proper* acoustical materials and the quality of the installation. This has been proven time after time. The selection of acoustical materials has to be tied very closely with the frequency range of the noise involved. If, for example, the noise is predominately high frequency, then the acoustical material chosen should be the one that has characteristics that operate most effectively at the same high frequency range. Similarly, if the noise is predominantly mid-frequency, then the acoustical material chosen should be most effective in the mid-frequency range. With the current cost of acoustical materials and the large number of materials available, it is senseless to have an “any material will do” attitude.

Acoustical materials, in addition to the frequency characteristics, can be classified into three groups. Each group performs somewhat differently. *Absorptive materials,* composed of a lightweight open cell or fiber structure, absorb sound energy, similar to a sponge absorbing water. Some common examples of absorption materials include fiberglass, various foams, and mineral wool. *Transmission-loss (barrier) material,* being a high-density material, actually reflects sound energy, thus preventing the sound from passing through it. Some common examples of transmission-loss material include brick, lead, and plywood. Finally, the *composite materials* are composed of a high-density material sandwiched between two layers of absorptive material. Fabricated in this manner, the composite material has both absorptive and transmission-loss properties. There is one other category of materials, not commonly thought of as acoustical material, that deals with vibration isolation and damping. These vibration materials are used to prevent one vibrating component from transmitting vibrational energy to another component by isolation, or to dampen a vibrating plate. As mentioned previously, each category of material has different properties and should be used accordingly.

One other item should be discussed regarding any type of acoustical or vibrational material. This is the flammability of the material. At the present time, MSHA does not have regulations pertaining to flammability of acoustical materials. However, it does have guidelines. These guidelines refer to the flame-spread index as measured by either ASTM E-162 or E-84 test standards. The investigator and mine operator should be very aware of the flammability properties of the acoustical materials that are intended for use. This is particularly important during any maintenance work where burning, welding, or any open flame is present near the installed acoustical materials.

After the proper acoustical materials have been selected, one more matter needs to be addressed for the most effective noise reduction to occur. This last item involves the quality of the installation, and it can be summarized as follows: *Regardless of how good the acoustical material is, maximum noise reduction will not be achieved by shoddy installation practices.* The proper installation of these acoustical materials requires forethought, planning, and last but not least, patience. To rush through an installation almost always results in unsealed holes or cracks,

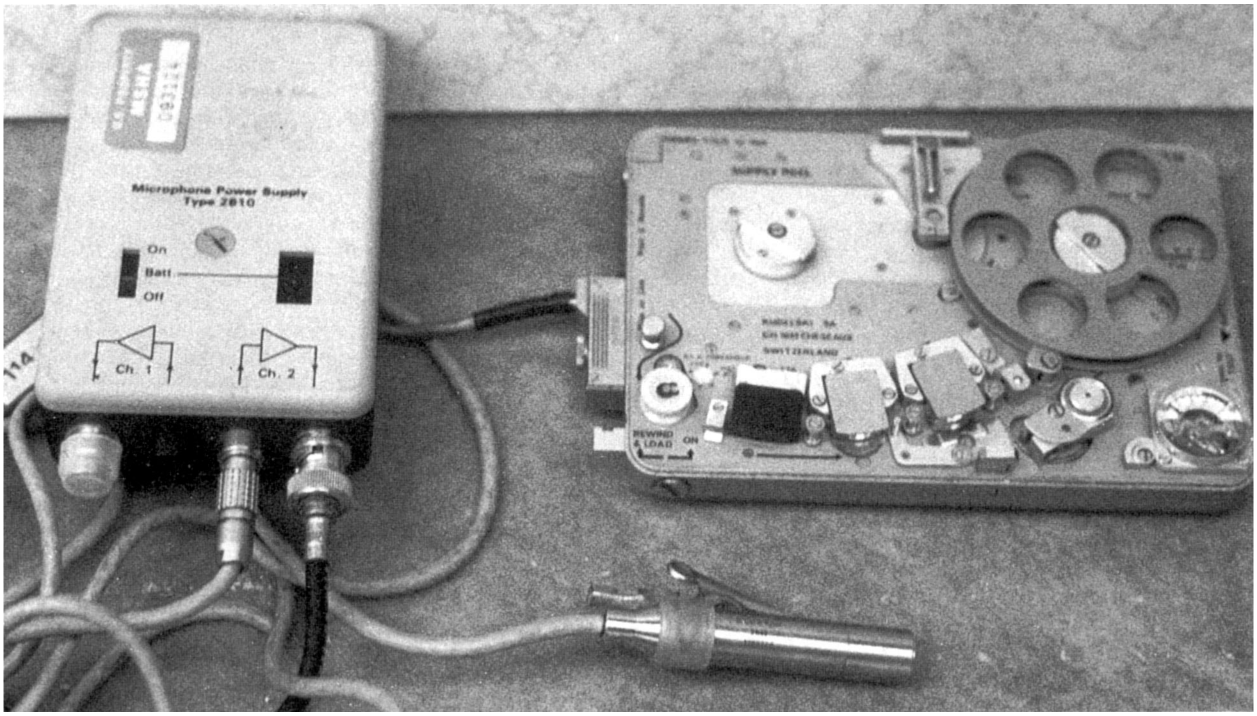

Fig. 11.8.4. Tape recorder and microphone system.

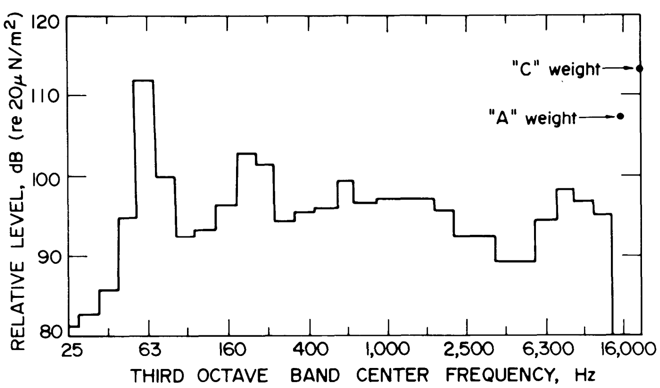

Fig. 11.8.5. Graph of frequency distribution of noise.

poorly fitted materials, etc. The end result is that the maximum noise reduction will not be achieved regardless of the quality of the acoustical materials used.

11.8.5 EXAMPLES OF NOISE CONTROL

Space requirements do not permit an in-depth discussion of noise controls for the wide range of mining equipment in use today. Instead, the discussion here is limited to "generic" machine types that can be characterized as mobile surface and underground equipment and preparation plants. It must be emphasized that these noise controls are general in content.

11.8.5.1 Mobile Surface Equipment

Most mobile surface equipment, such as bulldozers, front-end loaders, haulage trucks, etc., are diesel-powered. As such, the noise controls associated with this type of equipment are very similar, particularly from a retrofit aspect. Typical noise sources may include the diesel engine, exhaust, drive train, and torque converter. One of the most common techniques for noise control involves the installation of an exhaust muffler (properly matched to the engine specifications) to reduce the exhaust noise, and the isolation of the operator from various noise sources by installing acoustical materials to the inside surfaces of the operator's cab.

On equipment such as cranes, shovels, and draglines, a common noise control technique involves the installation of a barrier (flexible or rigid) between the operator and the engine area. This is done to prevent engine and gear noise from reaching the operator. In addition to this barrier, acoustical materials are installed in the area around the operator to absorb any noise that leaks through the barrier.

With most mobile equipment types, one of the more favorable aspects of retrofit noise control work is the availability of the equipment. The equipment can be taken out of production and relocated if needed to a shop area or adjacent to a shop area for the installation of the noise controls.

11.8.5.2 Underground Equipment

The task of performing retrofit noise control work on underground mining equipment is by far the most difficult in comparison to mobile surface equipment or preparation plant. One of the reasons for this is that in many situations, once the equipment is placed underground, it does not return to the surface unless it is scheduled for overhaul or rebuild. This time period may be as long as four or five years. However, in some cases, underground shops are available, thus making retrofit noise-control installation easier.

Many equipment types operating in the working face area incorporate cutting heads to extract the material and chain conveyors to move the mined material. Such equipment includes auger miners, continuous drum miners, and longwall shearers. Typically, the cutting head and chain conveyor system are two of the primary noise sources. Vibration isolation and vibration damping techniques have proven effective in reducing noise levels. Specially designed cutting heads incorporating vibration damping materials have been effective. Constraint-layer damping of the conveyor pans has also been effective.

In some of the larger pieces of equipment, such as drill jumbos, an acoustically treated operator's cab has been used successfully. Where several employees are working in close proximity to a noisy piece of equipment, treatment of the various sources or troublesome components is important because of reverberant conditions found in underground mines.

11.8.5.3 Preparation Plants

Noise control work in a mineral preparation plant or processing operation can present some very challenging situations. This can be due to the potentially large number of noise sources and the movement of people throughout the operation. For a plant or systems operator, the utilization of an acoustically treated booth is essential. In some situations, where an operator periodically walks a beltline or observes material levels in bins, he may feel that he must leave the treated booth. However, this need not be the case because economical video systems can be used along the belt lines or near the bins. The operator can remain in the treated booth and observe the operations on the video screens. Also numerous bin level indicators are commercially available that can also be incorporated with the booth. For employees who must move from one location to another throughout the day, small acoustically treated booths can be placed at various locations to create "quiet" areas where the employee can go for brief periods of time. Other noise controls available include chute liners, bin liners, treated screen decking, impact pads, and partial barrier curtains.

Regardless of the situation, be it either mobile surface equipment, underground equipment, or preparation plant equipment, a wide range of noise controls is available to reduce employee noise exposure.

11.8.6 PERSONAL PROTECTIVE DEVICES (HEARING PROTECTION)

When engineering noise controls and/or administrative controls such as changing work routines fail to bring employee noise exposures into compliance with regulations, personal hearing protection can be utilized. These devices are commercially available in a wide range of styles, models, and configurations. These include earplug devices, earmuff devices, and cap-mounted earmuff devices. All of these should be tested according to the latest ANSI standard. This standard is used to evaluate hearing protectors under optimum (laboratory) conditions. From these test data, a noise reduction rating (NRR) value is calculated. This rating value can give an estimation of the expected effectiveness of the devices. However, numerous organizations have conducted research that indicates that when the hearing protector is used in an actual work environment, the effectiveness of the device can be significantly less than that measured in the laboratory. There is a wide range of variables that can contribute to this difference, such as the various noise spectra of the equipment encountered, poor fit on the individual, use of other safety equipment such as safety glasses and hard hats, improper maintenance of the hearing protector, etc. Thus, individuals should not consider hearing protector devices as the complete "cure-all" from noise exposure. If the employee must depend upon hearing protectors, then that person should be properly trained, and be made aware of the limitations of the hearing protectors.

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Chapter 11.9
MINE ILLUMINATION

KENNETH WHITEHEAD AND GEORGE R. BOCKOSH

11.9.1 INTRODUCTION

The mining environment poses unique challenges to the illuminating engineer. It is unlike most other industrial settings, in that conditions cannot be designed but rather must be accommodated. Specifically, room dimensions and surface reflectance in underground mines are defined by the deposit mined and can only be modified at great expense. A potentially explosive atmosphere, dust, and the ever-changing nature of the underground environment further complicate the illumination task.

In surface mining operations, vast areas are active 24 hours a day, and large equipment confronts operators with poor visibility. In addition, reflectivity, weather, and task variations adversely affect visual performance and impose difficult demands on illumination system design.

It is these predetermined, and often adverse, conditions that cause extreme difficulty in providing adequate *illumination in mines.* The concerns that need to be addressed include adequate illumination for hazard and operational cue recognition, the quality of lighting, and health maintenance.

11.9.2 LIGHT PHYSICS

Two major systems of units are currently used for the quantification of light: Illumination Engineering Society (IES) and International System of Units (SI). The primary difference between IES and SI systems is that the IES system uses US standard measures for linear dimensions in the unit definitions while the SI system uses metric measures. Current US coal mine lighting regulations customarily use IES units; therefore, these will be adopted primarily in this chapter.

Systems of lighting units are unique in that they explicitly apply a human weighting function to the physical energy quantity they measure. That is, all unit systems take into account how the eye exhibits different sensitivities to various light wavelengths in terms of perceived brightness and weight the energy measurements according to the spectral luminous efficiency curve (Fig. 11.9.1).

All standard systems of light units employ certain fundamental concepts that are based on convenient and meaningful approaches to light energy measurement and quantification. These basic concepts are luminous flux, illumination (illuminance), luminous intensity, and luminance.

11.9.2.1 Luminous Flux

The luminous flux symbol is ϕ, and the lumen (lm) is the flux unit used in both the IES and the SI systems.

Luminous flux is the time flow rate of light energy. Flux is a power quantity in the same manner as horsepower or Btu per hour. The unit of luminous flux, the lumen, is most frequently used to describe the total lighting power of light sources. Other light energy concepts (e.g., illumination, luminous intensity, and luminance) use the lumen in conjunction with various geometric quantities to describe the distribution of light energy flow to the surroundings. Light sources are often evaluated for their total lumen output. For example, a 100-W incandescent lamp produces about 1740 lm.

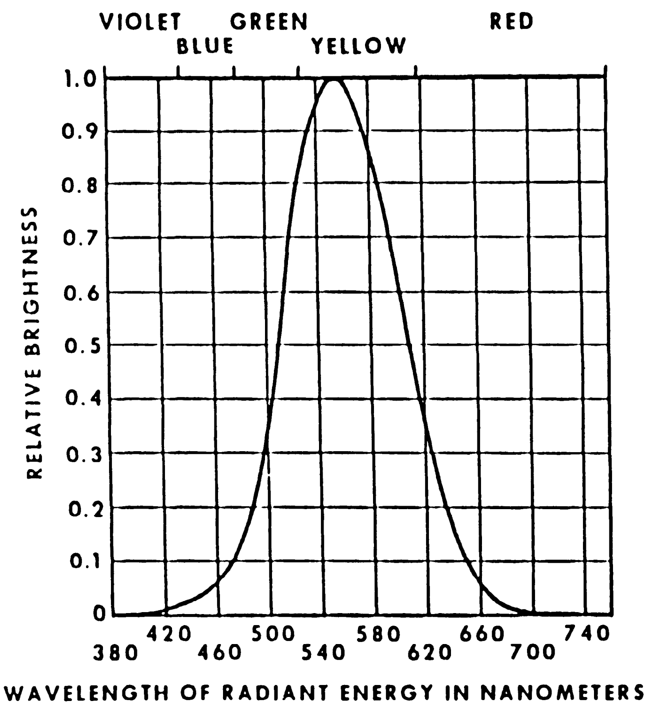

Fig. 11.9.1. Brightness sensitivity of the eye as a function of light wavelength.

Total lumen ratings of light sources, usually available from hardware manufacturers, are valuable for use in making preliminary approximations in lighting design problems. The number of lumens needed in a given situation will help determine the size and/or number of lighting fixtures necessary.

11.9.2.2 Illuminance

Illumination (illuminance) E is a measure of the density of luminous flux striking a surface. Its units are the footcandle (fc) and lumen/ft^2 in the IES system, and lux (lx) and lumen/m^2 in the SI system. Mathematically, illumination may be defined as

$$E = \phi/A_R, \tag{11.9.1}$$

where E is illumination produced by the luminous flux ϕ falling on a light-receiving surface of area A_R (Fig. 11.9.2).

As an example of the illumination concept, if light energy flowing at the rate of 9 lm is distributed over an area A of 1 ft^2, the illumination of area A is 9 fc. If the same energy is subsequently distributed over a larger area B of 3 ft^2, then the average illumination of area B is 3 fc, one-third as great as area A. The total illumination levels from two or more light sources shining on a surface are obtained by adding the illumination level produced by each source separately.

When determining illumination, ϕ in Eq. 11.9.1 concerns the total lumens striking the receiving surface area, regardless

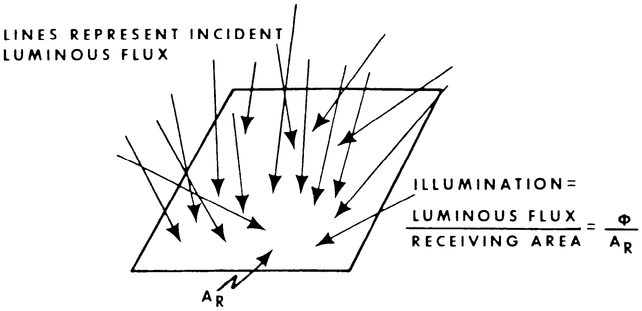

Fig. 11.9.2. Illumination.

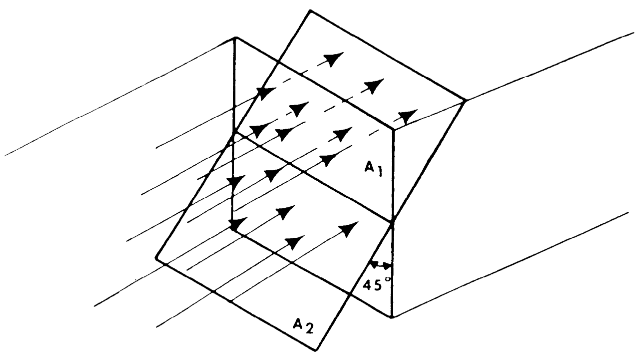

Fig. 11.9.3. Basis of the cosine law.

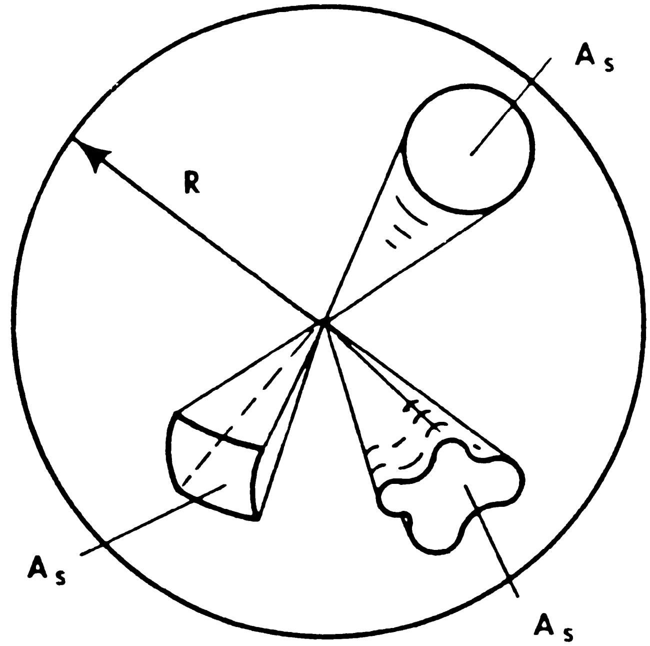

Fig. 11.9.4. Solid angle forms.

of the originating direction of the luminous flux and represents an average value. The illumination at any point on the receiving surface is determined by considering a very small area around the point as the receiving area. Point illumination for all points on a surface equals *average* illumination if the luminous flux is uniformly distributed over a surface. If the illumination is nonuniformly distributed, point illumination E_P would vary for each point on the surface.

Average illumination is an easy quantity to measure. The significance of the fact that illumination is an easy quantity to measure is that illumination measurements can be converted to other light quantities (e.g., luminous intensity), which are meaningful but more difficult to measure directly. Lighting design specifications, which are often presented in terms of illumination (footcandle or lux) levels, do not consider how a receiving surface reflects light. This is a definite shortcoming because reflected light determines what is seen. Various properties of light-receiving surfaces can affect reflectivity.

11.9.2.3 Illumination and the Cosine Law

The *cosine law* and the *inverse square law* (discussed later) are two very useful lighting laws. The cosine law states that the illumination of a surface varies as the cosine of the angle between an imaginary perpendicular line to the surface (i.e., the normal) and the actual direction of the incident light. To illustrate the cosine law, imagine a light beam consisting of uniformly distributed parallel rays traveling in a particular direction. Assume that 10 lm is incident upon surface A, with a surface area of 1 ft^2 in Fig. 11.9.3. Note that surface A is perpendicular (normal) to the direction in which the light is traveling. The average illumination of this surface could be calculated according to the discussion in the previous segment as follows: $E_1 = \phi/A$, or 10 lm divided by 1 ft^2 equals 10 fc.

Now imagine that instead of intersecting a surface perpendicular to the light's direction of travel, this same 10-lm beam intersects a surface rotated 45°, as surface A, is in Fig. 11.9.3. The illumination is now uniformly distributed over a larger area (A_2), which is equal to A_1 divided by the cosine of q.

Therefore, $E_2 = \phi/A_2$, which equals $\phi/A_1 X \cos q$ – or $E_1 X \cos q$. If q is 45°, $E_2 = E_1 \cos 45° = 10\ (0.5) = 5$ fc.

Although the cosine law derived was for average illumination, the law also applies equally to point illumination.

11.9.2.4 Luminous Intensity

Luminous intensity I is a concept used to describe how a light source (e.g., a lamp or luminaire) distributes the total luminous flux, or lumens, it emits into various portions of the space surrounding the source. Both the IES and SI unit is the candela c. The geometric concept of the *solid angle* is used to define the particular portion of surrounding space in question. Before luminous intensity is discussed, the geometric concept of solid angle must be explained.

A solid angle is simply a three-dimensional angle. It is formed by a point at the center of a sphere and a surface, of any shape, that comprises a part of the surface of the sphere. Fig. 11.9.4 illustrates various forms of solid angles.

The dimension of a solid angle w is determinated by finding the area A of the sphere subtended, or enclosed, by the angle divided by the radius R of the sphere squared. Hence,

$$w = A_s/R^2 \qquad (11.9.2)$$

The unit of the solid angle is the steradian (sr). A subtended surface area of 1 ft^2 on a sphere of l-ft radius forms a solid angle of 1 sr. Note that the spherical area subtended by a solid angle varies with the radius squared.

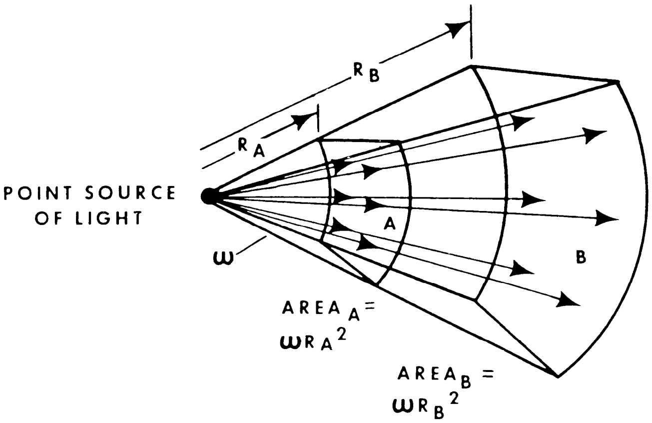

Fig. 11.9.5. Basis of the inverse square law.

Luminous intensity is defined mathematically as follows:

$$I = \phi/w \tag{11.9.3}$$

where I is equal to the luminous intensity of a point source of light in a direction defined by a particular solid angle, f is the total luminous flux (lumens) emanating from the point source within the specified solid angle, and w is the dimension of that solid angle in steradians. The candela c is the luminous intensity unit (both the SI and IES systems) and equals 1 lm/sr.

Intensity is a directional quantity, with the direction in question being defined by the line that forms the axis of the solid angle. The intensity of the light source may, indeed, vary with direction. Intensity, as calculated from Eq. 11.9.3, is an average intensity over the entire solid angle w. As w is subdivided into smaller and smaller solid angles, the direction of the intensity is better defined, and the distribution of light from the source is more accurately established.

11.9.2.5 Inverse Square Law

A common problem in lighting system design is determining the illumination on surfaces at various distances from a light source. This problem can be handled using the inverse square law.

Given the intensity of the light source depicted in Fig. 11.9.5 in the direction defined by the illustrated solid angle, the flow of luminous flux within that solid angle can be calculated using Eq. 11.9.3. The illumination of the depicted surface subtended by the solid angle would be the flux within the solid angle divided by the area subtended, as defined in Eq. 11.9.1. Since the flux is the same in both equations, we can solve for the flux and set the resulting expressions equal:

$$\phi = Iw\phi = EA_s \tag{11.9.4}$$

Solving for illumination gives

$$E = \frac{Iw}{A_s} \tag{11.9.5}$$

Since by definition, $w = A_s/R^2$, substituting this in the illumination equation gives

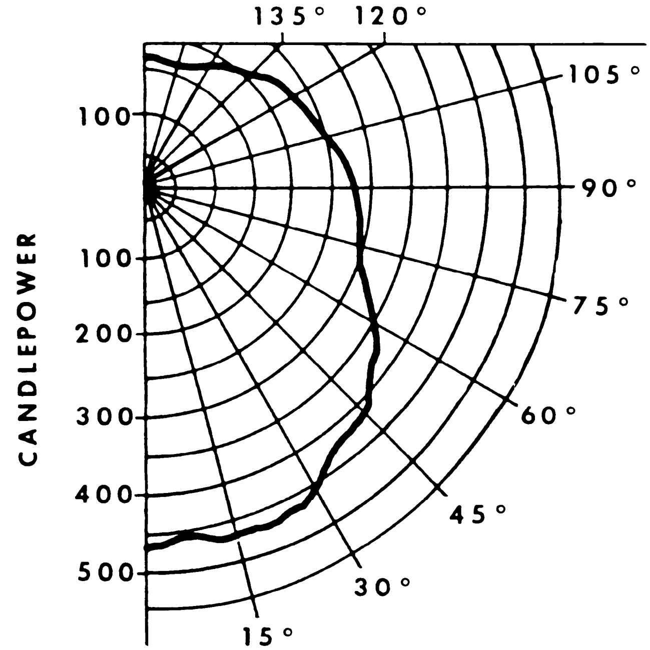

Fig. 11.9.6. Candlepower curve for a typical incandescent fixture. Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

$$E = \frac{IA_s}{A_sR^2} = \frac{I}{R^2} \tag{11.9.6}$$

This equation relating illumination, intensity, and the distance between the source and light-receiving surface is known as the inverse square law. It enables illumination of a surface to be calculated if the intensity of the light source and the distance between the light source and the surface are known.

In general practice, it is common to calculate the illumination at a point on a surface rather than an area on a sphere. The inverse square law can then be modified to use the distance D between the light source and the point rather than the radius R of the sphere. Eq. 11.9.6 then becomes:

$$E = I/D^2 \tag{11.9.7}$$

The inverse square law assumes a point source of light. Most real light sources are not point sources, however. Nevertheless, the law can be applied, with negligible error, if the distance between the light source and the illuminated area is greater than five times the maximum dimension of the light source. Consequently, using the law is practical for most purposes encountered in lighting design, except where long, tubular light sources (e.g., fluorescent lamps) are used.

A second assumption inherent in the inverse square law is that the surface area is perpendicular to the direction of light flow. When this is not the case, the inverse square law can be combined with the cosine law as follows:

$$E = E_{\text{normal}} \times \cos\theta = \frac{I\cos\theta}{D^2} \tag{11.9.8}$$

11.9.2.6 Candlepower Curves and Their Uses

The most common way of presenting lighting data on luminaries and lamps is by the candlepower curve or some variation of this curve. A candlepower curve is a plot of the intensities of a light source in a particular plane, at all angles around it. Fig. 11.9.6 illustrates a candlepower curve for a household incandes-

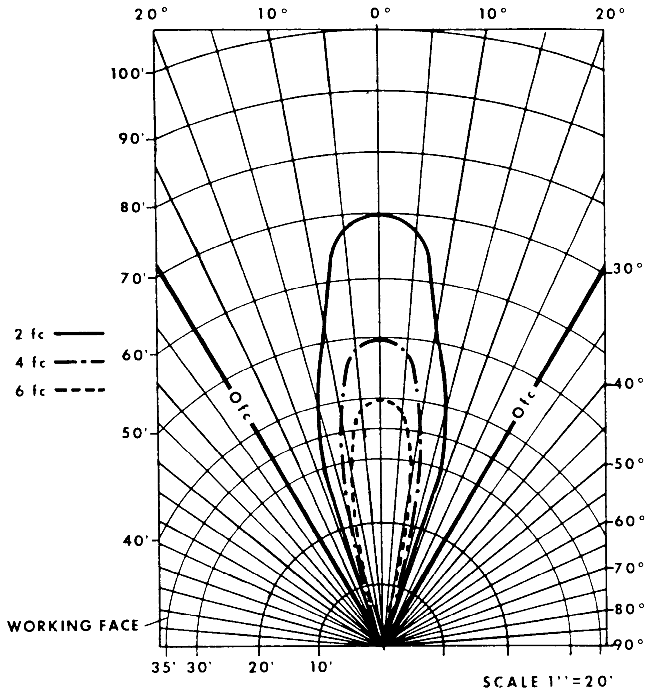

Fig. 11.9.7. Isofootcandle curve for a mining vehicle headlight.

cent fixture. As shown, these curves are usually plotted using polar coordinates.

Candlepower curves are extremely valuable for various design calculations. They depict how a light fixture distributes its total luminous flux into the surrounding space. Also, because of the relationship between illumination and intensity—the inverse square law—these curves may be viewed as depicting a light fixture's ability to illuminate in various directions. The higher the intensity in a given direction, the greater the ability of that source to illuminate surfaces in that direction. The curves are obtained by making *illumination* (fc) measurements at various orientations with respect to the lighting fixture and utilizing the inverse square law to calculate the intensity at that orientation.

A candlepower curve applies only to the intensities in a single plane that passes through the light fixture. To fully describe a fixture's light distribution, several candlepower curves in different planes may be necessary, depending on the symmetry of the light distribution.

One of the primary uses of candlepower curves is to utilize these data to calculate the surface illumination that the fixture would provide, given the geometry and dimensions of a particular setting.

Depending upon their intended use, candlepower curves can be converted to other forms for the sake of convenience. A commonly used alternative form is the isofootcandle curve shown in Fig. 11.9.7.

An isofootcandle curve shows the distance a source will illuminate to a certain footcandle level in the various directions around the source. A typical drawing includes a family of curves at different footcandle levels.

The ease of using candlepower and isofootcandle curves in the design process is described in detail by Lewis (1986), and an example of this process is given in 11.9.8.

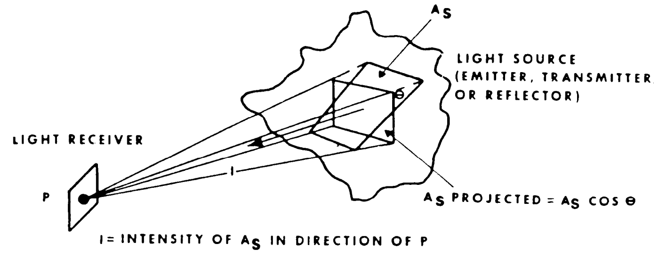

Fig. 11.9.8. Concept of luminance.

11.9.2.7 Luminance

In physical terms, luminance is a concept used to quantify the density of luminous flux emitted by an *area* of a light source in a particular direction toward a light receiver such as a human eye. The area of a light source, in practice, may be an area of a light reflecting surface, such as a wall or desk top; an area of a light emitter, such as a lamp; or an area of a light transmitter, such as a diffusing lens on a luminaire. The most common definition of luminance L is

$$L = \frac{I}{A_s \text{ projected}} = \frac{I}{A_s \cos\theta} \tag{11.9.9}$$

and its IES unit is candela per square inch (cd/in.^{2}) also referred to as foot lambert (fL), while its SI unit is candela per square meter. Referring to Fig. 11.9.8, I is the intensity of light produced by area A_s of the source in the direction of the receiver P, and A_s projected is the projected area of the source when viewed from the receiver. A, projected is equal to A_s cos where q is the angle between the normal to A_s and the angle of observation. Fig. 11.9.8 illustrates the concept. When A_s is large, L is the *average* luminance of A_s; as A_s decreases, L approaches a value of point luminance.

The luminance concept is extremely important in illumination design because it is a physically measurable quantity that correlates with the subjective evaluation of "brightness" when viewing a surface or object. By manipulating the variables (L, I, A_s) in the luminance equation, the subjective evaluation of "brightness" becomes apparent as light intensity (I) and area (A_s) are changed.

When applying the luminance concept, the principles to remember are that luminance (1) varies directly with the intensity (I) of a light source of constant area, (2) varies inversely with the area (A_s) of a light source of given intensity, (3) is independent of distance of observation, and (4) is affected by the angle of observation of the lighted surface.

11.9.2.8 Relationship Among Reflectance, Illumination, and Luminance

Reflectance p is the ratio of reflected to incident light energy, which may be defined as lumens emitted per unit area divided by lumens incident per unit area.

For perfectly diffuse reflectors, lumens emitted per unit area is luminance L', and lumens incident per unit area is illumination E. Therefore,

$$\rho = L'/E \tag{11.9.10}$$

This is an extremely important equation for illumination design because it can be used to determine the illumination that should be provided for an environment, given the desired luminance (i.e., brightness) level and the reflectance of the environment.

The simple relationship among luminance, illumination, and reflectance represented by Eq. 11.9.10 applies only to perfectly diffuse reflectors. Although no surface exactly meets this criterion, such a relationship often applies over a wide range of viewing angles making the use of the formula a practical matter in many cases. Although coal has a specular component in its reflection, the cleaved surfaces are generally not well oriented. In the main, coal can be considered a diffusing surface, and for practical purposes of analysis, perfectly diffusing. In some design problems, detailed reflectance measurements from many angles of observation are often made to assess the limits to which this equation may be applied.

11.9.3 LIGHT MEASURING TECHNIQUES AND INSTRUMENTATION

Instruments are required to evaluate lighting systems and components. The field of light measurement is called *photometry*, and the instruments used to measure lighting are called photometers.

Many types of photometers are available to measure light energy and related quantities, including illumination, luminance, luminous intensity, luminous flux, contrast, color, and visibility. The discussion in this chapter is limited to photometers used to measure illumination E and luminance L from which other quantities, such as reflectivity and luminous intensity, may be computed.

In underground mine lighting applications, the photometer is used to verify compliance with Mine Safety and Health Administration (MSHA) lighting regulations and in the design and evaluation of lighting systems.

Before taking measurements with a photometer, care must be taken to insure that a luminaire or illumination system is in the proper condition to satisfy the purpose of the measurements. An adequate warmup period should be allowed for fluorescent, sodium vapor, and mercury vapor luminaires; line voltage should be stable; and dirty luminaires, obstructions, or deviations in luminaire mountings and orientations from specifications should be avoided.

11.9.3.1 Portable Photoelectric Photometers

The portable photoelectric photometer consists of a photocell that receives light and converts it into an electrical signal that is conditioned through an electrical circuit and is displayed on a visual meter. The meter reading is proportional to the light energy level received by the photocell.

Even the more simple and durable photometers are delicate instruments that can give highly erroneous results when improperly used or calibrated. Proper instruction in the use and calibration of photometers cannot be overstressed. Many factors can cause significant errors in the measurements in the very low light levels typical of underground coal mines. Photometers are available in a variety of models ranging from the low-cost, hand-held units, which are very convenient but have limited accuracy and range, to more expensive, and more accurate but less portable units.

The following major factors affecting the accuracy of photometers should be understood and considered before purchasing and using a unit.

11.9.3.2 Photocell Characteristics

Photocells used in portable photometers have been improved significantly in the past few years. New design photometers that utilize silicon photocell technology have distinct advantages when compared with selenium photocell photometers. Silicon photocells are more stable and exhibit a more uniform (linear) response in output with a change in light level. Selenium photocells are more prone to change in calibration with time and also can exhibit a memory (hysteresis) effect when measured light levels vary significantly. When exposed to illumination, the output of photocells decreases over a period of time because of fatigue. Therefore, the meter should be exposed to the light level being measured for as long an adaptation period as necessary, that is until the meter reading stabilizes.

11.9.3.3 Color Correction

The color response of a typical photocell differs from that of the human eye. This difference would cause a significant error in the measurement of visible light if the cell were not color corrected. Thus photometers must be color corrected by filters to ensure accurate measurements.

11.9.3.4 Cosine Correction

The response of a photocell changes as the angle of light impinging on its surface changes. Errors in light measurement caused by this factor alone may be as much as 25%. Cosine correction is provided by placement of a diffusing cover over the photocell.

11.9.3.5 Sensitivity

Illumination and luminance levels in underground coal mines are very low. Regulations call for an average luminance of 0.06 fL and incident illumination levels in the range of 2.0 fc. For the engineering design and evaluation of underground coal mine illumination systems, it is recommended that the photometer sensitivity be such that a full-scale range of no more than 0 to 3 fc is available. The meter should, of course, have other selectable ranges for higher level measurements, perhaps to 50 fc or greater. A meter with high sensitivity and accuracy permits the very necessary fine tuning of lighting systems needed to meet the stringent lighting regulations.

In designing mine lighting systems, luminance measurements are often required to determine the reflectance of the simulated mine surfaces. For luminance measurements, meters with an available full-scale setting of approximately 0 to 0.1 fL are recommended.

Use of inadequate meters in the design and/or evaluation of mine lighting can result in overdesign of the system, which could be unnecessarily expensive due to the high cost of illumination system components.

11.9.3.6 Calibration

Calibration is a method by which the response of a photometer is set to match a working standard. Probably the most significant source of error in illumination measurements is inaccurate instrument calibration. Photometers are particularly susceptible to loss of calibration and should always be checked both before and after any series of light measurements. The meter can be calculated with a reference standard specifically designed by the manufacturer for the specific instrument. Whatever standards are used, they must be traceable to a primary lighting standard in a national physical laboratory.

11.9.3.7 Photometer Zeroing

It is important to check photometer zeroing prior to taking measurements. If an analog meter is used, this requires setting the meter reading to zero with the photocell completely covered. It should be verified that the meter remains correctly zeroed, when the photometer scale selector is changed. A photometer that cannot be properly zeroed on all scale ranges should be repaired and recalibrated. Improper zeroing can be the source of significant error in the low coal-mine illumination levels.

11.9.3.8 Temperature and Humidity Effects

Wide temperature variations affect the performance of photocells. Prolonged exposure of selenium photocells at temperatures above 120°F (49°C) will permanently damage them. Silicon photocells are less susceptible to temperature variation when compared with selenium photocells. Exposure of a photometer to corrosive high-humidity conditions should be kept to a minimum. Photometers should certainly never be stored underground. Hermetically sealed cells provide greater protection from the effects of both temperature and humidity. It is recommended that a desiccator be packed with meters taken underground.

11.9.3.9 Contamination

In mines, dust can rapidly accumulate on the photodetector surface and diminish measurement accuracy. Moisture and dust can enter photometer enclosures and cause component corrosion or wear. These factors can easily affect the accuracy and useful life of an instrument. Photometers should be kept in a well-sealed case and, to avoid contamination, should be removed only when they are to be used. The photodetector surface should be kept very clean, and care taken to use a cleaning method that will not scratch the surface.

11.9.3.10 Making Illuminance Measurements

Considerable error in measurements can occur if the light meter photocell is not positioned correctly for the type of illuminance measurement being taken. In mine lighting, illuminance measurements are typically taken for the following purposes:

1. To determine the incident luminous energy (footcandles) on a surface.
2. To determine the light output characteristics of a luminaire.
3. To determine if illuminance levels are sufficient to qualify the illumination system for MSHA approval.

First of all, it is important to recognize that the photocell is designed to generate an output signal that is proportional to all the light impinging on it and passing through an imagined hemisphere below which the photocell is placed with the lens facing directly up. Therefore, if a measure of illuminance impinging on a surface is required, the photocell should be placed flat against the surface, as shown in Fig. 11.9.9A. This reading will yield the footcandles (lumens per square foot) of luminous energy that is intercepted by the surface.

When determining the candlepower distribution of a luminaire, the photocell should be pointed directly at the luminaire, as shown in Fig. 11.9.9B. The candlepower curve can then be computed using the inverse square law, $E = I/D^2$ where E is measured illuminance, D is distance to the luminaire in ft, and I is luminous intensity in candelas. Remember, when the distance from the photometer to the luminaire is less than five times the maximum luminaire dimension, this method rapidly becomes inaccurate.

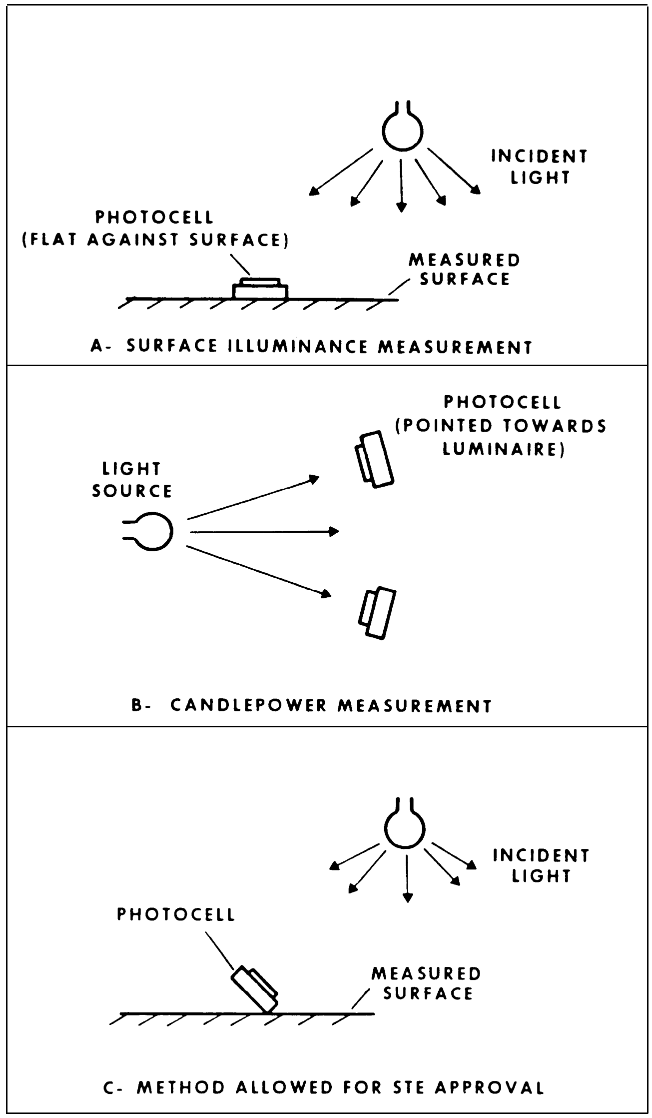

Fig. 11.9.9. Photocell orientation for making illumination measurements. STE—Statement of Test and Evaluation.

When illuminance measurements are made to qualify a lighting system for a statement of test and evaluation (STE, which will be discussed later in 11.9.6), MSHA allows them to be made by pointing the photocell in the direction that results in the highest measured value. This special case of measurement, as illustrated in Fig. 11.9.9C, yields the level of illuminance emanating in the direction of the photometer only. This measured illuminance does include the effect of the cosine law, and therefore, the values cannot be used for the computation of surface reflectance. This procedure is useful for STE measurements only.

11.9.3.11 Making Luminance Measurements

Luminance can be measured directly with photometers designed for this purpose. Typically, photometers have a luminance adapter, which allows light within a specific cone angle to be sensed by the photocell.

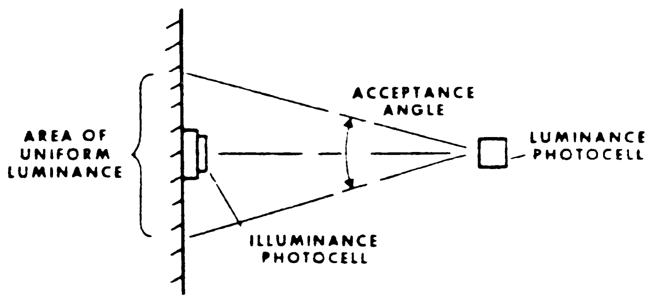

Fig. 11.9.10. Reflected-incident light comparison method for measuring reflectivity.

One method specified for making luminance measurements of the face, roof, ribs, and floor with a photometer is specified in the *Code of Federal Regulations* (CFR), Title 30, part 75.1719-3 (Anon., 1988). Luminance measurements specified in these regulations state that the photometer shall be held approximately perpendicular to the surface being measured. They also require that the sensing element be at a sufficient distance from the surface to allow the light sensing element to receive reflected light from a field not less than 3 ft^2 ($0.27m^2$) nor more than 5 ft^2 ($0.45m^2$). In areas where a photometer cannot be held sufficient distance from the surface to encompass at least 3 ft^2 ($0.27m^2$), the luminance can be computed by averaging four uniformly spaced readings taken at the corners and within a square field of approximately 4 ft^2 ($0.36m^2$). In the evaluation of large surface areas, the meter can be used in a scanning mode, which requires the meter be held 4 to 5 ft (1.2m to 1.5m) from the surface being measured and moved in a direction parallel to the surface. The scan rate must be less than 3 ft/sec (0.9m/s) to allow sufficient time for the sensor to react to changes in surface luminance. Such a procedure might be useful to mine operators to determine if they are in compliance with the lighting standards. The detailed regulations and procedures are discussed later in this chapter and should be reviewed when luminance measurements are to be taken to establish compliance of a mine lighting system.

11.9.3.12 Making Reflectance Measurements

The total reflectance of a surface was defined as $\rho = \phi$ reflected light/ϕ incident light, where ϕ reflected is the total reflected flux and ϕ incident is the total incident luminous flux.

Accurate measurements of the reflectance of surfaces such as a coal face can be a complex task because, when a surface is not a diffuser, the reflected flux can be scattered nonuniformly. Methods for the measurement of the reflectance of coal surfaces are briefly discussed in the following paragraphs. No standard method has been established for the measurement of coal reflectance.

The *reflected-incident light comparison method* is a convenient and simple means to measure reflectance if a surface is a good diffuser. For a perfect diffuser, reflectance can be computed using Eq. 11.9.10, $r = L'/E$. Although coal has a specular component in its reflective characteristics, the cleaved surfaces are often not well oriented and, therefore, it may be considered a diffusing surface and for some purposes perfectly diffusing. A standard portable photometer equipped to measure both illuminance and luminance can be used in this method of reflectance measurement. Illuminance is measured by placing the photocell flat on a uniformly illuminated surface as shown in Fig. 11.9.10.

With the photometer in the luminance measurement mode, the surface luminance is measured and care is taken that the area being measured is within the conical acceptance angle of the luminance photodetector. Surface reflectance can then be computed with the preceding equation. This is the method utilized by MSHA to approve coal mine simulators, which must have a surface reflectance no greater than 5%. Note that measurement error is introduced when the surface is not a perfect diffuser. Also care must be taken to insure that the presence of instrumentation or the operator does not interfere with the actual light levels.

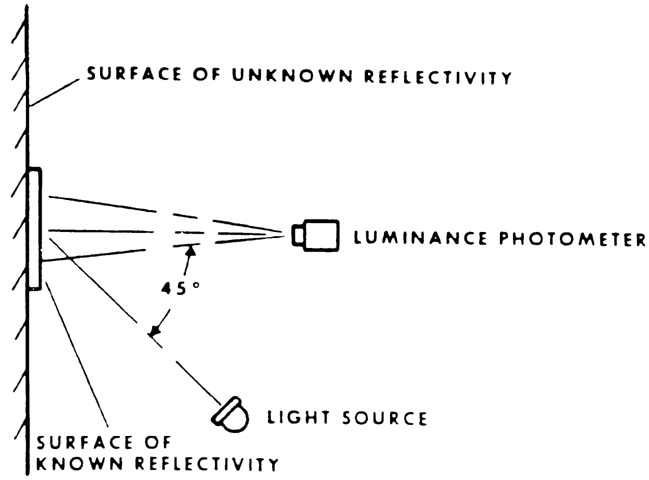

Fig. 11.9.11. Luminance standard comparison technique for measuring reflectivity.

The *reflectance standard comparison method* is a method by which the reflectance of an unknown surface is determined by comparing the luminance of a standard of known reflectivity to the luminance of the surface being tested. The reflectance of the surface being tested can be computed with the equation $r = r_s (L/L_s)$, where r_s is reflectance of the known standard surface, L is the luminance of surface being tested, and L_s is measured luminance of the standard surface. This method is depicted in Fig. 11.9.11. The acceptance angle of the photometer must be sufficiently small to insure that the measured luminance is within the area of the standard surface.

This method is adaptable to controlled laboratory measurement conditions because sample sizes need not be large. Commission International del'Eclairage (CIE) (Anon., 1981) standard conditions for using this method state that the source of illumination shall be at an angle of 45° from the perpendicular to the surface to be tested and that viewing should be perpendicular to the surface.

11.9.4 PHYSIOLOGICAL REQUIREMENTS FOR HUMAN VISION

Vision and the effect of illumination levels on visual perception is an extremely complicated process by which the brain receives the vast majority of information about the outside world. The operation of the eye has been described in great detail by many authors (Halldane, 1970); for this review, the most relevant information concerns the operation of the eye at low light levels. Light enters the eye through the lens and is focused on the retina. The retina contains a large number of light-sensing organs called rods and cones. The rods operate at low light levels and do not detect color; the cones can detect color but require higher levels of luminance to function. The distribution of these sensors is not uniform across the retina. Cones are dense at the center of the retina, and each is individually connected to the brain by a separate nerve fiber. As the distance from the retina's center increases, the density of cones decreases and rods predominate. The rods are not individually connected to the brain; instead, several transmit their information via a single nerve fiber. This characteristic causes the eye to see detail only in the center of the retina, utilizing predominately cone receptors. At luminance levels below the photopic or daylight vision threshold, the eye is night-adapted and is said to be in the scotopic or rod-dominate mode of operation. At these levels, cones do not receive enough energy to function, and the ability to see detail and color is lost. At luminance levels above the photopic threshold, cone vision is restored. A transitional mode of operation, between the photopic and scotopic range where the cones are not completely operational, has been defined as the mesopic region. Halldane (1970) reports that this transitional region lies between 10^{-3} to 10^{-1} fL, with the greatest changes occurring below 10^{-2} fL. Thus, at light levels above this value, the majority of photoreceptors are operational.

While it is difficult to precisely determine, the cone vision threshold lies between 0.03 fL and 0.05 fL, and it is generally agreed that illumination levels must maintain the eye above the scotopic, or night, vision range. Merritt et al. (1983) recommended 0.06 fL as a luminance level for reasonable response times in the detection of hazards. The reasons for this recommendation include not only maintenance of detail and color vision, but also the prevention of miners' nystagmus, a disease common to underground miners before 1940. Miner's nystagmus is caused by long periods of conditioning to low light levels and results in the individual's inability to look directly at an object, resulting in constant eye movement and nausea.

The introduction of the modern electric cap lamp eliminated miners' nystagmus and provided adequate illumination for detailed, near field tasks. However, the cap lamp's photometric distribution is very narrow (typically 5 to 10°) and in low-reflectivity environments does not provide enough illumination to ensure adequate peripheral vision. Standards that maintain levels above the scotopic threshold and provide adequate peripheral vision have been established in many countries (including the United States) to provide for area illumination in various high-activity locations in underground mines. International standards have been drafted by the Commission International de L'Eclairage TC 4.10 Committee (Anon., 1981) that follow these principles.

11.9.5 SURFACE MINE ILLUMINATION AND ILLUMINATION OF THE SURFACE AREAS OF UNDERGROUND MINES

Surface mining operations occupy vast areas, and activity is dispersed throughout the property, typically continuing 24 hours a day. It is the continuous operation throughout the night that demands consideration of adequate illumination. Federal standards that address illumination of these areas are general in nature and are published in the CFR, Title 30, parts 56.17001 and 77.207 (Anon., 1988.) These sections state, "Illumination sufficient to provide safe working conditions shall be provided in and on all surface structures, paths, walkways, stairways, switch panels, loading and dumping sites, and work areas."

The problem of providing adequate illumination for surface mining operations can be said to address a number of generic situations. These include the areas surrounding various slow-moving machines that typically perform their functions in a stationary mode such as draglines and highwall drills, high-speed machines such as haulers and front-end loaders, and finally, specialized task and area illumination for locations such as the interiors of large equipment and maintenance activities.

Table 11.9.1. Draglines, Shovels, and Bucket Wheel Excavators: Illumination Intensities While Equipment Is Operated

Location	Average illumination intensity in fc	Uniformity ratio
1. Interior walkways while persons are present.	10	10/1
2. All other interior areas, except operating cabs while persons are present.	10	10/1
3. All areas 20 ft (6m) in all directions from the main frame, including all work or travel areas beneath the main frame.	5	10/1
4. Exterior walkways on board draglines, shovels, and wheel excavators.	5	10/1
5. The area beneath the boom from 20 ft (6m) from the main frame to the farthest point the equipment is capable of excavating or discharging material.	1	10/1

Source: Anon., 1977.

Dragline, power shovel, and bucket wheel excavator illumination recommendations were published in the US Federal Register and are listed in Table 11.9.1.

Illumination levels for other self-propelled surface mining equipment are listed in Table 11.9.2.

Research to define appropriate illumination levels for other surface mining activities has not progressed to the extent that permits specific levels to be defined for all major tasks or situations, and so in many cases, the illumination engineer can only refer to standards that address similar but not identical conditions.

As an example, the IES Lighting Handbook (Kaufman and Haynes, 1981) lists recommended illumination levels for a large number of situations and therefore provides a good reference source. A specific example would be IES's recommendation of an illuminance level of 2 FC for building excavation. This condition is analogous to bench preparation activity. Until such time as definitive studies are completed, the reader should view these values as approximate, and consideration should be given to the local conditions, especially the contrast level between the overburden and ore.

11.9.6 UNDERGROUND COAL MINE ILLUMINATION

The lumination requirements for underground coal mines are defined in detail by CFR in Title 30, Section 75.1719 (Anon., 1988). This section consolidates and explains the regulations, interpretations, and official enforcement policies that govern the implementation of lighting in underground coal mines in the United States, including:

1. Specification of required illumination levels and the particular areas of the mine that must meet these levels. (75.1719-1)

2. Specification of standards (primarily electrical) to be observed when implementing lighting systems to prevent the sys-

Table 11.9.2. Self-propelled Equipment: Illumination Intensities While Equipment Is Operated

Location	Average illumination intensity in fc	Uniformity ratio
1. An area to the front and to the rear of all rubber-tired or crawler-mounted scrapers, front-end loaders, dozers, graders, loaders, and tractors. The areas shall start at the point the operator can normally see the ground surface and extend for a distance of 20 ft (6m) and for a width equal to the machine width.	1	10/1
2. An area to the side of equipment where operating visibility is normally to the side. The area shall start at the side edge of the blade or bucket and extend to the side and rear of the blade or bucket for a distance of 5 ft (1.5m).	1	10/1
3. All areas within 10 ft (3m) of the hole being drilled by vertical drills, except small pneumatic drills.	5	10/1
4. All areas within 10 ft (3m) of horizontal drills and coal augers. On-board work areas of horizontal drills and coal augers.	5	10/1

Source: Anon., 1977.

terns from posing additional hazard to worker or machine. (75.1719-2)

3. Specification of procedures for making light measurements to verify compliance with the illumination standards. (75.1719-3)

4. Specification of requirements, in addition to illumination, to increase visibility in the mine environment. (75.1719-4)

11.9.6.1 Primary Illumination Standard

Published lighting values are not the complete answer to the problem they are meant to help solve, for whenever a stage of knowledge is reached upon which a code is based, unsolved problems still remain.

This statement applies, of course, to US lighting regulations. Anyone implementing a lighting system on the basis of these regulations should not view the standards as a "cookbook prescription" for optimum lighting. Rather, these standards are only intended to specify *minimum* requirements for meeting the *priority* visual needs of the mine-face situation. The status of current knowledge and the necessary accommodations to account for variability of mining circumstances make specification of more extensive standards, at least for the present, inappropriate. To realize the optimum benefit from mine lighting, it may be desirable to exceed these minimum requirements in some instances and to consider additional factors.

The "cornerstone" of the US lighting code is expressed in provision 75.1719-1(d) of the *CFR* (Anon., 1988). "The luminous intensity (surface brightness) of surfaces that are in a miner's normal field of vision of areas in working places that are required to be lighted shall be not less than 0.06 foot-lamberts . . ."

The following points should be noted about this specification:

1. The requirement is expressed in terms of luminance rather than illuminance. This is the preferred method of expressing standards since only the physical measure of luminance correlates what the eye sees. Moreover, by specifying luminance rather than illuminance, the standard is, theoretically, uniform regardless of variability on environmental reflectivity. The disadvantages of expression of standards in luminance terms is that (a) luminance measurements are more difficult to make, and (b) design requires accurate knowledge of reflectivity.

2. The statement that 0.06 fL applies to "surfaces that are in a miner's normal field of vision" is an explicit recognition that mine lighting should accommodate peripheral vision. Peripheral vision is important in mining for (a) early recognition of conditions that might give prewarning of potential safety hazards in the peripheral field, and (b) performance of tasks that require knowledge of the relative spatial relationships among objects separated by significant distance in the surroundings. Light on all surfaces in a miners' normal field of vision eliminates the "tunnel vision" effect of the narrow cap lamp beam and overcomes beam cutoff by machine structure or posts. Provision that this light illuminates these surfaces to 0.06 fL insures response of all the eye's photoreceptors to light impulses in observer's field of view, particularly in the peripheral field.

3. Research shows that the 0.06-fL level is adequate for performance of a major portion of mining tasks (fine perception of details is generally not required). Most of those tasks where greater levels are desirable are performed at close range, and the cap lamp acts as a supplement to the general lighting for adequately meeting these lighting requirements.

4. The 0.06-fL general level of luminance is low enough that adaptation problems in going from the illuminated area to darker areas of the mine are not severe.

11.9.6.2 Applying Illumination Standards in the Mine Environment

The federal coal mine lighting standards apply to all self-propelled mining machines utilized inby the last open crosscut. The reason for requiring illumination around these machines and in this area of the mine—as opposed to other machines and other areas of the mine—is because here is where (1) machine-worker activity is most concentrated, and (2) hazards are most serious and most likely to develop.

The standards are expressed in terms of specific machine vicinities to be lighted to a surface brightness of at least 0.06 fL. These vicinities vary, depending upon machine type, mining method, and seam height. They have been selected primarily on the basis of the objectives stated previously; that is, lighting needs for adequate peripheral vision and adequate visibility for task performance and hazard identification.

The specific machine vicinities to be illuminated to the 0.06 fL standard are identified in CFR 75.1719-1(e)(1) through (e)(6) (Anon., 1988). In enforcing these standards, certain accommodations have been made by the MSHA so that the effective standards are consistent with the current status of mine lighting technology. The accommodations are in recognition of problems with discomfort glare, hardware maintenance, efficient utilization of fixture candlepower, etc., that currently cannot be alleviated through system design and installation with available hardware and techniques. Policy in enforcing the illumination standards is based upon extensive and continuing post audit of the impact of the standards and reflects a commitment by MSHA as a proponent of both good and practical lighting within the intent of scope of CFR 75.1719.

As an alternative to evaluating an illumination system in an underground coal mine, MSHA has established a program that permits the surface certification of a coal mine illumination system and the issuing of a *statement of test and evaluation* (STE).

The STE program is administered by MSHA's Approval and Certification Center (A&CC) group by system designers, including manufacturers and mine operators. This information is reviewed to determine system compliance with relevant requirements of the CFR and, if these requirements are satisfactorily met, A&CC issues an approval called a STE to the party submitting the design for review.

Mine operators may then implement the design in their mines and be guaranteed compliance with the illumination standards as long as the approved design and application specifications are initially and continually met in the actual installation. This includes proper maintenance of the system with respect to permissibility, cleanliness, and operativeness of system components.

The primary advantage of a STE-approved design over a non-STE-approved design is that the STE is a substitute for onsite light measurements in determining whether the system is in compliance with the illumination requirements. An inspector will not take light measurements as long as the system meets the design specified in the STE and is properly maintained.

The standards that apply to the various machines and environments, the latest enforcement policies for these standards, and details of STE application are described in USBM IC 9074, "Underground Coal Mine Lighting Handbook" (Lewis, W.H., 1986).

In complying with these standards, the mine operator has the option of mounting luminaires on the machine or separate from the machine, for example, temporarily mounting them along roof and ribs. The latter option is typically referred to as "area lighting." Because of practical difficulties with area lighting systems, nearly all systems now utilized employ machine-mounted fixtures.

The seam height used for lighting application requirements at a mine is equal to the normal extracted height. In cases where entry dimensions vary across the entries of a working section, the lowest dimension may be used to determine the lighting requirements that apply to that situation. On sections where the seam is extracted in two or more lifts (e.g., the upper 10 ft or 3m and lower 10 ft or 3m of a 20-ft or 6-m back seam), the lighting system must meet the requirements for both seam heights.

In addition to the illumination of face machine vicinities, it is a federal requirement that each person who goes underground wear an approved personal cap lamp.

11.9.7 ILLUMINATION FOR UNDERGROUND METAL AND NONMETAL MINES

Illumination requirements for underground metal and nonmetal mines are addressed in the CFR, Title 30, part 57.17010 (Anon., 1988). This section specifies lighting (requirements) only in general terms as follows: "Individual electric cap lamps shall be carried for illumination by all persons underground."

The metal and nonmetal mine environment offers a large variation in conditions that affect illumination requirements. The physical dimensions of the mine void and the reflectivity of the underground surfaces are of primary concern. Physical dimensions of worksites vary from a 4 by 4-ft (1.2 by 1.2-m) by 10-ft (3-m) development raise to a 124 by 148-ft (37.5 by 45-m) by 200-ft (60-m) open stope. Reflectivity varies from less than 2 to over 90%.

Table 11.9.3. Minimum Luminance Recommendations for Metal and Nonmetal Mines

Visual task	Luminance, fL Age, 40	Luminance, fL Age, 55	Maximum luminance ratio
Cracks			
Wide	0.029	0.058	27:1
Narrow	0.064	0.13	12:1
Rock Motion	0.066	0.13	12:1
Floor hazard			
Near holes	0.025	0.050	55:1
Far holes	0.014	0.028	13:1
Near rubble	0.019	0.038	72:1
Far rubble	0.014	0.028	36:1
Floor hazard			
Far holes	0.050	0.10	N.A.
Far rubble	0.024	0.048	N.A.
Peripheral			
Motion on foot	0.0026	0.0052	85:1
Mach. op.	0.013	0.025	N.A.

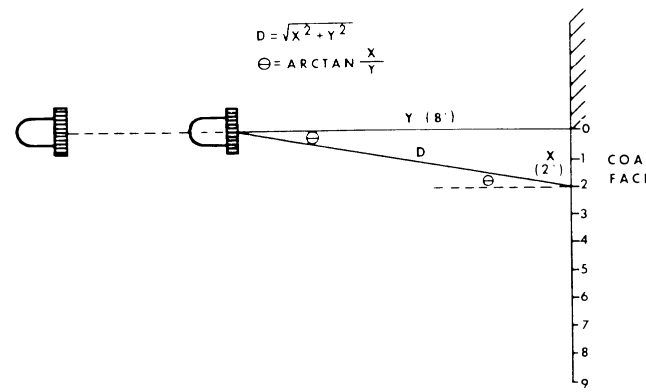

Fig. 11.9.12. Geometry for performing illumination calculations.

Merritt et al. (1983) studied the illumination requirements of the metal and nonmetal mine environment and determined minimum luminance requirements for a number of generic visual tasks as listed in Table 11.9.3. These values closely agree with other findings of a general minimum luminance value of 0.06 fL.

Research to establish appropriate illumination levels for other specific activities has not progressed to the extent that permits specific levels to be defined for all major tasks or situations. Therefore, the illumination engineer can only refer to standards that address similar but not identical conditions. The IES Illumination Handbook (Anon., 1981) lists recommended illumination levels for a large number of situations and, therefore, provides a good reference source.

11.9.8 EXAMPLE OF USE OF LIGHTING EQUATIONS AND RELATIONSHIPS

The following example illustrates the application of candlepower and isofootcandle curves in defining the illumination levels and light distribution produced by a typical mine headlight fixture, and the use of the cosine and inverse square laws, in conjunction with these curves, to calculate the illumination level at any point on the coal face.

Example 11.9.1. In this example the headlight is mounted on the inby end of a cutting machine that is assumed to be located 8, 12, and 16 ft (2.43, 3.65, and 4.87m) from the face. Calculation of illumination levels is made at 1-ft (0.3-m) intervals from the beam axis as shown in Fig. 11.9.12.

Table 11.9.4. Footcandle Values for Different Distances From Lamp Axis and Specified Distances From Coal Face

Distance from lamp axis, ft	Distance from headlight to face, ft		
	8	12	16
0	150	67	38
1	121	64	38
2	33	38	34
3	6	15	18
4	2	5	8
5	0	2	5
6	0	1	2
7	0	0	1
8	0	0	1
9	0	0	0

Conversion factor: 1 ft = 0.3048 m.

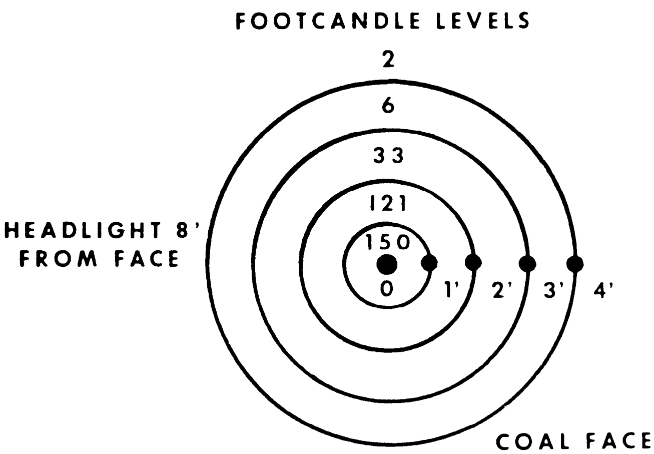

Fig. 11.9.13. Pattern of illumination with headlight 8 ft (2.4m) from mine face.

Solution. Illumination on the face normal to the direction of light travel is given by Eq. 11.9.7

$$E = I/D^2$$

For D = 8 ft and I = 2300 c (from candlepower curve),

$$E = 2300/64 = 36 \text{ fc}$$

Illumination on the face at points not normal to the direction of light flow is given by Eq. 11.9.8:

$$E = \frac{I \cos \theta}{D^2}$$

For a point 2 ft from the axis of the light,
$D^2 = 8^2 + 2^2 = 68$
θ = arc tan x/y = arc tan 2/8 = 14°
I = 2300 c
E = (2300/68) cos 14° = 33 fc

Using these relationships, the footcandle distribution on the face was calculated for the specified lamp distances from the face and different distances from the lamp axis. The results are presented in Table 11.9.4. Because the candlepower curve for the headlight is symmetric about the lamp axis, lines of equal illumination will form concentric rings on the face as shown in Fig. 11.9.13.

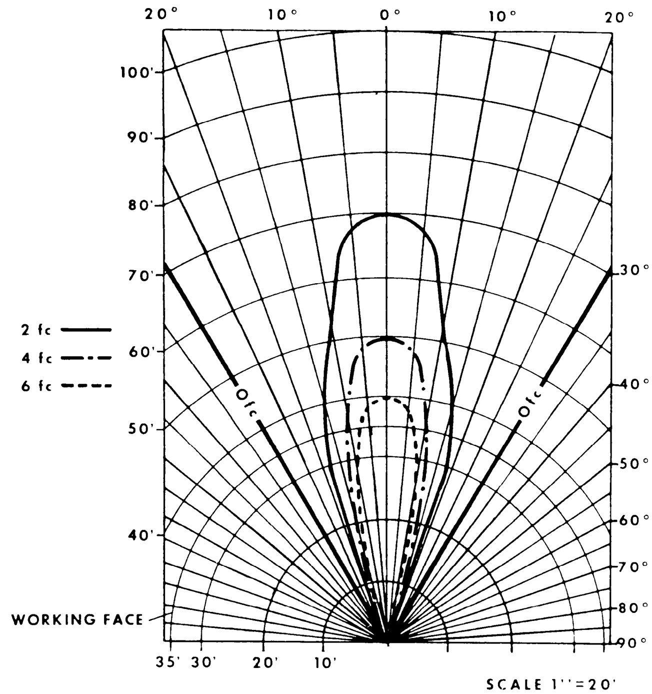

Fig. 11.9.14. Isofootcandle curve by a mining vehicle headlight.

When lamp candlepower curves are not symmetrical, similar calculations can be performed using candlepower curves for different planes and then drawing lines to connect points of equal footcandle levels.

The candlepower curve can be converted to an isofootcandle curve by using the following procedure:

Read the candela values from headlamp candlepower curve for a sampling of angles measured from the lamp axis, say, every 5° interval. For more accurate curves, smaller intervals are used.

Using the inverse square law, substitute values for the candlepower (read from the curve) and the desired isofootcandle curve level. Solve the equation for the distance at which the lamp will light to the desired footcandle level (Eq. 11.9.7):

$$E = I/D^2$$

Solving for D and substituting values for I and E,

$$D^2 = I/E = 2300/6 = 19.5 \text{ ft } (5.9 \text{ m})$$

Repeat this calculation for other candlepower/footcandle combinations and plot the results on polar-coordinate paper as shown in Fig. 11.9.14.

Illumination at any surface along the lamp axis and normal to the axis can be determined simply by drawing such a surface on the isofootcandle curve. Distances from the axis where the isolines intersect the surface may be scaled from the drawing and the isofootcandle curves plotted as shown in Fig. 11.9.15.

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Anon., 1981, "An Analytic Model for Describing the Influence of Lighting Parameters Upon Visual Performance," Vol. 1, "Technical Foundations," Vol. 2, "Summary and Applications Guidelines,"

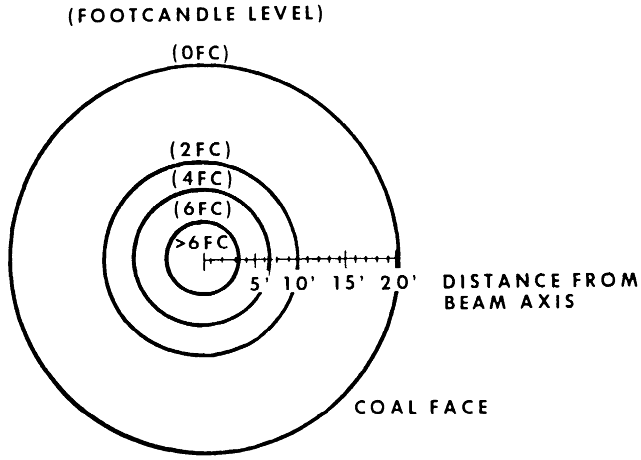

Fig. 11.9.15. Pattern of illumination with headlight 35 ft (11m) from mine face (determined by isofootcandle curve).

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Section 12 Auxiliary Operations

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Chapter 12.0
INTRODUCTION

CHRISTOPHER J. BISE

Although production operations at mines, representing the extraction and removal of the mined material, often appear to receive the greatest amount of attention, the emphasis given to auxiliary operations is continually increasing. This is due to the changing economic, social, and environmental climate in which modern mines must operate. *Auxiliary operations* can best be described as those activities that are not directly related to the extraction and removal of the mined material, but must receive proper attention if recovery is to take place in a safe, efficient, and socially responsible manner. In fact, certain mining auxiliary operations, such as strata control and ventilation, are considered to be of such importance that they warrant separate sections in this *Handbook*. Judging from a mine-personnel standpoint, it is not uncommon for a mine to hire the same number or more individuals responsible for auxiliary operations as the number assigned to production.

This section is organized into eight diverse chapters which are related to each other solely in their impact as auxiliary operations.

Chapter 12.1 deals with the design and management of water and sediment control systems. The proper design and implementation of a comprehensive stormwater and sediment control system can enhance overall mining productivity, meet regulatory requirements, and reduce time delays in permitting. Many cost-effective design procedures are illustrated throughout this chapter. Emphasis is placed on the types of control that are frequently utilized in water conveyance, retention, and energy dissipation. Likewise, current and innovative sediment control structure design procedures are explained. Methodologies needed to develop peak flow, a hydrograph, and sediment load are provided. Since sediment basins are the predominant stormwater and sediment control devices, cost-effective techniques, such as internal check dams and passive dewatering systems, are discussed in detail with respect to sediment trap efficiency. Various types of diversions and culvert design methods are described. Selection, design considerations, and troubleshooting procedures for pumps are illustrated through a dewatering design.

Chapter 12.2 focuses on the handling, transportation, and placement of mine waste, such as overburden, waste rock, low-grade and ore-grade stockpiles, refuse, mill tailings, leach heaps and dumps, slags, process sludge and residues, acid mine drainage, and seepage water. Designing waste disposal facilities that meet environmental goals is a complex task, resulting from a large number and variability of factors influencing waste, the many disciplines involved, the general inexperience of meeting environmental goals, and the inherent geotechnical uncertainty that must be managed. The intent of this chapter is to identify the basic design considerations for meeting both traditional structural goals and emerging environmental goals. Low-volume special wastes or stack effluent are not discussed; neither are methods for addressing many reclamation and environmental issues (these are covered elsewhere in the *Handbook*). The enormous potential for liability, the quick evolution of technology and regulations, and the complexity of factors make it imperative for a design engineer to develop the specific components of a design based on current information from the many disciplines involved.

Chapter 12.3 deals with the broad subject of reclamation. The scope of reclamation is addressed initially in order to establish the bounds of the text that follows. One issue addressed specifically is the regulatory framework related to reclamation. A major portion of the chapter focuses on reclamation planning since forethought can often lessen the burden placed upon operators and produce superior results. A general reclamation process is described in detail. Finally, several specific reclamation problems are considered, including such issues as mine waste disposal, soil-related problems, and stability of fills in mountainous terrain.

Chapter 12.4 provides an introduction and overview of electric-power components and systems commonly used in the mining industry. The intent is to familiarize the mining engineer with the importance, history, terminology, basic design criteria, types of equipment and protection, and system arrangements of mine power systems. To enhance coverage, many guidelines are included for this broad and complex subject. Even if the mining engineer is not directly responsible or involved with the power system, a basic understanding of its safety aspects, operation, and limitations is essential since the power system affects many other unit and auxiliary operations within the mine.

Chapter 12.5 deals with proper design of compressed air systems. This topic is approached from the perspective that the most cost-effective compressed air system requires careful design. To achieve the lowest cost system, this chapter uses a series of examples to lead the reader through the design of a network for a small mine. The lowest-cost system requires assessment of alternatives, which is facilitated with a computer program.

Mine monitoring, control, and communications are covered in Chapter 12.6. Effective communications among mine employees is crucial to the achievement of management goals for the safe and efficient operation of a mine. Technology has advanced considerably since the days of the crank-ringer phone to modern systems that often include computer-controlled private branch exchanges and hybrid wired/wireless systems. Second only to the need for good communications is management's need for information to improve the decision-making process. Mine monitoring systems are playing a pivotal role in providing timely information and allowing for operating economies. Traditionally, the mining engineer has had little exposure to this auxiliary aspect, but is now being called upon to utilize this technology. This chapter presents the necessary background to understand the technology and its appropriate uses, and suggests procedures that can be used to help insure that the technology will yield maximum benefit.

Chapter 12.7 addresses the topic of vehicles used in surface and underground mines for the purpose of transporting supplies and personnel. Both track-mounted and trackless applications are considered, and surface and underground applications are further subdivided into coal and noncoal. A selection procedure is included and encompasses suitability, safety, operating experience, and overall cost. Finally, a detailed example of the calculations required for analyzing the capital and operating costs for a vehicle is included.

Finally, Chapter 12.8 addresses the most direct method of increasing machine availabilities: equipment maintenance. This topic is approached from the perspective that the best maintenance program is one that anticipates and acts on impending problems rather than reacts to existing problems, and that the role of maintenance in mining is to provide the lowest possible cost to a mine in terms of maintenance labor, and material and production loss resulting from the maintenance program. To achieve these goals, strategies are proposed for productivity control, preventive maintenance, and predictive maintenance. As a recordkeeping aid for equipment repair histories and accountability, concepts and considerations for computerized management information systems for mine equipment maintenance are discussed.

Chapter 12.1
DESIGN AND MANAGEMENT OF WATER AND SEDIMENT CONTROL SYSTEMS

RICHARD C. WARNER

12.1.1 INTEGRATING ENVIRONMENTAL COSTS

During past decades, industry has sometimes been viewed by the public as an insensitive steward of the environment. Preoccupation with mineral and energy resource extraction caused the mining industry to neglect the integration of environmental cost into their companies' business plans and developmental policies. The results of this missed opportunity are mandated regulations, leaving many critical decisions outside the control of companies. With the passage of the Surface Mining Control and Reclamation Act of 1977 (SMCRA), Public Law 95-87, the goal of minimizing hydrologic impacts to offsite areas was enacted.

Specific interpretations of this law in the form of changing regulations do not provide for regional differences with respect to the quality of receiving waters and climatologic regimes. Regulations focus on many specific criteria, yet neglect the benefits/costs associated with an integrated overall strategy. For instance, sediment standards are set for a peak concentration associated with a design storm (the specified size of storm for which the strategy is designed), yet neglect the sediment duration-concentration relationship that influences aquatic invertebrates and disregards the water quality and multiple use of receiving waters. Criteria quite appropriate for some locales may be entirely unjustified, both environmentally and economically, for mine sites in other regions. Despite regulatory constraints, opportunities still exist to implement innovative approaches to achieving the goal of minimizing hydrologic impacts.

What is meant by minimizing the hydrologic impact of a mining operation? Surface mining and surface disturbance from underground mining create changes to the hydrologic regime, usually in the form of increased storm runoff and generated sediment loads. If minesite stormwater and sediment discharge can approximate predevelopment conditions, then no significant adverse offsite impacts are expected. The determination of predevelopment conditions should consider both onsite factors and the quality of receiving waters. A stormwater and sediment control plan is created to detain excess stormwater, reduce peak flow, and remove the higher sediment load caused by disturbing stable landforms. Similar to stormwater, the control of nuisance water is particularly important to mining operations.

For general coverage of environmental regulations, see Chapters 3.4 and 7.3.

12.1.2 TYPES OF CONTROLS

A comprehensive multifaceted approach to *storm water, nuisance water, erosion,* and *sediment control,* thoroughly integrated throughout the entire planned life of the mine, can reduce potential environmental impacts, increase mine productivity, and be achieved in a cost-effective manner. The types of controls frequently utilized are (1) water conveyance structures, (2) retention facilities, (3) energy dissipators, and (4) pumps. A wide spectrum of water and sediment controls is listed in Table 12.1.1.

Table 12.1.1. Stormwater and Sediment Controls

- Stormwater conveyance
  - Diversions
    - Temporary
      - Earthen
      - Mats of straw, nylon, etc.
      - Geotextiles
      - Flexible membrane liners
    - Permanent
      - Rock riprap
      - Gabion mattress
      - Concrete-filled geotextiles
      - Concrete
      - Asphalt
      - Grassed waterways
  - Culverts
    - Corrugated metal
    - Concrete
    - PVC
    - Polyethylene (PE)
    - Steel
  - Energy dissipators
    - Plunge pools
    - Rock aprons
    - Gabions
    - Concrete-filled geotextiles
    - Check dams
    - Filter fabric fences
    - Brush barriers
  - Dewatering
    - Sumps
    - System of wells
  - Sediment controls
    - Sediment basins
      - Contour
      - Dugouts
      - Embankment
      - Permanent pool
      - Passively dewatered
      - Infiltration
    - Sediment separators
      - Swirl concentrators
      - Inertial separators
    - Sediment traps and check dams
      - Excavation pits
      - Dozer basins
      - Filter fabric fences
      - Brush barriers
      - Vegetative filters

As can be seen from this list, numerous options exist for designing an integrated water and sediment control plan. Although not explicitly enumerated in this section, two other control methods are crucial to a program that is successful overall: (1) limiting temporal and spatial exposure of highly erodible areas, and (2) the implementation of onsite controls for concurrent reclamation through use of grading, terraces, mulches, soil amendments, grasses, etc.

The most common structures designed to minimize the hydrologic impact of mining are sediment basins, diversions, culverts, energy dissipators, and dewatering systems. Methods, equations, and discussions are provided to illustrate design procedures for these controls.

12.1.3 HYDROLOGY

12.1.3.1 Selection of Analysis Method

Stormwater and sediment control facilities are often based on a design storm of a specified frequency. For simple controls such as diversions or culverts, only a peak flow may be necessary. In contrast, the design of a sediment basin, which involves storage routing, requires a complete hydrograph. Thus the required control defines the complexity of the hydrologic analysis.

The rational equation (Anon., 1969a; Rossmiller, 1982) is widely used for peak flow estimation due to its simplicity. Development of a design storm hydrograph is more complex but has been greatly simplified through user-friendly application programs (Schwab and Warner, 1987; Anon., 1986b; Anon., 1970).

Numerous hydrology texts describe, in detail, all facets of peak flow estimation and hydrograph development (Hjelmfelt, 1975; Linsley et al., 1949; Overton and Meadows, 1976; Viessman et al., 1977; Eagleson, 1970; Barfield et al., 1981).

12.1.3.2 Design Rainfall Event

Stormwater management designs are based on a specified design storm (e.g., 10 yr–24 hr) precipitation amount, temporal distribution of precipitation, and the infiltration characteristics of watersheds. The design storm is normally specified by regulations. Storm depth-duration-frequency values are provided in HYDRO-35 (Frederick et al., 1977) and US Weather Bureau TP40 (Hershfield, 1961) for the eastern United States and in a series of National Oceanic and Atmospheric Administration (NOAA) Atlases for the 11 western states.

Where design situations require maximum assurance of safety, the probable maximum precipitation (PMP) events are utilized. PMPs represent the theoretically greatest depth of precipitation for a given duration that is physically possible over a given size area at a specific location at a given time of the year (Anon., 1985).

To develop a hydrograph, it is necessary to know the design storm depth-duration-frequency (e.g., 4.2 in. or 107 mm, 10 yr-24 hr) and the temporal pattern of rainfall throughout the duration of the storm. The rainfall pattern is usually associated with a synthetic rainfall distribution. The Soil Conservation Service (SCS) has developed four rainfall distribution curves, defined as SCS type curves, applicable to specific regions throughout the United States (Anon., 1986b).

12.1.3.3 Peak Flow Determination

For design of diversions and culverts, only the design storm peak flow is needed. The most widely used method for peak flow determination is the rational method (Anon., 1969a). The rational formula is

English units

$$q = CIA \tag{12.1.1}$$

SI units

$$q = 0.0028\ CIA \tag{12.1.la}$$

where q is peak flow in cfs (m^3/s), C is a dimensionless runoff coefficient, I is rainfall intensity in in./hr (mm/h) with a duration equal to the time of concentration t_c, and A is the watershed area in acres or ac (hectares or ha). The runoff coefficient accounts for all hydrologic factors such as rainfall interception, infiltration, antecedent moisture conditions, etc. Representative C coefficients are given in Table 12.1.2 for urban and rural areas. The rainfall amount is found in HYDRO 35 (Frederick et al., 1977).

Table 12.1.2. Runoff Coefficients for the Rational Method

Description of Area	Runoff Coefficients
Residential	
Single-family areas	0.30–0.50
Suburban	0.25–0.40
Industrial	
Light areas	0.50–0.80
Heavy areas	0.60–0.90
Unimproved areas	0.10–0.30
Lawns; sandy soil	
Flat, ≤ 2%	0.05–0.10
Average, 2–7%	0.10–0.15
Steep, > 7%	0.15–0.20
Rural Areas (clay and silt loam)	
Woodland	
Flat 0–5%	0.30
Rolling 5–10%	0.35
Hilly 10–30%	0.50
Pasture	
Flat 0–5%	0.30
Rolling 5–10%	0.36
Hilly 10–30%	0.42
Cultivated	
Flat 0–5%	0.50
Rolling 5–10%	0.60
Hilly 10–30%	0.72

Source: Adapted from Anon., 1969a; Schwab et al., 1962.

Time of concentration t_c, is the length of time for runoff from the hydraulically most remote point in the watershed to reach the watershed outlet. Several methods exist to estimate t_c, such as the SCS upland method (Anon., 1973), kinematic wave analogy (Ragan and Duru, 1972), and Kirpich (1940). The Ragan and Duru equation is

$$t_c = C_1\ (nL)^{0.6}/(i_e^{0.4}S^{0.3}) \tag{12.1.2}$$

where t_c is time in minutes, n is Manning's N, L is length in ft (m), i_e is rainfall excess intensity in in./hr (mm/h), S is slope in ft/ft (m/m), and C_1 = 0.928 in English units (C_1 = 1.977 in SI units). Note the equation is valid only when the product of i_e and L is greater than 500 ft (152 m). The Kirpich equation for estimating t_c is

$$t_c = C_2\ L_1^{0.77}\ (L_1/H)^{0.385} \tag{12.1.3}$$

where L_1 is the maximum flow length in ft (m), H is elevation difference in ft (m) between the hydraulically most remote point in the watershed and the watershed outlet, and C_2 = 0.0078 in English units (C_2 = 0.0195 in SI units).

12.1.3.4 Storm Hydrograph Development

A *storm hydrograph* is a graph of flow rate vs. time. It is necessary for evaluating alternative designs of basins, since routing of incremental storm volumes is used in determining peak

Table 12.1.3. Typical Topsoil *K* Factors for the USLE

Texture	Estimated *K* value (ton/ac/*R* unit)	(Mg/ha/*R* unit)
Clay, clay loam, loam, silty clay	0.32	0.72
Fine sandy loam, loamy very fine sand, sandy loam	0.24	0.54
Loamy fine sand, loamy sand	0.17	0.38
Sand	0.15	0.34
Silt loam, silty clay loam, very fine sandy loam	0.37	0.83

Source: Adapted from Anon., 1978.

stage, flow through principal and emergency spillways, detention time, and sediment trap efficiency. A description of hydrograph developmental methods is beyond the scope of this chapter, but many excellent texts provide detailed methodologies and illustrative examples (Barfield et al., 1981; Linsley et al., 1975; Overton and Meadows, 1976). Also computer programs are available that greatly simplify hydrograph development: SEDCAD+—Sediment, Erosion, Discharge by Computer-Aided Design (Warner and Schwab, 1990); SEDIMOT II (Wilson et al., 1982; Warner et al., 1982); TR-55 (Anon., 1986b); and HEC-1 (Anon., 1970).

12.1.4 SEDIMENTOLOGY

12.1.4.1 Quantity of Sediment Determination

The quantity of sediment eroded from a slope depends on the erosive power of rainfall, soil characteristics, slope length and gradient, and type and amount of soil cover and conservation practices. These parameters have been combined into the *universal soil loss equation* (USLE) (Wischmeier and Smith, 1965, 1978):

$$A = R\ K\ LS\ C\ P \qquad (12.1.4)$$

where *A* is soil loss per unit area in tons/ac (Mg/ha or metric tons/ha); *R* is a rainfall factor; *K* is soil erodibility in tons/ac/*R* unit (Mg/ha/R units); *LS* is a dimensionless length-slope factor that accounts for the actual length and slope compared to a standard slope of 9% and 72.6 ft (22.1 m) in length; *C* is a factor that accounts for the effectiveness of vegetal cover, mulches, etc.; and *P* is a factor that accounts for the effectiveness of conservation practices such as terraces. Values for these parameters may be found in Wischmier and Smith (1978).

A prediction of soil loss from a regraded spoil that has been recently topsoiled, seeded, and mulched at 2 tons/ac (4.5 Mg/ha) will be used to illustrate the applicability of the USLE. The design 10 yr–24 hr storm for western Kentucky (Hershfield, 1961) is 4.2 in. (107 mm). Assuming an SCS Type II rainfall distribution and applying Eq. 12.1.5 (Barfield et al., 1981),

$$R_{st} = 19.25\ P^{2.2}/D^{0.47} \qquad (12.1.5)$$

$$R_{st} = 2.48\ P^{2.2}/D^{0.47} \qquad (12.1.5a)$$

yields R_{st} of 101.6, where R_{st} is the rainfall factor on a storm basis, *P* is precipitation depth in in. (mm), and *D* is storm duration in hr.

A fine sandy loam topsoil has been used during reclamation. Table 12.1.3 lists estimated soil erodibility *K* values as a function of soil texture. The *K* value for this example topsoil is 0.24 (0.54).

Table 12.1.4. Selected *C* Factors for the USLE

Surface Conditions	*C* Factor
Bare soil, seed bed prepared, racked smooth	0.9
Bare soil, seed bed prepared, rough graded	0.8
Bare soil, compacted by a bulldozer	0.9–1.2
Undisturbed forest with > 90% ground litter and > 75% effective canopy	0.0001–0.001
Undisturbed forest with 75 to 85% ground litter and 40 to 70% effective canopy	0.002–0.004
Undisturbed forest with 40–70% ground litter and 20–25% effective canopy	0.003–0.009
Rangeland/idle land, no appreciable canopy, 0% groundcover	0.45
Rangeland/idle land, no appreciable canopy, 40% ground cover	0.10–0.15
Rangeland/idle land, no appreciable canopy, 80% ground cover	0.01
Rangeland/idle land, 50% tall weed or short brush canopy, 0% ground cover	0.26
Rangeland/idle land, 50% tall weed or short brush canopy, 40% ground cover	0.07–0.11
Rangeland/idle land, 50% tall weed or short brush canopy, 80% ground cover	0.01–0.04
Clear cut woodland, 20% residue on soil surface	0.06–0.44
Clear cut woodland, 60% residue on soil surface	0.05–0.20
Straw mulch at 1 ton/ac (2.26 Mg/ha)	0.1–0.2
Straw mulch at 2 tons/ac (4.52 Mg/ha)	0.02–0.08
Woodchip mulch at 2 tons/ac (4.52 Mg/ha)	0.65
Woodchip mulch at 4 tons/ac (9.04 Mg/ha)	0.4
Woodchip mulch at 8 tons/ac (18.08 Mg/ha)	0.1
Grass cover < 60 days after emergence	0.1–0.4
Grass cover > 60 days after emergence	0.05

Source: Adapted from Wischmeier and Smith, 1978.

The approximate original contour of a representative slope has a gradient of 12% and is 200 ft (70 m) in length. Applying these values to Eq. 12.1.6,

$$LS = (l/72.6)^{m}\ (0.043s^{2} + 0.30s + 0.43)/6.574 \qquad (12.1.6)$$

$$LS = (l/22)^{m}\ (0.043s^{2} + 0.30s + 0.43)/6.574 \qquad (12.1.6a)$$

where λ is slope length in ft (m); *m* is an exponential factor that equals 0.6, 0.5, 0.4, and 0.3 for slopes of > l0%, >4 to l0%, 4%, and < 4%, respectively; and *S* is sin *q*, *q* is the slope angle in degrees; an *LS* value of 2.8 is calculated.

Table 12.1.4 lists selected *C* factors reported in the literature (Barfield et al., 1981). Since the example problem has a regraded topsoil that has been recently seeded and mulched at 2 tons/ac (4.5 Mg/ha), the *C* factor ranges from 0.02 to 0.08. An average value of 0.04 is selected. Since no conservation practices such as terracing were included, P = 1. Applying Eq. 12.1.4 yields

$$A = (101.6)(0.24)(2.8)(.04)(l) = 2.7 \text{ tons/ac}$$

$$(A = (101.6)(0.54)(2.8)(.04)(l) = 6.1 \text{ Mg/ha})$$

If mulch was not applied, and rough-grade bare-soil conditions were predominant, the *C* factor would be 0.8, and the quantity of eroded sediment would be increased by the ratio of *C* factors (that is, 0.8/0.04), or by a factor of 20. Thus 54.6 tons/ac (122 Mg/ha) of eroded sediment would be predicted by the USLE.

The USLE provides an estimate of sediment delivered from a field size area. To determine the sediment yield at some point beyond the base of the field, the eroded sediment needs to be routed to the point of interest. As such, the USLE provides an estimate of the quantity of sediment eroded from a representative area within a relatively homogeneous subwatershed. A delivery ratio may be used to transfer eroded sediment to a point of interest such as a sediment basin (Barfield et al., 1981). A delivery ratio is simply the ratio of the quantity of sediment that reaches a structure to that quantity of sediment predicted by the USLE. It is a rough approximation often based on drainage area. As with hydrology, computer techniques have expanded predictive capabilities well beyond delivery ratio estimates by applying sediment detachment, transport, and deposition methodologies. Popular programs include SEDCAD$^+$ (Warner and Schwab, 1990) and CREAMS (Chemical, Runoff, Erosion from Agricultural Management Systems) (Knisel, 1980).

12.1.5 SEDIMENT CONTROLS

12.1.5.1 Sediment Basins

A wide variety of sediment basins has been used throughout the mining industry. Small elongated basins are often located along benches near the highwall in contour mining. Dugouts, which are primarily excavated basins with 4- to 8-ft (1.2- to 2.4-m) berms, are prevalent in area mining and mountaintop removal. The most common basins consist of a dam embankment 12 to 19 ft (3.6 to 5.8 m) in height with added storage capacity provided through excavation of soils in the pool area. These basins often use trickle tube (a straight pipe through the embankment) or drop inlet principal spillways, and an emergency spillway. The principal spillway of a dugout is often a trapezoidal channel that may also function as the emergency spillway. Contour basins often have trickle tube principal spillways and an emergency spillway.

Sediment basins are used to reduce peak flow and trap sediment. The efficiency of basins to accomplish these two functions is directly related to the basin design with respect to (1) location and gradient of inflow channel(s); (2) selection of type, size, and location of principal and emergency spillways; (3) length-to-width ratio of the pool measured at the crest of the principal spillway; (4) basin shape; (5) percentage of basin capacity allocated to sediment and permanent pool storage; (6) dewatering method(s); (7) detention storage; (8) inlet baffles, turbidity curtains, and internal check dams; (9) use of flocculation additives; etc. As can be seen from this list of engineering considerations, the performance of a sediment basin in reducing the incoming peak storm flow and retaining sediment is directly achieved through design rigor.

The objective of minimizing the hydrologic impact can be assured if the peak discharge emanating from the basin is reduced to near premining conditions and a significant percentage of the incoming sediment load is captured. To achieve this objective, consider the application of each of the design factors as explained in Table 12.1.5.

A cost-effective sediment basin design that is expected to minimize the hydrologic impact to offsite streams is illustrated in Figs. 12.1.1 through 12.1.5. (Warner and Schwab, 1989). The function of the identified features in these figures is described. The length-to-width ratio is approximately 2:1. It may be smaller than the recommended ratio due to the internal check dam. Incoming flow energy is dissipated by the rock impact pool at the basin entrance. The internal check dam is used to accomplish many functions. It further slows and spreads out the incoming flow by rapidly creating a pool of water in the primary chamber. The larger-sized sediments rapidly settle out in this forward chamber, which, in combination with the access road and/or a check dam wide enough for a backhoe, facilitates sediment cleanout.

Small storm runoff is contained in the primary chamber and is slowly dewatered by either a small drop-inlet perforated riser and/or a wide rock riprap French drain (see Figs. 12.1.2 and 12.1.3). As the primary chamber fills with the larger-sized sediment, it also acts as a slow sand filter that greatly increases the quality of discharged water. For larger storms, short circuiting and dead storage are substantially reduced since the internal check dam evenly distributes flow across the width of pool. The check dam is protected by a geotextile fabric and rock riprap or equivalent.

Optional principal spillway configurations are shown in Fig. 12.1.1 and detailed in Figs. 12.1.4 and 12.1.5. The drop-inlet perforated riser should be sized to reduce peak flow to near premining conditions. The tapered riser perforations slowly dewater the storm pool, thereby providing additional storage for subsequent storms. This also reduces the peak stage, compared to the permanent pool option, thus reducing embankment construction cost. Dewatering reduces peak discharge, increases sediment trap efficiency with respect to total sediment load, and reduces peak and average effluent sediment concentrations. The floating siphon tube (Fig. 12.1.5) provides an added advantage by dewatering only the uppermost water level, thus increasing sediment trap efficiency. Stationary or floating siphon tubes can be used in conjunction with either trickle tubes or drop-inlet principal spillways (Warner and Schwab, 1989, 1990).

Impact pools with depressed outlet channels dissipate the energy of concentrated pipe discharge to reduce downstream impacts (Warner and Schwab, 1990).

Design of stormwater retention basins is facilitated by using computer programs such as DAMS2 that routes water through principal and emergency spillways (Putnam et al., 1982; Anon., 1965). Principal and emergency spillway designs, stormwater and sediment routing, inflow and outflow hydrographs and sedimentgraphs, dewatering options, impact pools, sediment trap efficiency, embankment earthwork volumes, etc., can be calculated through use of the SEDCAD$^+$-Version 3.0 Model (Warner and Schwab, 1990).

12.1.5.2 Infiltration Basins

An infiltration basin has no direct discharge for surface waters and is used to recharge an aquifer constructed during mountaintop or area mining (Dinger et al., 1988). A vertical durable rock core is constructed during spoil placement to facilitate the rapid downward movement of stormwater. Infiltration basins have the advantages of recharging groundwater, complete containment of stormwater and sediment, relatively simple construction, avoidance of potential embankment stability problems, and allowance of a large degree of flexibility in a spatial siting.

12.1.5.3 Sediment Basin Hydraulics

Design aspects of basin hydraulics encompass (1) principal spillway design, (2) emergency spillway design, (3) pond stage-area-capacity, and (4) routing the inflow hydrograph. The basic steps in developing an outflow hydrograph for a sediment basin are listed in Table 12.1.6.

12.1.6 SEDIMENT SEPARATORS

The swirl concentrator and inertial separator have both been designed to intercept sediment-laden inflow. The swirl concentrator separates sediment from the incoming flow through the

Table 12.15 Application of Sediment Basin Design Principles

Design Consideration	Implementation
1. Location of inflow channels	Locate inlet(s) to provide the hydraulically longest flow path between inlet(s) and outlet(s). This will help reduce short circuiting of flow.
2. Inlet flow channel gradient	Provide a wide, shallow slope entrance. Stabilize steeper slopes with rock riprap or equivalent protection to avoid severe gully formation.
3. Size of principal spillway	Size to significantly reduce the incoming peak flow to near premining conditions.
4. Location of principal spillway	Depending on size, and associated flow rate, locate a minimum of at least 2 to 3 ft (0.6 to 0.9 m) above the sediment storage level. Also, locate farthest point away from inlet(s).
5. Pool length to width ratio (L/W)	The L/W ratio measured at the crest of the principal spillway should be 2:1 or greater.
6. Basin shape	Provide a L/W ratio of 2:1 or greater and the basin deep enough such that resuspension of settled sediment is avoided. Refer to design considerations 4 and 5.
7. Sediment storage	Base sediment storage needs depend on (1) estimated loads for at least the life of the project or (2) for the expected sediment load associated with the 10-yr, 24-hr design storm. Sediment storage volumes, based on project life, may be reduced dependent upon a sediment approval plan.
8. Permanent pool storage	Both advantages and disadvantages exist for permanent pools in contrast to a dewatered or partially dewatered pool. The major advantage is dilution of the incoming sediment concentration. Disadvantages are (1) higher peak flow discharged and greater peak stage, (2) increased fall depth of incoming sediment prior to retention in the sediment storage area, (3) shorter detention time, and (4) discharge of warm, low-oxygen water during summer months.
9. Pool dewatering (gated risers, perforated risers, stationary and floating siphon tubes, small trickle tubes)	Dewatering provides numerous advantages when compared to the permanent pool option—(1) lower peak discharge, (2) lower storage volume, (3) higher sediment trap efficiency.
10. Detention storage (time difference between inflow and outflow hydrograph peaks or hydrograph centroids.)	Detention storage is an indicator of basin performance. Longer detention times indicate reduced peak flow and poentially more efficient sediment trapping.
11. Inlet baffles, turbidity curtains, and internal check dams	All of these devices increase the flow path between inlet(s) and outlet(s). In addition, they decrease dead storage which improves sediment trap efficiency.
12. Flocculation additives	Long chain polymers create flocs, a group of soil particles, which settle at rates significantly faster than individual soil particles, thus increasing sediment trap efficiency.
13. Outlet stabilization	The outlet of principal and emergency spillways should be stabilized by energy dissipators such as impact pools constructed of rock riprap, gabions, or concrete-slurry-filled geotextiles to avoid downstream bed scour.

centrifugal force generated by the inherent inertia of the flow (Warner and Dysart, 1983). Effluent with a high sediment load is transmitted to a small sediment trap while the clearer flow is discharged directly to a stream. Fig. 12.1.6 shows a schematic of the swirl concentrator. The inertial separator is designed to retain incoming sediment in a tank between a series of V-cut bottom-slotted troughs (Sterling and Warner, 1984). The sediment-laden influent is transferred from shallow influent troughs into the tank, and clearer flow is withdrawn by deep troughs. Plan and elevation cross section of the inertial separator are shown in Fig. 12.1.7.

The numerous advantages of these devices call attention to their use as primary sediment control structures on surface-mined lands. Both devices are relatively simple, containing no moving mechanical parts, and no onsite energy requirements are needed. Since the devices are relatively small, they can be easily transported on flat-bed trucks and modularized to facilitate rapid installation and relocation once reclamation and bond release have been accomplished. The added advantage of the swirl concentrator is that no sediment maintenance is needed since it is self-cleaning; that is, the concentrated sediment-laden flow, which represents 5 to 15% of the incoming flow, is automatically discharged to a small sediment basin.

12.1.7 DIVERSIONS

Diversions are designed to be stable and convey a specified peak flow. Stability is usually interpreted as avoidance of significant erosion of the channel bed or sidewalls. Diversions can be classified as temporary or permanent structures. Temporary diversions may be constructed of soil and/or a soil/spoil mixture, and protected by straw or nylon blankets, geotextiles, and flexible membrane liners. For these diversions, the determination of stability is based on either a permissible velocity or a critical tractive force, the assumption being that if the specified velocity or tractive force is not exceeded, the diversion will remain stable. Conveyance capacity is calculated using $Q = VA$ and Manning's

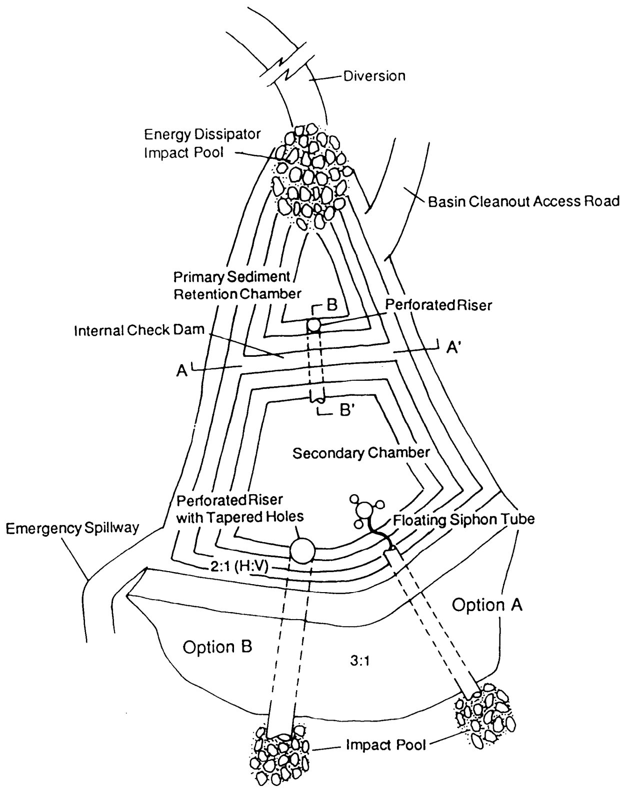

Fig. 12.1.1. Efficient and cost-effective sediment basins.

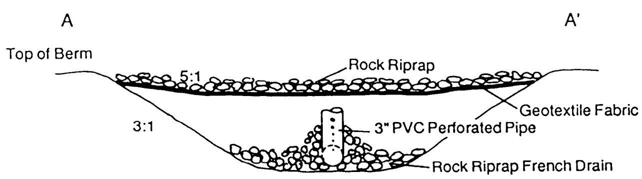

Fig. 12.1.2. Plan view (cross section A-A[1]) of dewatering type of internal check dam. Conversion factor: 1 in. = 25.4 mm.

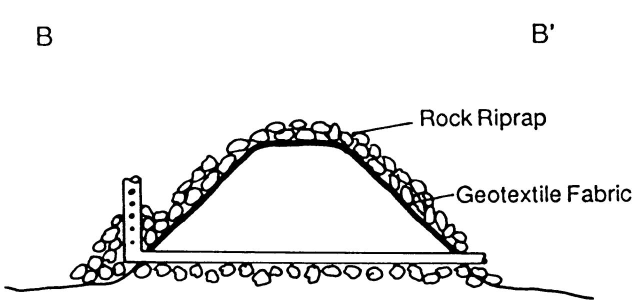

Fig. 12.1.3. Cross section B-B[1] of dewatering type of internal check dam.

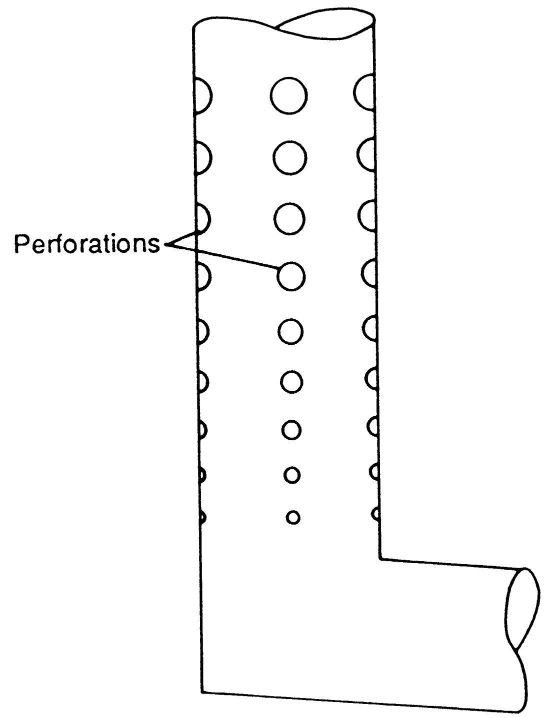

Fig. 12.1.4. Schematic of perforated riser with tapered holes for dewatering.

equation (12.1.7) where Q is discharge in cfs (m^3/s), V is the average channel velocity in fps (m/s), and A is the cross-sectional area in ft^2 (m^2), and Manning's equation is

$$V = 1.49\, R^{2/3}\, S^{1/2}/n \tag{12.1.7}$$

$$V = R^{2/3} S^{1/2}/n \tag{12.1.7a}$$

where R is the hydraulic radius in ft (m), which is the cross-sectional area divided by the wetted perimeter, and s is the hydraulic gradient (channel slope) in ft/ft (m/m).

Permanent diversions are often constructed of rock riprap, gabion mattresses, grass, asphalt, concrete, concrete slurry-filled, double-layer geotextile forms, and interlocking concrete blocks. Stability of these products is often related to discharge, diversion gradient, or velocity (Maccaferri, undated; Anon., 1989; Anon., 1982.) Rock riprap stability is based on selecting riprap that is large enough such that the gravitational force exceeds the overturning forces (Anon., 1982). Also an underlying rock filter, consisting of a range of smaller rocks, is required to avoid erosion of the base soil. The design of grassed waterways is a function of grass type and height, permissible velocity, slope, Manning's n, and hydraulic radius. Use of retardance classes simplifies the design process (Anon., 1969b). Computer programs, such as SEDCAD$^+$ Version 3.0 (Warner and Schwab, 1990), simplify design procedures.

12.1.8 CULVERTS

The selection of a properly sized culvert is based on design discharge, type of culvert, entrance configuration, pipe length and slope, headwater constraints, and tailwater condition (Fig. 12.1.8). Two of three primary design parameters must be known.

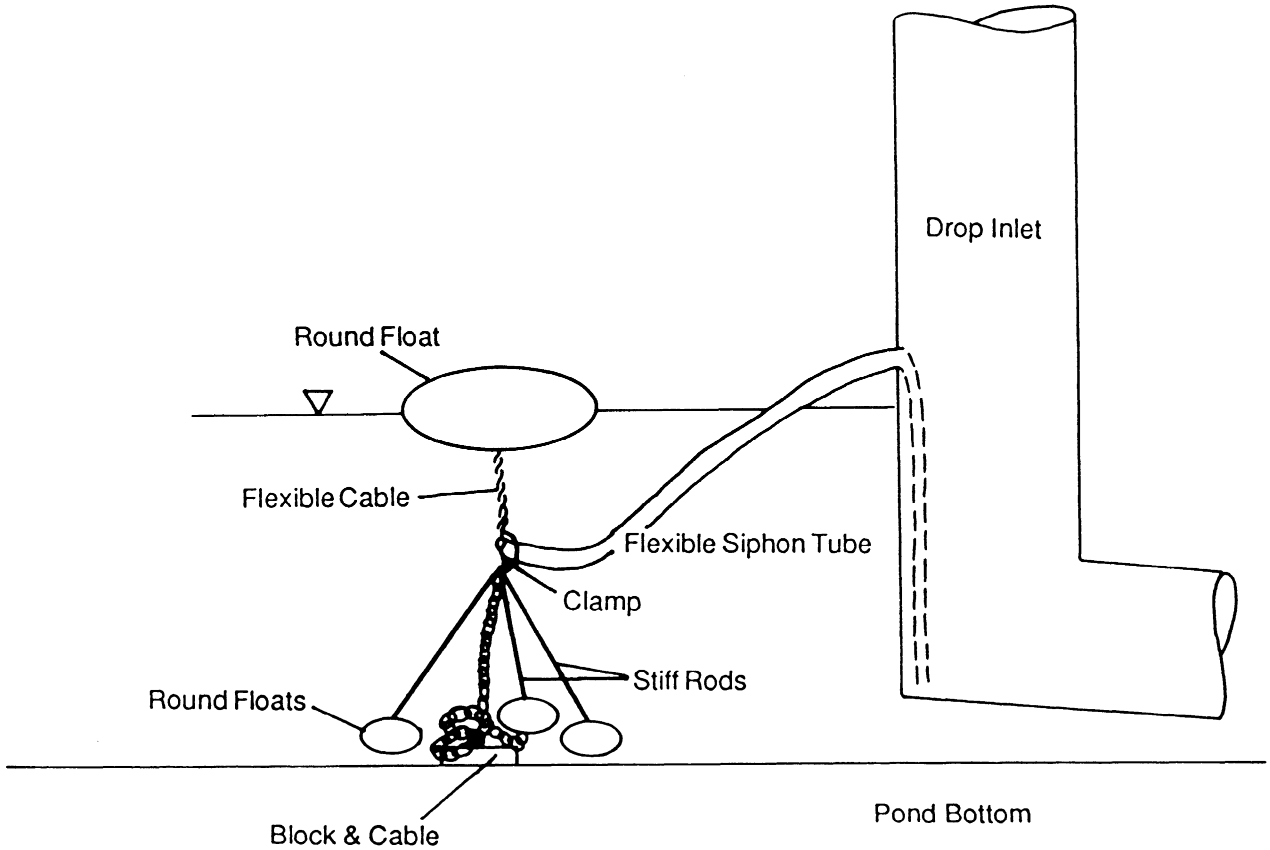

Fig. 12.1.5. Schematic of floating siphon tube used for dewatering.

The primary parameters are (1) discharge, (2) pipe size, and (3) headwater. Thus, if discharge is known, the headwater can be determined for a specific size of culvert. Depending on these parameters, the culvert may function with inlet, outlet, or full-pipe flow control.

Design procedures have been somewhat simplified through the development of culvert capacity charts (Anon., 1962). Such a chart is illustrated in Fig. 12.1.9. Headwater values are indicated on this chart on the ordinate and discharge values on the abscissa. The solid curves designate inlet control and the dashed curves indicate outlet control. Also listed is an index value I, defined as

$$I = \frac{L}{100S_o} \tag{12.1.8}$$

$$L = \frac{3.28L}{100\ S_o} \tag{12.1.8a}$$

where L is pipe length in ft (m), and S_o, is pipe gradient in ft/ft (m/m).

Example 12.1.1. Use of the culvert capacity chart is illustrated by a design example. Inputs are (1) a design discharge of 70 cfs (2m^3/s), (2) concrete culvert, (3) grooved-edge entrance, (4) culvert length of 200 ft (61 m), and (5) a slope of 0.02 and 0.002 ft/ft (m/m), respectively.

Solution. The solution procedure begins with the calculation of the index value. When the calculated index value is less than or equal to the index value of a specified culvert, the culvert will be in inlet control, and the solid line will describe culvert performance. Applying Eq. 12.1.8 to the design input values mentioned above and using a 0.02-ft/ft (m/m) slope yields an I of 100. A vertical line at 70 cfs (2 m^3/s) intersects the solid 300 index value for the 36-in. (910-mm) culvert (Fig. 12.1.9). Since the calculated index value is less than the 300 solid line, inlet control is specified, and the headwater height of 5.2 ft (1.6 m) is determined by extending a horizontal line to the ordinate. In a similar manner for the 0.002 ft/ft (m/m) culvert gradient, an index value I of 1000 is calculated. Since the calculated index value lies between the solid and dashed curve of the 36-in. (9l-cm) culvert, outlet control is specified. Referring to Fig. 12.1.9, a headwater of 6.1 ft (1.9 m) is determined from the interpolated index value. A calculated index value greater than the dotted curve indicates full pipe flow, and an applicable nomograph should be used. A detailed analysis of culvert design procedures is available (Normann et al., 1985).

The user of these culvert analysis charts should be cautioned that neither the open-channel flow regime nor submerged outlet conditions are accommodated by the culvert capacity charts. Computer programs for culvert designs are available (Anon., 1986; Warner and Schwab, 1990). The SEDCAD$^+$ Version 3.0 program provides complete performance curves of headwater versus discharge for all culvert flow regimes.

12.1.9 PUMPS

Pumps are widely used throughout the mining industry for dewatering deep mine operations, an active pit, and predraining

Table 12.1.6. Development of an Outflow Hydrograph for a Sediment Basin

Item	Method
Principal spillway relationship	1. Drop-inlet riser stage-discharge (Barfield et al., 1981) —Weir flow equation —Orifice flow equation —Pipe flow equation 2. Trickle tube Interactive solution to mild and steep slopes with submerged or unsubmerged outlets (Warner and Schwab, 1990) 3. Perforated riser (Warner and Schwab, 1990) —Step-function orifice flow equation 4. Siphon tubes (Warner and Schwab, 1989) —Pipe flow equation —Siphon flow equation
Emergency spillway	1. Broad crested weir with backflow analysis (Putnam et al., 1982)
Stage-area-volume	1. Stage-area from contour map 2. Volume from prismoidal rule
Routing	1. For each time increment, Δt a. determine volume of inflow b. from the stage-volume curve, determine the stage associated with the increase in inflow volume c. from the stage-discharge curve, determine the volume of outflow

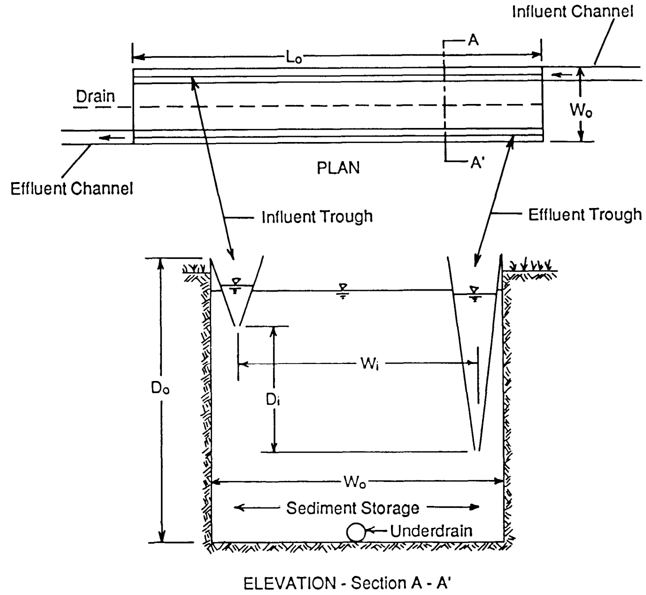

Fig. 12.1.7. Inertial separator.

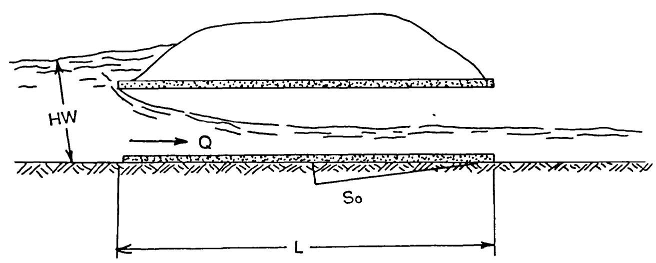

Fig. 12.1.8. Culvert input design parameters.

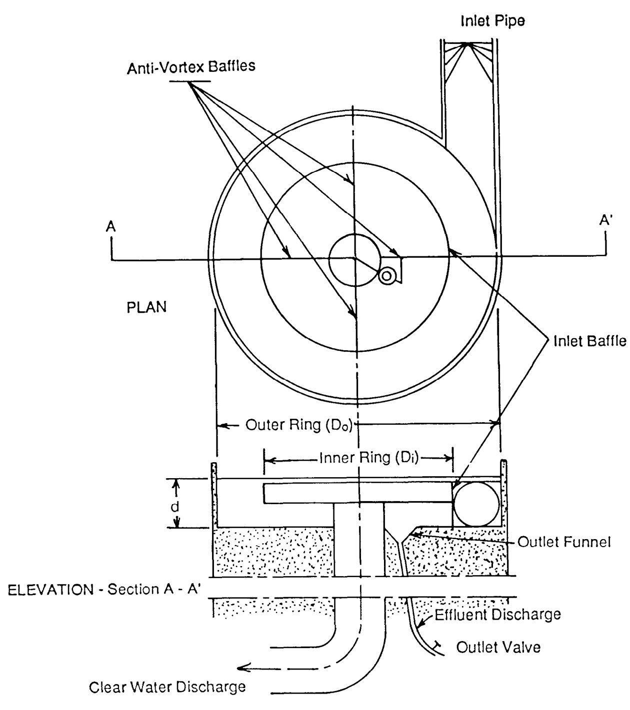

Fig. 12.1.6. Swirl concentrator.

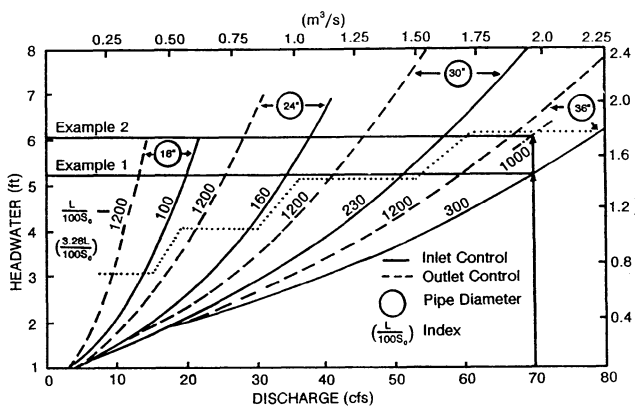

Fig. 12.1.9. Culvert design capacity chart.

adjacent areas prior to pit excavation. Control of groundwater and surface water can alleviate numerous production problems and provide safer working conditions. For example, a series of wells installed adjacent to an area to be mined can decrease groundwater migration, reduce construction of large sumps, and enhance slope stability.

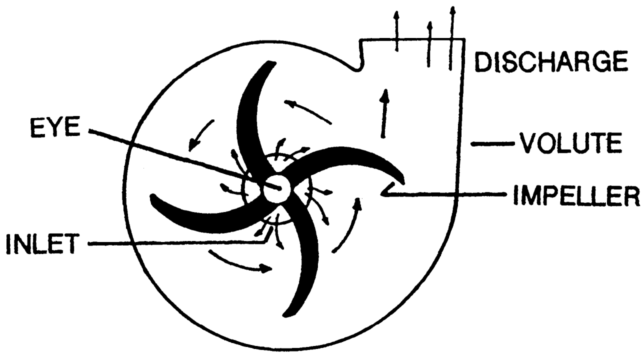

Fig. 12.1.10. Centrifugal pump components.

12.1.9.1 Basic Principles of a Centrifugal Pump

Centrifugal pumps are probably the most popular type in mining (Bise, 1986). The basic components of a centrifugal pump as illustrated in Fig. 12.1.10 are an inlet, the eye of the rotating impeller, curved impeller vanes, volute, and discharge connection. A centrifugal pump is a simple device with the only moving part being the impeller which is attached to the shaft of a motor or engine. Assuming that the suction pipe and pump housing (volute) is filled with water to the level of the eye (i.e. primed), as the impellers rotate, a partial vacuum is created that allows atmospheric pressure to lift more water into the pump housing. The water, which entered the inlet opening at the eye of the impeller, is set in rotation by the impeller and creates centrifugal force resulting in pressure at the outer perimeter of the impeller. Water moves outward from the impeller at a high velocity and pressure into an expanding volute and is discharged.

A single-stage centrifugal pump, that is, a single impeller, usually operate against low to moderate heads. For heads greater than approximately 250 ft (75 m), multistage pumps are generally used. A multistage centrifugal pump consists of two or more stages and is essentially a high-head pump. The single-stage pump is normally used for high volume and relatively low head.

12.1.9.2 Calculating Dynamic Head

The essential information needed to properly select a pump is the required flow rate in gpm (L/s) and the total dynamic head in ft (m). Flow rate is dependent primarily on operational factors such as the anticipated rate of flow into a pit or underground mine. Determination of the total dynamic head requires consideration of suction lift, elevation difference between the pump and discharge point, pressure requirements, if any, at the discharge outlet, friction along pipes and fittings, and velocity head.

Dynamic lift of water at the suction side is calculated as the required pressure needed to raise water from the water surface to the height of the centerline of the pump and to overcome frictional resistance in the intake pipe and fittings. Twenty-three ft (6.7 m) is considered the practical limit of suction lift. This is further reduced by long or undersized intake pipes, a large number of fittings, higher mine site elevation, and warmer waters. Table 12.1.7 contains the practical static suction lift as a function of site elevation and temperature.

Friction loss in pipes is a function of pipe diameter and length, type of pipe material, and flow rate. The Hazen-Williams equation is often used to calculate friction loss:

$$H_{f100} = \frac{K(Q/C)^{1.852}}{D^{4.87}} \tag{12.1.9}$$

where H_{f100} is friction loss per 100 ft (100 m) of pipe; K is the conversion constant, 1045 with English units and 1.22×10^{12} with SI units; Q is flow rate in gpm (L/s); C is retardation coefficient = 120 for coated steel and = 150 for *PVC*; and D is inside pipe diameter in in. (mm).

Friction loss in fittings is usually approximated by the equivalent pipe length (Table 12.1.8).

The velocity head h_v, in ft (m) is usually small and may often be neglected. It can be calculated using the following relation:

$$h_v = \frac{V^2}{2g} \tag{12.1.10}$$

where V is velocity of liquid in fps = $0.408 \times \text{gpm}/D^2$ (English units) and = $1273 \times L/s/D^2$ (SI units), D is pipe diameter in in. (mm), and g is acceleration due to gravity = 32.2 fps^2 (9.81 m/s^2).

Example 12.1.2. Consider a pump that is to be located 4 ft (1.2 m) above the water level of a sump. The mine is at 6000 ft (1830 m) and the water temperature is not expected to exceed 70°F (21°C). A flow rate of 900 gpm (56.8 *L/s)* is required. A 6-in. (152-mm), 40-ft (12.2-m) long suction line was specified with two 45°, 6-in. (152-mm) ells, a gate valve, a vacuum gage installed on a tee, and a check valve. Will the total practical lift of the pump be exceeded?

Solution. Total Dynamic Suction Lift = Lift Height + Friction Head + Velocity Head. Lift height is given as 4 ft (1.2 m). The equivalent pipe length of fittings totals 153.5 ft (46.8 m). Pipe friction head is calculated for the 40 ft (12.2 m) section from Eq. 12.1.9 as 7.1 ft/100 ft of pipe. For the total equivalent pipe length of 193.5 ft (59 m), the friction head loss is 13.7 ft (4.2 m).

Flow velocity is approximately 10.2 fps (3.1 m/s), and the velocity head is 1.6 ft (0.5 m) (Eq. 12.1.10). Thus the total dynamic suction lift is 4 + 13.7 + 1.6 = 19.3 ft (5.8 m). The practical lift at 6000 ft (1830 m) for water at 70°F (21°C) from Table 12.1.7 is 18.3 ft (5.6 m). Thus the practical lift of the pump approximated in Table 12.1.7, will be slightly exceeded, causing pump cavitation and a drop in pump performance.

Pump manufacturers provide either static suction lift or net positive suction head (NPSH) curves for each pump model over the entire operating range. The NPSH subtracted from the values listed in Table 12.1.7 provides an estimate of the maximum allowable dynamic suction lift for a given model at a specified operating point. Selection of a more efficient check valve, reduction of intake pipe length, lowering the pump closer to the water, eliminating fittings, or increasing the suction pipe size will reduce the total dynamic suction head. It should be noted that improper design and installation of the suction line are primary causes of poor pump performance.

In a similar manner to suction head calculations, the total dynamic pumping head must be calculated prior to pump selection. The total dynamic head is the sum of (1) suction head, (2) the elevation difference between the center line of the pump and the discharge point, (2) friction loss in pipes and fittings, (3) discharge pressure (zero if atmospheric discharge), and (4) velocity head.

Example 12.1.3. Assume that a 900-gpm (56.8-L/s) discharge is required to dewater a pit. The elevation difference between the pump and discharge point is 70 ft (21.3 m) and a 400-ft (122-m), 6-in. (152-mm) PVC pipe will be used to convey the pumped water. No fittings are needed.

Table 12.1.7. Practical Suction Lift[1]

Elevation ft	(m)	50°F ft	(10°C) (m)	70°F ft	(21°C) (m)	90°F ft	(32°C) (m)
0	0	23.8	7.2	23.2	7.1	22.6	6.9
1000	305	22.9	7.0	22.4	6.8	21.8	6.6
2000	610	22.0	6.7	21.5	6.6	20.9	6.4
4000	1220	20.4	6.2	19.9	6.1	19.3	5.9
6000	1830	18.9	5.8	18.3	5.6	17.8	5.4
8000	2440	17.5	5.3	16.9	5.2	16.4	5.0

[1]70% of theoretical siphon lift.

Table 12.1.8. Equivalent Length of Pipe, ft

	Nominal pipe size, in. (mm)				
	1 (25)	2 (50)	3 (75)	4 (100)	6 (150)
90° ell	2.7	5.5	8.0	11.0	16.0
45° ell	1.3	2.5	4.0	5.0	8.0
Tee	6.0	12.0	17.0	22.0	33.0
Globe valve, open	13.0	29.0	45.0	60.0	110.0
Gate valve, open	0.6	1.3	2.0	2.5	4.5
Check valve	8.0	19.0	32.0	43.0	100.0

Conversion factor: 1 ft = 0.3048 m.
Source: Adopted from Bise, 1986.

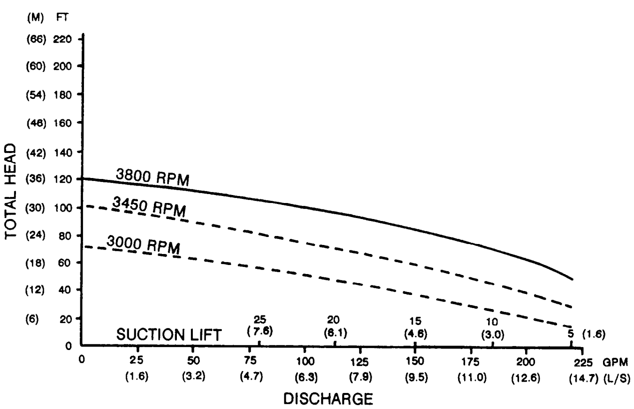

Fig. 12.1.11. Pump characteristic curves for engine-driven pump. (Modified from Anon., 1966.)

Solution. Applying Eq. 12.1.9 yields a friction loss H_f on the discharge side of 4.7 ft (1.4 m). The total dynamic head is the sum of the dynamic suction head 19.3 ft (5.8 m), elevation difference 70 ft (21.3 m), and discharge friction loss 4.7 ft (1.4 m), which yields 94 ft (28.6 m). Thus the selected pump must have the capacity to pump 900 gpm (56.8 L/s) at a total dynamic head of 94 ft (28.6 m), and a dynamic suction lift at this operating point of 19.3 ft (5.8 m).

12.1.9.3 Pump Selection

A pump characteristic curve shows the relationship among discharge in gpm (L/s), total dynamic head in ft (m), and either suction lift or NPSH in ft (m). A representative pump characteristic curve is illustrated in Fig. 12.1.11. Along the vertical axis, the amount of head that will be developed, expressed in ft (m), is shown. The x axis contains discharge in gpm (L/s). The lines marked 5 (1.5), 10 (3.0), 15 (4.6), 20 (6.1), and 25 ft (7.6 m) indicate the maximum flow in gpm (L/s) the pump is capable of delivering at designated suction lifts. To read the curve, start at either the x or y axis.

Example 12.1.4. Assume 200 gpm (12.6 *L/s*) is needed.

Solution. Follow across the bottom scale until 200 gpm (12.6 *L/s*) is reached; then proceed up this line until the 3800-rpm line is intersected. Reading across, it is found that the pump is capable of discharging 200 gpm (12.6 *L/s*) against a total head of 65 ft (19.8 m), provided the suction lift is no more than 7 ft (2.1 m).

Example 12.1.5. Similarly, if a total head of 80 ft (24.4 m) is required, proceed horizontally across to the 3800-rpm line and then vertically to the gpm (L/s) axis to read 160 gpm (10.1 *L/s*). If the suction lift is 15 ft (4.6 m), the maximum discharge is 150 gpm (9.5 *L/s*).

Pump characteristic curves for an electric-motor-driven pump contain additional information on pump speed, efficiency, horsepower, and suction head. Motors operate at a constant speed. Their rpm cannot be varied like a gasoline or diesel engine. A V-belt pulley arrangement is used to adjust a motor to the desired operating speed.

Example 12.1.6. (a) For a specified operating rpm, (e.g., 1750) the brake horsepower (brake watts) can be directly determined from Fig. 12.1.12. A ½-hp (0.4-kW) motor is needed. At 3450 rpm, a 3-hp (2.3-kW) motor is required since the 3450-rpm line is between the 2 and 3 hp (1.5 2.3 kW) scale. Pump efficiency is indicated by the U-shape curves, e.g., 56, 50, 45%. Brake horsepower can be calculated from Eq. 12.1.11:

$$BHP = \frac{Q \times H}{3960 \times \text{Eff}} \quad (12.1.11)$$

$$\text{brake watts} = \frac{Q \times H}{6.2 \times \text{Eff}} \quad (12.1.11a)$$

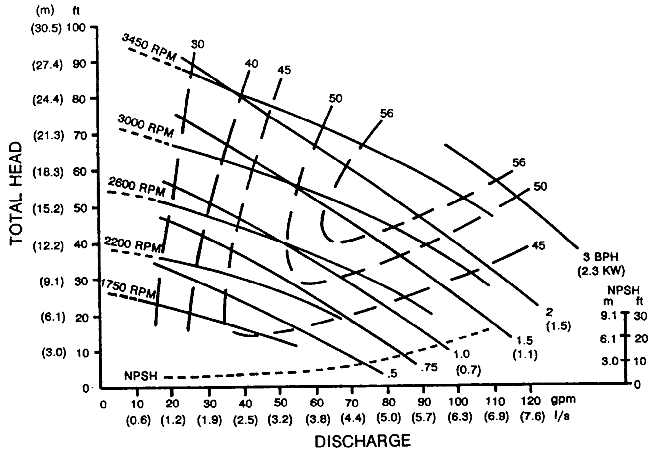

Fig. 12.1.12. Pump characteristic curves for electric-motor-driven pump. (Modified from Anon., 1966.)

where Q is flow rate in gpm $(\mathrm{L/min})$, H is total dynamic head in ft (m), and Eff is pump efficiency expressed as a decimal.

(b) The required net positive suction head (NPSH) is the minimum net energy required by the pump at the eye of the impeller to avoid cavitation and reduce pump performance. NPSH is a function of pump design. If the total dynamic suction lift (i.e., static suction lift plus friction loss and velocity head) exceeds the NPSH required by the pump, then cavitation will occur (Anon., 1983).

At a flow rate of 60 gpm (3.8 L/s), the NPSH, read from the lower dashed line and the NPSH axis, is 6 ft (1.8 m). The total dynamic suction lift is determined by subtracting the NPSH from the appropriate value listed in Table 12.1.7. For example, at 1600 ft (305 m) and 70°F (21°C), a 22.4-ft (6.8-m) practical suction lift is needed. Thus, for the above values, the total dynamic suction lift must be less than 16.4 ft (5.0 m), i.e., 22.4 to –6 ft (6.8 to –1.8 m).

The total head and pump discharge can be changed by placing pumps in series or parallel or by changing the operating speed. To plot characteristic curves for parallel and series pumping installations, it is required to secure each pump's characteristic curve. For parallel pump characteristic curves, the flow rates are added at a given head; for series pump characteristic curves, the heads are added at a given flow rate. As the pump speed is increased, (1) the discharge is proportionally increased, (2) head varies as the square of the RPM ratio, and (3) BHP (brake watts) varies as the cube of the RPM ratio. Refer to Bise (1986) for detailed examples.

12.1.9.4 Pump Troubleshooting

A properly designed and installed pump will reduce potential operational problems. A vacuum gage, installed a minimum of one pipe diameter from the pump inlet, can solve the majority of problems. A high gage reading indicates a partially blocked suction line, whereas a low reading often indicates air leakage through fittings on the suction side. A vacuum gage can easily detect common problems such as a non-primed pump; too high a dynamic lift; excessive air in the water due to the intake being near the water surface; air leakage in the inlet pipe, fittings, or stuffing box; or the strainer, foot valve, or suction pipe is too small or restricted by debris. If a discharge pressure gage reads too low, the pump may not be primed, rpms too low, total dynamic head is too high, impeller rotating in the wrong direction, excessive air in water, mechanical defects, etc. Excess pump vibration may indicate misalignment, worn bearings, bent shaft, or an obstruction has lodged in one side of the impeller.

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Chapter 12.2
WASTE DISPOSAL AND CONTAMINANT CONTROL

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12.2.1 INTRODUCTION

Waste disposal is the handling, transporting, and placing of mine wastes to meet stability and environmental goals by the most economical means. Mine waste includes overburden, waste rock, refuse, mill tailings, slag, process sludge and residues, acid mine drainage, and seepage water. Although technically not waste, low-grade and ore-grade stockpiles and leach heaps and dumps are included here because they must be given the same considerations as waste materials. Some of these materials may reside only temporarily in any one of the above categories, but it is still necessary to design the structures to meet stability and environmental goals.

Over the last few decades, construction methods have been developed for building structurally stable impoundments, waste storage structures, and effluent handling facilities (Vick, 1983; Williams, 1975; Argall, 1978), but few publications address both environmental and structural aspects of waste management comprehensively (Legge et al., 1982; Anon., 1989a; Hood et al., 1988). Design goals are quickly changing to meet the demands of increasingly stringent environmental regulations (Anon., 1985a), a topic that is covered in detail in Chapters 3.4 and 7.3 of this *Handbook.* The requirements to meet these environmental regulations are so extensive that substantial changes are often necessary in previously acceptable designs or, in some cases, entirely new disposal configurations are needed.

The intent of this chapter is to identify the basic design considerations to meet both traditional structural goals and emerging environmental goals. Low-volume special wastes, such as those classified as hazardous by the Environmental Protection Agency (EPA), and stack effluent are not discussed; neither are detailed methods for addressing many reclamation and environmental issues, even though these issues are integral to waste disposal.

Adverse environmental effects have primarily been the result of contamination of water resources by acids and metals and reclamation practices that were aesthetically unpleasing and that left few options open for future land use. Other less obvious impacts include contamination by process reagents, sediment loading of surface waters, and waste dispersal by wind. Methods to prevent or control these effects are usually the most costly and complex aspect of waste disposal and should be treated accordingly. Opening new operations, permitting ongoing operations, and minimizing future liability for environmental cleanup depend directly on preventing contamination or treating wastes to bring contamination to acceptable levels through appropriate disposal practices.

The enormous potential for liability, the quick evolution of technology and regulations, and the complexity of both site-specific and general factors make it imperative for a design engineer to develop the specific components of a design based on current information from many disciplines. It may be necessary to look to other industries, such as the coal-fired power plants industry, to find information relevant to environmental concerns (Anon., 1986a; Summers et al., 1983). References have been selected to provide the most current sources of information from which a design engineer may begin a search for appropriate information to address specific design constraints.

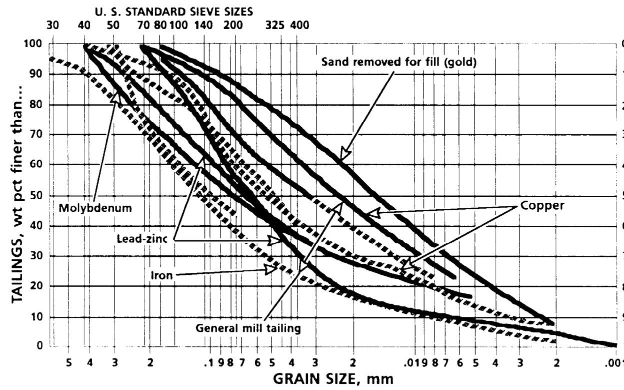

Fig. 12.2.1. Typical grain-size distributions for different types of mine tailings.

12.2.2 COMPOSITION OF WASTES

Solid mine waste usually consists of (1) waste rock from mine development or overburden, (2) stockpiles of low-grade ore or ore with unfavorable milling mineralogy, (3) leach dumps or heaps, and (4) milling waste. It is classified as coarse, fine, or slime. Fig. 12.2.1 illustrates common grain-size gradations for different mine waste types.

12.2.2.1 Waste Rock

Waste rock is usually coarse and is classified as cobbles, rocks, or boulders with associated fines. Disposal options for waste rock include valleyfill, crossvalley, sidehill, ridge, heaped, or diked dumps; embankments; and backfill (Fig. 12.2.2). In a dump, the most frequent concerns are the structural stability of the dump and wind and/or water erosion (Anon., 1972). When waste rock is used as backfill, it may be crushed, ground, blended, and/or chemically modified to give it characteristics that will meet structural and environmental constraints.

12.2.2.2 Fine Waste

Fine mine wastes usually result from milling or processing operations. Typical grain sizes range from 100% > 5 mm for iron ore tailings to 100% < 0.006 mm for phosphate slimes (Fig. 12.2.1). Water is often used to transport these fine materials to the deposition area. The water will usually be released slowly, which requires that special attention to be paid to controlling the level of the phreatic surface, which directly affects structural stability (Center and Zlaten, 1982; Coates and Yu, 1977; Soder-

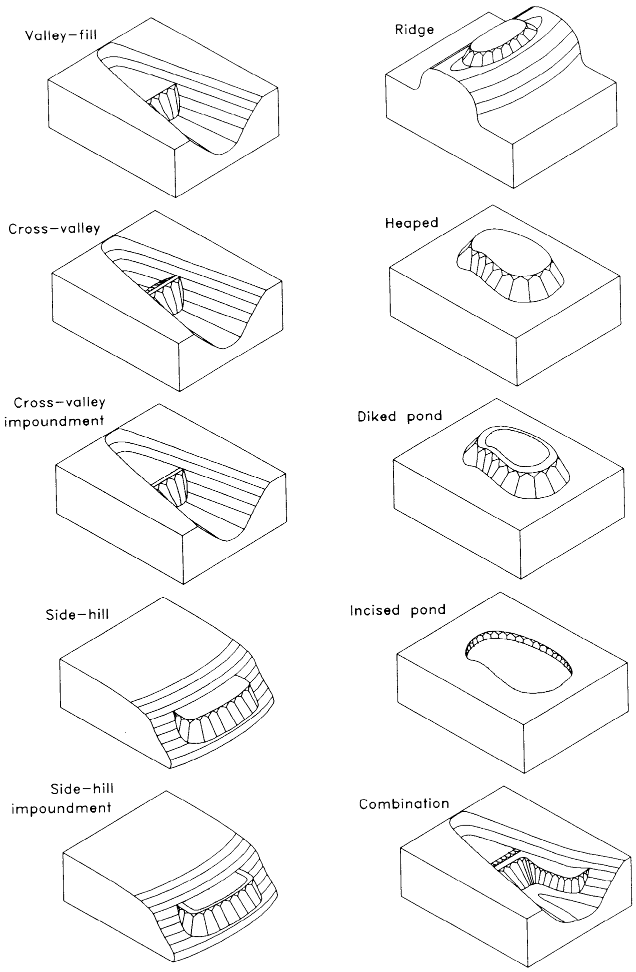

Fig. 12.2.2. Configurations of waste depending on topography.

berg and Busch, 1977). Embankments for fine waste impoundments are constructed of coarse waste or borrow materials or from the coarser fraction of fine materials.

Slimes are the smallest particles that make up fine waste. They are derived from both ore and waste rock and include clay, alumina, hydrated iron, near-colloidal common earths, and weathered feldspars. Slimes are characterized by low permeabilities that must be considered when designing for water control.

Sludges and slimes settle so slowly and incompletely that they typically remain in a liquid or easily reliquefied state. Tailings pond dike failure thereby constitutes a significant environmental threat, and the costs in human life and property damage of such disasters can be enormous.

Dewatering options are many, and choosing the appropriate method for a given site can be an important economic decision for the mine operator. For example, the Florida phosphate industry searched for many years for a viable solution to their ultrafine slimes disposal problem, and finally determined that the most effective method was simple compaction with a specially designed heavy-tired vehicle. Electrokinetic techniques have been shown to work well with responsive coal and metal-mine wastes (Kelsh, 1990). Flocculation, filtration, and use of mechanical settlers are the more traditional dewatering techniques. Cost and effectiveness will vary widely, and preliminary laboratory and pilot-scale tests are strongly recommended before choosing the dewatering method for a specific slurry.

12.2.3 WASTE STRUCTURES AND DISPOSAL METHODS

Planning waste disposal facilities requires determining regulatory constraints, locating an appropriate site, designing the structural and environmental integrity of the facility, developing an operating and maintenance plan, and developing a reclamation plan for future land use (Center and Zlaten, 1982; Ritcey, 1989). Specific factors to be considered are the engineering properties and contamination potential of the waste, effects of seismic loading, and ways to control water to maintain stability and limit contaminant release (Vick, 1983). These factors will not be treated separately here, but a designer should be aware that they need to be addressed in detail. The geomechanics of stability in relation to waste disposal is covered in more detail in Section 10 of this *Handbook.*

The layout of waste structures can be in one or more of the following forms, depending on the type of waste, the purpose of the waste structure, and the physical constraints at the site (Fig. 12.2.2).

As the name indicates, a *valleyfill dump* fills a valley. The top surface is usually sloped to eliminate water ponding. Construction begins at the upstream end of the valley and dumping proceeds along the downstream face. This type of embankment can also be started as a crossvalley structure where the area is subsequently filled upstream.

A *crossvalley* structure crosses the valley bottom, but the valley is not completely filled upstream. The structure is usually designed to control the storage and/or discharge of flood flows. If the structure is not designed to retain water, then a water diversion system must be installed to provide drainage around it. This type of structure is typically used as a retention dam for fine coal or waste slurries.

A *sidehill* structure lies along the side of a hill or valley but does not cross the valley bottom. It can be built to impound either water or mine waste slurries. As with a crossvalley structure, a sidehill embankment must control the storage and/or discharge of flood flows if it is to retain water or slurries.

A *ridge embankment* straddles the crest of a ridge, and waste material is placed along both sides of the area. This type of structure is not typically used to contain material or water.

A *diked embankment* is constructed on nearly level terrain and can either impound fine-grained or coarse-grained mine waste. If fine wastes are impounded by coarser waste, the structure is considered a dike. If the embankment is homogeneous and coarse, the embankment is termed a heap.

12.2.3.1 Leach Dumps or Heaps

Leach dumps or heaps consist of low-grade ores that are dumped loosely in piles so that fluids may be sprinkled on the piles to leach recoverable metals (Chapter 15.2). Heaps are placed on impermeable liners, while dumps are placed on soil surfaces. Care must be taken to ensure structural stability of the piles and control metals-laden seepage during and after leaching operations. Upon completion of the leaching process, reclamation of the piles must include environmental and structural stability. Cyanide is of particular concern with gold leach heap operations (Ingles and Scott, 1987; Scott, 1987).

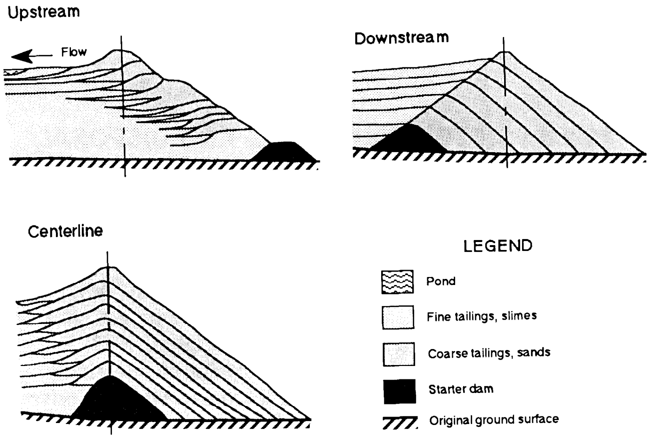

Fig. 12.2.3. Methods of dike construction for tailings impoundments.

12.2.3.2 Stockpiles

Stockpiles are ore that is stored much the same as waste rock except that they must be isolated from waste so that the ore may be recovered at some later time. This material may be chemically unstable, and so specific efforts must be taken to keep water from infiltrating the ore and becoming a contamination concern. Stockpiles must also be structurally stable and must be prevented from eroding.

12.2.3.3 Surface Impoundments

Impoundment structures are those designed to contain solid waste and effluents generated during the processing of coal and metal/nonmetal ores (see Chapter 7.4). Impoundment structures are typically water retention dams or embankments raised during waste disposal. Water retention dams are used where large amounts of water must be stored during specific periods (i.e., during spring runoff) or where the impounded water is not reusable for processing. Since design of a water retention dam does not need to incorporate rapid drawdown of water, the inside slope of an embankment can be steepened to reduce embankment volume and costs.

Raised embankments are favored over water retention dams because costs are significantly lower and mine waste materials can be used as part of the embankment, reducing volume requirements correspondingly. In addition, an embankment does not require that suitable embankment material be located and transported or that the entire structure be built before disposal begins. Three methods of constructing raised embankments have been developed to address various design requirements (Fig. 12.2.3). These are the upstream method, the centerline method, and the downstream method. Advantages and disadvantages of these methods may be seen in Table 12.2.1.

The upstream method is the least costly and the most widely used, but it is subject to a number of crucial constraints. Special consideration must be given to stability, particularly the stability of the phreatic surface within the embankment. Stability is controlled indirectly by (1) grain-size segregation near the embankment, which controls permeability and internal water flow, (2) location of the impounded water relative to the embankment crest, and (3) permeability of the foundation of the impoundment relative to the embankment. Successive levels of dikes rest on the beach of a previous dike. The weight-bearing capacity and settlement characteristics of these beaches must be considered. Waste is spigoted from single or multiple discharge points off a transport pipeline. The coarser portion of the waste settles out near the crest, and the finer material is carried toward the interior of the impoundment. Hydrocyclones can be used to improve the separation of the coarse materials; drained coarse material is used to raise the impoundment perimeter. Advantages of this technique include low construction costs, a minimum amount of dike fill to obtain storage capacity, and the capability of raising the embankment rapidly. The major disadvantages are that the embankment perimeter is built on previously deposited slimes, and the meandering flow of slurries from a spigot results in pockets of weak slime and sand along the periphery.

The downstream method of depositing waste requires a starter dike and, usually, an impervious layer of some sort on the upstream face of the dike. This layer acts as a seepage barrier to control the level of the phreatic surface in the dike and can allow water to pond near the inner face of the embankment. Hydrocyclones are used to separate the slurry. The coarse underflow is used to build the embankment downstream of the point of deposition, and the overflow is discharged into the interior of the impoundment. Blanket or finger drains should be provided downstream from the starter dam to collect and discharge seepage water deposited with the slurried waste. This technique avoids building the perimeter on previously deposited slimes; however, each successive downstream extension requires greater volumes of coarse material.

The centerline method begins with a starter dam and subsequent deposition of hydrocycloned underflow materials on both the upstream and downstream sides of the embankment. Blanket or finger drains are installed downstream of the centerline, much like in the downstream method of deposition.

Very fine waste, such as phosphate or coal slimes, are impounded by an embankment of coarse waste or borrow material. Internal drainage and reduction of pore water pressures within the waste may take many years; therefore, the impoundment structure must be designed and constructed as a water-retention-type dam.

Other deposition techniques include thickened discharge and subaerial. In some settings, these techniques may provide greater engineering strengths and more rapid dewatering, which may be critical for environmental or structural requirements. The thickened discharge technique consists of thickening the slurry and pumping the mix to a central deposition point or points to form a conical hill (Lighthall, 1987). A small embankment around the perimeter is used for collecting waste and runoff water. Chemical flocculants and/or mechanical dewatering devices can be used to accelerate thickening of the waste slurry.

The subaerial technique requires that the slurry be discharged through spray bars on a gently sloping beach of previously deposited waste. The laminar flow settles the solids in layers and the excess water flows over the beach to the collection pond. The method allows for continuous draining of the deposited material, thereby consolidating the mass through negative pore water pressures (Haile and Brouwer, 1988; Lighthall, 1987; Knight and Haile, 1983).

Many fine-waste disposal methods may need to be altered during freezing weather conditions, because freezing will alter deposition characteristics and may create ice lenses, areas of weakness, and other problems. Often during freezing conditions, spigoting or cycloning may be deferred in lieu of point discharge further into the pond interior to avoid creating embankment conditions that could weaken the structure.

12.2.3.4 Backfill

When possible, coarse waste rock from underground development operations is transported to previously mined-out open-

Table 12.2.1. Surface Impoundment Embankment Types

Embankment type	Mill tailings requirements	Discharge requirements	Water storage suitability	Seismic resistance	Raising rate restrictions	Embankment fill requirements	Relative embankment cost
Water-retention borrow	Suitable for any type of tailings	Any discharge procedure suitable	Good	Good	Entire embankment constructed initially	Natural soil	High
Upstream	At least 40–60% sand in whole tailings. Low pulp density desirable to promote grain-size segregation	Peripheral discharge and well-controlled beach necessary	Not suitable for significant water storage	Poor in areas of active seismicity	Less than 15–30 ft (5–10 m)/yr most desirable. Greater than 50 ft (15m)/yr can be hazardous	Natural soil, sand tailings, or mine waste	Low
Downstream tailings	Suitable for any type of details	Varies according to design	Good	Good	None	Sand tailings or mine waste if production rates are sufficient, or natural soil	High
Centerline	Sands or low plasticity slimes	Peripheral discharge on nominal beach size necessary	Not recommended for permanent storage. Temporary flood storage acceptable with proper design details	Acceptable	Height restrictions for individual raises may apply	Sand tailings or mine waste if production rates are sufficient, or natural soil	Moderate

Source: Vick, 1983.

ings to avoid hoisting time and cost, as well as surface waste disposal. Waste created during milling operations can also be transported underground or back to a pit and used as backfill (Boldt et al., 1989; Biggs et al., 1989). This backfill material can be "classified" waste (the coarse fraction of the total amount). Waste material is ordinarily pumped as a slurry by pipeline to the specific area to be backfilled, placed behind constructed bulkheads, and allowed to dewater or solidify. Classified waste material may be placed with or without the addition of cement, depending on the requirements of the mining technique being used. Total waste material is placed with lower water contents than classified materials, and any added cement tends to use the excess water in the hydration process. The excess water must be discharged (or consumed) from the waste backfill materials to eliminate the potential for liquefaction failures of the fill (Aref et al., 1989).

Backfill disposal in a surface mine requires enough room to place the waste material without interfering with the active mining (resource recovery) operation. In coal area mining (a variation of strip or open cast mining), the overburden is backfilled into the mined-out section, leveled, and the topsoil replaced.

12.2.3.5 Placer Waste

During placer mining, the practice of washing sand and gravel to recover minerals produces tailings with particle sizes ranging from coarse to fine and waste water streams loaded with fine sand, silts, and/or clays (slimes) (see Chapter 15.1). Proper placement of tailings and clarification of the waste water to meet effluent discharge requirements are primary design goals. Unneeded water should be diverted around the mining operation, and the amount of water directed to the operation should be kept as low as possible; lower water volumes and flow rates will result in more efficient settling pond systems. Water bypass systems will also reduce flood risk, reduce the amount of water going through settling ponds, allow fish to pass more freely, and provide a drier working area for equipment.

Settling ponds should have a length-to-width ratio of at least 2:l and should be large enough to retain the volume of expected sediments (Hanneman et al., 1987). Settling efficiency can be increased if more than one pond is used. Presettling ponds in series or in parallel can be used to remove up to 50% of the heavier sediment before the waste water reaches the main settling or recycling pond (Ault, 1986).

In general, a properly designed system of multiple settling ponds will allow requirements for suspended solids to be met. However, to meet turbidity and dissolved metals requirements at some placer mining operations, special techniques are necessary, such as flocculation (Phillips and Pollen, 1987), mechanical dewatering, liquid fluidized beds (Fuerstenau and Han, 1988) and/or a combination of these treatments. Environmental costs for waste stabilization and effluent treatment also need to be considered in feasibility studies of economic placer mining (Stebbins, 1987).

12.2.3.6 Uranium Waste

Because nuclear power plants are an important source of electrical power in the United States, and because of the threat and mobility of radionuclides, disposal of uranium mill tailings is one of the most regulated of mining activities and requires design goals at the onset of planning that will include cost-effective containment considerations for long-term stabilization, public health and safety, and environmental protection. After the best site has been selected, design and construction of tailings piles are critical to ensure meeting design goals. A final environmental impact statement on uranium milling (Anon., 1980a) lists alternative disposal systems for uranium mill tailings. The Nuclear Regulatory Commission has published several "how to" documents regarding disposal of uranium mill tailings. Particular topics of concern are treatment techniques (Sherwood and Serna, 1983), assessment methods (Serna and Peterson, 1983; Denham et al., 1985), long-term stability (Nelson et al., 1986), leachates (Opitz et al., 1985), erosion (Walters et al., 1986), and reclamation (Beedlow et al., 1985).

12.2.4 WASTE TRANSPORT SYSTEMS

Transport options for various disposal schemes should be considered when planning a waste disposal system. The selection and design of waste transport systems depend primarily on the type of waste produced by the mining or milling facility and the disposal method used. Transport methods can be categorized according to their suitability for handling coarse or fine mining wastes. Other considerations include how fast (rate) and for how long (duration) the waste is to be transported; moisture, specific gravity, cohesion, and abrasiveness of the waste; distance and terrain over which the waste is to be transported; and environmental constraints. Many disposal systems can be optimized by the use of more than one transport technique. In general, it is most desirable to select a transport system that can deal with large variations in particle size. Some of the available waste transport options are described briefly in the following, along with an explanation of the fundamental mechanics of the systems.

12.2.4.1 Coarse Waste Transport Systems

The size distribution of particles classified as coarse may cover a broad spectrum, from boulders to sands. The following methods are suitable for coarse waste transport. An in-depth treatment of haulage logistics may be found in Chapters 9.3 and 13.3.

GRAVITY FLOW. From an operations standpoint, the final disposal site should be at a lower elevation than the waste storage or production area so that transport costs can be minimized through the use of gravity. However, most waste transport systems cannot rely entirely on gravity flow but will require additional transport methods.

Coarse wastes can be dumped on an open slope or through rock passes to allow gravity flow, but this method is usually limited to the construction of waste dumps. Waste materials can also be transferred from individual systems through chutes. Poorly designed transfer chutes can limit the efficiency of the entire waste handling system.

The probability of blocking rock passes or conduits is a function of size of the waste particles being handled, ratio of conduit diameter to maximum particle size, shape of the particles, velocity profile across the flowing material, amount of moisture in the waste, and cohesion and frictional properties of the waste. Specific design criteria have been developed to evaluate the flow of materials through various components of a gravity flow system (Hambley, 1987; Hambley et al., 1983).

TRUCK HAULAGE. Trucks are the most efficient form of transport when great flexibility is needed to accommodate changes in disposal destination. However, this method would usually be considered only for operations using truck haulage for ore removal. Trucks are available in a wide variety of sizes and configurations to meet most needs. Compatible equipment should be selected for loading trucks at the desired production rate. Road maintenance should also be considered.

RAIL HAULAGE. If there are long distances between the storage or production area and the disposal site or if rail haulage is already being used to remove waste from the mine, rail haulage may be most applicable. While rail haulage affords little flexibility in changing disposal destinations, it may be suitable for waste-dump construction.

CONVEYORS. In common with rail haulage, conveyors have limited flexibility when disposal areas must be changed. However, several movable and/or modular conveyor systems have been constructed that greatly increase flexibility. In general, conveyor systems are suitable where large volumes of waste product need to be transported over a long period of time (Rajaram, 1982). Conveyors may be the most desirable method for transporting waste rock as backfill in underground mines (Watkins, 1978). Conveyors can operate in small entries, yet they provide a large hauling capacity. Additional equipment, such as reclaimers and spreaders, can increase the efficiency of conveyors in waste disposal. Sticky wastes can become a problem with belt conveyors, especially at transfer points.

PNEUMATIC TRANSPORT. Low-pressure, high-volume pneumatic transport of coarse wastes has been used primarily for backfilling (Reynolds, 1972). These systems are usually limited to handling rock with a maximum particle size of 3 to 4 in. (75 to 100 mm) (Tweedy, 1984). Pneumatic systems are only suitable for dry, noncohesive wastes; in some extreme cases, they may require that the moisture in compressed air be removed to prevent plugging. Pneumatic systems typically consist of three basic components: the compressor, which provides airflow; the feeder, which meters the flow of waste into a pipeline and prevents air leakage; and the pipeline. The main distinction among the systems offered by various manufacturers lies in the feeder device used to introduce solids into a stream of air. The output capacity of pneumatic systems is determined by the size and specific gravity of the material, flow rate of air, and pipeline dimensions, as well as layout (Sands et al., 1990).

12.2.4.2 Fine Waste Transport Systems

Fine wastes are typically the product of beneficiation processes, and the physical properties are highly dependent upon the nature of the specific process. For instance, milled wastes originate as a slurry of finely ground waste rock and water. These slurries may be placed in impoundments or used as backfill, both of which require dewatering at the disposal site. Given proper conditions, many fine wastes may be transported by the same methods described for coarse wastes. However, the following transport methods are usually considered more suitable for fine wastes.

HYDRAULIC TRANSPORT. As previously noted, many mine wastes originate as slurries and are transported most suitably in this form (see Chapter 25.2). The flow of slurries can be characterized as either heterogeneous, homogeneous, or compound. These flow regimes are determined by the grain-size distribution, specific gravity of the solid, and the flow velocity.

Several empirical tests to determine friction loss and critical velocity of deposition for heterogeneous slurries have been developed, but these tests are only applicable for laminar flow conditions (Verkerk, 1985). Homogeneous slurries, especially those of high solids concentrations, must be investigated with full-scale tests or theoretical models that require knowledge of the rheological properties of the slurry (Hanks, 1986). Most slurry operations have used gravity or centrifugal pumps for the heterogeneous flow regime. However, transport of waste backfill may not be limited to hydraulic transport techniques. Test results indicate that thickened, unclassified waste slurries, which would require no underground dewatering, could be pumped by equipment developed by the concrete industry (Vickery and Boldt, 1989).

PNEUMATIC TRANSPORT. Pneumatic systems for transporting fine wastes consist of the same basic components as those used for coarse wastes. The main application of pneumatic transport for fine wastes has been in European coal fields to stow mine wastes (Wood, 1983). Pneumatic systems may be characterized as dense phase or dilute phase, depending upon the concentration of the solids conveyed in the pipeline.

Dilute-phase conveyances use high-velocity air to suspend and transport discrete particles. In dense-phase conveyance, the transport material is forced through a pipeline in "slugs" separated by pulses of compressed air. Since the material is delivered relatively slowly, system wear and energy consumption are reduced as compared to pneumatic suspensions (Roberts et al., 1985). Theoretical solutions to determine a system's operating parameters have been developed for each phase of the conveying process, and preliminary designs have been simplified by the use of computers (Latincsics, 1982).

12.2.5 CONTAMINANT CONTROL

Designing waste disposal facilities that meet environmental goals is a complex task because of the large number of factors that influence waste, the many disciplines involved, a general lack of experience in meeting environmental goals, and the number of inherent uncertainties about geologic and geochemical parameters that must be managed. Spatial and temporal changes in any of these factors can vary by orders of magnitude even at one site.

The processes that control the rate of contaminant mobilization and transport also vary significantly from site to site, and many are very slow. For example, acid water and metals from wastes have been released years after sites have been reclaimed and revegetated, destroying the work of reclamation efforts and requiring resolution of the much greater problems involved with acid-generating wastes. Other problems arise in that some of the methods used to control one set of contaminants may cause another set to be released.

The purpose of contaminant control, as discussed here, is to prevent or limit the release of designated contaminants associated with waste disposal. Contaminant controls can be implemented anywhere along the contamination pathway from source to receptor. Usually, the most economical and environmentally effective approach is to implement controls as close to the source as possible.

The mechanisms underlying the chemical release of contaminants are not yet well understood. Oxidation is considered to be the primary cause of contaminant release; however, other mechanisms have been observed. Water contamination is most often a result of pyrite oxidation, which lowers the pH of interstitial water and mobilizes metals. Once pH is about 4.5, acid formation can be dramatically accelerated by action of bacteria such as *Thiobacillus ferrooxidans*. Some studies suggest this bacteria may speed the process by more than a factor of 1000. Contaminants can also be carried as solids in water or air in both surface and groundwater via surface runoff, failure of containment structures, seepage, and wind. The technology to address contamination associated with mining has been developed primarily for eastern US coalfields (Anon., 1979; Skousen et al., 1990) and Canadian hard-rock mines (Filion and Ferguson, 1990; Anon., 1988a, 1989b, 1990a). This technology is now being applied to noncoal mining in the United States (Anon., 1988b).

Environmental control measures, which are in various stages of development at the present time, involve the use of liners, hydrologic barriers, pH-control materials (Cravotta et al., 1990), chemical treatment, wetlands, and bactericides; drainage collection and treatment; and underwater disposal.

12.2.5.1 Investigations and Tests

Testing the properties of waste materials at a given site is a primary means by which mine operators manage waste and meet environmental regulations. Generally, an operator's main concerns are predicting (Anon., 1989c) and verifying effects on local surface water and groundwater systems (see Chapter 12.1). Understanding the chemistry of the water is a major factor in determining dissolution and contamination potential. Many good sources of information are available when considering these aspects of water chemistry (Hem, 1985; Anon., 1985b).

The physical groundwater system is delineated by taking depth-to-water measurements with piezometers and in wells drilled for other purposes (exploration wells, water wells). Care must be taken when a well intersects more than one aquifer because water levels will reflect the interaction. Bentonite seals should be placed between aquifiers if water quality is significantly different among them to prevent one aquifer from contaminating another. The hydrologic budget is determined from precipitation records, evapotranspiration estimates, and groundwater inflow/outflow measurements. The controlling equation is outflow = inflow + storage. Groundwater flow paths and residence times are determined through the use of isotopes such as tritium and carbon 14. A source of periodic information can be found in the *Groundwater Monitor Review*, a bulletin published monthly by the National Water Well Association, Dublin, OH. Different aspects of investigation and testing are listed below.

MODELING. Advances have been made recently in the use of hydrologic models as predictive tools to simulate the complex flow and fate of water and/or dissolved or suspended constituents (Anon., 1987, 1990b; Kincaid et al., 1984). A model can be used as a design aid to approximate the hydrologic and hydrochemical consequences of a planned mine waste disposal system quantitatively. There are three basic types of hydrologic model: (1) physical (a scaled-down version of the designed system), (2) analog (a model of the behavior of an analogous, but different, medium), and (3) mathematical. Physical and analog models tend to be impractical, so only mathematical computer models are commonly used.

Many different hydrologic and geochemical computer models are available, ranging from simple drawdown determinations to complex programs with three-dimensional graphics capabilities (Melchoir and Basset, 1988; Wang and Anderson, 1982; McDonald and Harbaugh, 1984; Nordstrom and Ball, 1984). A common assumption of most geochemical models is that equilibrium is reached; however, equilibrium is a function of temperature and kinetics and will not be valid for many components. Some codes suffer from inconsistency among the thermodynamic databases they utilize, making them inadequate for certain applications. Most geochemical programs were developed for specific applications, and consequently, no single program or program family has emerged that can be applied universally to mine waste management.

The user should select a program or group of programs that best meets project needs and incorporates as many of the following parameters as possible: constituents (including ion speciation), temperature, pH, redox potential, pressure, and alkalinity. The thermodynamic data base and the mineralogic data base should correspond as closely as possible to the field situation. One feasible approach is to use a program such as the US Geological Survey's WATEQ4F (Ball, 1987), which has excellent dissolved species and mineralogic data bases, to predict equilibrium conditions, and then use a mass balance code such as BALANCE (Parkhurst et al., 1982) to test the results. Others have published brief synopses of various geochemical codes (Nordstrom and Ball, 1984; Nordstrom et al., 1979), and the International Groundwater Modeling Center distributes other references as well as many of the programs. Other computer models are available from the USGS, EPA, and CANMET (Ottawa, Canada). Another newer code (not currently verified for mining applications) that simulates the transport and fate of inorganic chemicals in the subsurface environment is FASTCHEM (Hostetler et al., 1989). A wide variety of other codes and references is available at the International Groundwater Modeling Center, Holcomb Research Institute, Indianapolis, IN.

WATER SAMPLING AND TESTING. Monitoring surface and groundwater for possible contamination has been required for a number of years. Because only low levels of contaminants are now allowed, and because instruments now have the ability to detect levels as low as 1 ppb, sampling has become much more sophisticated (Driscoll, 1986; Skougstad et al., 1979) and regulated (Anon., 1986b). Great care must be taken to ensure that the sampling procedure, sampling devices, and preservatives do not contaminate the sample, react with the sample, or interfere with the analytical method, thereby leading to expensive and possibly unnecessary legal action. Information concerning well design criteria, sampling protocol, and preservation procedures is available from the EPA, National Well Water Association (NWWA), and state agencies.

Surface water samples are taken directly from the source, but groundwater samples require a device that can sample at depth. Usually this is a bailer or pump in combination with a piezometer. Where depths are very shallow, a bailer may be functional, but because of the EPA recommendation to purge two casing volumes prior to sampling, this is impractical for deeper wells, and a pump should be used. Lysimeters, which use a vacuum to "pull" a sample through a porous filter, may be installed in certain situations, such as beneath an impoundment during its construction or through the center of a hollow-stem auger. Unlike a piezometer, which must be installed below the phreatic surface, a lysimeter will attract a sample from the unsaturated zone by differential pressure. Proper selection of the filter pore size allows differential sampling of pore gas or pore water. Other more sophisticated devices are available, but they are expensive and are most applicable to special situations.

The selection of the construction materials and installation method for a monitoring system is governed by the anticipated chemistry of the water to be sampled (Anon., 1980b) because the installation method can contaminate the sampled area. Bentonite and additives in drilling mud have a profound chemical effect, and compressed air can act as an oxidant. Contamination by drillbit and stem shavings must also be considered. During installation, samples of the native material should also be obtained, preferably using a method that yields minimal disturbance, such as coring or by Shelby tube. Identification of precipitates can provide information relative to attenuation mechanisms and water chemistry, and a mineralogic analysis can be used to determine adsorption and ion exchange capacities of the soil or aquifer. Grain-size distribution can also be used to estimate the hydraulic parameters of an aquifer.

SOLID WASTE SAMPLING AND TESTING. Many standard procedures have been developed for collecting solid wastes and testing them to predict their contamination potential. Field sampling techniques are particularly difficult because of inherent radical changes within mine wastes. These variations are both temporal and spatial, and the fact that relatively few samples must represent large volumes of waste must be taken into account. However, there are a number of sources that provide direction on sample collection (Anon., 1989d).

Traditionally, slurry leach tests such as the EP Toxicity, TCLP, and Method 1312 (Anon., 1988c, 1989e) have been applied to mining wastes. However, these tests are being replaced by more accurate ones that better simulate the complex, long-term geochemistry that results as contaminants are being transported (Lawrence et al., 1989; Anon., 1989e; Doepker, 1988, 1990), and these tests are increasingly being used to make critical and costly regulatory and management decisions.

Care should be taken to ensure that the specific factors prevalent at a site are accounted for whatever testing procedure is chosen, because these factors will influence the reactivity and contamination potential of a waste. These include grain size, mineralogy, sulfate and metal content, water flow characteristics within and surrounding a waste, availability of oxygen and water to initiate geochemical reactions, pH, temperature, presence and degree of activity of sulfide-oxidizing bacteria, and presence or absence of neutralizing material such as carbonates. The mechanism that influences most contamination associated with mine waste is pyrite oxidation; however, the processes that control oxidation vary significantly and are not well understood (Lowson, 1982; Crundwell, 1988). The Acid Base Accounting test has been used extensively in the past with coal waste and is still used to distinguish whether sites have great or little potential for contamination from pyrite oxidation (Anon., 1983; Ferguson and Ericson, 1987; di Pretoro and Rauch, 1988; Smith and Brady, 1990; Sobek et al., 1978).

HYDROLOGIC FIELD MEASUREMENTS. Determination of hydrologic factors and their interactions at a mine is one of the tasks necessary to determine the best alternatives to prevent or remediate water contamination. Contamination as a result of acid drainage in underground mines requires an in-depth assessment of mine hydrology (Williams et al., 1986). Typically the problem in hydrologic field measurements is looking at the subsurface below waste disposal facilities. The aquifer properties from which transmissivity and storativity are derived are hydraulic conductivity, porosity, and compressibility (Freeze and Cherry, 1979; Ferris et al., 1962; Bentall, 1963). These are best determined with field tests, generally using one or more boreholes. Attempts to collect aquifer material for laboratory permeameter tests invariably alter the measurements of the properties.

The following field tests are used to understand groundwater flow.

Steady-state Pumping Injection—In these tests, water is pumped out of or into a borehole at a steady rate while the water levels in surrounding wells and piezometers are recorded (Doe and Remer, 1981). Water level drawdown is a function of distance from the pumped well and indicates the flow properties of the aquifer. Analysis requires several simplifying assumptions regarding the physical nature of the aquifer system. Once these assumptions are chosen, curve-matching and iterative numerical techniques are available to determine aquifer parameters. The results contain large inherent uncertainties.

Slug or Impulse Test—These tests are similar to the steady-state test, but measure instantaneous injection and withdrawal of a significant amount of fluid rather than steady flow over a period of time. Slug tests are also referred to as transient state tests. The results are used to calculate slightly different aquifer parameters, such as storativity.

Tracer Test—Tracers are introduced into surface or groundwater flowfields to determine speed and direction of flow. Tracer materials include colored dyes, fluorescent dyes (fluoroscein and rhodamine compounds), conducting materials (calcium chloride), and mildly radioactive elements (tritium, iodine, bromine, and others). Freon is gaining popularity because it is nonreactive and is detectable in very small concentrations.

12.2.5.2 Containment

LINERS. Liners may be made of synthetic membranes or of natural materials such as clays. For natural liners, recent research indicates that diffusion models may provide better estimates of the performance of low-permeability lining materials than the commonly used Darcy equations (Shackelford, 1988). Natural materials have fallen into disfavor with regulatory agencies because these materials may occasionally develop leaks and because agencies have typically used Darcy models, which may grossly overstate flow for materials of very low permeability, such as clay materials. Clay is often available locally, although each new source must be tested for its sealing capability. Slimes may reduce the permeability of waste impoundments fairly effectively. Slime layers often form naturally with time at the bottom of waste impoundments and provide an important barrier to water and contaminant movement. Bentonite is a type of clay widely used as a sealant because it expands and therefore closes off flow pathways when saturated.

Synthetic waste impoundment liner materials include elastomers, asphalt, and concretes (Koerner, 1989; Richardson and Koerner, 1990). These materials must be selected for their resistance to the liquids expected to form in the wastes and the length of time that a barrier is required. The effective life of such liners has not been clearly established. However, estimates have been made that are considerably shorter than the time that many wastes would still be releasing metal ions and acids. Elastomers must be sufficiently thick to resist punctures and tears, yet flexible enough to conform to the containment structure. Concerns include seaming sheets of elastomeric material together, damage caused by equipment operating on the impoundment, degradation by ultraviolet light, damage by animals, and settling of the structure. Asphalts and concretes are very expensive to place in large impoundments and are often more easily damaged by settling than are elastomeric liners. Drain systems are often constructed to detect and capture any leakage from lined impoundments. Natural and synthetic liners have been used in Sweden for capping and lining to control acid drainage from abandoned mine wastes (Lundgren, 1990).

GROUTS. Grouts are broadly categorized as cement or suspension grouts and chemical or liquid grouts (Nonveiller, 1989; Houlsby, 1990; Karol, 1983). The cements include cement/water suspensions, cement and activated mortars, and cement plus clay and sand. Cement grouts tend to be relative inexpensive compared to chemical grouts. If acidic groundwater conditions are expected, a sulfate-resistant portland Type V cement may be used. Chemical grouts include sodium silicate, sodium silicate plus reagent, bentonite, organic resins, and asphalts. Chemical grouts are usually from 2 to over 100 times more costly than cement grouts. Organic resins seem to have lost favor recently because of their toxicity, cost, and short life under field conditions. Boreholes are drilled to develop grout curtains and slurry trenches are dug to develop grout walls. In some cases, it is appropriate to pregrout before excavating a waste disposal pit, then test and grout from the pit bottom. In addition, Russian clay grouting technology is being adapted to limit groundwater contamination caused by mining (Kipko, 1988; Anon., 1990c).

12.2.5.3 Prevention and Treatment

CHEMICAL/BIOLOGICAL. There are a number of different approaches that are being tried to chemically bond contaminants, harness biological activity, or alter natural processes that lead to contamination. Success with any particular method is highly dependent on site conditions; what works well at one site may have disastrous consequences at another. Extensive investigation and assessment of techniques is normally warranted before a full-scale facility is constructed. Developing wetlands to treat acid mine drainage has been studied extensively for coal wastes (Hedin, 1989; Hedin and Nairn, 1990; McIntire et al., 1990), but very little for noncoal wastes (Dollhopf et al., 1988) because heavy-metal chemistry is typically more complex and difficult to control in wetlands. Biocides have been used successfully at some sites to dramatically reduce acid generation for the short term (Sobek et al., 1985; Erickson et al., 1985).

Many methods are being developed to prevent the production of contaminated waters by limiting the availability of oxygen, and therefore oxidation reactions, within the waste. For example, underwater disposal of tailings has been successful at some sites (Pederson and Drysdale, 1990; Pelletier and Birch, 1990). Control of acid generation from waste rock is also of concern (Anon., 1989b; Robertson and Barton-Bridges, 1990). Other processes are used to neutralize and/or inhibit releases. The most commonly used reagents are lime, cement, fly ash, sodium silicate, clay, polysilicates, or a combination of these (Anon., 1989b, 1990a; Cravotta et al., 1990). Research and field trials have also been completed on blending wastes to limit their acid-generating potential (Pelletier and Howe, 1990).

WIND. Movement of mining wastes by wind can be reduced by capping waste piles with coarser material, such as slag, that is not prone to being blown away by wind (Walters et al., 1986; Bohn and Johnson, 1983). Other approaches include periodic wetting, solidifying or agglomerating materials to increase particle density and material cohesion, or leaching contaminants of concern from near-surface to deeper levels. Some are temporary measures intended to prevent problems until final reclamation. The installation of windbreaks (trees and bushes) or planting vegetation on waste piles may also help reduce movement of contaminants by wind. However, revegetation can create problems as plants take up metals and bring them into the food chain.

WATER. Water control systems must be engineered to handle ponding and runoff from waste facilities and upgradient drainage basins. Such systems are normally ditches and/or stream diversions or barriers designed to prevent infiltration and recharge of groundwater (Johnson et al., 1985). Drains will reduce recharge head and the volume of water entering lakes and ponds, route water flow around waste, and draw off potential recharge water before it reaches the waste. Planting trees in low, flat areas may increase evapotranspiration, thereby reducing infiltration and recharge (Loofbourow, 1979). Waste piles and ponds may be capped to prevent infiltration of rainwater and snowmelt.

Control of groundwater becomes important when surface water controls have not been used or have failed. Control is accomplished by using interceptor wells to draw off recharge water before it reaches a waste. Water curtains create areas of artificially high head upgradient from a waste and these areas act as barriers to regional flow. Water curtains are often used in conjunction with interceptor wells. Groundwater may also be diverted by grouting with cement slurries or chemical grouts. Grout and waste in which the sand has been removed can be used to plug solution channels, and clay can be used to plug pores or fractures.

12.2.6 CONCLUSIONS

The trend in developing regulations is to specify disposal facility *performance* rather than disposal *techniques*. This places more responsibility on a design engineer to plan and defend creative disposal facilities, whether to obtain operating permits or to upgrade existing disposal systems. This chapter was written to delineate design consideration with numerous references so that the "how to" detail could be accessed. From this, the design engineer may plan waste disposal facilities as well as address new issues in waste disposal, including remediation of existing or abandoned waste disposal facilities, refinement of techniques to minimize contamination potential at a specific site, and exploration of alternative uses for a waste.

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Chapter 12.3
RECLAMATION

RICHARD J. SWEIGARD

12.3.1 INTRODUCTION

12.3.1.1 Scope of Reclamation

At the outset, it is appropriate to establish the meaning of the word reclamation and set it in the proper context since the term has many connotations. Reclamation is sometimes used synonymously with restoration; however, this is not necessarily appropriate. One definition of reclamation is the process of gaining or recovering land, bringing it into a condition for cultivation or other use (Anon., 1946). In the broadest sense, land reclamation need not be in response to some disturbance caused by human activity (e.g., reclamation of desert land). As it relates to mining, however, *reclamation* is a response to any disturbances to the earth and its environment caused by mining activity.

Restoration implies that the exact conditions of the site existing before disturbance will be replicated afterwards (Anon., 1974). In contrast, *rehabilitation* involves returning the disturbed site "to a form and productivity in conformity with a prior use plan." Since complete restoration is a virtual impossibility, the term reclamation, as it is applied to the mining industry, has taken basically the connotation of rehabilitation. This fact is evidenced further by the Surface Mining Control and Reclamation Act of 1977 (SMCRA). SMCRA states that, among other provisions, reclamation must "restore the land affected to a condition capable of supporting the uses which it was capable of supporting prior to any mining, or higher or better uses." Reclamation of land disturbed by mining, therefore, has been linked closely to the issue of land-use capability. It is not necessary, nor is it possible in most cases, to return the land to its original condition as long as its capability is not diminished.

While it is most common to consider reclamation as an aspect of surface mining, underground mining and mineral processing activities also impact the land in such a way as to affect its use. Reclamation activities must address all the impacts of mining as it impacts land-use capability. Land disturbance by the mining industry can be classified in several ways. Two useful classifications involve mineral commodities and mining functions. From Fig. 12.3.1 (commodity distribution), it is apparent that coal mining is by far the largest land user in the mining industry. In Fig. 12.3.2 (functional distribution), the affected total land area is classified according to mining function. Surface mining is responsible for 85% of the total land utilized. Underground mining accounts for a relatively small percentage (5%). Mineral processing wastes occupy the remainder (10%) of lands (Johnson and Paone, 1982). Therefore, a large component of the efforts in reclamation planning, research, and regulatory controls is directed toward surface mining, particularly the surface mining of coal.

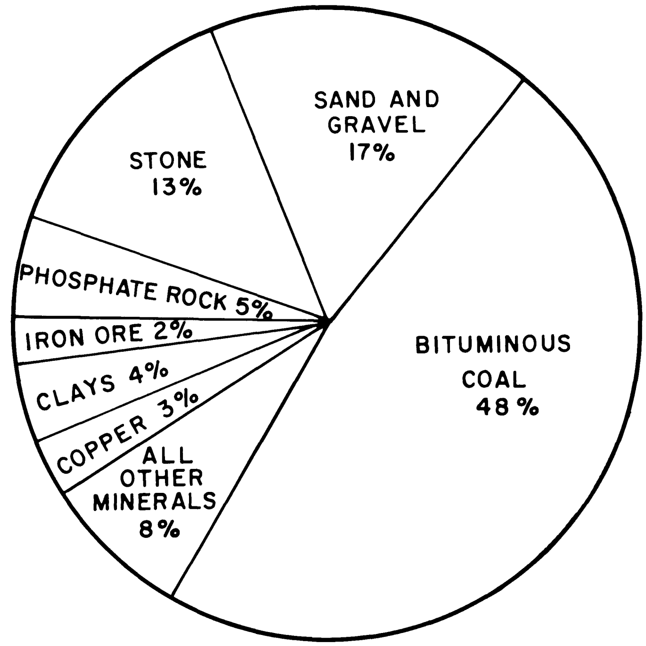

Fig. 12.3.1. Percentage of land disturbed to mine selected commodities, 1930–80 (Johnson and Paone, 1982).

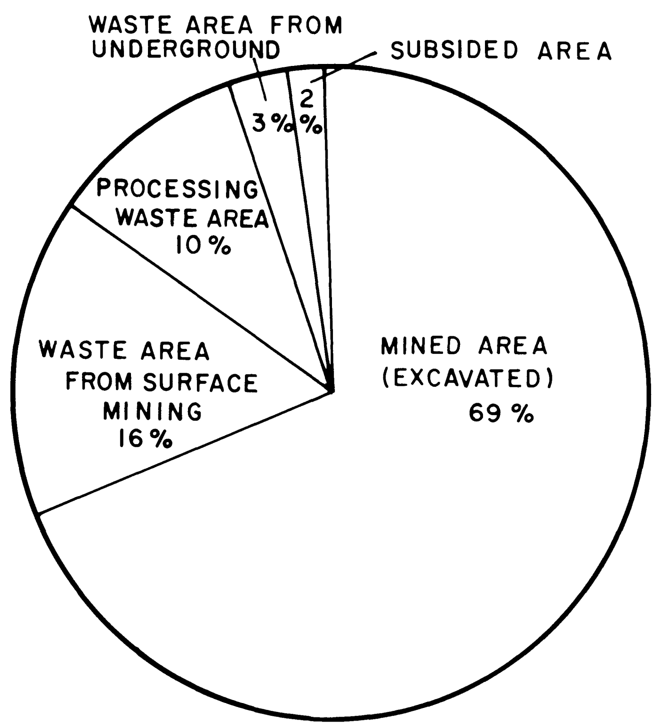

Fig. 12.3.2. Percentage of land disturbed by mining function, 1930–80 (Johnson and Paone, 1982).

12.3.1.2 Historical and Legal Framework

Historical developments in reclamation are linked closely to the regulatory systems that have been created to control environmental effects of mining. Since reclamation generally results in a cost incurred after the mineral has been extracted and does not add to the inherent value of the mineral, there was little incentive in the past for mine operators to reclaim the land. This was particularly true in a climate where the rights of the private landowner to use the land for whatever purpose he or she chose were unchallenged. This attitude had changed by the early- to mid-twentieth century to the point that the public viewed environmental pollution or any loss in land-use capability as a social cost that must be taken into account by the mineral industries.

The approaches taken to reclamation are as varied as the minerals in question and their methods of extraction. For example, the sand and gravel operators have had to address the issue of postmining land use due to the proximity of most operations to urban areas. This segment of the industry is recognized for its pioneering work in applying site-planning principles to mining and reclamation operations (Ahearn, 1964; Baxter, 1969).

The first recorded instance of reclamation by the US mining industry occurred in 1918 when some surface-mined land in Indiana was planted with fruit trees. Following this initial example, a group of mine operators comprising the Indiana Coal Association agreed in 1926 to plant five acres (two hectares) of trees for each shovel or dragline that they used (Bowling, 1978). These first voluntary reclamation efforts did not include any grading or topsoil replacement.

Concern by governments for effective reclamation of land disturbed by mining has been evidenced through the years in both the passage of regulations and increased research and development efforts (Chapter 3.4). The earliest state reclamation laws addressed only coal mining. Prior to 1965, only seven states had specific laws requiring postmining land reclamation. The first state surface mine reclamation act was adopted by West Virginia in 1939, followed by Indiana (1941), Illinois (1943), Pennsylvania (1945), and Ohio (1947). These state laws addressed limited grading and revegetation of spoil piles. Over the years, more comprehensive state laws evolved through modifications and amendments to existing legislation. In the 1950s and 1960s, the laws were modified to include stricter grading requirements, soil conservation measures and water quality control. Modifications in the late 1960s and 1970s went beyond simple reclamation standards to the control of all major activities before, during, and after mining. Not all state programs addressed each of the major areas of concern.

By 1977, legislation to regulate mine reclamation had been enacted in 38 states. Seven of the states regulated only coal mining activites, and several were concerned only with surface mining. However, 28 states had regulations that covered virtually all minerals and at least nine states regulated the surface effects of both surface and underground mining. The scope of performance standards found in the state reclamation regulations includes slope stabilization, grading, revegetation, neutralization of toxic materials, visual screening, waste disposal, control of drainage, and other sources of pollution and erosion (Anon., 1979).

After nearly ten years of legislative efforts and three presidential vetoes, the federal government attempted to standardize reclamation policy with the passage of SMCRA in 1977. Although the gradual evolution of various state laws and regulations had an impact on mining and reclamation operations, SMCRA had immediate and far-reaching effects. This law applies specifically to coal mining, but it does address the surface effects of both surface mining and underground mining. Although SMCRA is a comprehensive piece of legislation that deals with such matters as diverse as abandoned mine land, mining research, and enforcement of the federal government's policy, the most direct impacts upon reclamation were in the form of standardized reclamation planning requirements and extensive environmental protection performance standards.

In addition to SMCRA, there are several federal laws that, while not being directed specifically at the mining industry, require some reclamation activity or other form of environmental compliance on the part of mineral operations. These laws include, as a minimum, the Federal Water Pollution Control Act, the Clean Air Act, and the Resource Conservation and Recovery Act. There are also a number of laws that deal with special cases such as the Uranium Mill Tailings Act and those involving the use of federal land including the exploitation of minerals.

12.3.1.3 Reclamation Operations

For the purpose of discussion, the topic of reclamation can be subdivided into three general categories. The first area is reclamation planning. As the size of operations has grown and reclamation requirements have become more complex, the importance of reclamation planning has increased greatly. The next general category includes all of those reclamation activities that are either continuous or periodic but are related in some way to the ongoing cycle of mineral extraction. The final category encompasses all of the isolated or sporadic reclamation activities that are not related directly to the production cycle.

Although a document known as a reclamation plan is a legal requirement for many mining operations, the concept of reclamation planning extends beyond the simple preparation of a document for regulatory purposes. It is a valid consideration for any operation that results in land disturbance even if it is not required by a particular regulatory authority. Since one of the primary goals of reclamation is to restore the land-use capability of disturbed land, reclamation planning must be related, in some degree, to land-use planning. It should encompass the consideration of natural and cultural factors that impact land use as well as economic and technical feasibility of the mining and reclamation methods; it should also employ the most current planning processes available.

Those reclamation activities that are continuous or periodic are sometimes associated with the concept of contemporaneous reclamation. This is particularly important in the surface mining of coal where strip (open cast) mining methods are in common use. These mines are characterized by the disturbance of large surface areas within a fairly short time frame compared to the length of time that an open pit mine or quarry may be in operation. Although backfilling, grading, topsoil replacement, and revegetation are the major components of contemporaneous reclamation, a detailed list of all reclamation activities associated with surface mining of coal is given in Table 12.3.1.

The isolated or sporadic reclamation activities do not lend themselves to simple characterization. They may include such diverse operations as final mine closure, reclamation of box-cut spoil, or mitigation of mine subsidence damage. Therefore, it is necessary to describe these activities on an individualized basis.

12.3.2 RECLAMATION PLANNING

12.3.2.1 Planning Objectives

Since one of the primary goals of reclamation is to restore land-use capability to disturbed land, reclamation planning is necessarily related to land-use planning. The scope of this chapter, therefore, goes beyond the strict sense of reclamation plan preparation as mandated by federal and state regulations. The term *reclamation planning*, as used here, encompasses the consideration of natural and cultural factors that impact land use, economic and technical feasibility of reclamation plans, mining methods, and current planning processes employed by the mining industry and the public sector.

The planning problem with regard to mining is one of maximizing production and maintaining environmental quality, con-

Table 12.3.1. Time Sequence of Reclamation Activities for Surface Coal Mining

I. During site preparation:
1. Install control measures (diversion, sediment traps and basins, etc.).
2. Clear and grub—market lumber if possible; stockpile brush for use as filters; run brush through woodchipper and use chips for mulch.
3. Stabilize areas around temporary facilities such as maintenance yard, power station, and supply area.

II. During overburden removal:
1. Divert water away from and around active mining areas.
2. Remove topsoil and store it if possible and/or necessary.
3. Selectively mine and place overburden strata if possible and/or necessary.

III. During coal removal:
1. Remove all coal insofar as possible.
2. For the purpose of controlling postmining groundwater flows, break, or conversely prevent damage to, the strata immediately below the coal seam as desired.

IV. Immediately after coal removal:
1. Seal the highwall if necessary.
2. Seal the lowwall if necessary.
3. Backfill—bury toxic materials and boulders, dispose of waste, ensure compaction.

V. Shortly after coal removal:
1. Rough grade and contour, taking these factors into consideration—
   a. Time of grading—specific time limit; tied to advance of mining; seasonal considerations.
   b. Slope steepness.
   c. Length of uninterrupted slope.
   d. Compaction.
   e. Reconstruction of underground and surface drainage patterns.
2. If necessary, make mine spoil amendments (root zone), taking these factors into consideration—
   a. Type of amendment—fertilizers, limestone, flyash, sewage sludge, or others.
   b. Depth of application.
   c. Top layer considerations—temperature (color), water retention (size consist, organics), mulching and tacking.

VI. Immediately prior to first planting season:
1. Fine-grade and spread topsoil, taking seasonal fluctuations into consideration.
2. If necessary, manipulate the soil mechanically—ripping, furrowing, deep-chiseling or harrowing, or constructing dozer basins.
3. Mulch and tack.

VII. During the first planting season:
Seed and revegetate, considering—time and methods of seeding, choice of grasses and legumes.

VIII. At regular, frequent intervals:
Monitor and control—slope stability; water quality, both chemical (pH, etc.) and physical (sediment); vegetation growth.

Source: Ramani, 1978.

Table 12.3.2. Required Components of a Reclamation and Operations Plan

Component	*CFR* Reference
	30 *CFR* 780
Operation plan—general requirements	780.11
Operation plan—existing structures	780.12
Operation plan—blasting	780.13
Air pollution control plan	780.15
Fish and wildlife plan	780.16
Reclamation plan—general requirements	780.18
Reclamation plan—protection of hydrologic balance	780.21
Reclamation plan—postmining land uses	780.23
Reclamation plan—ponds, impoundments, banks, dams, embankments	780.25
Reclamation plan—surface mining near underground mining	780.27
Diversions	780.29
Protection of public parks and historic places	780.31
Relocation or use of public roads	780.33
Disposal of excess spoil	780.35
Transportation facilities	780.37
Underground development waste	—
Subsidence control plan	—
Return of coal processing waste to abandoned underground workings	—
Requirements for special categories of mining	30 *CFR 785*
Maps, plans, and cross sections	780.11–780.37

Source: Clar and Arnold, 1981.

sistent with the constraints imposed by the regulatory agencies and the need to satisfy the profit objectives of the companies. In the 1970s, several research projects were initiated to aid in the premining planning of surface coal mining (Grim and Hill, 1974; Anon., 1976; Imhoff et al., 1978; Ramani and Clar, 1978; Pugilese et al., 1979). The necessary planning steps are (1) make an inventory of the premining conditions; (2) evaluate and decide on the postmining requirements of the region, consistent with the needs and desires of the affected groups; (3) analyze alternative mining and reclamation schemes to achieve best the objectives; and (4) develop an acceptable mining, reclamation, and land-use scheme that is the most suitable under the technical, social, and economic conditions (Ramani et al., 1977; Riddle and Saperstein, 1978).

12.3.2.2 Reclamation Plan

SMCRA established very specific requirements for obtaining a mining permit. Section 508 of SMCRA states that a reclamation plan of sufficient detail must be submitted to demonstrate that reclamation can be achieved as required by federal or state programs. The various components that are required by law in the reclamation and operations plan are listed in Table 12.3.2. Such a plan must cover, among other aspects, the following important points with regard to the uses of land:

1. Uses existing at the time of the application, and if the land has a history of previous mining, the uses which preceded any mining.

2. The capability of the land prior to any mining to support a variety of uses, giving consideration to soil and foundation characteristics, topography, and vegetative cover.

3. The use that is proposed to be made of the land following reclamation, including a discussion of the utility and capacity of the reclaimed land to support a variety of alternative uses and the relationship of such use to existing land-use policies and plans, and the comments of any owner of the surface, state and local governments or agencies thereof which would have to initiate, implement, approve, or authorize the proposed use of the land following reclamation.

4. A detailed description of how the proposed postmining land use is to be achieved and the necessary support activities that may be needed to achieve the proposed land use.

5. The consideration that has been given to making the surface mining and reclamation operations consistent with surface owner plans, and applicable state and local land-use plans and programs.

6. The consideration that has been given to developing the reclamation plan in a manner consistent with local physical, environmental, and climatological conditions.

The requirement that companies must submit a reclamation plan is not new. However, that surface mine planners must investigate "the utility and capacity of the reclaimed land to support a variety of alternative uses" [SMCRA, Section 502(a)(8)] was not a requirement in most state laws. It is beyond the scope of traditional reclamation planning and enters into the area of environmental site planning or land-use planning. This provision of the law requires additional expertise in and involves the inclusion of many disciplines in the mine-planning process.

12.3.2.3 Information Requirements

Since the ultimate goal of the reclamation plan is to ensure that the surface-mined land is returned to a productive use, it is appropriate to consider information needs for postmining land use planning. Land-use factors can either be classified as natural or cultural. Natural land use factors include, at least, the geomorphic, climatic, hydrologic, and soil characteristics of a site. Although these characteristics can be altered by man, they were initially the result of nature. Cultural factors include all geographic, demographic, and economic characteristics that are the result of man's activities. Important factors that must be considered are listed in Table 12.3.3. Detailed discussion of these factors can be found in Clar (1982), Clar and Ramani (1986), and Ramani and Sweigard (1983).

12.3.2.4 Land-Use Analysis

Current land-use patterns may provide the best guidelines for use after reclamation. For example, forested hillside land adjoining a pasture may be more desirable as a pasture after mining if the need is for pasture lands. Similarly, orphaned mine land in a wilderness area may be more useful if restored as a forest. Future land-use plans contrary to premining use are unusual and not the common practice. Acceptance of mountaintop removal and valley-fill mining methods that create new flat areas in regions where such areas are in short supply reflect the permanent positive potential of the temporary drastic disturbance of lands due to mining.

Identification of probable future land uses can be summarized into the following categories that are defined in Section 701.5 of Title 30 of the *Code of Federal Regulations (CFR):*

1. Cropland.
2. Pastureland or land occasionally cut for hay.
3. Grazingland.
4. Forestry.
5. Residential.
6. Industrial/commercial.
7. Recreation.
8. Fish and wildlife habitat.
9. Developed water resources.
10. Undeveloped land.

Analysis of alternative long-term uses requires compliance with socioeconomic and public constraints as well as the economic goals of the mining company. Regardless of the choice of short-term and long-term land use, the plans must be linked to the mining plan in space, time, and method. Definition of the long-range plans do achieve a purpose by minimizing disturbance unnecessary for the determined future use and limiting those reclamation activities that are not a part of the normal mining or land-use plans.

The site-planning process has been applied to reclamation planning by Sweigard and Ramani (1986). Using this process, several alternatives can be evaluated through an iterative procedure applying social, economic, and environmental impact assessment techniques. For any mining area, there may be a number of feasible postmining land-use alternatives. Also since surface mining, in particular, affords an opportunity to completely reshape land forms, the potential for reclaiming the land parcel for different end uses is great.

Since each alternative plan will represent different amounts of earthmoving, grading, topsoiling, and planting, postmining topography is one of the most important factors to be considered. Postmining topography is approximated by applying mine-plan data, appropriate swell factors, and box-cut and end-cut locations to the premining topography. Other key factors are the location and thickness of high-capability soils. These factors are particularly important in determining the location of agricultural lands. Location of property boundaries and public right-of-ways and limitations on certain equipment are also taken into consideration. For example, the grade limitation on mechanical tree planters may influence the location of forest land or the type of planting pattern that is employed. Premining land use, however, may play the most important role in determining postmining land-use plans. Surface mine planners often endeavor to balance, as nearly as possible, premining and postmining areas devoted to various land uses. Although the distribution of land among various uses may not change significantly, planning skills are required in designing a site plan that spatially orients the land uses in an efficient and aesthetically pleasing manner.

In short, the planning of mining operations in general, and the planning of reclamation and postmining uses of land in particular, have become very complex. The premining definition of long-range land-use plans for minable lands makes it possible, in theory, to minimize any disturbance of the land that is not part of the normal mining or land-use plans and to limit reclamation activities to those required by the future-use plans. Thus the planning of mining operations must be very thorough, with preserving or enhancing the long-term use of the land as a major objective. The application of an integrated mining, reclamation, and land-use planning concept is required to achieve this end (Ramani, 1978).

12.3.3 SURFACE MINE RECLAMATION OPERATIONS

12.3.3.1 Backfilling and Grading of Spoil

General Requirements. Detailed backfilling and grading requirements are given in Section 816.102 of *CFR* Title 30. With few exceptions, backfilling and grading must achieve the approximate original contour that existed before mining. This includes the elimination of all highwalls, spoil piles, and depressions. Stable postmining slopes must also be achieved. Slopes may not exceed either the angle of repose or a lesser slope necessary to maintain a minimum long-term static safety factor of 1.3.

Minimization of erosion and water pollution, both onsite and offsite, is another objective. Exposed coal seams, acid- and toxic-forming materials, and combustible materials exposed, used, or produced during mining must be covered adequately. This burial process is performed to control the impact on surface and groundwater among other effects. The overriding objective of the backfilling and grading requirements is to prepare the land to support approved postmining land use.

Table 12.3.3. Information Needs for Reclamation and Postmining Land-Use Planning

- I. NATURAL FACTORS
  - A. Topography
    - 1. Relief
    - 2. Slope
  - B. Climate
    - 1. Precipitation
    - 2. Wind—airflow patterns, intensity
    - 3. Humidity
    - 4. Temperature
    - 5. Climate type
    - 6. Growing season
    - 7. Microclimatic characteristics
  - C. Altitude
  - D. Exposure (Aspect)
  - E. Hydrology
    - 1. Surface hydrology
      - a. watershed considerations
      - b. flood plain delineations
      - c. surface drainage patterns
      - d. amount and quality of runoff
    - 2. Groundwater hydrology
      - a. groundwater table
      - b. aquifers
      - c. amount and quality of groundwater flows
      - d. recharge potential
  - F. Geology
    - 1. Stratigraphy
    - 2. Structure
    - 3. Geomorphology
    - 4. Chemical nature of overburden
    - 5. Coal characterization
  - G. Soils
    - 1. Agricultural Characteristics
      - a. texture
      - b. structure
      - c. organic matter content
      - d. moisture content
      - e. permeability
      - f. pH
      - g. depth to bedrock
      - h. color
    - 2. Engineering Characteristics
      - a. shrink-swell potential
      - b. wetness
      - c. depth to bedrock
      - d. erodibility
      - e. slope
      - f. bearing capacity
      - g. organic layers
  - H. Terrestrial Ecology
    - 1. Natural vegetation, characterization, identification of survival needs
    - 2. Crops
    - 3. Game animals
    - 4. Resident and migratory birds
    - 5. Rare and endangered species
  - I. Aquatic Ecology
    - 1. Aquatic animals—fish, waterbirds; resident and migratory
    - 2. Aquatic plants
    - 3. Characterization, use and survival needs of aquatic life system
- II. CULTURAL FACTORS
  - A. Location
  - B. Accessibility
    - 1. Travel distance
    - 2. Travel time
    - 3. Transportation networks
  - C. Size and Shape of the Site
  - D. Surrounding Land Use
    - 1. Current
    - 2. Historical
    - 3. Land-use plans
    - 4. Zoning ordinances
  - E. Land Ownership
    - 1. Public
    - 2. Industry
    - 3. Private
  - F. Type, Intensity and Value of Use
    - 1. Agricultural
    - 2. Forestry
    - 3. Recreational
    - 4. Residential
    - 5. Commercial
    - 6. Industrial
    - 7. Institutional
    - 8. Transportation/Utilities
    - 9. Water
  - G. Population Characteristics
    - 1. Population
    - 2. Population shifts
    - 3. Density
    - 4. Age distribution
    - 5. Number of households
    - 6. Household size
    - 7. Average income
    - 8. Employment
    - 9. Educational levels

Source: Ramani, Sweigard, and Clar, in press.

IMPORTANT CONSIDERATIONS. In backfilling, three major areas of long-term concern are slope stability, groundwater control, and water-quality maintenance (Phelps et al., 1981). The primary source of slope failure is the prolonged action of water through erosion and mass wasting. The primary result of water action on slopes is erosion and sediment production. Long uninterrupted slopes are undesirable as compared to ones with terraces and diversions that decrease slope lengths and direct runoff water to safe outlets. Additionally, while backfilling, the permeability of the fill for water percolation can be modified. Such modifications are achieved by sealing or leaving the coal seam open, and by selective placement of the spoil.

Technical requirements of reclamation vary depending upon the degree of restoration. In today's standard surface mining approaches, selective replacement of material is relatively simple. Contour mining approaches such as block-cut, box-cut, haulback, mountaintop removal, and modified area mining are generally sufficiently flexible to allow segregation and selective replacement (Stefanko et al., 1973; Saperstein and Secor, 1973; Grim and Hill, 1974; Ramani et al., 1977). Larger-scale area methods employing traditional overcasting are less flexible. However, selective placement of spoils in these cases can also be achieved by a combination of equipment and methods. For example, selection of shovel-truck systems, as opposed to conventional overcasting, provides the needed flexibility in spoil placement (Ramani et al., 1990).

METHODS AND EQUIPMENT. The backfilling method to be employed is related closely to the method of overburden removal. In area stripping operations where direct casting methods of overburden removal are used, the primary stripping equipment backfills the spoil in the pit. Following are the most common methods of direct casting:

1. Dragline direct casting.
2. Stripping shovel direct casting.
3. Bucket wheel excavator (BWE) direct casting via cross-pit conveyor.
4. BWE in combination with a dragline or stripping shovel where the BWE strips a portion of the overburden and spoils it on top of direct-cast shovel or dragline spoil.
5. Dragline rehandle; can be done with the primary dragline or a second dragline (usually with smaller bucket capacity) working in tandem with the primary dragline.

Haulage backfilling methods are common in contour and open pit type operations. Due to pit configuration and limited reach of the overburden removal equipment, the material must be transported some distance before it can be disposed properly. These methods can be categorized into the following two types:

1. Separate loading and haulage units; this includes, primarily, loading shovel-truck operations and front-end loader (FEL)-truck operations.
2. Single units such as a scraper, dozer, and FEL used in a load-and-carry operation.

Regardless of the backfilling method, some grading is required. The amount of grading, however, is dependent upon the method of backfilling. Direct casting methods produce, in general, very large spoil ridges that require extensive grading. BWE backfilling may produce narrower ridges than stripping shovels or draglines, but leveling is still required. This grading is performed using large- capacity track dozers. Haulage methods of backfilling generally produce a more even surface than direct casting methods. Some leveling of the surface is still required prior to soil reconstruction. This is also accomplished primarily with dozers. However, graders may also be used in some specialized situations such as the construction of drainage structures.

The applicability of various types of equipment to regrading and backfilling was evaluated by Skelly and Loy (Anon., 1975) with respect to spoil configuration, rehandle percentage, transportation distance, and final surface contour. This evaluation is summarized in Table 12.3.4. General algorithms for calculating the production capacity of fixed-base stripping equipment, loading and haulage equipment, and single-unit load-and-carry equipment are given in Chapter 9.4 of this text.

Various refinements of the above methods are being practiced. For example, low-ground-pressure dozers have been employed in reclamation operations to minimize compaction. However, their primary application is in the grading of soil which is addressed in the following segment. Another refinement is the variety of dozer-blade configurations available (Fig. 12.3.3). Blades are selected upon such considerations as required maneuverability, distance the material must be transported, type of material, and special applications. Specific characterisitics of the various type blades are summarized in Fig. 12.3.3 (Anon., 1982).

12.3.3.2 Soil Reconstruction

General Requirements. Topsoil and subsoil requirements for surface mine reclamation are found mainly in Section 816.22 of *CFR* Title 30. They require all topsoil to be removed as a separate layer and segregated. If the topsoil is less than 6 in. (150 mm) thick, the topsoil and unconsolidated material immediately below must be removed and treated as topsoil. Proper methods of stockpiling are specified when it is not practical to redistribute immediately. Redistribution, when it occurs, must achieve a stable uniform thickness, prevent excess compaction of the materials, and protect materials from wind and water erosion. Provisions exist for the use of substitute or supplemental material. However, the responsibility lies with the operator to demonstrate that the resulting medium is equally or more suitable for sustaining vegetation than the existing topsoil and that it is the best available material in the permit area for that purpose.

The operator may be required to remove and redistribute the B horizon, C horizon, or other underlying strata if the regulatory authority finds that this is necessary to comply with the revegetation requirements. Such measures are required for all prime farmland reclamation (Sections 821.12 and 821.14) and will be addressed under a separate subheading.

Important Considerations. Jansen and Melsted (1988) have summarized some of the theoretical and practical considerations of soil reconstruction. As with reclamation in general, the proposed land-use objective should be identified to insure that the reconstructed soil is capable of supporting that use. Various land uses are more suited to soils exhibiting different characteristics. However, the ability of the mine operator to reconstruct soil with any particular set of characteristics is limited by the nature of the available material. For surface mine reclamation, it can be assumed that, as a minimum, the soil must be capable of supporting vegetation similar to that which existed before mining. While the chemical properties of the soil are important for successful revegetation, they are easier to manage through soil amendments than physical properties such as texture. Therefore, when adquate topsoil is not available, or when substitute material is otherwise needed, special attention should be given to the texture, coarse-fragment content, and mineralogy of the material. When crop production is the intended use, the soil should consist of medium-textured materials (silt loams, loams, or silty clay loams), be free of coarse fragments, and have a clay content within the general range of 20 to 35% (Jansen and Melsted, 1988).

Bulk density and soil strength are two additional physical properties that are very important. Unlike texture, which is of principal concern in the selection of suitable material, bulk density and soil strength are concerns while the soil is being handled. Compaction caused by the equipment used for soil reconstruction causes bulk density and soil strength to increase. This inhibits root penetration and movement of air and water through the soil. The result may be unsuccessful revegetation. Therefore, it is important to minimize compaction of the soil while it is being reconstructed.

Equipment and Methods. Scrapers are used almost exclusively to remove the topsoil. Scrapers are also used frequently for subsoil removal; however, alternatives do exist. The use of shovels for subsoil removal has been increasing for two reasons: (1) it has been suggested that subsoil haulage and replacement by trucks results in less compaction than by scrapers, and (2) some companies have realized a cost advantage with trucks and shovels in comparison to scrapers. Another alternative to scraper removal of subsoil is to use BWEs. This is practical only when there is a rather thick mantle of unconsolidated overburden and will result generally in mixing of the B and C soil horizons. In at least one case, a BWE was used without topsoil removal as an approved experimental practice when there was evidence that the resultant soil mixture would be more suitable for the vegetation requirements.

Haulage and replacement of soil are related to the method of soil removal. Topsoil or subsoil removed by scrapers is typically hauled and replaced by the same machine. The soil is spread evenly over the site in lifts until the desired depth is achieved. A minimal amount of grading is required in the operation; however, the number of scraper passes necessary to satisfy soil reconstruction standards is blamed most frequently for the compaction problem. Roll (1987) has described a unique use of scrapers in combination with a BWE where the topsoil is removed by scrapers and stockpiled on the highwall side of the pit. Dozers then

Table 12.3.4. Equipment Rating—Regrading and Backfilling

		Dozers	Graders	Scrapers	Draglines	Front-End Loader	Front-End Loader and Trucks
Spoil Configuration	High peaks	1	—	2	2	3	3
	Moderate peaks	1	3	1	2	1	2
	Low peaks	1	1	1	2	1	1
% Spoil Rehandled	> 75%	1	—	1	2	2	1
	75%–50%	1	—	1	2	1	1
	50%–25%	1	—	1	3	1	1
	< 25%	1	2	1	—	1	1
Transport Distance	50–150 ft (15–46 m)	1	2	1	2	1	—
	150–300 ft (46–91 m)	1	—	1	2	1	—
	300–500 ft (91–152 m)	2	—	1	—	2	4
	150–1000 ft (152–305 m)	—	—	1	—	—	1
	> 1000 ft	—	—	1	—	—	1
Final Surface Contour	Flat & smooth	1	1	1	—	3	—
	Flat & rough	1	2	1	2	2	—
	Steep & smooth	1	—	—	—	3	—
	Steep & rough	1	—	—	2	2	—

Source: Anon., 1975.

LEGEND: 1. Should be considered.
2. May be considered.
3. May be considered under certain conditions.
4. May be considered special situation.

A. High.
B. Moderate.
C. Low.

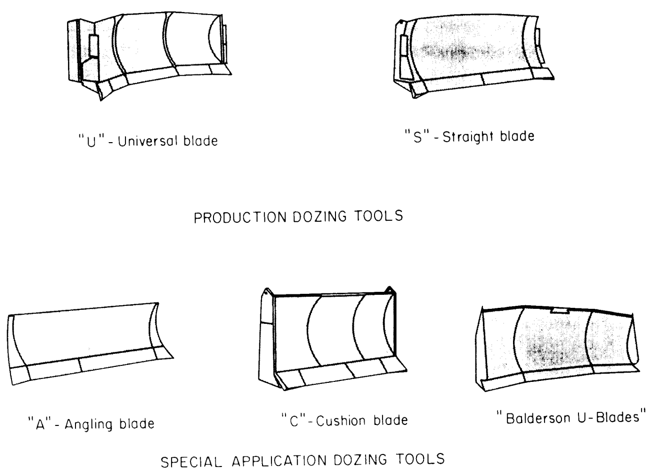

Fig. 12.3.3. Several types of dozing tools. The "U" blade is efficient for moving large loads over long distances. The "S" blade is the most versatile dozing tool. The "A" blade is designed for sidecasting. The "C" blade is used for on-the-go push-loading. "Balderson U-Blades" provide high-volume movement of light non-cohesive materials (Anon., 1982, by permission: Caterpillar, Inc.).

push the topsoil into slots in the highwall where the BWE loads the topsoil onto a cross-pit conveyor for replacement on previously graded wheel spoil. The BWE has also been used in conjunction with an around-pit conveyor transportation system and a spreader for soil replacement. More variability can be observed in the use of trucks for subsoil haulage and replacement than is seen in scraper operations. Both rear-dump and bottom-dump trucks are in use. Bottom-dump trucks traverse the graded spoil and previously replaced subsoil in a manner similar to scrapers. However, they are capable of replacing the subsoil in thicker lifts than scrapers, necessitating fewer passes. Also bottom-dumps can be used to windrow the subsoil, allowing dozers to spread it to the required thickness and thus minimize the extent of rubber-tired traffic on the soil. Rear-dump trucks can travel either on the graded spoil or on previously replaced subsoil. Operating conditions in the field determine to a great extent where the trucks must travel, but from a reclamation perspective, traveling only on graded spoil is preferred over traversing previously replaced subsoil. When direct placement of topsoil or subsoil is not possible, the material must be stockpiled, resulting in an additional loading and haulage component of the process (Sweigard and Siemsglusz, 1989).

An evaluation of various equipment types for topsoil removal and replacement, as conducted by Skelly and Loy (Anon., 1975), is summarized in Table 12.3.5.

12.3.3.3 Revegetation

General Requirements. Section 816.111 of *CFR.* Title 30 specifies that the vegetative cover required following mining must be diverse, effective, and permanent. It must consist of species that are native to the area. (Introduced species require special approval.) The vegetative cover must be at least equal in extent to the natural vegetation of the area. It must also be capable of stabilizing the soil surface from erosion. Section 816.114 requires that planting must be done during the first normal season for favorable planting conditions after replacement of the plant-growth medium. The standards for revegetative success are given in Section 816.116. Although the specific standards depend upon the approved postmining land use, the criteria used must be representative of vegetation on unmined lands in the area. In general, achieving 90% of the established

Table 12.3.5. Equipment Rating—Topsoil Removal and Replacement

		Dozers	Front-End Loaders	Scrapers: Elevating	Scrapers: Full-Power	Scrapers: With Push Tractor	Dragline	Shovel and Truck	Bucket Wheel Excavator	Front-End Loader and Truck
Topsoil Thickness	0–2 ft (0–0.6 m)	1	1	1	1	1	3	—	—	1
	2–5 ft (0.6–1.5 m)	1	1	1	1	1	3	4	3	1
Haul Distance	0–300 ft (0–91 m)	1	1	2	2	1	3	—	—	—
	300–500 ft (91–152 m)	2	2	1	1	1	4	—	3	—
	500–1000 ft (152–305 m)	—	—	1	1	1	—	4	—	3
	1000–1500 ft (305–457 m)	—	—	1	1	1	—	4	—	2
	1500–5000 ft (457–1524 m)	—	—	2	2	2	—	4	—	1
Flexibility Under Varied Field Conditions	good	A	A	A	A	A	A	A	B	A
	fair	A	A	A	A	A	A	B	B	B
	poor	B	B	B	B	B	A	C	C	C

Source: Anon., 1975.

LEGEND: 1. Should be considered.
2. May be considered.
3. May be considered under certain conditions.
4. May be considered special situation.

A. High.
B. Moderate.
C. Low.

standard is considered successful revegetation. Revegetative success cannot be achieved in one successful growing season, however. In areas that receive more than 26 in. (660 mm) average annual precipitation, the responsibility period is five years. Where the average annual precipitation is 26 in. (660 mm) or less, the responsibility period is 10 years.

IMPORTANT CONSIDERATIONS. Revegetation operations must be planned and executed in light of the approved postmining land use. Many considerations such as species selection, soil amendments, and methods of planting are related specifically to the land use. Other considerations such as soil stabilization, water infiltration, and compaction are more independent of the end use. Powell (1988) has addressed many of the major considerations involved in reclaiming surface-mined land to a variety of uses. Some of these considerations are summarized in the following discussion.

In the case of reforestation, one of the primary concerns is competition from herbaceous vegetation. This requires control either by scalping or proper uses of herbicides. In reclaiming land for pasture and hayland, one must be concerned with water control, selection of best species for the climate, and time of seeding, fertilizer, final seedbed preparation, seeding method, and proper use of surface mulches. The exact method of final seedbed preparation should depend upon the species selected. However, for reclaimed land, a rougher surface seems better because it results in greater infiltration and a less dense rooting zone. Smoother and firmer seedbeds necessitate the use of surface mulches.

The primary considerations for establishing cropland are similar to those for pasture/hayland such as seedbed preparation. An additional concern in cropland reclamation is the addition of organic matter required for good crop production. This is often accomplished by establishing a quick-growing cover crop of grasses and legumes, then using this as “green manure” when the cropland species are planted at a later time. Rangeland, which is an important postmining land use in the western United States, presents the major concern of less than adequate water availability. Tillage tools can be used that leave a rough surface allowing the available water to be trapped and to infiltrate the ground. Seed drills developed especially for rough terrain can be used to provide good contact between the seed and soil particles. For reclaiming rangeland, mulching is important and supplemental irrigation is desirable in establishing the initial stand.

METHODS AND EQUIPMENT. The selection of appropriate species is a key concern once the land-use objectives have been established. Various extensive lists of recommended species are available in the literature. The reader is directed to a recent USDA Forest Service publication by Vogel (1987) for a detailed listing of species recommended for surface mine reclamation in both the eastern and western United States.

Seedbed preparation can consist of two phases: (1) primary tillage and (2) secondary tillage (Lyle, 1987). Primary tillage penetrates rather deeply, breaking the soil into large clods through the use of some type of plow (moldboard, chisel, disc, etc.), a subsoiler, or a rotary tiller. This produces an uneven surface with large clods. Therefore, secondary tillage is performed, usually with some type of harrow (disc, spike-tooth, spring-tooth, tine, roller, etc.), to produce a suitable seedbed.

Drilling and broadcasting are the two basic methods of seed planting. Seed drills create a furrow with an implement, drop a seed into the furrow, and cover the seed immediately with soil. In the broadcast method, seeds are scattered over the surface of the ground by mechanical means or by hand for very small areas. Large areas can be seeded by hydroseeders or even airplanes. Direct seeding of most tree and shrub species has not been very successful on reclaimed land. Therefore, hand planting or the use of a mechanized seedling planter is usually required (Vogel, 1987).

Common approaches to soil amendments involve the addition of lime, fertilizer, and mulch as needed. Many mulches have been used to assist in establishing vegetative cover. Straw or hay can be manually applied, but they as well as wood chips are often spread mechanically or hydropneumatically. Mulches should be held in place with tar or chemical emulsion (Ramani et al., 1990).

Fly ash and stabilized sewage sludge are other amendments that can add needed nutrients to reclaimed mined land. However, use of these recycled amendments requires considerable testing and characterization of the material. For example, the composition of different fly ashes varies, and some may prove toxic to plant growth. Also care must be taken to avoid poisoning the soil with too high a level of heavy metals and salt from sewage sludge (Grim and Hill, 1974).

A summary of the revegetation equipment evaluation performed by Skelly and Loy (Anon., 1975) is given in Table 12.3.6.

12.3.4 RECLAMATION OF MINE WASTE DISPOSAL SITES[1]

12.3.4.1 Waste Disposal Problem

Solid wastes consist mostly of overburden and waste rock from mining, together with tailings, slurries, and slimes from processing (Andrews, 1975), but include some waste from the treatment of air and water, ordinary garbage, and debris from construction and discarded equipment. About twice as much waste results from mining as from processing, most of the mining waste being overburden.

The physical and chemical properties of this waste influence its stability and handling, as well as the potential for further beneficiation and for recovery of byproducts. These properties further influence the procedures that are needed to protect public health and to control environmental effects. The physical properties also affect the appearance of the solid waste and may strongly influence its ultimate form on the land surface, as in the case of disposal of tailings as a slurry (Brawner and Campbell, 1972). The chemical properties of this waste, besides being primarily responsible for any possible hazards to public health, are the factors that mainly determine its suitability for further processing and its effect on the physical and biological environment.

[1] This section is reprinted with permission from “Surface Mining of Non-Coal Minerals,” a report prepared by the Committee on Surface Mining and Reclamation, National Research Council, National Academy of Sciences, Washington, DC, 1979.

12.3.4.2 General Reclamation Practices

Certain areas of agreement are found in the technical literature on what approaches and goals are appropriate for reclamation in mining, including the economic desire for maximum results from a given outlay of money (O’Neil, 1977). These goals clearly influence current practices in that most mining companies now reclaim or rehabilitate land and disposal areas to some degree.

Effective reclamation, of course, serves many purposes. For most sites, reclamation is understood as rehabilitating the disturbed area by grading in a manner compatible with the surrounding landscape, or with the intended postmining use, and establishing a self-sustaining vegetative cover. Reclamation is intended to control erosion, to restore the hydrologic balance, and to limit the offsite impacts, during and after mining. The ultimate purpose is to restore the mining site to productive use, for example, for agriculture, forestry, industry, housing, recreation, or fish and wildlife habitat. Desirably, this use should be at least as beneficial as the capability of the site before mining, although not necessarily for the same purposes.

In assessing the rehabilitation potential of a particular mining area, a number of factors are relevant. Some of these, according to O’Neil (1977), are (1) magnitude and topography of the disturbed land, (2) chemical properties of the mining wastes, (3) physical properties of the waste, (4) an economic analysis of the mining and reclamation plan, and (5) climatic factors. In arid regions, the quantity and availability of water for irrigation is an especially important consideration, although for most commodities the predominant water use is for processing and waste disposal. The possibility of reclaiming a mining area for social needs may also be pertinent.

A long-term ecological approach to land reclamation deemphasizes the agricultural aspects and places greater weight on successional processes so as to develop self-perpetuating plant and animal communities (Curry, 1975; Ludeke, 1977; Wahlquist, 1976). This ecological approach stresses the concept of minimal impact and recommends that a “mining ecology” be developed,

Table 12.3.6. Equipment Rating—Revegetation

		Airplane	Hydroseeder	Broadcast Spreader	Drill Seeding	Disk and Row Planting	Hand Planting or Spreading	Mulching Machine
Acreage (hectares)	3	—	1	1	1	1	3	1
	3–10 (1.2–4.0)	—	1	1	1	1	4	1
	10–25 (4.0–10.1)	—	1	2	1	1	—	1
	25–50 (10.1–20.2)	3	1	2	2	2	—	1
	> 50 (20.2)	2	1	3	4	3	—	1
Seed Mixture	Grasses	1	1	1	1	3	3	—
	Crops	—	3	2	1	1	3	—
	Seedlings	—	—	—	—	1	2	—
Mulching	—	—	1	—	—	—	3	1
Fertilizer and Lime	—	1	1	3	1	1	3	—

Source: Anon., 1975.

LEGEND:
1. Should be considered.
2. May be considered.
3. May be considered under certain conditions.
4. May be considered special situation.

A. High.
B. Moderate.
C. Low.

integrating mining and nonmining uses of the land under a system that reflects an understanding of the ecology of the area (Bonham, 1976; Wali and Kollman, 1977). That is, the ecological approach attempts to limit long-term impacts that would reduce the plants and animals of an area or limit biological productivity.

Rebuilding the soil is thought to be the key to lasting reclamation success (Cryderman and Shetron, 1976; Dean et al., 1969). Soil development is greatly influenced by the available moisture, a factor that may severely inhibit reclamation success in arid regions (Berg, 1972; Brown, 1976; Hodder, 1977). Large-scale contouring and grading, when used with small-scale surface treatment, is often beneficial in conserving moisture and reducing erosion (Brown, 1976; Draskovic, 1973; Anon., 1973; Ludeke, 1977; Stephan, 1977). Machines have been developed to meet these needs (Hodder, 1976, 1977).

To plan for an ecological approach to reclamation, specific baseline data are needed, beginning with exploration and continuing through mine development (Clark, 1974; Anon., 1973; O'Neil, 1977; Wahlquist, 1976). Such an inventory includes information on all the physical and biological features—for example, the groundwater hydrology and characteristics of the overburden (Schuman et al., 1976; Dean et al., 1973; Packer, 1974).

Although aesthetic appeal is seldom a primary reclamation objective, it deserves some consideration in most mining and reclamation plans (Beverly, 1968; Anon., 1974; Arthur et al., 1977). Surfaces can be shaped and waste material from mining and processing can be placed such that a desired landscape design is achieved, thus helping to control erosion by wind and water and leaching of unwanted constituents, as well as creating areas having new and productive uses. Recreational lakes are an example, particularly in old quarries. For this purpose, the services of landscape architects can be helpful (Chironis, 1977; Down and Stocks, 1977; Greenwalk, 1976; Matter et al., 1974).

Revegetation is currently the preferred method of stabilizing disturbed sites (Dean et al., 1973, 1974). Some of the recommended practices are as follows:

1. Burying materials poisonous to plants (Farmer et al., 1976).
2. Leaching of tailings and other chemical methods of treatment to ensure success of plant growth (Anon., 1973; James, 1966; Kenahan and Flint, 1972; Nielson and Peterson, 1973; Watkin and Winch, 1974); e.g., allowing the land to lie fallow after its initial grading can increase leaching (Gemmell, 1975; Ludeke, 1977).
3. Combining chemical stabilizers with vegetation to enhance control of erosion (Brown, 1974; Kay, 1977).
4. Adding fertilizer in beneficial amounts, but avoiding continuous use that can hinder the development of self-perpetuating vegetation (R.W. Brown, 1976; Gemmell, 1975; Gordon, 1969; Shetron and Duffek, 1970).
5. Using sewage sludge as a soil amendment in order to control erosion and to stimulate plant growth (Dean and Shirts, 1977; Dean et al., 1973; Anon., 1973; Gordon, 1969; Hodgson and Townsend, 1973; Ludeke, 1977; Peters, 1974; Watkin and Winch, 1974). Other organic soil additives such as mulches can hold seed in place and improve the water balance of the soil (Hodder, 1977; Ludeke, 1977).
6. Encouraging nutrient recycling (Peters, 1974) and nitrogen-fixing plant growth as a means of increasing soil productivity (Berg, 1976).
7. Using microbial soil inoculants from neighboring undisturbed soils to provide beneficial microorganisms that enhance vegetation growth (Aldon, 1976; Reid and Grossnickle, 1978).
8. Adding topsoil, where feasible, to improve revegetation success (Brown, 1976; Anon., 1973; Schuman et al., 1976).
9. Increasing plant diversity and the number of native plants to improve the potential for plant survival (Aldon, 1976; Brown, 1976; Chosa and Shetron, 1976; Curry, 1975; Ludeke, 1977; Usai and Suzuki, 1973).
10. Planting more easily grown, non-native plants as a nurse crop for native plants (Berg, 1976; Dean et al., 1969; Hodgson and Townsend, 1973). Species capable of withstanding harsh site conditions, particularly high concentrations of metallic ions, may be advantageous for initial planting (Goodman et al., 1973; Kenahan and Flint, 1972; May, 1967; Stephan, 1977).
11. Collecting native seeds or plants in an area and then planting them on disturbed areas (Ludeke, 1977).

Although revegetation is the most commonly accepted approach to reclamation, alternatives do exist. One procedure would be to use the mine waste as building or construction material (Kenahan and Flint, 1972; Spendlove, 1977).

RECLAMATION OF WASTE ROCK. Control of air and water pollution from piles of waste rock may be enhanced by suitable placement, construction of diversion ditches, and the like, but a stable cover on the waste pile can further reduce fugitive dust and impede the loss of surface moisture. The cover could consist of rocks, gunite, asphalt, concrete, smelter slag, bark, or replaced topsoil. Among these materials, topsoil is the only cover that blends with the natural surroundings and that serves biological needs.

Chemical stabilizers, although not intended to be permanent, can effectively hold down dust in extreme situations and inhibit erosion by water. These chemicals are particularly useful in areas of harsh climates or where topsoil is not available.

RECLAMATION OF OPEN PITS. It is axiomatic that a mineral can be mined only where it is found. Thus, if an ore deposit is mined by open pit methods, the resulting pit may be more or less conspicuous, depending on geological circumstances. Active open pits, as compared with surface (open cast) coal mines, provide little opportunity, if any, for simultaneous mining and reclamation because the pit continues to expand and deepen as long as the mine is producing. Also the ultimate depth and shape of the pit, although roughly predictable, are dictated by the economics of mining and the geometry of the ore deposit rather than by particular reclamation goals. Indeed, the very size of a large open pit would make restoration by backfilling, or even by reshaping, an enormous economic burden of uncertain benefit, and inactive open pits could be reactivated if economic conditions became favorable. Thus, in practical terms, reclamation of open pits is limited to planning for the placement of rock dumps and tailings ponds that will remain when the mining operation is closed. Principles of landscape design can be applied at little additional cost in placing these materials in a manner that achieves beneficial postmining land use (Matter et al., 1974). The landscaping could be done by the mining company itself, or redevelopment rights could be leased or sold. The potential forms of redevelopment vary significantly from one mine setting to another, as do considerations of economic, social, and physical feasibility.

In summary, where large volumes of material are handled in open-pit operations, redevelopment could be a legitimate post-mining land use, in lieu of reclamation aimed only at establishing the original conditions. Such use can yield an economic return to offset the costs of reclamation and maintenance.

RECLAMATION OF TAILINGS. Of all the mining wastes, mill tailings are the most intractable to reclamation, in part because their amounts can be very large, from 30,000 to 100,000 tons (27,000 to 90,000 t)/day (Soderberg and Busch, 1977). A complicating factor is that reclamation of waste rock and tailings could be premature if future technology or economic conditions make them attractive as deposits of lean ore (Mantell, 1975). However, the need to limit airborne dust may require that tailings be reclaimed in a timely manner.

Revegetation of tailings is inherently difficult, regardless of the ore mined, because tailings seem to be physically incapable of supporting higher plants (Dean et al., 1969). Their fine-grained particles, which are easily picked up by wind when dry, and which are physically unstable when saturated, are also a major problem for reclamation. In addition, tailing ponds may have steep outer slopes, deficient water for plant growth, and severe surface temperatures (Down and Stocks, 1976). Finally, the tailings may be cemented or compacted, and old tailings can develop a hard oxidized layer about 8 in. (200 mm) below the surface (Peters, 1974).

Water is an extremely limiting factor in revegetating tailings (Cook, 1976; Ludeke, 1977; Hodder, 1977). Where the precipitation exceeds 20 in. (500 mm), revegetation problems are considerably simplified, but reclamation in the arid West and at high altitude or high latitude requires special techniques and comparatively greater effort.

Reclamation of tailings must start by building physically stable gradients and by then developing a medium for plant growth on the tailings. This cover should resemble undisturbed soils near the mine in order to minimize long-term maintenance and to promote the growth of native plants (Curry, 1975). Adverse characteristics of the tailings must be mitigated, namely deficient nutrients, extreme alkalinity or acidity, excessive salts and heavy metals, deficient microbial populations, and frequent blowing dust (Dean et al., 1974). Soil stability can be achieved through the use of mechanical surface manipulation, mulches, planting of annual species, and finally, by establishing permanent vegetation (Hodder, 1976). Soil moisture can be increased by impeding runoff, increasing infiltration, and reducing evaporation.

Soil neutralization, which enhances plant growth and reduces the mobility of heavy metals (Nielson and Peterson, 1973), can be accomplished through either natural or induced leaching. Spraying an area with a fine mist until leaching progresses below the root zone has been successful in some cases (James, 1966). Drip irrigation also has been used to leach the root zone, but its practicality in large revegetation projects is limited (Shoji, 1977; DeReemer and Buch, 1977). In humid areas, letting a site lie fallow for one or two years will allow natural leaching of salts (Gemmell, 1975). Other neutralization procedures include adding sulfuric acid to saline and alkaline soils, and treating acidic soils with limestone or fly ash (Dean and Shirts, 1977). If fly ash is used, the toxic elements that are commonly found in fly ash can be bound up as organic complexes by use of compost, manure, or barley straw. Placing the fine and coarse fractions of tailings in layers also helps to reduce the salinity (Dean et al., 1973).

Tailings are poor in nutrients, and use of commercial fertilizer is often recommended, even though the resulting plant cover may then require continued applications. Cook (1976) says that commercial fertilizers should not be applied until the second year, if used at all. Inoculation of tailings waste with native soil or commercial mycorrhiza (a microorganism found in soils) has been shown to be particularly helpful in establishing vegetation (Aldon, 1976; Reid and Grossnickle, 1978).

Applying sewage sludge to tailings appears to enhance revegetation (Peters, 1974). Furthermore, Hodgson and Townsend (1973), from revegetation experiments on pulverized fuel ash, found that any bulky ameliorant such as sewage sludge, shale, or peat improved the growth medium. Spreading the sludge can alleviate blowing dust on tailings ponds (Peters, 1974) and is more effective than topsoil in creating a growth medium where soils are poorly developed (Ludeke, 1977). Sludge increases the water-holding capacity of the soil, moderates the pH, helps to dilute any phytotoxic materials that may be present, and is a good source of nutrients (Dean and Shirts, 1977).

Legumes and other nitrogen-fixing plants should be encouraged to enrich the soil in nutrients (Hodgson and Townsend, 1973), and early growth on tailings should be plowed back in to conserve nutrients and reduce the need for fertilizer (Peters, 1974). For this reason, harvesting during the initial years is not recommended, and grazing by livestock and wildlife should be prevented.

Native species are often recommended (Aldon, 1976; Curry, 1975; Ludeke, 1977), and transplanting them to disturbed areas has been found to be effective (Ludeke, 1977). Whether or not native species are used initially, they may become established on the tailings after non-native plants have been growing for a few years.

Both broadcast seeding (Hodder, 1976) and drilling (Cook, 1976) have been recommended as methods of planting. Also most authors agree on the importance of some type of mulch to protect seeds, keeping them in place and providing some additional soil moisture (Hodder, 1977; Kay, 1977). Hydroseeding with a combination of mulch, seeds, and chemical stabilizers can provide a stable medium for germination (Day and Ludeke, 1979); the most successful chemicals are elastomeric polymers (Kay, 1977; Berg, 1972).

12.3.5 SPECIALIZED RECLAMATION PROBLEMS

12.3.5.1 Prime Farmland

The reclamation of prime farmland that has been disturbed by surface mining presents a special challenge to reclamation personnel. Land designated as prime farmland by the Soil Conservation Service must be reclaimed according to standards given in Part 823 of *CFR* Title 30. There are two general classes of regulations that must be satisfied: (1) soil reconstruction standards and (2) revegetation success standards. Soil reconstruction standards require that all topsoil must be removed and replaced as a separate layer. The B or C horizons or other suitable soil material must be removed also and replaced as a separate layer so that a minimum reconstructed soil thickness of 48 in. (1.2 m) is achieved. Revegetation success standards specify that the average yield must equal or exceed the average yield of a reference crop grown on similar nonmined soils in the surrounding area. This yield must be reached for a minimum of three years, and the reference crops must be selected from the crops produced most commonly on the surrounding prime farmland. The row crop requiring the greatest rooting depth must be one of the reference crops. In many cases, this crop will be corn.

Surface mine operators have attempted, in general, to satisfy the prime farmland soil reconstruction requirements using conventional construction equipment. The method used most often has been scraper haulage. Ground pressure from reclamation equipment (especially rubber-tired equipment) causes the bulk density and strength of the soil to increase, resulting in restricted root growth (Albrecht and Thompson, 1982). Soil structure is also affected by soil handling, resulting in less water and air being available to plant roots. The resultant soil condition is referred to generally as compaction.

Excessive soil compaction is cited as the major obstacle to returning agricultural land to high levels of productivity (Anon., 1985). At present, two general solutions to the problem are being considered. The first is to employ alternative methods of soil reconstruction that limit the amount of soil compaction (Albrecht and Thompson, 1982; Jansen and Dancer, 1981; Blakely, 1980; Roll, 1987; Sweigard and Escobar, 1989).

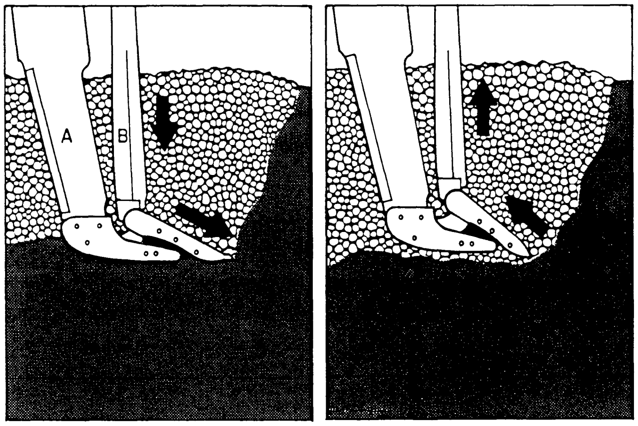

Fig. 12.3.4. Cut-lift system of the Kaelble-Gmeinder TLG-12 shanks: a) cutting—preloosening phase that causes coarse fracturing; b) lifting—fine-loosening phase that results in soil shattering (Chugh and Prasad, 1983).

Table 12.3.7. One-Year Corn Yield Data from Illinois Test Plots

Treatment	Area #2	Area #3	Mean
	yield, bu/a		
Scraper/TLG	136	108	119 a*
Scraper	117	56	91 b
Undisturbed	94	94	94 b

*Yields within a column with a similar letter are not significantly different at the 0.05 level. Conversion factor: 1 acre = 0.404 hectare.

Source: Hooks, Jansen, and Holloway, 1987.

Several of these methods involving truck-shovel combinations and BWEs are addressed in 12.3.3.2. In many cases, the alternative method is used only for subsoil replacement, requiring the use of scrapers to replace the topsoil.

The second general solution is to augment the soil after it has been reconstructed. Deep tillage has been the primary means of augmentation (Albrecht and Thompson, 1982; Hooks, Jansen, and Holloway, 1987). Soil-loosening techniques include but are not limited to ripping, plowing, discing, and rotary tillage (Chugh and Prasad, 1983). Most devices have been developed specifically for agricultural applications and are effective, under favorable conditions, in the upper 12 to 30 in. (300 to 760 mm) of replaced soil. A few deep-tillage machines have been designed to loosen soil below 30 in. (760 mm). One such device is illustrated in Fig. 12.3.4. This machine has three moving shanks that break the soil by a “cut-lift system.” The “cutting” portion of the system consists of the downward and forward motion of the shanks (Fig. 12.3.4a), which fractures the soil coarsely. The fine loosening is accomplished by the “lifting” portion of the system through the upward and backward motion of the shanks (Fig. 12.3.4b). Actual yield data given in Table 12.3.7 indicate a positive response to this method of deep tillage at one site studied. It is important to note that the effectiveness of deep tillage is greatly dependent upon soil moisture. Soils with moisture contents near the plastic limit will resist fracturing. Therefore, deep tillage will be most effective when it is performed under dry field conditions.

Another problem encountered in returning reclaimed land to full agricultural productivity is differential settlement of the ground surface. Differential settlement results in the formation of depressions where the soil remains saturated for extended periods of time. Impeded aeration resulting from poor drainage can inhibit revegetation. This presents a problem not only in terms of increased reclamation costs but also in potential delays in obtaining final bond release.

The mechanism by which differential settlement occurs has not been identified definitively. However, its occurrence seems to be related to such factors as the method of spoil placement, amount of unconsolidated material in the overburden, the moisture content of the spoil when it was placed, and reestablishment of the water table within the spoil. The methods of dealing with the problem are fairly straightforward. The effects of differential settlement can be reduced greatly by eliminating perfectly flat surfaces. In recontouring the surface, a slope of at least 1” will prevent most closed depressions from forming. When closed depressions do occur, remedial work can involve land leveling, construction of grassed waterways to drain low areas, and additional earthmoving in the most extreme cases (Sweigard, 1987).

12.3.5.2 Adverse Soil Conditions

ACIDIC SOILS. In addition to an acidic environment for plant roots, high acidity in the reconstructed soil contributes to higher concentrations of such toxic elements as iron, aluminum, manganese, selenium, chromium, nickel, and cadmium. It also contributes to a reduction in the availability of phosphorus (Vogel, 1987).

Application of lime is the most common method of treating acidic soil. Lime is available in various forms with different neutralizing capabilities. Liming materials are classified according to their calcium carbonate equivalent (CCE), that is, the relative neutralizing ability of the material expressed as a percentage of the neutralizing capacity of pure calcium carbonate ($CaCO_3$). Agricultural lime, which is finely ground limestone, is used most frequently. This material varies in quality and may have a CCE as low as 80%. The particle size of agricultural lime is important. If it is not ground finely enough, additional quantities may be needed. Hydrated lime (calcium hydroxide) has a CCE of 135%; however, it is more expensive than agricultural lime. Dolomitic limestone (calcium-magnesium carbonate) usually has a higher neutralizing potential than agricultural lime but may be toxic to plants in soils that already have high magnesium concentrations. Burnt lime (calcium oxide) is used infrequently. Some wastes such as calcium silicate slag and alkaline fly ash or bottom ash can be used, but care must be exercised to prevent toxicity problems caused by other elements in the waste (Vogel, 1987).

The rate of lime application is based upon soil pH. It is desired generally to achieve a soil pH of 5.5 to 6.5, depending upon the type of vegetation to be supported. The higher values in this range are desirable for most agricultural uses. A crude estimate of the lime application rate based upon soil pH is given in Table 12.3.8. It is recommended that lime be incorporated at least 6 in. (150 mm) into the soil using a disc or chisel plow. For extremely acidic material, deeper incorporation is suggested, and the application rate should be adjusted accordingly. When topsoil is being replaced on top of acidic spoil material, it is advisable to apply the lime and incorporate it into the spoil before the topsoil is replaced (Vogel, 1987).

SALINE AND SODIC SOILS. High concentrations of soluble salts in mine soil result in an increase in osmotic pressure of the soil solution with impedance of water uptake by plants. Therefore, plants growing in this environment are extremely suscepti-

Table 12.3.8. General Guide for Lime Application to Mine Soil

pH (soil-water suspension)	Rate (tons/acre CCE)
6.1 and higher	None
6.0 to 5.5	1 to 2
5.4 to 4.6	3 to 4
4.5 to 4.0	5 to 6

Conversion factors: 1 ton = 0.907 t, 1 acre = 0.404 hectare.
Source: Vogel, 1987.

ble to moisture stress. The effects of excessive sodium are somewhat different. Sodium has a dispersive effect on clay particles and dissolves organic matter. This results in a soil that has poor structure, low infiltration capacity, and is highly erodible. Any soil with an exchangeable sodium percentage (ESP) greater than 15 is considered sodic. Very fine-textured sodic surface mine spoils are common in the western US (Sandoval and Gould, 1978; Jansen and Melsted, 1988).

Several management and improvement principles have been reported by Sandoval and Gould (1978). These include (1) establishment of drainage to lower water tables, (2) leaching of excess soluble salts, (3) replacement of exchangeable sodium, and (4) rearrangement and aggregation of soil particles to improve structure. In areas of low precipitation, irrigation is suggested as a means of leaching. Application of gypsum (calcium sulfate) is a common method of facilitating sodium ion replacement. Calcium chloride, ammonium nitrate, and ammonium sulfate can also be used (Vogel, 1987).

Wherever possible, undesirable material should be buried deep in the spoil and covered with better quality overburden. In particular, the graded spoil should be covered with nonsaline and nonsodic soil material (Jansen and Melsted, 1988). The upward migration of soluble sodium may have an adverse effect on vegetation over time, even on topsoiled sites. However, incorporation of gypsum does seem to deter this migration. As a final precaution, it is recommended that sodium-tolerant species be selected for revegetation (Vogel, 1987; Sandoval and Gould, 1978).

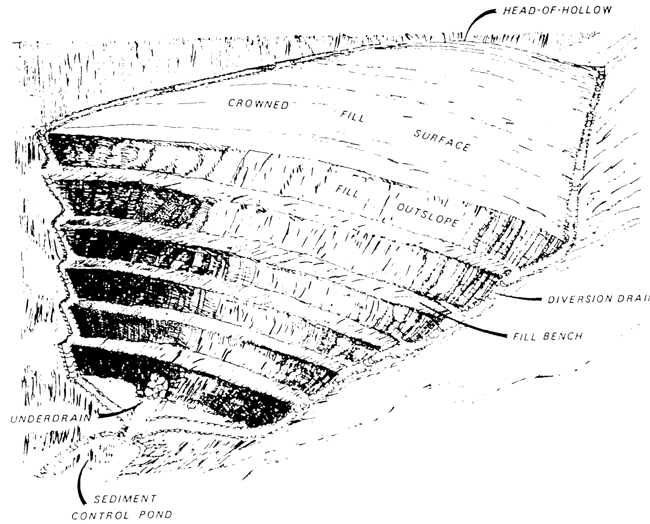

Fig. 12.3.5. Sketch showing typical head-of-hollow fill structure (Anon., 1979).

Table 12.3.9. Minimum Size of Main Underdrains for Different Amounts and Types of Valley-fill Material

Total amount of Fill Material	Predominant Type of Fill Material	Minimum Size of Drain ft (m) width	height
< 1 million yd^3 (764,000 m^3)	Sandstone	10 (3)	4 (1.2)
	Shale	16 (5)	8 (2.4)
> 1 million yd^3 (764,600 m^3)	Sandstone	16 (5)	8 (2.4)
	Shale	16 (5)	16 (5)

Source: Anon., 1981.

12.3.5.3 Excess Spoil Disposal in Mountainous Terrain

Disposal of excess spoil is a problem for surface coal mining operations employing both steep-slope mining and mountaintop removal methods. Steep-slope mining includes those methods employing lateral movement of overburden that result generally in a return to the approximate original contour (AOC), while mountaintop removal results in a flat or gently rolling surface that is exempted from AOC requirements due to an approved alternative land use. Various safety and environmental problems can be associated with these disposal sites that are situated generally in valleys and head-of-hollows. A typical head-of-hollow fill is illustrated in Fig. 12.3.5. Specific standards for their construction are given in federal and state surface mining regulations. This problem was studied in detail by the Committee on Disposal of Excess Spoil (CODES) of the National Research Council (Anon., 1981). Following is a summary of the committee's findings, including an overview of the engineering concerns. However, the reader is referred to the full report by CODES for a more in-depth handling of the subject.

The general requirements for disposal of excess spoil are given in Sec. 816.71 of *CFR* Title 30. To ensure mass stability, the design must provide a minimum long-term static safety factor of 1.5. The fill must be constructed on the most moderately sloping and naturally suitable area available. When spoil is disposed on a downslope greater than 2.8:1, either a keyway cut to bedrock or a rock-toe buttress must be provided. Before any material can be placed, all vegetation and organic material must be removed. The excess spoil must be placed in horizontal lifts no greater than 4 ft (1.2 m) thick and must be compacted concurrently as necessary to provide stability. Adverse effects on surface water and groundwater must also be minimized. Surface diversions are necessary to prevent spoil erosion and infiltration into the fill. Also the fill is to be underlain by drains constructed of durable, nonacid-, nontoxic-forming rock. Minimum sizes of main underdrains for different types of valley fills are given in Table 12.3.9. Lateral drains are to be constructed from seeps and springs to the main drains. Rock-core chimney drains may be used in head-of-hollow fills instead of underdrains and surface diversions. The fills must have no permanent impoundments. They must be revegetated and suitable for the approved postmining land use.

A site investigation should precede the design of any fill for excess spoil disposal. This includes geologic and geotechnical investigations of the foundation conditions. The primary function of these investigations is to identify any materials or conditions that could lead to failure (e.g., a layer of weak shale with a particularly unfavorable orientation). Information that is required before a design can be attempted includes (1) engineering properties (density, cohesion, and internal angle of friction) of the spoil and underlying material; (2) pore-water pressure anticipated within and beneath the spoil; and (3) the nature of external forces (e.g., blasting or earthquake). After this information is obtained, various embankment configurations can be considered and their stability analyzed. Various methods of analysis are available. However, the method used most commonly is the limiting equilibrium analysis. Using this method, driving forces and resisting forces can be estimated considering different failure surfaces (both curved and planar), and a factor of safety determined for each case. The design process leads to the selection of an embankment configuration that satisfies the factor of safety requirement (1.5 in most cases) for the critical failure surface. After the fill has been completed, it is advisable to monitor its performance through the measurement of such parameters as surface settlement and pore-water pressure.

12.3.5.4 Highwall Elimination

Another specific reclamation problem encountered, particularly in surface coal mining, is highwall elimination. The AOC provisions of SMCRA require that, with very few exceptions, all highwalls must be removed. Exceptions may be granted for remining in steep-slope regions where adequate spoil is not available to satisfy the requirements. In those cases, the requirements must be satisfied to the extent that is feasible. Exceptions also may be granted in some western states where portions of highwall can replace cliff-type habitats disturbed by mining. The provisions for a variance relating to AOC based upon an alternative approved land use do not generally include leaving highwalls, however. The problem of highwall elimination differs between contour mining and area mining operations. The Committee on Highwalls and Approximate Original Contour (COHAOC) of the National Research Council has investigated this problem on a national level (Anon., 1984). A portion of the findings are summarized in the following discussion.

In the steep-slope regions where contour methods are employed, there seem to be some differences among state regulatory authorities on the interpretation of highwall elimination. Some states allow no exposed rock face while others permit a few feet (meters) of exposed rock when it forms one side of a drainageway. The major engineering problem related to this requirement seems to be whether it is possible to eliminate the highwall completely and still achieve the required factor of safety (in the range of 1.3 to 1.5). This is a valid concern since many natural slopes in the region are known to be unstable. The theoretical relationship between slope angle, internal angle of friction, and factor of safety is illustrated in Fig. 12.3.6. This relationship is based upon an assumption of zero pore water pressure within the spoil. Water within the spoil, however, can both increase the driving force due to gravity and decrease the shearing resistance. Pore water pressure can be related to several factors including (1) infiltration into settlement cracks at the contact between spoil and highwall, (2) undisturbed barrier at the outcrop that restricts drainage, and (3) auger holes that may drain into the spoil. The required safety factors may be difficult to achieve in slopes greater than approximately 25°, but several measures will enhance the stability of these slopes. Compaction of the fill material increases the effective friction angle and should be done carefully. The build-up of pore pressure at the toe of the fill can be minimized by providing drains at acceptable intervals in the outcrop barrier. Also diversion ditches or terraces can be constructed to prevent water from flowing over the face of the slope. These diversions must have adequate slope to minimize infiltration.

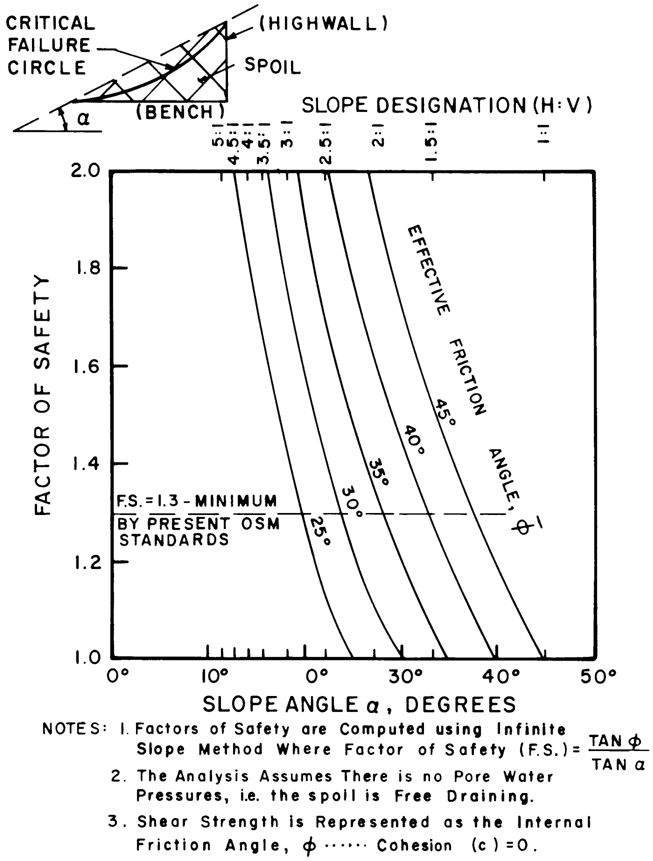

Fig. 12.3.6. Graphic representation of relation of slope stability to slope inclination for varying spoil shear strengths (Anon., 1984).

Finally, a layer of coarse-textured durable rock on the pit floor helps to improve internal drainage.

Highwall elimination is less of a problem in area mining situations. One reason is that the relative length of highwall to be reclaimed is smaller, since it is only the final cut in area mining that is affected. Secondly, stability is not a serious problem, since steep slopes are eliminated more easily in flatter terrain. Two general methods are being practiced depending upon the site conditions and specific state regulations. One method is to backfill the pit and grade the side slopes to a gradient that is less than or equal to a value specified by the regulatory authority. In this case, drainage is provided so that no water is impounded. Another method practiced in some states allows water to fill the final cut with grading of the highwall and spoil to acceptable slope angles down to an elevation slightly below the estimated minimum water level. Both methods may require some additional blasting of consolidated material in the highwall to satisfy the maximum slope requirements. Final cut impoundments present some interesting land-use potentials including recreational use and wetlands. However, impoundments may not diminish the quality or quantity of water used by nearby landowners, and the quality of the impounded water must be permanently suitable for its intended use. Potential water quality problems can be minimized by covering the exposed coal and segregating toxic spoil.

12.3.5.5 Subsidence Mitigation

Surface subsidence can result from underground mining of any laterally extensive stratified deposit. Most often, it is associated with underground coal mining. Surface subsidence damage caused by underground coal mining is addressed in Sec. 817.121 of *CFR* Title 30. The regulations recognize two distinct types of damage: (1) damage to the land surface and (2) damage to structures or facilities on the surface. Operators are required to correct any material damage to the surface. This implies maintenance of the presubsided land-use capability. To the extent required under state law, operators must also correct material damage done to structures or compensate the owner for the damages. Repair of subsidence damage may include rehabilitation, restoration, or replacement of the structure. In addition to the types of damage, two types of subsidence can be characterized as well. The first type is the result of longwall and high-extraction retreat mining. It occurs fairly rapidly and predictably, producing a regularly shaped subsidence trough. This is referred to as planned subsidence. A second type of subsidence is associated with failing pillars in room and pillar mining. Considerable time may pass before pillars begin to fail, making this type of subsidence much less predictable with respect to location and severity. It is frequently associated with abandoned underground mines. Mitigation measures are influenced by both the type of damage and the subsidence characteristics.

Subsidence damage to the land surface results from changes in topography and overall drainage. This type of damage is usually associated with the extensive troughs produced by planned subsidence. Its greatest impact is upon the use of the land for agricultural purposes. Darmody et al. (1988) reported that farmability can be impaired by alteration of subsurface soil drainage, soil chemistry and structure, and surface drainage caused by subsidence. Corrective measures include improvements to drainage such as installing subsurface drain tiles or constructing surface drainageways. Regrading of the surface may also be necessary to remove depressions. When this is done, topsoil should be removed before grading and replaced afterwards to avoid loss or contamination during the grading process.

Most structural damage caused by subsidence is due to strain. This can result in distortion, cracking, and failure of foundations and structural members. Unplanned subsidence under existing structures requires structural repairs as damages occur. Repairs may include repointing of masonry walls, shoring and bracing of the superstructure, and possible replacement of foundations and floor slabs (Bruhn et al., 1983). New construction in areas prone to subside can employ some protective measures to minimize damage. Peng (1986) describes both a flexible design approach, which allows structural elements to deflect according to the subsidence profile, and a rigid design approach using heavily reinforced structural elements that are intended to cantilever or span across subsided areas. When subsidence due to longwall or high-extraction retreat mining is planned under existing structures, it is possible to lift the superstructure from its foundation using jacks, maintain it in a level position until the surface is stable, and then return the superstructure to a rebuilt foundation.

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Chapter 12.4
ELECTRIC POWER AND UTILIZATION

LLOYD A. MORLEY AND THOMAS NOVAK

12.4.1 INTRODUCTION

Mine electrical power systems have traditionally held little interest for the mining engineer, who has tended to avoid them, or for the electrical engineer, who has given them scant attention. However, today's mine power systems are complex and subject to numerous legal constraints, and it is no longer possible to treat the subject with the indifference shown in the past. Mining equipment is usually mobile and self-propelled; most is powered electrically through portable cables and, for safety, must be part of an elaborate grounding system. The machines and power distribution equipment are seldom stationary and must resist extreme levels of dust, moisture, and vibration. The electrical loads created by mining machinery are cyclic and can be extremely variable. In the ever-moving mining scene, where distribution of power must be constantly relocated and subjected to abuse by machine and worker alike, the potential for safety hazards is always present. Designing and maintaining such an electrical system is demanding and challenging. It requires a specialist with knowledge of both mining and electrical engineering. Also effective management of a mine requires that anyone responsible for production and safety also be familiar with the mine electrical system.

This chapter is intended to provide the engineer with an introduction to electrical power use in mining. Many guidelines are included but, because the subject is so extensive, comprehensive examples cannot be presented. For more in-depth coverage, the references should be consulted, in particular, other works by the authors.

12.4.1.1 History of Mine Electrical Use

Electricity was first introduced into mines shortly before the beginning of the twentieth century in the form of direct current (dc) for rail haulage. *Direct current* was used because, at that time, most systems were powered by dc generators. Direct current has a number of advantages for haulage, the most outstanding being the fact that the dc series-wound motor has excellent traction capabilities. Batteries served as the first power source; hence battery-powered vehicles were extremely mobile even though constrained by rails. However, it was not long before trolley wires were introduced in several mines. Allowing the trolley wire to act as one conductor and the rail as the other provided the simplest form of power distribution yet known to the mining industry. Thus the dc system at a voltage of 250 or 550 V became firmly entrenched in mines.

Underground, the first electrically driven mining machine, the coal cutter, was introduced in the early 1920s. Although dc offered no special advantage, it was readily available, and the machine was equipped with a dc motor and added to the system. The cutter was followed almost immediately by the loader, and it was driven by dc motors as well. In mines with rail haulage, trailing cables supplied power from the trolley wire and the rail to the machines.

The next significant increase in power consumption came with the introduction of the shuttle car, almost 20 years after the first use of the coal cutter. Actually, when the shuttle car was first invented in 1937, it was battery-powered. The addition of an automatic reeling device to handle a trailing cable came later in an attempt to overcome the deficiencies of batteries. These trailing cables were also connected to the haulage power system, and this equipment, when combined with the cutters and loaders, placed additional demands on the dc distribution system.

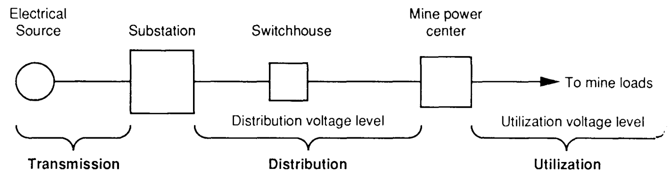

Fig. 12.4.1. Basic power system.

In the late 1940s, when continuous mining machines first began to be used extensively in coal mines, dc was again expected to provide the power. The continuous miners normally needed more energy input than the sum of the various conventional mining machines they replaced, and because this required horsepower caused high current demands, dc was found to be entirely unsatisfactory in most cases. The attendant current demand created enormous voltage drops in the distribution system. As a result, the dc supply system was separated from the haulage system; however, even this proved to be inadequate. During peak operating periods, voltages at the machines were so far below rated values that even moderate efficiency was impossible. Additional problems were caused by the increasingly large cable sizes required to supply the needed power, and cable handling was difficult. The use of *alternating current* (three-phase ac) distribution and motors was an obvious answer. However, some mining companies were reluctant to make the change for another decade. In many instances, this was because the laws in some states limited the maximum voltage in underground mines.

When higher voltages were finally permitted, the desirable economics of ac employment could be realized, and there was a swift transformation from dc to ac for both distribution and high horsepower loads in underground mines. As a result of conversions, mine power systems generally have two voltage levels, *distribution* and *utilization,* which are illustrated in Fig. 12.4.1. Here the substation transforms the utility voltage down to distribution levels, most often at voltages greater than 1000 V. Power at this voltage is distributed through conductors from the substation to the power center; hence it is termed the *distribution system.* The power center or load center, in actuality a portable substation, transforms the voltage to utilization levels, typically at low voltage (in US mining practice, defined as 660 V or less), or medium voltage (661 to 1000 V). It is at this level, or face voltage, that power is normally delivered to the equipment. Despite this reference to voltage levels, it should be noted that distribution and utilization describe functions of a power-system segment, not a specific voltage range.

Originally, primary ac distribution was made at 2300 or 4160 V. In most mines, these levels were later increased to 7200 V, and some operations even increased the voltage to 12,470 or 13,200 V. The principal reason for increasing the voltage was that, for the same load, current would be correspondingly smaller, and lower distribution losses would result. From the beginning, 440-V ac was the most popular for utilization, despite the fact that, when the continuous miner proved so successful, its horsepower was progressively increased. The additional horsepower resulted in an increase in trailing-cable sizes, until their weight was almost more than personnel could handle. To compensate, the most common move was to raise the rated motor voltages to 550 V. More recently, manufacturers have produced machines with 950-V, 2400-V and 4160-V motors to further overcome trailing-cable problems. The two higher voltages have recently gained increased popularity for high-capacity longwall mining systems (Morley, Kohler, and Smolnikar, 1988; Morley, Novak, and Davidson, 1989).

Whereas dc series motors were universally employed in underground rail haulage, the first large motors used in surface mining were dc shunt wound because of their constant speed characteristics. The initial distribution for electric shovels was also dc because of the nature of the power generation. Technological advances soon made ac power systems superior, and ac motors were tried with some success. However, by 1927, ac/dc motor-generator (m-g) sets and the invention of the Ward-Leonard control concept caused these efforts to be abandoned. The new control system enabled motor speed to be varied and controlled by varying the armature voltage while maintaining a constant voltage across the shunt field. The m-g sets functioned as on-board power-conversion units, thereby establishing the use of ac distribution in surface mines. Motor-generator sets driven by synchronous or induction ac motors, Ward-Leonard control, and dc motors established the standard, and even now the combination is still used on many mining excavators.

Distribution and utilization voltages in surface mines increased to keep pace with rapid increases in mine machinery horsepowers. Until the mid- 1950s, 4160 and 2300 V were the usual mine levels. Then, with the advent of larger concentrated loads, it was found that 7200 V was advisable (Rein, 1968). However, this level proved inadequate when machines larger than 100 yd^3 (76 m^3) were introduced, and 13,800-V mine and excavator voltage became a standard. With machines greater than 200 yd^3 (153 m^3) in capacity, 23,000-V utilization was established, but even with these substantial increases in distribution, some loads up to 1000 hp (746 kW) continued to be driven at 480 V. Production shovels up to 18 yd^3 (14 m^3) commonly stayed at 4160 and 7200 V, while in general 4160 V became standardized for machinery with 1500 hp or less, and as a result more than one voltage level could be required at a mine when excavators of different sizes were employed.

12.4.1.2 Power Terminology

Several power terms are used to describe the electrical operation of a power system. These terms are applicable not only in system design, operation, and maintenance, but also in utility company billing practices. If the sum of the electrical ratings is made for all equipment in an electrical operation, the result will provide a total connected load. The measure could be expressed in horsepower, but electrical units of kilowatts (kW), kilovolt-amperes (kVA), or amperes (A) are more suitable. (The connected horsepower can be easily converted to connected kilowatts simply by multiplying by 0.746.) Many loads operate intermittently, especially mining production equipment, with varying load conditions. Accordingly, the demand upon the power source is frequently less than the connected load. This fact is important in the design of any mine power system, as the system should be designed for what the connected load actually uses, rather than the total connected load. Obviously, these considerations have great impact on power-system investment or the capital required to build the system.

Because of the importance of assessing equipment power demands, the Institute of Electrical and Electronics Engineers, Inc. (IEEE) has standard definitions for load combinations and their ratios. Several of the important definitions follow (Anon., 1986):

1. *Demand* is the electrical load for an entire complex or a single piece of equipment averaged over a specified time interval. The time or demand interval is generally 15 min, 30 min, or 1.0 hr, and demand is generally expressed in kilowatts, kilovolt-amperes, or amperes.
2. *Peak load* is the maximum load consumed or produced by one piece or a group of equipment in a stated time period. It can be the maximum instantaneous load, the maximum average load, or (loosely) the maximum connected load over the time period.
3. *Maximum demand* is largest demand that has occurred during a specified time period.
4. *Demand factor* is the ratio of the maximum demand to the total connected load.
5. *Diversity factor* is the ratio of the sum of the individual maximum demands for each system part or subdivision to the complete-system maximum demand.
6. *Load factor* is the ratio of the average load to the peak load, both occurring in the same designated time period. This can be implied also to be equal to the ratio of actual power consumed to total connected load in the same time period.
7. *Coincident demand* is any demand that occurs simultaneously with any other demand. All these definitions may be applied to the units of average power, apparent power, or current. Thus they are invaluable in power-system design, and some examples will illustrate their versatility.

Consider a power cable supplying several mining sections in an underground mine. The sum of the connected loads on the cable, multiplied by the demand factor of these loads, yields the maximum demand which the cable must carry. When applied to current, this would be the maximum amperage. Good demand factors for mine power systems range from 0.7 to 0.8, depending upon the number of operating sections. The lower value is used when there are fewer producing units (e.g., from two to four). The demand factor can be extended to include estimates of average load. For instance, the sum of the average loads on the cable multiplied by the demand factor provides the average load on the cable. A prime applications here is for approximating the current that a conductor is expected to carry. If, for example, 10 identical mining sections would each draw 53 A, the conductors feeding all these sections would be expected to carry

$$\begin{aligned}&(\text{total average load})\ (\text{demand factor})\\&\quad = (\text{average load})\end{aligned} \tag{12.4.1}$$

or

$$(10)(53\ \text{A})(0.8) = 424\ \text{A}.$$

The demand factor and the diversity factor can be applied to many other mine electrical areas, such as estimating transformer capacities, protective-circuitry continuous ratings, and the load that a utility company must supply.

The load factor can be used to estimate the actual loads required by equipment. Here the total connected load multiplied by the load factor is an approximation of the actual power consumed. It should be noted that the average load factor in underground mining tends to be rather low, mainly due to the cyclic nature of machine operation but also to the employment of high-horsepower motors needed to perform specific functions and sometimes operating only for a small fraction of possible running time. For instance, when cutting and loading, a continuous miner may have all motors operating, thus have a total connected load of 385 hp or (0.746) (385) = 287 kW. The average load factor might be 0.6; therefore, the actual power consumed is (0.6) (287) or 172 kW. The load factor can also be applied to equipment combinations.

The maximum power demand normally forms one basis that utility companies use to determine power bills; most often, one month is the specified time period. Demand meters are often installed at the utility-company metering point.

12.4.2 DESIGN CRITERIA

The goal of the engineer is to provide an efficient, reliable electrical system at maximum safety and for the lowest possible cost. The types of information made available to the engineer include the expected size of the mine, the anticipated potential expansion, the types of equipment to be used, the haulage methods to be employed, and whether or not power is available from a utility company. The amount of capital assigned for the electrical system is also designated.

The designed system must meet certain minimum criteria. IEEE (Anon., 1986) has defined the basic criteria for industrial electrical systems that must be applied to mines:

1. Safety to personnel and property.
2. Reliability of operation.
3. Simplicity.
4. Maintainability.
5. Adequate interrupting ability.
6. Current-limiting capacity.
7. Selective-system operation.
8. Voltage regulation.
9. Potential for expansion.
10. First cost.

Of these, safety, reliability, and simplicity are closely related, and most are dependent upon good preventive maintenance. In the cramped and inhospitable environment of an underground mine, these are of vital concern. Most routine maintenance should be capable of being performed by nontechnical personnel, since it will most often be done by the miners themselves. Training for these tasks must be provided.

Adequate interrupting capacity, current-limiting capability, and selective-system operation are projected at safety through reliability. The first two areas ensure protection during a disturbance. Current limiting, when applied to grounding, is perhaps the most significant personnel safety feature of mine electrical systems. Selective-system operation is a design concept that minimizes the effect of system disturbances. *Voltage regulation*—the fluctuation of voltage resulting from change in electrical load—is a limiting factor in system design, particularly underground, and is often the main constraint to mine expansion. It should be anticipated that when the size of the mine is increased, this might involve augmenting the power-system supply through additional power sources.

While first cost is important, it should never be the determining factor, since high-cost equipment, projected at maximizing safety and reliability, can easily offset the increased first cost through the reduction in operating costs. At times, this fact appears to elude some company purchasing agents.

Using the data available, the task of the mine electrical engineer is to select one combination of power equipment over another; provide power or circuit diagrams; estimate equipment, operating, and maintenance costs; set the specifications for the system; and receive and assess the proposals from suppliers. For success, the engineer requires a firm knowledge of mine power systems, but this understanding cannot be based on a standard mine electrical system because such a standard does not exist: no two mines are exactly alike. The engineer must resort to the fundamental concepts, an awareness of what has worked in the past, and a clear understanding of the legal constraints.

12.4.3 POWER DISTRIBUTION EQUIPMENT

A few pieces of power equipment have already been mentioned but only to the extent necessary to describe the concepts of distribution and utilization. The evolution of mine systems has resulted in major items of power apparatus, each serving a specific function. In general, they can be listed as (Anon., 1964; Lordi, 1961)

1. Mine power centers.
2. Switchhouses.
3. Main substations.
4. Portable and unit substations.
5. Distribution (conductors and connectors).
6. Generation.

12.4.3.1 Power Equipment Types

MINE POWER CENTERS. The power or load center is one of the more essential power system units for underground mines and, in a simpler form, for surface mines. Its primary function is to convert the distribution voltage to utilization voltage for operating equipment throughout the mine. It must also incorporate protective circuitry to ensure safe, efficient, and reliable operation.

The electrical components of the power center are usually metal clad, that is, housed in a heavy-duty steel enclosure that may be tire-mounted, skid-mounted, or track-mounted. Towing lugs or pin-and-link couplers are commonly provided on each end of the enclosure to permit towing as mining advances or retreats. Bumpers or check plates are often installed to protect externally mounted components, such as couplers, from damage by mobile equipment. Similar enclosures can be used in surface mines, but here the simplest power center can consist of outdoor components assembled on a flat-bed trailer with a fence or gates to discourage unauthorized entry.

Fig. 12.4.2 shows typical internal components of a mine power center, and a discussion of the various devices follows. It should be noted that a "standard" power center does not exist because of the variety of mining practices and the numerous types of mining equipment used. The power center may supply only one motor or as many as 20 pieces of machinery; it may be totally ac, dc, or a combination of ac and dc. The distribution voltage received by the power center can be 4.16, 7.2, 12.47, 13.2, or 13.8 kV. The outgoing ac utilization voltage may be 480, 600, 995, 1040, 2300, or 4160 V or a combination of one of the higher voltages with one of the lower voltages. Direct current can be at 300 or 600 V, but is almost always 300 V for face applications. As a result of this variety, manufacturers custom build units to meet the individual needs and specifications of the customer.

High-voltage Compartment—The *disconnect* or *load-break switch* is a mechanically operated, air-type switch. Its primary

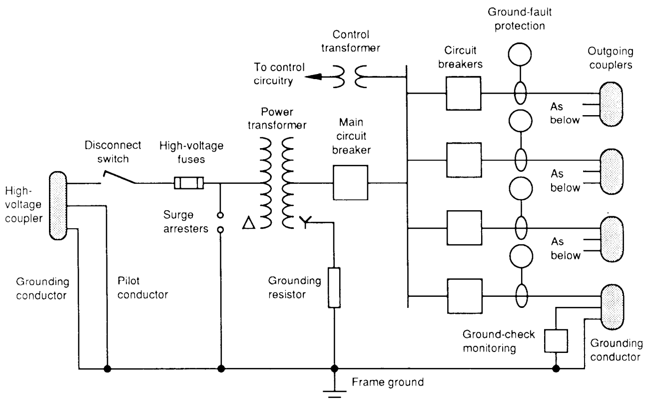

Fig. 12.4.2. Simplified diagram of a mine power center.

function is to allow a quick means of disconnecting the primary of the power-center transformer. A spring-loaded or torsion-bar mechanism provides quick-make and quick-break operation, independent of the speed of the manually activated handle. With observation windows in the power-center enclosure, the switch can serve as a visible disconnect.

Current-limiting fuses are typically used in mine power centers to protect the primary of the transformer. With high fault currents, they operate very quickly, and only a portion of the available destructive short-circuit energy is passed through.

Surge arresters are used to protect the transformer from transient overvoltages, which can be caused by lightning, switching surges, and some types of faults. The functions of a surge arrester are to discharge the energy associated with a transient overvoltage, to limit and interrupt the current that follows the transient current through the arrester, and to return to an insulating state without interrupting the supply of power to the load. The rating of the surge arrester should be coordinated with the insulation level (basic impulse level) of the transformer.

Power-center Transformers—The main transformer can be considered the heart of the power center since its primary function is to convert the distribution voltage to utilization. Proper selection is imperative from the standpoints of safety, efficiency, and reliability. IEEE (Anon., 1979) has established classifications and specifications for determining transformer characteristics as an aid in design and application, and these standards are in general use in the mining industry. A transformer may be classified by the insulation system as liquid or dry. Liquid insulation includes mineral oil or synthetic fluids, and dry transformers are ventilated or sealed gas-filled types. Ventilated dry units are used almost exclusively in mine power centers. Dry transformers used in mine power centers should have Class-220°C insulation (a rating always given in degrees Celsius). The classification places an absolute allowable maximum temperature on the transformer when it is operating continuously under full capacity at rated voltage and current.

There is no set formula for determining the capacity rating for a power-center transformer. For constant loads, it is a common rule-of-thumb to allow 1 kVA/horsepower of connected load. However, the mining process does not produce a constant load (i.e., all connected motors are not operating at the same time on a continuous basis); hence using this rule-of-thumb will normally result in oversizing the transformer. Past experience and demand factors established by manufacturers and operators, along with the horsepower of the connected load, are essential for determining the transformer capacity. For typical underground mining sections, the kVA rating may lie within the range of 50 to 80% of the connected horsepower. Over the years, power-center manufacturers have arrived at certain standard or typical transformer capacities, created mainly by repeat demand of the industry. Many manufacturers supply capacities at any increment of 50 kVA.

As an example of transformer sizing, consider the following continuous-mining section:

1. 370-hp continuous miner (gathering head, pump, tram, and cutter motors)
2. two 125-hp shuttle cars (traction, conveyors, and pump motors)
3. 80-hp roof bolter (traction and pump motors)
4. 50 hp of auxiliary equipment (section fan, sump pump, hand tools, etc.).

The total connected horsepower is the sum of the individual loads or 750 hp. If the demand factor has been determined to be 75%, the effective load (maximum demand) would be (0.75)(750) = 562 hp. Thus a 600-kVA capacity would be indicated. However, flexibility must always be considered in mining applications and, as a result, it may be necessary to select the next higher kVA rating, or 650 kVA, to accommodate anticipated additional loads.

All mine power-center transformers are three-phase, being either three single-phase units where each transformer is rated at one-third of the total required capacity, or integral three-phase types with construction allowing field replacement of failed windings. The integral three-phase transformers are preferred because they have equal reliability, lower cost, higher efficiency, take less space, and have fewer exposed interconnections than three single-phase units.

Delta primary and wye secondary are the preferred connections for standard two-winding power transformers and are the connections commonly used in mine power centers. The wye secondary provides an easy means for resistance grounding, and the delta-connected primary provides isolation of the distribution circuit from the utilization circuit with respect to ground currents. The delta-wye connection also stabilizes the secondary neutral point and minimizes the production of harmonic voltages. In certain situations, a delta-connected secondary may be specified or required, and the primary may be delta or wye in this case. If neutral grounding is desired or required, a grounding transformer is needed to derive a neutral.

Mine power centers are usually cooled by natural convection, and the side panels of the transformer compartment normally have louvers to allow air circulation. The transformer windings are designed so the heat generated is exposed to an adequate amount of cooling to handle the expected loads. The effective cooling areas are the inside of the winding, the outside of the winding, and the cooling ducts within the winding. The movement of air by convection carries away the heat generated by the winding.

Low- or Medium-Voltage Compartment—Molded-case circuit breakers are normally used to protect ac utilization equipment and their associated cables for applications up to 1000 V. High voltage (greater than 1000 V) requires vacuum or oil circuit breakers, which are discussed in the segment on switchhouses. Molded-case breakers are also used for 300-V dc applications.

Molded-case circuit breakers can provide both short-circuit and overload protection. Each circuit breaker usually has an undervoltage release, which is an auxiliary solenoid that trips the operating mechanism of the breaker whenever its coil voltage drops below 40 to 60% of its rated value. Other protective relaying external to the breaker, such as ground-fault relaying or ground-check monitoring, can use the undervoltage release to initiate circuit-breaker tripping. The selection of molded-case

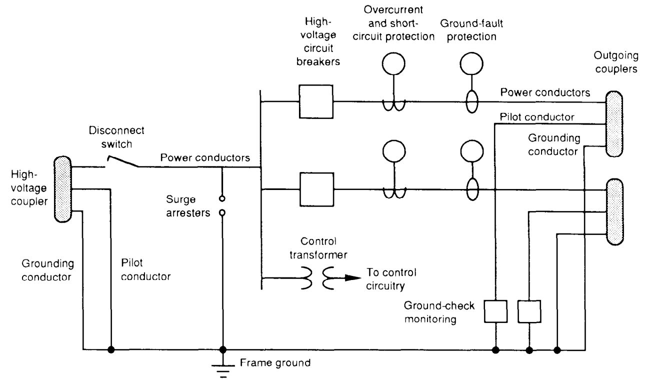

Fig. 12.4.3. Simplified diagram of a switchhouse.

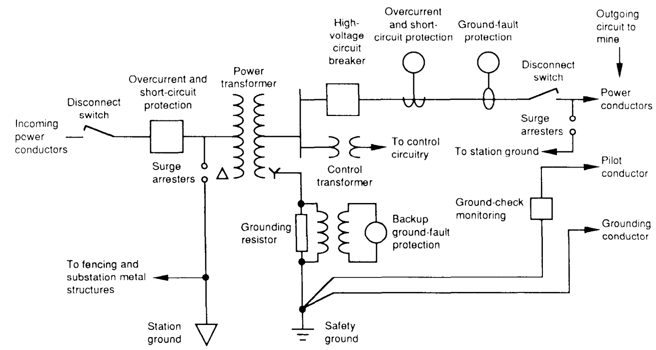

Fig. 12.4.4. Simplified diagram of a substation.

circuit breakers is based on voltage, frequency, interrupting capacity, continuous-current rating, and trip settings.

Fig. 12.4.2 shows *ground-fault protection* on each outgoing machine circuit. *Zero-sequence relaying* is the most common method of ground-fault relaying. With this method, the three-line conductors are passed through a window-type current transformer, and the secondary of the current transformer is connected to a ground-trip relay. The normally open contacts of the ground-trip relay are connected in series with the undervoltage release of the associated molded-case circuit breaker. If a ground fault occurs, the current transformer senses an imbalance in the three-line conductors, and current is induced in its secondary winding which activates the ground-trip relay. The contacts of the ground-trip relay then open and de-energize the undervoltage release which results in tripping the circuit breaker.

Fig. 12.4.2 also shows *ground-check monitoring* on each outgoing circuit. Its purpose is to continuously monitor the continuity of the grounding conductor. Monitors in common use in mining are impedance types and continuity types. *Impedance-type monitors* require the trailing cable to have a pilot conductor. The monitor is calibrated to the impedance of the loop formed by the pilot and grounding conductors. The device then monitors the change in impedance from the initial calibration. If the impedance of the loop increases beyond a preset value, the monitor trips its associated circuit breaker by opening a set of contacts in series with the undervoltage release. *Continuity monitors,* also termed wireless or pilotless monitors, do not require a pilot conductor. The monitor generates an audio frequency that is coupled to the grounding conductor by means of a transmitting coil. The pilot wire is eliminated by using the line conductors as a return path. Filters within the monitoring location and within the monitored machine are necessary for coupling and uncoupling the audio signal from the line conductors. If the grounding conductor is open, the receiver coil will not pick up a signal, and the monitor will cause a set of contacts to open in the undervoltage circuit of its associated circuit breaker.

SWITCHHOUSES. Switchhouses are portable equipment that protect and provide a means for branching of the distribution circuit. Switchhouse components (Fig. 12.4.3) are contained in metal-clad enclosures, similar in construction to those of mine power centers. The internal components consist of visible disconnects and sectionalizing units. As with power centers, the principal function of the disconnect switch is to remove power manually from downstream mine power equipment that is connected to the distribution system. The sectionalizing aspect is to provide protective relaying in the distribution system and to allow branching. The principal component is the high-voltage circuit breaker. Both *vacuum* and *oil circuit breakers* have been used in switchhouses, but vacuum breakers are more popular.

Induction-disk and, more recently, *solid-state relays* are used for protective-relaying applications in switchhouses. These relays have a range of adjustment for both the pickup current and time delay so they can be applied to a variety of situations. The protective relays in a switchhouse must be coordinated with other upstream and downstream protective equipment. Unlike molded-case circuit breakers, the overcurrent and short-circuit sensing devices are not an integral part of the high-voltage breaker. Instead, current transformers are used in conjunction with the induction-disk or solid-state relays to provide overcurrent and short-circuit protection.

The equipment name is modified depending upon the number of circuit breakers and associated protected outgoing circuits: for example, a double switchhouse contains two breakers and circuits. Because of size limitations, the units in underground mine power systems are rarely larger than double switchhouses, but surface mines often incorporate four-breaker switchhouses, or switching skids as they are commonly called. The schematic diagram for surface or underground applications is practically the same: the only basic difference is the repetition of internal components to correspond to the number of circuit breakers used.

SUBSTATIONS. It is common mining practice to purchase all or most power from utility companies, if it is available. As utility voltages usually range from 24 to 230 kV, a main (primary) substation is required to transform incoming levels down to a primary distribution voltage for the mine. Typical components are shown in Fig. 12.4.4. Main substations may range from 500-kVA capacity supplying 480 V for only pumps and conveyors, to 50,000 kVA servicing a large surface mining operation and processing plant (Rein, 1968). In addition to the transformer, substations contain a complex of switches, protection apparatus, and grounding devices, all serving a safety function. The nature of the mining operation and its power requirements dictate how many main substations are required and where they should be placed. These may be owned by the utility or the mining company; the decision on ownership is commonly dependent upon economics. However, if the total connected load is greater than 1000 hp (746 kW), mine ownership is often more favorable.

Main substations are usually permanent installations with components mounted on concrete pads. Power-conductor terminal structures, commonly made of galvanized steel but also of wood, support the conductors and cables that provide connections for transformers and circuit breakers. Insulators, usually made of glazed porcelain or glass, insulate the overhead conductors from the supports.

Portable or unit substations are popular for small load requirements and, in larger installations, they may serve to transform the primary distribution voltage to a lower distribution level. The term unit means that the substation and power equipment is designed and built as a package. In a surface mining deployment, a large dragline may require 24 kV while the production shovels and other mining equipment need 4160 V.

Substation Transformers—Transformers for permanent surface substations are almost always liquid (oil) immersed and built to IEEE standards (Anon., 1987). Capacities commonly range from 5000 to 30,000 kVA (sometimes less but rarely more). In addition to the mine, the loads may include a preparation plant and general surface loads, such as pumps, ventilation fans, maintenance shops, and office and bathhouse facilities. When an underground mine is involved, it is recommended that a separate transformer be used for the underground power system.

The selection of transformer capacity must be based on an estimation of the electrical load. Up to 2000 or 3000 kVA, a general rule of 1 kVA/hp of connected load is perhaps satisfactory. This provides more than adequate capacity for the load and will allow minimal load growth. Note that a worst-case demand factor of 0.75 to 0.85 is built into the selection. Such oversizing will also allow for an adverse power factor. When the requirements are greater than this, a more precise load estimate is required if optimal equipment costs are to be achieved. Considering the common capacity needs for the majority of mining operations, a load-flow analysis must be performed. In some cases, capacity might be determined on a per-connected kVA basis (about 0.75 per connected hp) to allow for the growth in demand factor that always exists.

Past practice in the industry was to use three single-phase transformers for easy replacement of failed units, and a spare was often included to facilitate the changeover. However, improved manufacturing techniques have now resulted in transformers being among the most reliable components; thus the common practice today is to install three-phase integral units (Wade and Kunsman, 1978). The same advantages of three-phase transformers given for power centers hold for substations.

Protective Equipment—To provide a visual disconnect for maintenance, *knife-blade load-break switches* or *fused cutouts* are located on the primary and secondary of the substation transformer. The switches are usually housed in metal-clad enclosures, while the cutouts are pole-mounted. High-voltage fuses or circuit breakers may be used for transformer protection. A general rule is to use a primary circuit breaker with protective relaying when the transformer capacity is 5000 kVA or greater. The short-circuit, overcurrent, and ground-fault protection is essentially the same as employed in switchhouses. Fig. 12.4.4 shows potential relaying across the grounding resistor; this is used as backup ground-fault protection. If a ground fault occurs, a voltage will develop across the grounding resistor. A potential transformer is used to step down this voltage to within the rating of the ground-trip relay. When the ground-trip relay is energized, it trips the circuit breaker.

Two physically separated ground beds are required at the substation. Surge-arrester grounding conductors, static conductors, metallic frames, and fencing are connected to the station bed, while only the distribution grounding conductors are tied to the safety ground bed. The purpose for the separation is to prevent voltages produced across the station bed from being transferred to the safety bed. The segment on mine grounding systems will further discuss this subject.

DISTRIBUTION. This category of major power equipment is often referred to as the weakest link in mine power systems. It encompasses all the overhead power lines, cables, cable couplers, and trolley lines used to carry power and grounding between the power equipment and eventually to the loads.

Cables—Cables carry the electricity from the substation, where the power is taken from utility company lines, to the point of utilization by a mining machine, pump, conveyor belt, or other pieces of equipment. There are many possible variations in mine distribution, and several types of cables can be put to a similar use.

The type of cable is dependent upon the application; some cables remain in stationary locations for several years, while others are moved frequently. The cables that are connected to mining machines are termed portable (Anon., 1978; Anon., 1980). Federal regulations use the term *trailing cables* for the specific variety of portable cables used in a mine (Anon., 1981). Trailing cables are flame-resistant flexible cables or conduits through which electrical energy is transmitted to a machine or accessory.

In underground mines, trailing cables are generally attached to the low-voltage side of the power center or distribution box. The portable cables that feed the power center, or are attached to the high-voltage side, have to be moved when the power center is moved, but they are not moved as often as the trailing cables. The most stationary cables are those that bring power into the mine, for instance, down the borehole and from the borehole to the portable switchhouses. These are the *feeder cables*, and a special type, designated *mine power feeder*, can be used for installations that may not be moved for several years. However, the use of the word “feeder” here is to denote a cable type rather than the application in distribution. Both feeder and portable cables can be used for feeder applications; here the cable supplies two or more major loads.

Similarly, in surface mines, the cables that feed from the switchhouses or unit substations to mobile equipment are trailing cables. Those moved only occasionally that are not connected directly to a machine are portable cables. Stationary cables can be feeder or portable types.

Moving the cable is a recurring task both underground and aboveground. Some trailing cables are placed on reels or spools to facilitate moving. Prime instances of reeled cables are those associated with the reeling devices on-board shuttle cars and mobile cable reels used with surface excavators.

An identifying code related to standard specifications designated by the Insulated Cable Engineers Association (ICEA) is embossed on the cable throughout its entire length. The code includes any federal approval number for flame resistance and approval by the Commonwealth of Pennsylvania (indicated by the letter P preceding the federal approval number). The code also includes identification of the number of power conductors in the cable (e.g., 3/c for three conductors), the approved voltage designation, and the cable type. Fig. 12.4.5 shows the cross sections and components of several popular mine cable types. The three basic cable components are the *conductors, insulation,* and *jacket,* although there may also be fillers, binding, shielding, and armor. Conductors are surrounded by insulation, and the jacket covers the insulation. Cable selection is based on a number of parameters, including, current-carrying ability, voltage rating, and configuration (see Chapter 8 of Morley, 1990, for full details). Components and selection parameters are discussed in the following paragraphs.

In order to obtain the required flexibility, cable conductors are composed of many wires combined into strands, and a number of strands form the conductor. The conductors are either copper or aluminum; the latter is cheaper and lighter but has lower conductivity. Aluminum is sometimes used in feeder cables, but poor flexibility eliminates its use in trailing cables. The cross-sectional area of conductors is important for mechanical

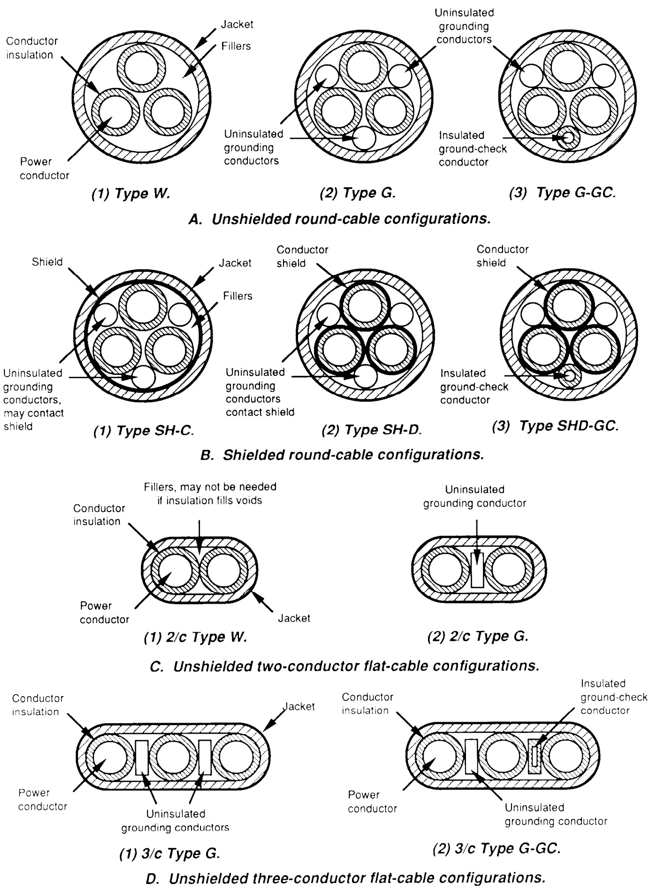

Fig. 12.4.5. Cross-sectional sketches of popular mining cables.

strength and is closely related to the current-carrying capacity. In the United States, both the American Wire Gauge (AWG) and circular-mi1 designations are used.

Insulation is required to withstand stress from heat, voltage, and physical abuse. It must be specially designed not only to protect mine personnel from electric shock but also to separate power and grounding circuits effectively. Excessive heat is particularly destructive to insulating compounds, and the main sources of heat are the ambient temperature and power loss in cable conductor resistance. The maximum, normal, continuous current the conductors can carry safely is directly connected to cable heating, and the term ampacity is often used to describe this current level. The ampacity rating is usually based on the maximum conductor temperature rise, with the temperature limit chosen on the basis of the specified life expectancy of the cable insulation. The *temperature class* (always given in degrees Celsius) describes the maximum allowable sustained conductor temperature in an ambient temperature of 40°C (104°F).

A cable used in a confined space can become overheated with continuous current at the ampacity rating. Perhaps the best example is a cable wound on a reel, either for storage purposes or to increase mining machine mobility. The ampacity of the cable must be derated in these cases; ampacity and derating factor tables are available in federal regulations (Anon., 1981) and other standards (e.g., Anon., 1978; Anon., 1980; and Anon., 1990).

The most common insulating compounds are neoprene, styrene butadiene (SBR), ethylene propylene (EPR), and cross-linked polyethylene (XLP). SBR is used in 600-V trailing-cable insulation. It has good elasticity and flexibility, a 75°C temperature rating, and resists damage by crushing from runovers and rock falls. EPR has replace SBR in many trailing cables because it allows the cable rated voltage to be increased to 2000 V and the temperature rating to 90°C, while maintaining the same insulation thickness as SBR and neoprene. XLP is also rated at 90°C and is used in high-voltage mine-feeder and portable strip mining cables. XLP is rather stiff and is not recommended for reeling applications.

The thickness of a specific insulating material yields the *voltage rating* of the cable, and the most common ratings are 600 V, 2 kV, 5 kV, 8 kV, 15 kV, and 25 kV. The 5-kV, 8-kV, 15-kV, and 25-kV ratings are primarily used for stationary feeder cables and are generally not connected to mining machines, except in surface mines. Usually 4.16-kV distribution requires 5-kV rated cables, 7.2 kV requires 8 kV, and 12.47 kV and 13.2 kV require 15-kV ratings. Utilization voltages of 250 V dc, 440 V ac, and 550 V ac usually call for 600-V or 2-kV cables, and medium-voltage applications need 2-kV insulation.

The main purpose of the cable jacket is to provide protection for the inner components and hold the assembly in the designed configuration. Common jacket materials are neoprene, nitrile-butadiene/polyvinylchloride (NBR/PVC), and chlorosulforated polyethylene (CSP, also called Hypalon). Armored cables are used in some borehole applications; the jacket is a heavy metallic covering that affords extra protection to the conductors and insulation.

Flat cable configurations are commonly used on mining machinery that incorporate cable-reeling devices (e.g., shuttle cars), whereas round cables are typical on all other mining equipment. The flat shape allows an increased length on cable reels and is less susceptible to run-over damage than round cables. Direct-current machines employ either Type W or Type G, where the Type G has a separate grounding conductor. Low-voltage ac mining machines commonly use the unshielded Type G or Type G-GC configuration. Grounding conductors are placed in the interstices between the power conductors in Type G, and a pilot (or ground-check) conductor replaces one of the grounding conductors in the Type G-GC.

There are two basic configurations for shielded cables: SH-D and SH-C. The shields of the SH-D cable surround each insulated conductor; in the SH-C cable, one shield encloses all power conductors and grounding conductors. The SH-D shielding is preferred because the grounding conductors are in intimate contact with the shields. The round shielded-cable configuration, SHD-GC, is also used extensively for medium-voltage and high-voltage trailing cables. Stationary power cables are often mine power feeders of the MP-GC type. Though similar in cross section to SHD-GC, MP-GC cables are less flexible, have higher tensile strength, and use different shield materials. Mine-power feeder cables are also designed for use in boreholes, aerial installations, ducts, and direct burial.

Current and voltage regulation are the major concerns in sizing cable power conductors for an intended application. The effective continuous current through the cable power conductors must be less than the the cable ampacity, with correct derating factors applied. The voltage drop across the distribution and utilization systems must be such that the voltages at the loads are within allowable tolerances. Current is often the determining factor for machine trailing cables, since these cables are almost always short in comparison with the balance of the system. Feeder cables serving many loads, however, are often so long that voltage drop becomes a principal concern. Even though the cable size may be found adequate in terms of ampacity and voltage drop, other factors may enter into the conductor sizing, such as tensile load, weight, and available short-circuit current. An important trailing-cable limitation is weight, as the cable cannot be too heavy for miners to physically handle. This maximum conductor size is usually considered to be 4/0 AWG for three-conductor cables.

There are several methods which can be used to find cable current, including full-load current, effective current demand, and applying a load factor. Regardless of the method employed, it should be realized that typical current requirements of mining machinery change continuously over time and may be described as unsteady in nature. The extremely wide variability of mining conditions makes it difficult to define current levels for any part of a given duty cycle with precision. Thus, unless current measurements are taken, current values can only be estimates, and a worst-case situation, such as assuming full-load current, is recommended to insure actual currents will be less than cable ampacities.

The primary concern for voltage conditions is that satisfactory voltage must be at the machine terminals for proper starting and operation. The allowable voltage tolerance on machine motors is usually ± 10% of rated voltage. Maintaining adequate voltage is one of the more difficult problems in mining, and is often the main constraint on mine expansion from a point of power delivery to the operation. Good practice calls for limiting the maximum voltage drop under normal load conditions to not more than 10% of the nominal system voltage across each voltage level, that is the distribution system, the utilization system, etc.

For a thorough voltage study of a mine, all impedances and all loads in the power system must be known. A circuit diagram must then be prepared and calculations performed to see if there are satisfactory voltage levels at the machines. Analyses are performed not only for normal load conditions but also for the starting of critical motors. If calculated voltages are below those tolerated, system impedance must be reduced, and the most convenient way is to increase cable conductor sizes. The process is continued by trial and error until desired results are obtained. Even with a small system, the computations can become so involved that load-flow computer programs are the only practical solution medium (Trutt and Morley, 1988).

Cable Couplers—Couplers are complex plugs and sockets used throughout the mine distribution system to connect mobile machinery to trailing cables, to connect cables with one another, and to connect cables to power centers, switchhouses, and substations. Their complexity is a direct result of the mine environment in which they are used; they must resist damage, be sturdy enough to withstand repeated use, prevent electrical hazards, be watertight and dustproof, and withstand heat and cold. Some models are rated explosion-proof. The coupling mechanism must be easy to accomplish yet secure. Couplers in the 15-kV, 500-A range are used as connections to switchhouses and mine power centers, to join high-voltage cables, and for high-voltage machines. These high-voltage couplers accommodate three power conductors, one or more grounding conductors, and one or more ground-check conductors. The standard sizes for low-voltage and medium-voltage couplers are 225 A, 400 A, 600 A, 800 A, and 1200 A. Their primary use is to connect mobile equipment to power centers and junction boxes, and to connect cables in the 600 V to 1000 V range. Their construction is sturdy but less complex than high-voltage couplers.

TROLLEY SYSTEMS. The conductors that provide power for electric track-haulage systems form a major part of the power distribution system in many underground mines. Several conductors are used in the trolley circuit: trolley wire, feeder cable, rail-bond cable, and steel track rails. The trolley wire supplies power directly to a rail-mounted vehicle, such as a mine locomotive, through a collector called a shoe or harp. The trolley-wire and collector connection can cause frequent severe arcing that may damage either part and cause an obvious ignition hazard. Proper positioning of the trolley wire, particularly at curves and switches, correct holding force on the collector, and the required amount of lubrication are necessary to minimize arcing.

A feeder cable supplies power to the trolley wire; both must be sized properly to provide enough current-carrying capacity yet minimize heating and voltage drop. In addition, rectifiers must be positioned at adequate intervals to supply the proper voltage to the feeder. The current return path utilizes the steel rails, which much have adequate conductivity to minimize the total system resistance. Rails are laid in segments, and the connections between them can loosen or the rails could break; hence, rail-bond cable is installed to maintain continuity. Rail-bond cable is attached at each rail joint and, as a further precaution, between the two rails at specified intervals (cross bonds).

SURFACE OVERHEAD LINES. The most common method used for surface electric power transmission and distribution is overhead conductors. Although their size and detailed construction can vary widely, overhead power lines normally consist of bare metallic conductors supported by insulators from some elevated structure. The conductors use air space for insulation over most of their length, while their elevation protects them from contact with personnel and equipment.

Overhead line installations use numerous types of conductor arrangements and support structures in various combinations. Utility systems range from single wooden poles carrying conductors at low voltages to self-supporting steel towers bearing major transmission lines. Wooden-pole lines with or without crossarms, for example, may be part of a single-phase or three-phase distribution system with voltages of 2.3 to 35 kV. By contrast, steel towers often carry lines transmitting large amounts of power at 115 kV and up, connecting major load centers of a utility company grid (Anon., 1964b). Utility-owned lines are commonly classified by function, which is related to voltage. There is no utility-wide standard for voltage classification, but the system typically used differs from the classification employed in the mining industry.

Overhead conductors are arranged in various configurations to reduce line-to-line contacts due to wind, ice loading, or sudden loss of ice load, and may include different combinations of power, neutral, and static conductors. Aluminum conductors with steel reinforcement (ACSR) are commonly used because of their strength and relatively low price, but special applications may call for such other materials as copper.

The types of overhead line installations used for mining applications are similar to those in utility distribution systems. Typical are pole lines to supply equipment in surface mining and lines feeding surface facilities related to mining. These lines are normally installed on single wooden poles and may carry only two conductors, as in single-phase supplies, or have up to six conductors, including three power, one grounding conductor, one ground check (pilot), and one static. The pole lines may be relatively permanent installations such as those feeding plants, shops, and other surface facilities, and long-term pit baselines or ring mains. Sometimes, temporary poles are mounted in portable bases (such as concrete-filled tires) for ease of relocation, and these are commonly used in surface mining operations to carry power into the pit. Conductors are again usually ACSR, but hard-drawn copper is used where blast damage is a problem (Anon., 1978).

12.4.4 POWER SUPPLY

12.4.4.1 Utilities

As utility companies are the principal power supplier for mines, an understanding of utility system power transmission and distribution is important. Often this system greatly affects the power available to the mine, including voltage regulation, system capacity during power failures in the mine, and overvoltage occurrences. In a nearby substation, power from a generating station is transformed up to a transmission voltage, commonly 69,000 V or more (Anon., 1986). This power is carried on transmission lines to major load areas, either supplying large industrial users directly or powering the utility's own distribution substations. Distribution substations step the voltage down, this time to a primary distribution level ranging from 4160 to 34,500 V, but most often at 12,470 or 13,200 V. The utility service, therefore, can be any of the following standard values: 138, 115, 69, 46, 34.5, 23, 13.8, 12.47, 6.9, 4.16, and 1.4 kV. Generally, the delivered voltage ranges from 23 to 138 kV, but others such as 480, 2300, and 7200 V are also found. What is available to the mine depends on whether the possible connection is to the power company transmission system, a primary distribution system, or a distribution transformer.

It is the responsibility of the mining company to select that voltage best suited to its needs. Primarily, the choice depends on the amount of power purchased. It is not safe to assume that the power company has the capability to serve a large mine complex from existing primary distribution lines or even from the transmission system. The problem stems from the fluctuating nature of mine loads. For example, large excavators in surface mines can require high peak power for a short time followed by regenerative peak power, cycling within the span of 45 sec. The fluctuating load may create voltage and frequency variations beyond the limit set for other utility customers. Accordingly, most large draglines and shovels require power from 69-kV to 138-kV transmission systems to get adequate operational capacity, and the construction of several miles of transmission equipment can result in a sizable cost for the mine budget.

Regardless of where the mine substation is tied into the utility complex and who owns that equipment, its outgoing circuits will here be termed the *mine primary distribution system,* or simply distribution. The incoming power will most often be referred to as the *transmission system.*

12.4.4.2 Generation

The purchase of electric power from a utility company is generally less expensive and more reliable than operating a small generating plant. However, a mine may be located in a remote area, a long distance from a utility transmission or distribution system. Thus self-generated electric power may be the only feasible alternative. Generating plants for mines are typically powered by diesel engines or by coal-, oil-, or gas-fired boilers. In some instances, even water-powered plants have been used. It should also be noted that some mines that purchase electric power also use diesel-powered generators for emergency electric power. If the primary source of power is lost, a diesel-powered generator can be quickly started and used to supply emergency power to critical equipment, such as ventilation fans and personnel or elevator hoists.

12.4.5 POWER DISTRIBUTION ARRANGEMENTS

12.4.5.1 Basic Arrangements

The basic distribution arrangements available for industrial applications are radial, secondary-selective, primary-selective, primary-loop, and secondary-spot networks (Anon., 1986). Radial systems are the most popular arrangements in mining, though other configurations can be found where special circumstances call for greater system reliability (Anon., 1976). Surface mines have, of course, greater flexibility than underground mines and employ a wider range of distribution arrangements. Secondary-spot networks, which are the most popular system for large facilities in other industries, are uncommon but could be applied to preparation and milling plants. The following descriptions of the main distribution patterns are based on IEEE definitions.

RADIAL SYSTEM. *Radial distribution* in its simplest form consists of a single power source and substation supplying all equipment. Radial systems are the least expensive to install as there is no duplication of equipment, and they can be expanded easily by extending the primary feeders. A prime disadvantage is tied to their simplicity; should a primary component fail or need service, the entire system is down.

An *expanded radial system,* or load-center radial, is commonly used in the mining industry. As in Fig. 12.4.1, two or more voltages levels are established, but the feeders form a tree-like structure spreading out from the source. This system has the advantages of the simple system and others too. If the load centers or distribution transformers are placed as close as practical to the actual loads, most distribution will be at the higher voltage. This allows decreased conductor investment, lower electrical losses, and better voltage regulation.

SECONDARY-SELECTIVE SYSTEM. In a *secondary-selective system,* a pair of substation secondaries are connected through a normally open tie circuit breaker. The arrangement allows greater reliability and flexibility than a radial system. Generally, the distribution is radial from either substation. If a primary feeder or substation fails, the bad circuit can be removed from service, and the tie breaker is closed either manually or automatically. Maintenance and repair of either primary circuit is possible without creating a power outage by shedding nonessential loads for the period of reduced capacity operation. Economics often justify this double-ended arrangement if substation requirements are above 5000 kVA.

OTHER DISTRIBUTION SYSTEMS. In the *primary-selective system,* each substation can receive power by switching from either of two separate primary feeders. During normal service, each feeder should handle one-half the load. The system is simple and reliable, but costs are somewhat higher than radial or secondary-selective because of the duplication of primary equipment.

Though found in some mines, the *primary-loop system* is not considered good practice. The cost can be slightly less than primary selective, but this configuration can result in dangerous conditions when a primary-feeder failure occurs. A failed portion can be energized at either side creating a hazard to maintenance personnel.

In the *secondary-spot network,* two or more distribution transformers are supplied by separate primary distribution feeders. The secondaries are tied together through special circuit breakers, called network protectors, to a secondary bus. Radial secondary feeders are tapped to the bus and feed the loads. This system creates the most reliable distribution system available for industrial plants. However, the arrangement is expensive, and the reliability gain is not warranted for the majority of mining applications.

12.4.5.2 Surface Mine Distribution

Mine power systems can be divided into three categories depending upon the purpose of a specific portion: (1) transmission or subtransmission, (2) distribution, and (3) utilization. Power system installations can vary greatly at distribution and

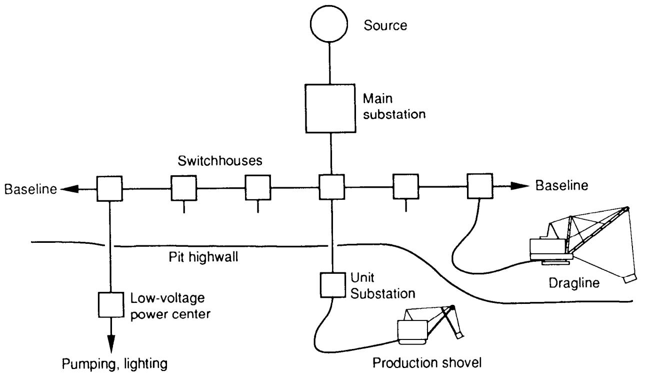

Fig. 12.4.6. Simple all-cable radial distribution system applied to strip mining.

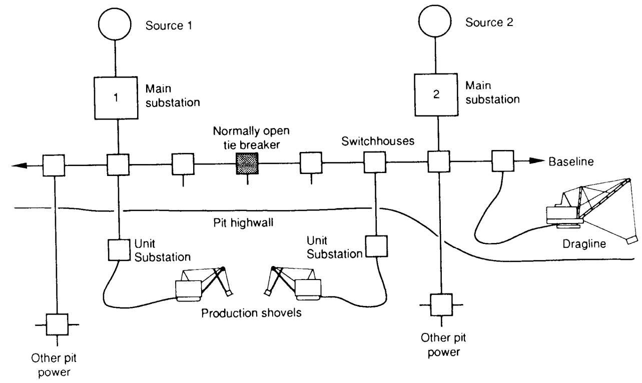

Fig. 12.4.7. Secondary-selective all-cable distribution system applied to strip mining.

utilization levels, but in some mines, distribution and utilization can be the same system.

The location of the mine substation is usually an economic compromise between the cost of running transmission lines and power losses in primary distribution. From the main substation, power is distributed to the various centers of load in the operation. However, individual loads or complexes, such as preparation plants and other surface facilities, may have large power requirements or be so isolated that primary distribution operation is not practical. In these cases, or for safety reasons, incoming utility transmission should be extended close to the load. The extension is designated a subtransmission system, and the conductors are usually suspended as overhead lines. Subtransmission commonly requires a switchyard of high-voltage switching apparatus for power tapping.

Subtransmission circuits, primary switchyards, and main substations are almost always located in areas unaffected by the mining operation. The main substation is where the grounding system for the mine is established. This ground is carried along the power lines through overhead conductors or in cables and is connected to the frames of all mobile mining equipment.

Mine power distribution, in its simplest radial form, has already been shown in Fig. 12.4.1 to consist of a substation, distribution, and a power center feeding the mining equipment. The arrangement is very common in small surface operations where the distribution voltage is commonly 4160 V but can be 2300 V in older equipment. In the smallest mines, power is purchased at low-voltage utilization (often 480 V) and fed to a distribution box to which motors and equipment are connected. At times, simple radial systems are employed in large surface mines where only one machine must be served or an extensive primary distribution network cannot be established, as in some contour operations.

The great majority of strip mines employ radial distribution, but secondary-selective can also be found. Simplified examples of these systems are provided in Figs. 12.4.6 through 12.4.9. In all configurations, a portion of the primary distribution is established as a baseline or bus. The baseline is usually located on the highwall, paralleling the pit for the entire length of the cut. Its location is typically maintained 1500 ft (457 m) ahead of the pit, and it is moved as the pit advances (Anon., 1976). Distribution continues from the baseline to the mining equipment, with the connections maintained at regular intervals. As the machines move along the pit, the baseline connections are changed to other convenient locations.

The baseline can consist of overhead polelines as shown in Fig. 12.4.9. The overhead poleline plus cable arrangement is

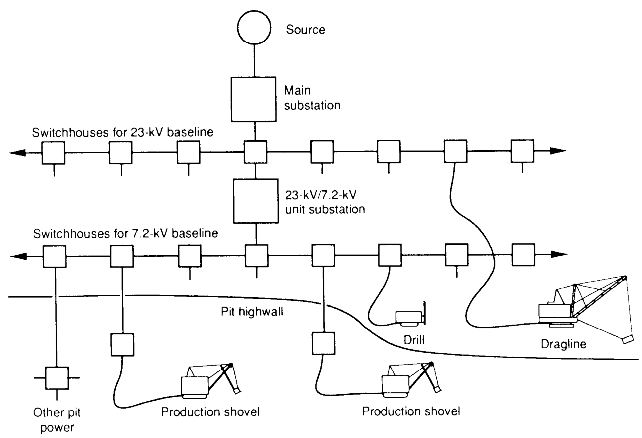

Fig. 12.4.8. Dual-baseline all-cable radial distribution system applied to strip mining.

common in older mining operations, especially when utilization is at 7200 V or less (Anon., 1976). Typical spacing between poles, or line span, is 200 ft (60 m), and drop points are noted in the figure by triangles. These are terminations between the overhead conductors and the cables, mounted about 8 ft (2.4 m) above the ground on poles. The drop points are spaced at regular intervals of around 1000 to 1500 ft (305 to 457 m). Cables connected to the drop points deliver power to skid-mounted switchhouses located on the highwall or in the pit. The skids can be either boat design with flat bottoms or have fabricated runners, depending upon the allowable bearing pressure of the mine terrain. Cable couplers are commonly used for both feeder and trailing-cable connections. Disconnect and circuit-protection functions are required for each distribution load, and double switchhouses (two-breaker skids) are frequently employed for two loads. Unit substations often contain internal circuit protection on the incoming side and thus do not require a breaker skid.

Trailing cables are usually 1000 ft (305 m) in length, although lengths to 2000 ft (610 m) can be found. When longer cables are necessary to reach a breaker skid, in-line coupling systems can be used, and these are commonly mounted on small skids for easy movement. Trailing-cable handling for stripping equipment is often assisted by cable reels mounted on skids or self-propelled carriers. Large excavators can require the self-propelled variety.

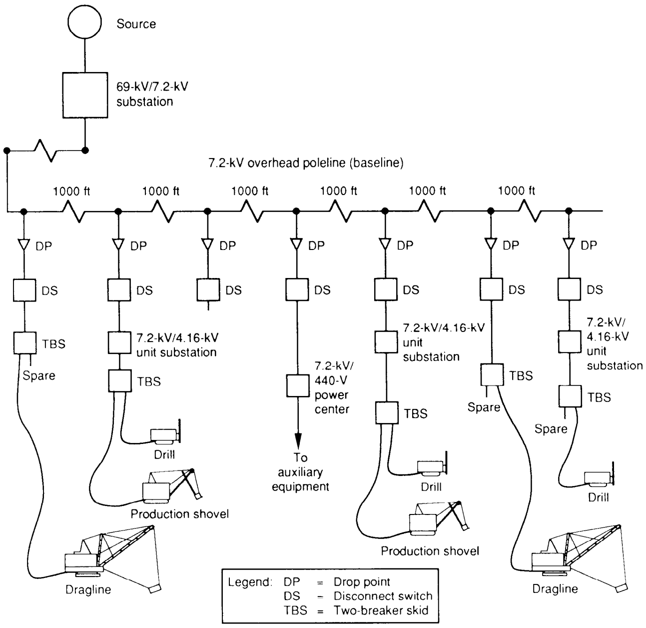

Fig. 12.4.9. Radial distribution system in strip mining using overhead polelines.

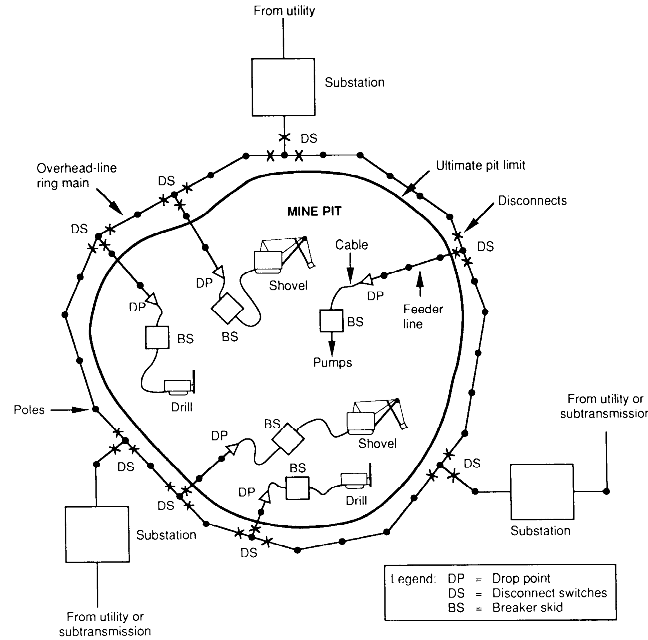

Fig. 12.4.10. Distribution system for an open pit mine illustrating a ring main.

The layout for all-cable mine distribution (Figs. 12.4.6 through 12.4.8) is very similar to that just described. In this case, however, the baseline is assembled using cable-interconnected switchhouses. The common approach is to use disconnect skids with three internal switches in the baseline and have separate breaker skids in line with the cables feeding the machinery. Another approach is to combine the single breaker skids into the baseline switchhouses.

When a secondary-selective configuration is used, as shown in Fig. 12.4.7, a normally open-tie circuit breaker is placed in the baseline in a location approximately equidistant from the main substations. In some operations, the two substations and the tie circuit breaker may be in the same location with two feeders running from the substation area to the baseline. More than two main substations may be established in very large operations.

Distribution voltage for the surface mine may be 7.2, 13, 23, and to a lesser extent 4.16 kV. Regardless of the level, drills and production shovels usually operate at 7200 or 4160 V. Therefore, when higher distribution levels are needed, portable unit substations are commonly used in the pit. One instance would be when the load created by a large machine is several times that for auxiliary machines. Another method is to establish two baselines on the highwall for two distribution voltages, as shown in Fig. 12.4.8. Here, a large unit substation interconnects the two baselines. Even in this situation, as can be seen in the preceding drawings, low-voltage unit substations or power centers are often required for 480-V auxiliary equipment.

The primary purpose of any distribution scheme in a surface mine is to provide a flexible, easily moved or modified power source for the highly mobile mining equipment. System designs must also be considered as an integral part of the total mine operation. Those described have these objectives in mind. The distribution system in any surface or underground mine that serves portable equipment is subject to damage from the mining machinery itself, and as a result, the system must be designed with optimum flexibility and consideration for personnel safety.

Open pit power systems are quite similar to those of strip mines with one main exception: distribution typically establishes a ring bus or main that partially or completely encloses the pit. Radial ties to the bus complete the circuit to switchhouses located in the pit, and portable equipment again uses shielded trailing cables. An example is shown in Fig. 12.4.10. Distribution voltage is normally 4.16 kV, but 7.2 (or 6.9) and 13.8 kV are sometimes used. Unit substations are employed if equipment voltages are lower. The distribution portion of the system is almost always constructed with overhead lines.

12.4.5.3 Underground Mine Distribution

As shown in Fig. 12.4.11, underground mine power systems are somewhat more complicated than those for surface applications. A coal-mine power system is depicted as an example. By the nature of the mine and its service requirements, distribution must almost always be radial; the freedom in routing distribution enjoyed by surface mines is not available underground. For increased reliability, secondary-selective main substations are employed (Fig. 12.4.12). Distribution voltage is most commonly 7200 V; however, older 4160 V systems can still be found, and 12,470 V has increased in popularity in recent years.

Power and mine grounding are fed underground in insulated cables, either through a shaft or borehole or an intake entry. In coal mines, these cables are required to terminate in disconnect switches within 500 ft (152 m) of the point of power entry into the coal seam. These switches allow total removal of underground power in an emergency. From the disconnects, which may be part of a switchhouse, the power is distributed through cables to power centers or rectifiers that are located as close to the machinery as practical. All the cables on high-voltage circuits, usually involving only distribution, have shielding around each power conductor.

The prime load concentrations in underground mining are created by the mining sections. Distribution terminates at the section power center that is a transformer combined with a

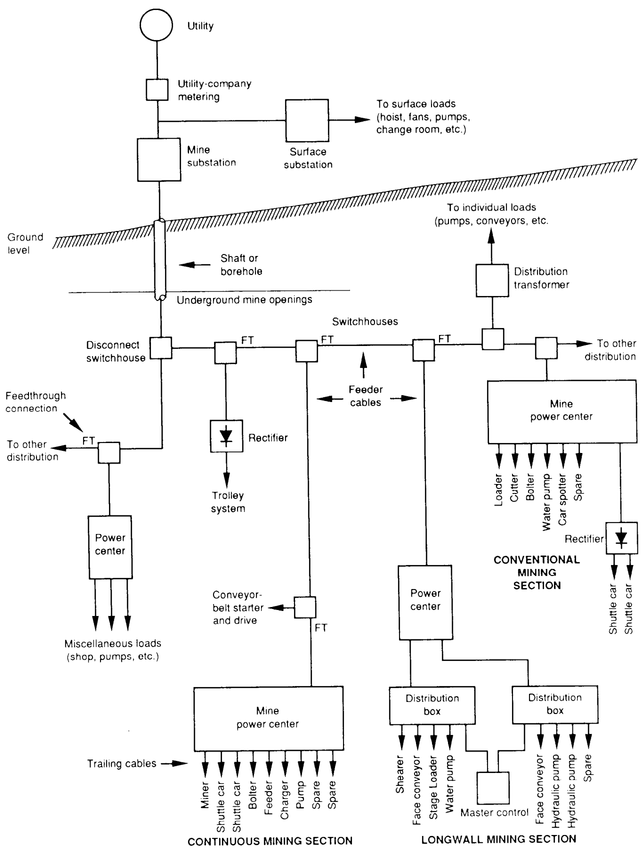

Fig. 12.4.11. Simplified radial distribution system for an underground coal mine.

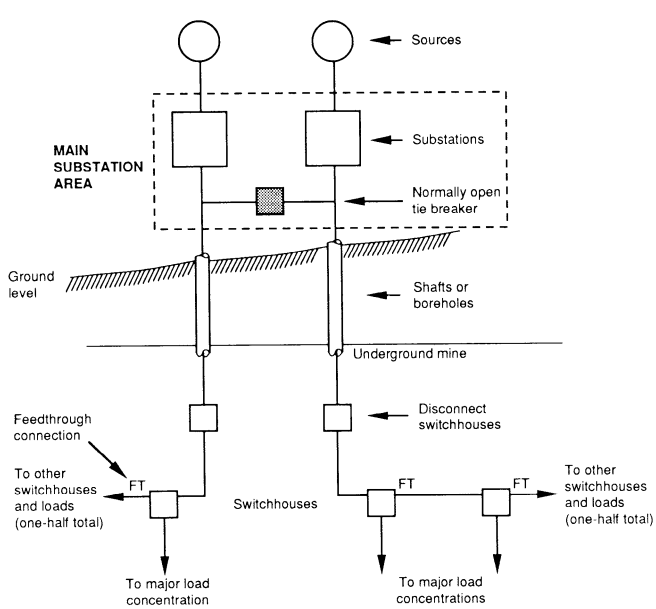

Fig. 12.4.12. Secondary-selective distribution applied to an underground mine.

utilization bus and protective circuitry. From this, several face machines are powered through couplers and trailing cables.

If belt haulage is used, distribution transformers are located close to all major conveyor belt drives and are referred to as belt transformers. After transformation, power is supplied through starter circuitry to the drive motors. With rail haulage, distribution terminates at rectifiers that contain a transformer and rectifier combination. The rectifiers are located in an entry or crosscut just off the railway. Direct-current power is then supplied through circuit breakers to an overhead conductor or trolley wire and the rail, with additional rectifiers located at regular intervals along the rail system. For further protection, the trolley wire is divided into electrically isolated segments. The typical rectifier supplies the ends of two segments of trolley wire, and each feeder has its own protective circuitry to detect malfunctions. In some mines, dc face equipment and small dc motors are powered from the trolley system through a fused connection (or nip) to the trolley conductor and rail. The dc distribution can also serve large motors directly through switchgear; however, this practice is rare.

All power equipment used underground must be rugged, portable, self-contained, and specifically designed for installation and operation in limited spaces. In addition, all equipment and the cables connecting them must be protected against any failures that could cause electrical hazards to personnel. This is provided primarily by protective relaying built into each system part, with redundancy to maximize safety.

12.4.5.4 Mine Grounding Systems

The concept of protecting mine electrical equipment and personnel against the consequences of electrical failures by suitable grounding has existed since electricity was first introduced into mines. As early as 1916, the US Bureau of Mines recommended equipment frame grounding as a means of preventing electrical shock to miners working on or around electrical equipment (Clark and Means, 1916). For the mining industry, a suitable grounding system has always been a difficult problem, more complex and difficult than in other industries.

GROUND BEDS. For mine usage, the electrical distribution cables and overhead transmission circuits carry into the mine one or more grounding *conductors* in addition to the line conductors. Each piece of ac equipment has its frame solidly connected via these grounding conductors to a safety ground bed commonly located near the main substation and consisting of buried horizontal conductors, driven rods, or a combination of both. The neutral of the substation-transformer secondary is also connected to the safety ground bed through the neutral grounding resistor (Trasky, 1965).

The substation actually requires two ground beds, seen in Fig. 12.4.4, maintained some distance apart. Lightning discharges and other transformer primary surging conditions are directed to the system or station ground. The system and safety grounds must be kept separate so current flow intended for one will not enter the other. It is essential for the safe operation of the mine power system that the resistance of the beds be maintained at 5.0 Ω or less (Anon., 1964a; King et al., 1978; Lordi, 1963). A ground bed with this resistance range is often termed a low-resistance ground bed.

GROUNDING IN UNDERGROUND MINING. Early practice in underground mining was to drive a metal rod into the mine floor and use that as a ground. In almost every case, this arrangement proved to be totally unacceptable. With the exception of pumps, the contact resistance of mining machinery with the mine floor also proved to be too high for adequate grounding. Rail-haulage track systems, even though often poorly bonded, showed much lower resistance to ground than most metallic rods driven specifically for that purpose. As a solution, Griffith and Gleim (1943) stated that consideration should be given to a grounding circuit carried to the outside of the mine. Present mine practice adheres to this concept.

A simple form of grounding systems in use in underground mines today is described as follows. After transformation, three-phase ac power enters the mine to supply the various three-phase ac loads. Some of the ac power is converted to dc at rectifier stations to power the trolley system and occasionally dc face equipment. More often, any dc face machinery is powered from rectifiers located in the mine section. Except for the trolley system, all dc as well as the ac equipment frames are connected to a common junction that is tied to the surface safety ground bed. In order for the system to be effective, grounding conductors must be continuous, and this continuity must be verified.

Trolley locomotives generally utilize the overhead trolley wires as the positive conductor and the tracks as the negative. Neither of these is tied to the rectifier-station frame ground. However, because the track is in contact with the mine floor, the negative conductor for the trolley system is grounded. The dc system that supplies power to face equipment normally employs trailing cables that have neither the negative nor positive conductor grounded.

At each transformation step within the power system, such as in a power center, an additional neutral point must be established on the transformer secondary. The neutral is tied through a grounding resistor to the equipment frame, and from there through the grounding conductors to the safety ground bed.

GROUNDING IN SURFACE MINES. The typical grounding system for a surface mine is similar to that for underground mining. One or more substations with resistance-grounded secondaries are employed to transform the incoming utility voltage to the lower potential used by the mining machines. At this level, pit distribution is carried on overhead lines or cables to supply switchhouses located near the particular piece of equipment. A trailing cable completes the power circuit from the switchhouse to the machine. A switchhouse is sometimes connected by cable to a portable substation which supplies lower voltage power to production, auxiliary, or lighting equipment. Substation grounding includes both a system and a safety ground bed, each physically removed and electrically isolated from the other.

12.4.5.5 Protective Relaying In Mining

There are two groups of protective relaying equipment within a typical power system: primary and backup. Primary relaying has the goal of clearing all faults and overloads and aims to isolate offending power-system segments with minimum interruption to the system balance. Backup relaying operates only in the event of a primary relaying failure; its action is only for uncleared faults. For overload, short-circuit, or ground-fault relaying in mining, both groups are used extensively to the point of redundance.

ZONES OF PROTECTION. Protection to the entire system is principally related to primary relaying and is accomplished by establishing zones of protection. Each zone has an associated circuit breaker and fusible disconnect, or fuses with the required sensing devices, and adjacent zones overlap (Mason, 1956). If a failure occurs within an individual zone, only the switching apparatus within that zone should open. If there was no overlap, there could be an unprotected region in the system in some situations; a failure within this area would produce no safety tripping. The coordination of protective relaying between the zones is extremely important. The aim is to isolate faults downstream from the power source without disturbing upstream zones. Unfortunately, obtaining this coordination is perhaps the most serious problem of relaying.

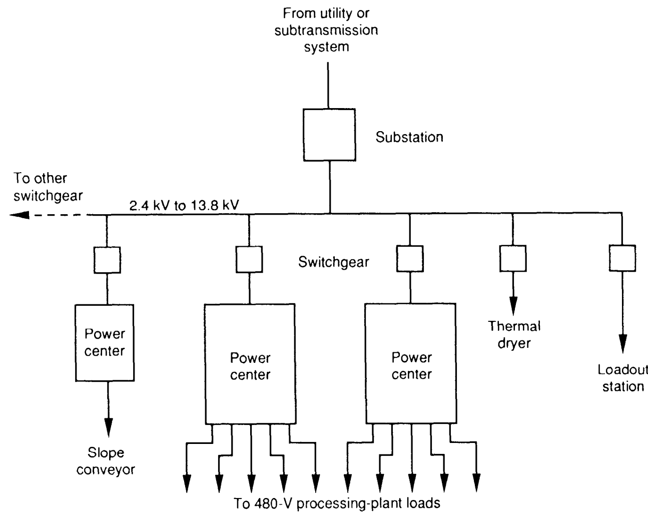

Fig. 12.4.13. Simple radial distribution applied to a mineral processing plant.

The objective of coordination is to determine the optimum characteristics, ratings, and settings for the protective relaying devices (Anon., 1986); consequently, fault analysis of the system must be involved. The two common coordination schemes that are utilized are pickup setting and time. One technique or the other may be applied to provide coordination between zones in the mine power system; at times, a combination might be needed. The design of this protective relaying system can be a substantial problem, as exemplified by the fact that many engineers consider protective relaying more of an art than science.

12.4.5.6 Mineral Processing Plants

The surface activities of any mine, which may include shops, change rooms, offices, ventilation fans, hoisting equipment, processing plants, and so forth, can have large power requirements. For safety, these facilities should at least have an isolated power source and, at times, a separate substation.

In mineral processing plants, the distribution arrangements are almost always expanded radial or secondary-selective (Lordi, 1956). Representative system layouts are shown in Figs. 12.4.13 and 12.4.14. In both, distribution is at 2.4 to 13.8 kV, with 4160 V the most common. Power is distributed at one of these levels to centers of electric load. This power may be used directly for high-voltage motors, but usually the voltage is stepped down to supply groups of motors or single high-horsepower motors. The power centers must be in an elevated location or totally enclosed. The rooms used for these and other electrical components may also be pressurized to exclude coal dust.

The most popular voltage for processing plant utilization is 480 V. This voltage is used to drive all motors throughout the plant except those with high horsepower demands, such as centrifugal driers and large fan drives where 2300 or 4160 V is commonly employed. These higher voltages may also be preferred for any motor that requires continuous service or independence from the power-center loads. Most modern mineral processing plants use group motor control instead of individually housed control units, since this method facilitates maintenance and enables the interlocking of the various motor functions re-

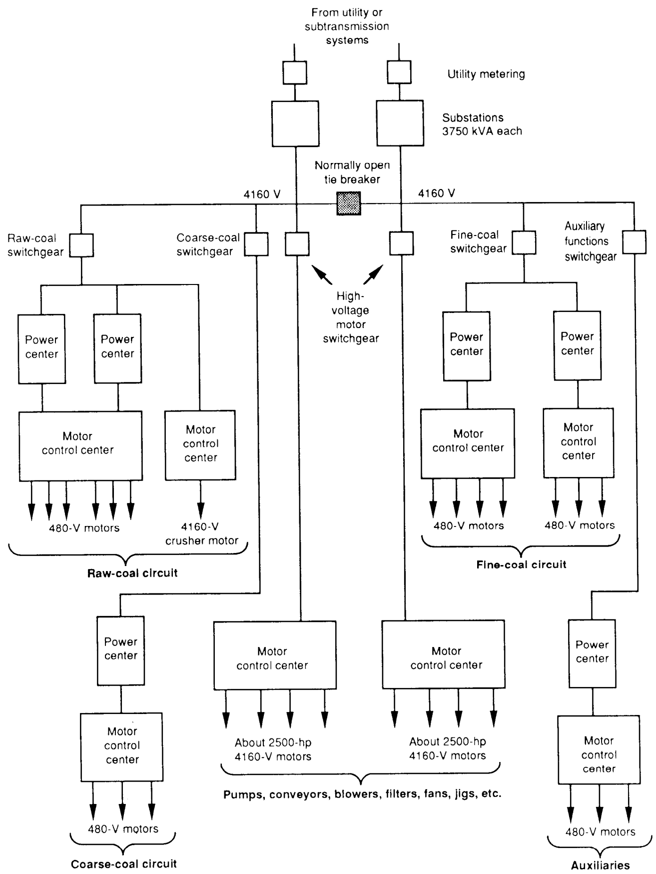

Fig. 12.4.14. Secondary-selective distribution applied to a coal processing plant.

quired for semi-automatic facilities. All manual controls, indicating lights, and so on are grouped in one central operating panel to allow easy access and visual indication of plant operation. The panel is often called a motor control center, as noted in Figs. 12.4.13 and 12.4.14.

12.4.6 SPECIAL CONCERNS

12.4.6.1 Mine Motors

Mine motor functions can usually be divided into two groups: auxiliary and face (Tsivitse, 1969). Auxiliary motors are employed for fans, pumps, conveyors, hoists, compressors, and other vital functions in mines aside from the actual process of mineral extraction. These operations commonly call for direct use of general purpose industrial motors. Face motors are associated with mining equipment, such as continuous miners, shuttle cars, loaders, roof bolters, and locomotives, where they are mounted in the machine. Their duty usually involves cyclic or random loading as well as the possibility of shock loading. The result is higher electrical and mechanical demands than those placed on equipment in other industrial applications. Because mining equipment operates at the tail end of the distribution system, voltage drop becomes an important factor in the selection and utilization of motors. This is more critical for ac equipment than it is for dc.

The horsepower rating for a motor is based on the maximum winding temperature for a continuous duty or intermittent duty. Continuous duty is quite obvious and refers to a substantially constant load (torque) over an indefinitely long period. Intermittent duty, however, means that loading is at alternate intervals of load and no load (motor running idle); load and off; or load, no load, then off (Lloyd, 1969). Each portion of the cycle is equal, and the time interval is specified. In some cases, face-motor intermittent duty is given a definite time interval of 15, 30, or 60 min, but it is often just listed as mine duty (Hugus, Buss, and Parker, 1955; Tsivitse, 1969).

Alternating-current motors in mining machines normally have four or six poles with synchronous speeds of 1800 and 1200 rpm, whereas dc motors often have comparable base speeds of 1750 and 1175 rpm. These are high enough to provide adequate horsepower, but low enough to have reasonable reliability. Series-wound motors for traction are built to withstand rotation up to 6000 rpm, such as might occur during maintenance.

Mine motors are presently standardized at a 90°C (194°F) allowable temperature rise based on a 25°C (77°F) ambient temperature. Older motors may have a 75°C (167°F) temperature-rise limit but are still based on the 25°C (77°F) ambient. The base temperature closely fits the typical conditions found in underground operations. The allowable temperature rise is effective to elevations of 3300 ft (1000 m); above this, the allowed temperature rise is reduced 1% for every 330 ft (100 m). In addition, for maximum ambient temperatures exceeding 25°C (77°F), the allowable rise must also be reduced by the difference above the base temperature. For example, a motor with 75°C (167°F) temperature-rise insulation, operating at 6600-ft (2000-m) elevation in 30°C (86°F) ambient temperature, has only a 62.5°C (144.5°F) allowable temperature rise.

Although the vast majority of mine motors are three-phase and dc, single-phase motors do find some use for auxiliary functions aside from the mining process. As a general rule, single-phase induction motors have one running speed and require a separate means for starting rotation, usually a separate stator or starting winding. Motors are classified by their starting method. The most common techniques are split-phase and capacitor-start.

12.4.6.2 Permissibility and Hazard Reduction

Any industrial area in which flammable or explosive gases, vapors, and dust can be encountered is designated as a hazardous location. Since the occurrence of this hazard depends upon the presence of an ignitable mixture, an ignition source, and contact between them, the chance of ignition is always present when electrical apparatus is used in hazardous atmospheres. The possibility of ignition cannot be totally eliminated in any portion of an underground mine, nor in selected parts of surface mines and surface facilities such as preparation plants. Hence electrical hazard-reduction techniques must be applied in these areas to protect both personnel and equipment.

TERMINOLOGY. Important measures used to reduce incendive hazards in underground mines include provision of adequate ventilation, control of flammable coal dust through mandatory rock dusting and watering, and the regulation of equipment. There are specific terms used by federal regulatory agencies in the United States for electrical equipment intended for use in underground gassy mines (Anon., 1981).

Accessories mean associated electrical equipment such as a distribution or splice box that is not an integral part of the machine. *Approval* applies to completely assembled electrical machines and accessories; it means that a formal document has been issued by the regulatory agency which states that the machine or accessory has met the applicable requirements of the regulation. An *approval plate* is then attached to the approved machine or accessory identifying it as suitable for use in hazardous locations. Such equipment is subsequently referred to as *permissible equipment*. This process is mandatory for all electrical equipment used in the face area and in return air of gassy mines. The term *certification* applies to an electrical component, that is, an integral part of an electrical machine or accessory that is essential to the functioning of the machine. Certification means that a formal written notification has been issued by the regulatory agency, stating that the component complies with federal requirements and is suitable for incorporation in a permissible machine. *Acceptance* is connected to flame-resistance requirements and is applied to auxiliary equipment such as a cable, hose, or belt. Acceptance means that written notification has been received from the regulatory agency, designating the equipment as meeting requirements for flame resistance. *Acceptance marking* is the identification that appears on the equipment. *Intrinsically safe* identifies equipment that is incapable of releasing enough electrical or thermal energy under normal or abnormal circumstances to cause ignition of a flammable mixture.

Hazardous locations in US surface mines and surface portions of underground mines are often classified by guidelines specified in the National Electrical Code, Article 500 (Anon., 1990). In this classification system, the nature of the hazard and the degree of hazard are the main considerations, and a location is specified by a class, group, and division designation. The class refers to the generic nature of the hazardous material, and the following are of specific importance in mining:

Class I: locations containing flammable gases or vapors that may be present in the air in sufficient quantity to produce an explosive or ignitable mixture.

Class II: locations having combustible dust in quantities that can cause a hazard. The group designation is a subclassification and refers to the nature of the hazard. Each group contains a listing of materials that present the same general hazard. The groups of interest in coal mining include:

Group B: atmospheres containing hydrogen or gases or vapors of equivalent hazard such as manufactured gas.

Group D: atmospheres containing gasoline, hexane, naptha, benzine, butane, propane, alcohol, acetone, benzol, lacquer-solvent vapors, or natural gas (methane).

Group F: atmospheres containing carbon black, coal, or coke dust. When more than one hazard is involved, the class and group describing the most serious situation usually applies. The division defines the probability of a hazardous material being present in an ignitable concentration:

Division 1: locations where the hazards exist continuously, intermittently, periodically, or where they may exist during maintenance or equipment failures.

Division 2: locations where hazards are presumed to exist only under abnormal conditions.

HAZARD REDUCTION METHODS. Hazard reduction is a practice in low probabilities, specifically low incremental probabilities, and relies on the fact that a safe electrical installation in a hazardous location does not significantly raise the probability of fire or explosion above that existing without the equipment (Magison, 1978). The methods adopted to reduce hazards are based on the principle that the occurrence of a hazard depends upon the presence of an ignitable substance, an ignition source, and contact between them. The methods focus on one or another of these interrelated factors. Among hazard-reduction methods used in or around mines, explosion-proof and dust-ignition-proof containers are the most important. Here internal ignition is possible, but the resulting combustion is so well controlled and contained that hazard is prevented.

Explosion-proof Enclosures—The explosion-proof enclosures used in US mines comply with applicable federal design requirements (Anon., 1981). They are able to contain internal explosions of methane-air mixtures without undergoing damage or excessive distortion of its walls or covers, without causing an ignition of a surrounding methane-air mixture, and without discharging flame from the inside to the outside of the enclosure. Outside the United States, the same definition often refers to flameproof enclosures, but the words "discharge of flame" are omitted (Anon., 1971). Although the terms flameproof and explosion-proof commonly have the same connotation, flameproof enclosures are not presently allowed in US underground gassy mines unless they also meet the federal requirements.

Explosion-proof enclosures are found on all electrically powered face mining machines in US gassy mines and all motor applications in Class I, Division 1 locations in all US industries. These enclosures have heavy-walled cast or welded construction and bolted or threaded close-fitting flanges; however, they are not necessarily vapor tight. Ignitable gas may enter a properly secured explosion-proof enclosure in several ways (Jones, Altimus, and Myers, 1971). Enclosures on machines allowed to stand for several hours in a gas-filled place can become completely filled with that gas from diffusion through openings, even though those openings are very small. Another process, "breathing," results from the expansion of the enclosure atmosphere during operation and contraction as it cools at rest. Air is forced out during expansion, and the atmosphere outside including any gas present around the machine is drawn in when cooling. Gas may also enter the enclosure when covers are removed for inspection and repair.

Ignition of the explosive gas by electrical means can be triggered in several ways. Arcs of sufficient energy, termed incendive, can result from normal or abnormal operation of electrical devices. Another source of ignition is contact with hot surfaces, but this depends on the surface heating sufficiently above the auto-ignition temperature (Magison, 1978). One reason for a hot surface would be excessive current through fine conductors, such as in a damaged cable.

Although explosion-proof enclosures have been researched since the turn of the century, little concrete information has been gained about the phenomena that make them effective, but it is accepted that cooling and inhibition are two of the more probable mechanisms (Phillips, 1971). The hypothesis can be explained considering that an explosive methane-air mixture can exist both inside and outside the chamber. After an explosion is initiated inside the enclosure, a flame front propagates toward the chamber walls, burning the available combustible material. This raises the temperature and pressure inside the chamber, and unburned gas, then heated burned gas, are forced through the flange gap. The jet of heated gas is cooled first by heat transfer within the flange gap. As it exits from the enclosure, it may be further cooled by adiabatic expansion into the surrounding atmosphere; during this process, combustible gas from the outside methane-air mixture is also entrained in the jet. If the jet entrains an excessive amount of combustible gas in this last phase, the heat supplied by the jet will be less than that lost through cooling, and no ignition of the outside gas will occur.

The action within the flange gap is believed to go beyond cooling. Combustion of a gas-air mixture does not proceed with a single chemical reaction but rather a chain of reactions. If the chain carriers or active molecules from a preceding step are inhibited, for instance by contact with the flange wall, then combustion stops. This theory is also used to explain the protection properties of flame safety lamps.

Reducing Dust Hazards—In Class II hazardous locations, the types of enclosures in common use are dust-ignition-proof and dust-tight. Pressurized and intrinsically safe systems are used less frequently (Magison, 1978). The objective of these enclosures is to keep dust away from ignition sources and to prevent ignition of layered dust. Dust-ignition-proof enclosures meeting Underwriters Laboratories, Inc. requirements conform to both objectives (Anon., 1974). The requirements are similar to those for explosion-proof containers but less severe. The goal is to provide an enclosure that will prevent hot particles from escaping and dust from entering. The maximum allowable temperature for the surface of such equipment is 150°C (302°F) for equipment that can experience overload and 200°C (392°F) for equipment that will not usually overload. Dust-ignition-proof motors are suitable for both Division-l and Division-2 locations.

Dust-tight enclosures are intended only for Division 2 locations. The standards for their dust-tight construction are less restrictive than for equipment in Division 1 locations, but they undergo the same tests by the Underwriters Laboratories, Inc. (Anon., 1973).

OTHER HAZARD REDUCTION METHODS. *Increased safety* is an approach in hazard reduction in which special design considerations are given to ensure an extremely low probability of electrical or mechanical breakdown that could produce a spark or temperature rise. Like intrinsic safety, the principle is to remove the ignition source. The technique is widely used in West Germany and is frequently applied in mine lighting fixtures. Increased safety designs include such features as large spacings and creepage distances between live parts, protection against hot spots, large rotor-stator clearances for motors, superior amounts and quality of insulation, and special enclosures and fastenings to prevent unauthorized entry.

Immersion systems rely on controlled environments to reduce or remove hazardous atmospheres from ignition sources. In sealing or potting methods, granular material such as sand is used to encapsulate the potential hazard. This does not actually isolate the source but quenches any incipient flame, thus preventing any effective contact between the source and the hazardous atmosphere. Purging or pressurized systems use liquid (oil) or inert gas to provide protection. The pressurized approach is rarely used in mining applications.

The basic problem with most of these hazard-reduction methods is their complexity. The overriding considerations for electrical hazard reduction in mining are robustness, interchangeability, and mobility; hence there is no indication that bolted explosion-proof enclosures will lose their popularity in the near future.

INTRINSIC SAFETY. As previously defined, intrinsically safe equipment is incapable, under normal or abnormal conditions, of releasing sufficient energy to cause an ignition of the most ignitable methane-air atmosphere (Widginton, 1968). The principle applies to complete electrical circuits and not to individual components. The advantage is that safety is inherent in the design and is difficult to defeat. Normal conditions are taken to include the effect of extreme power-supply and environmental variations (within the equipment specifications) as well as the opening, faulting, or grounding of all conductors leading to the apparatus (Magison, 1978). Abnormal conditions involve all failures of internal components and wiring.

12.4.6.2 Batteries

Batteries have a variety of uses in mines, including powering cap lamps, instruments, and equipment such as locomotives, articulated ram-dump haulers, tractor-trailer units, and scoops (front-end loader tractor units). The battery-powered off-track units eliminate problems involved with shuttle car trailing cables. There are inherent hazards in battery use: batteries emit hydrogen, an explosive gas, while charging; batteries and battery chargers are capable of delivering a fatal electric shock; and batteries can be a potential fire hazard. A number of less catastrophic hazards may also be encountered, ranging from acid burns from spilled electrolyte to pinched fingers from careless handling.

A *storage battery* can be defined as a battery in which the electrochemical action is reversible; that is, after an output of electrical current (discharge), the battery can be returned to the original state (recharged) by passing current through it in the opposite direction. Two types of storage batteries have been employed in underground traction, alkaline and acid. The nickel-iron or Edison cell is an alkaline cell because of the type of electrolyte used. Edison batteries were once popular in the mining industry because of their high-reliability and low-maintenance characteristics, but lead-acid batteries have now entirely replaced the Edison type as a result of their high energy per unit volume and high power capability.

About 110% of the ampere-hours discharged must be returned to fully charge a lead-acid battery. The rate at which charge is restored is an important consideration in attaining the maximum number of charge cycles and maximum life for the battery. Manufacturers may publish a normal or finish rate; this is also the current level at which the battery can be safely charged any time charging is required. The charge cycle usually takes eight hours. Most of the electricity supplied to a cell being charged is used to transform water and lead sulfate into sulfuric acid, lead, and lead peroxide, but some of the current causes electrolysis, breaking the water down into its constituents, which is called gassing. A certain amount of gassing is a necessary consequence of a good charge, which explains why water must periodically be added to batteries, but excessive gassing or overcharging causes damage to the plates, excessive water consumption, and excessive hydrogen emission. For this reason, the charging current must be controlled as the battery charges.

Proper battery maintenance has a very significant influence on battery life. Battery manufacturers normally include a recommended maintenance program with their batteries based on specific gravity levels and equalizing schedules. Equalizing is the process by which all the cells in a battery are brought to the same voltage. In lead-acid batteries, the electrolyte specific gravity is a function of the state of battery charge. Consequently, a plot of electrolyte specific gravity vs. discharge depth for a particular battery is important. All lead-acid batteries require periodic equalization, but excessive equalization can cause unnecessary battery deterioration.

Daily battery maintenance activities should include the monitoring of one battery cell, which is called the pilot cell. Any battery cell can be used as the pilot. The following cell characteristics should be recorded during each charge: (1) specific gravity before and after charging and (2) water level in the cell. If any of the pilot parameters falls outside those specified as acceptable by the manufacturer, all cells should be checked and corrective action should be taken. Other daily maintenance activities should include checking the battery for physical defects such as cracked cell plugs and ensuring that the charger output voltage is correct. The weekly maintenance program includes checking all battery cells for proper water level (the water consumption of a good battery is generally equally distributed among the individual cells) and routine cleaning of the battery tops. Every three months, it is good practice to take a complete set of cell voltage and specific gravity readings at the end of an equalizing charge to ensure that these parameters meet manufacturer specifications.

CHARGERS. Most mine batteries are charged from the ac distribution system, using a transformer-rectifier combination. Several methods can be used to control the rate of charge. Most are designed to initiate the charge at a fairly high current or starting rate. The level is then tapered off to the finish rate as the battery charge is restored. Some chargers reduce the starting rate to the finish rate in one step when the charge is about 80% complete, but control devices that taper the charge rate are more common in mining. Regardless of the type, the charge is usually stopped automatically when full charge is reached.

CHARGING STATIONS. Special charging stations are required in underground mines to charge vehicle batteries. These stations must be designed and constructed to meet specific ventilation requirements, but because mining methods and plans vary widely, it is extremely difficult to define a rigorous set of guidelines.

BATTERY-BOX VENTILATION. The ventilation of the traction battery enclosure is not only important while charging but also during operation. Following the charge cycle, the covers are closed and the vehicle is placed in service. From chemical reactions and entrapped gas within the cells, lead-acid batteries continue to emit gas for several hours after receiving a charge, no matter whether they are open-circuited or on discharge. Considering the ignitability of hydrogen, it is possible that a dangerous hydrogen-air mixture will accumulate. Hence it is necessary to provide adequate ventilation for the evolved hydrogen in the closed box while the battery is in operation.

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Chapter 12.5
COMPRESSED AIR POWER

J.L. GENT AND R.H. KING

12.5.1 INTRODUCTION

12.5.1.1 Scope and Objective

Compressed-air-powered equipment is widely used in many underground metal and nonmetal mines and even in some coal mines. Compressed air power is expensive due to the high costs of energy, equipment, and supplies. Furthermore, if air quantity and pressure are not available when needed, resulting delays and decreased operating efficiency can severely reduce productivity. In fact, lost production is probably the highest cost of an inadequate compressed air system.

Therefore, optimal system design is necessary to obtain the required air power at the lowest cost. This chapter demonstrates a correct design procedure using manual calculations and explains how computer-aided design is useful for alternative analysis in large systems.

12.5.1.2 Design Method Summary

The procedure described herein addresses determination of the following compressed air requirements:

1. Air quantity, pressure, and compressor horsepower.
2. Valve and fitting selection and their proper placement.
3. Optimal pipe diameter(s).
4. Hose diameter(s).
5. Receiver size.
6. Overall system capital and operating costs..

To determine these requirements, a sizable amount of information must be gathered or estimated for either manual calculations or the computer program. Initial resources should include:

1. A scaled mine map or schematic showing possible routes so that preliminary pipe lengths and fitting locations can be determined. The drawing should also include probable locations of all air consuming equipment.
2. An overall production schedule for the longest planning horizon possible is needed if compressed air quantity and maximum pressure requirements are to be determined for various critical times in the future. This schedule may be mapped, graphed, or otherwise presented so it can be analyzed by stages. For example, mine management may wish to defer capital expenditures by purchasing a smaller compressor initially and adding additional units later in the mine life.
3. All available or planned compressed-air-powered equipment and the corresponding manufacturer's specifications in psia and scfm or lbm/sec (kPa and m^3/s).
4. Diversity factor estimation, equipment scheduling, and operational priority.
5. The types and sizes of hoses, pipes, valves, compressors, and receivers that are currently available or are preferred by the engineer.
6. Leakage location and magnitude, when known.

After this information is gathered, the following design steps are undertaken:

1. Determine machine requirements.
2. Estimate leakage locations and quantities and apply them to the system.
3. Calculate equivalent lengths for valves and fittings and add to the downstream branch.
4. Decide equipment placement.
5. Balance flow rates at nodes (where possible).
6. Calculate pressure losses.
7. Calculate inlet pressures and quantities for the network.
8. Calculate compressor horsepower and receiver capacity.
9. Calculate optimal pipe diameters. Repeat steps 6 thru 9 until optimal diameters become fixed.
10. Analyze the system to further minimize costs, if possible.
11. Calculate capital and operating costs.

The required calculations are computationally intensive, but available computer programs remove the repetitive calculations, allowing the engineer to concentrate on design improvements.

A simple mine layout is used as an example throughout this chapter to illustrate the design procedure. The procedure developed can be used to obtain a quick approximate solution or a precise result, depending on the availability of good input data and the user's requirements.

For additional information on these methods, the reader is referred to CAGH, *Compressed Air and Gas Handbook* (Rollins, 1973), or *Flow of Fluids Through Valves, Fittings, and Pipe,* Technical Paper 410 of the Crane Co. (Anon., 1978). Both references focus on volumetric analyses. Mass flow analysis is more accurate but requires longer solution times. The design procedure covers steady-state analysis only. A computer program designated COMPAIR (Gent, 1986) was used to develop the numerical examples in this chapter.

12.5.2 DESIGN CONDITIONS

The example mine encompasses radial and combined (loop and radial) compressed air systems (Fig. 12.5.1). Equations are presented for the optimum pipe diameter, air quantity and pressure, compressor horsepower, and receiver size for both types of distribution networks.

Figs. 12.5.2 and 12.5.3 are schematics of the example mine radial and combination networks, respectively, which include the section numbers, and valves and fittings. Pipe intersections are defined as *nodes,* and a *branch* is defined as the pipe and fittings between nodes.

Eight air hammers requiring 175 scfm (0.08259 m^3/s), two air hammers requiring 123 scfm (0.05805 m^3/s), and three overshot muckers that require 210 scfm (0.09911 m^3/s) per motor are used in the example. All equipment requires 90 psig (620.5 kPa) minimum. The example mine surface facilities are at 7800 ft (2377 m).

Table 12.5.1 lists the abbreviations for drift headings, raises, and stopes used in Figs. 12.5.1 through 12.5.3. Tables 12.5.2 and 12.5.3 show pipe lengths and diameters for radial branch pipe and the combination network respectively. Symbols used in the figures are American National Standards Institute (ANSI) standards.

Although the example mine is small, the iterative design requires the same series of steps as a larger operating mine: (1) determine machine requirements; (2) calculate the quantity in each branch and in total; (3) adjust quantities for altitude and leakage; (4) select line sizes; (5) place the appropriate valves and

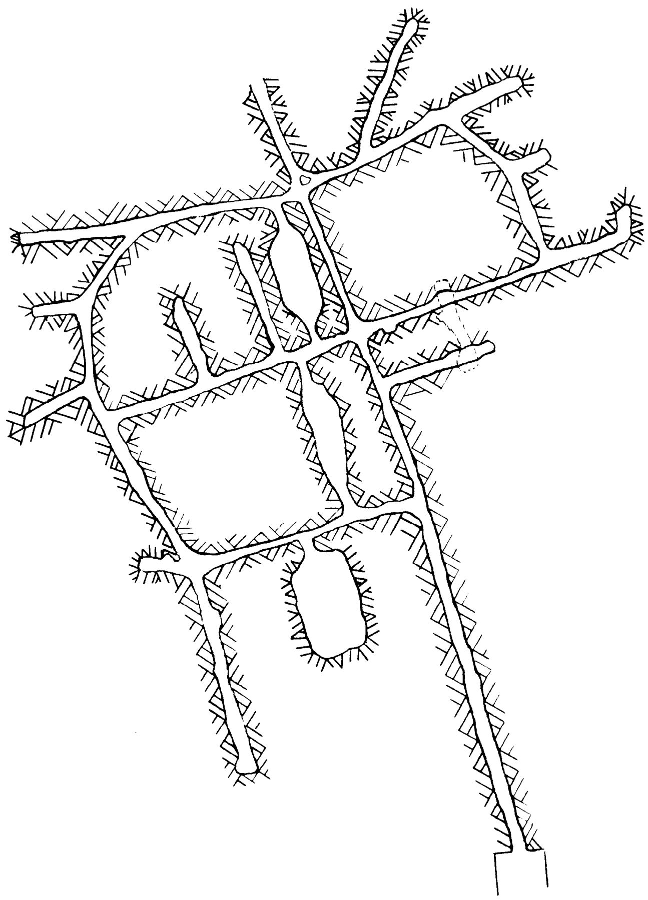

Fig. 12.5.1. Example mine layout.

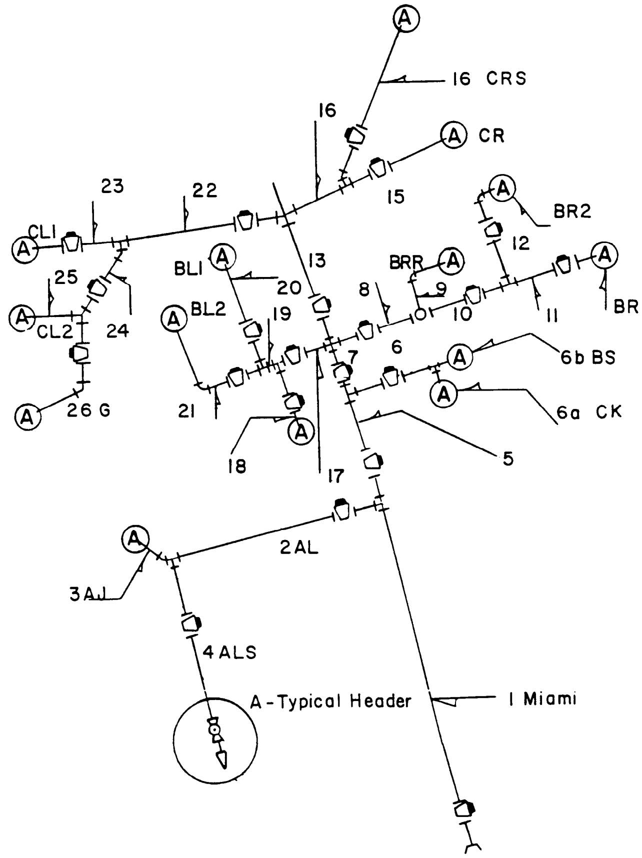

Fig. 12.5.2. Radial network layout.

fittings; (6) calculate the equivalent length for each branch; (7) calculate the corresponding pressure drop; (8) determine compressor air quantity, discharge pressure, and horsepower; (9) determine optimal diameters; (10) calculate the required receiver volume; (11) estimate the overall cost; and (12) reiterate parts of the solution to analyze effects on the assumed data.

12.5.3 MACHINE REQUIREMENTS

Compressed air requirements for most equipment can be obtained from the manufacturer. For the equipment to perform optimally, its air horsepower requirements must be met.

Quantity is normally expressed as the volumetric free air flow rate at a specific pressure. Most manufacturers define *standard free air* volume as 1 ft^3 of air at 60°F, 14.7 psia, and 0% relative humidity (0.002832 m^3, 15.6°C, 101.4 kPa)

Manufacturers always include the operating pressure with their stated quantities because operating pressure affects rotation rates and blow frequencies. The operating pressure is usually expressed as gage pressure (psig or kPa), which is the pressure differential between the compressed air and free atmospheric pressures. The corresponding quantity is normally denoted scfm or cfm (m^3/s).

Mass is also used to designate flow rates throughout the chapter (lbm/sec). When a *mass flow* is stated, temperature and pressure accompany the rate. If temperature is not designated, standard conditions are assumed.

An acfm is 1 ft^3 (0.002832 m^3) of air at any condition. As an illustration, consider the internal volume flowing in a pipe. It is equal to scfm divided by the compression ratio *r*, using the assumption of isothermality. The compression ratio is defined as the absolute pressure of the compressed air divided by the atmospheric pressure:

$$r = \frac{\text{psia of compressed air}}{\text{psia of free air}} \qquad (12.5.1)$$

If drill quantity requirements are unknown and manufacturers' data are unavailable, they can be estimated using data from Staley (1949) or calculated if additional information is known.

To calculate the air requirements for a drill or other machinery, additional information (the bore, stroke, frequency of blows, or total horsepower) must be known. For a piston-type machine such as a drill, the calculation procedure is as follows:

1. Calculate the per stroke displaced air volume.
2. Multiply the stroke volume by the frequency to obtain acfm at operating conditions.
3. Convert acfm to scfm by multiplying by the compression ratio.

This calculation should always be based on standard air to insure compatibility with altitude correction factors. If so, results will closely parallel published data. However, manufacturers base some data on bench tests that differ from calculated values.

Example 12.5.1. Calculate the required scfm if a drill has a 3.0-in. (76.2-mm) bore and a 2.5-in. (63.5-mm) stroke. The drill delivers 2400 bpm at 90 psig (620.5 kPa).

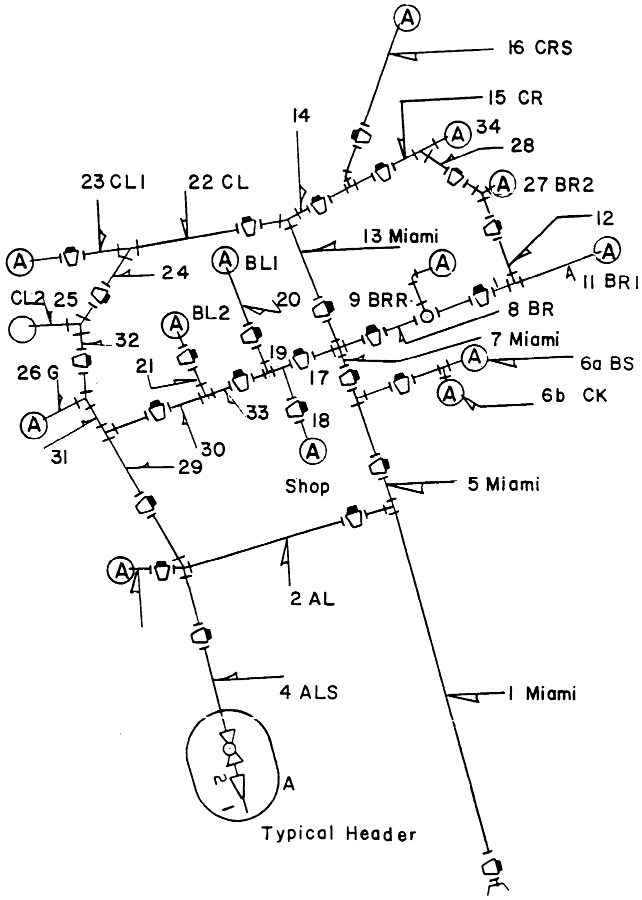

Fig. 12.5.3. Combination network layout.

Table 12.5.1. Heading Abbreviations

AJ	Absorbine Junior	BR2	B Right Second
AL	A Left	CK	CooperKendall Level
ALS	A Left Spur	CL	C Left
B	Bator Stope	CL1	C Left First
BL1	B Left First	CL2	C Left Second
BL2	B Left Second	CR	C Right
BRR	B Right Raise	CRS	C Right Spur
BR1	B Right First	G	Galena

Solution. Determine the actual flow rate through the drill by Eqs. 12.5.1 through 12.5.3:

$$V_c = \pi D^2/4\ S_c \tag{12.5.2}$$

$$acfm = V_c\,(Hz) \tag{12.5.2a}$$

Definitions for mathematical symbols are provided at the end of the chapter.

Cylinder volume by Eq. 12.5.2 is

$$V_c = [\pi\ (3.0\text{ in.}/12\text{ in./ft})^2/4]\ (2.5\text{ in.}/12\text{ in./ft})$$
$$= 0.0102\text{ ft}^3\text{/blow } (0.0002899\text{ m}^3\text{/blow})$$

Actual flow rate by Eq. 12.5.2a is

Table 12.5.2. Radial Branch Pipe Lengths and Initial Diameters

Branch No.	Branch Name	Diameter (in.)	Length (ft)
1	Miami Tunnel	6	750
2	AL	3	400
3	AJ	2	60
4	ALS	2	310
5	Miami Tunnel	6	205
6	B	2	165
7	Miami Tunnel	6	100
8	BRight	2	175
9	BRR	2	120, 60 vert.
10	BRight	2	180
11	BR1	2	170
12	BR2	2	190
13	Miami Tunnel	6	240
14	CRight	2	120
15	CR	2	205
16	CRS	2	300
17	BLeft	2	100
18	Shop	2	120
19	BLeft	2	33
20	BL1	2	210
21	BL2	2	240
22	CLeft	3	310
23	CL1	2	180
24	CLeft	3	150
25	CL2	2	130
26	G	2	250

Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm.

$$acfm = (0.0102\text{ ft}^3\text{/blow})\ (2400\text{ bpm})$$
$$= 24.48\ acfm\ (0.01156\text{ m}^3\text{/s})\text{ at }104.7\text{ psia }(721.9\text{ kPa})$$

From Eq. 12.5.1,

$$r = (90\text{ psig} + 14.7\text{ psia})/14.7\text{ psia} = 7.12$$

The required scfm by Eq. 12.5.3 is

$$scfm = acfm\ (r) \tag{12.5.3}$$

$$scfm = (24.48)(7.12) = 175\text{ scfm }(0.08259\text{ m}^3\text{/s})\text{ at }2400\text{ bpm}$$

The preceding example used a jackleg rated at 175 scfm (0.08259 m^3/s) and 100 psig (690 kPa), which varies less than 2% from the calculation. Ex. 12.5.1 ignores volumetric efficiency, which may become important for old equipment where the volumetric efficiency is reduced.

An overshot mucker has two motors that rarely operate simultaneously at their peak demand, so manufacturers recommend derating one motor by 50%. Therefore, a quantity of 305 scfm (0.1439 m^3/s) is appropriate for each mucker in this example.

Exact quantity requirements are difficult to determine because mining machines are operated dynamically, and actual consumption may vary considerably from manufacturers' data. For example, Gent (1986) tested four S83F drills and found that the average flow was 205 scfm (0.09675 m^3/s) at 88 psig (606.7 kPa) with a 59.9-scfm (0.02826-m^3/s) standard deviation. Peak values varied from 237 to 284 scfm (0.1119 to 0.1340 m^3/s). Measurements on 12B muckers resulted in an average 286 scfm (0.1350 m^3/s), with a 55 scfm (0.02596 m^3/s) standard deviation,

Table 12.5.3. Combination Network—Pipe Lengths and Initial Diameters

Branch No.	Branch Name	Diameter (in.)	Length (ft)
1	Miami Tunnel	6	750
2	AL	3	400
3	AJ	2	60
4	ALS	2	310
5	Miami Tunnel	6	205
6	B	2	165
7	Miami Tunnel	6	100
8	BRight	2	175
9	BRR	2	120
10	BRight	2	180
11	BR1	2	170
12	BRight	2	170
13	Miami Tunnel	6	240
14	CRight	2	120
15	CR	2	205
16	CRS	2	300
17	BLeft	2	100
18	Shop	2	120
19	BLeft	2	33
20	BL1	2	210
21	BL2	2	120
22	CLeft	3	310
23	CL1	2	180
24	CLeft	3	150
25	CL2	2	130
26	G	2	100
27	BR2	2	20
28	BRight	2	130
29	ALeft	2	280
30	BLeft	2	190
31	CLeft	2	70
32	CLeft	2	150
33	BLeft	2	120

Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm.

and peaks of 347 scfm (0.16377 m^3/s) when tramming and lifting and 180 scfm (0.08495 m^3/s) when lifting.

Quantity varies with time because the air consumed by equipment depends on the task being performed at a specific time. Dynamic simulation of duty cycles is one way to approach the problem. The other, used here, is to rely on manufacturers' data that represents an average during machine run time. However, if actual operating data is available, it is preferable to manufacturers' data. For example, King's and Gent's (1989) measurements on the S83F varied from the manufacturers' data because the equipment studied was old and worn.

Manufacturers generally specify pressure and quantity because minimum pressure must be provided if the advertised capabilities are to be achieved. Higher pressures are acceptable and will improve machine performance (especially in drill penetration rates) as long as the maximum pressure ratings are not exceeded.

12.5.4 QUANTITY ADJUSTMENT

Free air quantities are published at standard conditions; however, equipment rarely operates at standard conditions, and consumption rates must be adjusted.

Altitude adjustment can be accomplished in three ways: (1) Use the correction factors in Table 12.55, (2) calculate mass requirements and correct for changes in density, or (3) use the following volumetric formula:

$$F_v = P_s (P_o + P_e)/P_e (P_o + P_s) \qquad (12.5.4)$$

Note that acfm at the new atmospheric pressure = F_v (scfm).

The volumetric approach accounts for the metering effects of air tools. Air tools operating at steady state are essentially a meter passing a constant volume of compressed air acfm, regardless of the inlet pressure provided. Thus the same volume of compressed air must be provided by the compressor at any altitude.

When calculating pressure drop in pipes using CAGH tables, the altitude adjustment factor should not be used because the tables are based on scfm rather than acfm. Consequently, only the compressor is corrected for altitude. The pressure-loss segment of this chapter will demonstrate the procedure.

Adjustment, based on the mass of air, can be calculated in two ways. The inlet approach is based on a ratio of free air densities while the outlet approach uses compressed air density ratios. Both approaches assume that a constant mass of air must be supplied to the drill irrespective of altitude. Each technique has advantages. However, it is believed that the inlet approach is the easiest to understand and apply. For the inlet approach,

$$F_i = P_s/P_e \qquad (12.5.5)$$

and for the outlet approach,

$$F_o = (P_o + P_s)/(P_o + P_e) \qquad (12.5.6)$$

The most conservative answer results when using the ratio of inlet densities. Less conservative answers result when using volumetric adjustments and the least conservative using the ratio of outlet densities.

Example 12.5.2. Determine the altitude correction factor and the free air quantities required by the machines for the example mine.

Solution. The mine portal is at 7800 ft (2377 m) elevation, and the atmospheric pressure is 10.40 psia (71.7 kPa). If a drill requires 90 psig (620.5 kPa) at the header, the correction factor is

1. by volumetric approach,

$$F_v = 14.70(90 + 10.40)/10.40(90 + 14.70) = 1.35$$

2. by mass flow at the inlet,

$$F_i = 14.70/10.40 = 1.41$$

3. by mass flow at the outlet,

$$F_o = (90 + 14.7)/(90 + 10.40) = 1.04$$

The average of the results is 1.27, compared with a value of 1.29 in Table 12.5.4, an altitude correction listing.

For the example problem, the inlet density correction factor will be used. The free air quantity that must be supplied by the compressor for drills and muckers (refer to the problem statement) is given in Table 12.5.6.

In addition to the quantity used by the machine, it is necessary to account for leakage in the system. In mines, it is estimated that leakage ranges between 5 to 20% of total consumption. Leakage is very costly in terms of fuel and power costs. Estimated costs run as high as \$0.20/1000 acfm (\$0.42/m^3/s) (Loomis, 1980). There are many ways to account for leakage; two methods are presented here.

Table 12.5.4. Multipliers to Determine Air Consumption of Rock Drills at Altitudes and for Various Number of Drills

Altitude (ft)	Number of Drills														
	1	2	3	4	5	6	7	8	9	10	12	15	20	25	30
	Multiplier—Assuming 90 psi (620.5 kPa) Air Pressure														
0	1.000	2.0	3.0	4.0	5.0	6.0	6.3	7.2	8.1	9.0	10.8	12.0	16.0	20.0	22.5
1000	1.032	2.1	3.1	4.1	5.2	6.2	6.5	7.4	8.4	9.3	11.1	12.4	16.5	20.6	23.2
2000	1.065	2.1	3.2	4.3	5.3	6.4	6.7	7.7	8.6	9.6	11.5	12.8	17.0	21.3	24.0
3000	1.100	2.2	3.3	4.4	5.5	6.6	6.9	7.9	8.9	9.9	11.9	13.2	17.6	22.0	24.8
4000	1.136	2.3	3.4	4.5	5.7	6.8	7.1	8.2	9.2	10.2	12.3	13.7	18.2	22.7	25.6
5000	1.174	2.3	3.5	4.7	5.9	7.0	7.4	8.5	9.5	10.5	12.7	14.1	18.8	23.5	26.5
6000	1.213	2.4	3.6	4.9	6.1	7.3	7.6	8.7	9.8	10.9	13.0	14.6	19.4	24.2	27.4
7000	1.255	2.5	3.8	5.0	6.3	7.5	7.9	9.0	10.2	11.3	13.6	15.1	20.0	25.1	28.2
8000	1.298	2.6	3.9	5.2	6.4	7.8	8.2	9.3	10.5	11.7	14.0	15.6	20.8	26.0	29.2
9000	1.343	2.7	4.0	5.4	6.7	8.1	8.5	9.7	10.9	12.1	14.5	16.1	21.5	26.9	30.3
10000	1.391	2.8	4.2	5.6	7.0	8.3	8.8	10.0	11.3	12.5	15.0	16.7	22.3	27.8	31.3
12500	1.520	3.0	4.6	6.1	7.6	9.1	9.6	10.9	12.3	13.7	16.4	18.2	24.3	30.4	34.2
15000	1.665	3.3	5.0	6.7	8.3	10.0	10.5	11.9	13.5	15.0	18.0	20.0	26.6	33.3	37.5

Source: Rollins, 1973. Conversion factor: 1 ft = 0.3048 m.

Table 12.5.5. Atmospheric Pressure and Barometer Readings at Different Altitudes

Altitude above sea level, ft	Atmospheric Pressure, psi	Barometer Reading, in. Hg	Altitude above sea level, ft	Atmospheric Pressure, psi	Barometer Reading, in. Hg
0	14.69	29.92	7500	11.12	22.65
500	14.42	29.38	8000	10.91	22.22
1000	14.16	28.86	8500	10.70	21.80
1500	13.91	28.33	9000	10.50	21.32
2000	13.66	27.82	9500	10.30	20.98
2500	13.41	27.31	10000	10.10	20.58
3000	13.16	26.81	10500	9.90	20.18
3500	12.92	26.32	11000	9.71	19.75
4000	12.68	25.84	11500	9.52	19.40
4500	12.45	25.36	12000	9.34	19.03
5000	12.22	24.89	12500	9.15	18.65
5500	11.99	24.43	13000	8.97	18.29
6000	11.77	23.98	13500	8.80	17.93
6500	11.55	23.53	14000	8.62	17.57
7000	11.33	23.09	14500	8.45	17.22
			15000	8.28	16.88

Source: Rollins, 1973.
Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm, 1 psi = 6.895 kPa.

Table 12.5.6. Free Air Equivalent Quantities at 7800-Ft Elevation

Drill quantity (83)	= 1.41 × 175	= 247 acfm at 10.40 psia
Drill quantity (53)	= 1.41 × 123	= 173 acfm at 10.40 psia
Mucker quantity	= 1.41 × (1.5 × 210)	= 444 acfm at 10.40 psia

Conversion factors: 1 ft = 0.3048 m, 1 acfm = 0.0004719 m^3/s, 1 psi = 6.895 kPa.

In a simplified approach, leakage factors can be applied at a small number of discrete locations, for example, at the operating machines only. This is justified because the majority of leaks occur closest to the working areas where frequent damage, improper installation, and little preventative maintenance exist. This method is conservative but adequately models the real environment.

A more comprehensive method is to decide where all probable and possible leakage points exist in the system and assign leakage quantities. This is a more accurate and time-consuming method. The computer program reduces the computation time for this technique.

Very few measurements have been made to quantify leakage in mines. A simple test to determine overall leakage is to start a compressor and record the operating time to maintain minimum pressure. This type of test has some disadvantages. It is not possible to determine the quantities at each leakage point or determine where compressed air is being used for ventilation. Compressed air used for ventilation and cooling should be considered a load. Example 12.5.3 adjusts the quantities at the example mine for leakage.

Example 12.5.3. Determine leakage quantities at the example mine.

Solution. The design at the mine under consideration assumes poor installation and maintenance. For these conditions, 10% leakage (L_f = 1.1) will be assumed for two different points at each heading: (1) the connection between the pipe and hose and (2) the connection between the machine and hose.

Table 12.5.7. Design Quantities (scfm) at Sea Level

Drills:		
(83) Quantity at hose outlet	= (175) (1.1)	= 193 scfm at 90 psig
(83) Quantity at hose inlet	= (193) (1.1)	= 212 scfm at 90 psig
(53) Quantity at hose outlet	= (123) (1.1)	= 135 scfm at 90 psig
(53) Quantity at hose inlet	= (135) (1.1)	= 149 scfm at 90 psig
Mucker:		
Quantity at hose outlet	= (1.5) (210) (1.1)	= 347 scfm at 90 psig
Quantity at hose inlet	= 347 × 1.1	= 381 scfm at 90 psig

Table 12.5.8. Design Quantities at 7800-Ft Elevation

Drills:		
(83) Quantity at hose outlet	= (247) (1.1)	= 271 acfm at 10.4 psia
(83) Quantity at hose inlet	= (271) (1.1)	= 299 acfm at 10.4 psia
(53) Quantity at hose outlet	= (173) (1.1)	= 191 acfm at 10.4 psia
(53) Quantity at hose inlet	= (191) (1.1)	= 210 acfm at 10.4 psia
Mucker:		
Quantity at hose outlet	= (444) (1.1)	= 489 acfm at 10.40 psia
Quantity at hose inlet	= (489) (1.1)	= 537 acfm at 10.40 psia

Conversion factors: 1 psi = 6.895 kPa, 1 acfm = 0.0004719 m^3/s.

Tables 12.5.6 and 12.5.7 display design quantities at the hose inlet and outlet for the drill and mucker, respectively. Table 12.5.7 shows design quantities for use with the CAGH tables. Table 12.5.8 shows quantities that were corrected for altitude and used for the volumetric pressure loss calculations.

Leakage assumed at any location requires certain decisions that are best left to experience or actual field measurements. The guidelines that are presented here should be replaced by other methods where appropriate.

12.5.5 EQUIVALENT LENGTHS

Before analyzing radial or combination networks, valves and fittings must be placed appropriately. Fittings placement is relatively obvious, but valve placement and selection is not and requires careful consideration for maintenance and safety. For mining applications, quick shut-off valves such as butterfly or plug valves are necessary. Globe valves are usually avoided due to high frictional loss and purchase cost. Gate valves should be avoided because they require many revolutions to stop airflow.

Valve and fitting placement is a trade-off between (1) potential maintenance and lost production and (2) initial cost. At a minimum, valves must be placed at the compressor outlet and at each working face. However, if this minimum cost scheme were followed, every broken line or an extension to the existing network would shut down the entire mine. Lost production easily justifies the cost of more valves; therefore, valves should be placed at the start of each heading, at the face, and at major feeder intersections, similar to those shown in Fig. 12.5.2. Other shut-off valves may be required at particular sites.

Replacing the valve or fitting with an equivalent pipe length is an accepted and convenient method for calculating the additional resistance. This method determines the length of straight pipe that will produce the equivalent pressure loss through the specified valve or fitting. Fig. 12.5.4 shows victaulic piping symbols.

Before equivalent length can be determined, pipe diameters must be known or estimated. Therefore, some iteration occurs within the design process. At this point, pipe-size estimates must be made, and the designer must later calculate optimal diameters. Consequently, the designer may not wish to determine equivalent lengths until approximate pipe diameter are analyzed throughout the system. Diameters assumed for the example mine are given in Tables 12.5.2 and 12.5.3.

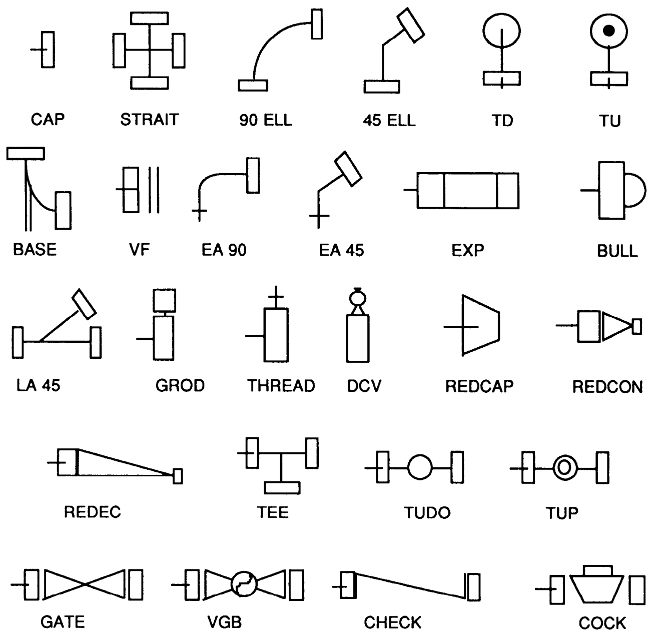

Fig. 12.5.4. Victaulic piping symbols

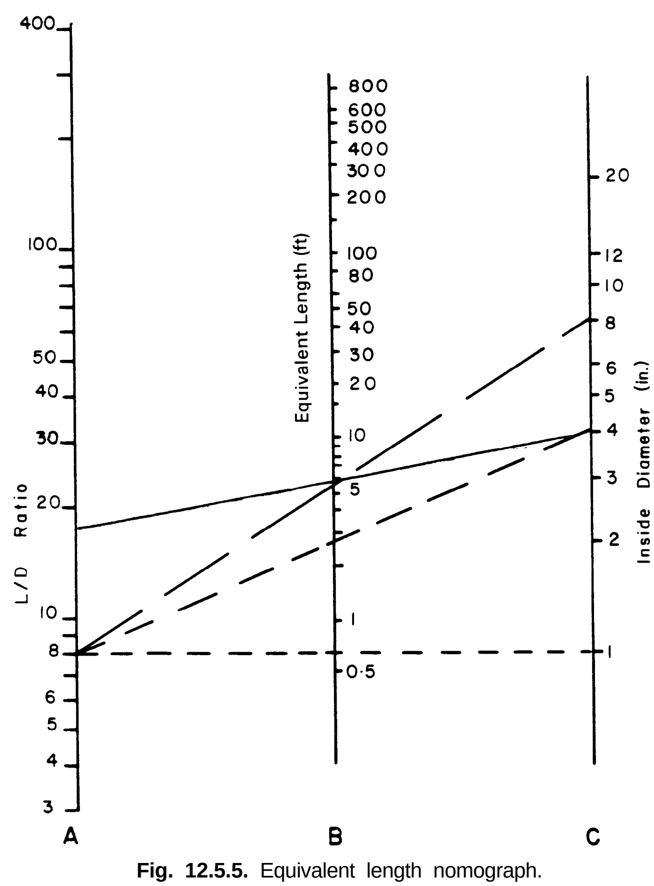

Fig. 12.5.5. Equivalent length nomograph.

There are many tables and charts available on equivalent length. However, most of these are taken from Crane's Technical Paper 410 (Anon., 1978). Figs. 12.5.5 and 12.5.6 are based on

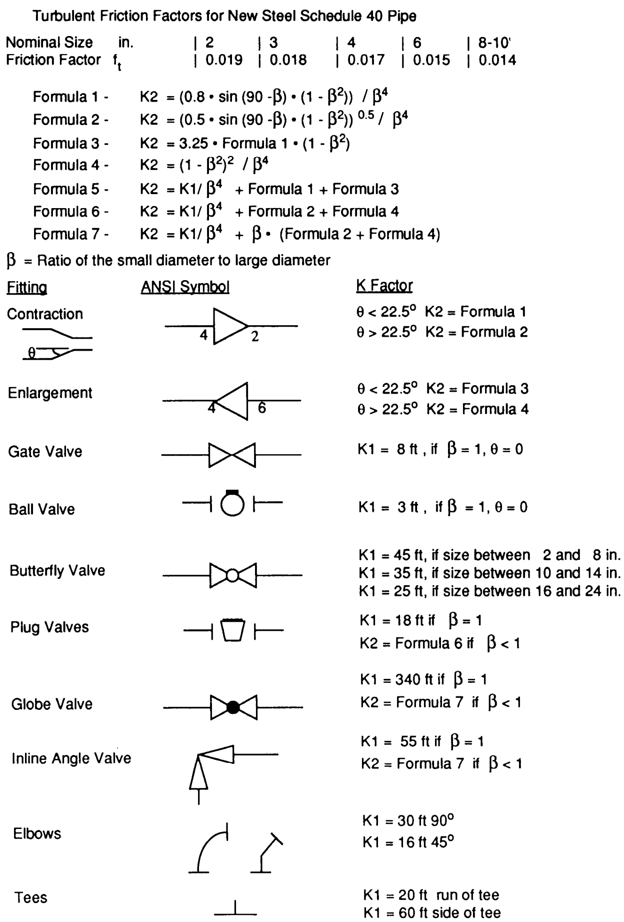

Fig. 12.5.6. Equivalent length formulas (Anon., 1978).

this document and are used in equivalent length calculations in Ex. 12.5.4.

Example 12.5.4. Determine the equivalent length of a 4-in. (101.6-mm) plug-cock valve.

Solution. Determining equivalent lengths using the charts from Technical Paper 410 is a relatively simple and very versatile procedure. The first step is to determine the L/D ratio from Fig. 12.5.6. The L/D ratio is the K_e number in ft that appears before the turbulent friction factor b. For instance, the L/D ratio for a straightway plug-cock valve is 18. For some β values, K_e must be calculated from the Fig. 12.5.6 formulas:

$$L_e = D\,(L/D) = (4.026/12)\,(18) = 6 \text{ ft } (1.829 \text{ m}) \qquad (12.5.7)$$

Equivalent length can be determined from Fig. 12.5.5 if L/D is known. Locate the L/D ratio on the ordinate labeled A. Next locate the internal pipe diameter on ordinate C, and draw a line between these two points. The point where the line intersects ordinate B determines the equivalent length. For the 4-in. (101.6-mm) plug-cock valve, the equivalent length is 6 ft (1.829 m).

Figs. 12.5.2 and 12.5.3 show valves and fittings for the example mine problem. Equivalent length was added to the pipe lengths from Figs. 12.5.2 and 12.5.3 to obtain total lengths shown in Table 12.5.9.

Table 12.5.9 tabularizes the fittings for each branch, the physical branch length, and the total calculation length. Equivalent length for valves and fitting should be added to the downstream branch, as shown in Table 12.5.9, because this is where the additional loss will occur.

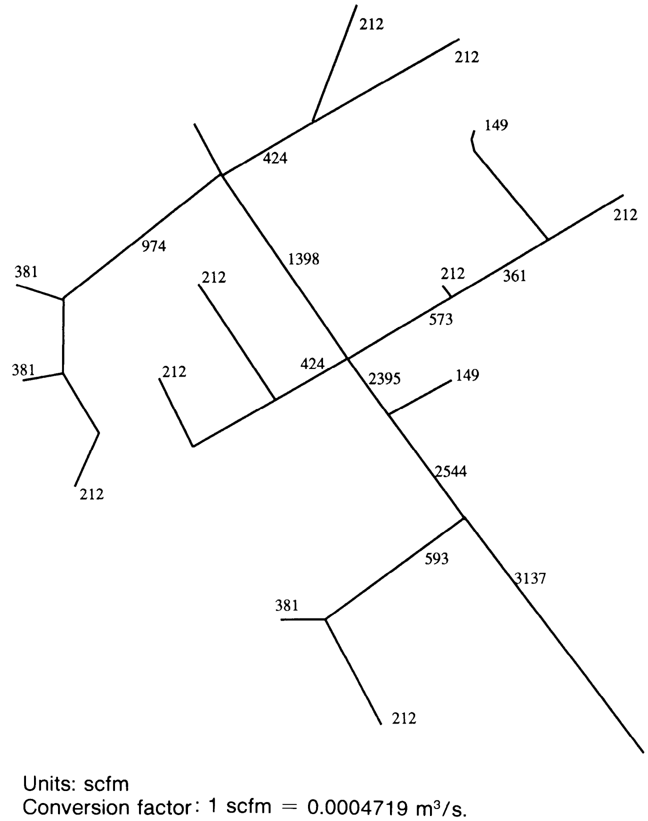

Fig. 12.5.7. Flow rates for the radial network (CAGH method).

12.5.6 MACHINE ALLOCATION

Deciding equipment configuration (machine placement in working headings) is one of the most difficult design steps because limitations are being imposed. One procedure is to design for the worst condition; another is to design for the most likely conditions. Both have advantages and disadvantages.

The worst condition may only occur for a limited time or during certain shifts. Since short-duration low-pressure periods may be tolerable, worst-condition design is not cost effective, or the most efficient utilization of resources. Designing a system for average load, with no consideration for peak demand, may limit productivity.

As with any design, a balance must be maintained between the two and this balance is a site-specific decision. The best method is to calculate the requirements for several cases using the computer program and compare costs and limitations.

When placing equipment at specific headings, it is difficult to predict the worst-case condition. However, equipment arrangements that increase pressure drop and simulate worst-case conditions can be achieved by placing equipment with the highest consumption rates at headings on the largest resistance paths.

A method of equipment placement for worst-case simulation is to divide the branch length by its proposed diameter and sum the diameter-weighted lengths to each working face, as shown in Table 12.5.10. Next, place equipment with the highest pressure

Table 12.5.9. Actual, Equivalent, and Total Lengths for Each Branch

Branch No.	Length (ft)	Diameter in.	Fittings	Equivalent length (ft)	Total length (ft)
1	750	6.0	1pv;	9.0	759.0
2	400	3.0	1st, 1pv;	19.5	419.5
3	60	2.0	1rt, 14e, 1bf;	12.5	72.5
4	310	2.0	1st, 1pv, 1bf;	21.0	331.0
5	205	6.0	1rt, 1pv;	19.0	224.0
7	100	6.0	1rt, 1pv;	19.0	119.0
6	165	2.0	1st, 1rt, 1pv, 1bf;	24.5	189.5
17	100	2.0	1st, 1rt, 1pv;	13.0	113.0
19	33	2.0		0.0	33.0
20	220	2.0	1st,1pv,1bv;	14.0	234.0
21	242	2.0	1rt, 1pv, 19e, 1bf;	17.5	259.5
8	175	3.0	1st, 1pv;	13.0	188.0
9	120	2.0	1st, 19e, 1bf;	23.0	143.0
10	167	2.0	1rt, 1pv;	19.0	186.5
12	190	2.0	1st, 1pv, 19e, 1bf;	26.0	216.0
11	170	2.0	1rt, 1pv, 1bf;	14.5	184.5
13	260	6.0	1rt, 1pv;	19.0	279.0
22	310	3.0	1st, 1pv;	19.5	329.5
14	120	2.0	1st;	10.0	130.0
16	300	2.0	1st, 14e, 1pv, 1bf;	24.0	324.0
15	205	2.0	1rt, 1pv, 1bf;	14.5	219.5
23	180	2.0	1rt, 1pv, 1bf;	14.5	184.5
24	150	3.0	1st, 14e, 1pv;	24.0	174.0
25	130	2.0	1st, 1bf;	18.0	148.0
26	250	2.0	1st, 1pv, 14e, 1bf;	24.0	274.0

pv = plug valve st = side outlet of tee rt = run of tee
e = ell bf = butterfly valve
Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm.

Table 12.5.10. Diameter-Weighted Distances

Heading	Path	Diameter-Weighted Distances (ft/in.)	
ALS	124	759/6 + 410/3 + 331/2	= 429
AJ	1–2–3	759/6 + 410/3 + 72.5/2	= 299
CK	1–5–6A	(759 + 224)/6 + 184/2	= 256
BS	1–5–6B	(759 + 224)/6 + 189.5/2	= 258
BR1	1–5–7–8–10–11	(759 + 224 + 106.5)/6 + 188/3 + (186.5 + 184.5)/2	= 430
BR2	1–5–7–8–10–12	(759 + 224 + 106.5)/6 + 188/3 + (186.5 + 216)/2	= 446
BBR	1–5–7–8–9	(759 + 224 + 106.5)/6 + 188/3 + 143/2	= 316
BL1	1–5–7–17–19–20	(759 + 224 + 106.5)/6 + (113 + 33 + 234)/2	= 371
BL2	1–5–7–17–19–21	(759 + 224 + 106.5)/6 + (113 + 33 + 259.5)/2	= 384
SHOP	1–5–7–17–18	(759 + 224 + 106.5)/6 + (113 + 141)/2	= 309
CRS	1–5–7–13–14–16	(759 + 224 + 106.5 + 279)/6 + (130 + 324)/2	= 455
CR	1–5–7–13–14–15	(759 + 224 + 106.5 + 279)/6 + (130 + 219.5)/2	= 403
CL1	1–5–7–13–22–23	(759 + 224 + 106.5 + 179)/6 + (310/3 + 194.5/2)	= 428
CL2	1–5–7–13–22–24–25	(759 + 224 + 106.5 + 279)/6 + (310 + 174)/2 + 148/2	= 463
G	1–5–7–13–22–24–26	(759 + 224 + 106.5 + 279)/6 + (310 + 174)/2 + 274/2	= 526

Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm.

requirements on the longest weighted paths, and concentrate the highest consumption equipment on the largest diameter-weighted path.

Example 12.5.5. Allocate equipment at the example mine using the diameter-weighted-distance (dwd) technique.

Solution. Table 12.5.10 lists the path from each heading to the portal using the branch numbers from Fig. 12.5.2. Pipe diameter and branch lengths from Table 12.5.9 were used to calculate the dwd.

Table 12.5.10 can be interpreted by using heading CK as an example; path 1-5-6A was divided into two segments because two different pipe sizes were assumed in the path. Branches 1 and 5 are 759 and 224 ft (231.3 and 68.28 m), respectively, and are 6 in. (152.4 mm) in diameter. The third branch, 6, has 184 ft (56.08 m) of 2-in. (50.8-mm) diameter pipe.

12.5.7 BALANCING FLOWS AT RADIAL NODES

When balancing mass or volumetric flow rates at nodes, radial networks must be treated differently from combination networks. However, regardless of the system being designed, mass flowing into a node must equal mass flowing out of the node.

With a compressible medium such as air, the only correct way to balance at nodes is to use mass rather than volume as the

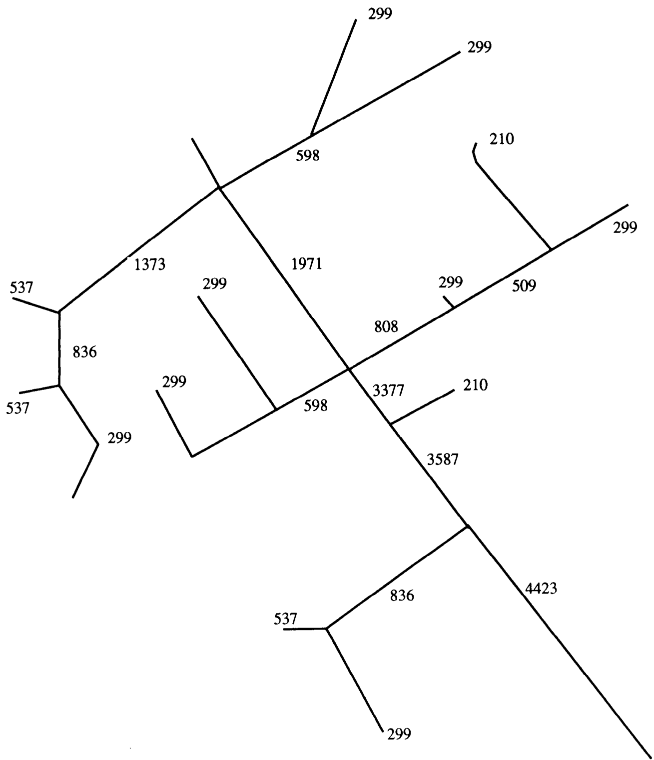

Fig. 12.5.8. Flow rates for the radial network (volumetric and mass flow).

basic flow unit. Only if the fluid is assumed incompressible will the quantities flowing into and out of nodes sum exactly to zero. However, quantities change in mine compressed air systems because decreased pressure caused by friction and shock loss allows air expansion within the system. For many situations, the loss of accuracy caused by the incompressible assumption is negligible, and inlet volume approximately equals the outlet volume.

Balancing a radial network is relatively simple since flow directions are defined by the physical layout, and splitting is determined by equipment placement. Air volumes by branch are determined by balancing nodes from the machines to the compressor.

Flow rate determination in loop networks is considerably more difficult. The Hardy Cross method or nodal iterative techniques must be used to determine splitting at nodes. Hardy Cross assumes continuity of mass and iterates until continuity of energy is obtained. Nodal techniques assume continuity of energy and iterates until continuity of mass is obtained.

Fig. 12.5.7 shows the flow rates for the example problem in a radial configuration.

12.5.8 PRESSURE LOSSES

Many methods can be used to calculate pressure drop, but only three are discussed here: (1) CAGH table formula, (2) the Darcy-Wiesbach equation, and (3) the mass flow equation.

Tables are generally used for manual calculations. The Darcy-Weisbach equation is most commonly used in existing fluid flow computer programs. The computer program, COMPAIR (Gent, 1986), referenced herein employs either the mass flow or the traditional volume approach.

12.5.8.1 CAGH Table Equation

The CAGH tables were generated from the following equation:

$$\Delta p = 0.1025\ LQ^2/3600\ r\ D^{5.31} \tag{12.5.8}$$

Example 12.5.6. A quantity of 560 scfm (0.2643 m^3/s) is flowing in a 375-ft (114.3-m), 4-in. (101.6-mm), Schedule 40 pipe with a 97-psig (668.8-kPa) inlet pressure. The pipe is located at sea level. Using the CAGH table formula, determine the pressure loss.

Solution. Use Eq. 12.5.8. The compression ratio is calculated using Eq. 12.5.1 as shown here:

$$r = (97 + 14.7)/14.7 = 7.6$$

and the resulting pressure loss is

$$\begin{aligned}\Delta p &= 0.1025(375)(560)^2/3600(7.6)(4.026)^{5.31}\\ &= 0.27 \text{ psig } (1.862 \text{ kPa})\end{aligned}$$

A table of standard pipe weights appears in Table 12.5.11 for reference.

12.5.8.2 Darcy-Weisbach Equation

Calculating pressure loss with the Darcy-Weisbach equation allows selection of absolute roughness and viscosity, thus allowing friction factors to be calculated for any flow regime.

The Darcy-Weisbach equation for pressure loss is

$$\Delta p = f\ p\ L\ V^2/(288\ D\ g_c) \tag{12.5.9}$$

Example 12.5.7. A quantity of 560 scfm (0.2643 m^3/s) is flowing in a 375-ft (114.3-m), 4-in. (101.6-mm), Schedule 40 pipe at 97 psig (668.8 kPa). The pipe is at sea level, and the internal temperature is 60°F (15.6°C). Determine the pressure drop using Eq. 12.5.9.

Solution. Determining the Moody friction factor is the most complicated part. The first step is to calculate the fluid velocity:

$$\begin{aligned}acfm \text{ at 97 psig} &= scfm/r = 560/7.60\\ &= 73.70\ acfm\ (0.03478 \text{ m}^3/\text{s})\\ &\quad \text{at 111.7 psia (770.1 kPa)}\end{aligned}$$

The cross-sectional area A of a 4-in. Schedule 40 pipe is 12.73 in.^{2} (8213 mm^2). The air velocity is:

$$\begin{aligned}V = Q/A &= (73.70)(144 \text{ in.}^2/\text{ft}^2/(60)(12.73 \text{ in.}^2)\\ &= 13.89 \text{ fps } (4.234 \text{ m/s})\end{aligned} \tag{12.5.10}$$

Reynolds number can be calculated from the following relation:

$$R = 10.325\ D\ (V/\nu) \tag{12.5.11}$$

The internal pipe diameter is 4.026 in. (102.3 mm), the veloc-

Table 12.5.11. Standard Weight of Welded and Seamless Steel Steam, Air, Gas, and Water Pipe

Nominal pipe size (in.)	Diameter (in.)		Thickness (in.)	Circumference (in.)		Transverse Area (in.^{2})			Pipe Length (ft per ft^2)		Pipe Length containing 1 ft^3
	External	Internal		External	Internal	External	Internal	Metal	External surface	Internal surface	
2.000	2.375	2.067	0.154	7.461	6.494	4.430	3.355	1.075	1.608	1.847	42.913
2.500	2.875	2.469	0.203	9.032	7.757	6.492	4.788	1.704	1.328	1.547	30.077
3.000	3.500	3.068	0.216	10.996	9.638	9.621	7.393	2.288	1.091	1.245	19.479
3.500	4.000	3.548	0.226	12.566	11.146	12.566	9.886	2.680	0.954	1.076	14.565
4.000	4.500	4.026	0.237	14.137	12.648	15.904	12.730	3.174	0.848	0.948	11.312
5.000	5.563	5.047	0.258	17.477	15.856	23.306	20.066	4.300	0.686	0.756	7.198
6.000	6.625	6.065	0.280	20.813	19.054	34.472	28.891	5.581	0.576	0.629	4.984
8.000	8.625	8.071	0.277	27.096	25.356	58.426	51.161	7.265	0.443	0.473	2.815
8.000	9.625	7.981	0.322	27.096	25.073	58.426	50.027	8.399	0.443	0.478	2.878
10.000	10.750	10.020	0.365	33.772	31.479	90.763	78.855	11.908	0.355	0.381	1.826
12.000	12.750	12.000	0.375	40.055	37.699	127.676	113.097	14.579	0.299	0.318	1.273
14.00 od	14.000	13.250	0.375	43.982	41.626	153.938	137.886	16.052	0.272	0.288	1.044
15.00 od	15.000	14.250	0.375	47.124	44.768	176.715	159.485	17.230	0.254	0.268	0.903
16.00 od	16.000	15.250	0.375	50.265	47.909	201.062	182.654	18.408	0.238	0.250	0.788
17.00 od	17.000	16.214	0.393	53.407	50.938	226.980	206.476	20.504	0.224	0.235	0.697
18.00 od	18.000	17.182	0.409	56.549	53.979	254.469	231.866	22.603	0.212	0.222	0.621
20.00 od	20.000	19.182	0.409	62.832	60.262	314.159	288.986	25.173	0.191	0.199	0.498

Source: ASTM Standard Specifications A53.33.
Conversion factors: 1 in. = 25.4 mm, 1 in.^{2} = 645.16 mm^2, 1 ft = 0.3048 m, 1 ft^3 = 0.02831685 m^3.

ity is 13.89 fps (4.234 m/s), the absolute viscosity is 0.018, and the fluid density r can be calculated as follows:

$$\rho = (144\ P_f)/(R_c\ T_o) \qquad (12.5.12)$$

For this example,

$$\rho = 144(97 + 14.70)/(53.34)(459.69 + 60)$$
$$= 0.58025 \text{ lbm/ft}^3 \ (9.295 \text{ kg/m}^3)$$

Relative viscosity is

$$\nu = 0.018/0.58025 = .031021 \text{ cp ft}^3/\text{lbm}$$

Reynolds number is

$$R = (10.325)(12 \text{ in./ft})(4.026 \text{ in.})(13.89 \text{ fps})/$$
$$0.031021 \text{ cp ft}^3/\text{lbm} = 223{,}400$$

A Reynolds number of 223,400 indicates a flow regime in the transitional region, the most common region for mine compressed air applications. If the absolute roughness is 0.00015, the relative roughness is

$$\epsilon/D = 0.00015 \text{ ft}/(4.026 \text{ in.}/12 \text{ in./ft}) = 0.0004471$$

Based on this relative roughness, the Moody friction factor is 0.019. The calculated pressure loss is

$$\Delta p = 0.019(0.58025)(375)(13.89)^2/288(4.026/12.0)(32.174)$$
$$= 0.26 \text{ psig } (1.79 \text{ kPa})$$

12.5.8.3 Mass Flow

Mass flow assumptions are similar to the other methods. An isothermal process is usually assumed, but this can be changed to isentropic or adiabatic. Steady state is assumed for the pressure loss calculation. The pressure loss can be calculated using the following expression (Anon., 1980):

$$\Delta p = \frac{P_1 + \text{m}^2}{\left[\dfrac{\rho\ 144\ g_c\ A^2}{fL/D + 2\log(P_1/P_2)}\right](P_1 + P_2)} \qquad (12.5.13)$$

Mass flow calculations are more complex and time consuming because the physics is more complicated. The mass flow approach readily lends itself to a nodal analysis because pressures at both nodes can be estimated; then the corresponding flow rates can be calculated. A full mass-flow nodal analysis discussion is given by Gent (1986).

Example 12.5.8. A quantity of 560 scfm (0.2643 m^3/s) is flowing in a 375-ft (114.3-m), 4-in. (101.6-mm), Schedule 40 pipe. The inlet pressure is 97 psig (668.8 kPa). Determine the pressure drop if the pipe is at sea level and the internal temperature is 60°F (15.6°C).

Solution. As with the Darcy-Weisbach equation, the most difficult part of this calculation is determining the friction factor. Since Exs. 12.5.7 and 12.5.8 have identical stated conditions, the friction factor and internal densities determined in Ex. 12.5.7 are used. The next step is to determine the mass flow rate.

From Ex. 12.5.7, the internal flow rate is 73.70 acfm at 97 psig, and the internal density is 0.58025 lbm/ft^3. Mass flow rate is

$$m = (73.70 \text{ acfm}/60 \text{ sec/min})\ (0.58025 \text{ lbm/ft}^3)$$
$$= 0.71274 \text{ lbm/sec } (0.3233 \text{ kg/sec}) \qquad (12.5.14)$$

The mass flow equation as presented is an iterative equation. If fluid acceleration is ignored or the inlet to outlet pressure ratio is close to one, then 2 log(P_1/P_2) can be neglected without an appreciable loss of accuracy. Substituting ΔP for $P_1 - P_2$, and combining similar terms (negligible acceleration), a closed-form equation results:

$$P_2^2 = P_1^2 + \left(\frac{m^2 P_1 f L}{144\, g_c\, D\, \rho\, A^2}\right) \qquad (12.5.15)$$

The area of a 4-in. (101.6-mm) Schedule 40 pipe is 0.088405 ft^2 (0.007815 m^2), and the internal diameter is 0.3355 ft (0.1023 m), the solution for P_2 is

$$P_2^2 = (111.7)^2 + [(0.7125)^2(111.7)(0.019)(375)/(144)(32.17)$$
$$(0.3355)(0.5803)(0.0884)^2] = 12{,}419.54$$

$$P_2 = 111.44 \text{ psia } (768.4 \text{ kPa})$$

$$\Delta p = P_1 - P_2 = 111.7 - 111.44 = 0.26 \text{ psig } (1.793 \text{ kPa})$$

Notice there is no difference between answers by the mass flow and volumetric methods; this, however, is not always the case. When the flow rates increase and the distances become longer, the solutions of the two methods will diverge.

All analyses presented are for new clean pipe. Pipe will not remain in this condition forever. The pipe will rust, become coated with oil, collect dirt and scale, and become bent and ovaloidal in cross section. To account for this, corrections must be made to the friction factor.

A 1% decrease in diameter will increase the pressure loss by 5%. For a 2-in. (50.8-mm) Schedule 40 pipe, this equals less than 1/64th in. (0.3969 mm) of buildup on the interior walls. A 5% decrease in diameter increases the pressure drop by 29%. Crane's Technical Paper 410 (Anon., 1980, p. 17) reports that aging effects increase the roughness factor 20% in as little as three years with only moderate use. It is recommended that the pressure drops be increased by 20 to 50% for fouling.

12.5.8.3 Altitude Pressure Differential

Compressed air distribution systems can be three-dimensional. None of the calculations presented thus far have accounted for changes in altitude. Ex. 12.5.9 illustrates corrections for altitude.

Example 12.5.9. Calculate the altitude pressure differential in a 500-ft (152.4-m) raise with a base elevation of 2500 feet (762 m).

Solution. A vertical air column (vertical pipe) will create a static pressure differential that is proportional to the change in elevation. This can be calculated through the use of the following formula:

$$\Delta P_h = \rho\, H/144 \qquad (12.5.16)$$

At l00-psig (690-kPa) average pressure, the altitude pressure differential is:

$$\Delta P_h = (0.5891)(500)/144 = 2.05 \text{ psia } (14.13 \text{ kPa})$$

As an alternative, altitude correction can be approximated as 4.1 psig/1000 ft (9.3 kPa/1000 m) of elevation change.

12.5.8.5 Pressure Loss in Hoses

Hose pressure loss can be found in the tables provided in CAGH. The tables assume smooth hose and use standard fluid flow theory or a two-phase flow regime.

Utilizing the Darcy-Weisbach method, pressure drop calculations for a hose supplying a jackleg drill are given in Ex. 12.5.10.

Example 12.5.10. Calculate the pressure drop for a l-in. (25.4-mm) diameter hose supplying 272 acfm (0.1284 m^3/s) at 10.4 psia (71.70 kPa) to a jackleg drill. The drill operates at 115.7 psig (797.7 kPa). The hose does not have an in-line oiler.

Solution. The compression ratio is

$$r = (115.7 + 10.4)/10.4 = 12.13$$

The inside area of a l-in. (25.4-mm) hose is

$$A = \pi\, (1.00/24)^2 = 0.005454 \text{ ft}^2\ (0.0005067 \text{ m}^2)$$

Internal velocity is

$$V = 272/(12.130)(0.005454)(60) = 68.55 \text{ fps } (20.89 \text{ m/s})$$

Internal density is

$$\rho = 144\, (115.7 + 10.4)/(53.34)(519.69)$$
$$= 0.6551 \text{ lbm/ft}^3\ (10.49 \text{ kg/m}^3)$$

Reynolds number is

$$R = 123.9\, (68.55)(1.00)/(0.018/0.6551) = 309{,}100$$

Relative roughness is

$$\epsilon/D = 0.000001 \text{ (smooth hose)}$$

The friction factor f is 0.0145.

Coupling equivalent length should be accounted for when calculating hose pressure drop. Assuming the couplings can be modeled as a sharp entrance or exit, the equivalent length is 6 ft (1.829 m). The pressure drop for 40 ft (12.19 m) of l-in. (25.4-mm) hose is:

$$\Delta p = (0.0145)\, (0.6551)(46)(68.55)^2/(288)(32.174)(1.0/24)$$
$$= 2.67 \text{ psig } (18.41 \text{ kPa})$$

Hose linings deteriorate with age and abuse. Used-hose friction may be 1.2 to 1.5 times greater than new hose. Assuming an average 35% deterioration,

$$\Delta p = (1.35)(2.67) = 3.59 \text{ psig } (24.75 \text{ kPa})$$

The pressure-drop calculation for hose containing an in-line oiler is most accurately calculated by the use of homogenous two-phase flow theory (Gent, 1986). The pressure loss of a two-phase fluid is calculated by mass weighting the viscosities and applying mass flow equations.

Publications such as Rollins (1973), Loomis (1986), and Anon. (1983) have reference tables for pipe and hose pressure losses. Many of these tables must be interpolated or extrapolated unless gross estimates are desired. The two-phase flow technique or careful interpolation provides more accurate results and can be obtained in a comparable amount of time using calculators or computers.

12.5.9 COMBINING NETWORK FLOWS AND LOSSES

To begin the network solution, pressures must be assumed at both inlet and outlet ends of the system. One way to determine

Table 12.5.12. Pressure and Pressure Losses (COMPAIR & CAGH Method)

Node				COMPAIR (psig)				CAGH (psig)				
In	Out	Branch No.	Flow (scfm)	In	Out	Drop	Cum Drop	In	Out	Drop	Cum Drop	Regulation
MO	ALMI	1	3137	106.1	103.8	2.29	2.29	105.6	103.4	2.15	2.15	1.85
ALMI	ALAJ	2	593	103.8	102.2	1.65	3.94	103.4	101.9	1.53	3.68	3.17
ALAJ	AJ	3	381	102.2	101.2	0.91	4.85	101.9	101.2	0.70	4.38	3.78
ALAJ	AL	4	212	102.2	100.8	1.38	6.23	101.9	100.5	1.34	5.82	5.02
ALMI	MIBS	5	2544	103.8	103.3	0.46	2.75	103.4	103.0	0.42	2.67	2.30
MIBS	BLMI	7	2395	103.3	103.1	0.22	2.97	103.0	102.8	0.18	2.85	2.46
MIBS	BS	6	149	103.3	102.9	0.41	3.16	103.0	102.6	0.36	3.03	2.61
BLMI	BLS	17	424	103.1	101.4	1.73	4.70	102.8	101.2	1.63	4.48	3.86
BLS	BLH1	19	424	101.4	100.9	0.50	5.20	101.2	100.7	0.48	4.96	4.28
BLH1	BL1	20	212	100.9	99.9	0.98	6.18	100.7	99.7	0.98	5.94	5.12
BLH1	BL2	21	212	100.9	99.8	1.13	6.33	100.7	99.6	1.09	6.05	5.22
BLMI	BRRC	8	573	103.1	102.4	0.72	3.69	102.8	102.2	0.62	3.47	2.99
BRRC	BRR	9	212	102.4	101.5	0.86	4.55	102.2	101.3	0.85	4.32	3.72
BRRC	BRSI	10	361	102.4	100.3	2.06	5.75	102.2	100.2	2.01	5.48	4.72
BRSI	BR2	12	149	100.3	99.9	0.48	6.23	100.2	99.8	0.43	5.92	5.10
BRSI	BR1	11	212	100.3	99.6	0.78	6.53	100.2	99.4	0.77	6.25	5.39
BLMI	CLMI	13	1398	103.1	103.0	0.18	3.15	102.8	102.6	0.16	3.01	2.59
CLMI	CL1S	22	974	103.0	99.6	3.37	6.52	102.6	99.5	3.14	6.15	5.30
CLMI	CR1S	14	424	103.0	101.0	1.94	5.09	102.6	100.5	1.88	4.89	4.22
CR1S	CRS	16	212	101.0	99.7	1.36	6.45	100.5	99.1	1.36	6.25	5.39
CR1S	CR	15	212	101.0	100.1	0.92	5.99	100.5	99.6	0.92	5.81	5.01
CL1S	CL1	23	381	99.6	97.1	2.44	8.96	99.5	97.7	1.82	7.97	6.87
CL1S	CL2S	24	593	99.6	98.9	0.71	7.23	99.5	98.8	0.66	6.81	5.87
CL2S	CL2	25	381	98.9	97.0	1.87	9.11	98.8	97.1	1.69	8.50	7.33
CL2S	GAL	26	212	98.9	97.7	1.19	8.42	98.8	97.7	1.16	7.97	6.87

Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm, 1 psi = 6.895 kPa, 1 scfm = 0.0004719 m^3/s.

starting pressures is to use the rated pressure of the compressor. Even if a compressor has not been selected, it is quite reasonable to use 125 psig (861.8 kPa) as an upper bound. Another way is to guess where the largest resistance path will be (see design step 4) and work from the machines to the compressor.

Fixing the compressor outlet pressure is the easiest way to determine starting pressures and, subsequently, pressure drops. It simplifies starting the network solution and relieves the engineer of the tedious calculations needed to determine the largest resistance path. It can, however, produce a network that has poor pressure regulation or has wasted capital. Excess pressure is rarely a problem in mining, but lack of pressure plagues many mines.

Improper equipment placement and fixed inlet pressure can produce a design that provides inadequate pressure to headings during certain operating conditions. A computer program, such as COMPAIR, allows multiple scenarios to be evaluated, eliminating many of the problems described by reducing calculation time.

Pressure regulation is a valuable tool for determining problem areas in the design. It is analogous to voltage regulation in electrical networks and calculated in a similar fashion. Notice from Eq. 12.5.17 that as pressure loss increases, P_r degrades, but a higher P_c improves P_r:

$$P_r = (P_c - P_h/P_c)\ 100 \qquad (12.5.17)$$

Loomis (1986) recommends pressure regulation for the main and primary feeders be limited to 3% each. This guarantees a maximum of 6.1% pressure regulation to any heading or stope.

The example mine design meets this specification in all but one instance, the primary feeder for C Left. Switching to a 4-in. (101.6-mm) pipe for branch 22 would correct the deficiency. To illustrate the difference between the design using the CAGH equation, Darcy-Weisbach analysis, and mass flow analysis, an arbitrary compressor outlet pressure of 123 psig (848.1 kPa) was selected. This value provides enough difference to illustrate the effects of pressure on pressure loss.

The next step in analyzing the network is to calculate the quantities and pressure drop for each branch. The assumptions used are

1. $\epsilon = 0.00015$.
2. constant temperature of 60°F (15.6°C).
3. deterioration factor = 1.35.

12.5.9.1 Radial Networks

Branch flow rates are determined by summing at each node with Kirchhoff's node equations, that is, quantity flowing in must equal quantity flowing out. The quantity at each machine location (shown in Fig. 12.5.2) was adjusted for leakage, then combined at nodes balancing from the machine to the compressor (Ex. 12.5.11).

Example 12.5.11. The mucker at heading CL1 (refer to Fig. 12.5.2) requires 315 scfm (0.149 m^3/s). Leakage adjustment (Ex. 12.5.3) brings the quantity to 381 scfm (0.180 m^3/s). This quantity is listed for branch 23 in Table 12.5.12. It is added to the machine quantities from branch 24 (593 scfm or 0.280 m^3/s) to obtain 974 scfm (0.460 m^3/s) in branch 22. See Fig. 12.5.7.

These quantities were used to determine pressure losses using the CAGH equation. The pressure loss for each branch and cumulative pressure loss along the paths (from Table 12.5.10) are shown in Table 12.5.12. The results from COMPAIR are also included for comparison.

Table 12.5.13 presents the pressure drops and inlet pressures calculated using the Darcy-Weisbach equation. Inlet pressures are used to calculate pressure losses. A better method is to calculate pressure drop based on the average pressure in the pipe.

Table 12.5.13. Pressures and Pressure Losses (Darcy-Weisbach Method)

Node			Flow	Pressure (psig)				
In	Out	Branch No.	Rate (scfm)	In	Out	Drop	Cum. Drop	Pressure Regulation
MO	ALMI	1	4423	123.0	121.0	1.97	2.0	1.50
ALMI	ALAJ	2	836	121.0	119.6	1.44	3.4	2.54
ALAJ	AJ	3	537	119.6	118.8	0.79	4.2	3.15
ALAJ	AL	4	331	119.6	118.4	1.20	4.6	3.44
ALMI	MIBS	5	224	121.0	120.6	0.40	2.4	1.80
MIBS	BLMI	7	3377	120.6	120.4	0.19	2.6	1.95
MIBS	BS	6	210	120.6	120.3	0.36	2.7	2.02
BLMI	BLS	17	598	120.4	118.9	1.51	4.1	3.07
BLS	BLH1	19	598	118.9	118.5	0.44	4.5	3.37
BLH1	BL1	20	299	118.5	117.6	0.85	5.4	4.05
BLH1	BL2	21	299	118.5	117.5	0.98	5.5	4.12
BLMI	BRRC	8	808	120.4	119.8	0.63	3.2	2.40
BRRC	BRR	9	299	119.8	119.3	0.51	3.7	2.77
BRRC	BRSI	10	509	119.8	118.0	1.80	5.0	3.75
BRSI	BR2	12	210	118.0	117.6	0.41	5.4	4.05
BRSI	BR1	11	299	118.0	117.3	0.68	5.7	4.27
BLMI	CLMI	13	1971	120.4	120.3	0.15	2.7	2.02
BLM1	CL1S	22	1373	120.3	117.4	2.86	5.6	4.20
CLMI	CR1S	14	598	120.3	118.6	1.69	4.4	3.30
CR1S	CRS	16	299	118.6	117.4	1.18	5.6	4.20
CR1S	CR	15	299	118.6	117.8	0.80	5.2	3.90
CL1S	CL1	23	537	117.4	115.3	2.11	7.7	5.77
CL1S	CL2S	24	836	117.4	116.8	0.61	6.2	4.65
CL2S	CL2	25	537	116.8	115.2	1.61	7.8	5.85
CL2S	GAL	26	299	116.8	115.8	1.01	7.2	5.40

Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm, 1 psi = 6.895 kPa, 1 scfm = 0.0004719 m^3/s.

In fact, the computer program iterates for the average pressure. However, with small line losses (less than 10% of inlet pressure), failure to average the pressure will not introduce significant error.

The mass flow technique results are presented in Table 12.5.14. Table 12.5.15 summarizes the nodal pressures, pressure loss, and pressure regulation for each of the methods described.

12.5.9.2 Combination Networks

There are advantages and disadvantages to utilizing loops in compressed air networks. Some of the advantages are:

1. Improved pressure regulation.
2. Decreased downtime for network maintenance.
3. Reduced wasted energy costs.

The disadvantages are

1. Increased initial capital cost.
2. Increased safety problems.
3. Higher maintenance costs.

Valve Placement. Valve placement in a combination network is not the same as placement in a radial network. Safety and serviceability are still the paramount placement criteria; however, more thought must be given to the zones of protection for each working place. A zone of protection is defined as the independent paths providing air to a heading. The minimum is one, and it may be desirable to have two or more.

With loop networks, valves placement can insure all headings will operate during repairs, if the zones of protection are established properly. This may be excessive if some headings are not worked on a regular basis or if several headings are assigned to the same crew.

Safety requires that, as a minimum, each heading or working place have two quick (quarter-turn) shut-off valves, one at the face and one where the heading intersects a loop. Three valves at an intersection are rarely necessary except when a four-way intersection is encountered—otherwise, two valves provide adequate protection with less confusion and reduced maintenance. The general rule for the number of valves required at an intersection is the number of branches minus 1.

12.5.9.3 Hardy Cross Technique

Combination network analysis is more difficult than radial network analysis because flow direction and quantity are unknown in loops. The Kirchhoff node equation is not adequate to determine flow direction; therefore, an iterative approach must be undertaken.

Two solution techniques, the Hardy Cross and nodal methods, were tested extensively (Gent, 1986). The Hardy Cross technique, probably the most widely used in simple fluid analysis (e.g., water lines, mine ventilation), assumes conservation of mass will govern the iteration process, and that conservation of energy will be obtained through successive iterations. The following discussion will explain the technique. The pressure loss for any pipe branch in the network can be written in simplified form as

$$\Delta p = KQ^2 \qquad (12.5.19)$$

K varies with flow rate because the friction factor is dependent on Reynolds number, which varies with fluid velocity. It is often assumed constant or changed every four or five iterations to reduce computation time. K equals the given pressure loss divided by the corresponding quantity squared.

Conservation of energy for a loop can be written:

$$\Sigma\ \Delta p_i = 0 = \Sigma\ (\pm)\ K_i Q_i^2 \qquad (12.5.20)$$

This equation will sum to zero when all quantities are cor-

Table 12.5.14. Pressures and Pressure Losses (Mass Flow Method)

Node In	Node Out	Branch No.	Flow Rate (lbm)	Pressure (psig) In	Out	Drop	Cum. Drop	Pressure Regulation
MO	ALMI	1	3.98	125.0	123.0	1.97	2.0	1.47
ALMI	ALAJ	2	0.75	123.0	121.4	1.60	3.6	2.66
ALAJ	AJ	3	0.48	121.4	120.6	0.83	4.4	3.25
ALAJ	AL	4	0.27	121.4	120.0	1.45	5.0	3.69
ALMI	MIBS	5	3.23	123.0	122.6	0.39	2.4	1.77
MIBS	BLMI	7	3.04	122.6	122.5	0.18	2.5	1.85
MIBS	BS	6	0.19	122.6	122.2	0.39	2.8	2.07
BLMI	BLS	17	0.54	122.5	121.2	1.31	3.8	2.81
BLS	BLH1	19	0.54	121.2	120.8	0.38	4.2	3.10
BLH1	BL1	20	0.27	120.8	119.7	1.03	5.3	3.91
BLH1	BL2	21	0.27	120.8	119.6	1.19	5.4	3.99
BLM1	BRRC	8	0.73	122.5	121.9	0.58	3.1	2.29
BRRC	BRR	9	0.27	121.9	121.3	0.62	3.7	2.73
BRRC	BRSI	10	0.46	121.9	119.8	2.09	5.2	3.84
BRSI	BR2	12	0.19	119.8	119.3	0.45	5.7	4.21
BRSI	BR1	11	0.27	119.8	119.0	0.82	6.0	4.43
BLMI	CLMI	13	1.78	122.5	122.3	0.16	2.7	1.99
BLMI	CL1S	22	1.24	122.3	119.6	2.68	5.4	3.99
CLMI	CR1S	14	0.54	122.3	120.8	1.46	4.2	3.10
CR1S	CRS	16	0.27	120.8	119.4	1.43	5.6	4.14
CR1S	CR	15	0.27	120.8	119.9	0.96	5.1	3.77
CL1S	CL1	23	0.48	119.6	117.4	2.22	7.6	5.61
CL1S	CL2S	24	0.75	119.6	118.9	0.68	6.1	4.51
CL2S	CL2	25	0.48	118.9	117.2	1.70	7.8	4.76
CL2S	GAL	26	0.27	118.9	117.7	1.22	7.3	5.39

Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm, 1 psi = 6.895 kPa, 1 scfm = 0.0004719 m^3/s.

Table 12.5.15. Pressure and Pressure Loss Summary Table

Branch No.	CAGH In	CAGH Out	CAGH Drop	CAGH Regulation	Darcy-Weisbach In	Darcy-Weisbach Out	Darcy-Weisbach Drop	Darcy-Weisbach Regulation	Mass-Flow In	Mass-Flow Out	Mass-Flow Drop	Mass-Flow Regulation
1	125.0	123.0	1.97	1.47	123.0	121.0	1.97	1.50	105.6	103.4	2.15	1.85
2	123.0	121.4	1.60	2.66	121.0	119.6	1.44	2.54	103.4	101.9	1.53	3.17
3	121.4	120.6	0.83	3.25	119.6	118.8	0.79	3.15	101.9	101.2	0.70	3.78
4	121.4	120.0	1.45	3.69	119.6	118.4	1.20	3.44	101.9	100.5	1.34	5.02
5	123.0	122.6	0.39	1.77	121.0	120.6	0.40	1.80	103.4	103.0	0.42	2.30
7	122.6	122.5	0.18	1.85	120.6	120.4	0.19	1.95	103.0	102.8	0.18	2.46
6	122.6	122.2	0.39	2.07	120.6	120.3	0.36	2.02	103.0	102.6	0.36	2.61
17	122.5	121.2	1.31	2.81	120.4	118.9	1.51	3.07	102.8	101.2	1.63	3.86
19	121.2	120.8	0.38	3.10	118.9	118.5	0.44	3.37	101.2	100.7	0.48	4.28
20	120.8	119.7	1.03	3.91	118.5	117.6	0.85	4.05	100.7	99.7	0.98	5.12
21	120.8	119.6	1.19	3.99	118.5	117.5	0.98	4.12	100.7	99.6	1.09	5.22
8	122.5	121.9	0.58	2.29	120.4	119.8	0.63	2.40	102.8	102.2	0.62	2.99
9	121.9	121.3	0.62	2.73	119.8	119.3	0.51	2.77	102.2	101.3	0.85	3.72
10	121.9	119.8	2.09	3.84	119.8	118.0	1.80	3.75	102.2	100.2	2.01	4.72
12	119.8	119.3	0.45	4.21	118.0	117.6	0.41	4.05	100.2	99.8	0.43	5.10
11	119.8	119.0	0.82	4.43	118.0	117.3	0.68	4.27	100.2	99.4	0.77	5.39
13	122.5	122.3	0.16	1.99	120.4	120.3	0.15	2.02	102.8	102.6	0.16	2.59
22	122.3	119.6	2.68	3.99	120.3	117.4	2.86	4.20	102.6	99.5	3.14	5.30
14	122.3	120.8	1.46	3.10	120.3	118.6	1.69	3.30	102.6	100.5	1.88	4.22
16	120.8	119.4	1.43	4.14	118.6	117.4	1.18	4.20	100.5	99.1	1.36	5.39
15	120.8	119.9	0.96	3.77	118.6	117.8	0.80	3.90	100.5	99.6	0.92	5.01
23	119.6	117.4	2.22	5.61	117.4	115.3	2.11	5.77	99.5	97.7	1.82	6.87
24	119.6	118.9	0.68	4.51	117.4	116.8	0.61	4.65	99.5	98.8	0.66	5.87
25	118.9	117.2	1.70	5.76	116.8	115.2	1.61	5.85	98.8	97.1	1.69	7.33
26	118.9	117.7	1.22	5.39	116.8	115.8	1.01	5.40	98.8	97.7	1.16	6.87

Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm, 1 psi = 6.895 kPa, 1 scfm = 0.0004719 m^3/s.

rect. If initial guesses are made, then each quantity will differ from the correct value by some delta quantity as expressed in Eq. 12.5.20:

$$Q_i = Q_i' + \Delta Q \tag{12.5.21}$$

Substituting Eq. 12.5.21 into Eq. 12.5.20,

$$\Sigma\, \Delta p_i = 0 = \Sigma\,(\pm)\, K_i(Q_i' + \Delta Q)^2 \tag{12.5.22}$$

Using the binomial theorem and expanding Eq. 12.5.22, discarding the second- and higher-order terms (the quantity ΔQ is assumed small), and solving for ΔQ yields Eq. 12.5.23:

$$\Delta Q = \frac{\Sigma\,(\pm)\, K_i Q_i^2}{-\Sigma\,|2K_i Q_i|} \tag{12.5.23}$$

The mesh equation (Eq. 12.5.23) can be interpreted as follows. The numerator is the algebraic sum of pressure losses around a given loop (conservation of energy). The denominator is the summed K-weighted quantities representing the conservation of mass. Thus the ratio of conservation of energy to conservation of mass defines the correction. The negative sign makes the adjustment in the proper flow direction, and the n in the denominator (n = 2 in Eq. 12.5.23) reduces the adjustment size to control divergence. Adjusting quantities by ΔQ in successive iterations results in a ΔQ which is close to 0. Branches that are in multiple loops are adjusted by each loop's differential quantity.

There are two drawbacks to the Hardy Cross analysis. As with all numerical techniques, the solution does not always converge or is slow to converge, and can be time-consuming if poor first estimates are made. It is also tedious to define the mesh equation.

The first step in solving a combination network after all of the physical parameters are determined is flow rate estimation in loop branches. It is strongly recommended that zero flow rates be avoided because they increase convergence time. If a computer program is unavailable, the K values for each branch will have to be estimated as demonstrated in Ex. 12.5.12.

Example 12.5.12. Calculate the K factor for a 4-in. (101.6-mm) diameter Schedule 40 pipe that is 450 ft (137.2 m) long and has two plug valves along its length. The flow rate is 3000 scfm (1.4158 m^3/s) at 125 psig (861.8 kPa).

Solution. The equivalent length of two 4-in. (101.6-mm) plug-cock valves is 15.2 ft (4.632 m), and the total length is 465.2 ft (141.8 m). The pressure loss is 6.3 psig (43.44 kPa).

The K value for this branch is

$$K = \Delta p/Q^2 = 6.3/(3000)^2 = 7.00 \times 10^{-7}$$

Sign convention is clockwise positive; therefore, flow in branches that are in two independent loops will have opposite signs as the sign convention applies to all loops. If one loop has positive flow defined in a clockwise direction, all loops must be similarly defined.

Fig. 12.5.9 shows initial combination network assumptions for flow rates at the example mine.

Example 12.5.13. Explicitly write the mesh equations for the example mine.

Solution. The mesh equations are, for loop I,

$$\Delta Q = \frac{-K_7Q^2 - K_5Q^2 - K_2Q^2 + K_{29}Q^2 - K_{30}Q^2 - K_{33}Q^2 - K_{17}Q^2}{K_7Q + K_5Q + K_2Q + K_{29}Q + K_{30}Q + K_{33}Q + K_{17}Q} \tag{12.5.24}$$

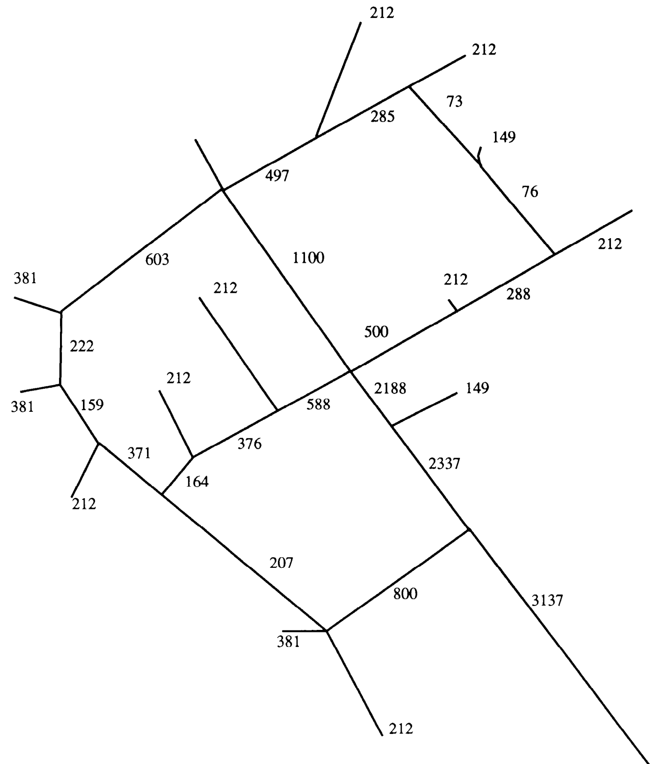

Fig. 12.5.9. Initial flow rates (combination network).

Table 12.5.16. *K* Factors for Hardy Cross Calculations

2 = 2.8890 E-06	5 = 4.5230 E-08
7 = 2.3735 E-08	8 = 2.8890 E-06
10 = 1.0192 E-05	12 = 1.2285 E-05
13 = 5.6537 E-08	14 = 6.8722 E-06
15 = 2.0240 E-05	17–19 = 6.0613 E-06
22 = 2.2389 E-06	24 = 1.3151 E-06
28 = 1.3493 E-05	29 = 1.9103 E-05
30 = 1.2486 E-05	31 = 5.5637 E-06
32 = 1.4453 E-06	33 = 6.0612 E-06

For loop II,

$$\Delta Q = \frac{K_{30}Q^2 + K_{33}Q^2 + K_{17}Q^2 + K_{31}Q^2 - K_{32}Q^2 - K_{24}Q^2 - K_{22}Q^2 - K_{13}Q^2}{K_{30}Q + K_{33}Q + K_{17}Q - K_{31}Q + K_{32}Q + K_{24}Q + K_{22}Q + K_{13}Q} \tag{12.5.25}$$

For loop III,

$$\Delta Q = \frac{K_{13}Q^2 + K_{14}Q^2 + K_{15}Q^2 + K_{28}Q^2 - K_{12}Q^2 - K_{10}Q^2 - K_8Q^2}{K_{13}Q + K_{14}Q + K_{15}Q + K_{28}Q\, K_{12}Q + K_{10}Q + K_8Q} \tag{12.5.26}$$

There are two ways to account for flow reversals. The first is to maintain the mesh equations in their present form and allow negative flow rates. The other is to maintain positive flow rates and adjust the equations. The second is the most popular, especially in computer programs, since they generally update the K factors during every iteration, and negative velocities create problems when determining friction factors.

The K factors for the example mine, given in Table 12.5.16, were calculated in the same manner as presented in Ex. 12.5.12.

Table 12.5.17. Results from Hardy Cross Analysis

Node				Length		Diameter	Pressure (psig)			
In	Out	Branch No.	Flow (scfm)	(ft)	(eq)	(in.)	In	Out	Drop	Regulation
MO	ALMI	1	3137.0	750	9.2	6.0	125.0	123.6	1.44	1.06
ALMI	ALAJ	2	741.0	400	20.1	3.0	123.6	122.0	1.58	2.22
ALAJ	AJ	3	381.0	60	14.1	2.0	122.0	121.4	0.57	2.66
ALAJ	AL	4	212.0	310	21.3	2.0	122.0	121.1	0.87	2.88
ALMI	MIBS	5	2396.0	205	19.4	6.0	123.6	123.3	0.26	1.26
MIBS	BLMI	7	2247.0	100	19.4	6.0	123.3	123.2	0.12	1.33
MIBS	BS	6	149.0	165	24.8	2.0	123.3	123.0	0.26	1.48
BLMI	SHOP	17	451.0	80	10.4	2.0	123.2	122.1	0.99	2.14
SHOP	BLH1	19	451.0	20	6.6	2.0	122.1	121.9	0.24	2.19
BLH1	BL1	20	212.0	243	14.1	2.0	121.9	121.3	0.68	2.73
BLH1	BLH2	33	239.0	100	6.6	2.0	121.9	121.6	0.35	2.51
BLH2	BL2	21	212.0	173	10.9	2.0	121.6	121.1	0.49	2.88
BLH2	BLCI	30	27.0	200	6.6	2.0	121.6	121.6	0.01	2.51
ALAJ	BLCI	29	148.0	280	23.9	2.0	122.0	121.6	0.40	2.51
BLCI	CLG	31	175.0	70	20.8	2.0	121.6	121.4	0.17	2.66
BLMI	BRR	8	579.0	175	20.1	3.0	123.2	122.7	0.47	1.70
BRR	CKL	9	212.0	120	21.3	2.0	122.7	122.3	0.37	1.99
BRR	BRI	10	367.0	180	6.6	2.0	122.7	121.4	1.34	2.66
BRI	BR1I	12	155.0	170	13.5	2.0	121.4	121.1	0.27	2.88
BR1I	BR2	27	149.0	20	10.9	2.0	121.1	121.0	0.04	2.95
BR1I	BCRI	28	6.0	135	10.1	2.0	121.1	121.1	0.00	2.88
BRI	BR1	11	212.0	170	14.4	2.0	121.4	120.9	0.49	3.03
BLMI	CLMI	13	1217.0	240	19.4	6.0	123.2	123.1	0.09	1.90
CLMI	CLI	22	799.0	310	20.1	3.0	123.1	121.7	1.43	2.44
CLMI	CRI	14	418.0	120	13.5	2.0	123.1	121.9	1.22	2.29
CRI	CRS	16	212.0	300	24.1	2.0	121.9	121.0	0.86	2.95
CRI	BCRI	15	206.0	170	6.6	2.0	121.9	121.1	0.78	2.88
BCRI	CR	34	212.0	65	11.3	2.0	121.1	120.9	0.20	3.03
CLI	CL1	23	381.0	180	14.4	2.0	121.7	120.1	1.51	3.62
CLI	CL2I	24	418.0	150	24.2	3.0	121.7	121.4	0.23	2.66
CL2I	CL2	25	381.0	130	18.2	2.0	121.4	120.3	1.15	3.47
CL2I	CLG	32	37.0	110	9.4	2.0	121.4	121.4	0.01	2.66
CLG	GAL	26	212.0	140	21.3	2.0	121.4	121.0	0.43	2.95

Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm, 1 psi = 6.895 kPa, 1 scfm = 0.0004719 m^3/s.

Table 12.5.17 summarizes the flow rates, pressures, pressure losses, and cumulative pressure regulation for each branch in the combination network. Both the upstream and downstream pressures are given.

Table 12.5.18 lists the gage pressures, pressure drops, and pressure regulations for the volumetric, radial, and Hardy Cross analyses. Improved pressure regulation was obtained with the combination network. Example 12.5.13 shows pressure losses around closed loops sum to zero.

Example 12.5.14. Sum pressure losses around the example loops to insure conservation of energy has been maintained.

Solution. Loop I

$$PL_I = 1.58 + 0.40 - 0.01 - 0.35 - 0.99 - 0.24 - 0.12 - 0.26 = 0.01 \text{ psig } (0.07 \text{ kPa})$$

Loop II

$$PL_{II} = 0.01 + 0.35 + 0.99 + 0.24 + 0.17 - 0.01 - 0.23 - 1.43 - 0.09 = 0.00 \text{ psig } (0.00 \text{ kPa})$$

Loop III

$$PL_{III} = 0.09 + 1.22 + 0.78 - 0.00 - 0.27 - 1.34 - 0.47 = 0.01 \text{ psig } (0.07 \text{ kPa})$$

When the pressure losses around all loops sum to zero (or within a specified tolerance), conservation of energy has been achieved, representing convergence of the Hardy Cross algorithm.

12.5.9.4 Nodal Technique

Nodal analysis assumes conservation of energy will govern all iterations until conservation of mass is achieved. The derivation of the adjustment scheme for nodal analysis is almost identical to the Hardy Cross method, except pressure is the independent variable and flow rate is the dependent variable that is calculated from the nodal pressures.

Manually analyzing combination networks with the nodal technique requires much computation, even more than the Hardy Cross method. It does, however, produce a small computer algorithm that removes the burden of user-specified meshes.

The nodal form of the mesh equations is

$$\Delta P_{ij} = \frac{n \, \Sigma \, m_{ij}}{\Sigma \, m_{ij} \, P_{ij}} \qquad (12.5.27)$$

Table 12.5.19 presents the initial flow rates and fittings for the combination analysis utilizing the nodal method. Table 12.5.20, generated by COMPAIR, shows the results. Solution was achieved after 30 iterations (approximately 5 minutes on a

Table 12.5.18. Radial and Combination Network Pressure Comparisons

Branch No.	Flow (scfm)	Pressure (psig) In	Out	Drop	Regulation	Branch No.	Flow (scfm)	Pressure (psig) In	Out	Drop	Regulation
1	3137.0	125.0	123.6	1.44	1.06	1	4423	123.0	121.0	1.97	1.50
2	741.0	123.6	122.0	1.58	2.22	2	836	121.0	119.6	1.44	2.54
3	381.0	122.0	121.4	0.57	2.66	3	537	119.6	118.8	0.79	3.15
4	212.0	122.0	121.1	0.87	2.88	4	331	119.6	118.4	1.20	3.44
5	2396.0	123.6	123.3	0.26	1.26	5	224	121.0	120.6	0.40	1.80
7	2247.0	123.3	123.2	0.12	1.33	7	3377	120.6	120.4	0.19	1.95
6	149.0	123.3	123.0	0.26	1.48	6	210	120.6	120.3	0.36	2.02
17	451.0	123.2	122.1	0.99	2.14	17	598	120.4	118.9	1.51	3.07
19	451.0	122.1	121.9	0.24	2.19	19	598	118.9	118.5	0.44	3.37
20	212.0	121.9	121.3	0.68	2.73	20	299	118.5	117.6	0.85	4.05
33	239.0	121.9	121.6	0.35	2.51	21	299	118.5	117.5	0.98	4.12
21	212.0	121.6	121.1	0.49	2.88	8	808	120.4	119.8	0.63	2.40
30	27.0	121.6	121.6	0.01	2.51	9	299	119.8	119.3	0.51	2.77
29	148.0	122.0	121.6	0.40	2.51	10	509	119.8	118.0	1.80	3.75
31	175.0	121.6	121.4	0.17	2.66	12	210	118.0	117.6	0.41	4.05
8	579.0	123.2	122.7	0.47	1.70	11	299	118.0	117.3	0.68	4.27
9	212.0	122.7	122.3	0.37	1.99	13	1971	120.4	120.3	0.15	2.02
10	367.0	122.7	121.4	1.34	2.66	22	1373	120.3	117.4	2.86	4.20
12	155.0	121.4	121.1	0.27	2.88	14	598	120.3	118.6	1.69	3.30
27	149.0	121.1	121.0	0.04	2.95	16	299	118.6	117.4	1.18	4.20
28	6.0	121.1	121.1	0.00	2.88	15	299	118.6	117.8	0.80	3.90
11	212.0	121.4	120.9	0.49	3.03	23	537	117.4	115.3	2.11	5.77
13	1217.0	123.2	123.1	0.09	1.90	24	836	117.4	116.8	0.61	4.65
22	799.0	123.1	121.7	1.43	2.44	25	537	116.8	115.2	1.61	5.85
14	418.0	123.1	121.9	1.22	2.29	26	299	116.8	115.8	1.01	5.40
16	212.0	121.9	121.0	0.86	2.95						
15	206.0	121.9	121.1	0.78	2.88						
34	212.0	121.1	120.9	0.20	3.03						
23	381.0	121.7	120.1	1.51	3.62						
24	418.0	121.7	121.4	0.23	2.66						
25	381.0	121.4	120.3	1.15	3.47						
32	37.0	121.4	121.4	0.01	2.66						
26	212.0	121.4	121.0	0.43	2.95						

Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm, 1 psi = 6.895 kPa, 1 scfm = 0.0004719 m^3/s.

microcomputer) with the largest deviation from exact conservation of mass equivalent to 10 scfm (0.004719 m^3/s) at the intersection of B Left and C Left. Pressure differences were less than 0.0001 psig (0.0007 kPa).

There is no significant difference between the flow rates calculated with the Hardy Cross and nodal techniques.

12.5.10 COMPRESSOR HORSEPOWER

Compressor horsepower calculation is simple if the compression cycle is assumed to be an adiabatic process. In actuality, the cycle is neither adiabatic nor isothermal but somewhere between the two. If the horsepower for a single-stage compressor is desired, and the process is assumed adiabatic, the following relation applies:

$$ihp/100 \text{ acfm} = 1.542\ P_s(r^{0.283} - 1) \qquad (12.5.28)$$

To calculate the horsepower for a two-stage compressor, a slightly different formula is used:

$$ihp/100 \text{ acfm} = 3.084\ P_s(r^{0.1415} - 1) \qquad (12.5.29)$$

Eqs. 12.5.28 and 12.5.29 determine indicated horsepower not brake horsepower. To obtain brake horsepower from indicated, divide the indicated horsepower by the appropriate efficiency, which is commonly around 85%.

Example 12.5.14. For the example mine, a two-stage piston compressor with a mechanical efficiency of 87% is assumed available. Determine the indicated and brake horsepowers for this compressor.

Solution. 1. For CAGH analysis, the compressor indicated and brake horsepowers are as follows:

$$r = (105.6 + 10.40)/10.40 = 11.154$$

$$ihp = [(3137)(1.41)/100](3.084)(10.40)(11.154^{0.1415} - 1) = 577.0$$

$$bhp = 577.0/0.87 = 663.2, \text{ or approximately } 675 \text{ (503.3 kW)}$$

2. For volumetric analysis:

$$r = (123.0 + 10.40)/10.40 = 12.827$$

$$ihp = (4423/100)(3.084)(10.40)(12.827^{0.1415} - 1) = 616.9$$

$$bhp = 616.9/0.87 = 709.0, \text{ or approximately } 725 \text{ (540.6 kW)}$$

3. For mass flow analysis:

$$r = (125.0 + 10.40)/10.40 = 13.019$$

$$ihp = (4423/100)(3.084)(10.40)(13.019^{0.1415} - 1) = 621.1$$

$$bhp = 621.1/0.87 = 713.9, \text{ or approximately } 725 \text{ (540.6 kW)}$$

Table 12.5.19. Example Mine Nodal Analysis Initial Conditions

Node In	Node Out	Branch No.	Flow (scfm)	Length (ft)	Diam (in.)	Valves and Fittings
MO	ALMI	1	3137.0	750	6.0	1pv;
ALAJ	ALAJ	2	593.0	400	3.0	1st, 1pv;
ALAJ	AJ	3	381.0	60	2.0	1rt, 14e, 1bf;
ALAJ	AL	4	212.0	310	2.0	1st, 1pv, 1bf;
ALMI	MIBS	5	2544.0	205	6.0	1rt, 1pv;
MIBS	BLMI	7	2395.0	100	6.0	1rt, 1pv;
MIBS	BS	6	149.0	165	2.0	1st, 1rt, 1pv, 1bf;
BLMI	SHOP	17	424.0	80	2.0	1st, 1rt, 1pv;
SHOP	BLH1	19	424.0	20	2.0	1rt;
BLH1	BL1	20	212.0	243	2.0	1st, 1pv, 1bv;
BLH1	BLH2	33	212.0	100	2.0	1rt, 1pv;
BLH2	BL2	21	212.0	173	2.0	19e, 1bf;
BLH2	BLCI	30	0.0	200	2.0	1rt, 1pv;
BLCI	ALAJ	29	0.0	280	2.0	2st, 1pv;
CLG	BLCI	31	0.0	70	2.0	2st;
BLMI	BRR	8	573.0	175	3.0	1st, 1pv;
BRR	CKL	9	212.0	120	2.0	1st, 19e, 1bf;
BRR	BRI	10	361.0	180	2.0	1rt, 1pv;
BRI	BR1I	12	149.0	170	2.0	1st, 1pv;
BR1I	BR2	27	149.0	20	2.0	19e, 1bf;
BR1I	BCRI	28	0.0	135	2.0	2rt, 1pv;
BRI	BR1	11	212.0	170	2.0	1rt, 1pv, 1bf;
BLMI	CLMI	13	1398.0	240	6.0	1rt, 1pv;
CLMI	CLI	22	974.0	310	3.0	1st, 1pv;
CLMI	CRI	14	424.0	120	2.0	1st, 1pv;
CRI	CRS	16	212.0	300	2.0	1st, 14e, 1pv, 1bf;
CRI	BCRI	15	212.0	170	2.0	1rt, 1pv;
BCRI	CR	34	212.0	65	2.0	1rt, 1bf;
CLI	CL1	23	381.0	180	2.0	1rt, 1pv, 1bf;
CLI	CL2I	24	593.0	150	3.0	1st, 14e, 1pv;
CL2I	CL2	25	381.0	130	2.0	1st, 1bf;
CL2I	CLG	32	212.0	110	2.0	1rt, 1pv, 14e;
CLG	GAL	26	212.0	140	2.0	1st, 1bf, 1pv;

Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm, 1 psi = 6.895 kPa, 1 scfm = 0.0004719 m^3/s.

The mass flow rate of 3.98 lbm/sec was converted to scfm to perform the above calculation.

The reader should notice that the required horsepower increases as pressure increases. When calculating horsepower using the CAGH method results, the flow was corrected for altitude since compression ratio affects horsepower, and therefore all calculations must be done at actual conditions. The other methods already accounted for altitude.

When selecting compressor(s), consideration must be given to the number of compressors to be purchased. One large compressor or multiple smaller compressors can be selected as long as capacity requirements are satisfied. Multiple compressor arrangements are more expensive in both initial capital costs and ongoing maintenance costs; however, they are insurance against complete shutdown, and their purchase can be spread over time as the mine expands.

Determining receiver volume is the next step in the design. Typical recommendations for receiver size are 10 to 20% of the acfm compressor capacity, or alternatively one can use manufacturers tables. For the example mine, the above rule of thumb requires a receiver capacity between 400 and 800 ft^3 (11.3 and 22.7 m^3). A 400-ft^3 (11.3-m^3) receiver will be selected as additional capacity is provided by the network. In fact, with large networks where flow rates are very high, surface receiver capacity requirements are generally negligible.

Receivers can be useful to supply short duration demands in excess of compressor capacity., for example, if numerous air starters are activated at the beginning of a shift. In this instance, another method for estimating receiver capacity should be used (Rollins, 1973):

$$V_r = T\ C\ P_e/(P_{ri} - P_{rf}) \tag{12.5.30}$$

12.5.11 OPTIMAL PIPE DIAMETERS AND SYSTEM COSTS

Capital and operating cost estimates vary by location and the method presented is by no means a complete treatment of the subject. Valves and fittings costs vary widely throughout the world, and the values presented in Table 12.5.21 are estimates.

Steel pipe cost can be estimated by weight. A complete treatment of this method is given in the optimization section. Table 12.5.22 provides a sampling of pipe costs.

Initial compressor cost (Table 12.5.23) and installation also vary widely due to local differentials in labor rates, options installed, and size. Options include dewatering, oil removal, pre- and post-cooling, and monitoring. The provision of a receiver can be another major cost.

Determining optimal diameters can be extremely beneficial in reducing the overall cost of mine compressed air networks. The following equation can be used to determine the optimal inside diameters required. The equation developed by Gent

Table 12.5.20. Example Mine Nodal Analysis Results

Node In	Node Out	Flow (scfm)	Length (ft)	Length (eq)	Diam (in.)	Pressure In	Pressure Out	Pressure Drop	Pressure Regulation
MO	ALMI	3137.0	750	9.2	6.0	125.0	123.5	1.45	1.09
ALMI	ALAJ	734.0	400	20.1	3.0	123.5	122.0	1.55	2.20
ALAJ	AJ	381.0	60	14.1	2.0	122.0	121.4	0.57	2.64
ALAJ	AL	212.0	310	21.3	2.0	122.0	121.1	0.87	2.86
ALMI	MIBS	2402.2	205	19.4	6.0	123.5	123.3	0.26	1.25
MIBS	BLMI	2251.6	100	19.4	6.0	123.3	123.1	0.12	1.39
MIBS	BS	149.0	165	24.8	2.0	123.3	123.0	0.26	1.47
BLMI	SHOP	449.0	80	17.0	2.0	123.1	122.1	1.01	2.13
SHOP	BLH1	448.4	20	3.5	2.0	122.1	121.9	0.25	2.27
BLH1	BL1	212.0	243	14.1	2.0	121.9	121.2	0.68	2.79
BLH1	BLH2	236.1	100	6.6	2.0	121.9	121.5	0.34	2.57
BLH2	BL2	212.0	173	10.9	2.0	121.5	121.0	0.49	2.93
BLH2	BLCI	23.3	200	6.6	2.0	121.5	121.5	0.01	2.57
ALAJ	BLCI	152.1	280	23.9	2.0	122.0	121.5	0.44	2.57
BLCI	CLG	175.1	70	20.8	2.0	121.5	121.3	0.17	2.71
BLMI	BRR	563.9	175	20.1	3.0	123.1	122.7	0.45	1.69
BRR	CKL	212.0	120	21.3	2.0	122.7	122.3	0.37	1.98
BRR	BRI	351.8	180	6.6	2.0	122.7	121.4	1.24	2.64
BRI	BR1I	139.5	170	13.5	2.0	121.4	121.2	0.23	2.79
BR1I	BR2	149.0	20	10.9	2.0	121.2	121.2	0.04	2.79
BCRI	BR1I	10.4	135	10.1	2.0	121.2	121.2	0.00	2.79
BRI	BR1	212.0	170	14.4	2.0	121.4	121.0	0.49	2.93
BLMI	CLMI	1235.9	240	19.4	6.0	123.1	123.0	0.09	1.47
CLMI	CLI	801.1	310	20.1	3.0	123.0	121.6	1.44	2.49
CLMI	CRI	434.7	120	13.5	2.0	123.0	121.7	1.31	2.42
CRI	CRS	212.0	300	24.1	2.0	121.7	120.9	0.86	3.01
CRI	BCRI	222.6	170	6.6	2.0	121.7	121.2	0.51	3.80
BCRI	CR	212.0	65	11.3	2.0	121.2	121.0	0.20	2.93
CLI	CL1	381.0	180	14.4	2.0	121.6	120.1	1.51	3.59
CLI	CL2I	419.3	150	24.2	3.0	121.6	121.4	0.23	2.64
CL2I	CL2	381.0	130	18.2	2.0	121.4	120.2	1.16	3.52
CL2I	CLG	37.9	110	9.4	2.0	121.4	121.3	0.02	2.71
CLG	GAL	212.0	140	21.3	2.0	121.3	120.9	0.43	3.01

Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm, 1 psi = 6.895 kPa, 1 scfm = 0.0004719 m^3/s.

Table 12.5.21. Valve and Fitting Cost

Valve or Fitting	Nominal Size (in.) 2	3	4	6	8
Butterfly Valve	40.10	66.30	89.10	198.65	339.55
90 Elbow	5.80	10.50	15.60	44.45	93.30
45 Elbow	5.80	10.50	15.60	44.45	93.30
22 1/2 Elbow	5.80	11.55	15.60	44.45	93.30
11 1/4 Elbow	5.80	11.55	15.60	44.45	79.55
Tees	8.35	16.05	24.60	67.00	147.55
Crosses	14.35	37.60	62.75	165.35	217.85
Wyes	37.00	52.70	69.05	122.65	176.85
Couplings	5.80	7.80	11.45	20.85	34.10
Check valve	—	—	165.60	242.45	351.15
8 inch Reducer	—	42.35	42.35	54.50	—
6 inch Reducer	23.60	23.60	20.55	—	54.50
4 inch Reducer	10.05	12.60	—	20.55	42.35
3 inch Reducer	7.55	—	12.60	23.60	42.35
2 inch Reducer	—	7.55	10.05	23.60	—
Vic Groover	219.75	284.60	311.85	430.10	453.05

Source: Anon., 1978.
Conversion factor: 1 in. = 25.4 mm.

(1986) balances energy costs with initial installation costs to determine the optimal diameter

$$D_o = 1.465 \left[\frac{m^3 f y C_e P/A}{[(1 + P/A) b + F + z] P_1^2 X E}\right] \qquad (12.5.31)$$

Optimal diameters in the range of 1 to 16 in. can be determined with this model. Caution should be exercised with diameters outside this range.

Table 12.5.22. Pipe Cost and Weight Information, Schedule 40

Diameter (in.) Nominal	Actual	Cost $/ft	Weight lb/ft	Cost $/lb
2	2.067	1.44	3.65	0.395
3	3.068	2.46	7.58	0.325
4	4.026	3.44	10.79	0.319
6	6.065	5.75	18.97	0.303
8	7.981	8.45	28.55	0.296
10	10.020	11.90	40.48	0.294
12	11.938	15.05	49.56	0.304

Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm, 1 lb = 0.4536 kg.

Example 12.5.15. The following assumptions will be used to determine the optimal pipe sizes for the radial and combination networks:

1. Internal rate of return is 15%.
2. Installation is 20% of pipe cost.
3. Cost of electricity is 0.07 $/kWh.
4. Mine operates 3000 hr/yr at full capacity.
5. Life of the design is 10 years.
6. Minimum size pipe is 2 in. (50.8 mm).
7. Maximum size pipe is 12 in. (304.8 mm).

Table 12.5.23. Compressor Costs

Compressor	Acfm 125 psig	Motor hp	Cost $ (1972)	$/100 acfm (1972)	Installation Costs $ (1972)	$/100 acfm (1972)
125 BT 40	547	125	11,016	20.0	200	21
VC 600 A	583	150	14,530	25.0	150	25
24 × 12 × 8 XLE 2	1340	250	43,875	33.0	12,500	42
24 × 14 × 10 XLE 2	1880	350	60,455	33.0	17,500	41
30 × 19 × 10 XLE 2	6340	1250	157,775	25.0	44,000	32
2C40M4	3440	900	82,000	24.0	11,000	27
4C110M4	9935	2500	165,000	16.5	34,000	20

Source: Cummins and Given, 1973.
Conversion factors: 1 psi = 6.895 kPa, 1 scfm = 0.0004719 m³/s, 1 hp = 0.7457 kW.

Table 12.5.24. Optimized Radial Network Cost

Node		Branch No.	Diameter (in.)	Pipe Cost $	Valve Cost $	Fitting Cost $	Coupling Cost $
In	Out						
MO	ALMI	1	10.0	8925.00	446.69	262.30	1844.90
ALMI	ALAJ	2	4.0	1376.00	89.10	111.00	229.00
ALAJ	AJ	3	4.0	206.40	89.10	57.70	34.35
ALAJ	AL	4	3.0	762.60	66.30	55.20	124.80
ALMI	MIBS	5	10.0	2439.50	446.69	147.65	534.05
MIBS	BLMI	7	10.0	1190.00	446.69	217.85	242.75
MIBS	BS	6	3.0	405.90	66.30	141.60	70.20
BLMI	BLS	17	4.0	344.00	89.10	111.00	57.25
BLS	BLH1	19	4.0	103.20	—	24.60	22.90
BLH1	BL1	20	3.0	541.20	66.30	55.20	85.80
BLH1	BL2	21	3.0	871.92	66.30	65.70	101.40
BLMI	BRRC	8	4.0	602.00	89.10	102.00	103.05
BRRC	BRR	9	3.0	295.20	—	65.25	46.80
BRRC	BRSI	10	4.0	619.20	89.10	24.60	103.05
BRSI	BR2	12	3.0	467.40	66.30	74.05	78.00
BRSI	BR1	11	3.0	418.20	66.30	57.70	70.20
BLMI	CLMI	13	8.0	2028.00	339.55	147.55	409.20
BLMI	CL1S	22	6.0	1782.50	198.65	98.95	333.60
CLMI	CR1S	14	4.0	412.80	—	16.05	68.70
CR1S	CRS	16	3.0	738.00	66.30	55.20	117.00
CR1S	CR	15	3.0	504.30	66.30	55.20	85.80
CL1S	CL1	23	4.0	619.20	89.10	57.70	103.05
CL1S	CL2S	24	4.0	516.00	89.10	40.20	91.60
CL2S	CL2	25	4.0	447.20	—	73.30	80.15
CL2S	GAL	26	3.0	615.00	66.30	74.05	101.40
Totals				27,230.43	3098.67	2191.60	5139.50

Conversion factor: 1 in. = 25.4 mm.

8. 95.2 psig (656 kPa) required at C Left 2nd.
9. Maintenance is 7% of initial pipe cost per year.
10. List prices given in Tables 12.5.11 and 12.5.12 apply.
11. Overall energy conversion is 85% efficient.

Solution (Radial Network). Table 12.5.24 gives the radial network initial and optimized pipe and fitting costs, respectively.

To provide 95.2 psig (656 kPa) to C Left 2nd requires 104.5 psig (721 kPa) in the unoptimized case and 95.9 psig (661 kPa) in the optimized case. The required horsepower to produce the required quantity and pressure is as follows:

$$bhp = (3.084)(44.23)(10.40)\,[(104.5 + 10.40/10.40)^{0.1415} - 1]/0.85 = 676 \text{ (504.1 kW)}$$

$$bhp = (3.084)(44.23)(10.40)\,[(95.9+10.40/10.40)^{0.1415} - 1)]/0.85 = 650 \text{ (484.7 kW)}$$

$$\text{Energy saved} = (676 - 650)(3000)(0.7457) = 58{,}164.6 \text{ kWh/yr}$$

$$\text{Cost} = (58{,}164.6)(0.07) = \$4071.52/\text{yr}$$

$$\text{Present value at } 15\% = 5.019(4071.52) = \$20{,}435$$

The cost of using the larger-diameter pipe is $20,501 including the additional 20% for installation. The yield on the investment is 15%.

12.5.11.1 Combination Network

Solution. Initial and optimized costs are presented in Table 12.5.25 and Table 12.5.26, respectively. It is evident from the total costs that the optimized design is more capital intensive.

Table 12.5.25. Initial Combination Network Cost

Node In	Node Out	Branch No.	Diameter (in.)	Pipe Cost $	Valve Cost $	Fitting Cost $	Coupling Cost $
MO	ALMI	1	6.0	4312.50	198.50	67.00	792.30
ALMI	ALAJ	2	3.0	984.00	66.30	61.20	156.00
ALAJ	AJ	3	2.0	86.40	40.10	47.65	16.40
ALAJ	AL	4	2.0	446.40	40.10	47.65	92.80
ALMI	MIBS	5	6.0	1178.75	198.65	67.00	229.35
MIBS	BLMI	7	6.0	575.00	198.65	165.35	104.25
MIBS	BS	6	2.0	237.60	40.10	118.90	52.20
BLMI	SHOP	17	2.0	144.00	40.10	52.60	29.00
SHOP	BLH1	19*	2.0				
BLH1	BL1	20	2.0	349.92	40.10	47.65	75.40
BLH1	BLH2	33	2.0	144.00	40.10	16.70	29.00
BLH2	BL2	21	2.0	249.12	40.10	47.65	52.20
BLH2	BLCI	30	2.0	288.00	40.10	8.35	58.00
BLCI	ALAJ	29	2.0	403.20	40.10	—	81.20
CLG	BLCI	31	2.0	100.80	—	—	23.20
BLMI	BRR	8	3.0	430.50	66.30	61.20	70.20
BRR	CKL	9	2.0	172.80	—	53.45	34.80
BRR	BRI	10	2.0	259.20	40.10	8.35	52.20
BRI	BR1I	12	2.0	244.80	40.10	8.35	52.20
BR1I	BR2	27	2.0	28.00	—	47.65	11.60
BR1I	BCRI	28	2.0	194.40	40.10	8.35	40.60
BRI	BR1	11	2.0	244.80	—	47.65	52.20
BLMI	CLMI	13	6.0	1380.00	198.65	122.65	250.20
CLMI	CLI	22	3.0	762.60	66.30	16.05	124.80
CLMI	CRI	14	2.0	172.80	40.10	31.95	34.80
CRI	CRS	16	2.0	432.00	40.10	8.35	87.00
CRI	BCRI	15	2.0	244.80	40.10	—	52.20
BCRI	CR	34	2.0	93.60	—	47.65	40.60
CLI	CL1	23	2.0	259.20	—	47.65	52.20
CLI	CL2I	24	3.0	369.00	66.30	15.90	62.40
CL2I	CL2	25	2.0	187.20	—	47.65	40.60
CL2I	CLG	32	2.0	158.40	40.10	8.35	34.80
CLG	GAL	26	2.0	201.60	—	47.65	40.60
Totals				15336.19	1701.40	1376.55	2925.30

Average fitting-to-pipe cost ratio = 0.389
* Accounted for in the previous branch
Conversion factor: 1 in. = 25.4 mm.

Table 12.5.26. Optimal Combination Network Cost

Node In	Node Out	Branch No.	Diameter (in.)	Pipe Cost $	Valve Cost $	Fitting Cost $	Coupling Cost $
MO	ALMI	1	10.0	8925.00	446.69	262.30	1844.90
ALMI	ALAJ	2	4.0	1376.00	89.10	83.20	229.00
ALAJ	AJ	3	4.0	206.40	89.10	73.25	34.35
ALAJ	AL	4	3.0	762.60	66.30	60.25	124.80
ALMI	MIBS	5	10.0	2439.50	446.69	262.30	534.05
MIBS	BLMI	7	10.0	1190.00	446.69	262.30	242.75
MIBS	BS	6	3.0	366.54	66.30	86.40	70.20
BLMI	SHOP	17	4.0	344.00	89.10	90.55	57.25
SHOP	BLH1	19*	4.0				
BLH1	BL1	20	3.0	597.78	66.30	67.80	102.40
BLH1	BLH2	33	4.0	344.00	89.10	24.60	57.25
BLH2	BL2	21	3.0	425.58	66.30	55.10	70.20
BLH2	BLCI	30	2.0	288.00	40.10	8.35	58.00
BLCI	ALAJ	29	3.0	403.20	66.30	—	109.20
CLG	BLCI	31	3.0	177.20	—	—	31.20
BLMI	BRR	8	4.0	602.00	89.10	66.95	103.05
BRR	CKL	9	3.0	295.20	—	78.30	46.80
BRR	BRI	10	4.0	619.20	89.10	24.60	103.05
BRI	BRI1	12	2.0	244.80	40.10	8.35	52.20
BR1I	BR2	27	2.0	28.00	— M	47.65	11.60
BR1I	BCRI	28	2.0	194.40	40.10	8.35	40.60
BRI	BR1	11	3.0	418.20	—	67.90	70.20
BLMI	CLMI	13	6.0	1380.00	198.65	122.65	250.20
CLMI	CLI	22	6.0	1782.50	198.65	87.55	333.60
CLMI	CRI	14	4.0	412.80	89.10	66.95	68.70
CRI	CRS	16	3.0	738.00	66.30	73.25	117.00
CRI	BCRI	15	2.0	244.80	40.10	—	52.20
BCRI	CR	34	3.0	159.90	—	62.65	31.20
CLI	CL1	23	4.0	619.20	—	57.60	103.05
CLI	CL2I	24	4.0	516.00	89.10	24.60	80.15
CL2I	CL2	25	4.0	447.20	—	57.60	80.15
CL2I	CLG	32	2.0	158.40	40.10	10.35	34.80
CLG	GAL	26	3.0	344.40	—	62.85	54.60
Totals				27046.60	3022.90	2264.55	5197.15

Average fitting-to-pipe cost ratio = 0.389
* Accounted for in the previous branch
Conversion factor: 1 in. = 25.4 mm.

However, energy cost savings provide a 17% rate of return on the investment.

To provide 95.2 psig (656 kPa) to C Left 2nd requires 103.1 psig (711 kPa) from the compressor in the unoptimized case and 95.9 psig (661 kPa) in the optimized case. The horsepower required is as follows:

$$bhp = (3.084)(44.23)(10.40)\,[(103.1 + 10.40/10.40)^{0.1415} - 1.0]/0.85 = 672\ (501.1\ \text{kW})$$

$$bhp = (3.084)(44.23)(10.40)\,[(94.9 + 10.40/10.40)^{0.1415} - 1.0)]/0.85 = 650\ (484.7\ \text{kW})$$

$$\text{Energy saved} = (672 - 650)(3000)\ 0.7457 = 49{,}216.4\ \text{kWh/yr}$$

$$\text{Cost} = 49{,}216.4\ \text{kWh}\ 0.07\$/\text{kWh} = \$3445$$

$$\text{Time zero value at } 15\% = 5.019(3445) = \$17{,}291$$

The cost of using the larger diameter pipe is $18,554. The yield on the additional investment is 14%.

SYMBOL CONVENTIONS

A pipe area in ft^2
acfm actual flow rate in cfm
b maintenance as a % of initial pipe cost
bhp brake horsepower
C short duration air requirement in cfm
C_e cost of electricity in $/kWh
D pipe valve or fitting diameter in ft
D_c air hammer internal cylinder diameter in ft
D_o optimal pipe diameter in ft
E compressor efficiency
F fitting-to-pipe cost ratio
F_i inlet mass altitude adjustment factor
F_o outlet mass altitude adjustment factor
F_v volumetric altitude adjustment factor
f Moody friction factor (dimensionless)
g_c experimental constant = 32.174 ft lbm/(lbf set^2)
H elevation difference in ft
Hz air hammer blow frequency in blows/min
i interest rate in decimal %
ihp indicated horsepower
K linearized constant for combination network algorithms
K_e L/D formula coefficient for specific valves and fittings
L pipe length in ft
L/D equivalent length ratio for valve or fitting (unitless)
L_e valve or fitting equivalent length in ft
m mass flow rate in lbm/sec
P_c compressor outlet pressure in psia
P/A present value of an annuity based on internal ROR

P_e atmospheric pressure at any elevation in psia
P_f fluid pressure in psia
P_h heading pressure in psia
P_o operating pressure in psig
P_r pressure regulation in %
P_{ri} initial receiver pressure in psia
P_{rf} final receiver pressure in psia
P_s standard atmospheric pressure = 14.696 psia
P_1 pressure at branch beginning in psia
P_2 pressure at branch end in psia
Q flow rate in cfm
R Reynolds number (dimensionless)
R_c gas constant for air = (unitless) 53.34 ft lbf/(lbm °R)
r compression ratio
$scfm$ standardized flow rate in cfm
S_c air hammer stroke length in ft
T time in min
T_o absolute temperature in °R
V fluid velocity in fps
V_c air hammer per stroke air volume in ft^3
V_r receiver volume in ft^3
X price of 2-in. schedule 40 pipe in $/ft
Y hours/year that pipe is to operate
z installation-to-pipe cost ratio
Δp pressure drop in psig
Δp_h altitude pressure drop in psia
ΔQ flow rate adjustment in cfm
ϵ absolute roughness
ρ fluid density in lbm/ft^3
ν relative (u/r) fluid viscosity in $cp/(lbm/ft^3)$
θ absolute fluid viscosity in cp

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Chapter 12.6
MONITORING, CONTROL, AND COMMUNICATIONS

JEFFERY KOHLER

12.6.1 INTRODUCTION

The application of technology to improve all aspects of mining is especially evident in the areas of communications and monitoring systems as well as in automation and robotics. It is important to recognize that decisions to apply these technologies are based on certain expectations that tangible benefits, such as improved safety or reduced costs, will be realized. Mine monitoring systems can provide quality information in a timely fashion; timely information can be used to improve the decision-making process, and in so doing allow realization of the desired benefits. Similarly, mine communications systems that have been carefully planned and correctly designed facilitate information flow throughout all levels of the organization. However, the achievement of expected benefits will only occur when the technical characteristics of the selected systems are capable of satisfying the mine's needs for information. All too often, the mine's information requirements are not carefully considered prior to the purchase and installation of a system, and once a system is installed, it is very difficult to correct mistakes. Thus it is important to thoroughly address these issues.

Regulations establish minimum requirements for communications systems in general and for mine monitoring systems in specific instances. Beyond these required minimums, every aspect of the system synthesis must be driven by information requirements. One means of defining these requirements is to provide answers to the following (Kohler et al., 1987):

1. *Who* needs what information?
2. *How often* is this information needed?
3. What *level of detail* is needed?
4. What *presentation format* will satisfy the need?

System synthesis must be based on justifiable needs rather than desires and fads; this list is useful for identifying these needs. The tradeoff between information and cost must also be considered. Almost any piece of information can be obtained if economic constraints are removed. Accordingly, steps should be taken to identify the added incremental cost to obtain various information elements from the monitoring system. The following specific comments apply more to monitoring rather than communication systems, though there are some common aspects, especially in the case of systems that combine both monitoring and voice communication capabilities.

The determination of *what* information is needed will define the type of sensors and, ultimately many parameters relating to system communications and architecture. The who aspect affects the architecture, generally reflecting a requirement for a distributed architecture, when certain information is required in different locations with different levels of processing. After this question is answered, the locations for terminals, display screens, or annunciator panels will be evident. It is important to allow easy access to the monitored information in locations where it can be utilized.

The responses to the first question will have a significant impact on the system hardware; the other three questions/answers will tend to dictate the software structure of the system, although there are obvious hardware ramifications as well. All three questions focus on the difference between data and information. Computer-based data collection systems in general—and monitoring systems in particular—are good at collecting large amounts of data within short time periods, and overwhelming amounts of data over longer time frames. In fact, the sheer volume of data collected has often discouraged its use. All data, however, are not information.

Information, by definition, reduces uncertainty. Thus, for example, if the concentration of gas is known, an additional piece of data pictorially representing the gas concentration does not reduce any uncertainty, and as such is not information. Anyone who has ever sorted through a stack of printer paper from a monitoring system output can attest to the fact that there were few data elements that reduced uncertainty in a decision-making framework. Accordingly, the challenge is to process the large volume of raw data and extract and display useful information. The 1970s was a period in which the utility of mine monitoring was restricted by hardware inadequacies and shortcomings; the 1980s was a decade when the utility of monitoring systems was limited by their inability to handle information. Just as research and development efforts overcame the hardware limitations of the 1970s, similar efforts successfully addressed the information issues of the 1980s. Actually, information management became a key concern in many aspects of engineering operations. Despite the advances that have contributed to improved information management within monitoring systems, the burden is still on the user to match the local information needs with the technical capabilities of the system.

The second question of *how often* has an operational side in addition to the information management aspect. In general, there should be two modes for accessing information: alarm and interrogation. In alarm mode, information elements such as sensor values are not displayed unless the value is outside a predefined region; in the interrogation mode, information is available on demand, whereby the user can "interrogate" the system for sensor values and other information. Sometimes a third mode is available, wherein the user can configure one or more standard reports that will be automatically generated on a periodic basis. The reports may include graphs as well as tables.

It is not the purpose of this chapter to examine information management issues, other than to point out that the aforementioned questions must be answered before the monitoring system software for a management information system (MIS) can be selected. Careful attention to the questions of information need and usage will result in an accurate and comprehensive understanding of the mine's information needs. These can be fully documented; such a document will be most useful in developing the specifications for the proposed monitoring system.

The design and fabrication of mine monitoring systems is essentially the activity of manufacturing companies, not mining companies. Although a few mining companies have become involved in these systems at the circuit-board level, it is not necessary nor desirable for mining personnel to become involved in these details. However, if the monitoring system is to be well matched to the mining application, it is essential for mining personnel to clearly and accurately state the systems-level requirements to manufacturers and vendors. Toward this goal, it is helpful to understand the composition and function of the building blocks that constitute a monitoring system, and how the specification of these will directly affect the performance of the system.

12.6.2 GENERALIZED MONITORING SYSTEM

Fig. 12.6.1 illustrates the building blocks of a monitoring system insofar as a user is concerned. Each of these aspects will be discussed as they relate to the specification of a mine monitoring system, and those aspects that are appropriately addressed by the mining engineer are emphasized in the following discussion. It is also assumed that the specified components will be *mine-worthy,* a term suggesting that the component can withstand the harsh mine environment, including moisture, dust, temperature extremes, vibration, and shock.

12.6.2.1 Sensors

The choice of the right sensor is dependent upon an understanding of the property that is to be measured (the *measurand)* and the performance parameters of the sensor. Sensor selection is an aspect of the overall system specification in which the mining engineer should actively participate. In fact, many situations could be improved through monitoring if it were recognized that a wide variety of sensor types can be used with a mine monitoring system, even though they are not necessarily sold by the manufacturer of the mine monitoring system.

Characteristics of the measurand include the expected range of values, significant and insignificant variations that must be detected or ignored, respectively, and the presence of factors or compounds that might introduce error into the measurement or cause other problems. It is always best if the interaction between the sensor and the measurand and its environment are understood. Otherwise, in the case of a gas sensor for example, the presence of one gas in the environment may result in an erroneous indication of the gas that is of interest, or in some cases, it may even "poison" or ruin the sensing element. Taking the time to understand all aspects of the measurand can pay unexpected dividends. It may turn out that the property of interest can be sensed indirectly, and that the sensor for this secondary indicator of the measurand is more reliable or perhaps less expensive than a sensor for directly measuring the measurand.

The key performance parameters for sensors are accuracy, resolution, dynamic range, and stability. Cost is an important factor, especially if large numbers of a particular sensor must be purchased for many locations. For the purpose of this discussion, *accuracy* is defined as the deviation of the measured value from the true value; *resolution* is the smallest part of the whole that can be detected; *dynamic range* is the range of values over which the sensor can be used with the stated accuracy and without causing permanent damage to the sensor and is sometimes called *full-scale range* (FSR); *stability* is a measure of the sensor's tendency to drift, that is, for the output to change due to unrelated factors such as elapsed time or temperature.

It is noteworthy that these parameters are somewhat interrelated. For example as the sensor's accuracy is increased, the dynamic range will decrease, stability will tend to worsen, the cost will increase, and the resolution will probably increase. Oftentimes, accuracy is even given as a percentage of the full-scale range (dynamic range). Accordingly, it is important to match the performance parameters as closely to the application as possible, as illustrated in the following example.

Example 12.6.1. A current with an expected range of 90 to 120 A is to be measured. Information on available sensors is obtained, and the choice is narrowed to one of two models. The first has a rated range of 0 to 100 A with an accuracy of +2% of FSR; the second has a range of 0 to 500 A, also with an accuracy of 2% of FSR. Since the expected value of 120 A exceeds the FSR of the first model, the second model would normally be selected. The error associated with this model could be 10 A (2% of FSR), making the actual error as a percentage of the measured value around 10%. This might be satisfactory for some situations, but regardless, the actual measurement accuracy in this application will be five times worse than the mistakenly expected value of 2%.

Upon discussion with the manufacturer, it is learned that the FSR of either sensor can be doubled without harming the sensor, although the accuracy deteriorates to 3% at 100% overload. Given this, the first model could be used, and assuming a worst-case accuracy of 3%, the actual measurement accuracy would also be on the order of 5%, which is significantly better than that of the second model.

A published accuracy often reflects a fixed error or bias in the sensor in addition to a variable and sometimes random component. In marginal cases, this should be investigated; it is often possible to perform onsite calibration, resulting in significant improvement in the actual measurement accuracy as compared to the published accuracy.

Resolution is often given little consideration or is unneccessarily sacrificed for dynamic range.

Example 12.6.2. Suppose that a pressure sensor is to be selected to measure air pressure in automated pressure/quantity mine ventilation surveys. It is determined that a resolution of 0.2 in. water (49.8 Pa) is necessary. A piezoresistive pressure sensor is selected with a dynamic range of 0 to 15 psia (103.4 kPa), and an accuracy of 0.1% FSR with an output of 0 to 1 V. The transfer function is assumed to be linear and is 0.0024 V/in. (0.094 mV/mm); given an accuracy of 0.1 %, it is clear that the resulting error of 0.42 in. water (104.5 Pa) is larger than the desired resolution of 0.2 in. (44.8 Pa), although a workable resolution on the order of 0.4 in. water (99.5 Pa) would be possible, and for many applications satisfactory. However, a more serious problem may exist.

The monitoring system uses a 10-bit *analog-to-digital convertor* (ADC) with a FSR of 10 V. The resolution, *R,* of an n-bit ADC is given as

$$R = \mathrm{FSR}/2^n \tag{12.6.1}$$

which in this case is calculated to be 9.8 mV. Given the foregoing transducer specification, it becomes clear that the smallest detectable change, that is, 1 bit, is going to be 4.07 in. water (1013 Pa). If the accuracy of the ADC is a typical value of 0.1% FSR (for a 10-bit ADC), a worst-case error would be in excess of 4.5 in. water (1120 Pa). This is absurd, and yet installed systems can be found with such gross mismatches as attempting to resolve an air pressure to within 0.5 in. water (124.4 Pa) with an overall system limitation that is almost 10 times larger than that of the measured quantity.

There are, of course, a variety of options available to rectify this problem if it is recognized before components are purchased. A 12- or 16-bit ADC would be appropriate, as would a reduced FSR. However, in particularly demanding applications such as this one, additional steps would be required. By recognizing that the pressure sensor will be used only over a small part of its dynamic range, it is possible to amplify only that portion, resulting in a significantly larger swing in sensor output for a small change in pressure. A second stage of amplification prior to the input of the ADC may also be desirable. By fully utilizing the ADC's FSR, acceptable levels of both resolution and accuracy can be achieved. Generally, monitoring system manufacturers

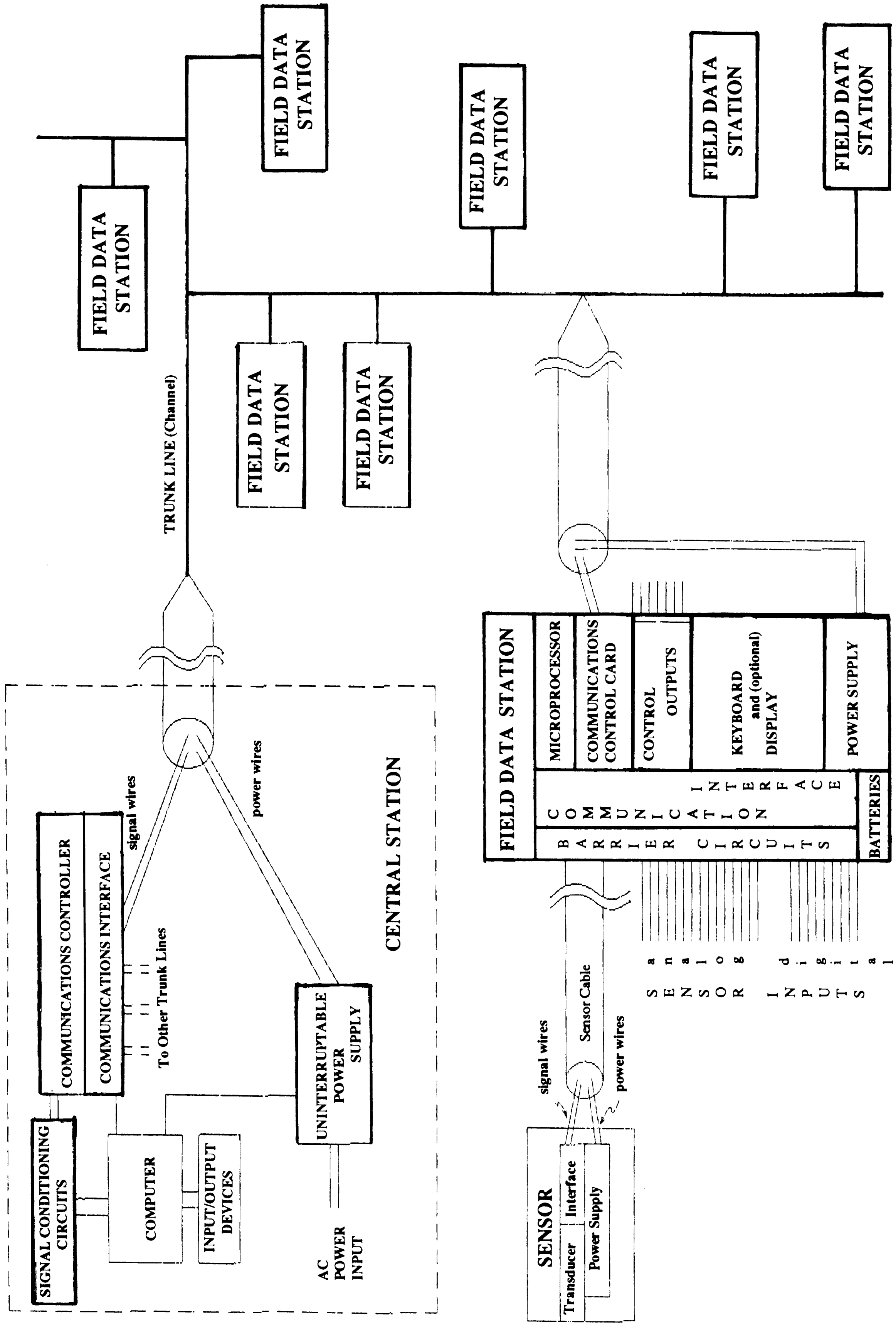

Fig. 12.6.1. Block diagram of a mine monitoring system.

can provide the type of custom interfaces and circuitry that would be needed here.

The latter problem illustrates the importance of matching performance specifications throughout the system, and calculating resolution and accuracy at all points in the system. Additional information on sensors, in general and for mining applications, can be found in the literature (e.g., Anon., 1986; Anon., 1988; Chilton and Cohen, 1983).

12.6.2.2 Interfaces

The function of an *interface* is to match the output of one device with the input requirements of the next device in the system to which it is connected. Just as a three-pronged electrical plug will not fit into a two-pronged socket without an adapter plug (interface), the electrical characteristics must be compatible at all points where two different components are connected. In addition, the sensor interface may be employed to provide general signal conditioning to improve overall performance. Interfaces maybe an integral part of a component, such as a sensor, or maybe installed as a separate device. Some common signal conditioning techniques are described here.

Attenuation or *amplification* of a signal to improve the signal-to-noise ratio, or to utilize the FSR of an ADC to improve resolution, as illustrated previously, is a common practice. The transducer output within a sensor may not be in the form desired for transmission of the signal. Voltage outputs are sometimes converted to frequency where the transmitted frequency is proportional to the magnitude of the original voltage. Such a scheme allows for a signal to be transmitted over long distances without suffering deleterious effects from attenuation. Another scheme more commonly employed to avoid problems from line loss is to convert the voltage to a current; this also allows the use of a non-zero baseline.

The 4- to 20-mA current loop interface transmits a signal of 4 mA when the sensor output is zero, and up to 20 mA for the sensor's maximum value. This non-zero baseline is particularly attractive in an application like mining when there is a risk that the transmission line between the sensor and the central station can be damaged. Consider, for example, a gas sensor that has an output of 0 V when the gas concentration is 0%. The monitoring system will register 0 V and report a gas concentration of 0%. Now assume a rockfall cuts the transmission line between the sensor and the central station; the monitoring system will still register 0 V. However, if a 4- to 20-mA current-loop interface had been used, the system would have registered 4 mA (or the equivalent voltage drop across a known impedance) when the sensor detected 0% gas, and 0 mA after the transmission line was damaged.

Isolation is another important function performed by an interface. Isolation is desirable when an event (normal or abnormal) on one side of the circuit could cause an undesirable event (ignition, shock hazard, failure, etc.) on the other side. Typically, isolation will be performed to isolate voltage levels and to achieve intrinsic safety; this is usually achieved by using contacts and, transformer or optical isolator circuits.

Contacts, either from a relay, as auxiliary contacts on a contactor, or as an inherent part of a sensor, can be checked for closure by the monitoring system. For example by detecting a set of auxiliary contacts on a motor contactor, such as for a belt drive, the status of the device in the power circuit is determined without connecting the monitoring circuit to the high-power circuit. In other situations it may be possible to insert a relay into a circuit: the relay coil, when energized, will open or close contacts that can be sensed by the monitoring system. As with the auxiliary contacts, information is obtained without directly sensing high voltages, and to the data-acquisition equipment anything over about 15 V may be considered high and potentially damaging.

Isolation circuits utilizing either transformers or optical couplers are also readily available and often used. However, optical couplers usually provide a higher degree of isolation and better frequency characteristics and would be used if high-frequency information were of interest. Transformer-based isolation circuits are satisfactory if only increased reliability is desired. For example, if a control circuit for a machine is being monitored, a failure in the monitoring circuit might affect the machine's control circuit, thereby idling the machine; in these cases, transformer isolators can be used to insure that this will not be a problem.

If a sensor is going to be located in an area of the mine where there is concern that an ignition may occur, or in an area where permissible equipment is required, then a different kind of isolation may be needed. Circuits, known as barriers, are often used to prevent potentially dangerous levels of energy from moving from the monitoring-system circuitry to a connected sensor.

Outstations contain sufficient energy to cause an ignition and, accordingly, are normally not located in areas that require permissible circuits. The sensors connected into the outstation, however, may be located in hazardous areas and, even though the sensor may be permissible, a failure inside of the outstation could allow sufficient energy to reach the sensor to cause an ignition. To prevent this, the wires going to the sensor have to pass through an intrinsic-safety barrier strip. These barriers contain circuitry that limits the energy to non-incendive levels, by clamping the voltage and limiting the current. The class of the barrier strip must be matched to the operating characteristics of the sensor to achieve satisfactory performance.

A separate but related issue is the permissibility of the sensor, if it is going to be located in a hazardous area. If the sensor is intrinsically safe (i.e., under any normal or abnormal mode of operation, it cannot produce an incendive arc or spark in a methane-air mixture), and if it has been certified by the Mine Safety and Health Administration (MSHA) as permissible (legal for use in gassy mines), then the sensor may be located in the restricted area of the mine. Nonetheless, barriers will still be required, in virtually all cases, because a failure in the monitoring system's circuits could allow incendive levels of energy to pass to the intrinsically-safe sensor. Some sensors cannot be made intrinsically safe, because, to operate, they require more than 0.3 mJ of energy, which under certain conditions could cause an ignition. For these, part of the sensor must be placed inside of an approved explosion-proof (XP) enclosure.

Filtering is another signal conditioning technique available to improve the quality of the signal. The most common application is to remove unwanted signals that are interfering with the legitimate signal. Commercially available filters include lowpass, highpass, bandpass, and notch filters. Lowpass filters pass frequencies below a specified value and attenuate frequencies above it; highpass filters attenuate frequencies below a specified value; bandpass filters pass only frequencies between two specified values; and notch filters pass everything except for the frequencies between two specified values. Lowpass filters should be used when a Fourier transform will be performed on the signal, for example, in a predictive maintenance application, to prevent aliasing; another application is to use a 60-Hz notch filter to eliminate this component from sensor signals.

Other signal conditioning techniques are employed, but the ones described are the most likely to be encountered in a mining operation. Some techniques that could be grouped here, such as analog-to-digital conversion, encoding, and modulation, are discussed in the next segment.

12.6.2.3 Communications

The communication blocks in Fig. 12.6.1 facilitate the interchange of data between different parts of the monitoring system. The choice of communications parameters in the system will directly affect both the quality and quantity of information that passes through the communications channel. *Channel* is a general term for the medium through which the signal is sent; channels may be either wired or wireless. Data communication are emphasized in this section, although voice communication is merely a special case of data communication.

The central issue in communications is preserving information in a signal. During a telephone conversation when there is a "bad connection," one experiences some difficulty in trying to hear and understand the other person. When the connection deteriorates to the point where information is lost or misinterpreted, a real problem exists. The same is true in data communications. To prevent these problems, it is necessary to understand the nature of the noise (unwanted interference signals), the source of the noise, and the technology that can be employed to overcome the problems. Fortunately, this is an area that has been investigated for in-mine applications and is well understood (Bredeson, Kohler, and Singh, 1981).

Many of the potential problems with noise are circumvented by using a *digital communications scheme.* In such a scheme, the analog signal, which is a continuous function of time, is digitized using an analog-to-digital convertor. As a result, the original signal is represented by a string of "zeros" and "ones," which is then transmitted. To illustrate the simplified task of accurately receiving the transmitted state, assume that a "zero" can be considered as any pulse less than 1.5 V, and a "one" as any pulse greater than 3.5 V. Then a 5-V pulse will be transmitted for a "one" and no pulse for the digital "zero." Obviously, these pulses would have to undergo extensive noise corruption before it would be impossible to differentiate between the two allowable states. An analog signal, however, loses information with the least amount of noise. The quality of digital communications is further enhanced by utilizing a transmission scheme known as *frequency-shift keying* (FSK) in which the amplitude is constant but the frequency is changed to correspond to a binary 1 or 0; this is particularly useful if the distances involved are large. Regardless of distance, however, this scheme is used almost exclusively in mine monitoring systems for communication between the outstation and the central computer; communication between the sensor and the outstation is usually accomplished with an analog scheme.

The digitization of the analog signal by the ADC can be accomplished by a number of means. The successive-approximation technique is quite common and is well-suited for mining applications; integrating convertors are less expensive, but the longer conversion time is often unacceptable. The ADC's specifications will impact significantly the information-collection capability of the monitoring system. While many of its characteristics are of more concern to the manufacturer than to the user, there are some factors with which the user should be familiar. The number of bits, as indicated previously, is important because it determines the system resolution; the settling time and conversion time of the ADC may limit the sampling rate, which will limit the number of channels that can be scanned in a certain time period. Finally, the accuracy of the ADC must be considered in determining the overall accuracy of the system.

Wired channels are commonly used as the transmission medium, although wireless channels are particularly suited to receive signals from mobile equipment or from non-mobile equipment over short distances where it would be cumbersome to run a signal cable. Frequency-modulated (FM) transmitters are often used for this purpose. The selection of a wired-channel transmission medium is based on a number of factors, but for mining applications, twisted pair is satisfactory for most applications, except when a larger bandwidth requires the use of coaxial cable. Large and sophisticated applications in the future will probably utilize optical fiber data links. Although these are even finding some application in the mineral industries today, extensive use will come only after further reductions in the cost of the technology.

A number of decisions must be made in the selection of a cable for the mine monitoring system, but it first should be recognized that it is not just "a cable." Many problems have occurred in too many mine monitoring installations where the company settled for low-cost or surplus telephone cable. If funds for the proper cable are not available, then the monitoring system should not be purchased.

The physical configuration of the cable used for the monitoring system varies depending on the type of monitoring system, as well as the mine's requirements. A simpler cable can be used when it is unnecessary to provide power to the monitoring system through the cable. A more sophisticated, and relatively unpopular, configuration consists of insulated signal conductors, insulated power conductors, a bare ground wire, metallic shields, and possibly a messenger wire. Filler materials are added and these components are cabled and covered with a jacket or sheath.

The signal conductors almost always consist of twisted pairs of wire. Each twisted pair will carry the signal to or from one actuator or sensor. Each pair may be enclosed by a metallic shield, and cables with varying numbers of twisted pairs can be purchased. The power conductors, if needed, provide power to sensors. The signal and power conductors are stranded to increase their flexibility. The purpose of the steel messenger, when used, is to carry the weight of the cable; when it is suspended, very little stress is imparted to the signal or power conductors. Further, it provides a convenient place to attach the hardware that is used to hang the cable. The whole assembly is enclosed in a jacket which provides moisture resistance and reduces the chance of abrasion damage to the conductors.

The following parameters should be considered in the selection of a cable: losses, number of conductors, and shielding. The signal-conductor size must be large enough to minimize the resistive voltage drops that will occur over the length of the longest-expected trunkline. Usually, a 14- or 16-gage wire is preferred for its mechanical strength; these sizes are more than adequate for the signal lines. A second component of signal loss is due to the distributed capacitance of the signal wires. Over several thousand feet (meters), this component can become a limiting factor. Therefore, dielectric materials with relatively low capacitance (e.g., 25 pF/ft or 82 pF/m) are preferred. If power conductors are used, the voltage drop can become a limiting factor that should be checked. The monitoring-system manufacturer can provide the appropriate information to correctly size the conductors.

Shielding is used to reduce the reception of electromagnetic noise on the signal conductors by surrounding them with a grounded metallic covering. However, for mine monitoring applications, this practice is no longer recommended, except under very special circumstances, due to problems that develop through ground loops. Ground loops occur whenever a shield or ground wire becomes externally grounded at points that are physically separated. This can occur at a sensor, if it has a single-ended output or if one side of the differential output has been tied to a chassis or ground; it may occur within outstations and at the central station. As a result of ground loops, noise pickup can be increased dramatically, and chassis grounds can become elevated, causing a myriad of problems with ADCs and microprocessors. Since these ground loops can be difficult or impossible to prevent in an extensive system, recent experience in mining suggests that it is better to use unshielded twisted pairs, especially with the higher common-mode rejection available at the system inputs.

The purchase of a cable with surplus twisted pairs is often a good investment. Additional pairs are, on an incremental basis, inexpensive and offer considerable flexibility for future expansion—further, they make it easier to maintain the link over time. From a reliability point of view, the use of the embedded messenger is thought also to be desirable, as mentioned before. Unfortunately, there are no data available to support this position, and the incremental cost of including a messenger with an external sheath is significant. Few mines utilize a messenger.

The use of a twisted pair of wires to connect each sensor to the central-station computer is impractical except in small or local applications. The cost of installing the communication channel (the wires) throughout the mine is a major expense in implementing mine monitoring and, in fact, is an impediment to instituting monitoring in large existing mines. Thus various multiplexing schemes are used to allow information from multiple sensors to travel over the same twisted pair.

A *space-division multiplexing* (SDM) scheme is one in which separate wires are run for each sensor. This approach is appropriate when the distance is on the order of a several hundred feet (few hundred meters) or less, and when the sensors are not located along a straight line. It is a simple and straightforward scheme but too expensive to use over long distances. *Time-division multiplexing* (TDM) is a scheme where the sensors or other sources of information take turns utilizing a twisted pair. This is an economical system with few disadvantages other than speed. As the number of information sources becomes larger, it will take increasingly longer to query the sources. This time is quantified as the polling rate, and in larger systems this rate would become unacceptably long (on the order of minutes) if TDM were used exclusively.

Frequency-division multiplexing (FDM) offers the continuity of information flow and speed of SDM, with the use of only one twisted pair as in TDM. This is accomplished by partitioning the available frequency band and then giving each information source its own frequency band. It is necessary, of course, to utilize slightly more complex electronics to accomplish this. However, the cost of the signal-controlled oscillators and related circuits is more than offset by the reduced cost of cable and the improved performance.

An extensive monitoring system will typically employ all three forms of multiplexing. SDM is often used to connect sensors at outstations. FDM is usually used to communicate with an outstation; sometimes each outstation is interrogated by the central computer using a TDM scheme. If the monitoring system uses multiple trunklines, it would not be unusual to find that TDM was being used to access each individual trunk.

While many of the parameters that define data communications within the monitoring system are transparent to the user, such as those related to improving the signal-to-noise ratio or data security schemes, there are others that are of concern to the user. The minimum amount of time required to sample each sensor is of concern since this will determine the total time required to scan all of the sensors. In some schemes all sensors are effectively off-line when the system is scanning, except for the one that is currently being polled. In other systems, if a sensor goes into an alarm condition, it will interrupt the normal polling sequence so that this condition will be immediately known at the central computer. The *polling rate* or number of sensors sampled per unit time and the interrupt scheme must be commensurate with the needs of the user.

The *sampling rate* per sensor (the number of samples taken to define one reading) and the use of these samples (e.g., arithmetic averaging, etc.) should be defined. The system may take a sample in a few milliseconds. Thus, if there are time-dependent variations in the sensor signal, as is often the case with atmospheric parameters as well as others, then misleading readings will result. The ability to take multiple samples at the time of polling is useful because time-averaged values can then be calculated and displayed.

The sampling rate is also of concern if frequency information must be extracted from the measurement. In this case, the sampling frequency must be theoretically twice the highest frequency to be resolved, but practically it must be greater than the theoretical minimum. Mine monitoring systems, by their nature, are not well suited to sampling individual channels at high rates, particularly if there are a number of channels with this requirement. A better alternative is to use a dedicated microprocessor to sample at the high rate, perform the desired computations, and then output the result to the mine monitoring system data bus. Essentially, the microprocessor behaves like a *"smart sensor,"* and is treated simply as a sensor by the monitoring system.

Another consequence of the communications structure used by the manufacturer is system performance if the communications trunk line is severed. With some schemes, only those stations inby the cut will be lost. In other schemes, all stations on the trunk will be lost. A discussion of the technical alternatives is beyond the scope of this chapter, but the practical ramifications of the communications architecture is not. In addition to the aforementioned consideration of system performance when the trunk line is cut, the user should also understand, before purchasing a system, any limitations on the number of devices that can be connected to a trunk line and the maximum distance permissible between any two outstations or an outstation and the central station.

12.6.2.4 Outstations

Outstations or *field data stations* represent a point of aggregation in a hierarchical structure of data flow. They serve as an economical compromise by allowing access to the communications trunk by a number of sensors. Finally, they are convenient locations to provide local monitoring and control. The latter can be particularly crucial if there is a failure of the mine-wide system. The major function, however, is to provide a communications interface between a sensor and the trunk connected to the central station computer. Outstations cannot be located in areas that require permissible equipment, unless they have an MSHA approval number, which would be quite unusual. However, *blue outstations* can be used with sensors that are located in areas that require permissible equipment, whereas *red outstations* cannot. Intrinsically safe and permissible systems can be obtained, but they normally do not use the outstation-type architecture. An intrinsically safe system would have some advantages in gassy mines (Watson, 1982). Fig. 12.6.2a illustrates a typical outstation used with a mine monitoring system. The barrier strips, required in a "blue" outstation, can be observed in the figure. An outstation installed in an underground coal mine is shown in Fig. 12.6.2b.

It is possible to add a communications interface right into a sensor, thereby eliminating the need for an outstation. Such a sensor with a built in accessor card is shown in Fig. 12.6.3. If there are several sensors in close proximity, such an approach is not cost-effective; if there are many instances of isolated locations with only a few sensors, then elimination of the field data station would be cost-effective. However, it is usually not possible to mix and match within a given system: either the system is designed to

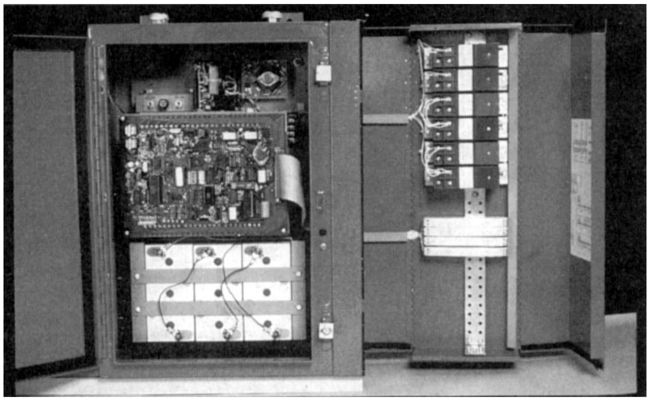

Fig. 12.6.2a. A field data station with the cover open to show the circuitry and barrier devices (*courtesy:* National Mine Service).

Fig. 12.6.3. Photograph of a CO sensor with built-in accessor card (*courtesy:* Conspec Controls, Ltd.).

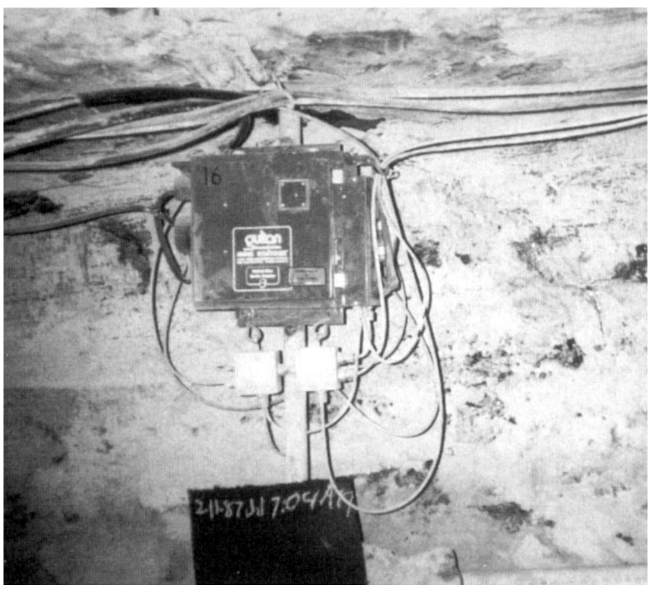

Fig. 12.6.2b. A field data station installed in a mine.

use outstations or it is not. In general, more sensors can be used on a trunk if outstations are employed than if they are not. This limitation, due to the number of unique addresses that can be defined in an n-bit communications signal, is rarely a problem, but should always be checked, especially in the larger and more comprehensive systems. Sometimes it is necessary to run multiple trunk lines in extensive installations. Intrinsically safe systems have been manufactured using the accessor-card type architecture, rather than the outstation type.

12.6.2.5 Central Station

This station, regardless of what it is called, contains the main monitoring system computer(s), terminals, data storage devices, and sometimes other components such as uninterruptable power supplies (UPS) and so forth. Often this area is staffed around the clock and may also serve as a communications center for the mine. A basic workstation in the central station will consist of one computer that is usually the equivalent of a personal computer, a keyboard, a monitor, a printer, and a UPS. Typically, the basic workstation is upgraded to include additional computers and monitors. Industrial-grade components should be used whenever available. Rather than review all of the components of a central station and the various options, all of which can be obtained from system vendors, the emphasis here will continue to be on specific aspects that should be given special consideration but are sometimes overlooked.

The availability of service and spare parts is always considered in the purchase of mining equipment but not always in the purchase of monitoring equipment. Yet, over time, these systems become just as indispensible to the operation. Some monitoring systems heavily utilize “off-the-shelf” hardware and software. Generally, this translates into better than average availability of parts and support if needed. On the other hand, systems utilizing “custom” processors may offer better performance; in some cases, this may offset any inherent disadvantages of the custom processor. Of course, the requirements document, discussed earlier, can again be consulted to determine if the additional performance is significant in the proposed installation. A backup computer was often included in early monitoring installations. However, with the higher reliability units available today, combined with the availability of replacement boards, it is seldom necessary to purchase a backup system. An inventory of spare boards is sometimes recommended.

The number of terminals available at the central station should be at least two and preferably three. One terminal is always needed to track alarm conditions; a second can be used to display specific graphics related to areas of interest, such as the belt system and the status of all sensors. A third terminal can be useful for displaying additional information or in performing off-line development and maintenance of system software, assuming this is supported by the monitoring system software. Naturally, if the utility of the system is to be realized, then the system should be able to support multiple terminals in remote locations. This would allow a terminal in the engineering office, the superintendent’s office, and so on. These locations would be determined through the information definition stage outlined previously. Most systems allow for the establishment of various access levels and security codes so that the information accessible by any user is easily controlled.

The usefulness of a monitoring system is determined, in part, by the usefulness of its displays. The display should contain as much relevant information as is practical, and this information must be displayed in a format that is clear and easy to understand. Achievement of this is quite a challenge, though advances in graphic displays (e.g., resolution, size, speed, software support) have permitted display formats that were impossible only a few years ago.

Color graphics allow an order of magnitude increase in the information that can be presented on a single display, and they are easier to interpret as well. Systems that easily allow the user to create custom maps and displays are desirable. In too many instances, however, complex graphics are used to display trivial or simple pieces of information, thereby unnecessarily tying up system resources. In other cases, systems are selected based more on the attractiveness of the graphics than the performance of the system.

The individual displays, including maps, flow charts, diagrams, and tables, will have to be created at the mine site, usually by engineers with little specialized training in computer software. Therefore, it is important that the graphics software provided with the system be suitable for the intended application. The suitability of a graphics package can be evaluated by determining the ease of creating a new display, the ease of modifying an existing display, the total number of display screens allowed, and the number of active elements that are allowed per screen (an active element will display the value of one sensor). The maximum requirement for active elements generally occurs on either the display for a diagram of the ventilation network or the belt system, where it is important to simultaneously show several sensor values.

It is not essential that an environmentally isolated and conditioned space be provided for the system in all cases, but a relatively dust-free and air-conditioned room will reduce failures and heat-related shutdowns. It is essential that the system be supplied with a quality power source; in most cases, an uninterruptable power supply (UPS) that is often required anyway, will satisfy this need.

Finally, it is desirable for the user to be able to set up data files and easily access these from another computer in order to facilitate analysis and use of the collected information. This cannot be accomplished unless the monitoring system stores files in a manner that allows retrieval by another computer system, such as a DOS-based personal computer. A personal computer, interfaced to the monitoring system computer and used as a file server, would certainly facilitate use of the collected information. A central station computer system, representative of the minimum required, is shown in Fig. 12.6.4. The next acquisition for the illustrated station would be an additional monitor and possibly another processor.

12.6.2.6 Control

Mining applications involving control are relatively rare (remote control of mining machines, automation, and robotics is excluded here and is covered in Chapter 22.2; process control is also excluded and discussed in Chapter 25.3). This is in part because of the resistance of regulatory bodies to recognize new technology and allow its application, and is also due to the absence of a strong need for such control. Currently, control in an integrated monitoring and control system consists solely of straightforward on/off control of equipment. Notable applications include drive motors for materials handling equipment, switchgear, and pump motors. An application offering promise is energy management, and it is believed that more mines will adopt this technology as it becomes better known (Toni, 1988).

Mine ventilation offers several notable opportunities for control. In a general sense, the control objective would be to exactly match the mine's need for air with the output of the mine ventilation system. Technologically, this can be accomplished by controlling regulators and the mine fan (Aldridge, 1976; Kohler, 1986); however, a change in the statutory requirements will be necessary before it will be possible to alter the ventilation network while workers are in the mine. Under less restrictive legislative environments, the development and implementation of ventilation control has been reported (Ching, 1987).

Most commercially available mine monitoring systems offer a control option. Typically, there are options for digital and analog output. A digital output might be used to control a device, such as a pump that has a built-in digital speed controller or any other similarly equipped device. The analog output might be used with a proportional controller or other device that responds to the amplitude of the control signal. For most applications in mining, however, it is only necessary to control a relay or contactor. Manufacturers provide relay boards that have a number of low-power contacts, each controllable by the software, that can be used in many control applications. If large amounts of power must be controlled, these contacts would be used to energize larger relays better suited to this requirement.

Programmable controllers (PC) have served, over the years, as a crucial evolutionary step between hard-wired relay logic and computer control of industrial circuits. PCs have undergone quite an evolution in which powerful microcomputers and sophisticated software have been incorporated into them. Today, it is possible for a user with a modest knowledge of the technology to custom configure a mine-wide monitoring and control system using PCs and associated devices. In certain situations, doing so may offer distinct advantages over purchasing an off-the-shelf system. Of course, such an approach could also be fraught with problems. Most commercially available systems can be interfaced to a PC. Thus local and specialized control loops are possible and straightforward to achieve. Typically, such control applications would be characterized by a need to monitor multiple variables, process the information in a timely fashion, and assert control on one or more actuators.

12.6.2.7 Standards

There are few standards to govern the construction of mine monitoring systems. MSHA controls aspects of the system design and construction that are related to permissibility. Some state mining agencies require an approval code for use of the systems within their states, but none have any specific criteria for approval. Some European countries have specific standards for monitoring systems, but again these apply more to permissibility issues than performance. The Institute of Electrical and Electronic Engineers (IEEE) has adopted and published a Recommended Practice for Atmospheric Mine Monitoring Systems, Std. No. P1203-1990. This document provides a wealth of information on features which should be a part of any monitoring system and even includes some information on recommended installation, maintenance, and operation practices.

12.6.3 REPRESENTATIVE APPLICATIONS

An overview of some applications in mine monitoring will serve to illustrate the potential of this technology as well as current practice; many others can be found in the literature (e.g., King and Eros, 1988). The most prevalent application is fire monitoring, typically using carbon monoxide sensors, in belt entries. Interest in this application is a result of the MSHA's

Fig. 12.6.4. A typical workstation for a mine monitoring system. (*courtesy:* Mine Safety Appliances Company).

requirement that a fire monitoring system be installed if the belt entries are to be used for intake air. This requirement resulted in the installation of many mine monitoring systems that otherwise would never have been purchased. As the technology developed and more systems were installed, companies looked for ways to more fully utilize the capabilities of monitoring systems. As a result of this effort, the second most frequent application is now conveyor-belt monitoring. Other less common applications include monitoring of production, pumping station parameters (both degasification and water), rock parameters, and equipment for predictive maintenance purposes. The results of a survey to determine the types of monitoring that are perceived to be important by managers of underground coal mines is shown in Table 12.6.1. The survey results shown in this table have been augmented to show the approximate percentage of installed monitoring systems that is being used for that purpose.

It is clear from Table 12.6.1 that companies are installing monitoring systems to satisfy perceived needs for information. (The only exception in the table is for production monitoring; this is because the technology to monitor machine state and related information has only recently been demonstrated.) Justifying these perceived needs is not often an easy task. Certainly if the installation of an atmospheric monitoring system is prerequisite to the use of belt air to ventilate a coal mine working place, then a cost-benefit analysis can be done, and the benefit of the monitoring technology can be demonstrated. In the case of a monitoring system to detect problems in the materials handling system, the monitoring system's economic benefits can be related to delay costs. For many other applications, the benefits may be difficult to quantify in a rigorous analysis. It is in these cases that the engineer must apply judgment and intuition to evaluate the worth of an investment in the technology.

Table 12.6.1. Mine Monitoring Applications Ranked by Decreasing Management Interest, Showing the Relative Number of Systems with that Capability

Application	Percentage of Installed Systems[a] that Perform this Application[b]
Production Monitoring	0 %
Materials Handling	45 %
Ventilation and Environmental	85 %
Maintenance	< 2 %
Power	10 %
Geomechanics	5 %

a. Based on 110 monitoring systems installed and operating in US mines in 1988. (Some data for this table provided by the US Bureau of Mines and MSHA.)
b. A mine-wide monitoring system is often used for multiple applications.

The examples of monitoring applications that have been included in the next part of this chapter are not meant to be exhaustive in coverage but rather to illustrate relatively common and successful applications of the technology, and in a few instances, to suggest extensions to current practice. It is always helpful to bear in mind a few often overlooked aspects of monitoring systems: (1) the monitoring system should be kept as simple as the application will allow; (2) once a system is installed, it will cost little extra money to add additional sensors, up to the systems capability; and (3) the types of things that can be monitored are limited more by one's own ingenuity and creativity than by technology.

Suppose that rail cars are to be counted at an automatic underground tipple. After reviewing the vendor catalogs, it is found that no one sells a "rail car counter." Thus it will be the engineer's responsibility to devise a sensor. The first step is to identify all of the unique aspects that characterize the event that is to be detected, in this case the number of cars that have been loaded. Next the fundamental parameters (e.g., pressure, displacement, event, current, concentration, etc.) that are associated with the event and for which transducers can be purchased must be noted. Then one of these parameters would be selected based on transducer cost, reliability, and implementation considerations. Finally the appropriate interface card would be purchased from the monitoring system vendor to allow connection of the sensor to the system.

For this example, a number of sensing options are available to determine the number of cars that have been loaded. These include weighing the coal, assuming a relatively constant capacity per car, or detecting an interrupted light beam when a car moves into position. A choice between these two alternatives is simple. The optical sensor employing the light beam will be far cheaper and probably more accurate than a weight sensor implementation. An infrared beam would provide better service than one in the visible spectrum due to the expected interference from airborne dust in the vicinity of the tipple. Still the sensor will require frequent attention. Another alternative is to sense a pressure change in the hydraulic system of the car spotter; assuming constant displacement each time the spotter is activated, this will be an inexpensive and accurate means of determining the number of cars. Further, it would require no routine cleaning or adjustment. In addition to a pressure transducer, a threshold detector with a binary output would be required at a comparable cost to the optical sensor. The key point is that a parameter or event can often be detected through many different means.

12.6.3.1 Atmospheric Monitoring

Monitoring of environmental and ventilation parameters is practiced in mines throughout the world. Carbon monoxide is a good early indicator of a fire, carbon dioxide of spontaneous combustion, and methane of an ignition hazard. A variety of other gases, including hydrogen, oxides of nitrogen, hydrogen sulfide, and oxygen, may be of interest in a particular instance. In addition to the use of this information in setting an alarm condition, it can be used for planning purposes and to mark the occurrence of events such as gas outbursts.

Ventilation parameters, including airway pressure and airflow velocity (which may be multiplied by cross-sectional area to obtain quantity) and fan parameters can be monitored and used for a variety of purposes. Real-time pressure/quantity surveys are technically and economically feasible but rarely incorporated in present day monitoring systems. Taken individually, velocity (quantity) and pressure can be used for both alarm and planning purposes. Ventilation control devices such as doors and regulators could be monitored if the position or status of these were of interest.

A wealth of useful information can be obtained by either adding sensors to the fan and connecting them into the mine monitoring system or by using a separate fan monitoring system and interfacing it with the existing mine monitoring system. Information on pressure and quantity is useful, as is information on power consumption. Fan speed can be sensed as well as blade pitch. A stall condition, or the conditions leading to fan stall, can be detected. Finally, predictive-maintenance monitoring should be performed at the fan. Present technology supports detection of bearing problems and mechanical unbalance as well as a number of incipient mechanical and electrical problems in the drive assembly.

A significant aspect in the implementation of atmospheric monitoring is sensor location. There are two issues: (1) strategic placement of sensors at locations in the ventilation network to satisfy information objectives, and (2) local placement at a desired network location to satisfy technical constraints. A methodology for the strategic location of sensors is necessary to minimize the number of sensors necessary to achieve a certain level of information; such a methodology is discussed by Cohen et al. (1987). Technical placement at a specific location is necessary to insure that the sensor reading is representative of the actual quantity. Since the sensors are of the single fixed-point variety, this is a concern. Furthermore, some sensors may give erroneous readings depending on their placement in the air course. A complete discussion of these, along with recommendations for placement, can be found in the literature (Kohler, 1987; Kohler and Thimons, 1987).

Atmospheric monitoring systems have been in use for three or more decades. Simple and early forms consisted of little more than remote telemetry systems to monitor an isolated mine fan. Despite the simplicity of these early systems, they performed an important function. Today mine-wide systems are becoming routine in new mines, and systems are being installed in many existing mines. The monitoring of CO and CH_4 accounts for the majority of system usage, although there is increasing interest in adding other parameters to the system. As incentives for monitoring are incorporated into mining regulations, this trend will be hastened.

For more detailed treatment of computer applications for monitoring in mine ventilation, see Chapter 11.10.

12.6.3.2 Materials Handling Monitoring

The most readily observed benefits of mine monitoring have occurred in this area, and the greatest benefit has been the immediate identification of a belt conveyor delay. By knowing the location and cause of a belt shutdown, it is possible to dispatch the appropriate personnel to the problem drive or location without losing time to determine the location and cause. An unexpected benefit that also occurs in some cases is that the record of delays tends to highlight problem areas, such as certain transfer points, that should be corrected. Munz (1988) discusses specific considerations for belt monitoring.

An analysis of materials handling problems that occur during peak-load periods should include a study of the material flows over time. Often it will be found that additional system capacity can be obtained by introducing small delays into the system. This can be accomplished by utilizing variable-speed drives in which the speed is controlled by a feedforward monitoring and control system. Such an alternative can be particularly attractive in a large system compared to increasing capacity by increasing the belt size. Other materials handling applications include skip and rail monitoring and automated tipples used in both surface and underground mines. Truck dispatching in surface mining operations has sometimes included an automated monitoring component, such as keeping track of the number of loads of ores of various grades, and automatically directing the transportation of these depending on the needs of the concentrating plant.

12.6.3.3 Maintenance Monitoring

Many predictive maintenance measurement techniques are well suited for continuous monitoring applications (on-board machine diagnostic monitoring is not considered here). Further, the use of such techniques on a continuous, rather than periodic, basis allows for more accurate detection of incipient problems and eliminates the chance that the periodic measurement will be overlooked. In particular, vibration, temperature, and current sensors can be used effectively to detect problems in time to schedule corrective actions, thereby eliminating the costs and hazards associated with an unexpected failure and the associated emergency maintenance actions. Typical candidates for vibration and temperature measurements are large drives and fans; in some cases, large pumps might also qualify. Large or crucial motors would qualify for current sensing.

The use of vibration data to detect bearing deterioration, coupler or shaft misalignment, and balance problems is well known. This type of monitoring is easily accomplished with a mine monitoring system, provided that a satisfactory sampling rate (typically, a minimum of 5 kHz) can be achieved and that an FFT algorithm can be obtained for the system. Temperature sensing is useful for detecting bearing deterioration and lubrication problems. Current sensing is somewhat useful if magnitudes are trended over time, but other more useful techniques have been developed that detect incipient motor failures (Kohler, Sottile, and Trutt, 1989).

12.6.3.4 Production Monitoring

A method of economically monitoring production-related information (machine state and raw tonnage) has been developed for use with existing mine monitoring systems and is described by Kohler and Gentzler (1989). The production monitoring system consists of a microcomputer and voltage and current sensors that are located in the load center. The machine states are determined for each machine; the machine identification number, machine state, and time are then transmitted over the existing mine monitoring system channel to the central station computer. The collected information is useful for checking the status of equipment on any section and identifying related problems, and is particularly useful for tracking and scheduling preventive-maintenance operations based on actual hours of operation. The accumulated information can also be used for planning purposes because of its similarity to time study information.

12.6.3.5 Electrical Power System Monitoring

The monitoring systems at some mines have included some power system parameters. Most commonly, the status of crucial circuit breakers in the distribution system were monitored. This yields much the same benefit as monitoring belt conveyor drives. Today mine power equipment such as sectionalizing switchhouses and load centers is being manufactured with built-in monitoring and control interfaces so that a number of parameters can be monitored in addition to the breaker's status. One such equipped load center is shown in Fig. 12.65. In many cases, the circuit breakers can be controlled remotely as well. Voltage, current, and power factor can be monitored at key locations; such monitoring provides important information for planning and diagnostic purposes.

Energy management systems have been commonplace in commercial installations for many years; manufacturing and other industries have also embraced this technology. Even some mining companies in other countries have installed such systems, and yet these are virtually unheard of in the US mining community. These systems have the potential to reduce power costs per ton in excess of 10 to 25% (Toni, 1988). An energy management system consists of sensors that measure a parameter that can be related to power consumption, a processor that can interpret the measurements and implement a control strategy to shed loads, and control devices that can automatically take loads off-line according to the predetermined strategy. It is even possible, and sometimes feasible, to utilize manual intervention, such as a call from a dispatcher, to shut down certain loads.

The inherent justification for an energy management system in mining is the demand charge levied by the electric utility (for a discussion of demand, see Croyle et al., 1987). As a result of the mechanics for computing this charge, a power peak within a 15-minute interval can cost thousands of dollars in penalty fees for one month and may even establish a minimum penalty that will be assessed every month over the next year. Such peaks can be avoided by means that have little or no effect on production. Shutting off a large pump for a few minutes, delaying a load of cars on the slope, or turning off electric heaters in the plant for a few minutes are just a few examples of actions that can result in savings of several thousand dollars per month. A case study for an underground coal mine and preparation plant is presented by Toni (1988).

12.6.3.6 Other Applications

Other less common applications include microseismic monitoring to facilitate prediction of coal bumps or rockbursts, water level monitoring in sumps, and monitoring of methane drainage equipment. Once it is recognized that the mine monitoring system is really a powerful data-acquisition system that can accept many sensor types, then the possible use of monitoring to solve a host of problems will become clearer.

12.6.3.7 Smart Sensors

Certain monitoring applications have special requirements that may restrict their implementation with a mine-wide monitoring system. These special requirements could include the need for sensors that cannot be directly connected to the system, the need for high sampling rates, or the need for large computation resources. Examples of such applications have been mentioned earlier, and include production monitoring, microseismic monitoring, and condition monitoring. When this situation is encountered, it is best to treat it as a separate monitoring system that will interface to the mine-wide monitoring system. This separate system will be addressed by the mine-wide system as though it were simply a sensor, hence the designation *smart sensor*.

Usually the smart sensor consists of a few sensors, signal-conditioning electronics, a microcomputer, power supply, and a communications interface, mounted in an enclosure that is appropriate for the application. In certain cases, local input and output devices may be supported or even included. Regardless of the specific application and hardware used to satisfy the application requirements, smart sensors function in a similar manner. They sample data and execute computational routines, therein relieving the mine-wide system of the necessity to provide these system-intensive services. The smart sensor then makes available key results to the mine-wide system. These can be available on a periodic basis whenever the mine-wide system interrogates the smart sensor, or the user may choose to have the smart sensor report to the mine-wide system on an exception-only basis, that is, when an alarm or otherwise notable event is detected.

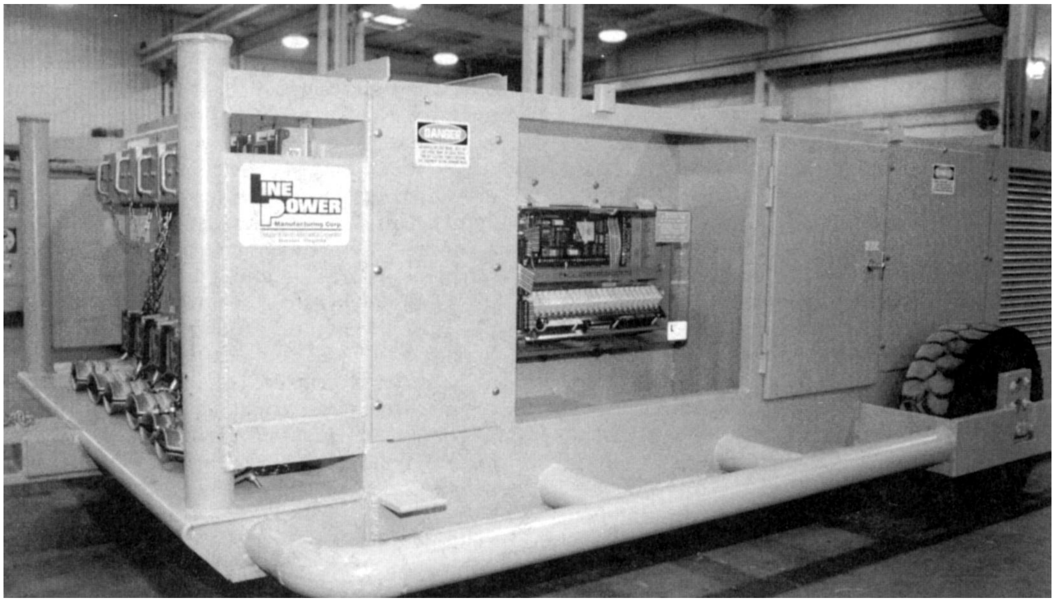

Fig. 12.6.5. Load center with built-in mine monitoring system interface (*courtesy:* Line Power Manufacturing).

12.6.3.7 Monitoring System Costs

The exact costs of a mine monitoring system are difficult to quantify given the almost infinite combination of features in a system as well as the differences in specific mine application sites; further, few people have kept records that would reflect their costs to install and maintain a system. Nonetheless, it is possible to estimate levels-of-effort and costs based on the industry's experience with the technology.

The purchase cost of a system, that is, sensors, outstations, and central station equipment, can range from $15,000 for a small system with a limited scope of application, such as only fire monitoring, to well over $250,000 for systems in larger mines that include a variety of monitoring functions, such as atmospheric, materials handling, and power, among others. Systems can be expanded— many times, a central station and a few outstations are installed, and then additional sensors and outstations are added as needed and as the budget permits. Quite useful systems that service the mine's needs now, but permit future expansion, can be purchased for $25,000 to $50,000.

Table 12.6.2 lists representative costs of system components. While this table is useful for estimating an order-of-magnitude cost, the reader is cautioned that many factors can change these costs. For example, the cost of certain sensors can increase by 50% depending on the enclosure; outstations fully loaded with digital and analog I/O cards, isolation, local display and control, and so on, can cost 100% more than shown in this table. The values in the table reflect the cost of components with typical options and of average quality, and as such are representative of equipment being sold today. Finally, it should be noted that more expensive components may actually cost less through reduced operating costs resulting from longer calibration intervals, automatic verification of operational status, and so forth.

Labor requirements will include hardware installation, programming, routine maintenance and calibration, and operation. The magnitude of the labor requirement is often underestimated or even neglected. Yet an average-sized installation will require one to two employee-years of effort. Often one or more permanent "monitoring-system" positions are created. The different components that comprise the labor requirement are discussed here.

Table 12.6.2. Representative Costs of Mine Monitoring System Components[1,2]

1. Central Station Computer & Software (basic)		$10,000
Additional graphics terminal & assoc. graphics Software (incl. license fees)		10,000
2. Outstation-Red, 8 channel	3500.	
Upgrade to Blue		500
Barrier, per channel		125
3. Sensors		
Belt Monitoring (1 drive, 8 voltages)		0
Methane Sensor		1000
Carbon Monoxide Sensor		550
Carbon Dioxide Sensor		550
Oxygen		550
Air Speed Sensor (vortex shedding)		1500
Air Pressure		750
Hydraulic Pressure		125
Electrical Current		100
Voltage		20
Temperature		40
Accelerometer		450
Event (proximity)		150
Speed (rpm)		250
4. Uninterruptable Power Supply		1000
5. Cable/1000 ft (4-16 Awg twisted pairs, shielded, Polypropylene insulation, PVC jacket)		500

1. In 1990 dollars.
2. These costs are representative of actual prices paid by the mine and are not list prices.

OPERATION. Daily operation of the monitoring system usually is assigned to the dispatcher or communications person at the mine and as such does not require that an additional person be employed. These people become quite adept with the systems with only a few hours of training and a few weeks of experience.

A few larger mines and preparation plants have created positions, such as monitoring system operator. However, these people generally assume additional system responsibilities such as software development, data analysis, and coordination of system maintenance.

PROGRAMMING. The amount of programming required at the mine site depends somewhat on the system that was purchased and on the level of customized graphic displays that are desired. The system programming is frequently done by an engineer at the mine after a week of training and study of the system manuals. As a minimum it is necessary to program system-level parameters, such as information on the connected sensors, threshold values, and display options. Modern systems provide user-friendly and interactive routines that allow this programming to occur with a minimum level of skill and time, often in less than a few days. The development of custom graphic displays can be much more involved, even with the best graphics software being sold with monitoring systems today. It is useful to remember that the objective of graphic displays is simply to facilitate information transfer, not to create graphic masterpieces for all to admire.

Engineers report spending as little as two weeks to as much as twelve weeks—with the median being close to four weeks—on the development of an average complement of display screens. This would include schematic diagrams with active elements of the ventilation and materials handling systems, and perhaps the power system. Other displays (e.g., for sumps, methane drainage stations, and fans) may be included if they are monitored. Various tabular displays, for both the video terminal and the printer, would be included as well. Once developed, these displays will require periodic updating to reflect changes in the mine. Such updating usually requires less than one day each month. Companies with several mines generally prefer to have a few people at the division or corporate level who handle all programming. Usually they will also be responsible for equipment selection and will be well-versed on the application of monitoring technology to mining problems.

MAINTENANCE. Routine maintenance includes battery replacement, sensor calibration, repair of damaged cable, and repair or replacement of failed components. This work is typically handled by the maintenance department, often by one or two electricians who have become familiar with the system. Sometimes the electrician responsible for the mine communications system will be assigned responsibilities for the mine monitoring system as well. A gas sensor in an atmospheric-monitoring system will take an average of 15 minutes to calibrate, excluding travel time. If the sensor must be calibrated monthly, and there are 30 such sensors, the monthly time committment to this task alone is significant. In many cases it is desirable to hire a full-time technician to calibrate and maintain the system. This person will often assist in the installation of the system.

INSTALLATION. The installation can be divided into three parts: the central station, the in-mine cabling, and outstation installation. Setting up the equipment in the central station, which is often in the dispatcher's room, requires as little as one day, although elaborate control rooms requiring weeks of effort can also be found. Hanging cable as the mine advances is easily accomplished as part of the same task as extending power and communications lines. Unfortunately, many new monitoring systems must be installed in large existing mines where it is necessary to install miles of new monitoring system lines. This can be a formidable task, requiring thousands of employee-hours of effort. Fortunately, it is usually possible to get a part of the system operational while in-mine wiring continues. Installation of an outstation, including the associated sensors, in which all of the sensors are in the immediate vicinity of the outstation requires from one shift to several shifts, depending on the amount of wiring required to connect the sensors. Installing voltage pickups in a belt center may take a shift whereas hanging and connecting a gas sensor can take less than an hour. If a sensor is located a few thousand feet (meters) from an outstation, additional time will be required to route and hang a twisted pair of wires.

Many mines using monitoring systems have found it best to create one or two positions, for example, for an engineer and a technician, specifically for the monitoring system. These people are responsible for all aspects of installation, maintenance, and often programming.

12.6.4 VOICE COMMUNICATIONS

The need for a quality mine communication system is evident in virtually every aspect of daily mine operation, and during an emergency, good communication is crucial. The importance of the communication system is reflected in statutory requirements that specify minimum system requirements. However, technological advances have made it possible for the communications system to serve a variety of functions beyond these minimums.

Voice communications is a specialized form of data communications, and much of the material presented previously is applicable here, especially the comments made in 12.6.1 on the information aspects of mine monitoring. The type of voice communication system and its degree of sophistication should be determined by well-defined needs to transmit and receive information in a clear and timely fashion. Fortunately a wide variety of systems, designed to meet virtually any need, are now commercially available. A comprehensive overview of mine communications is given by Anon. (1984) and examples by Anon. (1983).

12.6.4.1 Telephone

Telephone communication was first practiced in mines using crank-ringer phones. In the late 1940s and 1950s, pager phones were introduced, and pagers in some form are in use today in many mines. The pager phone, like the crank-ringer before it, is a party-line system. One line services all of the phones in the mine. A "call" is placed by paging, that is, saying the person's name while holding the page button; this message is then heard through the loudspeaker at each pager phone in the mine. If the page is answered, then the individuals can talk without having their conversation broadcast at each pager phone. However, it is not a private conversation because anyone who picks up another pager phone will be able to eavesdrop on any ongoing conversation.

The quality, that is, intelligibility, of pager phone conversations is good if the telephone line is maintained in reasonable condition. Although these phones continue to meet the needs of many mines, they have several disadvantages. First, as a party-line system, there is a lack of privacy to discuss sensitive issues. Second, in all but the smallest mines, there is often difficulty in finding the line free when a call must be placed, a problem that tends to be especially bad around shift change, and one that can be a real hazard when the phone is needed for emergency purposes. Third, as the volume of calls on the system increases, there will be a more or less constant series of pages. This results in two undesirable effects: people begin to "tune out" these background pages, and eventually they do not hear when they are being paged; all of this paging, at each pager phone in the mine, results in an accelerated draining of the batteries that power these units.

One improvement of the pager phone is the addition of selective paging, in which each pager is electronically assigned a unique identification number; then a phone will only broadcast those pages that have been prefaced by its identification number. In this way the needless paging at all pager phones is eliminated, thereby eliminating the background noise and battery problems mentioned previously. Nonetheless, all of the other problems still exist: to eliminate these, a dialer phone is needed.

Dialer systems, usually with a paging capability often involve a private branch exchange (PBX) at the mine. A pair of wires is needed to connect between each phone and the PBX. Since this can result in thick bundles of cables in large installations, it is desirable to utilize frequency-division multiplexing to allow multiple conversations to use the same wire. While the cost of a PBX system is higher than for a pager system, it offers significant advantages that can result in fewer delays in locating people and communicating a message. PBX systems are the system of choice for many newly opened mines, and a number of older mines have also been outfitted with this type of system.

Some of the features, in addition to the private line aspect, which make PBX systems both popular and useful are summarized here. First, the ability to call outside the mine—to an engineering office or parts supplier, for example—is a timesaver. The systems are often interfaced with pagers, both loudspeaker and personnel or pocket pagers that are carried by key personnel. This makes it easier to locate these people and lowers the average time needed to locate a person from 30 minutes, with a pager phone, to a few minutes. If the page is not answered, many systems can be set up to automatically retry the page every few minutes, and then when it is answered, an automatic callback is generated to the person who initiated the page.

12.6.4.2 Radio

Radio communication is attractive because it permits a degree of mobility by removing wiring constraints. The need and utility of wireless communication is evident in the widespread use of cap lamps and lamp signals to communicate simple messages among miners. Further, wireless systems eliminate the problem of not having access to a phone, and they make it easier to advance communication as the working area advances. The impediment to widespread use of radio communications in underground mines is the attenuation problem. Simply put, low-power transmissions, compatible with portable equipment, occur at wavelengths which tend to pass through obstructions, such as pillars, losing energy as they do so. As a result, good line-of-sight distances can be achieved, but the signals generally have only enough energy to travel around a few corners or bends (Lagace et al., 1975).

The transmission range is optimized, within the constraints, by transmitting within the medium frequency (MF) band (i.e., 300 kHz to 3 MHz). The characteristic of this band that makes it well suited for in-mine use is its tendency to couple onto water pipes, power cables, etc., for a "free ride" and then to re-radiate. Because of this, the effective transmission distance is much greater than would be expected. Most radio communications gear for the mine utilizes this frequency band.

One option to increase the transmission range is to utilize repeaters, devices that receive and retransmit signals to improve the range. The disadvantages are that they are typically required in some numbers, are not inexpensive, and must often be relocated as mining advances. As such the use of repeaters is limited in mining applications. Another option is to use a hybrid, combination wired and wireless system, in which a "leaky coaxial feeder" is hung in a main entry. This feeder radiates signals along its length to portable radios, and acting like an antenna, receives portable radio transmissions. Mines utilizing leaky feeder systems in conjunction with medium-frequency radios report unqualified success and complete satisfaction. Still, leaky feeder systems are used in relatively few mines, probably due more to a lack of understanding than anything else, although the purchase cost of the system can be slightly higher than a PBX system.

12.6.4.3 Trolley

Trolley communications are important to insure safety on the haulageway as well as for the efficient dispatching of traffic. Trolley phones utilize a carrier communications scheme in which the high-frequency signal is superimposed on the dc trolley wire. Filters are then used to isolate the communications signal at the trolley phone and to reject the dc component. In practice, the quality of the signal tends to be poor, for two reasons. First, there are a number of parallel loads that provide paths to ground for the signal; these paths include motors and rectifiers. As a result, the signal power is reduced at each parallel path. Second, high-frequency noise from arcing further degrades the quality of the signal.

A variety of technological improvements has resulted in greatly improved trolley phone systems. However, an attractive alternative is to use a hybrid system, in which a dedicated communications wire is installed adjacent to the trolley wire, and radio transmitters/receivers are installed in haulage vehicles. For a slight increase in capital cost, a superior communications system is available through this option. Of course, mines utilizing the leaky feeder system have no need for a separate trolley-phone system.

12.6.4.4 Shaft

Shaft or hoist communications are necessary to permit communication between persons in the cage and the surface or underground. In many cases, and particularly in underground coal mines, a phone line directly connects the cage to the mine communication system. In other cases, notably deep hard-rock mines, it would be difficult or costly to maintain such a cable. An alternative is to use a special system that employs the wire rope itself as a channel or low-impedance propagation path. An annular coupling antenna is placed around the rope at the top of the shaft and at the top of the cage; the transmitters/receivers are connected to these antennas, resulting in high-quality, relatively inexpensive, and reliable communication between the cage and the mine or surface.

12.6.4.5 Emergency

Communication systems play a crucial role during an emergency, and accordingly, the design of the system must reflect the different conditions that might exist. For example, it is likely that power will be interrupted, and it is also possible that some communication lines will be damaged. The traffic on the system is likely to increase as many people attempt simultaneous use of the system. Hardware to satisfy the mine's emergency communications needs, as well as the routine requirements is readily available from several manufacturers. There is a variety of other products available to meet emergency requirements (e.g., Hislop and Rundle, 1986). Over the past few years a number of high quality wireless and portable systems have been introduced into the marketplace. The durability and long-range characteristics of these units uniquely qualify them for use in rescue operations.

Through-the-earth communication systems, utilizing ultra-low frequencies (ULF), were developed to facilitate the location of trapped miners. Although the efficacy of such systems was successfully demonstrated, they never really gained acceptance.

Recently, however, this technology has been used to overcome an old problem common to metal mines: notifying personnel to evacuate when a fire has been detected. Historically, stench gas has been used by injecting it into the ventilation system and often the compressed air pipelines. The long time delays, often a half an hour or more, can prove fatal. The new ULF system will require a modified cap lamp which, when it detects the signal from the ULF transmitter on the surface, will begin to blink. A switch on the cap lamp must then be used to stop the blinking. This warning system, which has been successfully demonstrated to work over several miles, will provide virtually instantaneous warning to evacuate the mine, regardless of where the miner is located. The cost of the system is a few dollars per cap lamp and a few thousand dollars for the surface transmitter (Hjelmstad and Ackerson, 1990).

12.6.5 SUMMARY

Mine monitoring and communications systems allow mining personnel access to information in a timely fashion. Technologically, it is possible to achieve virtually any degree of sophistication desired in these systems. The constraint is the cost of obtaining the information; obviously, this must be commensurate with the information needs at the mine.

A variety of quality products are available to satisfy the mining industry's needs in monitoring and voice-communications systems. Several manufacturers even specialize in products for the mining industry. Given the types of sensors available, and a little imagination, it is possible to monitor virtually anything. However, given the many technical options and costs, it is essential that the mining engineer have some basic understanding of the technology, and have the ability to articulate the mine's needs to the manufacturer. By doing so, the many benefits of this technology can be realized.

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Part IV
Surface Mining

Section 13 Surface Mine Development

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Chapter 13.0
INTRODUCTION

W.A. HUSTRULID

The subject of surface mine development is large. This section covers the fundamental concepts involved in both the open pit mining of thick or steeply dipping deposits as well as the strip mining of thin, shallow, and relatively flat-lying seams or beds. Design involves the marrying of geometries, ore and waste characteristics, equipment, and economics so that the desired result is achieved. Generally, there are many possible combinations from which to choose. For example, extraction over a short time period requires one strategy while quite another is applied for longer times. Variations in supply and demand require some flexibility of operation. Customer requirements impose certain constraints regarding the set of possible designs.

Over recent years, planning and operating considerations for surface mining imposed by environmental requirements have increased markedly. There have, at the same time, been enormous advances made in the design aids available to the mine planner. Computer simulation of the various options, whether it be for mine design or equipment selection, provides detailed information for decision making in real time. A significant challenge to the engineer/planner is how to best use the aids available. Because of the widespread use of commercially available and in-house computer software by industry, this section does not deal with the details of planning (for basic coverage of mine development, see Section 7). Rather, some of the basic concepts are stressed. The principles involved in equipment sizing and selection are discussed. A highly important but sometimes overlooked aspect of planning, haul road design, is also covered. For surface mining operations of the future, in-pit crushing and conveying will become even more important. Some of the considerations for application evaluation are included. The section concludes with some future concepts for surface mining.

Chapter 13.1
OPEN PIT PLANNING AND DESIGN

G.A. FOURIE AND GERALD C. DOHM, JR.

13.1.1 BASIC CONCEPTS

G.A. FOURIE

13.1.1.1 Introduction

An *open pit mine* is an excavation or cut made at the surface of the ground for the purpose of extracting ore and which is open to the surface for the duration of the mine's life. To expose and mine the ore, it is generally necessary to excavate and relocate large quantities of waste rock. The main objective in any commercial mining operation is the exploitation of the mineral deposit at the lowest possible cost with a view of maximizing profits. The selection of physical design parameters and the scheduling of the ore and waste extraction program are complex engineering decisions of enormous economic significance. The planning of an open pit mine is, therefore, basically an exercise in economics, constrained by certain geologic and mining engineering aspects.

Several factors are significant indicators of economic and technological trends in surface mining. Among these are increasing production, a shift in emphasis from underground to surface mining, a decline in ore grade and quality of some crude materials, and, with few exceptions, an increase in productivity of labor. Increasing production is the result of a growing demand for more mineral commodities, induced by an increase in population and per capita consumption.

Paradoxically, productivity has increased even with declining grade and quality, which is indicative of the rapid technological improvement taking place in open pit mining techniques. This pattern of change has permitted production from many formerly uneconomic mineral resources occurring near the surface at a time when higher-grade ores are inadequate to meet increasing demand.

It is generally conceded that surface mining is more advantageous than underground mining in terms of recovery, grade control, economy, flexibility of operation, safety, and the working environment. There are, however, many deposits that are too small, irregular, and/or deeply buried to be extracted economically by surface mining methods. Furthermore, even where mineralization extends to a greater depth in open pit mines, the rapidly increasing amount of overburden to be handled imposes economic limits beyond which mining must either be abandoned or converted from surface to underground operations. The conclusions, although valid, are sometimes outweighed by a new set of factors that arise from differences in the physical character of mineral deposits available for future exploitation and from changes in technology, markets, and public policy.

Open pit design is conducted in several stages. They consist, technically, of devising a scheme or set of alternative schemes, followed by an evaluation and selection of the optimum scheme, as illustrated in Fig. 13.1.1.1.

The most economic final pit design often depends on factors that are largely outside the mining engineer's control, such as the geometric outline of the ore body, the distribution of ore within the ore body, topography, maximum allowable slope angles, etc. The economics of the mining program, however, depends upon the choice of mining ratio, production rates, and equipment, all of which are determined by the mining engineer.

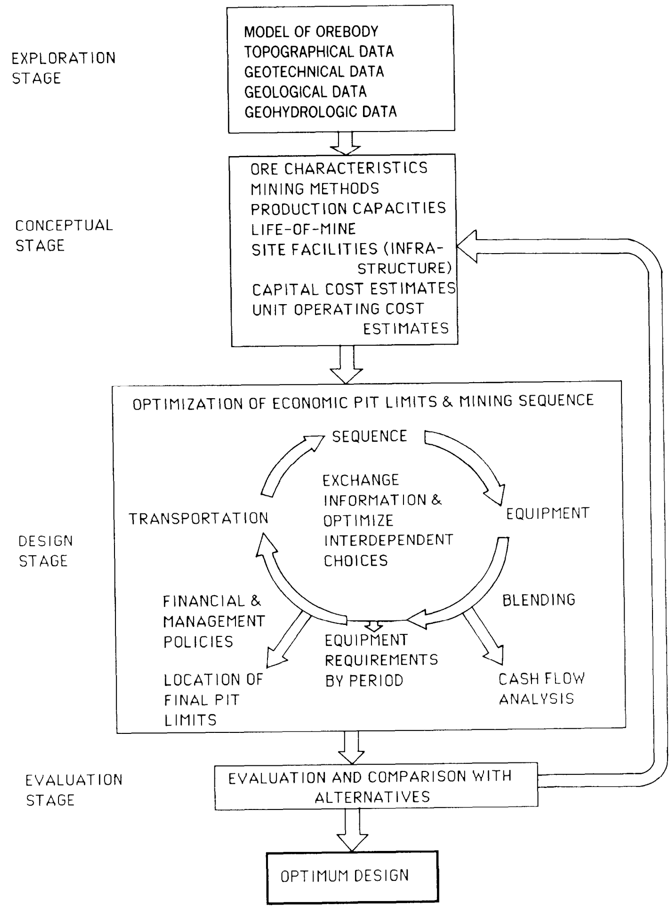

Fig. 13.1.1.1. Diagrammatic presentation of the design process.

13.1.1.2 Open Pit Terminology

Persons closely associated with open pit mines have coined terms or phrases for certain operations or for defining commonly occurring geometric parameters in open pit mining. The following are terms that commonly occur in open pit mine planning and are used throughout this chapter. These terms are illustrated in Fig. 13.1.1.2, which shows a pit section through an idealized tabular ore body.

A *bench* may be defined as a ledge that forms a single level of operation above which mineral or waste materials are mined back to a bench face. The mineral or waste is removed in successive layers, each of which is a bench. Several benches may be in operation simultaneously in different parts of, and at different elevations in the open pit mine.

The bench *height* is the vertical distance between the highest point of the bench, or the *bench crest,* and the *toe* of the bench.

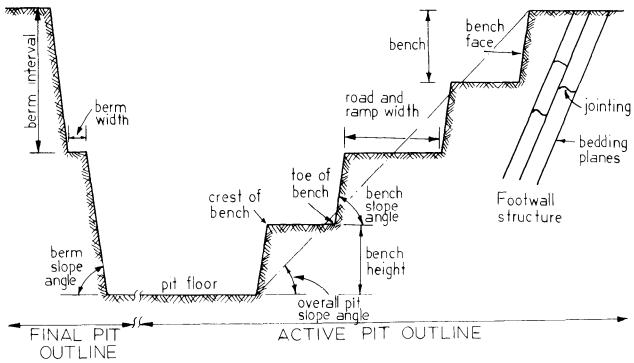

Fig. 13.1.1.2. Section through an open pit in an idealized tabular ore body.

The bench height is normally governed by the specifications of operating machines, such as drills and shovels, and by government mining regulations.

The *bench slope* is the angle, measured in degrees, between the horizontal and an imaginary line joining the bench toe and crest.

Pit limits are the vertical and lateral extent to which the open pit mining may be economically conducted. The cost of removing overburden or waste material vs. the minable value of the ore is usually the prime factor controlling the limits of the pit. Other factors that can influence pit limits are existing surface infrastructures, such as townships, rivers, etc.

In order to enhance the stability of a slope within the pit—and for safety reasons—*berms* may be left. A *berm* is a horizontal shelf or ledge within the ultimate pit wall slope. The *berm interval, berm slope angle,* and *berm width* are governed by the geotechnical configuration of the slope. The *overall pit slope angle* is the angle at which the wall of an open pit stands, as measured between the horizontal and an imaginary line joining the top bench crest with the bottom bench toe.

For the duration of open pit mining, a *haul road* must be maintained into the pit. A *spiral system* is an arrangement whereby the haul road is arranged spirally along the perimeter walls of the pit so that the gradient of the road is more or less uniform from the top to the bottom of the pit. A *zigzag* or *switchback* system is an arrangement in which the road surmounts the steep grade of a pit wall by zigzagging, generally on the footwall side of the pit. The choice of spiral or zigzag is dependent upon the shape and size of the ore body, truck economics, and pit slope stability.

Haul road width is governed by the required capacity of the road and type of haulage unit. The *grade* may be defined as the inclination of the road in terms of degrees from the horizontal or percentage of rise to the horizontal.

The *angle of repose* or *angle of rest* is the maximum slope at which a heap of loose material will stand without sliding.

The *suboutcrop depth* represents the depth of waste that has to be removed before any ore is exposed. This waste is often referred to as *preproduction stripping.*

13.1.1.3 Exploration Input for Open Pit Planning

In addition to defining the extent and value of a mineral deposit, exploration supplies data vital to mine development and exploration. Decisions regarding pit size and layout, production rate, and the mineral processing flowsheet are critically dependent on exploration data input. Procedures and particulars of mineral prospecting and exploration are provided in Section 4 and of project geology in Section 5.

A successful exploration program culminates in drilling and sampling information, useful in establishing mine operation parameters, geotechnical design, geohydrologic conditions, and mineral processing or metallurgical extraction. Two of three—geotechnical and geohydrologic investigations—warrant further discussion here. For detailed coverage of geomechanics, see Section 10.

Geotechnical Investigations. While soil masses are often homogeneous and relatively isotropic, this is rarely true of rock masses. The main reason for this is the presence of ubiquitous discontinuities in rock masses compared with soils. The behavior and stabilities of rock masses are controlled mainly by the nature and orientation of these discontinuities, while the stability of soil masses is controlled mainly by the strength of the intact material.

The importance of geotechnical and geohydrologic factors on open pit mining operations has been a very popular topic among researchers and cannot be overemphasized.

However, there is no definite system available prescribing geotechnical data-collection procedures; for this reason, mining companies seldom take into account geotechnical considerations during the initial exploration stage of the investigations, resulting in drill cores being disposed of shortly after completion of the drilling program. Geotechnical engineers are often consulted at an advanced stage of project evaluation, resulting in additional expenditures in order to obtain sufficient information.

Requirements for effective geotechnical drilling are briefly as follows:

1. Boreholes must be cored from as close as possible (or practicable) to surface through the overburden to at least 35 ft (10 m) into the footwall. If there is any suspicion of permeable zones at greater depth, the boreholes should be extended.
2. Cores should preferably be at least 2 3/8 in. (60 mm) in diameter and drilled with triple-tube core barrels to ensure minimal core disturbances.
3. In many cases, it may be desirable to drill inclined rather than vertical holes for the purposes of
   a. Orientating the core, either from the bedding traces or by the use of a core orientation.
   b. Intersecting steeply dipping joints, which are common in flat-lying sediments and which are poorly sampled by vertical holes.

The density of geotechnical drilling will be dependent primarily on the degree of variability of the material over the proposed mining area.

Drilling must be supervised by an experienced exploration geologist to record hole depths, core losses, and any other significant aspects (e.g., water losses) during drilling. Continuous liaison between the exploration geologist and the mine planning engineer is important to ensure that all mining-related data are collected.

As each core run is recovered, it should be carefully transferred to the core tray and allowed to dry before being

1. Photographed in color, ensuring that the core trays are adequately labeled and oriented.
2. Logged in detail for the following features:
   a. Rock type.
   b. Degree of weathering.
   c. Description of weak rocks or weak zones.
   d. Location, description, and orientation of any breaks in the core.

3. Sampled for materials testing.

Color photographs are an invaluable record of the original core condition. The core, even if available at a later date, may have deteriorated or be incomplete as a result of sampling.

Geologic data should be presented in graphical form to highlight the similarity or variations in properties across the proposed mining area. Rock hardness and bedding, or joint frequency, can be plotted in histogram form along the borehole paths. Such plots can be made with an exaggerated vertical scale to provide adequate detail, but it is also important to plot natural-scale sections in order to place the data into its actual physical content.

A basic program of materials testing is required to determine significant material properties and to confirm visual estimates during core logging. The relevant testing for open pit mines includes

1. Slaking and classification tests on weathered or suspected weak materials that could influence dump and pit stability.

2. Shear strength tests on soft or weak layers, particularly in the floor, and on overburden materials showing strong slaking behavior, for use in analysis of pit stability.

3. Compressive strength, point load strength, and drillability tests to provide information for estimation of overburden and ore excavation characteristics.

Geohydrologic Investigations. The presence of groundwater in the rock mass surrounding an open pit has a detrimental effect on mining operations for the following reasons:

1. Water pressure reduces the stability of pit slopes by reducing the shear strength of potential failure surfaces. Water pressure in tension cracks, or similar near-vertical fissures, reduces stability by increasing the forces tending to induce sliding.

2. High moisture content results in an increased unit weight of the rock and hence gives rise to increased transport costs. Changes in moisture content of some rocks, particularly shales, can cause accelerated weathering with a resultant decrease in stability.

3. Freezing of groundwater during winter can cause wedging in water-filled fissures due to temperature-dependent volume changes in the ice. Freezing of surface water on slopes can block drainage paths, resulting in a buildup of water pressure in the slope with a consequent decrease in stability.

4. Erosion of both surface soils and fissure infilling occurs as a result of the velocity of flow of groundwater. This erosion can give rise to a reduction in stability and also to silting up of drainage systems.

5. Discharge of groundwater into an open pit gives rise to increased operating costs because the water must be pumped out, and also because of the difficulties of operating heavy equipment on very wet ground. Blasting problems and blasting costs are increased by wet blastholes.

6. Liquefaction of overburden soils or waste dumps can prevail where water pressure within the material rises to the point where the uplift forces exceed the mass of the soil. This can occur if drainage channels are blocked or if the soil structure undergoes a sudden volume change, as can happen under earthquake conditions.

By far, the most important effect of the presence of groundwater is the reduction in stability resulting from water pressures within discontinuities in the rock mass.

There are two possible approaches to obtaining data on water pressure distribution within a rock mass:

1. Deduction of the overall groundwater flow pattern from consideration of the permeability of the rock mass and sources of groundwater.

2. Direct measurement of water levels in boreholes or wells or of water pressure by means of piezometers installed in boreholes.

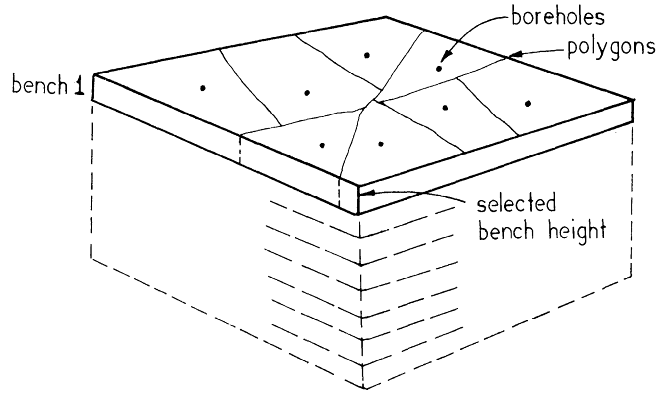

Fig. 13.1.1.3. Block model of an ore body.

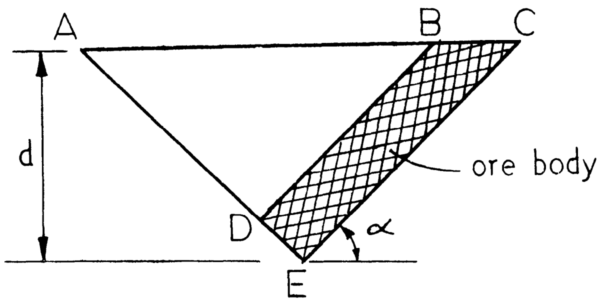

Fig. 13.1.1.4. Overall stripping ratio representation.

Both methods abound with practical difficulties; however, because of the very important influence of water pressure on slope stability, it is essential that the best possible estimates of these pressures be available before a detailed open pit stability analysis is attempted.

13.1.1.4 Bench Plan Preparation and Presentation

A key point in the design and operation of a modern mining operation is the construction of what is called an ore body model or *block model.* This model is a representation of reality constructed from predicted information. The blocks involved are merely subsets of the overall model that allows manipulation of the contained information on a local scale. In general, block models enable mine planners to effectively select the most promising means of extracting the ore both physically and economically. The uses of a block model can be quite diversified, but one must realize that a single model that satisfies all curiosities and forms of expertise is difficult to construct.

Because material removal or mining takes place along a particular bench, it is firstly important to prepare a series of bench plans that, when combined, forms the overall block plan of the modeled ore body. This is illustrated in Fig. 13.1.1.3.

With the assistance of modern computer facilities, sophisticated and complex bench plans can be prepared.

13.1.1.5 Stripping Ratio Considerations.

The parameter known as the *stripping ratio* is almost universally used and represents the amount of uneconomic material that must be removed to uncover one unit of ore. Fig. 13.1.1.4 represents an idealized open pit ore body, dipping at angle *a*.

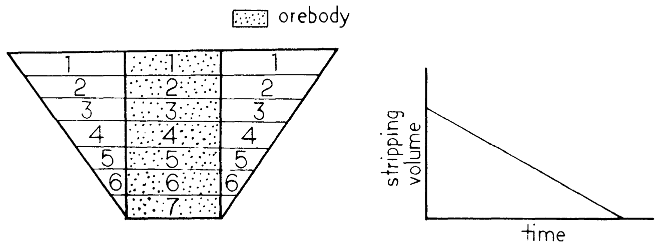

Fig. 13.1.1.5. Declining stripping ratio method.

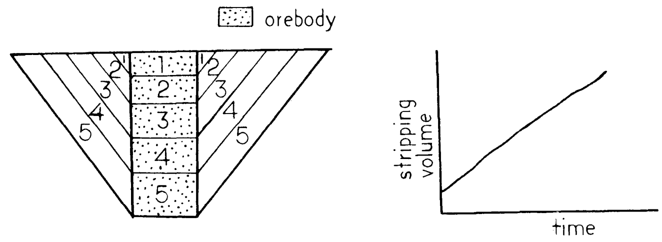

Fig. 13.1.1.6. Increasing stripping ratio method.

The ratio of the total volume of waste to the ore volume is defined as the *overall stripping ratio:*

$$R = \frac{\text{volume of waste removed to depth } d}{\text{volume of ore recovered to depth } d} = \frac{ABD}{BCED} \qquad (13.1.1.1)$$

While a volume relationship, calculated in cubic yards/cubic yards (cubic meters/cubic meter), it is more commonly expressed as tons/ tons (tonnes/tonne). Note that in mining certain mineral commodities, however, stripping ratio is expressed in units of cubic yards/ton (cubic meters/tonne).

A simple and widely used method of analysis to obtain pit limits is to utilize the cutoff stripping ratio. First, the pit slope is determined from geotechnical and other considerations and the stripping ratio then calculated. The *cutoff stripping ratio* is the one for which the costs of mining the ore and waste are matched by the revenue from that block of ore. Factors used to determine costs should include the added costs of mining as the pit deepens and the interest charges on the prestripping of waste.

In a more complete analysis, the entire ore body is mined on paper. The production from each time period is determined, the costs and revenues listed, and a cash flow generated. The profits from each year are discounted to reflect the time value of money in one of several ways, depending on management preference (investment analysis is discussed in Section 6). The result is considered to be the value of the mine or production. Mining is continued until it no longer increases the value, and so a pit limit is determined. The ratio of the total volume of waste to total volume of ore is then the overall stripping ratio.

With fluctuating commodity prices, increasing mining costs, and the introduction of more sophisticated mining techniques, the overall mining plan and overall stripping ratio can change over the total life of any mine.

For this reason, it is necessary to update the long-term plan of the mining project at regular intervals. Having determined the final pit limits and overall stripping ratio, the mining plan can be executed in a number of ways, as illustrated in the following.

Declining Stripping Ratio Method. This method (Fig. 13.1.15) requires that each bench of ore be mined in sequence, and all the waste on the particular bench is removed to the pit limit. The advantages of this method are the operating working space available, the accessibility of the ore on the subsequent bench, all equipment working on the same level, no contamination from waste blasting above the ore, and equipment requirements are a minimum towards the end of the mine's life.

The primary disadvantage of this method is that the overall operating costs are a maximum during the initial years of operation when maximum profits are required to handle interest and repayment of capital.

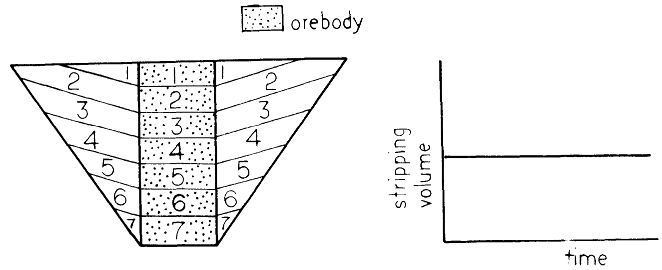

Fig. 13.1.1.7. Constant stripping ratio method.

Increasing Stripping Ratio Method. In this method, stripping is performed as needed to uncover the ore (Fig. 13.1.1.6). The working slopes of the waste faces are essentially maintained parallel to the overall pit slope angle. This method allows for maximum profit in the initial years of operation and greatly reduces the investment risk in waste removal for ore to be mined at a future date. This method is very popular where the mining economics or cutoff stripping ratio is likely to change on very short notice.

The disadvantage of this method is the impracticality of operating a large number of stacked, narrow benches simultaneously to meet production needs.

Constant Stripping Ratio Method. This method (Fig. 13.1.1.7) attempts to remove the waste at a rate approximated by the overall stripping ratio. The working slope of the waste faces starts very shallow, but increases as mining depth increases until the working slope equals the overall pit slope. This method, from an advantage and disadvantage point of view, is a compromise that removes the extreme conditions of the former two stripping methods outlined. Equipment fleet size and labor requirements throughout the project life are relatively constant.

In actual practice, the best stripping sequence for a large ore body is one in which the rate is low during the initial stages and towards the end of the project's life. The advantages can be summarized as follows:

1. A good profit can be generated initially to increase cash flow.
2. The labor and equipment fleet can be increased to maximum capacity over a period of time.
3. The labor and equipment requirements decrease gradually toward the end of the mine's life.
4. Distinct mining and stripping areas can be operated simultaneously, allowing for flexibility in planning.

13.1.1.6 Mine Plan Frequency

Mine plans vary in frequency from short- to long-range. The most common are (1) daily, (2) weekly, (3) monthly, (4) yearly, and (5) life-of-mine. The prime difference among these plans is the degree of detail. The shorter the time span covered by the plan, the greater the degree of accuracy and confidence that the actual performance will meet the forecast. For example, the daily mine plan details the production schedule on a shift basis for 24 hours, while long-range plans can be used to establish financial forecasts with regards to purchase of new equipment, changes in the work force, variations in operating costs due to increased haul distances, and dewatering requirements. Changes in the ore type may dictate the long-term blending criteria and possible changes in revenue from the sale of ore.

13.1.2 OPEN PIT FEASIBILITY STUDIES

GERALD C. DOHM, JR.

After a mineral deposit has been discovered and evaluated sufficiently to be considered an ore deposit, the problem then becomes how to mine and process that ore body in a way that *maximizes the net present value* (NPV) *within a practical operating format.*

The problem facing planners who must recommend mine plant size, equipment selection, and long-range scheduling is how to optimize a property not only in terms of mechanical efficiency but also in project life. Mine planning is dependent upon the interaction of contributing factors that lead to maximizing the NPV. Realistic mine planning is basic to the analysis. The process discussed here is best used when evaluating and planning undeveloped properties. This procedure is not intended to replace exacting mine analysis but rather to focus in on property evaluation for feasibility studies.

Derivation of the various input parameters is discussed only to the point where a meaningful correlation can be made to the overall analysis. The interrelationship of the parameters is the key as most company guidelines and financial analysts have their own standards when setting the various inputs.

Assume that the data are acceptable and provide at least the following categories of information and guidelines: (1) detailed drillhole data; (2) mineral and waste inventory; (3) geologic, hydrologic, and geotechnical criteria; (4) topographic layout, including property boundaries; (5) metallurgical flowsheet, recovery, and design criteria; (6) access, water and power information; (7) environmental baseline data; and (8) financial criteria (minimum rate of return, payback period, etc.).

Any evaluation of a property involves bringing all these parameters together. All, however, are interrelated and dependent upon one another. This means that without knowing the cutoff grade, the ultimate ore reserves cannot be calculated; without the ore reserves, the final pit limits cannot be established; without knowing the overall tonnages, the production schedule cannot be selected; and without the required capacity-generated capital and operating costs, the cutoff grade and total ore reserves cannot be derived. The optimization process is outlined in Fig. 13.1.2.1.

The iterative process closes in on the evaluation by going through the endless loop analysis cycle until no appreciable change is noted after continued refinement of the input.

Where does one start? After establishing the preliminary geologic and process data, one proceeds with a cursory treatment of equipment, facilities, and production scheduling. Historical operating parameters for a region or similar deposit provide a sound basis from which to start the evaluation. The data establish the basis for calculating preliminary ore reserves, mining techniques, and process design criteria.

Financial evaluations should utilize a discount rate that is equivalent to the cost of capital to a company plus associated risks. The resulting NPV is a functional yardstick by which to compare investment alternatives. Maximizing the NPV is primarily a function of production scheduling and ore cutoff grades. As a basic premise, the sooner the capital write-off occurs, the better the optimization level achieved. Therefore, some initial high-grading of an ore body or commencing operations in a low waste-to-ore ratio area would improve the cash flow.

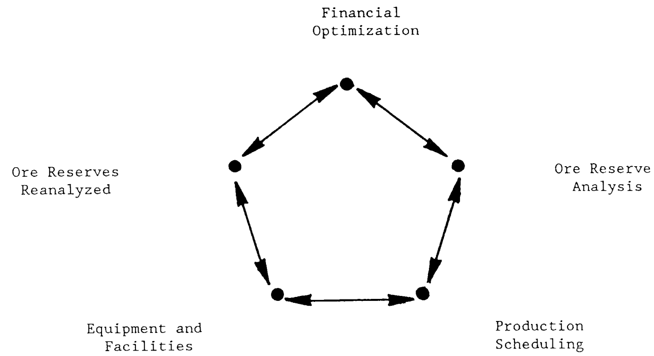

Fig. 13.1.2.1. Analysis components.

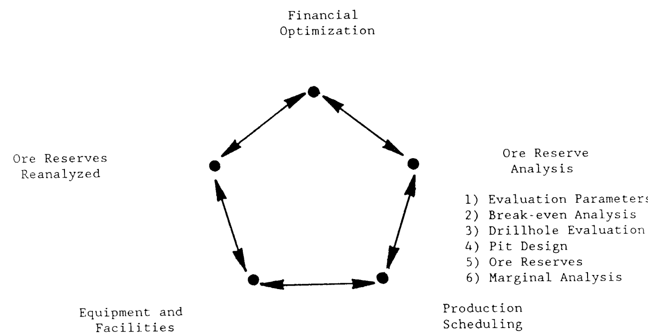

Fig. 13.1.2.2. Ore reserve analysis.

All capital and operating costs are expressed in current year dollars to forego having to project future cost and value trends. It will also be assumed that the ratio of income to production costs will remain constant throughout the life of the property. In reality, periodic reviews should be made during the operating life. Modifications to the ore reserves and operational plans may result if significant changes occur in the actual operating costs, commodity value, improved technologies, etc.

For the purpose of demonstrating the development of open pit evaluation, a simplified numerical case history is presented and evaluated step-by-step through one cycle.

For detailed discussion of ore reserve estimation procedures, see Chapter 5.6; for mine feasibility studies, see Chapter 6.2; and for equipment selection, see Chapters 9.3, 13.3, and 17.2.

13.1.2.1 Ore Reserve Analysis

A large, low-grade uranium deposit has been discovered and drilled to outline mineral at depths amenable to surface mining. The mineralization is in sandstone that lends itself to conventional mining techniques. The property is located in a rather remote region, and while there are unimproved roads into the area, all utilities have to be developed. The minimum acceptable rate of return will be set at 15%.

The first step is an analysis of ore reserves. The components of this analysis are indicated in Fig. 13.1.2.2, each of which will be briefly discussed.

Table 13.1.2.1. Overall Pit Slope (horizontal : vertical)

Vertical Height, ft	Overall Pit Slope
100	0.75:1
200	1.25:1
500	1.50:1

Conversion factor: 1 ft = 0.3048 m.

Table 13.1.2.2. Mineral and Waste Inventory

Grade % U_3O_8	Million* Tons	Avg. Grade % U_3O_8	Classification	Tonnage Ratios 1st	Tonnage Ratios 2nd
>0.080	8	0.129			
0.070	10	0.118			
0.060	14	0.103	Potential Ore	1	1
0.050	18	0.092	and Low-Grade		2
0.040	26	0.079			2
<0.040	16	N/A	Associated Waste**	0.6	
-0-	364		Primary Stripping	14	26

Conversion factor: 1 ton = 0.9072 t.
* Cumulative through 0.040 only
** Cutoff based on lower grades' inability to support mining, milling, and costs to do business [$14.80 cost vs. $13.50 value (0.030% × 20 × $25/lb × 0.9 recovery = $13.50 value)].

EVALUATION PARAMETERS. There are physical and financial evaluation parameters to be imposed.

Physical—A geotechnical evaluation was undertaken that indicated the overall pit slopes in Table 13.1.2.1 as acceptable for design purposes.

The specific gravity SG of representative rock samples was tested yielding an average value of 2.00. To convert the volume of rock to be mined and milled, a tonnage factor TF is calculated:

$$TF = \frac{2000 \text{ lb/ton}}{SG\ (2.00) \times 62.4 \text{ lb/ft}^3} = 16 \text{ ft}^3\text{/ton } (0.50 \text{ m}^3\text{/tonne}) \tag{13.1.2.1}$$

Initial metallurgical flowsheets and design criteria were also developed. The process is to be conventional acid leach-solvent extraction with an average recovery of 90%.

Financial— As a starting point for the calculations, regional operating and capital costs for operations having similar size, geology, and equipment are used. Amenability studies on which to base process specifications, reagent consumption, and product recovery are performed.

Regional operating economics (expressed in dollars/ton) include primary stripping (0.65); associated waste and ore mining (1.50); milling (7.00); direct costs such as salaries, utilities, maintenance, consumables, etc. (3.50); and indirect costs such as taxes, interest, depreciation, and corporate overhead (2.80).

For purposes of this example, the selling price of the recovered uranium oxide is set at $25/lb.

The drillhole grid is on 200-ft (60-m) centers, sufficient to outline the mineralized zone and associated overburden.

BREAKEVEN ANALYSIS. Establishing a cutoff grade for ore is necessary for drillhole evaluations. The process involves two steps and utilizes a tabulation of the mineral inventory in the area being considered for development (Table 13.1.2.2).

Table 13.1.2.3. Production Costs

Category	$/ton	1st Ratio	Total Cost, $	2nd Ratio	Total Cost, $
Stripping	0.65	14	$ 9.10	26	$16.90
Associated Waste	1.50	0.6	0.90	2	3.00
Ore Mining	1.50	1	1.50	1	1.50
Milling	7.00	1	7.00	1	7.00
Directs	3.50	1	3.50	1	3.50
Indirects	2.80	1	2.80	1	2.80
			$24.80		$34.70

Conversion factor: 1 ton = 0.9072 t.

The initial cutoff grade does not take into account primary stripping and associated waste. Using the ratios established in the first pass, the cutoff grade is recalculated.

Cutoff grade calculations are based on total operating costs, utilizing the overall ratio of tons of waste to tons of potential ore (Table 13.1.2.3).

$$\begin{aligned} \text{grade} &= \frac{\text{production cost}}{\text{value} \times \%\ \text{recovery}} \\ &= \frac{\$24.80\text{/ton}}{\$25\text{/lb} \times 0.90} \\ &= 1.10 \text{ lb/ton } (0.55 \text{ kg/t}) \\ &= \frac{1.10 \text{ lb/ton}}{20 \text{ (conversion factor)}} = 0.055\%, \text{ say, } 0.06\% \end{aligned} \tag{13.1.2.2}$$

The higher cutoff grade creates a new category of material, between 0.040% and 0.060%, which is classified as low grade. Under certain circumstances, it may be economically advantageous to process this material. This detail is reviewed later under marginal analysis.

DRILLHOLE EVALUATION. A simple approach for illustration is to use the polygon method to assign an area of influence for each drillhole. With the drilling grid on 200-ft (60-m) centers, the area of influence is 40,000 ft^2 (3720 m^2). Dividing the area by the tonnage factor, one finds:

$$\frac{40{,}000 \text{ ft}^2}{16 \text{ ft}^3\text{/ton}} = 2500 \text{ tons/vertical ft } (7440 \text{ t/m})$$

For simplicity, the deposit is assumed to be dry. The weight of water, when present, must be added to the weight of the rock. The total weight will then be used in the determination of operational costs, disposal considerations, and capital requirements.

All drillholes are evaluated from top to bottom. The economic bottom limit of mining is the depth at which the maximum net profit is realized. No credit is assigned to any values contained in material below the cutoff grade as they, in themselves, cannot justify any change in pit outline. This process is shown in Fig. 13.1.2.3.

The evaluation is repeated for all drillholes. This evaluation should also include those holes that obviously carry a negative value. In the expansion of the total reserves, many instances will be found where profitable holes will offset some negative values. The value of each must be known before a final determination can be made. The complete analysis is essential in establishing the ultimate pit limits, as discussed next.

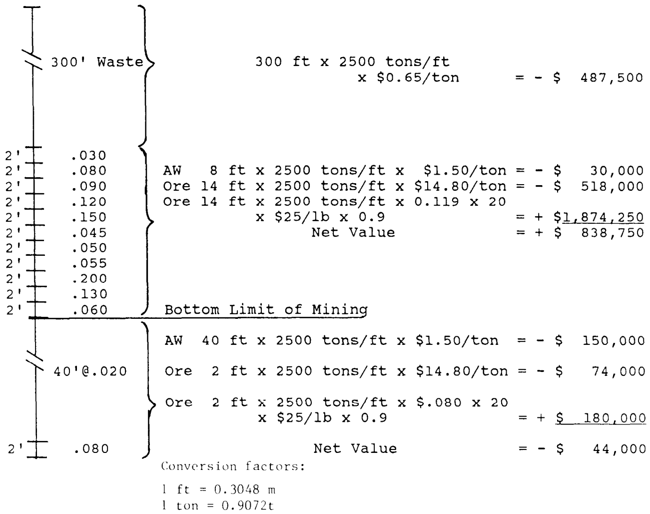

Fig. 13.1.2.3. Drillhole profit/loss calculation.

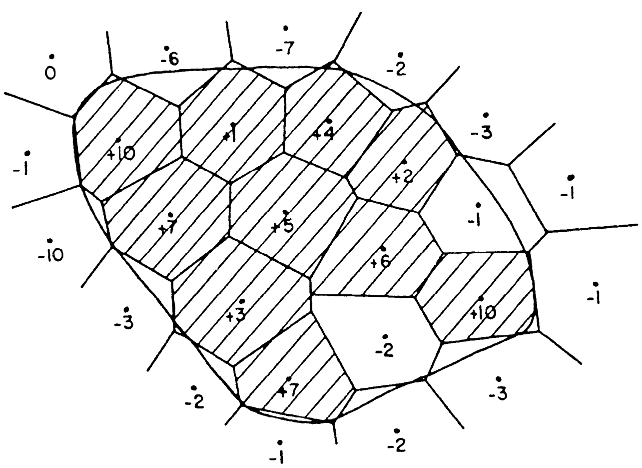

Fig. 13.1.2.4. Ore block smoothing.

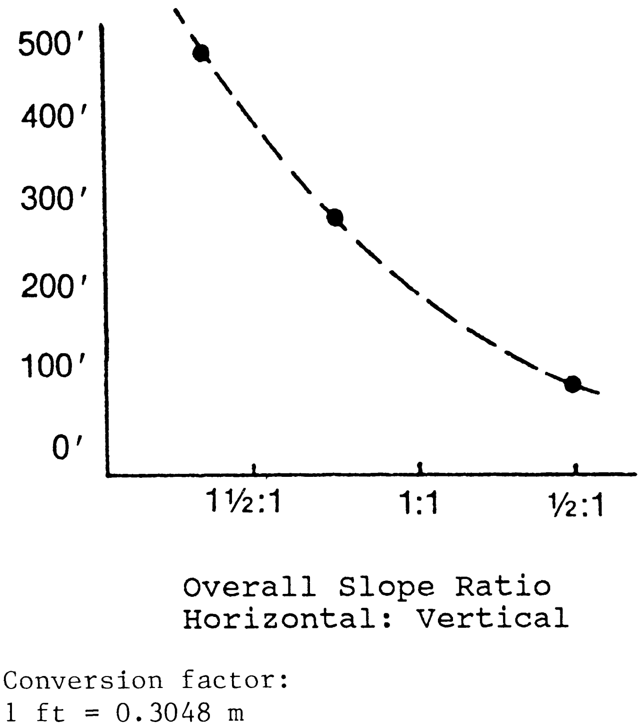

Fig. 13.1.2.5. Pit slope angle determination.

OPEN PIT DESIGN. The ultimate limits of the pit do not precisely follow the economic limits established in the drillhole evaluation. Refinements to the pit outline result from various working, financial and safety constraints. In practice, close-spaced development drilling will eventually be undertaken to better define pit boundaries and to develop information on where to commence mining activities. The bottom of mining may also be affected by the adjacent drillholes if bottom ore elevations vary to any appreciable extent.

Operation Efficiency—Operational efficiency increases advantageously as the horizontal section is smoothed by eliminating projections and generally irregular configurations. The drillhole evaluation helps with the smoothing of the pit walls and inclusion of waste by showing where positive values offset negative values. The objective is to maximize the total value and, at the same time, provide adequate access and working room (Fig. 13.1.2.4).

Overall Pit Slope—Use the steepest possible pit slope with regards to the safety factor deemed acceptable for the particular pit.

Overall pit slope angles are not only a function of the actual rock conditions encountered and the total vertical lift but also of the duration of exposure and the amount of water indigenous to or added to the formation. The effects of the latter two can be lessened by timely back filling and by either or both a mine dewatering program and a surface runoff diversion system. Where variable heights affect the ultimate design, a graph can be utilized relating height to slope angle at a safety factor where the slope is close to failure but does meet risk standards set within corporate guidelines.

In the example utilized herein, all slope angles are a direct function of height (Fig. 13.1.2.5). It must be remembered that in rough terrain, it is the difference in elevation from the surface to the economic bottom of the pit that ultimately establishes the final slope angle.

Design—One must provide adequate operating room for maneuverability and mining flexibility in ore/waste segregation requirements. A 0.5:1 pit slope between benches is used for design purposes. The desired overall slope angle is achieved by adjusting the widths of the benches:

$$\text{bench width} = \frac{\text{horizontal displacement}}{\text{no. benches required}} \tag{13.1.2.3}$$

$$= \frac{\text{vertical height} \times (\text{overall slope ratio} - \text{slope ratio between benches})}{(\text{vertical height} \div \text{bench height}) - 1}$$

In this example,

$$\text{width} = \frac{500 \text{ ft} \times (1.5 - 0.5)}{(500 \text{ ft} \div 50 \text{ ft}) - 1} = \frac{500 \text{ ft}}{9 \text{ benches}}$$

$$= 56 \text{ ft (17m)}$$

Table 13.1.2.4. Marginal Analysis Ore Reserves

Tons, million	Grade % U_3O_8	Material Classification	Ratio ton/ton
22	0.084	ore	1
20	N/A	associated waste	0.9
364		primary stripping	16.5

Conversion factor: 1 ton = 0.9072 t.

Table 13.1.2.5. Optimized Net Profit

300 ft stripping × 2500 tons/ft × $0.65/ton	−	$ 487,500
2 ft associated waste × 2500 tons/ft × $1.50/ton	−	7,500
22 ft ore × 2500 tons/ft × $14.80/ton	−	740,000
20 ft ore × 2500 tons/ft × $44.10/ton (0.098% × 20 × $25/lb × 0.9)	+	2,205,000
	+	$ 970,000

Conversion factors: 1 ft = 0.3048 m, 1 ton = 0.9072 t.

Individual bench widths may be adjusted to allow for ramps, lithology changes, etc., as long as the aggregate remains unchanged.

Having engineered a preliminary design, the ore reserves, associated waste, and stripping tonnages are now established.

ORE RESERVES. Total ore reserves are a direct result of the relationship between the drillhole evaluation and the open pit design. The calculation utilizes the concept of a break-even cutoff grade based on known or predicted operating costs. Another method, the break-even stripping ratio, that deals with the average relationships of the ore body is best suited to homogenous deposits. In this example, the grade is markedly different from one drillhole to the next, which requires separate ratios for adjoining areas of influence.

MARGINAL ANALYSIS. Once the pit limits are established and the resulting ore and waste inventories calculated, direct stripping and mining zone costs are estimated and are considered thereafter as sunk costs. Ore reserve optimization then involves the deletion of sunk costs from the analysis of the cutoff grade. The ensuing marginal analysis not only increases the ore reserves and tons to be milled but also lowers the waste-to-ore ratio:

marginal cutoff grade = operating cost – sunk cost + load and haul cost from stockpile to crusher ÷ (value × recovery cost) (13.1.2.4)

In this example,

$$\text{grade} = \frac{\$34.70/\text{ton} - \$24.00/\text{ton} + \$0.75/\text{ton}}{\$25/\text{lb} \times 0.9}$$

$$= \frac{\$11.48/\text{ton}}{\$22.50/\text{lb}} = 0.51 \text{ lb/ton} \ (0.25 \text{ kg/t})$$

$$= \frac{0.51 \text{ lb/ton}}{20} = 0.026\%, \text{ say, } 0.03\%$$

The calculation demonstrates that low-grade material can be milled profitably under the right circumstances. For planning purposes, however, it may not be prudent to consider all the low-grade material millable. A cushion should be established to cover profit margin, ore dilution, additional capital requirements, and uncertainties at this stage of the evaluation. Of the total low-grade from 0.03 to 0.06%, this example will consider only one-half the material, from 0.045 to 0.06%, as being economically justified to process (Table 13.1.2.4).

It can be seen that the ore tonnage increases from 14 million tons (12.7 million t) (Table 13.1.2.2) to a total of 22 million tons (20 million t) (a 57% increase), and the grade goes from an average of 0.103 to 0.084% (an 18% decrease). Recoverable pounds increase by 7.3 million, a gain of 28%.

While larger reserves appear to be the more efficient utilization of the mineral present, it is the cash flow and resulting NPV that dictate the cutoff grade. It should also be noted that marginal reserves cannot support capital expenditures for process capacity expansion. For the purpose of determining a cash flow, the marginal reserves are used in the total tonnage of ore to be milled, with the ends of the analysis spectrum being whether or not the marginal ore should be blended in with the ore or milled separately at the end of mine life.

A recalculation of the drillhole profit/loss calculation (Fig. 13.1.2.3) utilizing the marginal ore results in an improved net profit is shown in Table 13.1.2.5.

The net value increases by $131,250, an increase of 16% over the initial evaluation.

To recap, the break-even analysis is utilized to establish the ultimate pit limits within justifiable economic constraints. Once established, the marginal ore reserve calculation allows a greater utilization of lower-grade material, resulting in improved overall economics.

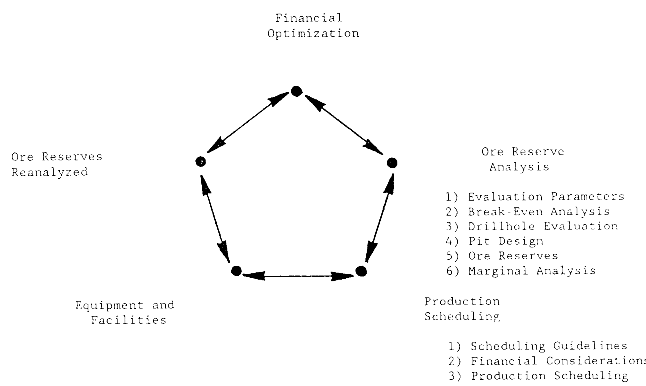

Fig. 13.1.2.6. Production scheduling.

13.1.2.2 Production Scheduling

Production scheduling (Fig. 13.1.2.6) is an important facet of mine planning. Once the mill feed grade has been established, maximizing the NPV of a property becomes highly dependent on scheduling. Scheduling determines mine life and, therefore, cash flows including capital requirements, operational costs, and revenues.

Initial production scheduling is based on marginal analysis ore reserves, a haulage study based on a conceptual pit design and overall facility layout.

SCHEDULING GUIDELINES. The following parameters provide guidelines for scheduling of the operation:

1. Minimizing preproduction costs.
2. Assuring adequate working room.
3. Smoothing of the stripping ratios.
4. Timely exposure of ore grade material.
5. Reclamation accountability.
6. Maximizing production.

Each point will be discussed in some detail.

Table 13.1.2.6. Cash Flow

Discount Factor @ 15% (end of period)		$ in thousands: Case I 2-yr preproduction ACF	DCF	Case II 1-yr preproduction ACF	DCF
Year 1	0.87	$ −500	$−435	$ −600	$−522
2	0.76	−500	−380	+500	+380
3	0.66	+600	+396	+500	+330
4	0.57	+600	+342	+500	+285
5	0.50	+600	+300	+500	+250
6	0.43	+600	+258	+1100	+473
7	0.38	+1100	+418		
Net Present Value			$+899		$+1196
Undiscounted Value		$ 2500		$ 2500	

Minimizing Preproduction Costs—Preproduction operational costs are treated as capital costs because they are incurred before production starts. These costs are not discounted and, in fact, should be assessed an interest charge for the time period used.

The following example only illustrates the time value of money, where two cases having the same total production costs ($5 million) and generated revenue ($7.5 million) are compared on slightly different production schedules.

The comparison involves a two-year preproduction period vs. a one-year preproduction period with a total five-year mill (revenue-generating) life for each case. Each example has a balanced production rate with stripping completed one year prior to the final production date in both cases.

Case I. Two-Year Preproduction
Total Cost:
Stripping $3 million ÷ 6 yrs = $ 500,000/yr
Mining and milling $2 million ÷ 5 years = $ 400,000/yr
Total revenue $7.5 million ÷ 5 yrs = $1,500,000/yr

Case II. One-Year Preproduction
Total Cost:
Stripping $3 million ÷ 5 yrs = $600,000/yr
Mining and milling $2 million ÷ 5 yrs = $400,000/yr
Total revenue $7.5 million ÷ 5 yrs = $1,500,000/yr

The *actual cash* flows (ACF) are the dollars spent each year, while the *discounted cash flows* (DCF) indicate the value today of future dollars.

The cash flow differences resulting from the different preproduction periods are shown in Table 13.1.2.6.

This example is oversimplified, but it stresses the importance of maximizing cash flow by lessening the preproduction commitment.

Contract stripping may also prove to be advantageous if used to the point where a positive cash flow is realized. Capital commitment for equipment and facilities can then be deferred. Here again, a cash flow analysis should be the determining factor.

Assuring Adequate Working Room—Especially in projects where the stripping ratio varies from one area to another, flattened working slopes should be used up to the point where the final pit configuration is intercepted. As mentioned under pit design, the flattening is accomplished by widening the benches. The resulting working areas are less congested, which provide for a smoother operational flow and minimizes traffic hazards.

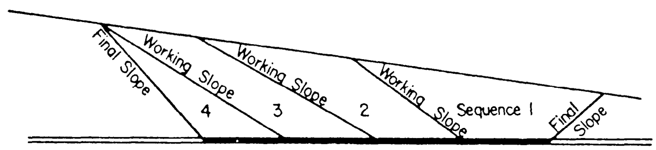

Fig. 13.1.2.7. Working slopes.

Also slope stability is improved as the back slope is flattened. Allowing the degree of slope flattening to fluctuate with the overburden ratio allows for a constant production rate and a predictable ore availability (Fig. 13.1.2.7).

Smoothing of the Stripping Ratios—For initial scheduling of operations, the average stripping ratio (tonnage basis) is used for determining working shifts and equipment required. It is important to avoid peaking the ratios as scheduling or capital requirements may become excessive.

Obvious exceptions exist, however. In preproduction stripping, it could be advantageous to commence operations in a low overburden-to-ore-ratio area that would only require utilization of a portion of the eventual fleet. Also, in instances where the stripping ratios change predictably over large areas such that equipment can be added or subtracted at a future date, it then becomes the average ratio of each area that becomes the scheduling basis.

Timely Exposure of Ore Grade Material—Proper sequencing is achieved through incremental pit design. Each increment is directly related to mill requirements within specific time constraints. Mining sequences are tentatively established and then analyzed to set the most logical development program. The mine sequencing then assures a predictable mill feed.

Reclamation Accountability—Environmental regulations require proper production planning and scheduling to minimize the costs associated with mine reclamation, The items most effectively handled by efficient planning are (1) returning ground contours to approximate premine conditions, (2) minimizing surface depressions to the greatest extent possible, and (3) revegetating disturbed areas.

Selected backfilling can create new slope angles within open pits and, in some instances, can completely fill portions of pits to achieve original contour. Generally, backfilling shortens waste haul cycles by reducing haul grades and distances. Waste dumps can be constructed on receding lifts, allowing the creation of flatter than "angle of repose" slopes with only a minimum of dozer work required. Whatever topsoil exists should be stripped and stockpiled for reemplacement at a future date.

With scheduling and design addressing reclamation requirements up front, operational efficiency is improved. The equipment and operating personnel required to do the work can then be added into the overall analysis.

Maximizing Production—The following points provide for more efficient scheduling and equipment utilization: (1) avoid excessive shovel moves, (2) minimize number of working areas, (3) work the lowest number of benches possible at any one time, and (4) reduce haul distances and ramp grades when practical. Increased productivity lowers operating costs. Additionally, rotating four crews through a seven-day week (20 shifts/week) gives the best utilization of equipment.

FINANCIAL CONSIDERATIONS. Property life directly influences the amount of capital expenditure required to produce a future income. While the evaluation of alternatives may identify an optimum-sized operation, it is wise to remember that mining plans are based on present-day parameters and, therefore, subject to changeover time.

Influencing factors are (1) capital and operating cost changes, (2) new mining and milling innovations, (3) increased knowledge of the deposit, and (4) changes in commodity value.

Ore and low-grade stockpiles allow the flexibility to supply a constant feed grade to the mill and also enable the producer to adjust to changing conditions. In addition, setting aside lower-grade ore for processing at the end of the operation could extend mill operations and financial profitability. Another form of optimization involves limited high-grading to cover spot sales, as outlined in the following example.

As applied to the example set forth in the text, the following considerations are offered. During mine life, a spot sale was contracted for over and above the commitments under existing contracts. The sale generates a $5/lb or $11/kg premium ($30/lb or $66/kg). The extra pounds are only available as above-average-grade ore, as the mill is operating at the designed capacity of 3000 tons/day (2720 t). The spot sale contract would be for an additional 500,000 lb (226,800 kg) during the third year of a 20-year production life.

The requirements for added production are:

$$\text{Tons/year @ } X\% = \text{original contract} + \text{spot sale}$$
$$1{,}095{,}000 \text{ @ } X\% = 1{,}655{,}640 \text{ lb/yr} + 500{,}000 \text{ lbs}$$

$$X\% = \frac{2{,}155{,}640}{1{,}095{,}000 \times 0.9} = 2.19 \text{ lb/ton} = 0.109\%$$

The high-grading would reduce the remaining grade of the ore body:

Ore Reserves	22 million tons	@ 0.084%
− 2 yrs production	− 2.2 million tons	@ 0.084%
− 3rd year production	− 1.1 million tons	@ 0.109%
Remaining Reserves	18.7 million tons	@ .082%
	(17.0 million t)	

Using the precalculated production costs of $34.70/ton (Table 13.1.2.3), a comparison of cash flows is calculated from the beginning of production through property life.

The value per ton (grade converted to recovered pounds × selling price) minus production cost equals the net value per ton. The results are summarized in Table 13.1.2.7.

Case I. Uniform Production

Discounted @ 15% on a 20-year life:

$3,394,500/year × 6.712 (cumulative PW factor) = $22,783,884

Table 13.1.2.7. Annual Cash Flow

Grade %	Net Value/ton	Annual Value @ 3000 tons/day
0.084	$ 3.10	$ 3,394,500
0.109	$14.35	$15,713,250
0.082	$ 2.20	$ 2,409,000

Conversion factor: 1 ton = 0.9072 t.

Table 13.1.2.8. Working Efficiencies

Classification	%	Working Time min/hr
Favorable	92	55
Average*	83	50
Unfavorable	75	45

*Used in Table 13.1.2.10

Case II. Inclusion of Spot Sale

Yrs 1 and 2	$ 3,394,500 × 1.74 (cumulative PW factor)	= $ 5,906,430
Yr 3	$15,713,250 × 0.71 (mid-period PW)	= $11,156,407
Yrs 4 thru 20	$ 2,409,000 × 4.26 (cumulative PW factor) 20th yr − 3rd yr)	= $10,262,340
	NPV	= $27,325,177

The illustration serves to point out that the time value of money is as important when optimizing production scheduling and resulting income as preproduction capital outlays and resulting income, as previously illustrated in Table 13.1.2.6.

Production scheduling guidelines provide the basis for alternative comparisons on which the highest NPV is determined. The basis for establishing alternatives is to vary the project duration (size).

The following discussion on production scheduling concentrates on the development of generalized operating costs for different production rates. These costs are applied to the current example.

PRODUCTION SCHEDULING. Scheduling is limited by constraints derived from historical data, regional information, and estimates based on industry-generated records.

Constraints by Equipment Type—

1. ELECTRIC SHOVEL APPLICATION.

(a) Diversity Factor: This refers to time lost due to moving, cleanup, and queuing of trucks. It is expressed as a factor of effective loading time. A common factor of 0.83 (based on industry time studies) is used here.

(b) Effective Loading Time: This means full shift minus time lost to lunch and waiting on trucks. Typical times are (minutes/shift)

480/full shift
− 30 lunch period
− 10 beginning of shift truck travel time
− 10 (5 lost on each side of lunch period)
− 10 end of shift truck travel time
420/net shift

2. TRUCK APPLICATION.

(a) Job Efficiency: This refers to the minutes of actual production during a 60-min hour. Manufacturers' data that apply to all equipment, except electric shovels, are listed in Table 13.1.2.8. The average efficiency may be used for evaluations until actual times are verified.

Table 13.1.2.9. Shovel Shifts (based on a 7-day working week)

No. Shovels	Shifts/ week	Working Shifts/ week	Availability %	Designation
1	21	10	48	unacceptable
	21	15	71	acceptable
	21	20	95	unrealistic
2	42	30	71	acceptable
	42	35	83	marginally achievable
3	63	45	71	acceptable
	63	50	79	acceptable
	63	55	87	unrealistic

(b) Production Time: This refers to operating time (full shift minus lunch period) times job efficiency (queuing, spotting, fueling, maintenance checks, etc.) minus travel time (to and from parking lot):

$$8 \text{ hr} - 0.5 \text{ hr} = 7.5 \text{ hr} \times 50 \text{ min/hr}$$
$$= 375 \text{ min} - 20 \text{ min} = 355 \text{ min/shift}.$$

Travel time (to and from parking lot) is optional. Many operations utilize 50 min/hr to account for both inefficiency and the extra travel time. Distance could be a factor when making this determination.

3. SUPPORT EQUIPMENT. Operating time is also estimated to be 7.5 hr/shift, but the actual production time will vary with property layout, scheduling procedures, etc.

4. FRONT-END LOADERS. Production time is calculated in the same manner as for trucks when the unit returns to the parking lot at the end of each shift, and as for a shovel when it remains in the work area between shifts.

Overall Constraints—

1. ANNUAL WORKING DAYS. This calculation includes the total days per year (365) minus holidays and unscheduled shutdowns (weather, labor disruptions, and major equipment breakdowns):

$$365 \text{ days/yr} - 10 \text{ holidays} - 5 \text{ unscheduled days} = 350 \text{ days/yr} \div 7 \text{ days/wk} = 50 \text{ wks/yr}$$

2. AVAILABLE SHIFTS. Selection of the number and duration of working shifts per week depends on required production and equipment utilized. Property life alternatives set annual production requirements. An 8-hr shift is the most efficient time allocation, based on overtime considerations and long-term productivity. It is also possible to operate on a 24-hr/day basis if required. Only 8-hr shifts are used in this example.

3. MECHANICAL AVAILABILITY. *Mechanical availability* is the availability after mechanical repair, preventive maintenance, and servicing have been accounted for.

A practical, long-range availability for shovel scheduling is in the range of 70 to 80% of total shifts available, depending upon working conditions and preventive maintenance procedures (Table 13.1.2.9).

Enough trucks should be purchased to provide 100% coverage of scheduled loader capacity. Using a 75% mechanical availability factor, full coverage is ensured by adding the appropriate number of extra trucks to the fleet.

Table 13.1.2.10. Scheduling Constraints

Shovels	Diversity Factor	0.83
	Effective loading time	420 min/shift
Trucks	Job efficiency	50 min/hr
	Production time	355 min/shift
Working weeks/year		50
Working shift duration		8 hours
Coverage: Shovels		70–80%
Trucks		100%
Manpower Availability Factor		1.10

$$\frac{\text{number of trucks required for loader coverage}}{\text{availability}} = \text{fleet required} \qquad (13.1.2.5)$$

$$\frac{8 \text{ trucks}}{0.75} = 10.7 \text{ or } 11 \text{ trucks}$$

4. MANPOWER. The manpower requirement multiplied by the availability factor yields the total number of individuals required. The availability factor is a variable based on local averages of vacations and absenteeism. For this example, absenteeism is set at 6% and a vacation average of 2.0 wk/yr as 4%, making the total time loss 10%. The manpower availability factor, then, is

$$100\% + 10\% \text{ loss} = 110\% \text{ or } 1.10$$

The total manpower required is

$$\frac{\text{No. working shifts} \times \text{hr/shift}}{\text{working hr/shift}} \times \text{availability factor} \qquad (13.1.2.6)$$

For example, assume there are 35 truck shifts/week:

$$\frac{35 \text{ shifts} \times 8 \text{ hr/shift}}{40 \text{ working hr/wk}} = 7 \text{ men} \times 1.10$$
$$= 7.7 \text{ or } 8 \text{ employees required}$$

In this case, the extra person is assigned to the labor pool and is used as required.

Table 13.1.2.10 recaps the constraints that are used later for example calculations.

5. PROPERTY LIFE. Maximum property life should be another limiting factor. Permanent installations (i.e., shops, mills, etc.) tend to become outmoded and wear beyond the point of reasonable repair. As a practical limiting factor, this example uses a maximum property life of 30 years as a basis for evaluation.

13.1.2.3 Equipment and Facilities

Open pit mining is becoming increasingly capital intensive as pits are enlarged and lower-grade ore is utilized. Contributing factors are (1) increased mechanization to handle the larger tonnages, (2) lower-grade ore resulting in higher plant investment per unit of refined product, (3) larger operations requiring greater preproduction cash outlays, and (4) environmental impact mitigation requirements.

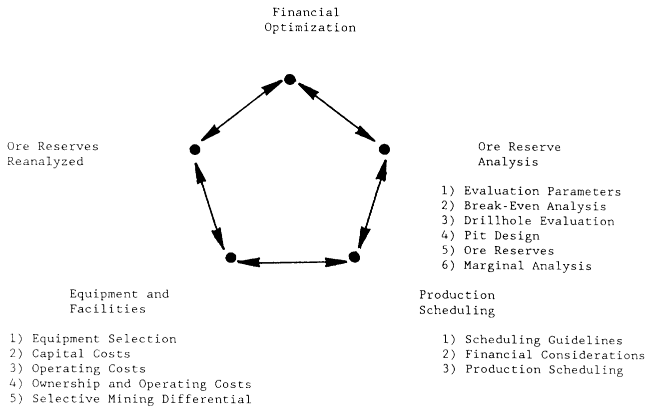

Fig. 13.1.2.8. Equipment and facilities.

Table 13.1.2.11. Overburden Swell

Lithology	% of total	% swell (decimal equiv.)	Weighted average % swell
Gravel	20	0.12	2.4
Clay	20	0.25	5.0
Sand-loose	30	0.10	3.0
Sandstone cemented	30	0.50	15.0
		Total	25.4

The major capital requirements that are a part of the final investment analysis are (1) plant, mobile units, and ancillary equipment, (2) preproduction stripping and mining, (3) spare parts inventory, (4) plant start-up, (5) working capital, (6) licensing and environmental assessments, and (7) cost of capital.

Both capital requirements and subsequent operational costs are developed at length to provide the basis for the selection of loading and haulage equipment. One should minimize and/or delay capital expenditures as long as feasible to improve the NPV. Capital expenses that do not lower production costs and increase earnings should not be considered. This segment is divided as shown in Fig. 13.1.2.8.

EQUIPMENT SELECTION. The selection of a fleet of equipment is accomplished by comparing production requirements against the capital and operating costs of appropriate alternatives. A pit haulage study is required to size and select types and number of haul trucks. Production requirements, rock characteristics, and desired flexibility are parameters guiding selection of the primary loaders. Equipment size should be standardized whenever possible.

Basic requirements for loader selection include rock breakage patterns, mine life, size of haulers utilized, tonnage factor, and swell factor. From drillhole log composites, lithologies are expressed as a percentage of the overburden. Using published tables, the weighted averages are made (Table 13.1.11).

$$\text{Swell factor} = \frac{1}{1 + 0.254} = 0.8$$

The tonnage factor for broken muck is

$$\text{TF} = \frac{\text{in-place TF}}{\text{swell factor}}$$

$$= \frac{16 \text{ ft}^3/\text{ton}}{0.8} = 20 \text{ ft}^3/\text{ton } (0.62 \text{ m}^3/\text{t}) \quad (13.1.2.7)$$

Table 13.1.2.12. Typical Electric Shovel Capacities

Shovel Size, kW	250	375	525	625	725
Nominal dipper, yd³	6	12	17	22	30
Nominal cycle time, sec	22	25	27	29	31
Tons/dipper	7.3	14.6	20.7	26.7	36.5
Nominal production Tons/shift	6,940	12,198	16,035	19,278	24,626

Conversion factors: 1 ton = 0.9072 t, 1 yd³ = 0.7646 m³, 1 hp = 0.7457 kW.

Table 13.1.2.13. Stripping Requirements

		Mill size, tons/day			
	Shifts/week	2,000	3,000	5,000	7,000
Property Life, yrs		30	20	12	8.6
Required Stripping, 10⁶ tons/yr		12.1	18.2	30.3	42.3
Required Stripping, tons/shift	15	16,178	24,267	40,400	56,400
	20	12,100	18,200	30,300	42,300
Required Stripping—multiple shovels, tons/shift/shovel	30	8,089	12,133	20,200	28,200
	40	—	9,100	15,150	21,500
	45	—	8,089	13,467	18,800
	60	—	—	10,100	14,100

Conversion factor: 1 ton = 0.9072 t.

$$\frac{27 \text{ ft}^3\text{yd}^3}{20 \text{ ft}^3/\text{ton}} = 1.35 \text{ tons/yd}^3 \ (1.60 \text{ t/m}^3)$$

Using an average bucket fill factor of 0.9, the dipper capacity is bucket capacity (yd^3) × tons/yd^3 × fill factor = tons/dipper:

$$6 \text{ yd}^3 \times 1.35 \text{ tons/yd}^3 \times 0.9 = 7.3 \text{ tons/dipper } (6.6 \text{ t})$$

Shovel production is

min/shift × diversity factor × cycles/min × bucket capacity = tons production/shift:

$$420 \text{ min} \times 0.83 \times \frac{60 \text{ sec}}{22 \text{ sec}} \times 7.3 \text{ tons} = 6940 \text{ tons (6296 t)/shift}$$

For the sake of simplification, only electric loading shovels are evaluated in this exercise. However, large front-end loaders and hydraulic face shovels offer advantages where increased mobility and lower capital investment may be required. Table 13.1.2.12 is based on typical manufacturers' recommendations.

The example in Table 13.1.2.13 utilizes the ratio between the stripping tonnage of 364 million tons (330 million t) and 22 million tons (20 million t) of ore as generated in the topic on marginal analysis (see Table 13.1.2.4). Stripping requirements are based on production levels and the number of shovel shifts scheduled.

Using the required stripping tons per shift, the shovels are selected by matching the nominal shovel production per shift to the production requirements for each case (compare Tables

Table 13.1.2.14. Dipper Selections

Shovel size kW	Dipper size yd^3	Tons/dipper	Tons/number passes 4	5	6
250	6	7.3	29.2	36.5	43.8
	7	8.5	34.0	42.5	51.0
	8	9.7	38.8	48.5	54.5
	9	10.9	43.6	54.5	65.4
375	12	14.6	58.4	73.0	87.6
	13	15.8	63.2	79.0	94.8
	14	17.0	68.0	85.0	102.0
525	17	20.7	82.8	103.5	124.2
	18	21.9	87.6	109.5	115.5
	19	23.1	92.4	115.5	138.6
625	22	26.7	106.8	133.5	160.2
	23	28.1	112.4	140.5	168.6
	24	29.2	116.8	146.0	175.2
	25	30.4	121.6	152.0	182.4
	26	31.6	126.4	158.0	189.6
	27	32.8	131.2	164.0	196.8
725	30	36.5	146.0	182.5	219.0
	31	37.7	150.8	188.5	226.2
	32	38.9	155.6	194.5	233.4
	33	40.1	160.4	200.5	240.6
	34	41.3	165.2	206.6	247.8

Conversion factors: 1 ton = 0.9072 t, 1 yd^3 = 0.7646 m^3, 1 hp = 0.7457 kW.

13.1.2.12 and 13.1.2.13). A property should not be dependent on one prime earthmover because of downtime considerations and reduced flexibility in the operation, and to reiterate, equipment should be standardized whenever possible. Therefore, the following examples consider only two or more of the same shovels as viable options.

It should be noted that the production from a nominally sized bucket does not always meet requirements. However, by upsizing the bucket within the manufacturer's guidelines, the required capacity will be generated.

Once a shovel is selected, the dipper capacity is sized to accommodate various truck capacities. Manufacturers' guidelines suggest that with the overburden specific weight of this example, dipper capacities shown in Table 13.1.2.14 may be utilized. Four or five passes per truck is considered optimum, although six is acceptable under normal conditions.

Truck fleet options are generated by matching dipper sizes to number of passes to rated hauler capacities. Maximizing truck size to shovel capability is an effective objective, although adverse haulage grades, equipment availability, and cumulative bucket capacities also contribute to the final selection.

A modifier to be used when establishing shovel cycle times is to increase cycle times by 1 second for each 2 yd^3 (1.5 m^3) above nominal dipper capacity. It also should be noted that nominal truck capacity may be exceeded by up to 5%; however, increased maintenance and tire wear can be expected under these conditions.

Although not limited to the shifts or truck sizes in Table 13.1.2.15, the choices that follow have been made to simplify the comparison process.

After sizing the truck, it is important to select the engine horsepower, gear ratios, and tire size necessary to provide satisfactory performance under anticipated working conditions. Manufacturers provide performance charts that relate the power provided by the vehicle power train (rimpull) to the vehicles weight and total rolling resistance (RoRi).

Rolling resistance is the measure of force required to overcome the retarding effect between the tires and the ground. RoRi is primarily the cumulative effect of vehicle weight, tire penetration, and road grades.

Vehicle weight is the primary factor in the amount of force required to overcome RoRi. Using level, compacted roads as the base, the RoRi is 40 lb/ton (18 kg/t) vehicle weight, or expressed as a percentage:

$$\frac{40 \text{ lb}}{2000 \text{ lb/ton}} = 2\%$$

Tire penetration or road flex is an added resistance factor due to soft conditions. Use 30 lb RoRi/in. (0.54 kg/mm) tire penetration. When expressed as a percentage:

$$\frac{30 \text{ lb}}{2000 \text{ lb/ton}} = 1.5\%/\text{in.} \ (0.06\%/\text{mm}) \text{ penetration}$$

In the planning stage some basic assumptions must be made regarding penetration in various areas. This example utilizes the following:

- Loading areas and waste dumps — 2 in.
- Approaches — 1 in.
- Established haul roads — 0 in. (base)

For grades both positive and negative, use 20 lb RoRi/ton (10 kg/t) per % grade. Expressed as a percentage, this becomes

$$\frac{20 \text{ lb}}{2000 \text{ lbs/ton}} = 1\%/\% \text{ grade}$$

To bring all the factors together, consider a truck leaving the shovel approach area on a gentle 2% uphill grade:

Table 13.1.2.15. Truck-Shovel Combinations

Property Life, yr	Shovel No., kW	Shifts	Dipper size, yd^3	Tons/ dipper	Cycle time, sec	No. passes	Tons/ load	Truck Capacity tons	Tonnage/shovel shift calculated	Tonnage/shovel shift required	Capacity %
30	2-250	30	8	9.7	23	5	48.5	50	8,821	8,089	109
20	3-250	45	8	9.7	23	5	48.5	50	8,821	8,089	109
	2-375	30	13	15.8	25	5	79.0	85	13,219	12,133	109
12	3-375	45	14	17.0	26	5	85.0	85	13,676	13,467	102
	2-525	40	19	23.1	28	5	115.5	120	17,256	15,150	114
	2-625	30	24	29.2	30	5	146.0	150	20,358	20,200	101
8.6	4-525	60	17	20.7	27	4	82.5	85	15,977	14,100	113
	3-625	45	24	29.2	30	5	146.0	150	20,358	18,800	108
	2-725	40	30	36.5	31	4	146.0	150	21,548	21,150	102

Conversion factors: 1 ton = 0.9072 t, 1 yd^3 = 0.7646 m^3, 1 hp = 0.7457 kW.

$$\begin{aligned}\text{Total RoRi} &= \text{weight } (2\%) + \text{tire penetration } (1 \times 1.5\%) \\ &\quad + \text{grade } (2 \times 1\ \%) \\ &= 2\% + 1.5\% \text{ (say } 2\%) + 2\% = 6\% \text{ total RoRi}\end{aligned} \tag{13.1.2.8}$$

The RoRi reduces the rimpull available from the power train. A loaded 50-ton (45-t) truck has a gross weight of approximately 175,000 lb (80,000 kg). A 6% RoRi reduces available rimpull by 10,500 lb (175,000 × 6%)(4760 kg). The remaining rimpull is available for acceleration.

Vehicle speed under a given set of conditions results from the relationship of the truck's weight and RoRi to the vehicle's performance chart, published by the manufacturer. The maximum speed, as determined by this relationship, is then modified by speed factors that convert maximum speeds to average speeds over segments of the haul.

The first step in establishing speed factors is to determine the vehicle weight to flywheel horsepower (FWHP) ratio:

$$\frac{\text{vehicle weight (with or without load)}}{\text{FWHP}} \tag{13.1.2.9}$$

A loaded 50-ton (45-t) hauler (175,000 lb or 80,000 kg) with 600 FWHP (450 kW) has a ratio of 282 lb/hp (175,000 ÷ (600) (170 kg/kW). This ratio is used to select the proper table to determine the speed factor. The factors are then obtained by matching the haul road length to the point where the truck is starting from 0 mph (or stopping), or where the grade is level, downhill or uphill and the unit is in motion when entering this section.

The average speed for each haul segment is obtained by multiplying the maximum speed times the speed factor (Table 13.1.2.16).

Speed limits become the final modifier in the calculation. Speeds should be based on reasonable guidelines to ensure safe operation. This example utilizes the following:

• Loading and dump areas	5 miles/hr
• Long uphill and level roads	25 miles/hr
• Downhill grades	15 miles/hr

The number of trucks needed to move the required tonnage is a time function involving the sum of fixed and travel times. Travel time is

$$\frac{\text{haul road segment length (ft)}}{\text{average speed (mph)} \times 88 \text{ (conversion factor)}} \tag{13.1.2.10}$$

The example in Table 13.1.17 typifies a truck cycle time of a 50-ton (45-t) unit traversing a complete load, haul, dump, and return cycle.

Haulage studies should consider the following:

1. Overall layout indicating centroids of pit excavation sequences and incremental waste dump growth.
2. Haul road widths, passing lanes, intersections, and curves
3. Pit profiles detailing level hauls and grades and the length of each.
4. Waste dump geometry including height, width, and traffic patterns.

Truck fixed times are nonproductive periods that include truck spotting, loading, and the time required to turn and offload at the waste dump. The times are based on industry norms and can be used or modified as local situations warrant.

A typical example for a 333-hp (250-kW) loader and a 50-ton (45-t) hauler is as follows:

Shovel spotting	0.42 min
Shovel loading	1.92 min
Turn and dump	0.85 min
Total fixed time	3.19 min

The total truck cycle time for the 50-ton (45-t) unit is

10.04 min travel time + 3.19 min fixed time = 13.23 min

The number of trucks required for the 30-year property life shown in Table 13.1.2.15 is determined as follows:

No. shovels and size (kW)	2 @ 250
Hauler capacity, tons	50
Individual truck cycles/shift (355 min ÷ 13.23)	27
Payload tons	48.5
Tons/shift/truck (27 × 48.5)	1,310
Required tonnage/shift	16,178
Required trucks/shift (16,178 tons ÷ 1,310)	12.4
Trucks required at 75% availability (12.4 ÷ 0.75) = 16.5 − use	17

Production scheduling is set by limiting stripping tonnage by truck coverage and not shovel capacity. The latter is usually

Table 13.1.2.16. Speed Conversion Factors (under 300 lb/hp only)

Haul Road Length, ft	Lever Haul Unit, starting from 0 mph	Unit in Motion When Entering Haul Road Section: Level	Downhill Grade	Uphill Grade Factor
0–200	0 – .40	0 – .65	0 – .67	1.00
201–400	.40 – .51	.65 – .70	.67 – .72	(Entrance speed greater than maximum attainable speed on section)
401–600	.51 – .56	.70 – .75	.72 – .77	
601–1000	.56 – .67	.75 – .81	.77 – .83	
1001–1500	.67 – .75	.81 – .88	.83 – .90	
1501–2000	.75 – .80	.88 – .91	.90 – .93	
2001–2500	.80 – .84	.91 – .93	.93 – .95	
2501–3500	.84 – .87	.93 – .95	.95 – .97	
3501 & up	.87 – .94	.95 –	.97 –	

Source: Adapted from Terex, 1970.
Conversion factors: 1 ft = 0.3048 m, 1 mph = 1.609 km/h.

Table 13.1.2.17. Pit Haulage Study

Segment length, ft	Grade %	RoRi %	Net Resistance %	Max. vel. (mph)	Speed factor	Avg. vel. (mph)	Travel time min	Remarks
200	0	5	5	11.5	0.4	4.6	0.49	@ shovel
200	0	4	4	22.0	0.65	14.3	0.16	
1875	8	3	11	9.0	1.0	9.0	2.37	ramp
3000	2	2	4	22.0	0.94	20.7	1.65	main road
200	0	5	5	11.5	0.33	3.8	0.60	dump
200	0	5	5	12.0	0.4	4.8	0.47	return
3000	−2	2	0	34.0	0.96	25.0*	1.36	cycle
1875	−8	3	−5	15.0	—	15.0*	2.27	
200	0	4	4	34.0	0.64	21.8	0.10	
200	0	5	5	12.0	0.33	4.0	0.57	
						Total Travel Time	10.04	

* Speed Limit
Conversion factors: 1 ft = 0.3048 m, 1 mph = 1.609 km/h.

Table 13.1.2.18. Total Truck Requirements

	Yr Mine Life/million tons/yr: 30/12.1	20/18.2		12/30.3			8.6/42.3		
No. shovels and power, kW	2-250	3-250	2-375	3-375	2-525	2-625	4-525	3-625	2-725
Truck size, tons	50	50	85	85	120	150	85	150	150
Total truck cycle time, min	13.23	13.23	13.83	13.83	14.50	15.05	13.61	15.05	14.62
Truck cycles/shift	27	27	26	26	24	24	26	24	24
Payload-tons	48.5	48.5	79	85	115.5	146	82.5	146	146
Tons/truck shift	1,310	1,310	2,054	2,210	2,772	3,504	2,145	3,504	3,504
Required tons/shift	16,178	24,267	24,267	40,400	30,300	40,400	56,400	56,400	42,300
Required operating trucks/shift	12.35	18.52	11.81	18.28	10.93	11.52	26.29	16.10	12.07
Total trucks required @ 75% availability	17	25	16	24	15	15	35	21	16

Conversion factors: 1 ton = 0.9072 t, 1 hp = 0.7457 kW.

slightly higher. When determining the total number of trucks required, round 0.00 to 0.29 to zero, and above 0.29 is rounded up. A recap of truck requirements is shown in Table 13.1.2.18.

CAPITAL COSTS. No attempt is made in this example to derive the costs of the mill and related administration and maintenance facilities nor of any offsite facilities. Offsites include access road, electric power, water, and natural gas. The assigned costs are listed in Table 13.1.2.19.

One should establish specifications for the major equipment. Work closely with vendors who not only have access to their technical data, but are also experienced with applications over diverse conditions. Once the major rolling stock has been speci-

Table 13.1.2.19. Mill and Offsites Costs

	Mill tons/day, cost $ millions			
	2000	4000	5000	7000
Offsites	$ 4	4	4	4
Mill	30	35	45	55
Total	$34	39	49	59

Conversion factor: 1 ton = 0.9072 t.

Table 13.1.2.20. Ancillary Capital Cost Breakdown

Classification	% of Total
Drilling	8
Blasting	2
Field support	10
Utilities	3
Buildings	5
Tools and spare parts	2
Total	30%

fied, send out requests for quotations to the manufacturers being considered. A cost pattern should emerge on which to base the estimate.

Accurate capital cost estimating of production equipment, both shovels and trucks, is essential to the evaluation for two reasons:

1. Overall costs are usually the basis for fleet selection.
2. The cost of ancillary equipment can be estimated on the basis of the major rolling stock.

As a general rule, capital committed to shovels and trucks can be the basis for prorating ancillary equipment capital costs for estimating purposes. The combined ancillary capital costs represent approximately 30% of the total capital costs. When expressed in terms of the shovel-truck cost, this percentage becomes 43% (3/7).

The breakdown in Table 13.1.2.20 is based on averages generated from equipment lists sufficient to handle the total material generated from 10,000- to 200,000-ton (9000- to 180,000-t)/day operations. Individual situations may dictate different percentages, but they should be altered only if sufficient detail has been developed to warrant changes.

Drilling utilizes rotary production units with average rock condition suggesting the following relationship: one large diameter (9 to 12 in., or 229 to 305 mm) rotary drill per 30,000 to 40,000 tons (27,000 to 36,000 t)/day of stripping. Smaller drills (6 to 9 in., or 152 to 229 mm) will suffice for the lower tonnage operations at the rate of one per 10,000 to 15,000 tons (9000 to 13,600 t)/day.

Blasting covers the explosive-handling vehicles and the air-track drills that are required for secondary breakage in most operations.

Field and miscellaneous support equipment includes dozers, patrols, light duty and maintenance vehicles and cranes. The support equipment is very dependent upon local conditions, making detailed relationships an impractical matter. However, the overall relationship to the total capital remains fairly constant.

Utilities include transformers, power distribution, water, air, gas, firewater, and telephone lines.

Buildings cover administration, maintenance, welding, lube and tire shops, changerooms, and warehousing.

Table 13.1.2.21. Production Equipment Capital Costs

Shovels		Trucks	
Size, kW	Cost, $ millions	Capacity, tons	Cost, $1000
250	2.0	50	400
375	2.5	85	500
525	3.0	120	600
625	3.5	150	700
725	4.0		

Conversion factors: 1 ton = 0.9072 t, 1 hp = 0.7457 kW.

Tools and initial spare parts inventory include only those purchased prior to operations start-up. Preproduction mining is also a capital item, expensed for tax purposes. However, it will be addressed only as a function of production scheduling in this example.

Capital costs of the shovels and trucks are given in Table 13.1.2.21.

The overall capital costs for each case (Table 13.1.2.15) are assembled in Table 13.1.2.22.

It should be noted that with options utilizing more units of smaller-sized equipment, a higher initial capital cost is generally experienced.

OPERATING COSTS. Capital requirements alone cannot serve as the basis for selection of production and ancillary equipment. Combine operational and capital costs to arrive at a single-fleet choice for each property life alternative. The four fleets selected provide the framework for the financial analysis and ultimate pit optimization.

Again, as in capital costs, no attempt is made to derive mill operating costs. Assigned costs are listed in Table 13.1.2.23.

The first step is to determine the operating hours/year for the loaders and the haulers involved in each option (Table 13.1.2.24).

Hourly operating costs are generated based on field experience and manufacturer's guidelines.

A 525-kW shovel will be costed out in detail. The operating costs of other shovels are calculated in the same manner, but they are only summarized here. The primary costs include electric power, labor, maintenance, and repair.

Power costs are generated by multiplying kilowatt hours by the commercial rate. Consumption is rated at 525 kWh with the rate being $0.05/kWh:

$$525 \text{ kWh} \times \$0.05/\text{kWh} = \$26.25/\text{hr}$$

Labor costs are calculated by multiplying combined operator and oiler rates by payroll burden. Next multiply that figure by 8 hr/shift divided by 7 productive hr/shift. Hourly rates are $12 and $11/hr. The payroll burden is 35%. Payroll burden includes vacation, holidays, overtime, unexcused absenteeism, payroll benefits, FICA, health insurance, unemployment insurance and workmen's compensation. Labor costs become

$$(\$12 + \$11) \times 1.35 \times 8 \div 7 = \$35.50/\text{hr}$$

Repair and maintenance is expressed as a cost/ton multiplied by the tons/hr. Assume that the cost/ton is $0.02 ($0.018/t) (experience factor). Production is 17,314 tons (15,707 t)/shift divided by 7 hr/shift equals 2473 tons (2244 t)/hr, or,

$$\$0.02/\text{ton} \times 2473 \text{ ton/hr} = \$49.50/\text{hr}$$

The total cost of power, labor, and repair and maintenance is $111.75/hr (Table 13.1.2.25).

Table 13.1.2.22. Overall Capital Costs For Equipment and Facilities

Property life, yr	Production combinations: Shovels No., kW	Trucks No., tons	Costs, $ millions: Fleet	Ancillaries	Mill and Offsites	Total
30	2-250	17-50	$10.8	$ 4.6	$34	$ 49.4
	3-250	25-50	16.0	6.9	39	61.9
20	2-375	16-85	13.0	5.6	39	57.6
	3-375	24-85	19.5	8.4	49	76.9
	2-525	15-120	15.0	6.5	49	70.5
12	2-625	15-150	17.5	7.5	49	74.0
	4-525	35-85	29.5	12.7	59	101.2
	3-625	21-150	25.2	10.8	59	95.0
8.6	2-725	16-150	19.2	8.3	59	86.5

Conversion factors: 1 ton = 0.9072 t, 1 hp = 0.7457 kW.

Table 13.1.2.23. Mill Operating Costs

Costs $/ton	Mill Capacity, tons/day: 2000	3000	5000	7000
Operating	$ 7.25	$ 7.00	$ 6.50	$ 6.00
Directs	3.60	3.50	3.30	3.10
Indirects	2.90	2.80	2.60	2.40
Totals	$13.75	$13.30	$12.40	$11.50

Conversion factor: 1 ton = 0.9072 t.

Table 13.1.2.24. Equipment Operating Hours per Year

Property life, yr	Option	Shovels and Trucks kW and ton	Hr/shift	Operating Units No.	Shifts/wk	Hr/yr
30	(1)	250	7	2	15	10,500
		50	7.5	13		73,125
20	(1)	250	7	3	15	15,750
		50	7.5	19		106,875
	(2)	375	7	2	15	10,500
		85	7.5	12		67,500
12	(1)	375	7	3	15	15,750
		85	7.5	18		101,250
	(2)	525	7	2	20	14,000
		120	7.5	11		82,500
	(3)	625	7	2	15	10,500
		150	7.5	12		67,500
8.6	(1)	525	7	4	15	21,000
		85	7.5	26		146,250
	(2)	625	7	3	15	15,750
		150	7.5	16		90,000
	(3)	725	7	2	20	14,000
		150	7.5	12		90,000

Conversion factors: 1 ton = 0.9072 t, 1 hp = 0.7457 kW.

The operating costs of a 120-ton (109-t) hauler is detailed utilizing experience factors and local wage rates.

Operator costs are obtained in the same manner as the shovel labor is calculated:

$$\frac{\$11.00/\text{hr} \times 1.35 \times 8 \text{ hr/shift}}{7.5 \text{ productive hr/shift}} = \$15.85/\text{hr}$$

Tire cost is the cost/tire times the number of tires divided by the hours of expected life:

$$\frac{\$7500/\text{tire} \times 6 \text{ tires}}{4000 \text{ hrs}} = \$11.25/\text{hr}$$

Fuel cost is the gallons consumed/hour times cost/gallon:

$$30 \text{ gal/hr} \times \$0.80/\text{gal} = \$24.00/\text{hr}$$

Maintenance, service, and repair are covered by utilizing the relationships between:

1. Mechanics wages, including burden ($\$11/\text{hr} \times 1.35 = \$14.85/\text{hr}$).
2. Wearable items, historically related to truck horsepower.
3. Working conditions.

The first relationship, between mechanics' wages and truck size, generates a cost factor of wearable items and mechanical labor. Once derived, the factor is related back to truck size to generate the cost per hour of maintenance, service, and repair. The cost per hour is then modified by the working conditions, expected to be experienced-based on haulage analyses and overall site conditions.

Working conditions impact the costs as follows:

Average:	costs @ 100%
Favorable:	Includes light duty resulting from gentle grades, low RoRi, minimum traffic restrictions, etc. Cost × 80% (lower limit)
Adverse:	Includes long ramps, steeper grades, high RoRi, adverse weather, etc. Costs × 130% (upper limit)

Manufacturers' graphs are utilized to obtain costs suitable for estimating purposes. For this case, 120-ton (109-t) haulers and a cost of $25.90/hr under average conditions for maintenance, service, and repair will be utilized.

Total cost of operators, tires, fuel and maintenance, service and repair for the 120-ton (109-t) hauler is $77.00/hr. Table 13.1.2.26 lists operating costs for all trucks used in this comparison.

Table 13.1.2.25. Shovel Operating Costs

Shovel size, kW	Operating cost, $/hr
250	$ 71.45
375	87.65
525	111.75
625	121.80
725	134.65

Conversion factor: 1 hp = 0.7457 kW.

Table 13.1.2.26. Truck Operating Costs

Haul Truck, tons	Operating cost, $/hr
50	49.22
85	69.50
120	77.00
150	83.74

Conversion factor: 1 ton = 0.9072 t.

Table 13.1.2.27. Ancillary Operating Cost Breakdown

Classification	% of Total
Drilling	7
Blasting	3
Roads	6
Dumps	4
Total	20%

The fleet operating costs become the basis for estimating the ancillary operating costs. Allocation of operating costs is highly variable. Expected operating conditions are influenced heavily by weather conditions, rock characteristics, general terrain, etc., and should so be anticipated when assigning operating costs. This example utilizes the generalized percentages outlined in Table 13.1.2.27.

The total ancillary operating costs represent 20% of the total operating costs. When expressed in terms of shovel-truck cost, this percentage becomes 25% (2/8).

The production fleet overall operating cost/hour for mine lives and equipment combinations is given in Table 13.1.2.28 based on the same equipment options shown in Table 13.1.2.22.

Owning and Operating Costs. Capital costs are depreciated, using the straight-line approach for simplicity, over the expected useful life of the individual components. The results are expressed in dollars/operating hr and, when combined with operating costs, provide total *ownership and operating costs* (0 and 0) from which a single fleet will be selected for each mine life.

Ownership costs are obtained by dividing capital costs by the expected life of each type of equipment. Ancillary costs are split between mobile and stationary equipment because of the difference in their expected lives. Operating hours for mobile ancillary equipment should be the same as the truck fleet. The stationary equipment is utilized for 8 hr/shift (Table 13.1.2.29).

A compilation of the 0 and 0 costs for the 12-year life case, utilizing 525-kW shovels and 120-ton (109-t) trucks, is shown in Table 13.1.2.30. Total 0 and 0 costs for the equipment combination options are listed in Table 13.1.2.31.

Only the cases with the asterisk are used for further mine comparisons. The following points should be noted:

1. Utilizing 20 shifts/week results in the lowest initial capital investment but not necessarily the lowest 0 and 0 cost.
2. The largest equipment produces the lowest overall 0 and 0 cost.
3. In the 8.6-year example, case 2 was selected over case 3 because of the increased scheduling and operations flexibility resulting from more available equipment and shifts if required. This option should only be used where 0 and 0 costs are similar.

A summation of stripping costs based on total required operating hours is shown in Table 13.1.2.32. The costs are expressed in dollars/ton for ensuing financial analyses.

Selective Mining Differential. Under many circumstances, enough information now has been generated to recalculate the ore reserves based on appropriate 0 and 0 costs for the alternatives being considered. However, this example is somewhat more involved in that the selective mining requirements of the ore zone require a separate calculation of direct mining costs. As in any instance of highly selective ore mining, operating costs run higher than stripping costs because of slower rates, tighter supervisory control, and extra equipment requirements.

Refined mine operating costs are not calculated here in detail. However, by using parameters as previously generated, new operating costs are established.

The equipment shown in Table 13.1.2.33 is considered capable of producing 3500 tons (3175 t)/shift of combined ore and associated waste. Being a mid-range calculation, the operating costs/ton are appropriate for all four options.

In cases where mining has no bearing on stripping fleet selection, additional costs are not added to the total 0 and 0 costs. Mining costs shown in Table 13.1.2.33 will contribute to revised ore reserves and are included in the cash flow analysis.

The direct operating cost/ton is

$$\frac{\$6102.50/\text{shift}}{3500\ \text{tons/shift}} = \$1.74/\text{ton}\ \ (\$1.92/\text{t})$$

The mining fleet will be depreciated over an average life 50,000 hours. The ownership cost/ton milled is the same in all four cases as mining-fleet size is directly proportional to mining tonnage requirements.

$$\text{Ownership period: } \frac{50{,}000\ \text{hrs}}{5625\ \text{hrs/yr}} = 8.89\ \text{yr}$$

Tonnage: 8.89 yr × 2,100,000 tons of ore, low-grade and associated waste/yr. = 18,699,000 tons (16,964,000 t)

$$\text{Ownership cost: } \frac{\$3{,}975{,}000\ \text{capital cost}}{18{,}669{,}000\ \text{tons}} = \$0.21/\text{ton}$$

($0.23/t)

Table 13.1.2.28. Overall Fleet Operating Costs

Mine Life, yr	Fleets: Shovels No., kW	Fleets: Trucks No., ton	Production Costs, $/hr	Ancillary Costs, $/hr	Overall Cost, $/hr
30	1) 2-250	13-50	782.76	195.69	978.45
20	1) 3-250	19-50	1,149.53	287.38	1,436.91
	2) 2-375	12-85	1,009.30	252.33	1,261.63
12	1) 2-375	18-85	1,513.95	378.49	1,892.44
	2) 2-525	11-120	1,070.50	267.63	1,338.13
	3) 2-625	12-150	1,248.48	312.12	1,560.60
8.6	1) 4-525	26-85	2,254.00	563.50	2,817.50
	2) 3-625	16-150	1,705.24	426.31	2,131.55
	3) 2-725	12-150	1,274.18	318.55	1,592.73

Conversion factors: 1 ton = 0.9072 t, 1 hp = 0.7457 kW.

Total 0 and 0 costs for the mining fleet are $1.74 + $0.21 = $1.95/ton ($2.15/t).

13.1.2.4 Ore Reserves Reanalyzed

The original ore cutoff grade of 0.06% was based on preliminary data. New cutoff grades are now established reflecting respective operating costs of each case. Reanalysis utilizes projected operating costs to modify criterion illustrated in Fig. 13.1.2.9.

CUTOFF GRADE. The calculation utilizes the initial cutoff grade of 0.06% and original waste and low-grade-to-ore ratios to establish new cutoff grades. The second step uses the revised cutoff grades to establish new waste-to-ore ratios.

Direct operating costs are tabulated in Table 13.1.2.34 and extended to calculate total direct operating costs for each ton of ore processed. The resultant cutoff grades are then calculated.

Ore reserves generated for each case are different. Ore grades change as do the waste to ore ratios, as shown in Table 13.1.2.35.

DESIGN ALTERNATIVES. At this point it would be prudent to reanalyze drillhole data as some alteration in pit design may result. This example will not make that iteration, because all four cases bracket the original example without significant deviation (± 15% of grade).

MARGINAL ANALYSIS. The new stripping ratios have the potential to further alter cutoff grades. This process can be re-

Table 13.1.2.29. Anticipated Equipment Life

Equipment	Life, hr
Shovels	100,000 hr
Trucks	50,000 hr
Ancillary—mobile 67%	25,000 hr
—stationary 33%	96,000 hr

Table 13.1.2.30. Total Fleet Ownership and Operating Cost Breakdown (12-yr life case)

Case	Capital $ million	Ownership $/oper hr		Operating $/hr	Total/unit $/hr	Hours/unit/yr	Total/yr $1000
2,525-kW shovels	6.0	60.00		223.50	283.50	7000	1985
15,120-ton trucks	9.0	180.00	(11)	847.00	1,027.00	7500	7703
Ancillary, mobile @ 67%	4.4	176.00		267.63	443.63	7500	3327
stationary @ 33%	2.1	21.88		—	21.88	8000	175
Totals	21.5	437.88		1,338.13	1,776.01		13,190

Table 13.1.2.31. Summary of Total Fleet Ownership and Operating Costs

Mine Life, yr	Option	Capital $ million	Ownership $/oper. hr	Operating $/hr	Total $/hr	Max. operating hr/yr	Total $1000/yr
30	*1	15.4	315.63	978.45	1294.08	6000	7,216
20	1	22.9	467.96	1436.91	1904.87	6000	10,621
	*2	18.6	380.75	1261.63	1642.38	6000	9,161
12	1	27.9	568.17	1892.44	2460.61	6000	13,725
	2	21.5	437.88	1338.13	1776.01	8000	13,190
	*3	25.0	506.04	1560.60	2066.64	6000	11,517
8.6	1	42.2	853.75	2817.50	3671.25	6000	20,455
	*2	36.0	724.50	2131.55	2856.05	6000	15,903
	3	27.5	556.13	1592.73	2148.86	8000	15,956

Table 13.1.2.32. Stripping Costs

Mine Life, yr	$1,000/ yr	Million tons/yr	$/ton
30	5,450	12.1	0.45
20	7,031	18.2	0.39
12	8,687	30.3	0.29
8.6	11,853	42.3	0.28

Conversion factor: 1 ton = 0.9072 t.

Table 13.1.2.33. Mining Capital and Operating Costs

Equipment No. & description	Capital cost $1000	Operating cost $/hr	Hr/shift	Total Operating $/shift
2 Front-end loaders, 6 yd^3	600	52.00	7.5	780.00
2 Dozers with ripper, 400-hp	950	90.00	7.5	1350.00
7 Trucks, 35-ton	1925	46.00	7.5	2415.00
1 Rotary drill	450	37.00	7.5	277.50
4 Ore controllers & equipment	50	40.00	8.0	1280.00
Total	3975			6102.50

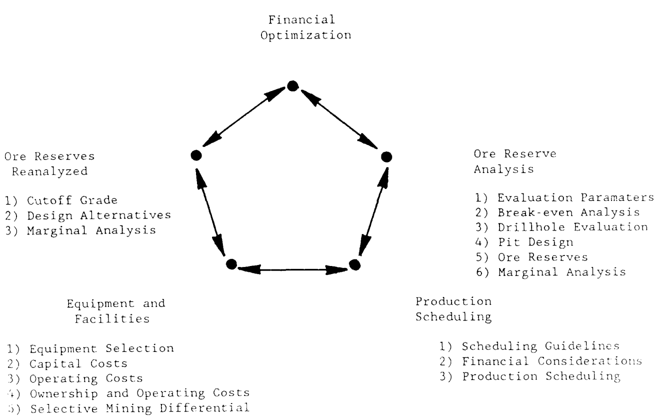

Fig. 13.1.2.9. Ore reserves reanalyzed.

peated, with diminishing returns, until no further change is noted. However, the accuracy at this stage is considered sufficient for a feasibility study.

The milling rate needs no adjustment as the upper half (0.045 → 0.06%) of the marginal ore reserves was used to set mill processing tonnage. Verification that marginal reserves (0.03 → .06%) still cover the four cases is analyzed in Table 13.1.2.36.

In each case there is sufficient marginal ore to satisfy milling requirements at the previously established milling rates. This example will then use the previously established milling grade of 0.084% for all four cases.

13.1.2.5 Financial Evaluation

The steps to be followed to obtain the financial evaluation are outlined in Fig. 13.1.2.10.

CASH FLOWS. Annualized cash flows need to be generated for use in the economic evaluation.

Annual revenue is determined by using the marginal analysis ore reserve grade of 22 million tons (20 million t) at 0.084%, a mill recovery of 90% and a selling price of $25/lb ($55/kg) (Table 13.1.2.37). Operating costs/ton of ore processed are summarized in Table 13.1.2.38.

Ownership costs thus far have included both *initial capital investment* (ICI) and replacement capital for direct comparative purposes. Setting up a cash flow statement requires that these costs be separated. Replacement capital applies to the stripping and mining fleets only. The mill and related facilities will be considered functional for the life of the property. Mill maintenance is a component of direct operating expenses.

Equipment replacement is based on scheduled hours of operation and expected useful life. Where equipment is spared or only selected shifts are worked, equipment operating hours should be adjusted accordingly (* in Table 13.1.2.39). For example, it can be demonstrated that mining capital costs already calculated are adequate for the 2000-ton/day (1800-t/day) at 2 shifts/day and 3000-ton/day (2700-t/day) cases, while the 5000- and 7000-ton/day (4500- and 6400-t/day) cases require capital increases of 50% and 100% respectively. The replacement factor is based on capital investment. Table 13.1.2.39 details the derivation of the replacement costs for case 1 at 2000 tons/day (1800 t/day) and 30- years life.

The annual replacement cost is then $37,240,000 ÷ 30 yr = $1,241,000.

Straight-line depreciation may be applied to total capital expenditures over the life of the mine for tax calculations. For simplicity, this evaluation does not consider salvage value at the end of mine life as the present value would be heavily discounted and would not influence the outcome substantially.

In Table 13.1.2.40, the initial capital investment and the total replacement capital are summed and then divided by mine life to obtain the average annual depreciation.

The next step is to establish a cash flow statement that provides the basis for calculating investment criteria such as net present value, *investment efficiency* (IE), and *return on investment* (ROI). The statement provides a common basis for evaluating project alternatives.

ROI is the interest rate earned on an investment. ROI standards are the minimum acceptable rates of return on a project. A higher rate than required by capital markets is used to provide a safety factor to cover risks inherit in any new project.

NPV is the value today of future net cash flows. The NPV of a project is calculated by discounting the cash flow at the ROI standard used for the project.

IE is the ratio of the NPV of the project to the amount of initial capital investment. The ratio provides a quick visualization of value added per initial dollar invested.

Depletion is a deduction from taxable income that approximates the value of the exhaustible mineral deposit. The percentage allowed is multiplied by the gross revenues, but the total amount cannot exceed 50% of a property's net income.

The present value factors used are for mid-period to average the flow of money over a year. Standard tables are available for this purpose. Table 13.1.2.41 compares the four selected cases.

In the cash flow statement, it is obvious that the NPV and ROI continue to grow with the increasing size of operation. Theoretically, it could be demonstrated that a duration of one year would yield the highest NPV. That is an absurd situation, but it does highlight the need to approach the evaluation from a practical operating standpoint and also to evaluate project gains incrementally.

With an extremely short duration project, an inordinate amount of capital equipment is required, most of which cannot be properly sized nor fully depreciated. The equipment becomes salvage and must be transferred or sold. Operating problems

Table 13.1.2.34. Revised Production Costs

Categories	Tonnage ratios	2000 $/ton	2000 total	3000 $/ton	3000 total	5000 $/ton	5000 total	7000 $/ton	7000 total
Stripping	26	0.45	11.70	0.39	10.14	0.29	7.54	0.28	7.28
Associated waste	2	1.74	3.48	1.74	3.48	1.74	3.48	1.74	3.48
Ore mining	1	1.74	1.74	1.74	1.74	1.74	1.74	1.74	1.74
Milling	1	7.25	7.25	7.00	7.00	6.50	6.50	6.00	6.00
Directs & indirects	1	6.50	6.50	6.30	6.30	5.90	5.90	5.50	5.50
Operating cost/ton ore milled			30.67		28.66		25.16		24.00
Cutoff grade Cost divided by value of recovered lb/ton			0.068%		0.064%		0.056%		0.053%

Mill Capacity, tons/day (column groups 2000, 3000, 5000, 7000)

Conversion table: 1 ton/day = 0.9072 t/day, 1 lb/ton = 0.500 kg/t.

Table 13.1.2.35. Revised Ore and Waste Inventory

Case, tons/day	Ore million tons & grade	Low Grade and Associated Waste million tons	Stripping million tons
2000	10.8 @ 0.114%	31.20	364
ratios	1	2.89	33.7
3000	12.4 @ 0.108%	29.60	364
ratios	1	2.39	29.4
Original example	14.0 @ 0.103%	28.00	364
ratios	1	2.00	26
5000	15.6 @ 0.098%	26.40	364
ratios	1	1.69	23.3
7000	16.8 @ 0.095%	25.20	364
ratios	1	1.50	21.7

Conversion factor: 1 ton = 0.9072 t.

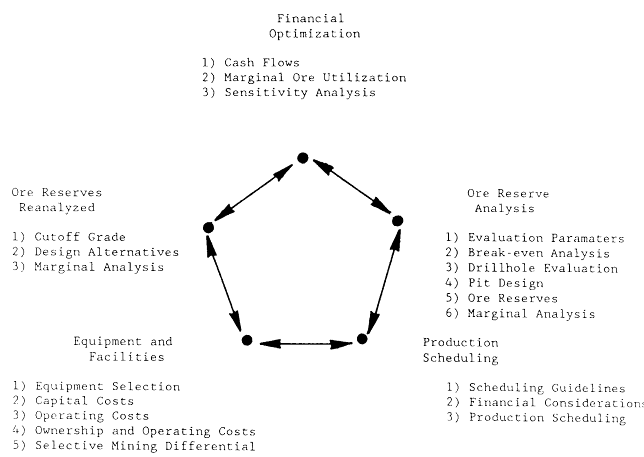

Fig. 13.1.2.10. Financial evaluation.

Table 13.1.2.36. Marginal Analysis

Mill Size tons/day	Milling $/ton	+	Load, Haul and Dump $/ton	÷	Value $/lb × 0.09	=	Required lb/ton	=	Grade required %
2000	13.75		0.75		22.50		0.64		0.032
3000	13.30		0.75		22.50		0.62		0.031
5000	12.40		0.75		22.50		0.58		0.029
7000	11.50		0.75		22.50		0.54		0.027

Conversion factor: 1 ton = 0.9072 t.

such as traffic congestion, skilled manpower availability, etc. become magnified. The optimum plant size is, in part, obtained only by increasing the size to the point where the most workable plant operation exists.

The 30-year, 2000-ton/day (1800-t/day) case fails to meet the 15% ROI standard and is eliminated from further consideration. The remaining three cases all satisfy the minimum acceptable rate and continue to increase in value as the plants become larger. However, by analyzing incremental growth a different picture emerges.

Using the data available from Table 13.1.2.41, the delta gains between the four cases are determined. Table 13.1.2.42 examines the incremental returns on investment.

Table 13.1.2.37. Annual Revenue

Mill Size tons/day	Ore 1000 tons/yr	Product 1000 lb	Revenue 1000 $
2,000	730	1,102	27,550
3,000	1,095	1,653	41,325
5,000	1,825	2,756	68,900
7,000	2,555	3,858	96,450

Conversion factor: 1 ton = 0.9072 t.

Table 13.1.2.38. Operating Costs

Costs, $/ton	Ratios	Mill Throughput, tons/day 2,000	3,000	5,000	7,000
Stripping	16.55:1	$ 7.45	$ 6.45	$ 4.80	$ 4.63
Mining Zone	1.91:1	3.32	3.32	3.32	3.32
Processing	1:1	13.75	13.30	12.40	11.50
Totals		$24.52	$23.07	$20.52	$19.45
1000 tons/yr		730	1,095	1,825	2,555
1000 $/yr		17,900	25,262	37,449	49,695

Conversion factor: 1 ton = 0.9072 t.

Table 13.1.2.39. Replacement Costs

Fleet	ICI $ million	Usage hr/yr	Life 1000 hr	Life yr	Replacement factor	Replacement cost $1000
Shovels, 2 @ 250-kW	4.0	5250	100	19.05	0.57	2,280
Trucks 17 50-ton (13)	6.8	*4301	50	11.63	1.58	10,744
Ancillary—mobile	3.1	5625	25	4.44	5.76	17,856
—stationary	1.5	6000	96	16.00	0.88	1,320
Mine (2 shifts)	4.0	*3769	50	13.27	1.26	5,040
Mill	34.0	N/A	N/A	30.00	0	0
Total	53.4					37,240

Table 13.1.2.40. Replacement and Depreciation Costs

Case tons/day	Life yr	$1,000 ICI	Total Replacement	Annual Replacement	Total Capital	Annual Depreciation
2,000	30	53,400	37,240	1241	90,640	3,021
3,000	20	61,600	24,520	1226	86,120	4,306
5,000	12	80,000	11,440	953	91,490	7,620
7,000	8.6	103,000	6,768	787	109,768	12,764

Conversion factor: 1 ton = 0.9072 t.

It is apparent that the incremental gains do not warrant the expenditures necessary to increase the size of the project to 7000 tons (6400 t)/day. After 5000 tons (4500 t)/day, the IE drops substantially and the capital investment requirement, per percent gain, blossoms almost fourfold.

Based on the preceding analysis, the optimum plant size is in the range of 5000 tons (4500 t)/day. Adjustments may be possible by bracketing the case with 4000 and 6000 tons/day (3600 and 5400 t/day) options and repeating the analysis using data generated in the first iteration.

MARGINAL ORE UTILIZATION. As earlier mentioned, postponing the milling of the marginal ore to the last few years of the property enhances the NPV. The ore reserves and values for the 5000-ton (4500-t)/day case are listed in Table 13.1.2.43.

To illustrate the difference in values, cumulative NPV factors are applied to each scenario.

Option 1. Marginal ore milled after ore milled:

7.6 yr @ $43,034,000/yr × 4.672	= $201,054,840
4.4 yr @ $4,435,000/yr × 1.141	= $ 5,060,335
Total NPV	= $206,115,175

Option 2. Uniform milling of combined reserves:

12 yr @ $31,536,000/yr × 5.813	= $183,319,760

Postponing the milling of marginal ore to the last years of mine life results in a net NPV increase of $22,795,415 or 12%.

From a practical standpoint, unless the higher-grade ore is located in distinct areas of the pit and can be accessed in a normal operational flow, the preceding illustration is not possible. However, the engineer should always keep his eyes open to the potential of opening the pit in a higher-than-average grade area and sequencing the advances such that higher-grade material is exposed first. If that opportunity exists, an improved cash flow often is possible.

The time value of money is critical when planning property development. Deferment of the ICI is possible by the following steps:

1. Contract out the initial stripping until a positive cash flow is experienced.
2. Strip initially in areas with lower overburden to ore ratios to reduce equipment requirements.
3. Minimize the preproduction construction and mine development period to the greatest extent possible.

A final consideration is what to do with the low grade between 0.030 and 0.045%. This material will not contribute to the economics of the project, but it should not be treated as waste. Low-grade stockpile areas should be provided so that an

Table 13.1.2.41. Cash Flow Statement

	Mill Size, tons/day			
	2,000	3,000	5,000	7,000
Cash Flow				
Revenue	27,550	41,325	68,900	96,450
Operating Costs	17,900	25,262	37,449	49,695
Operating Cash Flow	9,650	16,063	31,451	46,755
Annual Replacement Capital	1,241	1,226	953	787
Before Tax Cash Flow	8,409	14,837	30,498	45,968
Tax Calculation				
Operating Cash Flow	9,650	16,063	31,451	46,755
Capital Depreciation	3,021	4,306	7,620	12,764
Taxable Income before Depletion	6,629	11,757	23,831	33,991
Depletion at 22% (50% limit)	3,315	5,878	11,915	16,995
Taxable Income	3,315	5,879	11,916	16,996
Tax at 39%	1,293	2,293	4,647	6,628
Evaluations				
Before Tax Cash Flow	8,409	14,837	30,498	45,968
Tax	1,293	2,293	4,647	6,628
Cash Flow After Tax	7,116	12,544	25,851	39,340
Cumulative PW Factor at 15% (Mid-Period)	7.041	6.712	5.813	4.995
Discounted Cash Flow After Tax	50,104	84,195	150,272	196,503
Initial Capital Investment	53,400	61,600	80,000	103,000
Net Present Value at 15%	3,296	22,595	70,272	93,503
Net Present Value at X%	13,- 3,207	25,- 6,218	40,- 4,935	45,- 2,448
ROI (Interpolation)	14.0	22.8	38.4	44.2
Investment Efficiency	0.06	0.37	0.88	0.91

Conversion factor: 1 ton = 0.9072 t.

Table 13.1.2.42. Incremental Gains Analysis

	Mill Size, tons/day			
	2,000	3,000	5,000	7,000
Initial Capital Investment, $1,000	0	8,200	18,400	23,000
Net Present Value, $1,000	0	25,877	47,706	23,347
Return on Investment, %	0	8.8	15.6	5.8
Investment Efficiency	0	3.16	2.59	1.02
$1,000/% ROI Increase	0	932	1,179	3,966

Conversion factor: 1 ton = 0.9072 t.

option exists to mill the material after the mine has closed or if market conditions improve substantially.

The first evaluation has now been completed. The degree of accuracy presented herein is adequate for many engineered feasibility studies. Further optimization is probable, first by re-analyzing the ore reserves using the recalculated ore cutoff grade. Reconfiguration of the pit boundaries may result. Secondly, the 5000-ton (4500-t)/day target can be bracketed and ancillary costing performed in addition to the loading and hauling costs. The analysis should then be repeated in the same manner until one is satisfied that more significant improvement is not likely.

SENSITIVITY ANALYSIS. The analysis is sensitive to the uncertainties inherent in any new project. The most critical parameters in terms of effect on the evaluation are

1. Ore reserves: Has the mineral been properly sampled?
2. Mill recoveries: Are the amenabilities studies based on representative samples?
3. Product selling price: What is the market volatility in this commodity? Are long-term sales contracts obtainable?

Each can have a profound effect on the ultimate outcome of the property as they all have a direct influence on the total value of the deposit. Capital and operating cost variations also can have an effect, but because these are generally more predictable and less volatile, their variations have less impact on the overall cash flow.

As the major sensitivities directly influence the anticipated revenue, it is a relatively simple exercise to generate a spectrum of annual revenues based on variations in ore reserves, mill recov-

Table 13.1.2.43. Ore Reserve Values

Case	Reserves	1,000 Tons and grade	Value, $/ton	Cost, $/ton	Net Value $/ton	Annual Value $1,000	Reserve, yr
1	Ore	15,600 @ 0.098	44.10	20.52	23.58	43,034	7.6
	Marginal Ore	6,400 @ 0.051	22.95	20.52	2.43	4,435	4.4
2	Combined Reserves	22,000 @ 0.084	37.80	20.52	17.28	31,536	12.0

Conversion factor: 1 ton = 0.9072 t.

eries and selling price. Recalculation of the cash flow statement for the 5000-ton (4500-t)/day option will indicate how much revenue loss can be experienced before the project drops below the 15% minimum ROI.

In any event, a decision must be reached on whether or not the potential project will become a reality. Decisions must be based on sound engineering and financial judgments to decide on the basic merits of the investment. It is the engineer's responsibility to provide as detailed a basis as possible upon which a particular project can be evaluated.

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Chapter 13.2
STRIP MINE PLANNING AND DESIGN

MATTHEW J. HREBAR AND THOMAS ATKINSON

13.2.1 PLANNING PROCEDURE AND RESERVE ESTIMATION

MATTHEW J. HREBAR

13.2.1.1 Planning Procedure

Planning a surface mine using the strip (open cast) mining method requires consideration of many variables with complex interrelationships (Table 13.2.1.1). Following a collection of general deposit and project-related information, a development and extraction plan is conceived. Project economics is determined, and an economic analysis is performed to determine project viability. Note that steps III and IV are iterated until an "optimum" solution is found. These iterations may include consideration of various mining method/equipment combinations, mine size/equipment combination, mining method/pit layout combinations, etc.

As seen in reviewing the list of variables and information, the problem areas relate to geology, engineering, environmental sciences, and economics. As a result, the planning and development process is interdisciplinary in nature and requires inputs and guidance from numerous individuals with diverse backgrounds and training. The correct mine plan will result in a feasible operating plan, as well as one that optimizes the economic return subject to the numerous contractual, environmental, legal, and other constraints related to the specific property (see Section 7 for general details of mine development).

Jones (1977) has outlined the 10 major steps involved in planning and developing a surface coal mine. These steps are detailed in Table 13.2.1.2 and shown in a Gantt chart in Fig. 13.2.1.1. The planning and development process can take up to 10 years and require millions of dollars of expenditure exclusive of that for actual mine preparation and equipment purchase.

13.2.1.2 Reserve Estimation and Stripping Ratio Calculations

A major portion of the planning process centers on the reserve estimate and geologic "model" of the deposit. These estimates can be quite preliminary or can be made in great detail. The level of effort should correspond to the stage in the exploration/development program. Prospecting and exploration are covered in Section 4 and project geology in Section 5.

SAMPLING METHODS. In most cases, a combination of rotary and diamond core drilling methods is used to obtain sample data from the deposit. Rotary drilling is used to determine overburden depth and coal thickness. A rotary rig is used to drill through the coal seam(s). Then a geophysical probe is lowered into the hole and a log of the hole is produced. Combinations of parameters such as gamma logs and density logs are used to pick the coal/overburden interface. Quality data (e.g., Btu/lb, % ash, % S, etc.) are obtained by diamond core drilling. The solid cores of the coal seam are processed in a lab with the analysis returned for appropriate posting. Core data of the overburden are taken occasionally to determine if toxic overburden exists on the property.

Sample spacing will vary depending on the variability of thickness and quality parameters of the seam. An initial rotary grid spacing (e.g., 2000 ft or 600 m) might be followed by a closer-spaced grid (e.g., 1000 ft or 300 m) or a hole at the center of the original grid. This follow-up drilling is usually conducted to provide further resolution in a second phase of drilling that would be contingent upon success in the more widely spaced initial program. The coal core drilling usually is performed on a lesser spacing than rotary drilling. Closely spaced fences of holes normally are drilled perpendicular to the cropline to determine the extent of the burned or oxidized zone. This contact usually corresponds to some depth of overburden. However, in areas where the overburden is badly fractured, the burned or oxidized zone may not correlate with depth of overburden, and more extensive drilling is required. Overburden coring is usually performed with a hole density of approximately four to five holes/section.

CALCULATION METHODS. The method of calculation used should be a function of the seam variability and to some extent the mining history of the seam in the particular locale.

Tons/Acre Method—The tons-per-acre approach is commonly employed where there is little seam variability or where a rapid approximation of reserves, overburden, and stripping ratio is required. As seen in Table 13.2.1.3, the method involves calculating the tons of coal/acre-ft (tonnes/hectare-meter) and then multiplying this figure by the average thickness of the coal to determine the tons/acre (tonnes/hectare) of coal. The area of reserves in acres is then multiplied by the tons/acre (tonnes/hectare) of coal to arrive at the reserve. Two reserve figures are generally presented: (1) the in-place reserve and (2) the recoverable reserve, which includes allowances for mining and for preparation plant recoveries.

Similarly, overburden/acre is calculated using the average overburden height, and the reserve area is multiplied by this figure to arrive at total overburden.

The average stripping ratio (i.e., bank cubic yards of overburden/ton of coal, or cubic meters/tonne) is calculated by dividing total volume of overburden by total weight of coal, usually on a recoverable basis. In these approximate calculations, average coal and overburden thickness dimensions are usually based on simple means of the drillhole data. The area is usually found by planimetering the area bounded by the outcrop, subcrop, and/or property lines. If more precision is required, techniques such as the polygon or triangular approach can be used to more precisely assign areas of influence to the drillholes. However, the iso-line approach is most commonly used when a detailed reserve and overburden analysis is required.

Iso-Line Method—The iso-line method involves construction of lines of equal values for overburden thickness and coal thickness in an effort to present a model of these parameters representing the deposit.

The iso-overburden map is generated from the surface topography map and a topographic map of the top of the coal seam. The latter is constructed by using drillhole data (depth to top of

Table 13.2.1.1. Salient Factors Requiring Consideration In Mine Planning And Feasibility Studies

I. INFORMATION ON DEPOSIT
- A. Geology: Overburden
  1. Stratigraphy
  2. Geologic structure
  3. Physical properties (highwall and spoil characteristics, degree of consolidation)
  4. Thickness and variability
  5. Overall depth
  6. Topsoil parameters
- B. Geology: Coal
  1. Quality (rank and analysis)
  2. Thickness and variability
  3. Variability of chemical characteristics
  4. Structure (particularly at contacts)
  5. Physical characteristics
- C. Hydrology (Overburden and Coal)
  1. Permeability
  2. Porosity
  3. Transmissivity
  4. Extent of aquifer(s)
- D. Geometry
  1. Size
  2. Shape
  3. Attitude
  4. Continuity
- E. Geography
  1. Location
  2. Topography
  3. Altitude
  4. Climate
  5. Surface conditions (vegetation, stream diversion)
  6. Drainage patterns
  7. Political boundaries
- F. Exploration
  1. Historical (area, property)
  2. Current program
  3. Sampling (types, procedures)

II. GENERAL PROJECT INFORMATION
- A. Market
  1. Customers
  2. Product specifications (tonnage, quality)
  3. Locations
  4. Contract agreements
  5. Spot sale considerations
  6. Preparation requirements
- B. Transportation
  1. Property access
  2. Coal transportation (methods, distance, cost)
- C. Utilities
  1. Availability
  2. Location
  3. Right-of-way
  4. Costs
- D. Land and Mineral Rights
  1. Ownership (surface, mineral, acquisition)
  2. Acreage requirements (onsite, offsite)
  3. Location of oil and gas wells, cemeteries, etc.
- E. Water
  1. Potable and preparation
  2. Sources
  3. Quantity
  4. Quality
  5. Costs
- F. Labor
  1. Availability and type (skilled, unskilled)
  2. Rates and trends
  3. Degree of organization
  4. Labor history
- G. Governmental Considerations
  1. Taxation (local, state, federal)
  2. Royalties
  3. Reclamation and operating requirements
  4. Zoning
  5. Proposed and pending mining legislation

III. DEVELOPMENT AND EXTRACTION
- A. Compilation of Geologic and Geographic Data
  1. Surface and coal contours
  2. Isopach development (thickness of coal and overburden, stripping ratio, quality, costs)
- B. Mine Size Determination
  1. Market constraints
  2. Optimum economics
- C. Reserves
  1. Method(s) of determination
  2. Economic stripping ratio
  3. Mining and barrier losses
  4. Burned, oxidized areas
- D. Mining Method Selection
  1. Topography
  2. Refer to previous geologic/geographic factors
  3. Production requirements
  4. Environmental considerations
- E. Pit Layout
  1. Extent of available area
  2. Pit dimensions and geometry
  3. Pit orientation
  4. Haulage, power, and drainage systems
- F. Equipment Selection
  1. Sizing, production estimates
  2. Capital and operating cost estimates
  3. Repeat for each unit operation
- G. Project Cost Estimation (Capital and Operating)
  1. Mine
  2. Mine support equipment
  3. Office, shop, and other facilities
  4. Auxiliary facilities
  5. Manpower requirements
- H. Development Schedule
  1. Additional exploration
  2. Engineering and feasibility study
  3. Permitting
  4. Environmental approval
  5. Equipment purchase and delivery
  6. Site preparation and construction
  7. Start-up
  8. Production

IV. ECONOMIC ANALYSIS
- A. Sections III and IV repeated for various alternatives

coal) and the surface drillhole collar elevations. The two maps are then overlaid, and by subtraction of the surface and top of coal elevations, the thickness of overburden can be established either at the topography intersections or on a regular grid basis. These values are then contoured to produce the iso-overburden thickness lines (Fig. 13.2.1.2).

The iso-coal thickness map is generated either by using the top and bottom coal contours or by interpolation of actual drillhole thickness intercepts. When coal structure contours are used, the top and bottom coal elevation maps are overlaid and the values calculated at either the line intersections or at regular grids. These thicknesses are contoured to produce the iso-coal

Table 13.2.1.2.
Major Steps in Surface Mine Development

I. Assembly of the Minable Coal Package
   1. Lease acquisition
   2. Mapping the area
   3. Drilling program
   4. Surface drilling rights acquisition
   5. Drilling, sampling, logging, analysis
   6. Mineral evaluation (determination on commercial quantities present)
   7. Drilling on closer centers (development drilling)
   8. Sampling, logging analysis
   9. Surface acquisition
II. Market Development
   1. Market survey
   2. Potential customer identification
   3. Letter of intent to develop and supply
   4. Contract negotiation
III. Environmental and Related Studies
   1. Initial reconnaissance
   2. Scope of work development
   3. Consultant selection
   4. Implementation
   5. Environmental impact report
   6. Environmental monitoring
IV. Preliminary Design, Machine Ordering
   1. Conceptual mining development
   2. Economic size determination
   3. Mining system design, layout and development
   4. Equipment selection
   5. Stripping machine ordering
   6. Mine plan development
V. NEPA Process (National Environmental Policy Act of 1969)
   1. Identification of lead agency for Environmental Impact Statement (EIS)
   2. Draft EIS
   3. EIS review and comments
   4. EIS hearing and record
   5. Federal EIA review
   6. Council on Environmental Quality filing
   7. Mining and/or reclamation plan approval
VI. Permits
   1. State water well rights appropriation permits
   2. State special use permit, such as a reservoir
   3. State mining permit
   4. State industrial siting permit
   5. Federal NPDES permit
   6. US Forest Service special land use permit
VII. Design and Construction
   1. Preliminary design and estimation
   2. Material ordering and contracting
   3. Water well development
   4. Access road and site preparation
   5. Railroad construction
   6. Power supply installation
   7. Facilities and coal handling construction
   8. Warehouse building and yards
   9. Coal preparation and loading facilities construction
   10. Overland conveyor construction
VIII. Mining Preparation
   1. Stripping machine(s) erection
   2. Loader erection
   3. Support equipment readying
   4. Manpower recruitment and training
IX. Production Buildup
X. Full Production

Source: Jones, 1977.

thickness lines. If coal intercepts are used directly, the values are interpolated to produce the thickness iso-lines. The most common approach is utilization of the coal structure topography, because these maps are utilized in pit haulage and drainage planning.

The coal thickness and overburden thickness iso-lines are then utilized to calculate the amount of overburden and coal, using procedures outlined in Table 13.2.1.4. The data are usually summarized by overburden range, as indicated in step 8 of the table.

Iso-stripping ratio lines can also be constructed by overlaying the overburden and coal thickness map, calculating the stripping ratio at the line intersection or on a regular grid, and then interpolating and contouring the stripping ratios. The iso-stripping ratio map is extremely useful in determining the economic limits, as discussed under stripping ratio.

Computer Models—Many operators utilize computer software to perform the previously described reserve calculations and to generate the associated graphics utilizing flat-bed plotters. A typical software system will have the following capabilities:

1. Drillhole data loading and editing with data including drillhole identification, location, lithology on a from-to basis, rock type, seam codes, coal seam analysis, etc.
2. Drillhole data listing and abstracts.
3. Drillhole data quality summaries by seam, list, and area with averages for specified parameters.
4. Polygonal reserves with reserves by hole and seam with maps generated showing polygonal shapes.
5. Topography loading by drillhole or by digitizing contour maps.
6. Vertical cross sections, to scale, showing overburden, coal, and other input data.
7. Plan maps showing drillhole locations, crop lines, property lines, etc.
8. Interpolation of data to produce a gridded model of overburden thickness, coal thickness, top and bottom coal elevations, etc.
9. Seam reserves based on gridded data for any area or series of areas with ability to limit reserves based on numerous parameters or combinations of parameters (e.g., minimum coal thickness, maximum overburden thickness, maximum percent sulfur, etc.)
10. Mining cut data generated on a cut-by-cut basis to allow simulation of alternative mine plans and comparison of coal and overburden amounts, coal quality, etc.

Such software is available from numerous companies. A fairly complete directory of programs available has been compiled by Gibbs (1987). Use of computer-assisted reserve calculation and mine planning must be considered state-of-the-art in the surface coal mining industry.

LIMITATIONS. In arriving at a minable reserve for a specific property, consideration must be given to various limitations and losses that will be encountered. These can be grouped into economic limits, equipment limits, and mining and barrier losses. All of these factors will result in coal left in place or lost in the process of extraction. The minable reserve is usually significantly less than the in-place or geologic reserve. For a discussion of economic considerations, see Section 6.

Economic Stripping Ratio—The economics of stripping increasing amounts of overburden is a factor controllable by mine management. The basic premise involves setting a minimum acceptable profit/ton of coal recovered and then adjusting the amount of stripping to satisfy that profit constraint. Such an approach results in a maximum allowable stripping ratio and places a boundary on the area that can be mined economically.

The concept is illustrated in Table 13.2.1.5. As seen in the example, the portion of the property with a limit greater than 32 yd^3/ton (27 m^3/t) would not be considered a surface minable

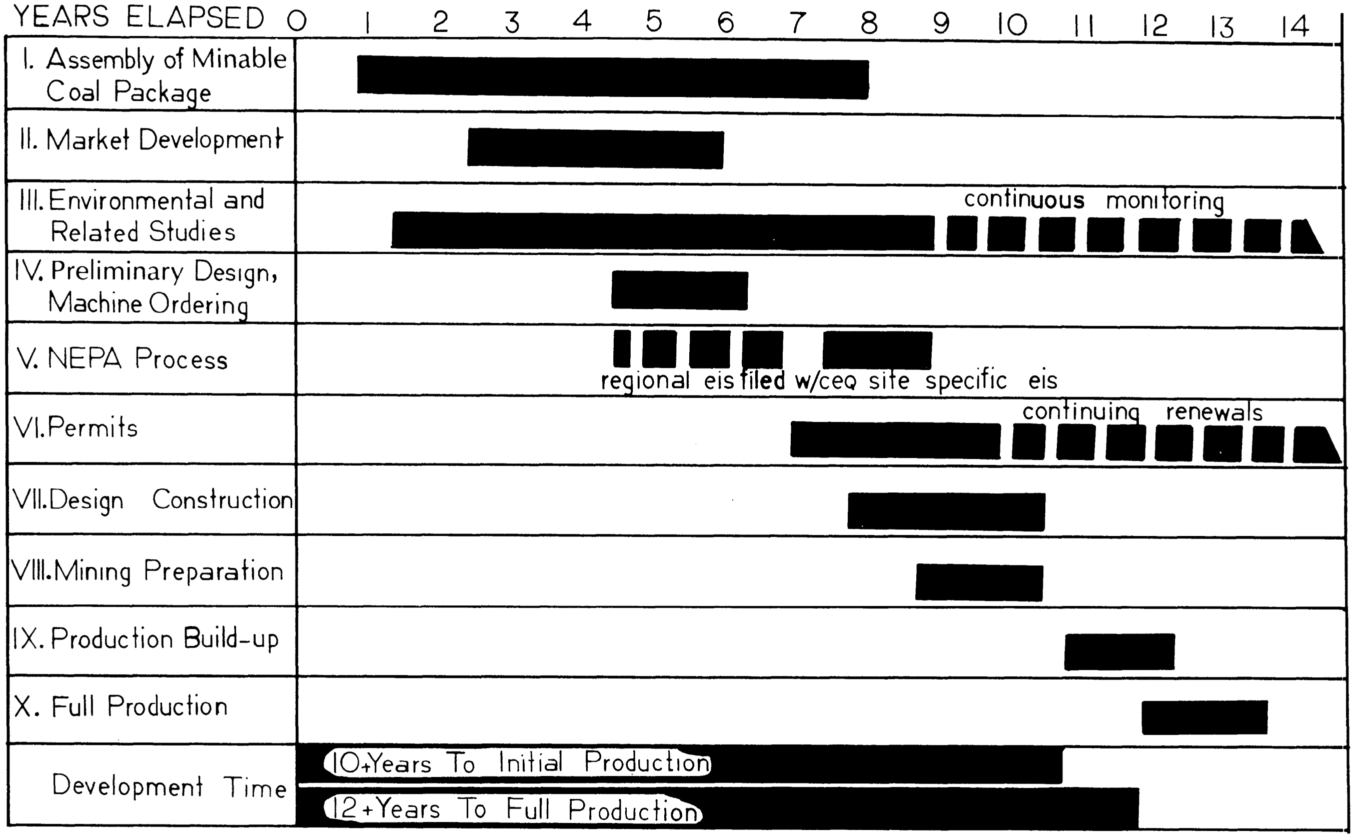

Fig. 13.2.1.1. Illustrative surface mine development schedule (Federal Coal-West) (after Jones, 1977).

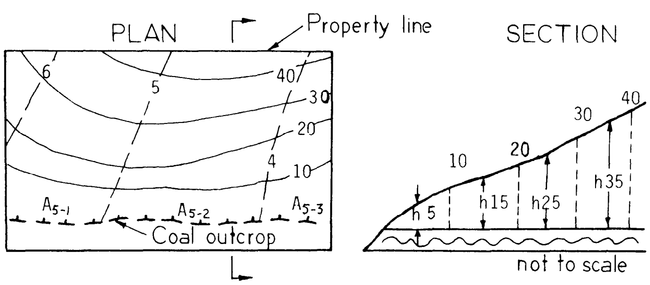

Fig. 13.2.1.2. Plan and section views for a reserve calculation using the iso-line method.

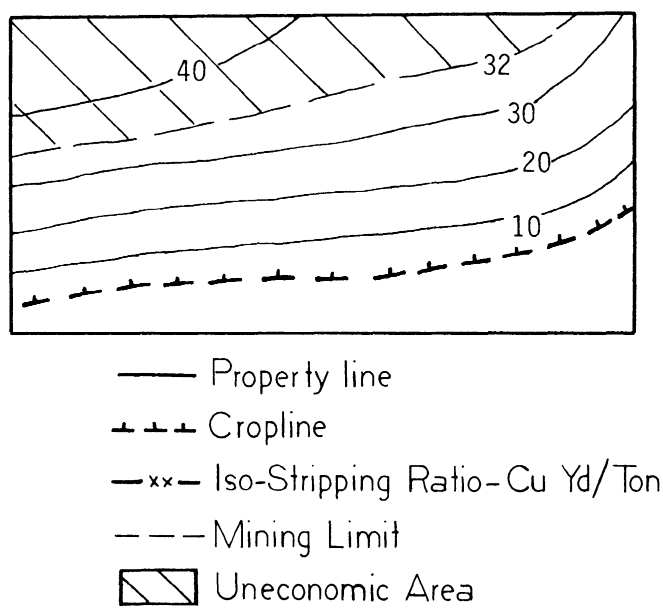

Fig. 13.2.1.3. Plan view used in economic or ultimate stripping ratio calculation.

reserve, and thus the reserve would exclude that area (Fig. 13.2.1.3).

It should be noted that costs of production exclusive of stripping and stripping costs should be "marginal" costs and not average costs. The marginal costs should represent the best estimate of actual costs at that specific area on the property. For example, consideration must be given to any additional costs of stripping caused by greater depths of overburden or any additional haulage costs resulting from increased haulage distances.

Equipment Limitations—In some cases, stripping equipment digging-depth limitations or coal-loading equipment height limitations may limit the minable area. In certain deposits, highwall or spoil instability may preclude use of deep-digging techniques and limit the minable area with a maximum digging depth. In this situation, extended bench or spoil-side pullback may not be possible, causing a depth limit on the reserve (deep-digging methods are described in a subsequent chapter).

A minimal coal thickness is required to economically recover thin seams using surface techniques. In very thin seams, dilution from top and bottom material will preclude recovery of a marketable product. As a result, depending on the coal loading tech-

Table 13.2.1.3. Reserve Calculations: Tons/Acre Method

1. $$\frac{tons}{ac\text{-}ft} = 43{,}560\,\frac{ft^2}{ac} \times \frac{coal\ density}{2000 lb}$$

e.g., density = 82 lb/ft³

$$\frac{tons}{ac\text{-}ft} = 43{,}560 \times \frac{82}{2000} = 1786\,\frac{tons}{ac\text{-}ft}$$

2. $$\frac{tons}{ac}\ (raw) = \frac{tons}{ac\text{-}ft} \times average\ thickness\ (ft)$$

e.g., average thickness = 4.5 ft

$$\frac{tons}{ac}\ (raw) = 1786 \times 4.5 = 8037\,\frac{tons}{ac}\ (raw)$$

3. $$\frac{tons}{ac}\ (recoverable) = \frac{tons}{ac}\ (raw) \times mining\ recovery \times preparation\ plant\ recovery$$

e.g., mining recovery = 90% prep. plant recovery = 85%

$$\frac{tons}{ac}\ (recoverable) = 8037 \times .90 \times .85 = 6148\,\frac{tons}{ac}$$

4. $$tons\ (raw) = \frac{tons}{ac}\ (raw) \times ac$$

e.g., 640 ac

tons (raw) = 8037 × 640 = 5,143,700 tons

5. $$tons\ (recoverable) = \frac{tons}{ac}\ (recoverable) \times ac$$

e.g., 640 ac

tons (recoverable) = 6148 × 640 = 3,934,700 tons

Stripping Ratio

6. $$\frac{overburden}{ac} = 43{,}560\,\frac{ft^2}{ac} \times \frac{avg.\ overburden}{27\ ft^3/yd^3}\ (ft)$$

e.g., overburden thickness = 60 ft

$$\frac{overburden}{ac} = 43{,}560 \times \frac{60}{27} = 96{,}800\,\frac{yd^3}{ac}$$

7. stripping ratio or overburden ratio

$$stripping\ ratio = \frac{overburden/ac}{tons/ac\ (recoverable)}$$

e.g., tons/ac = 6148

$$stripping\ ratio = 96{,}800/6148 = 15.7\,\frac{yd^3}{ton}$$

Conversion factors: 1 ft = 0.3048 m, 1 ton = 0.9072 t, 1 acre = 0.405 hectare, 1 yd³/ton = 0.8428 m³/t, 1 ton/acre-ft = 7.37 t/hectare-m, 1 lb/ft³ = 16.0185 kg/m³.

Table 13.2.1.4. Reserve Calculation: Iso-Line Method (see Fig. 13.2.1.2)

Step 1. Construct iso-overburden lines.

2. Measure areas bounded by iso-overburden lines with planimeter. (e.g., A_5 (outcrop-10), A_{15}(10-20), etc.)

3. Assign thickness to overburden, calculate volume for each "slice":

Area (A)	Height (h)	Yardage (A × h/27)
A_5	h_5	CY_5
.	.	.
.	.	.
A_n	h_n	CY_n
		Sum CY

4. Construct iso-coal thickness lines.

5. Measure sub-areas bounded by iso-overburden and iso-coal thickness lines for each A_n or "slice".

6. Assign coal thickness to each sub-area and calculate coal quantity. Summarize, calculate stripping ratios, etc.

Sub-area	Coal Thickness	Tonnage $A_{5-i} \times T_{5-i} \times$ Density/2000	Stripping Ratio CY_n/T_n
A_{5-1}	T_{5-1}	Ton_{5-1}	
.	.	.	
.	.	.	
A_{5-m}	T_{5-m}	Ton_{5-m}	
		T_5	SR_5

7. Repeat 5 and 6 as required.

8. Data usually summarized as follows:

Overburden Range	Overburden Quality	Coal Quality	Stripping Ratio
0–10			
10–20			
20–30			
30–40			
40+			

Conversion factor: 1 yd³ = 0.7646 m³.

Table 13.2.1.5. Economic or Ultimate Stripping Ratio Calculations (See Fig. 13.2.1.3)

$$\text{Economic Stripping Ratio} = \frac{\text{Recoverable Value/Ton} - \text{Production Cost/Ton*} - \text{Minimum Profit/Ton}}{\text{Stripping Cost/Yd}^3}$$

*Exclusive of Stripping

Example:

	Recoverable Value	=	\$17.00/Ton
	Minimum Profit	=	\$ 3.00/Ton
	Stripping Cost	=	\$ 0.25/Yd³
	Production Cost*	=	\$ 6.00/Ton

$$\text{Economic Stripping Ratio} = \frac{17.00 - 6.00 - 3.00}{0.25} = 32\,\frac{\text{Yd}^3}{\text{Ton}}$$

Conversion factors: 1 ton = 0.9072 t, 1 yd³ = 0.7646 m³, 1 yd³/ton = 0.8428 m³/t.

nique employed, areas with coal thicknesses below that minimum would be considered unminable.

Mining and Barrier Losses—Mining losses are those associated with the extraction process. These losses include the following:

1. Top of coal: losses at upper coal contact with overburden as a result of cleaning coal after stripping.
2. Bottom of coal: losses at lower coal contact with bottom as a result of loader losses; both top and bottom losses, as a percentage, are a function of seam thickness.
3. Rib: losses at side of seam adjacent to spoil as a result of spoil piled on coal rib during stripping or slides of spoil on coal; function of pit width.
4. Other: includes fly rock (blasting) and transportation losses (dust and spillage).

In eastern area operations, these losses account for some 10% of the reserve (Anon., 1977). In thicker-seam western operations, these losses are probably a lesser percentage because of the thicker seams and wider pits generally encountered.

Barrier losses are caused by coal left in place for various practical and regulatory reasons. Barrier losses include outcrop (coal left as a low-wall barrier or for quality reason), right-of-way, stream, underground mine, oil and gas wells, property line, building, and cemetery. In eastern area operations, these losses account for 5.2% of the reserve (Anon., 1977). These losses are quite site-specific and depend on the geologic and geographic setting of a property.

Mining and barrier losses are significant and are given thorough consideration in estimating minable reserves.

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13.2.2 STRIPPING AND PIT DEVELOPMENT

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The surface coal mining method selected for a specific property is generally keyed to the overburden removal system, since this unit operation usually involves the greatest capital and operating cost components. Once the stripping method and complementary equipment have been selected, compatible equipment systems are selected for the other unit operations (i.e., coal fragmentation, coal loading and hauling, etc.).

The following segment describes development for the major systems commonly encountered in surface mining of coal, especially at large western operations. In addition, major performance standards and Office of Surface Mining (OSM) regulations are integrated into the presentation of unit operations.

As previously mentioned, standard engineering practice dictates that various alternatives be investigated, not only for the stripping system, but also for the other unit operations. The goal is to provide maximum resource extraction at a minimum cost including both capital and operating cost considerations.

13.2.2.1 Alternative Stripping Methods

Three alternative stripping procedures are described.

AREA/DRAGLINE METHOD. The area/dragline method (Fig. 13.2.2.1) involves opening an initial box cut, removing the coal exposed in the box cut, and then placing the overburden from the next longitudinal cut into the mined-out, box-cut area. The procedure is then repeated on a cut-by-cut basis. The method is also referred to as “deep plowing.”

The method is generally employed in flat to moderately dipping coal seams with relatively constant overburden depths. Such a method can also be employed in areas where coal dip or overburden slope result in reaching the economic limit in relatively few cuts. In this mode, the method is referred to as a modified area or box-cut contour operation. The major difference is the rate at which the limit is reached.

The advantages and disadvantages of the area/dragline method combine the characteristics of the dragline excavator and the method. Generally, the alternative piece of equipment used to perform stripping is the stripping shovel. The industry has purchased few large stripping shovels in recent years, and there are only approximately 20 stripping shovels (i.e., direct casting machines mounted on eight crawlers) currently in operation. Draglines have been chosen in these applications primarily because of their flexibility in varying operating conditions. The dragline can handle varying overburden depths, varying overburden characteristics, and multiple seams by changing the operating mode. Although these changes often cause some loss in machine productivity, they allow mining through the specific set of conditions without major additions of auxiliary equipment that would be required with a stripping shovel. For a more detailed discussion of stripper selection, see Atkinson (1971) and Steidle (1977).

MODIFIED OPEN PIT/SHOVEL-TRUCK METHOD. Modified open pit or terrace mining is generally used in thick-seam properties with low stripping ratios. In these applications, seams are generally flat-lying, gently dipping, or rolling. The method shown in Fig. 13.2.2.2 involves opening an initial pit and placing the overburden in temporary off-site storage. Coal is then removed from the initial pit area. The next cut is taken in the direction of mine advance and the overburden is hauled around the existing pit and dumped in the mined-out area. Coal is removed and the haul-back process is repeated as the relatively small pit advances.

In new pitching or steeply dipping seam operations, the initial pit is opened and coal removed. The initial box cut is opened to the economic limit down dip. Subsequent cuts are advanced on-strike, and overburden hauled back into the mined-out initial pit.

BLOCK AREA/DOZER-SCRAPER METHOD. The block area method, which utilized construction-type equipment, was first conceived in the mid- 1970s as an alternative to the area/dragline method. At that time, draglines were not readily available because of the high demand and long lead times in procurement. Alternative methods were sought that would employ readily available construction-type equipment (i.e., dozers and scrapers). This method takes advantage of the dozer’s ability to move material over short distances at low costs and the scraper’s ability to elevate material over steep grades for short distances at reasonable costs.

In this method, it must be noted that there is a relatively narrow window of application from a material standpoint. As the degree of consolidation of material increases, wear and tear on the scrapers makes the method cost prohibitive because of high repair, maintenance and supply costs, and reduced machine

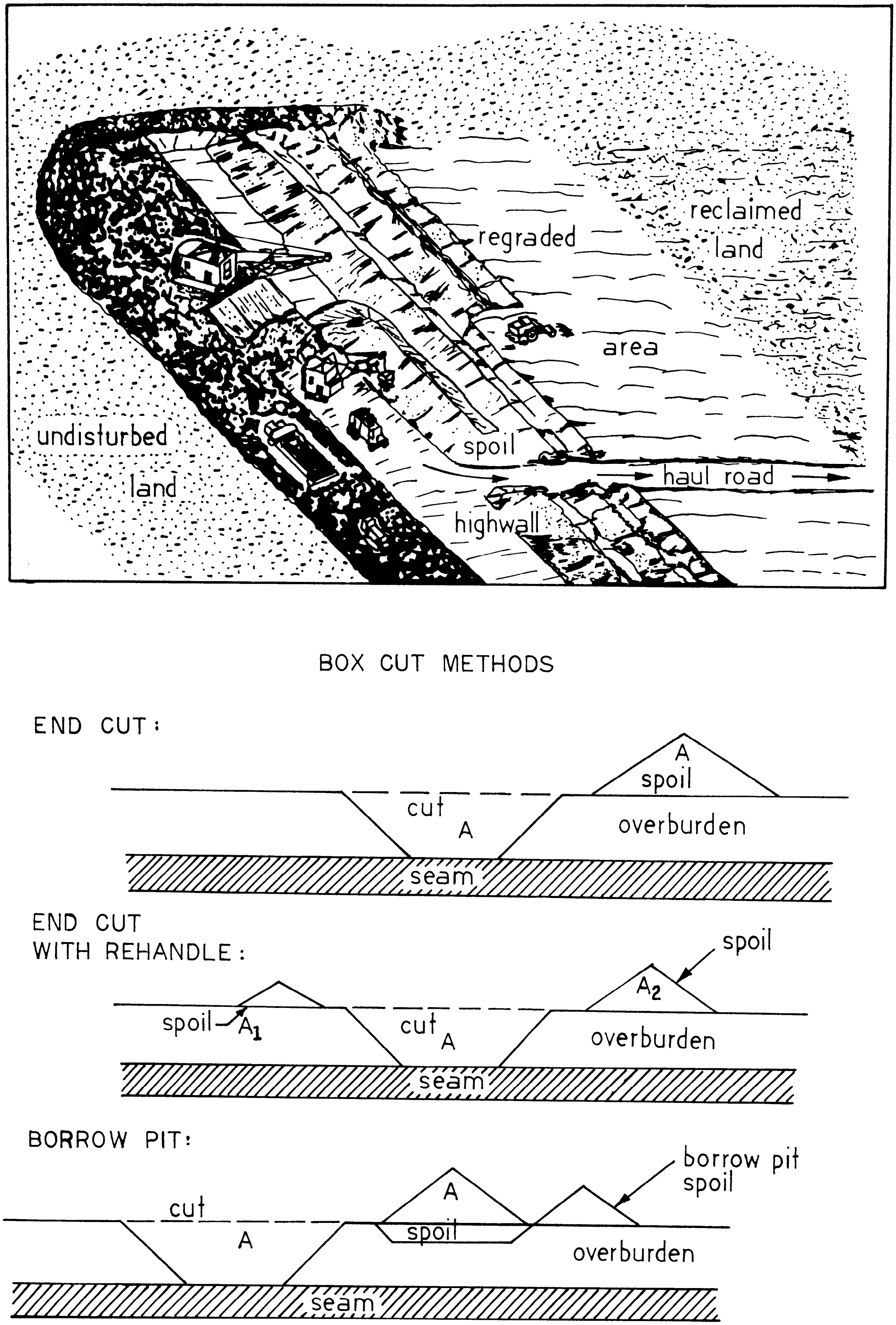

Fig. 13.2.2.1. Area/dragline stripping (Anon., 1975; by permission from Skelly and Loy, Harrisburg).

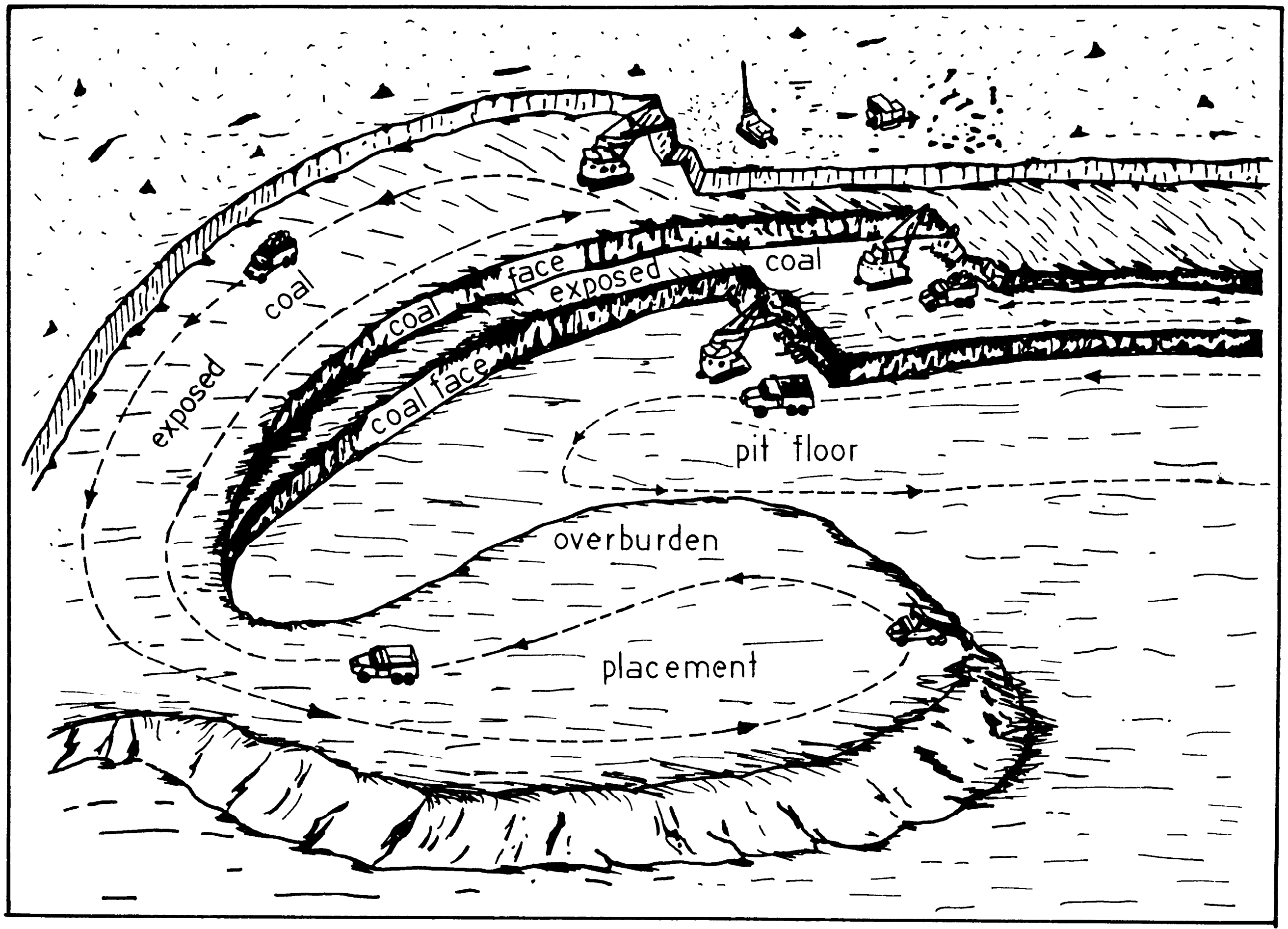

Fig. 13.2.2.2. Modified open pit: flat-lying seams (Anon., 1975; by permission from Skelly and Loy, Harrisburg).

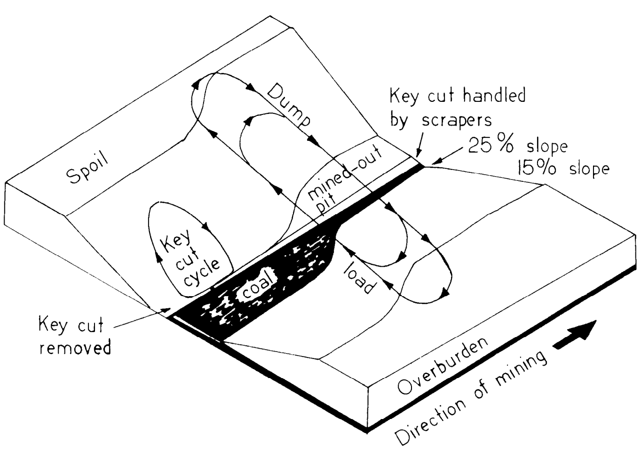

Fig. 13.2.2.3. Block area/dozer-scraper method (after Ryan, 1982).

productivity. One version of this method is shown in Fig. 13.2.2.3.

13.2.2.2 Dragline Pit Design

Pit parameters and configuration selected for a given site are a function of deposit geometry and prevailing regulations. The following points must be given consideration.

PIT LENGTH. The average length of pit in the central United States is approximately 5500 ft (1.67 km), and although specific data are not available for the western part of the country, lengths are similar. The following factors influence pit length.

1. Property dimensions: Can limit maximum length.

2. Dragline efficiency: Longer pits (Over 3000 ft or 900 m) preferred.

3. Coal inventory: In some cases, a pit inventory is kept to meet seasonal demand or compensate for low stripper availability; amount is a function of seam thickness, pit width, and length of block in place.

4. Spoil instability: If spoil stability is a function of time, shorter pits are preferred to limit spoil failures.

5. Pit flooding: In high-rainfall areas, short pits limit water quantities in pit.

6. Power system: In low mine-voltage systems, voltage drop becomes excessive with long pits.

PIT WIDTH. Pit width in large midwestern mines ranges from 95 to 105 ft (29 to 32 m) and averages 120 ft (37 m) in western mines. In deep cover, widths range to 180 ft (55 m). The following factors influence pit width:

1. Stripping machine reach: Strippers are limited by reach, and therefore a limit is placed on the pit width and overburden depth combination if rehandling is to be avoided.

2. Overburden depth and dragline productivity: Simulation studies indicate that narrow pits should be chosen in shallower overburden because of decreased swing angles, and that wide

pits should be selected in deep overburden where extended bench is utilized to minimize percent rehandle. (Anon., 1976b; Hrebar and Dagdelen, 1979) These selections improve dragline production.

3. Reclamation: Narrow pits minimize spoil peak strike-off in regrading, while wide pits increase spoil area and strike-off quantities.

4. Coal loading and hauling: Some minimum width is required for efficient equipment operation; width is a function of equipment size and type combinations.

5. Coal inventory: Width will influence tonnage/unit length of pit.

PIT ORIENTATION. Selection of pit orientation and sequencing is critical to the mine plan, because it will determine equipment selection, operating costs relative to production life, coal quality relative to production life, and so forth.

Straight vs. Curved Pits—Where a property contains a curved outcrop or subcrop where stripping will begin, the operator must decide if the subcrop will be followed or whether straight pits will be employed. Where curved pits are selected, a series of inside and outside curves are usually encountered. The inside curves, where the advancing highwall arc is greater than the spoil side arc, can cause significant spoil room problems, particularly when overburden depth increases in the direction of mine advance. In this case, each new cut contains more overburden volume than is available in the prior cut spoil volume. Excess spoil is soon generated and costly rehandling is required.

Outside curves, with the advancing highwall arc less than the spoil side arc, can be used to advantage in areas of increasing overburden depths. In this case, each new cut has less area than the adjacent spoil area, and this difference can be utilized to accommodate the added volume caused by the increase in overburden height. Outside curves can be used to advantage where an operation encounters ridges perpendicular to the cropline or subcrop.

In addition to the inside-curve problem, curved pits are difficult to lay out in the field and difficult to execute from an operating standpoint. They also require coal haulage on curves in the confined pit. As a result of these problems, many operators choose to develop straight pits, which alleviate these problems.

Straight pits are developed either by a series of short pits or by varying pit width. A series of short pits on the cords of an outside curve can be utilized to straighten the pits. In many cases, auxiliary equipment such as a scraper fleet is used to strip these pits so that the dragline is not utilized until long straight pits are available. This has the advantage of not applying the dragline in inefficient short pits and of postponing dragline investment while low capital cost equipment is used to begin the operation. The other alternative is to utilize the dragline and employ wide pits at the outside curves and narrow pits at the inside curves until a straight pit configuration is achieved. This alleviates the short pit inefficiency but is more difficult to engineer, lay out, and execute. Two papers (Anon., 1976b; Anon., 1977) provide an in-depth discussion of inside and outside curves.

Average vs. Minimizing Stripping Requirements—Orientation of pits and pit sequencing are crucial to the entire mine planning scheme. The common alternatives are averaging and minimizing the stripping ratio over time.

When the ratio is averaged, approximately the same stripping ratio and the same quantity of overburden are handled each year. Overburden depths range from low to high in a given year to balance and average the ratio. The advantages of this approach include fairly constant stripping capacity requirements, constant costs/ton (exclusive of inflationary effects), and constant manpower requirements.

When the ratio is minimized, areas with the low stripping ratio are mined in early years, mid-range ratio in midyears, and high ratio in later years. These plans must also consider such constraints as minimum pit length, avoidance of excessive dragline walking by selecting a logical progression throughout the property, and allowance for efficient haul-road access. The primary advantage of this approach is that the best economics is generally achieved because of postponement of capital expenditure and low direct operating costs in early years of production. However, the disadvantages include continually increasing equipment and manpower requirements in order to strip increasing amounts of overburden. Management of the constantly changing operation is more difficult. However, the economic benefits of minimizing stripping ratio are significant, as described in the case study in the final section.

HAUL ROAD LAYOUT. The average strip mine haul distance in the central United States is approximately 5 miles (8 km) one way. Strip mine roads are usually spaced on 1500- to 1800-ft (450- to 540-m) centers through the spoil to facilitate one-way hauls if possible. In deeper overburden, fewer haul roads through the spoil may be used as a result of significant spoil volume losses. Where contemporaneous haul-road reclamation is required, the trend appears to be fewer haul roads through the spoil. In addition, there seems to be a trend toward steeper ramps. Both of these trends cause decreases in truck productivity because of increases in turnaround areas in the pit and increases in time spent hauling up ramps. Generally, these roads are located to minimize haul distance and designed to facilitate high-speed haulage using width and construction guidelines similar to those presented by Kaufman and Ault (1977).

Two other haul-road arrangements are used in certain situations. Highwall side ramps are used where soil stability problems, spoil volume losses, or unusual property geometry are factors. The prime disadvantage of the highwall side ramps is the dragline inefficiency and auxiliary equipment time associated with continuous rebuilding as the mine advances. Pit-end haul roads are used where spoil volume losses or spoil instability are factors. This requires unique pit geometry, cropline on three sides, or deep box cutting to maintain the access. Long in-pit haul distances are generally associated with pit-end haul roads.

13.2.2.3 Pit Development

CLEARING AND GRUBBING. This involves clearing the property of trees and shrubs and then removing the stumps and roots to insure a homogeneous topsoil. Grubbing is often done with rake-like grubbing attachments on agricultural tractors or dozers.

TOPSOIL REMOVAL. Regulations dictate that the topsoil be removed and ultimately replaced upon graded spoils. Topsoil can either be stockpiled at the side of the pit area for later redistribution or hauled immediately to the graded area for redistribution. In the latter case, the topsoil can either be hauled around the pit or across the pit on spoil bridges. The decision is made on the basis of economics considering topsoil quantities and haul distances.

Self-elevating scrapers are the most common means of removing and redistributing topsoil, although loader/truck or wheel excavator/belt systems find application where topsoil quantities are large and/or haul distances are long.

Topsoil thicknesses are usually determined by auger drilling, and the regulations dictate that a minimum of 6 in. (150 mm) of material be removed. In cases where multiple soil horizons (i.e., A, B, and C) exist, these layers must be removed separately and stockpiled and redistributed to maintain the integrity of each horizon. Topsoil storage, if longer term, must be revegetated to prevent water and wind erosion.

MINE DRAINAGE AND EROSION AND SEDIMENT CONTROL. For many years, it has been good operating practice to divert surface water from active pit areas to eliminate in-pit water problems. Diversion ditch systems were utilized to deflect the water and direct it into natural drainages.

Under current regulations, this practice has been expanded to include the following:

1. Surface drainage from disturbed areas must pass through a sediment pond.

2. Effluent from the ponds must meet limitations of pH, iron, manganese, total suspended solids, etc.

3. Sedimentation ponds must be constructed to standards on capacity, detention time, dewatering, location, slopes, etc.; these standards, particularly the capacity standards, have been contested by industry.

4. Discharge compliance must meet 10-year, 24-hour precipitation events.

5. Treatment is required, if necessary to meet effluent standards.

General design measures taken to meet these standards economically include minimizing the disturbed area, stabilizing backfill, diverting overland flow around or through disturbed areas to reduce pond size, revegetating immediately to reduce sediment load, and separating pit water from other water to minimize treatment. Skelly and Loy (Anon., 1975) have provided an in-depth treatment of drainage and erosion control (see also Chapter 12.1 in this *Handbook*).

REGRADING. Regrading or striking-off the spoil is usually accomplished with large-horsepower dozers, and final grading is done with large graders. Regulations provide a number of general guidelines including:

1. Restoration to approximate original contour.
2. Elimination of highwalls.
3. Restoration of natural drainages to the extent possible.
4. Construction of final slopes not exceeding original slopes.
5. Productivity equal or greater than premining productivity. In addition, the regrading must be done within 180 days or with regrading kept within four spoil ridges of the active pit.

TOPSOIL REPLACEMENT OR REDISTRIBUTION. As previously discussed, scrapers, dozers, or loader/trucks are utilized in this operation with topsoil replaced continuously or from stockpile. Some preparation of the graded spoil, plowing, etc., is usually done to stabilize the topsoil bed. Traffic patterns are designed to prevent over-compaction of the bed.

REVEGETATION. Planting is accomplished either by hydroseeding or with conventional farm equipment and must be done as soon as practical with seed selection based on postmining land use. Success is usually judged by comparison with a reference area, with production rated as some percentage of that area. Erosion must be controlled so that gullies or rills more than 9 in. (276 mm) in depth do not develop. A period of extended responsibility is involved (e.g., 10 years where rainfall is less than 26 in. or 660 mm annually).

More coverage of reclamation is provided in Chapter 12.3.

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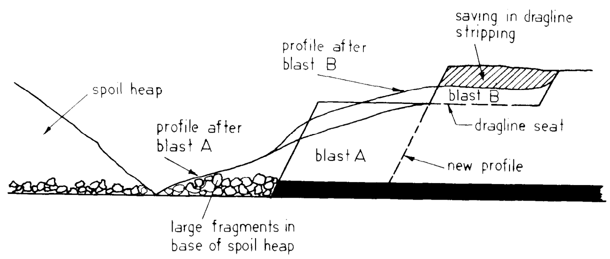


Fig. 13.2.3.1. Throw blasting for a standard dragline operation.

13.2.3 CAST BLASTING OF DEEP OVERBURDEN

THOMAS ATKINSON

For strip (open cast) mining, overburden must often be blasted so that it can be economically excavated by a dragline, that is, the overburden must be well fragmented and loosened, with relatively small lateral displacement. For many years mining engineers have been attracted to *cast blasting* (explosives casting) of overburden to take advantage of the abundant chemical energy of explosives (Brealey and Atkinson, 1968; McDonald, Smith, and Crosby, 1982). The favorable heaving effect of ammonium nitrate-fuel oil (ANFO) and its low cost increased this interest. The advantages of cast blasting become even more desirable when considering deeper, stronger overburdens requiring high powder factors (pounds of explosive/cubic yards of overburden blasted, or kilograms/cubic meters), 1.1 lb/yd^3 (0.65 kg/m^3) being typical for some very strong sandstones, for example, in South Africa and Australia. Recent developments have shown that in certain circumstances, cast blasting in deep overburden can be more economic than conventional stripping.

Early trials were based on reducing the primary overburden cast by the dragline (Fig. 13.2.3.1). In addition to reducing the dragline duty, this method has the added advantage of casting very large rock fragments from the base of the highwall onto the floor of the pit to form the base of the spoil heap, thereby improving spoil heap drainage and stability.

Blasting theory is covered extensively in Chapter 9.2.1.

13.2.3.1 Presplit Blasting

Presplit blasting has been used for dewatering permeable overburden. Where clay-rich and similar materials are absent,

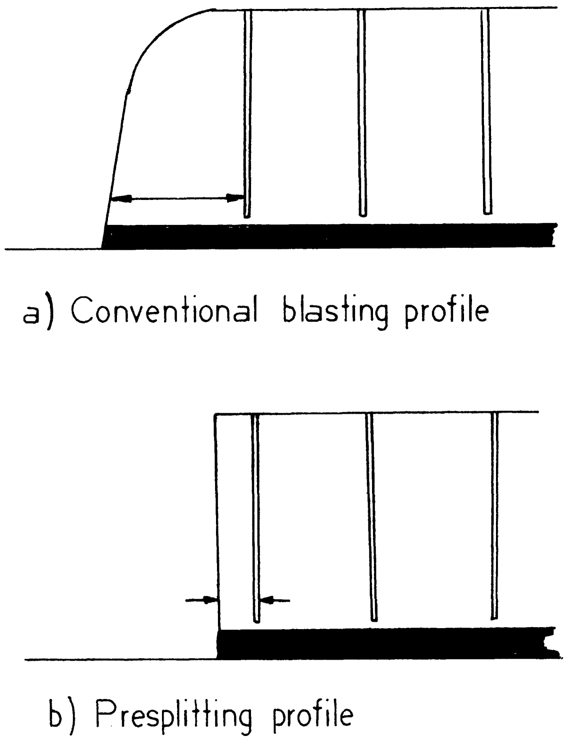


Fig. 13.2.3.2. Vertical presplit highwall.

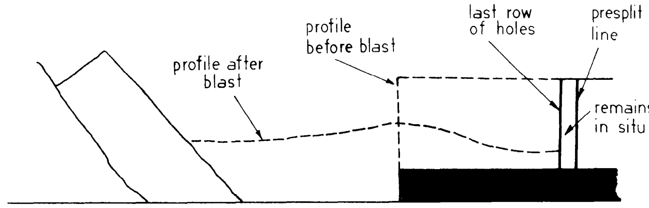
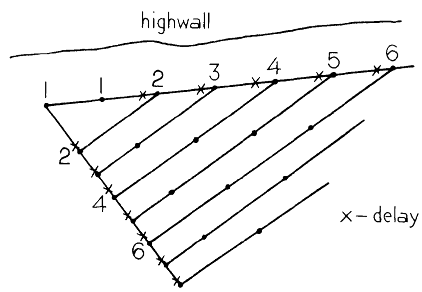


Fig. 13.2.3.3. Blasting with parallel tie up.



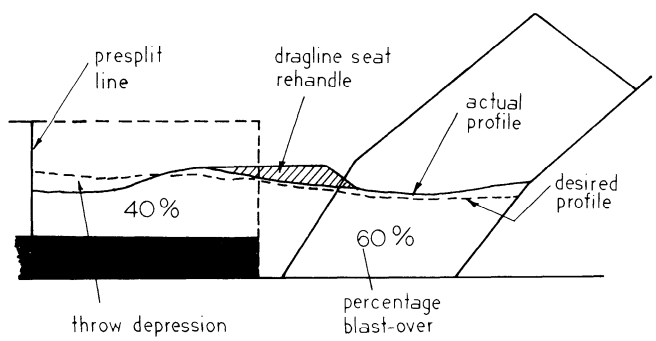


Fig. 13.2.3.4. Blasting method.

low-cost, bulk-placed ANFO has been used in place of more expensive water-resistant explosives, such as water-gels, heavy ANFO, etc. Additionally, in strong ground, a vertical face can be created after blasting the total burden as compared with the irregular sloping face produced by conventional blasting (Fig. 13.2.3.2). It is obvious from Fig. 13.2.3.2b that the vertical face, with greatly reduced distance from the front row of blastholes to the toe of the highwall, will result in far more efficient cast blasting. The combination of pre-splitting and inexpensive ANFO can result in economic cast blasting. It can also be economic with non-vertical highwalls, where an inclined presplit line is used and the blasthole pattern suitably modified.

13.2.3.2 Casting Characteristics

BLASTING PATTERNS. The geometry of the spoil after cast blasting must be controlled to provide a section suitable for dragline operation while moving as much spoil as possible across the pit. Conventional blasting patterns in strip mining are usually square or rectangular with a 90° tie up, that is, rows of holes at 90° to the highwall are detonated simultaneously, then sequentially, so that the spoil is thrown parallel to the highwall. It appears, therefore, that if rows of blastholes parallel to the highwall are detonated simultaneously, initiated from the highwall progressively from front to back, maximum spoil will be cast across the pit. Experience shows that marginally more spoil is cast by this tie up, but a section of overburden between the presplit line and the last row of blastholes remains in situ (Fig. 13.2.3.3).

The best results appear to occur with about a 30° tie up using a staggered "V1" pattern with relatively long, inter-row delays. Fig. 13.2.3.4 illustrates a typical section and the desired section. The "throw depression" can be greater than desirable, and some minor rehandling of the thrown spoil is necessary to form the "seat" for dragline operation.

The spacing:burden ratio of the blasting pattern is affected by site-specific factors and requires detailed consideration to obtain control of the trajectory of the thrown spoil.

Blasthole diameters of 10 to 12 1/4 in. (254 to 311 mm) are currently used. It is doubtful whether larger-diameter holes will provide additional advantages in strong ground, unless more energy can be packed into the blasthole, and closer spacing rather than larger blasthole diameters may be preferable.

DEPTH/WIDTH RATIO. A measure of the efficiency of cast blasting is the amount of spoil thrown beyond the edge of the mineral seam. This may be expressed as the percentage volume thrown by blasting across the edge of the mineral seam, that is, the *percentage blast-over*. Intuitively, for a constant strip width, the percentage blast-over will be expected to increase with increasing depth. There are few published data on this subject, but the results from two sources where strong sandstone overburden was cast blasted gave the following relationship:

$$\text{percentage blast-over} = 57.5\frac{d}{w} + 18 \qquad (13.2.3.1)$$

where d/w is depth/width ratio varying between 0.4 and 0.9, d is depth of the pit to base of the mineral seam, and w is width of the exposed strip.

The percentage blast-over obviously will vary considerably due to site conditions and blasting efficiency, but Eq. 13.2.3.1 is adequate for initial mine planning purposes.

13.2.3.3 Cast Blasting Methods

Major advantages accrue where the dragline can be located on the spoil side of the pit. These are

1. The access ramp roads can be located on the highwall side of the pit, eliminating the problems caused by lack of spoil space at the junctions of the strip with the access ramps (e.g., major rehandling of spoil), since generally, spoil must be piled higher at these junctions and often results in serious spoil-side failures (Walton and Atkinson, 1978).
2. The stripping rhythm is maintained since access ramp junctions do not have to be formed, and dragline productivity on prime overburden is increased greatly.
3. Equipment scheduling is simplified.
4. Access to mineral benches in multi-seam settings is greatly facilitated.
5. Surface reclamation is simplified and land is returned to its planned use at an earlier date due to the absence of access ramp roads through the spoil.

The main disadvantages are

1. The dragline may operate in the less efficient "chop-down" mode.
2. Separate electrical reticulation systems are required on both sides of the pit-on the spoil side for the dragline and on the highwall side for the blasthole drill rigs.

SINGLE-SEAM, CROSS-PIT, CHOP-DOWN OPERATION. A dragline used in the chop-down mode is about 60% efficient compared with the conventional drag mode. Bucket maintenance costs are also higher (although this is considerably alleviated by the better fragmentation achieved in cast blasting). The depth/width ratio of the pit should exceed 0.4 to achieve greater than 40% blast-over for chop-down to be considered; that is, the method is more suited to deeper pits.

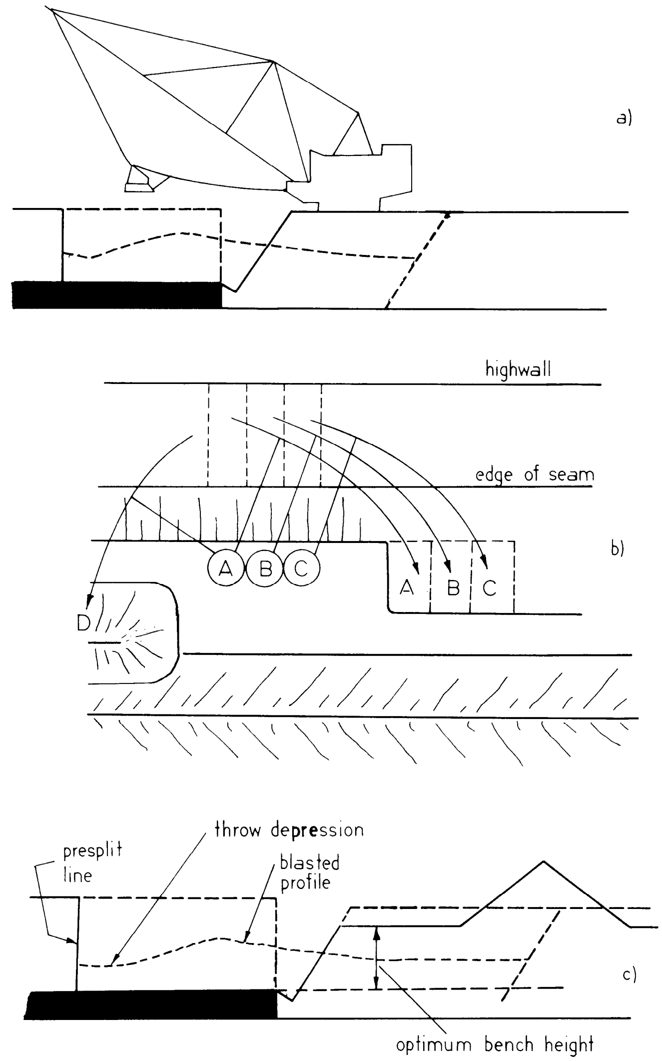


Fig. 13.2.3.5. Cross-pit, chop-down with single seam.

Fig. 13.2.3.5 shows the method of operation. The dragline seat bench height in (c) can be fixed so that all the spoil is required for leveling, that is, the need for leveling spoil peaks for land reclamation is eliminated. This advantage can only be fully realized, however, if the height of the dragline seat above the top of the mineral seam does not exceed the optimum digging depth. Where this height exceeds the optimum digging depth, the bucket must be dragged to a higher elevation so that it may be hoisted away for dumping, thereby increasing the cycle time. In these circumstances, it is usually more economic to level the spoil peaks with conventional mobile equipment. (Note that for a dragline to hoist a full bucket of spoil, the hoist and drag ropes must form an angle, 90°, i.e., be within a semicircle drawn through the fairleads and boom sheaves).

The single-seam, chop-down method may be used to strip seam partings where other seams exist beneath the seam being extracted. This operation results in a reduction of dragline productivity of up to 50%, but if the ratio of parting to lower seam thickness is low, it can be economic.

TWO-SEAM METHOD. Fig. 13.2.3.6 shows a sequence of operations for a typical two-seam setting.

(a) Pit section prior to blasting.

(b) Interburden blasted into the void left after extracting the mineral from the previous strip. Note that large rocks will form the base of the spoil heap and are not handled by the dragline.

(c) Blasted overburden. Note that the throw depression of the interburden blast is filled by overburden spoil.

(d) The dragline seat is prepared by bulldozer.

(e) A key cut is formed to the lower seam, and the upper seam is stripped; this spoil is dumped on the lowwall to provide material for the spoil bench.

(f) The spoil bench is dozed level, and the remaining spoil is "pulled back," being dumped on the spoil bench.

(g) The lower seam is stripped. The mineral extraction operation can now commence. After the mineral is removed, the cycle can be repeated.

The method results in an increase in dragline productivity above that of conventional dragline stripping, but insufficient experience is available in comparable conditions to quantify this increase. There are, however, other marked advantages:

1. Both seams are exposed together, allowing simultaneous mineral extraction. The upper seam can be dozed over the side of the bench (or by other suitable means) on to the lower seam, thereby concentrating and simplifying mineral loading opera-

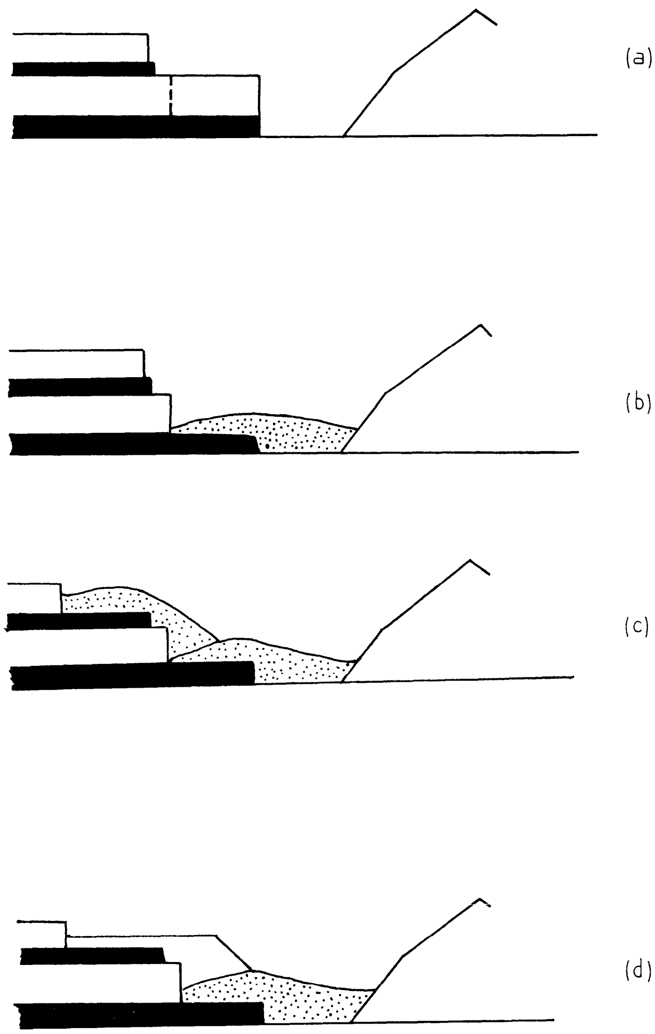


Fig. 13.2.3.6. Two-seam method (a), (b), (c) and (d).

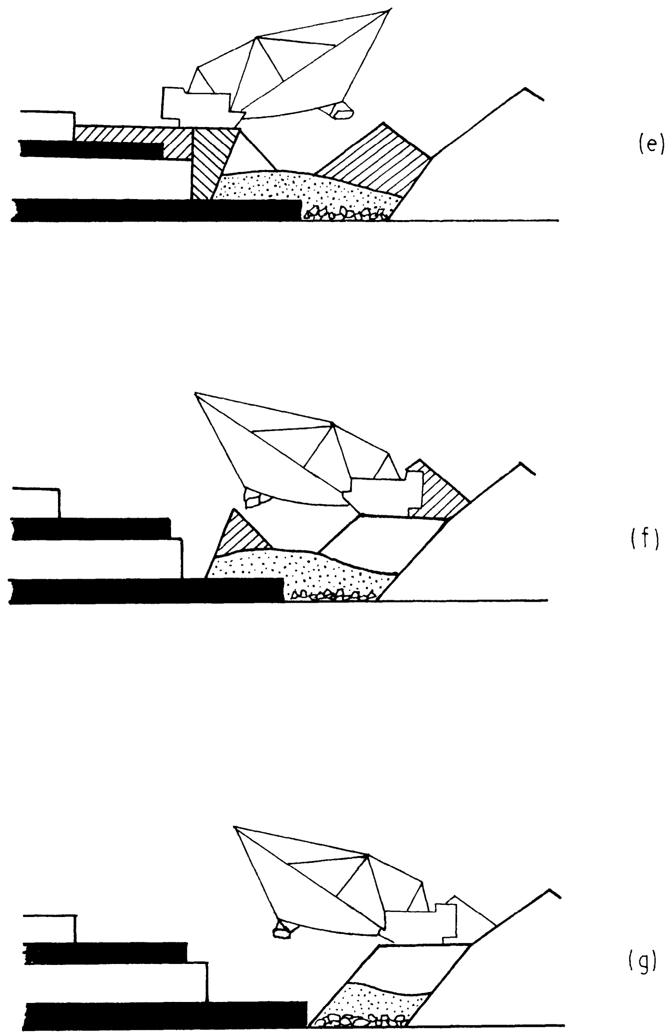


Fig. 13.2.3.6. (Continued) Two-seam method (e), (f) and (g).

tions and improving blending. The volume of in-pit exposed reserves is increased allowing greater flexibility in operation.

2. Spoil-heap stability is greatly improved as strong rocks from deep within the overburden form the base of the spoil heap. In conventional dragline stripping, weaker weathered material forms the toe of the spoil heap.

3. Both seams are exposed simultaneously. In conventional dragline stripping, two dragline passes are necessary. Dead heading is therefore halved.

4. The deeper, stronger interburden is cast by blasting into the void to form the base of the spoil heap, and is not handled by the dragline. Bucket maintenance costs are therefore greatly reduced.

The method can also be used for single-seam, thick-overburden settings to eliminate the chop-down operation of the upper overburden. Where weathered material exists near surface, the upper bench can be presplit with more closely spaced holes than for the lower, more competent overburden. Providing weathering is not too deep, the upper bench slope can often approach 90º (vertical) because the presplit face did not seriously deteriorate after blasting and due to the enhanced drainage effects.

STRIPPING PREVIOUSLY WORKED DEPOSITS. In many countries, thick coal seams that have been previously partially extracted by underground methods (e.g., room and pillar mining) can be economically recovered by strip mining. In some situations, the security of the dragline can be in question where conventional dragline stripping is proposed. Cast blasting provides an alternative stripping method where the dragline is not located over the pillared coal, thereby eliminating any dragline security hazard.

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Chapter 13.3
SELECTION AND SIZING OF EXCAVATING EQUIPMENT

THOMAS ATKINSON

13.3.1 INTRODUCTION

The main operations in a surface mine—ground preparation, excavation and loading, transport, and mineral treatment—are interdependent, and the optimum cost per ton may not be obtained by attempting to minimize each of the individual operational costs. It is not good practice to focus on any one item in isolation. The increased use of explosives may result in reduced loading, transport, and crushing costs, which could amply recover the increased ground preparation costs. Adverse rock materials cause numerous equipment selection problems in surface mining. Abrasion and stickiness cause major problems that are accentuated where intensive continuous systems are employed.

The importance of not selecting items of equipment in isolation and of material properties testing, both prior to selection and during excavation, cannot be overstressed.

The selection of the excavator or loader is of prime importance because it largely determines the other equipment required and the mode of operation; it is the key to low-cost production but is not usually the starting point in planning.

The competence and "diggability" of ground is of major importance in the selection of excavating equipment. It depends on many factors, for example, (1) the intact strength of the ground, (2) the competence of the ground as a whole, (3) the abrasive properties of the mineral constituents, (4) the bulk density of the ground, both in situ (bank) and broken (loose), (5) the flow properties of the broken ground, possible stickiness, etc., and (6) the degree of ground preparation contemplated to achieve a desired fragmentation.

At present, there is no generally accepted quantitative measure of diggability, but a fairly reliable indication can be obtained from (1) similar excavations in the area; (2) the behavior of ground excavated in trial pits; (3) physical tests on core samples recovered by drilling (some caution must be exercised, because exposed beds may have very different characteristics from the cores; the most widely used tests are uniaxial and triaxial compressive strength, shear box, abrasion hardness tests, and a variety of simple field tests; and (4) refraction seismology tests (these are easily carried out, but their interpretation can be difficult in disturbed geologic conditions). Fig. 13.3.1 shows excavation possibilities without blasting and the seismic velocities of various rocks. Generally, several of the above methods are used in conjunction with one another.

Other factors that must be taken into account are the reach of the excavator, both horizontal and vertical, above grade and below grade, both digging and discharging; the bearing strength of the working bench; maneuverability; power supply; capacity; etc. Where the mineral grade varies considerably and stockpiling is uneconomic or not possible because of the nature of the mineral, a larger number of smaller-capacity excavators may be required to meet the mill feed requirements than if the grade is reasonably uniform or if stockpiling is possible.

Basic principles of materials handling are discussed in Chapter 9.3.

13.3.1.1 Cyclic vs. Continuous Excavators

Numerous operational systems and machines are available, and it is necessary to summarize the factors that influence their selection. Fig. 13.3.2, a classification diagram, is based on the method of working, that is, continuous or cyclic, and also shows the materials handling alternatives.

Where it can be applied, *continuous* operation is preferable, since it provides fuller plant utilization and reduces margins added to plant capacity, as well as peaks of all forms (e.g., material flow, mechanical stresses, electrical maximum demands). In open pit mining, the continuous system is typified by the multi-bucket excavator (bucket wheel and bucket chain).

The *cyclic* system is represented by shovels, draglines, front-end loaders, scrapers, rippers, bulldozers, etc.

There is a great deal of overlap between the excavating and loading capabilities of the many machines available, and in order to confine this discussion to a reasonable length, it is proposed to consider here only those that are most widely used in mining. For mechanical and electrical considerations, see Atkinson (1969).

13.3.2 LOADING SHOVELS

The crawler-mounted loading shovel is the machine most capable of handling hard, dense, abrasive, badly fragmented

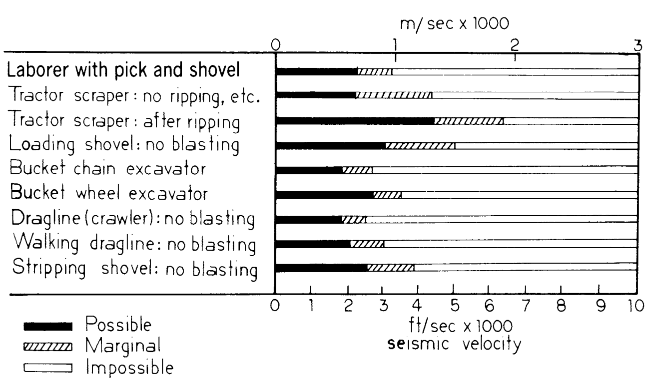


Fig. 13.3.1. Seismic velocity method for determination of excavation possibilities.

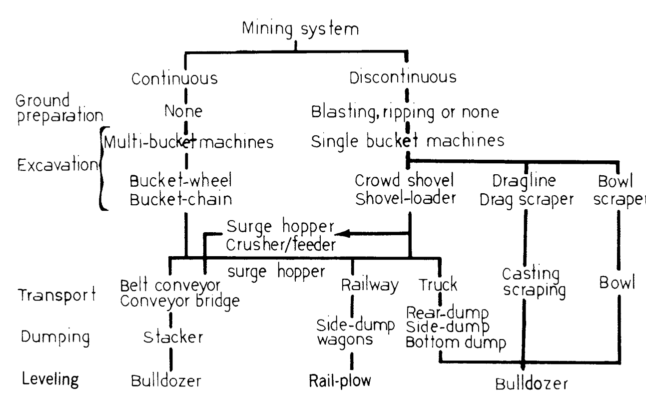


Fig. 13.3.2. Materials handling system classification diagram.

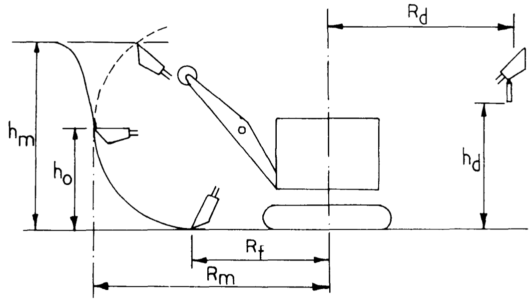
Table 13.3.1. Loading Shovel Cycle Times (sec)

B_c yd^3	m^3	E	Digging Conditions M	M–H	H
4	3	18	23	28	32
5	4	20	25	29	33
6	5	21	26	30	34
7	5.5	21	26	30	34
8	6	22	27	31	35
10	8	23	28	32	36
12	9	24	29	32	37
15	11.5	26	30	33	38
20	15	27	32	35	40
25	19	29	34	37	42
45	35	30	36	40	45

E:
Easy digging, loose, free-running material, e.g., sand, small gravel
M:
Medium digging, partially consolidated materials, e.g., clayey gravel, packed earth, clay, anthracite, etc.
M–H:
Medium–Hard digging, e.g., well blasted limestones, heavy wet clay, weaker ores, gravel with large boulders, etc.
H:
Hard digging–materials that require heavy blasting and tough plastic clays, e.g., granite, strong limestone, taconite, strong ores, etc.

Table 13.3.2. Correction Factor for Shovel Cycle Time Where Digging Depth is Less Than Optimum

Optimum digging depth, %	40	60	80	100
Cycle time correction factor	1.25	1.10	1.02	1.00



ground—by virtue of its positive crowd action and the possibility of applying a high breakout force. It can accurately spot for loading into dump trucks, rail cars, loading hoppers, etc. Because of its robust construction and simple action, it can have relatively high availability; but it is not very mobile and has a poor subgrade digging capability. A competent floor is essential unless oversize crawlers of low bearing pressure are used.

Loading shovels can be divided into three main categories: (1) heavy-duty, mine loading shovels (for dense, abrasive, badly fragmented ground, as is found in most metal mining operations); (2) general purpose loading shovels (for lighter, well-fragmented materials, e.g., sand and gravel, coal, bauxite, etc.); and (3) hydraulic shovels. Loading shovels with dippers as large as 45 yd^3 (35 m^3), are in service.

13.3.2.1 Size Selection

The first major step in shovel selection is the determination of dipper size. Since mine planners are mostly concerned with in situ volumes (bank volumes), these are generally used in calculations. Dipper size can be expressed as

$$B_c = \frac{Q}{C \times S \times A \times O \times B_f \times P} \tag{13.3.1}$$

where B_c is dipper capacity (volume), Q is production required (bank volume per hour), C is theoretical cycles per hour for a 90° swing = $60/t_c$, t_c is shovel cycle time for a 90° swing in minutes, A is mechanical availability during the scheduled hours of work, O is the job operational factor, B_f is the dipper factor, S is the swing factor, and P is the propel time factor.

C: Theoretical cycles per hour. Values of C may be obtained from manufacturers' literature or from time studies. Alternatively, the approximate times in Table 13.3.1 can be used in association with an approximate dipper size. The skill of the operator has some effect on the cycle time, and without the benefit of time

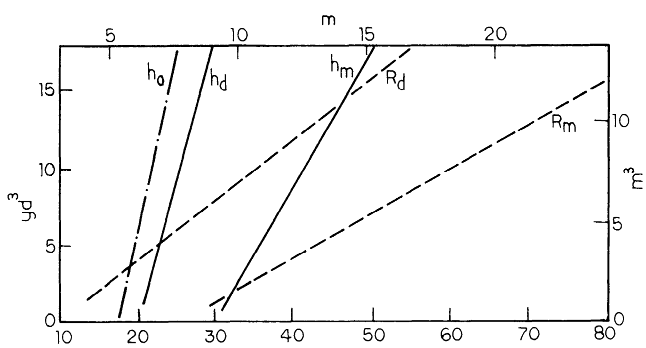


Fig. 13.3.3. Approximate dimensions of loading shovels for initial investigations.

studies or previous experience in similar conditions, precise figures cannot be justified initially.

For most open pit mining operations, shovels have no difficulty in loading up to their optimum digging depth. In strip (open cast) mines, however, the mineral bed may be relatively thin; hence the crowd-hoist time in the shovel cycle will be increased to obtain a full dipper. This may be corrected from Table 13.3.2. The optimum digging depth for a range of shovel sizes is shown in Fig. 13.3.3. In some strip mining operations, the bed underlying the mineral may be insufficiently competent (e.g., fireclays) for the passage of rubber-tired trucks. The haul roads are then located on top of the mineral bed, and the loading shovel may be fitted with a longer-than-standard boom and dipper handle to provide sufficient reach to load the trucks. In these circumstances, the loading shovel cycle times will be greater than those shown in Table 13.3.1, and manufacturers' literature should be consulted, if available. An approximate allowance of 7 to 12% may be added to the cycle times shown in Table 13.3.1.

Table 13.3.3. Shovel Swing Factor

Angle of swing, degrees	45	60	75	90	120	150	180
Swing factor	1.2	1.1	1.05	1.00	0.91	0.84	0.77

Table 13.3.4. Shovel Operating Efficiency

Job Conditions	Management Conditions			
	Excellent	Good	Fair	Poor
Excellent	0.83	0.80	0.77	0.70
Good	0.76	0.73	0.70	0.64
Fair	0.72	0.69	0.66	0.60
Poor	0.63	0.61	0.59	0.54

S: Swing factor. Loading shovel cycle times are normally based on a 90° swing. Variations in the angle of swing necessitate correction of the shovel cycle time; most manufacturers provide charts to determine this correction, but Table 13.3.3 may be used as a guide for preliminary calculations.

A: Availability. A is the availability of the shovel for work during the manned hours; it is generally defined as the mechanical availability during the scheduled hours.

O: Job operational factor. Loading shovels are part of a system and as such are subject to delays due to management, supervision and labor deficiencies, job conditions, climate, etc. The dipper capacity must be increased to compensate for these losses in production time. The job operation factor also can be determined by industrial engineering methods. Care must be taken to ensure that if the propel time *P* is included in *O,* it is eliminated from Eq. 13.3.1.

OA: Operating efficiency. Where no experience is available to enable *O* and *A* to be determined, their product *OA,* the *operating efficiency,* may be determined from Table 13.3.4.

Given a climate that is extreme, in a dusty environment where the ground is dense and abrasive, and if the quality of the labor is poor, performance will be affected adversely due to "poor" job conditions.

If management and supervision are excellent, with good workshops, planned maintenance programs, minimum transport delays, high availability, etc., the actual time spent in producing will be high. Conversely, poor management and supervision will reduce production, and the dipper capacity must be increased to meet production requirements (Drevdahl, 1960).

B_f*: Dipper factor.*

$$B_f = \frac{\text{Fillability}}{\text{Swell factor}} \qquad (13.3.2)$$

Fillability is the loose volume of material excavated in an average load as a ratio of the dipper capacity B_c; this is best determined by field measurements, but Table 13.3.5 provides an approximate guide. *Swell factor* is defined as

$$\text{Swell factor} = \frac{\text{Weight/unit volume (bank)}}{\text{Weight/unit volume (loose)}} \qquad (13.3.3)$$

This is felt to be the preferred method of defining swell factor as it indicates the degree of swell. Most American literature describes this figure as percentage swell, however, and its reciprocal as the swell factor.

Some caution is needed in placing a value on swell factor especially with less-consolidated materials that can decrease in volume with repeated handling. This must be taken into account when field measurements are being planned. Table 13.3.5 provides a guide for common materials.

P: Propel time factor. P depends upon the time required to propel the shovel during maneuvering. In strip mining, where a relatively thin narrow bed is exposed, the shovel must move more often than in the case where a high rock pile is being loaded. Again industrial engineering methods can be used to determine the propel time factor. Table 13.3.6 has been compiled from a large number of operations. The propel time factor is sometimes combined with *O* or *OA,* and care must be taken to avoid double compensation.

The approximate size having been fixed, a standard dipper size can be chosen and a range of shovels can be selected from manufacturers' information. The actual production capacity of each model must then be calculated, manufacturers' data being used to determine that production requirements will be met (e.g., see Table 13.1.2.11).

Detailed studies are required to determine space requirements for a loading shovel. Ample bench room is necessary for operational efficiency, but in general, the slope angle for a single bench, with double-spot loading, is too flat for most multi-bench, conical pits (Fig. 13.3.4) and several bench levels forming a machine loading group must be excavated in sequence. Space must also be available for drilling and blasting and to allow for the passage of trucks. For pits with rail haulage, clearance must be provided for rail track, trolley lines, and shovel; the whole sequence of drilling, blasting, and loading; and trolley and track shifting. The approximate leading dimensions of loading shovels are shown in Fig. 13.3.3 and the clearance heights of rear dump trucks in Fig. 13.3.5 These may be used for early planning, but the actual dimensions of the selected equipment must be used in subsequent mine plans.

13.3.2.2 Cost Estimation

Throughout this chapter, both *ownership* and *operating costs* are estimated. Where discounted cash flow rates of return are to be calculated, all cash outflows are negative cash flows and must be debited at the time of payment. Initial capital costs for equipment must be included in the cash flow at the date of purchase. Replacement capital costs must also be debited at the date of purchase. Ownership costs, other than capital charges (e.g., insurance, etc.) must be debited at the date of payment. Operating costs must be debited on an annual basis.

OWNERSHIP COSTS. Many shovels have a useful life in excess of 20 years, but it is unusual for this figure to be assumed in any assessment of depreciation. Individual company depreciation and replacement policies may determine the depreciation period, and advantage may be taken of tax regulations to write off a machine over a shorter period.

Table 13.3.5. Bulk Density, Swell Factor, and Diggability of Common Materials[1]

Rock	Bank Density (t/m^3)	lb/yd^3	Swell factor	Fillability[2]	Digability[3]
Asbestos ore	1.9	3200	1.4	0.85	M
Basalt	2.95	5000	1.6	0.80	H
Bauxite	1.9	3200	1.35	0.90	M
Chalk	1.85	3100	1.3	0.90	M
Clay (dry)	1.4	2400	1.25	0.85	M
Clay (light)	1.65	2800	1.3	0.85	M
Clay (heavy)	2.1	3600	1.35	0.80	M–H
Clay and gravel (dry)	1.5	2500	1.3	0.85	M
Clay and gravel (wet)	1.8	3000	1.35	0.80	M–H
Coal (anthracite)	1.6	2700	1.35	0.9	M
Coal (bituminous)	1.25	2100	1.35	0.9	M
Coal (lignite)	1.0	1700	1.3	0.9	M
Copper ores (low-grade)	2.55	4300	1.5	0.85	M–H
Copper ores (high-grade)	3.2	5400	1.6	0.80	H
Earth (dry)	1.65	2800	1.3	0.95	E
Earth (wet)	2.0	3400	1.3	0.9	M
Granite	2.41	4000	1.55	0.8	H
Gravel (dry)	1.8	3000	1.25	1.0	E
Gravel (wet)	2.1	3600	1.25	1.0	E
Gypsum	2.8	4700	1.5	0.85	M–H
Limonite	3.2	5400	1.4	0.85	M
Iron ore (40% Fe)	2.65	4500	1.4	0.8	M–H
Iron ore (+ 40% Fe)	2.95	5000	1.45	0.8	M–H
Iron ore (+ 60% Fe)	3.85	6500	1.55	0.75	H
Iron ore (taconite)	4.75	8000	1.65	0.75	H
Limestone (hard)	2.6	4400	1.6	0.80	M–H
Limestone (soft)	2.2	3700	1.5	0.85	M–H
Manganese ore	3.1	5200	1.45	0.85	M–H
Phosphate rock	2.0	3400	1.5	0.85	M–H
Sand (dry)	1.7	2900	1.15	1.00	E
Sand (wet)	2.0	3400	1.15	1.00	E
Sand and gravel (dry)	1.95	3300	1.15	1.00	E
Sand and gravel (wet)	2.25	3800	1.15	1.00	E
Sandstone (porous)	2.5	4200	1.6	0.8	M
Sandstone (cemented)	2.65	4500	1.6	0.8	M–H
Shales	2.35	4000	1.45	0.8	M–H

[1] These figures vary from location to location, and tests should be made where possible. Allowance should be made for operation in wet conditions as density varies with moisture content.
[2] Based on shovel dippers.
[3] For explanation, see footnote to Table 13.3.1.

Table 13.3.6. Propel Time Factor

Strip mines	0.75
Open pit mines	0.85
Sand and gravel pits	0.90
High-face quarries	0.95

In addition to the FOB price, freight and insurance costs, erection costs, insurance during erection, and interest charges up to the time of going into production must be included.

The following form is intended as a guide for the estimation of ownership costs for loading shovels and, with suitable modifications, for any production-cycle equipment. (Note: this is the format followed in Chapter 13.1.2 for the numerical example).

These figures include the cost of supervision of erection by the manufacturer, that is, time, subsistence, local transport, skilled and unskilled labor, erection equipment (e.g., cranes, tools, slings), workshop costs, etc., but not the travel expenses of the manufacturer's erectors. The costs should be adjusted to take account of (1) distance from port of entry, railhead, road, etc., from erection site (off-loading facilities, availability of heavy transport); (2) availability of skilled labor; (3) quality of mine supervisory staff; and (4) availability of cranes, tools, workshop services, etc.

OPERATING COSTS. Operating costs consist of maintenance and supply costs, electrical power, and labor. The list of steps initiated under the ownership cost is continued on the following form.

TOTAL OWNERSHIP AND OPERATING COSTS. The total cost is the sum of the ownership and operating costs, also appearing on the cost form.

It should be noted that no administrative, development, or other charges are included in these figures.

13.3.3 LOADING DRAGLINES

The crawler-mounted dragline finds limited application as an open pit loading machine because (1) it has a less positive digging action, particularly when chopping down, than the shovel, and the smaller sizes are not suitable for loading dense, badly fragmented rocks; (2) the cycle time is longer than that of an equivalent shovel; and (3) it has poor spotting ability when loading.

Guide for Estimation of Ownership and Operating Costs

1. FOB machine price, including optional extras, sales taxes, etc. ______
2. Freight and insurance (to site) ______
3. Import duty (if any) ______
4. Subtotal ______
5. Ballast, if locally manufactured ______
6. Erection costs (see Table 13.3.7) ______
7. Insurance during erection ______
8. Subtotal ______
9. Interest up to start production (allow interest on 30% of sub-total 8) ______
10. Subtotal ______
11. Shovel write-off period:
    (11 a) n = _____ years
    (11 b) _____ hours/year
    (11 c) _____ total hours
12. Machine depreciation and amortization

$$Cost/h = \frac{(Subtotal\ 10)}{(Item\ 11c)} = \underline{\qquad\qquad}$$

13. By use of average investment formula, assuming depreciation charges, replace original investment; average machine investment

$$= \frac{(Subtotal\ 10) \times (n + 1)}{2n} = \underline{\qquad\qquad}$$

14. Interest rate __________%
15. Insurance __________%
16. Taxes, etc. __________%
17. Total __________%
18. Interest, taxes, insurance, etc. (Costs/hour)

$$Machine = \frac{(Total\ \%) \times (Item\ 13)}{(Item\ 11\ b)} = \underline{\qquad\qquad}$$

19. Trailing cable costs (if electrically powered) Capital cost + import + insurance up to start of production (see Table 13.3.8)
20. Trailing cable life
    20 (a) _____ years
    20 (b) _____ hr/year
    20 (c) _____ total hr
21. Trailing cable depreciation

$$Cost/hr = \frac{(Item\ 19)}{(Item\ 20)} = \underline{\qquad\qquad}$$

22. Trailing cable, average investment

$$= \frac{(Item\ 19) \times (Item\ 20\ a + 1)}{2 \times (Item\ 20\ a)} = \underline{\qquad\qquad}$$

23. Trailing cable interest, taxes, insurance, etc.

$$Cost/hr = \frac{(Item\ 22) \times (Item\ 17)}{(Item\ 20b)} = \underline{\qquad\qquad}$$

24. Total ownership costs/hr: ______

$$(Item\ 12) + (Item\ 18) + (Item\ 21) + (Item\ 23)$$

25. Maintenance and supply costs/hr

$$= \frac{10\% \times (\text{Subtotal 4}) \times (\text{H (Table 13.3.9)}) \times (\text{M (Table 13.3.10)})}{(Item\ 11b)} = \underline{\qquad\qquad}$$

26. Electrical power consumption/hr × cost/kWh (see Table 13.3.11) ______
27. Labor rate/hr (to include social benefits, taxes insurance, etc.) ______
28. Total operating costs/hr
    (*Item* 25) + (*Item* 26) + (*Item* 27) ______
    * Where there are no figures available from other mining operations in the area, rates based on local wages rates are invariably too low. Opening a new mine usually stimulates the local economy and local wage rates often double within one year. The social benefits must be carefully investigated as these can be as high as 250% of the monetary wages.
29. Total ownership + operating costs
    = (*Item* 24) + (*Item* 28) ______
30. $\text{Cost/ton} = \frac{(Item\ 29)}{\text{tons/hr}}$ ______

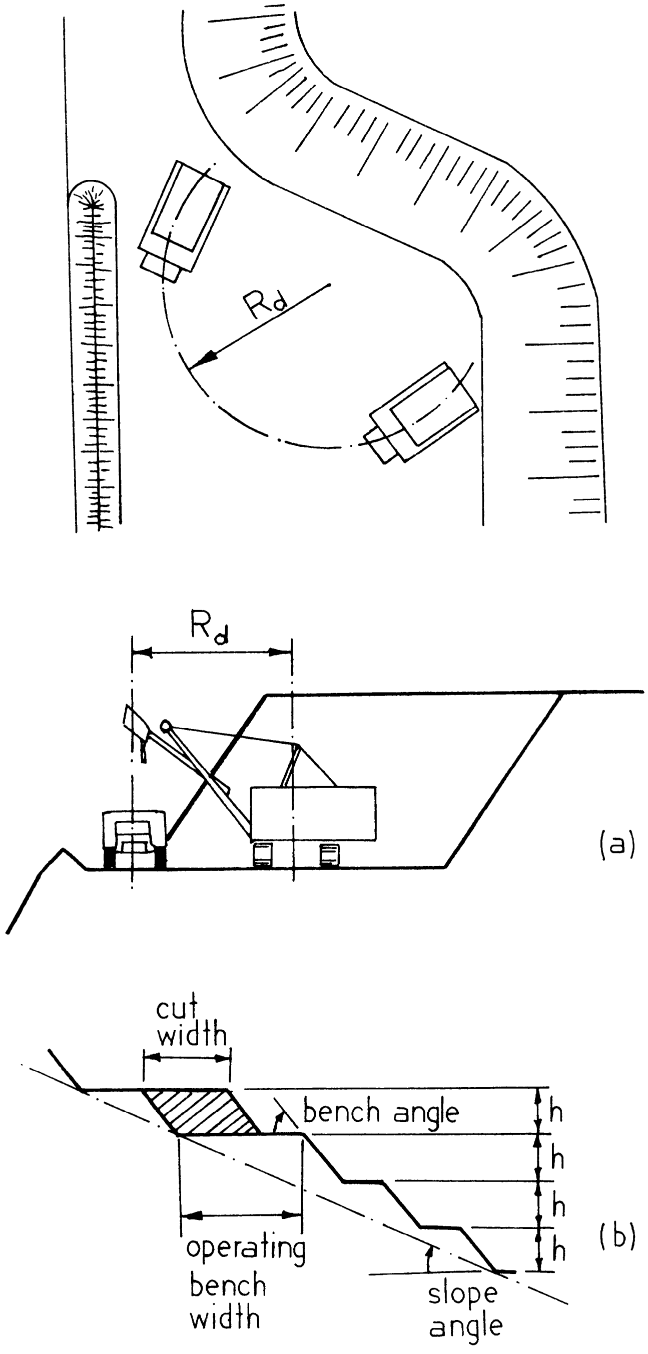


Fig. 13.3.4. Double-spot loading and machine excavation group.

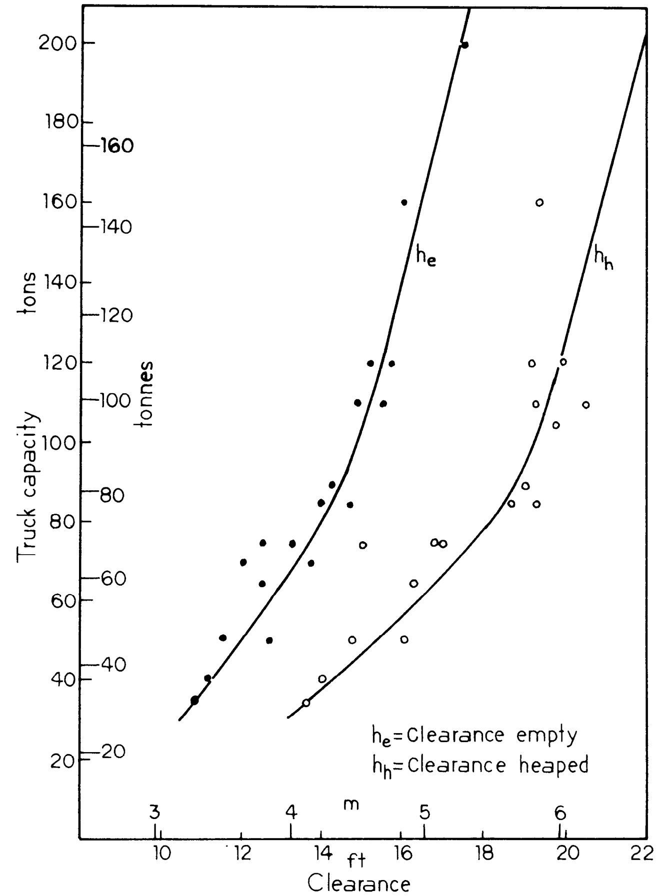


Fig. 13.3.5. Clearance heights for rear-dump trucks. For initial investigations.

Table 13.3.7. Loading Shovel Erection Costs (1989 projections)

Dipper capacity		Erection costs
yd³	m³	$
5	46	18,000
8	66	28,000
10	88	40,000
15	12	48,000
25	20	*

* Inadequate data available: manufacturers' figures must be obtained.

Table 13.3.8. Trailing Cable Prices for Loading Shovels ($/yd)*

Dipper capacity		Voltage, kV		
yd³	m³	4	7	13
5–10	4–8	36	30	30
10–15	8–12	67	55	44
15–25	12–20	—	75	61

* Based on a copper price of $2900/t. *These figures should be used for preliminary estimating purposes only.*
Conversion factor: $1/yd = $1.09/m.

The excellent deep-digging characteristics of the dragline result in reduced gradients for truck and rail haulage operations where the bucket excavates below grade. It can operate from any horizon within the excavation and, unlike the shovel, can stand on a selected competent bed. Because of its deep-digging capability, the crawler-mounted dragline finds application in wet pit operation (e.g., sand and gravel, chalk, etc.) excavating box cuts, recovering remnants from pit floors, etc., and in general work (e.g., digging sumps, extending inclines, etc.) The basic formula for loading shovels can be adapted to determine the output of crawler-mounted draglines, but, because of their limited use in

Table 13.3.9. Correction Factor H for Hours of Operation

hr/year	H
up to 3400	1.00
3401–4500	1.15
4501–5000	1.20
5001–5500	1.30
5501–6000	1.35
6001–6500	1.40
6501–7000	1.45
Above 7000	1.50

Source: Weis, 1970.

Table 13.3.10. Correction Factor M for Different Materials

Material*	M
Coal	0.70
Sand	0.85
Clay	0.90
Copper ores	1.00
Limestone	1.00
Hematite	1.00
Magnetite	1.00
Granite	1.15
Taconite	1.25

* The abrasive qualities and density of the materials should be carefully considered.
Source: Weis, 1970.

Table 13.3.11. Electric Power Consumption of Loading Shovels

Dipper size		Consumption, kWh/hr	
yd^3	m^3	(a)	(b)
5	4	110	85
8	6	260	160
10	8	355	200
15	12	450	250
20	15	670	370
25	19	900	500

(a) Good conditions, high operating efficiency.
(b) Bad conditions, poor operating efficiency.

open pit mining, a full treatment is not appropriate here. Table 13.3.12 provides a summary of the outputs of crawler-mounted draglines. Ownership and operating costs can be calculated by use of the method adopted for loading shovels.

13.3.4 CYCLIC STRIPPING MACHINES

Large single-bucket stripping machines mainly were developed in the United States to expose relatively near-surface coal deposits, whereas the major developments in multi-bucket machines took place in German brown coal mines. Application of these machines has now extended into all strip mining operations. The most important machines in this group are walking draglines, stripping shovels, and bucket wheel excavators.

Because of the huge capital investments involved, and because the investment decisions are usually irrevocable, the selection procedure for large stripping machines must be sufficiently comprehensive to cover all eventualities throughout the stripping of a deposit.

Table 13.3.12. Outputs of Crawler-mounted Draglines (bank volumes)[1]

Bucket capacity		Boom length		Production yd^3/hr (bank) Digging conditions[2]		
yd^3	m^3	ft	m	E	M	M–H
3	2.5	65	20	220	135	70
4	3	70	21	250	150	85
5	4	80	24	315	200	105
6	4.5	100	30	330	215	115
7	5.5	140	43	341	224	125
10	8	160	49	435	290	180

[1] Based on a job efficiency of 0.8, a fillability of 0.75, a swell factor of 1.35, and a swing angle of 110°. No proper time factor is included.
[2] For explanation, see footnote to Table 13.3.1.
Conversion factor: 1 yd^3/hr = 0.7646 m^3/h

The first stage in the selection procedure is to fix the production requirement of the mineral loading machines; from this, the output of the stripping machine is determined. This is not usually a single calculation since some optimization procedure within the market constraints is often essential. The maximum volume to be stripped is the main factor in the determination of the output of the stripping machine, and the maximum depth decides the reach. The geometry of most strip mining operations allows fairly limited reserves only to be exposed; stripping and loading facilities must therefore be correctly matched and machine availabilities must be accurately forecast.

Where the demand for a product is seasonal (e.g., fuel minerals, building materials, agricultural fertilizers, etc.), and stockpiling is not possible or economic, the viability of the operation must be based on the use of machines that can meet the maximum production requirements. If significant lateral variations in grade occur, and again if stockpiling is impractical, a number of small stripping operations may be needed to exploit a deposit. This entails the use of a greater number of smaller-capacity machines, but since the reach of the stripping machines is determined by the depth of overburden, the reach will remain substantially the same as for a larger-capacity single machine, and this invariably results in increased overall capital cost.

Because they are used for similar duties, it is convenient to consider walking draglines and stripping shovels together as single-bucket excavators. Multi-bucket machines are considered in a separate segment.

13.3.4.1 Walking Draglines

Walking draglines are extensively used for direct casting operations since they have a better (capacity × reach)-to-service weight ratio than any other single-bucket machine. They can be designed to create relatively low bearing pressures by the use of a large tub, but this, of course, results in the center of rotation being farther back from the edge of the cut, thus reducing the reach of the machine. Walking mechanisms are usually cam-operated, but, because of weight distribution problems, hydraulic systems have been adopted for large machines in the United States and the USSR. Walking draglines have been extensively developed over the past 20 years, and a machine with a 220-yd^3 (170-m^3) bucket, a 400-ft (122-m) boom, a machine mass (service weight) of 14,000 tons (12,700 t), and a connected (ac) load of 48,500 hp (36,000 kW) is in service in the United States.

Table 13.3.13. Approximate Cycles/Hour for Stripping Shovels and Draglines*

Bucket or dipper size			
yd^3	m^3	Dragline	Shovel
8–35	6–27	58	69
36–59	28–45	56	68
60–200	46–150	53	64

* These figures are based on a 90° swing for a shovel and a 120° swing for a dragline, which approximates most field conditions.

Table 13.3.14. Stripping Machines—Swell Factor and Fillability*

Overburden conditions	Swell factor	Shovel dipper fillability	Dragline bucket fillability
Light blasting	1.23	0.90–0.95	0.85–0.90
Medium blasting	1.33	0.85–0.95	0.80–0.90
Heavy blasting	1.40	0.80–0.90	0.75–0.85
Bad fragmentation	1.45	0.75–0.85	0.70–0.75

* For fillability, the lower figures refer to small machines and the higher figures to large machines

Although smaller machines work most efficiently in less-consolidated, well-fragmented rocks, large draglines are extensively used in strong but well-blasted ground. To prepare a highwall for dragline working, blasting must result in sufficient fragmentation to ensure efficient bucket loading, and the blasted burden must remain sufficiently stable to allow the dragline to dig and travel without danger of collapse. This may entail buffer blasting and decking of charges where some strata are likely to yield large lumps.

The bottom plates of the dragline tub can become distorted or punctured by point loading, and it is usual for a track to be prepared on which the machine will "walk." The costing of the operation must take into account the price of the bulldozer or other units used for this purpose.

13.3.4.2 Stripping Shovels

The stripping shovel has found application because it is usually more productive than the walking dragline by virtue of its positive dipper loading action, shorter swing time, and ability to handle strong, dense rocks. The largest machine reported in service to date has a 180-yd^3 (138-m^3) dipper on a 215-ft (65-m) boom with an ac installed load of 30,000 hp (22,500 kW). Self-cleaning crawlers and hydraulic leveling jacks are universal.

A competent floor is essential, and for large machines, some preparation of the floor may be necessary to reduce excessive local bearing pressures and to obtain satisfactory crawler life. The cost of this operation must be accounted for.

SHOVEL vs. DRAGLINE. The fields of application of stripping shovels and draglines greatly overlap. However, with the advent of inexpensive ANFO explosives, the superior digging depth and reach of the dragline, and its lower capital and operating costs, the walking dragline has virtually superseded the stripping shovel.

13.3.4.3 Size Selection

The main steps in selecting a stripping machine are (1) determination of the approximate dipper or bucket capacity, (2) determination of the machine geometry, and (3) selection within a standard range, where applicable, and reassessment of the model selected.

The approximate dipper or bucket capacity can be readily calculated from Eq. 13.3.1, the overburden stripping rate having been determined from the mineral production requirements. Because of the huge capital investments involved, stripping machines are usually scheduled to work 22.5 hr/day and up to 350 days/yr. Detailed investigation and planning are essential to ensure that adequate exposed reserves are available during periods allocated for deadheading, overhaul, etc.

C: Theoretical cycles per hour. C may be obtained from time studies or from Table 13.3.13.

S: Swing factor. No correction for swing factor need be made in the preliminary calculations unless, in the case of a dragline, a large proportion of chopping down is proposed. That portion of the operation where chopping down is used should have the number of cycles per hour reduced by 20%.

Example 13.3.1. A dragline performs 60 theoretical cycles/hour for a 120° swing; 30% of the overburden is cast by chopping down.

Solution. The cycles/hour when chopping down are

$$60 \times 0.8 = 48$$

The total approximate theoretical cycles/hour are

$$\begin{aligned} 0.3 \times 48 &= 14.4 \\ 0.7 \times 60 &= \underline{42.0} \\ & \underline{56.4} \end{aligned}$$

OA: Operating efficiency. Where no experience is available to determine *O* and *A*, their product *OA* may be determined from Table 13.3.4.

B: Dipper or bucket factor. If no data are available, the fillability and swell factor may be obtained from Table 13.3.14.

P: Propel time factor. In the absence of time studies, the following propel time factors can be used (Weis, 1969):

shovels	0.96
draglines	0.94

These figures are based on normal operations and include the deadheading associated with such operations. Should the mineral deposit be such that greater than normal deadheading is necessary, allowance must be made for this. Time studies often include the propel time in the operating efficiency and care must be taken to avoid overcompensation.

The approximate dipper size having been determined, the next step is to determine the machine geometry.

MACHINE GEOMETRY. The two main dimensions to be determined are the dumping radius and the dumping height. Fig. 13.3.6 illustrates an idealized dragline stripping operation.

Usually, to avoid an excessively long boom, no berm is used in dragline operations, the cut width being equal to the pit width and all exposed mineral being loaded out. If the dragline is located on the surface, as in Fig. 13.3.6, the minimum width of cut can be determined by the mineral loading and transport equipment requirements (see Figs. 13.3.3 through 13.3.5). For small loading shovels, pits can be as narrow as 50 to 60 ft (15 to 18 m); for large shovels, up to 15 yd^3 (11.5 m^3), pit widths of 80 to 100 ft (24 to 30 m) are required. Apart from a reduction of the boom length required, and, hence of the capital cost, a narrow

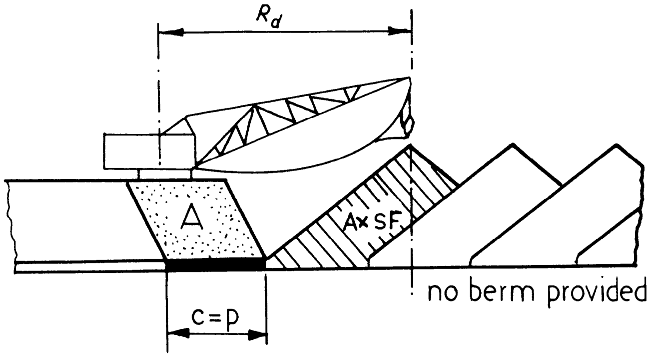


Fig. 13.3.6. Idealized dragline stripping operation.

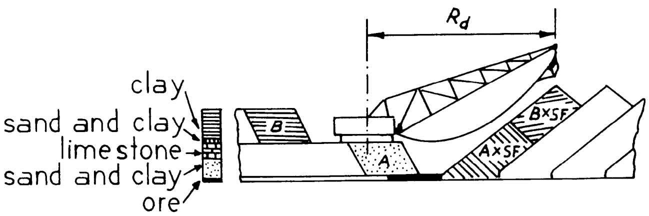


Fig. 13.3.7. Dragline on intermediate bench.

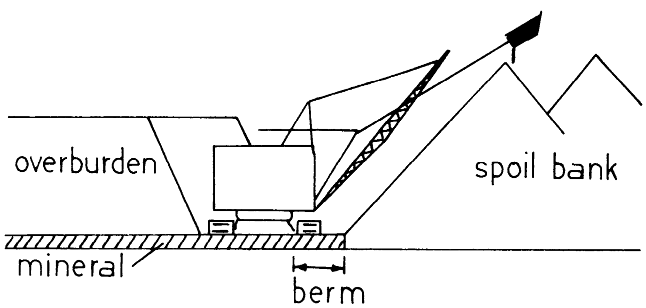


Fig. 13.3.8. Idealized stripping shovel operation.

cut (and in this case, pit) width provides more efficient use of the spoil space since it reduces the valleys between the spoil peaks. Short boom lengths reduce the dragline cycle time, some increase in stripping capacity being achieved.

Other factors affect pit width, for example, an increase may be desirable to allow greater flexibility of the loading operation, or safety of personnel and equipment may require a wider pit than the minimum. Fig. 13.3.7 illustrates the practice adopted where the ground bearing strength of the surface is too low to support the dragline. A competent bed is selected within the overburden as the horizon of the working bench. In this case, the minimum pit width must be sufficient to allow the dragline boom to swing through 90° when in the operating position nearest the edge of the highwall.

Fig. 13.3.8 illustrates an idealized shovel stripping operation. Obviously, the pit must be wide enough to allow the shovel to swing through 90° when it is in the operating position nearest the spoil bank. The stripping cut width may be less than the pit width, and a berm of mineral remains. The stripping cut width is dependent on the width required for the operation of the

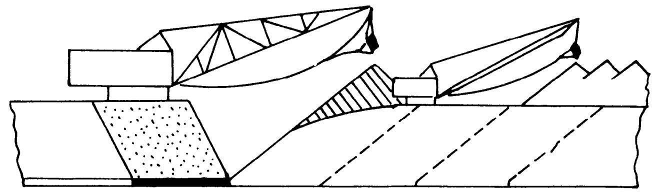


Fig. 13.3.9. "Hay-making" operation: double handling with two draglines.

mineral loading and transport equipment. It is, therefore, generally equal to the mineral cut width or a multiple of it.

The overburden thickness may vary considerably due to hillside outcrops, steeply inclined mineral beds, or areas of undulating surface topography. If, in these circumstances, the reach of the dragline is determined by the average overburden thickness, it will be inadequate for the thicker overburden. To avoid over investment where occasional areas of thick overburden occur, the following short-term measures can be adopted: (1) spoil the overburden on an outside curve to create additional spoil space; in contour mining operations with short irregular strips, this can often be achieved; (2) temporarily reduce the pit width to the minimum possible; (3) make a forecut, outside contractors possibly being employed; and (4) in the case of a shovel where the spoil is broken rock, operate with the spoil bank side crawler off the mineral bed.

If thick overburden is extensive, it may be necessary to extend the reach of the stripping machine to cater to the maximum overburden thickness, or to rehandle the spoil, usually by "hay-making" (Fig. 13.3.9) or by the spoil bridge methods.

The geometry of stripping machines is best determined by drawing or computing average and extreme pit sections and plans showing both stripping and loading operations. Some trial and error is involved. Pit sections for uniform conditions can be prepared quickly by use of the method shown in Fig. 13.3.10. The overburden and mineral thicknesses, the safe slope of the highwall, and the angle of repose of the spoil are known, and the cut width can be determined from the loading and transport requirements. The width of the berm (if any) required for transport, blasthole drilling, or other purposes can be decided and the pit width determined. The section can then be partially drawn, the slope of the pit side faces of the spoil heaps only being indicated. The broken line /–/ is then drawn at distance H × swell factor parallel to the floor of the mine. The /–/ line is then bisected as shown, and the peaks of the spoil heaps are determined by drawing in the other face of the spoil heaps at the angle of repose of the spoil. The dumping radii and the dumping heights of shovels or draglines can then be determined.

A method proposed by Rumfelt (1961) for the preliminary selection of stripping machines uses the "maximum usefulness concept." The *maximum usefulness factor* MUF = dipper or bucket capacity × reach of the machine. Rumfelt investigated a large number of stripping machines and produced MUF—machine mass (service weight) curves for shovels and draglines. To ensure valid comparison, the reach of each machine was modified to provide tubs or crawlers that gave the same ground bearing pressures, and flat, uniform, geological conditions were assumed. Rumfelt showed that the curves were straight lines with slopes (yd^3-ft/lb or m^3-m/kg) of $1/745$, for shovels and $1/575$, and $1/467$ for draglines.

The steeper curve for draglines represents newer and prospective designs, whereas the flatter curve relates to older designs. The method may be used for preliminary evaluation either

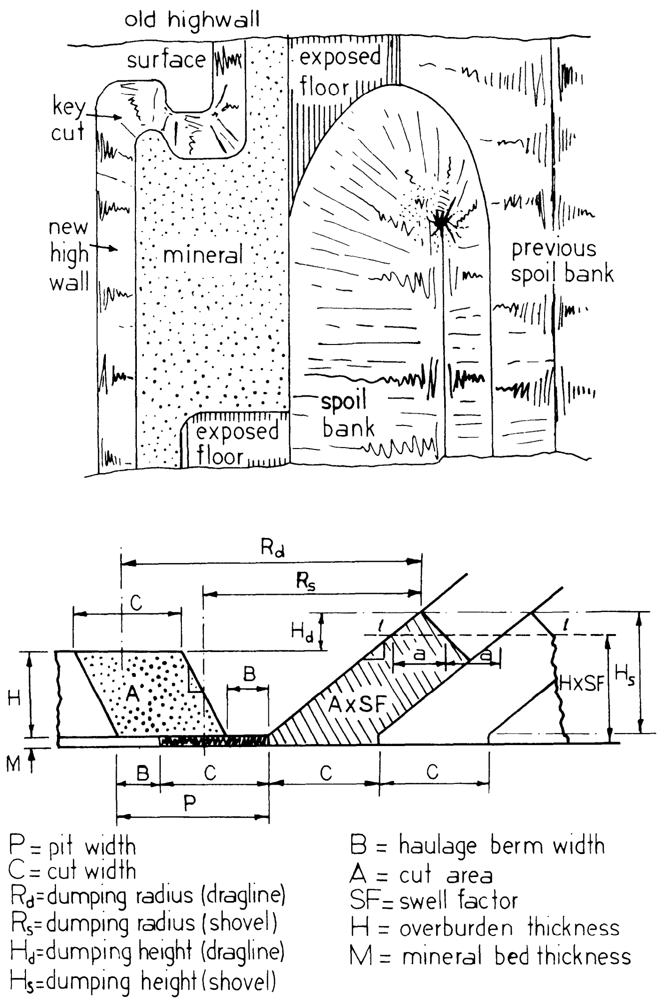


Fig. 13.3.10. Strip mining cut diagram for determination of stripping.

within or beyond the present range of designs. Final selection must, however, be based on more detailed analysis.

A further method (Weis, 1969) used computer-generated tables covering a range of pit geometry variables.

In thick mineral operations, it may be advantageous to load out ore with the stripping dragline. Pit plans and sections must be drawn in detail to determine the machine geometry, and account must be taken of the increased cycle time when loading, as the bucket must be accurately spotted.

FINAL SELECTION. The bucket or dipper size, the dumping radius, and the dumping height having been determined, it is not always possible to select a machine with perfect dimensions, and the nearest model may have to be adapted. Usually, an increase in dumping radius can be achieved by a reduction in dumping height, and vice versa. Careful investigation is essential to avoid selecting an oversized machine, and hence incurring over-investment (see Table 13.3.15). Often large stripping machines are not catered to by standard models, and investigations and discussions with manufacturers are essential before the final design is settled.

When the machine has been selected, it is necessary to recalculate its production by use of the manufacturer's cycle times and swing factors. If a single manufacturer has not been selected, Table 13.3.16 may be used for estimating cycle times. The figures given in the table are based on normal-geometry machines; (1) for stripping shovels, average overburden thickness = 0.4 × maximum dumping radius; and (2) for walking draglines, average overburden thickness = 0.3 × dumping radius. For extra-long-boom units, add 0.75 sec to the cycle time for each 1 yd (1 m) that the above-reduced radii dimensions exceed the overburden thickness.

13.3.4.4 Cost Estimation

OWNERSHIP COSTS. The format used to calculate ownership costs for loading shovels may also be used for single-bucket stripping machines. A stripping machine can have a useful life in excess of 30 years, but company policy, tax regulations, etc., influence the actual depreciation period. It is usual to schedule large stripping machines to operate up to 350 days/yr on a three-shift basis of 22.5 scheduled hours.

It is not possible to define erection costs as simply as for loading shovels, for both the dumping radius and dumping height, as well as the dipper or bucket size, influence the erection cost of stripping machines. The mass (service weight) may be obtained from manufacturers' literature or from Rumfelt's MUF—service weight curves (Rumfelt, 1961). It should be remembered that these curves use the reach of the machine, however, and not the dumping radius. The erection costs based on 1989 projections are

$$\text{Erection costs (\$US)} = cW \qquad (13.3.4)$$

where, in English units, W is machine mass (service weight) in lb, and c is a constant = 0.28 for stripping shovels and = 0.38 for walking draglines. In SI units, W is machine mass in t, and c = 128 for stripping shovels and = 176 for walking draglines. The figures obtained are on the same basis as those given in Table 13.3.7; they should be adjusted as necessary where conditions warrant.

Some care is necessary with very large machines as erection costs may include onsite fabrication and machining of large components. In these circumstances, detailed consultation with the manufacturer is essential.

The trailing cable should form part of an integrally engineered electrical system, but, for preliminary purposes only, the cost figures in Table 13.3.17 may be used.

OPERATING COSTS. Again the format used for loading shovels may be used for the calculation of operating costs for stripping machines. Maintenance and supply costs are calculated in the same way, except that the following figures should be used in item 25 (Weis, 1969):

Walking draglines	7%
Stripping shovels	8%

The correction factor H for hours of operation does not generally apply as it is usual to operate for the maximum possible number of hours, and the above figures are based on an annual operation of at least 7000 hr.

The correction factor M for different types of overburden is shown in Table 13.3.18. Some account should also be taken of the abrasiveness of the overburden; for example, if the overburden is highly abrasive and badly fragmented, then M would be taken as 1.25.

TOTAL OWNERSHIP AND OPERATING COSTS. Total ownership and operating costs, the cost per unit volume (bank), and the stripping cost per ton (tonne) of mineral can be calculated in the same general way as for loading shovels. Again no administrative or amortization charges are included.

Table 13.3.15. Approximate Specifications of Stripping Machines

Walking Draglines

Bucket size		Dumping radius		Dumping height		Width over shoes		Machine	
yd^3	m^3	m	ft	m	ft	m	ft	mass, t	Ballast, t
15	11.5	50	165	21	70	14	45	470	115
20	15	58	490	23	75	16	52	550	160
40	31	67	220	26	85	22	72	1250	200
50	38	79	260	30	100	23	76	1950	250
80	46	84	275	37	120	27	90	2900	320
90	70	92	300	41	135	32	105	4200	340
110	85	92	300	44	145	35	115	5700	370

Stripping Shovels

Dipper size		Dumping radius		Dumping height		Width over crawlers		Machine	
yd^3	m^3	m	ft	m	ft	m	ft	mass, t	Ballast, t
40	31	38	126	27	90	13.5	44	1300	350
60	46	43	140	30	100	14.5	48	2000	600
80	62	59	160	37	120	17.5	58	3500	870
90	70	55	180	43	140	18.5	60	4100	950
100	77	60	195	44	145	20	65	4900	1050
120	92	63	205	46	150	23	75	7200	1350
180	140	66	215	46	150	27	88	11000	1650

Conversion factor: 1 ton = 0.9072 t.

Table 13.3.16. Theoretical Cycle Times for Stripping Machines (sec)

Dipper or bucket size		Draglines				Shovels		
yd^3	m^3	90°	120°	150°	180°	90°	120°	150°
Up to 19	Up to 15	55	62	69	77	51	57	63
20–34	16–26	56	63	70	78	52	58	64
35–59	27–44	57	64	71	79	53	59	66
60–74	45–57	59	65	72	80	54	61	67
175–120	58–92	60	66	73	81	55	62	68
120–200	93–150	62	69	76	84	57	63	70

Table 13.3.17. Trailing Cable Prices for Stripping Machines ($/yd³)*

Dipper or bucket capacity		Voltage, kV		
yd^3	m^3	4–7	13	25
Up to 19	Up to 15	44	48	—
20–34	16–26	76	80	—
35–59	27–44	156	120	—
60–74	45–57	—	192	112
75–120	58–92	—	280	200
120–200	93–120	—	360	232

*Based on a copper price of $2900/t.
Conversion factor: $\$1/yd^3 = \$1.31/m^3$.

Table 13.3.18. Correction Factor for Different Overburden Materials

Overburden conditions	M
Light blasting	0.9
Medium blasting	1.0
Heavy blasting	1.1
Bad fragmentation	1.2

13.3.4.5 Selection of Buckets and Dippers

Abrasion and wear of buckets and dippers cannot be eliminated, but liner plates, shrouds, etc., can be used to obtain longer life. The increased weight involved, however, means that longer life can only be provided by a reduction in capacity. By the correct location of liner plates, etc., and the careful selection of the materials used, wear can be economically counteracted.

For walking draglines, lightweight buckets can provide maximum capacity, but the actual drag pull per cubic yard (cubic meter) excavated is reduced; thus they can only be used for relatively light, easily loaded materials. Their selection should be restricted to conditions where wear life and maintenance costs are not critical factors. If a spare bucket is available, or if maintenance can be scheduled to avoid downtime, a lightweight bucket may be economic.

Medium-weight buckets are generally selected for heavier rocks and have the widest application.

For severe duties, or operations where it is uneconomic to hold a spare bucket (e.g., small mines or short-life operations

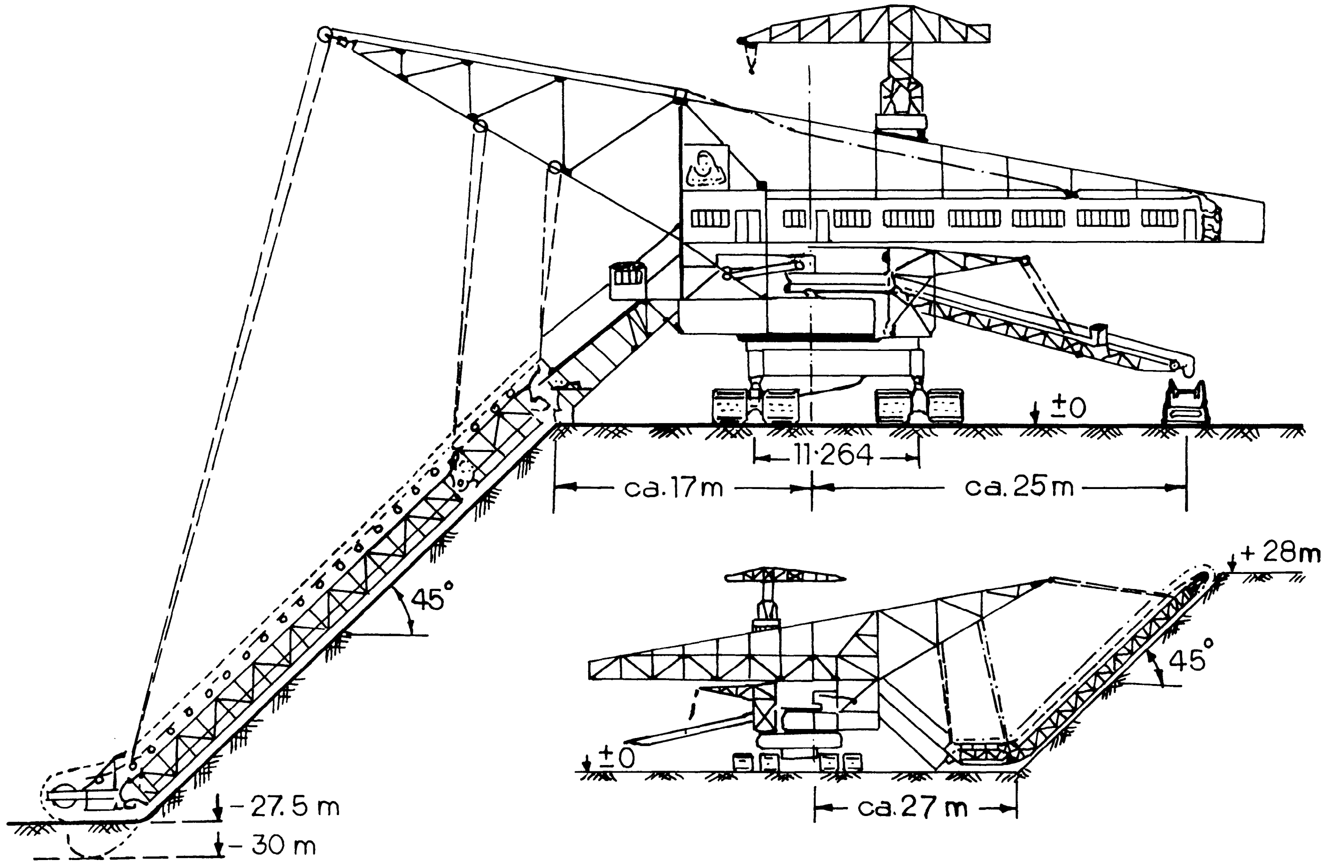


Fig. 13.3.11. Bucket chain excavator (*courtesy:* Friedrick Krupp GmbH Maschinen und Stahlbau Rheinhausen). Conversion factor: 1 ft = 0.3048 m.

which require at least 14,000 hr of operation before major repairs are carried out), heavy-duty buckets of lower capacity are needed.

Bucket capacity is primarily dependent on the difference between the maximum allowable suspended load of the dragline and the mass of the bucket plus rigging. Thus, for abrasive conditions, where an oversize drag chain is used to combat wear, the bucket capacity would have to be reduced.

13.3.5 CONTINUOUS EXCAVATORS

The two most important types of continuous excavator—the bucket chain excavator and the bucket wheel excavator—were developed mainly in the German brown coal fields. Some development has also taken place in Czechoslovakia, the United States, Australia, and the USSR.

13.3.5.1 Bucket Chain Excavator (BCE)

Earlier machines were track-mounted, but this involved frequent track shifting due to the inherently narrow width excavated at each pass of the excavator, and modern machines are invariably of the slewing type mounted on crawlers, performing block excavating operations. Fig. 13.3.11 shows a large BCE which is in service in an Italian mine; it has a theoretical output of 1920 yd^3 (1470 m^3) bank/hr, a machine mass (service weight) of 1640 tons (1495 t), and installed ac power of 3060 hp (2285 kW).

The BCE can mine high outputs in weak unconsolidated ground, but it cannot dig hard ground and is not selective in operation. Its most advantageous feature is its excellent downward-digging capability, combined with reasonable upward-digging capability.

13.3.5.2 Bucket Wheel Excavator (BWE)

The BWE is the most effective machine for mining large outputs in weak unconsolidated ground, although its application is extending into harder formations. It can selectively mine faulted or intercalated ground. It is possible to accurately cut bands as thin as 4 in. (100 mm), but with much reduced output. It has poor downward-digging capability, but this can be improved by a boom sandwich belt, or a special long-boom design which reduces the boom angle sufficiently to allow successful conveyor operation when digging below grade. The sandwich belt does not give satisfactory service when boulders or large frozen lumps are being handled.

The BWE is fitted with evenly spaced buckets on the periphery of the wheel. Excavated material is fed via a transfer point inside the wheel (e.g., a plow, rotating disc, etc.) to the belt conveyor system of the excavator for discharge. Machines are in service with theoretical outputs of up to 13,000 yd^3 (10,000 m^3)

Table 13.3.19. Approximate Power Consumption

Machine	Power consumption kWh/m^3	kWh/yd^3
Loading Shovel	0.45–0.71	0.35–0.55
Stripping Shovel	0.52–0.91	0.40–0.70
Walking Dragline	0.88–1.21	0.68–0.93
Bucket chain excavator	0.41–0.60	0.32–0.46
Bucket wheel excavator	0.30–0.50	0.23–0.38

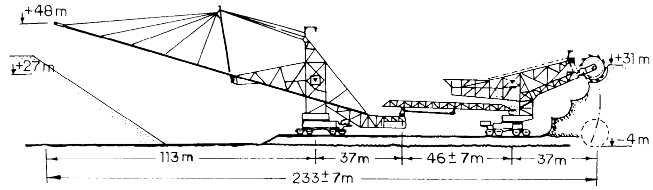


Fig. 13.3.12. Mobile boom stacker for BWE (*courtesy:* Friedrick Krupp GmbH Maschinen und Stahlbau Rheinhausen). Conversion factor: 1 ft = 0.3048 m.

bank/hour and with machine masses (service weights) exceeding 7700 tons (7000 t).

ADVANTAGE OF CONTINUOUS EXCAVATORS. Continuous excavators have lower impact loadings than comparable single-bucket machines. This tends to reduce dynamic stresses, machine mass (service weight), maintenance costs, and power consumption. The slower slewing (swing) speed and reduced digging impacts results in gradual load transfer across the crawlers, with a consequent reduction in ground bearing pressure; operation is therefore possible in conditions in which a single-bucket machine may work only with difficulty.

Land reclamation is generally easier with continuous excavators—many fine examples of reclamation can be seen in Germany and Czechoslovakia.

OUTPUT OF CONTINUOUS EXCAVATORS. Continuous excavators are usually rated in terms of theoretical output:

$$Q_{th} = \frac{60Fs}{\text{Swell factor}} \qquad (13.3.5)$$

where Q_{th} is theoretical output in yd^3/hr (m^3/h) bank, F is capacity of a single bucket, s is number of bucket discharges per minute, and the swell factor is that of the material being excavated. Table 13.3.19 lists approximate power consumption numbers.

Some caution is needed in comparing the theoretical outputs of similar BWEs because manufacturers of different nationality may define F, the bucket capacity, differently. In addition to the actual bucket capacity, some may add (for cell-less buckets) 50% of the associated ring volume or 100% of the associated ring volume, and (for bucket wheels with cells) the volume of the individual cell. It is suggested that the best basis for comparison is to use the bucket volume only. (Note that SI units are generally used as most BWEs and BCEs are of continental manufacture.)

BCE VS. BWE. Since World War II, the BWE has, to a large extent, superseded the BCE. The BCE has continued to be built in eastern Germany where ground conditions are comparatively easy and where designs and operational experience with the BWE were not available. The present trend in eastern Germany, however, is towards the BWE.

In selecting a continuous excavator, the following points favorable to the BWE must be considered.

1. The BWE is selective in operation because of its horizontal action in part-block operation. The BCE cannot selectively mine a deposit.

2. The BWE can, in certain conditions, excavate relatively hard ground (sandstone, shales, etc.) The BCE cannot dig hard ground or handle occasional boulders.

3. Because it has a much greater number of wear points, maintenance costs of the BCE are higher than those of the BWE. Consequently, the availability of the BCE is less than that of the BWE when working in the same conditions.

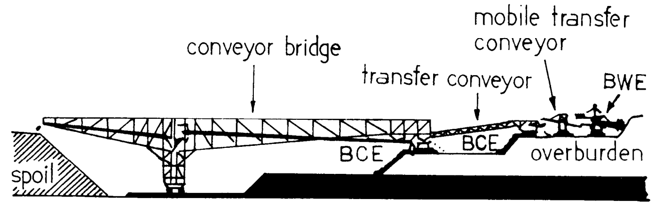


Fig. 13.3.13. Overburden bridge for BWE.

4. The BWE has excellent high-digging capability, whereas the BCE is not as efficient owing to bucket-filling difficulties in high digging. The BCE has excellent deep-digging capability, and the BWE has relatively poor deep-digging capability.

5. The bucket-clearing action of the BWE is superior to that of the BCE, and it has less difficulty in handling sticky materials.

The following conditions favor the BCE, provided boulders or hard bands are not present and selective mining is not necessary:

1. Soft, nonabrasive rocks where high and deep cuts must be performed by one machine.

2. Situations in which the initial box cut is opened in these conditions.

3. Wet pit operations.

4. Where it is necessary to reduce transport gradients.

5. Overburden bridge operations where cuts must be taken below grade level of the bridge crawlers (Strzodka and Piatkowiak, 1969).

6. Where exact slope profiles are required.

7. Where there are large undulations in the surface of the mineral bed.

Most of these advantages are due to the better downward-digging capability of the BCE. Most other applications, however, favor the BWE.

OPERATIONS. Continuous excavators are generally limited to the following gradients: excavating 1 in 20, traveling 1 in 10.

The BWE can be used in thin mineral/thin overburden strip mining casting and can strip deeper overburden or provide a greater pit width than single-bucket stripping machines. The BWEs of the River King and Northern Illinois mines of Peabody Coal Co. in the United States are typical examples of this form of machine.

To obtain the advantage of the shortest possible lift and transport distance and lowest stripping costs in thicker overburden and/or thicker mineral, the direct across-the-pit range of the BWE and the BCE can be extended by use of (1) the mobile stacker boom (Fig. 13.3.12) and (2) the overburden bridge (Fig. 13.3.13). Because of their limited operational flexibility, these units impose a strict discipline on planning, and the following conditions should exist: (1) sufficient proved reserves to jus-

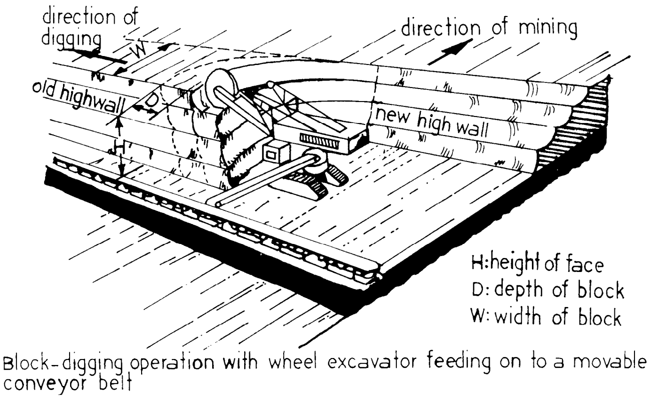


Fig. 13.3.14. Full-block operation, bucket wheel excavator.

tify the capital cost of the relatively expensive machine complex, (2) relatively horizontal stratification over a wide area of the deposit, and (3) uniform geologic conditions (i.e., absence of major faulting, severe undulations, large variations in overburden thickness, etc.).

The most important application of continuous excavators is in weak sedimentary deposits of relatively great thickness where direct transport of the overburden across the pit is impracticable because of excessive pit width. Most of these operations employ several benches, and overburden must be transported round the pit by a conveyor or rail transport system and dumped in the spoil space provided by the advancing pit.

The full-block method with BWEs (Fig. 13.3.14) is normally adopted where selective mining is not necessary. The part block method has lower volumetric efficiency since the thickness of the ground being mined is usually less than the optimum, it involves greater traveling time with increased crawler wear and reduced digging time, and is therefore restricted to selective mining operations. Short booms are not suitable for part block operations. If part-block operation is proposed at a horizon high up the bench face, an oversize bucket wheel is necessary to obtain clearance for the boom if a reasonable block width is to be maintained.

The mass (service weight) of BWEs fitted with booms having crowd action is greater than equivalent BWEs with rigid booms. For large BWEs cutting large bench heights, a crowd action boom requires a long crowd travel motion, which can introduce ground bearing pressure problems.

TRANSPORT SYSTEMS. Clearly, considerable planning is essential before the selection of a continuous excavator can be made. As with any continuous system, the excavators and their transport systems must be engineered initially as a complete system.

The two main systems used with continuous excavators for round-the-pit transport are (1) high-speed belt conveyors and (2) locomotive rail haulage.

The side-slewable, high-speed belt conveyor is gaining preference over rail transport because operation on steep slopes (1:3) is possible, large areas used for wide curves, inclines, sidings, etc., by rail track, especially in deep pits, being eliminated; and the time spent by the excavator in waiting for the transport system is greatly reduced, that is, the whole system becomes truly continuous.

The main disadvantages of high-speed conveyors are their high capital cost and high power consumption.

Rail transport can still prove economic where long distances on relatively horizontal grades have to be covered, for example, from the benches to treatment plant, pit to outside spoil bank, etc.

As each block width (Fig. 13.3.14) is mined, the transport system must be moved forward. To reduce the frequency of this operation and hence the production lost, a mobile, crawler-mounted, transfer conveyor can be used between the excavator and the transport system to increase the range of the excavator.

MECHANICAL DESIGN FEATURES. *Bucket Wheel Design*—Bucket wheels can be of the following construction: (1) with cells, that is, with individual chutes from the buckets to the transfer point supplying the boom conveyor; (2) with half cells; or (3) without cells.

Earlier bucket wheel designs were fitted with cells. They have low maintenance costs and power consumption, but are more suited to granular materials (sand, gravel, etc.), excavated material tending to compact within the cells if high wheel speeds are used or if the material tends to be sticky. Some difficulty is also experienced in clearing the cells when deep-digging with BWEs with reversed buckets. Wheel peripheral speeds between 200 and 400 fpm (1.0 and 2.0 m/s) are used.

The cell-less wheel was first developed in the United States in 1943 for the Kobe excavator. It has since been intensively developed elsewhere, particularly in Germany. It handles sticky materials better than the wheel with cells and provides better bucket clearance when down-digging. Wheel speeds up to 1000 ft/min (5 m/s) are used.

The half-cell wheel has found application in large machines without compaction occurring and with negligible spillage. Chain-backed buckets tend to flex in operation and are superior to solid buckets for handling sticky materials. At one bauxite operation with very sticky overburden material, flow has been improved in those parts of the buckets and wheel not subject to scouring action by the fitting of magnesium liner plates (Sluyeter and Bennett, 1970). Linatex rubber and similar lining have also proven successful in very sticky materials.

There are numerous bucket teeth configurations, and some trial and error may be necessary before a satisfactory arrangement is found.

In strong, highly abrasive material, tungsten-carbide inserts with welded, hardfacing of the parent metal may be necessary (Rasper and Rittner, 1961).

Bucket Wheel Drives—In variable ground it is necessary on occasions to adjust the wheel speed (and slewing speed) to meet the digging requirements and to avoid excessive vibration in hard ground. This may be achieved by the adaptions to the power transmission system.

Crawlers—As continuous excavators usually operate in less consolidated formations, it is often necessary to limit the ground bearing pressure to a low value, and rigid crawlers such as those used on shovels should only be used for very small machines up to about 55 tons (50 t) machine mass (Fig. 13.3.15 (a)). With this form of construction, it is economically possible to limit the average ground bearing pressure to 8.5 psi (60 kPa).

Up to 165 tons (150 t) machine mass, crawler tracks with two wheel bogies should be used so that the crawlers adapt at least partially to uneven terrain (Fig. 13.3.15 (b)). Larger machines generally require the type of crawler design shown in Fig. 13.3.15 (c).

The larger the excavator and the crawler bearing area, the greater may be the imposed ground bearing pressure since a large pressure cone is developed beneath the crawlers and average ground bearing pressures between 21 and 23 psi (150 and 160 kPa) can be used. Large spreaders with the crawler arrangement in Fig. 13.3.15 (c) working on unconsolidated spoil in the German Rhineland impose bearing pressures as high as 17 psi (120 kPa). With crawlers of this construction, tests show that the

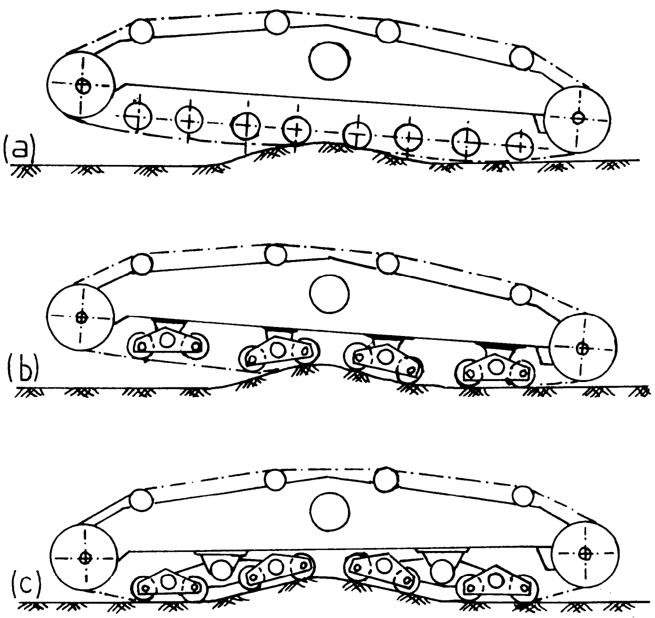


Fig. 13.3.15. Continuous excavator crawlers (courtesy: Friedrick Krupp GmbH Maschinen und Stahlbau Rheinhausen).

maximum ground bearing pressure is about 1.4 times the average figure. For the rigid construction shown in Fig. 13.3.15 (a), the maximum figure can be up to three times the average figure.

Steerable crawler track designs are available and should be selected for large machines to increase their maneuverability.

Specific Cutting Force—The bucket wheel drive power, the mechanical strength of the bucket, and machine mass (service weight) of the BWE depend on the digging resistance of the material being excavated. Investigations in Germany, Czechoslovakia, and the USSR have so far failed to produce a uniform quantitative approach to digging resistance. The relationships between digging resistance, intact rock strength, jointing, bedding, tooth shape and sharpness, angle of attack, and wedge angle of the bucket cutting lip have yet to be determined.

A specific cutting force *k* (expressed in kp/cm of bucket cutting lip edge) is widely adopted. For machine design, however, most engineers prefer to use the "specific digging force" (expressed in kp/cm^2 of bucket slice area). The values given in Table 13.3.20 have been suggested.

There is still much to be done in this field in relating conventional rock properties to specific digging force. Careful investigation is essential. If an insufficiently robust machine is designed, structural and drive failures will occur, but overdesign can, of course, be costly. The large capital investments required for continuous excavators warrant detailed study of similar operations and comprehensive discussions with manufacturers before the limits of specific digging force are specified.

Hard Ground Operation—Large, modern bucket wheel excavators with high wheel speeds and slower slewing speeds can mine materials that shovels cannot mine without blasting (e.g., hard clays, phosphates, sandstones, frozen oil sands). This is only possible with machines of sufficient mass to transfer the cutting forces through the crawlers to the ground. Provided that the rock does not contain large boulders that will not pass through the bucket wheel, ground with a uniaxial compressive strength of 2150 to 2550 psi (15 to 18 MPa) can be excavated without ground preparation. Fig. 13.3.16 shows a modern com-

Table 13.3.20. Specific Cutting and Digging Forces for BWEs

Material	kp/cm cutting lip	kp/cm^2 slice area	Reference
Compact sands	33–103		(Habermaas, 1969)
Chalk	26–113		
Clay (soft)	9–13		
Clay (hard)	40–142		
Sandstone	14–160		
Slate clay	200+		
Sand and gravel	20		(Himmel, 1963)
Sandy clay	29		
Sand and loam	33		
Greasy clay	56		
Argillaceous shale	156	23.8	(Krumrey, 1965)
Compact clay	86	7.82	
10% Sand + 90% clay	86	5.91	
Sandy clay and loam	72	5.67	
Light ground		1.8–2.5	(Dombrowsky, 1964)
Medium ground		3.0–3.5	
Heavy ground		7.0–18	
Light ground-soft loose soil of low cohesion that can be parted with a hand shovel, e.g., unconsolidated sand or fine gravel	15–50	2–5	(Krumrey, 1970)
Medium ground-soil of medium to strong cohesion which can be parted with a spade, e.g., sandy clay, soft and plastic clay, coarse gravel, loam, loess	30–100	5–8	
Heavy ground-strong to very strong cohesion which must be parted with a pick, e.g., compacted clay, sandstone	60–120	8–10	

Fig. 13.3.16. BWE modern compact machine working in difficult overburden (*courtesy:* O&K Orenstein & Koppel, A.G., West Germany).

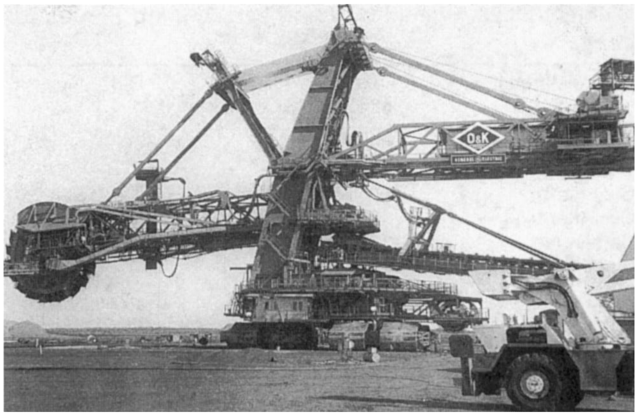


Fig. 13.3.17. BWE working in Australia excavating preblasted overburden (*courtesy:* O&K Orenstein & Koppel, A.G., West Germany).

pact BWE working in difficult overburden. Fig. 13.3.17 shows a large BWE working in Australia excavating preblasted overburden.

The production rate of the BWE operating in hard ground is usually much less than the theoretical output because of loading difficulties and flow difficulties through the bucket wheel. Some attempts have been made to excavate stronger rocks subjected to blasting, but with mixed success. Unless exceptionally uniform fragmentation is achieved, considerable difficulty is experienced owing to the breaking out from the face and spilling of large lumps.

13.3.5.3 Size Selection

MACHINE OUTPUT. The theoretical output of a continuous excavator provides a convenient means of comparing similar machines, but the actual production rate may vary considerably from the theoretical. The bucket filling can vary from 0.4 for difficult conditions (e.g., hard or sticky ground, ground containing boulders, etc.) to 1.32 for free-flowing granular materials, such as sands and gravels, when down-digging with a BCE. Some care must be exercised to use these values correctly since methods of calculating theoretical outputs vary. Most mine planners usually prefer to use a "field factor" that allows for (1) bucket filling, (2) planned maintenance stoppages, (3) stoppages due to breakdowns, (4) conveyor or rail track shifting, and (5) nonworking days.

Because of the large capital investments involved, continuous excavators are usually operated on a three-shift basis. In Germany and elsewhere, statistics show that a daily scheduled operating time of 19.2 hr is normally attained if the excavator is correctly matched to operating conditions. If Sundays and holidays are available for maintenance (including annual overhaul), 5000 scheduled hr/yr is the average operating time. In difficult operating conditions (e.g., hard, abrasive or sticky ground, severe climate, etc.), this will be reduced.

The annual production is then

$$Q/\text{yr} = Q_{th} \times 5000 \times \text{service factor, yd}^3/\text{yr (m}^3/\text{yr)} \qquad (13.3.6)$$

The service factor covers periods when the excavator is not working to full capacity because of poor bucket-filling, maneuvering, transport stoppages, etc. This service factor can vary from 0.5 to 0.8, depending on ground conditions, management efficiency, climate, etc., and may be estimated from plant records.

For initial planning, however, it is usual to base calculations on a field factor since the scheduled hours, management efficiency, climate, ground conditions, etc., are not independent of one another.

The annual production is then

$$Q/\text{yr} = Q_{th} \times \text{days/yr} \times 24 \times \text{field factor} \qquad (13.3.7)$$

where Q_{th} is based on the volume of the bucket only, and the cell or ring volumes are not included.

If studies from similar operations are not available, the values in Tables 13.3.21 and 13.3.22 may be used, where

$$\text{Field factor} = OE \times JC \qquad (13.3.8)$$

Table 13.3.21. Operating Efficiency (*OE*) for Continuous Excavators

Ground conditions	Management conditions			
	Excellent	Good	Fair	Poor
Light	0.70	0.63	0.55	0.47
Medium—heavy	0.65	0.58	0.50	0.42
Heavy	0.57	0.50	0.43	0.36
Hard ground	0.42	0.37	0.32	0.27

Table 13.3.22. Job Conditions Factor (*JC*) for Continuous Excavators

Excellent	0.92
Good	0.83
Fair	0.73
Poor	0.62

MACHINE GEOMETRY. The machine geometry of continuous excavators is determined by the pit layout. The overall slope angles involved are usually not steep because (1) the rocks involved are relatively weak, and (2) the deposits mined may be relatively flat, advance being lateral rather than vertical, so that advance stripping is not quite so economically undesirable as for a deposit of mainly vertical aspect.

Because of this, the benches are usually wide, and space restrictions are not often a major problem. If an excavator must perform upward and downward digging, it requires adequate reach to work on both sides of the transport system. In a rotating pit operation the excavator must be able to work up to the pivot point, where space is restricted, however; whereas in a parallel advancing pit, where end-bench, lift conveyors may be used, the excavator must be capable of digging up to the corner of the pit. The machine geometry may impose some limitations on pit layout, and considerable planning is required before the final machine geometry can be decided. The solutions to these problems are best achieved by plotting pit plans and sections. In some cases, it may be necessary to use a mobile, crawler-mounted, transfer conveyor; for large machines, a telescopic discharge boom conveyor may have to be incorporated into the excavator design.

Table 13.3.23. Constant for Calculating Erection Costs of Continuous Excavators

Machine Mass, tons	Constant a[1]
Below 1000	880
1000–2000	720
2000–3000	600
3000–4000	520
Above 4000[2]	432

[1]Erection cost = aM ($)
[2]Some discretion is needed with very large machines as increased erection costs may be necessary because of increased site fabrication due to the size of component parts, but a reduction in works cost may be expected.
Conversion factor: 1 ton = 0.9072 t.

Table 13.3.24. Trailing Cable Prices ($/yd) for Continuous Excavators*

Theoretical output, m^3/h	Voltage, kV		
	4–7	13	25
0–1000	32	40	—
1000–2000	60	64	—
2000–3000	84	84	—
3000–5000	108	104	108
5000–7500	—	—	132

* Based on a copper price of $2900/t, 1989 projected prices.
Conversion factors: 1 yd^3/hr = 0.7646 m^3/h, $1/yd = $1.09/m.

Table 13.3.25. Maintenance and Supply Cost Factor

Material	Factor
Light ground (free-flowing)	0.01
Medium ground	0.02
Heavy ground	0.03
Hard or frozen ground	0.05
Ground containing boulders	0.04–0.05
Abrasive sandstones, shales, etc.	0.05

13.3.5.4 Cost Estimation

OWNERSHIP COSTS. Continuous excavators can have service lives in excess of 30 years, and machines commissioned in 1935 are still in daily operation in eastern Germany; but tax concessions, company depreciation policies, etc., can influence the write-off period. It is difficult to compare single-bucket excavators with continuous excavators because they usually perform different duties, but for the same (production rate × reach), continuous machines are usually more expensive in capital cost than single-bucket machines. Because of the large capital sums involved and the irrevocable nature of the investment decision, comprehensive investigations and detailed discussions with manufacturers are essential before the selection of a continuous excavator can be finalized. As they also form part of a continuous system, which in a multi-bench mine, must be synchronized with other excavator-transport systems, detailed planning of mine operations is imperative.

Again ownership and operating costs can be estimated using the method for loading shovels (see 13.3.2.2). Tables 13.3.23 through 13.3.25 supply additional information.

13.3.6 MOBILE EQUIPMENT

Mobile earthmoving equipment originally developed for the construction industry has found considerable application in modern mining operations. The major types of equipment are front-end loaders (FEL), tractor-scrapers, bulldozers, rippers, etc. In general, their use has been limited to soils and the weaker, less consolidated rocks, but the introduction of inexpensive explosives has greatly extended their range, and there is some overlap between their application and that of conventional loading equipment.

13.3.6.1 Crawler vs. Rubber-tired Mounting

Crawler-mounted units have the following features:

1. They possess strong digging capability as a front-end loader or as a tractor scraper.
2. Ground bearing pressure is relatively low, and they have excellent tractive effort in bad ground conditions.
3. Performance on steep inclines is good.
4. The degree of maneuverability is high.
5. Travel speeds are relatively slow, and hence they can only be economically used where the distances between digging and discharge points are short.
6. Maintenance costs in abrasive materials are relatively high.

Rubber-tired units have the following characteristics:

1. They are very mobile.
2. Maintenance costs in easy to medium conditions are relatively low.
3. Tire costs are high where sharp, broken, or abrasive rocks are encountered (e.g., basalt, taconite, angularly broken schists, etc.).
4. They are less maneuverable than crawler-mounted units, but center-articulated units are still highly maneuverable in comparison with conventional excavating and loading equipment. They are well suited for operation on narrow benches. Four-wheel-drive units have reasonable maneuverability whereas two-wheel drive units have poor maneuverability.
5. Travel speeds are relatively high; they can therefore, economically dig and transport excavated material over longer distances than crawler-mounted machines. This feature has been responsible for the development of rubber-tired units as load-haul-dump machines, e.g., FELs and tractor-scrapers.
6. Ground bearing pressures are relatively high. The machines are not efficient in bad conditions where rolling resistance is high.

TIRE SELECTION. Considerable research has been undertaken by tire manufacturers, particularly in the United States, and a very wide range of tire types is available. In view of the importance of tire maintenance and replacement costs, it is surprising that many mine operators fail to keep accurate records of or make detailed investigations into tire life. Undoubtedly, the best selection and costing procedures are based on past records. Factors to be considered are surface condition, loading, speeds, curves, grades, temperature, maintenance standards, etc.

Table 13.3.26 shows the average life of tires used in their correct applications. Recapped tires have about 80% of the life of new tires, and usually three recaps are possible during the life of one tire.

13.3.6.2 Front-end Loaders

In recent decades, rubber-tired front-end loader applications have greatly expanded in open pit mining. Units with rock buckets up to 35 yd^3 (27 m^3) are in service, but in metal mines, the

Table 13.3.26. Tire Life

	Job Conditions			
	A	B	C	D
Average life, operating hours	4000–5000	3000–3500	2000–2500	400–1500

A: Good haul roads with operation over soft, non-abrasive rocks.
B: Clays, marls, soft shales, wet gravels, etc., with average haul roads.
C: Medium hard rocks, e.g., hematite, abrasive shales, etc., with short transport distances in average bench conditions.
D: Severe conditions, taconite, angularly fragmented basalt, etc., with poor bench surfaces.

Table 13.3.27. Front-end Loaders

Bucket capacity		Fixed time, sec			
yd^3	m^3	E	M	M–H	H*
5	4.0	32	33	41	—
6	4.5	33	34	42	—
7	5.5	33	35	44	—
10	7.5	37	39	51	—
12	9.0	39	42	56	—
15	11.5	41	44	60	—

* The application of FELs in "hard" digging conditions is marginal and requires comprehensive investigation.

FEL has been mainly restricted to weak overburden stripping and auxiliary duties. Compared with loading shovels, they are short-life machines not greatly affected by obsolescence. Many observers forecast increased acceptance of FELS in face loading, and larger machines capable of handling rock will undoubtedly be developed to take advantage of the following features:

1. Mobility—the FEL can quickly travel from one part of a pit to another, move out of blasting zones, etc.
2. Comparatively low capital costs—the correct selection procedure is to compare different machines by using a discounting method, but the FEL becomes attractive in situations of uncertainty (e.g., political, economic) where capital expenditure must be restricted.
3. Operational flexibility—FEL can perform many auxiliary operations (e.g., haul road construction and maintenance, stockpile and blending operations, drainage and bunding operations, ripping, truck "boosting" on inclines, etc.).

PRODUCTION RATE. Because of its mobility, the most important factor in determining the production rate of a wheel loader is the operating load (bucket load) rather than bucket capacity, since if a FEL operates at maximum load when handling a light material, it will be underpowered and unstable if transferred to handling a dense material. Bucket capacity has been defined by the Society of Automotive Engineers (Anon., 1967). In determining bucket capacity, the bucket load being known, the struck volume of the bucket should be used as most materials do not heap during the loading process. For granular materials (e.g., sand, gravel), a fillability of unity is normal, but for other materials, the fillability factors in Table 13.3.5 should be employed.

The bucket capacity B_c is

$$B_c = \frac{\text{Operating load}}{\text{Loose density} \times \text{fillability}} \tag{13.3.9}$$

The operating cycle is made up of (1) loading, (2) hauling, (3) dumping, and (4) return.

The loading and dumping times are combined as the *fixed time* and hauling and return as the *variable time.*

The fixed time may be based on time studies or, in the absence of experience, values may be taken from Table 13.3.27 for varying digging conditions. The variable time may be determined from time studies or from Fig. 13.3.18.

The production rate in tons/hour of a FEL can be calculated from:

$$\text{Production rate} = \text{Operating load} \times C \times A \times O \tag{13.3.10}$$

Operating Load: This is taken from manufacturers' literature

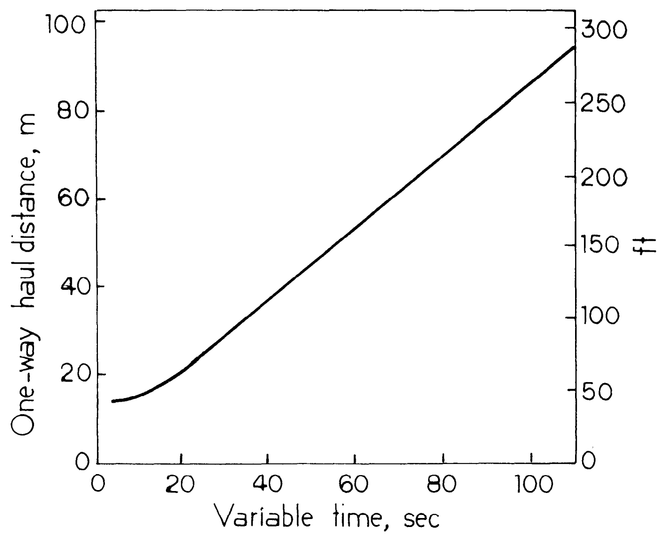


Fig. 13.3.18. Front-end loaders: variable time.

for the machines being considered, the bucket capacity being determined from the operating load.

C: Theoretical cycles per hour.

$$C = \frac{60}{\text{Fixed time (min)} + \text{variable time (min)}} \tag{13.3.11}$$

A: Availability. The mechanical availability during scheduled hours may be obtained by industrial engineering methods from plant records.

O: Job operational factor. This may also be obtained from plant records.

OA: Operating efficiency. Where no previous experience is available to determine *A* and *O,* their product *OA* may be obtained from Table 13.3.4. The job operational factor is very dependent on operator skill, and this factor must be suitably weighted.

COST ESTIMATION. The economic life of a FEL depends on the condition of service, the skill of the operator, and the standards of management supervision. Table 13.3.28 provides a guide to the life in operational hours where no previous experience is available.

If the duty involves long transport distances over good surfaces of nonabrasive materials on relatively flat grades and easy digging conditions, with highly skilled operators, the job condi-

Table 13.3.28. Economic Life of Front-end Loaders (Operating Hours)

Job Conditions	Management Conditions			
	Excellent	Good	Fair	Poor
Excellent	12,000	11,500	11,000	10,000
Good	11,000	10,500	10,000	9,000
Fair	10,500	10,000	9,000	8,000
Poor	9,000	8,500	8,000	7,000

Table 13.3.29. Approximate Fuel Consumption for Rubber-tired Equipment with Diesel Engines

	Operating Conditions		
	Light	Normal	Heavy
Fuel Consumed, Gal/hp-hr	0.02	0.04	0.065

Conversion factor: 1 gal/hp-hr = 1.411×10^{-9} m^3/J.

Table 13.3.30. Crankcase Lubricating Oil Consumption for Mobile Equipment Diesel Engines (gal/hr)

	Operating Conditions		
Engine hp	Light	Normal	Heavy
150	0.1	0.11	0.13
250	0.17	0.18	0.2
350	0.2	0.22	0.27
500	0.3	0.33	0.39
750	0.38	0.42	0.49

Conversion factors: 1 hp = 0.7457 kW, 1 gal/hr = 3.79 L/h.

Table 13.3.31. Filter Costs for Mobile Equipment Diesel Engines ($/hr)

Engine hp	Dusty Conditions	Normal Conditions
150	0.2	0.14
250	0.24	0.18
350	0.50	0.35
500	0.60	0.48
750	0.64	0.50

Conversion factor: 1 hp = 0.7457 kW.

tions are classed as *excellent.* For very short transport distances over abrasive rocks, where dense, badly fragmented material must be dug and the quality of the operating labor is poor, the job conditions are classed as *poor*.

If management and supervision are excellent and maintenance is good, management conditions are classed as *excellent;* conversely, poor management and supervision combined with bad maintenance require the management conditions to be classed as *poor*.

Because of the relatively short life of FEL tires, their cost is deducted from the machine cost and treated as an operating cost item. Otherwise, the format used to calculate ownership and operating costs follow that provided in 13.3.2.2 for shovels. See Tables 13.3.29 through 13.3.31 for additional necessary information.

FRONT-END LOADERS vs. LOADING SHOVELS. Since the introduction of the 10-yd^3 (8-m^3) FEL in 1967, controversy has abounded regarding the relative merits of FELs and loading shovels as primary loading equipment. Briefly restated, the features of both machines are as follows.

FEL Advantages—(1) Excellent mobility—can quickly travel to any part of the pit. (2) Good versatility—can be used for a wide variety of duties, including face loading, cleaning up stockpile operations, haul-road construction and maintenance, etc. This must not be over-emphasized, however, as in one mine in which a large FEL was used, the following applied: primary loading, not sufficiently robust; cleaning up, satisfactory with bucket teeth fitted; difficulties experienced with continuous bucket lip; stockpile duties, excessive degradation of iron pellets when bucket teeth fitted, but satisfactory with continuous bucket lip. Changing duties, therefore, may not simply mean traveling to a different location. (3) Can operate on moderate grades without difficulty. (4) Lower capital cost—has obvious advantages in situations of capital rationing, overseas investments where there is political and economic uncertainty, etc. (5) Not affected by obsolescence. (6) Only one operator needed per machine. (7) Large lumps are not trapped in the bucket, as can happen with a shovel dipper.

FEL Disadvantages—(1) Unsuitable for hard, dense rocks that fragment badly due to open fissures, joints, bedding planes, and other discontinuities. (2) Ground preparation must be extremely thorough. There are, of course, overall savings to be made by increased blasting with most forms of loading machines. (3) Tires require special attention to avoid excessive costs. (4) High operating costs. (5) Greater operator fatigue and proneness to operator abuse. (6) Relatively high ground bearing pressures, poor performance with bad floor conditions.

Shovel Advantages—(1) Well proven in the field of digging hard, dense, badly fragmented rocks. (2) Low operating costs. (3) Less sensitive to poor quality maintenance than wheel loaders. (4) Reduced operator fatigue and less prone to careless operation. (5) Lower ground bearing pressures and not so badly affected by poor floor conditions.

Shovel Disadvantages—(1) Lack of mobility—for multi-bench, sequence operations in open pits with a number of shovels, this need not be a particular disadvantage. (2) Single-purpose loading machine. (3) Higher capital cost. (4) Cannot travel on steep grades. (5) Affected by obsolescence, but many 30-year-old shovels are still performing economically in severe condition. (6) Support equipment may be needed for bench clean up, etc.

Operational Experience—Of about 20 mine operators having experience with both FEL and loading shovels, those handling hard, dense rocks, particularly metallic ores, considered that the FELs presently available are insufficiently robust for regular primary loading duties. Almost all agreed that FELs were indispensable for general duties and emergency primary loading. All were appreciative of their mobility and versatility.

Some operators engaged in mining weaker rocks felt that the FEL had some application, but the majority preferred the loading shovel because of its long-term reliability. Prejudice against the FEL possibly exists as some operators complained of excessive operating costs towards the end of its life when, probably, it should have been scrapped.

Those engaged in mining weaker materials or soft overburden were slightly in favor of FELs. Some who were producing materials for the construction industry felt that they could not plan realistically ahead beyond four years, and they therefore favored the FEL because of its lower capital costs.

CRAWLER-TYPE FRONT-END LOADERS. Recent developments of the front-end loader have resulted in an extension of its use into job applications previously considered suitable only

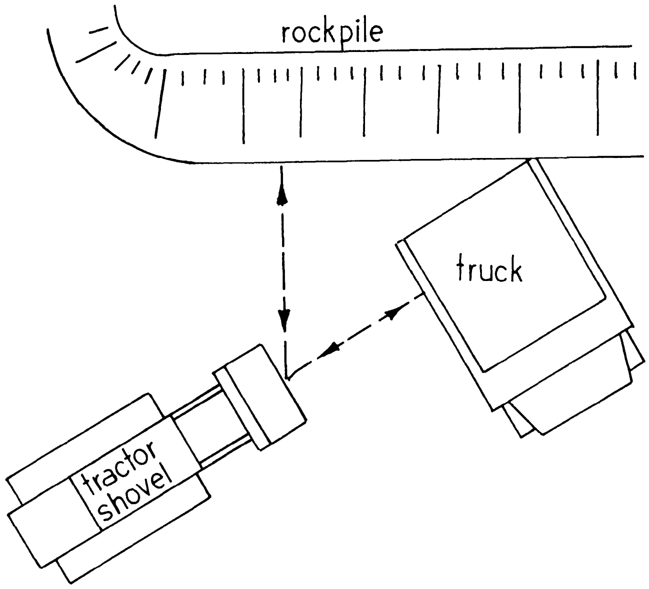


Fig. 13.3.19. V-shaped path: crawler-mounted, front-end loader operation.

for crawler-mounted machines. The main duty of the crawler-type FEL in open pit mining is as a support machine. Although it has not the mobility of the rubber-tired loader, it is much more mobile than the loading shovel, and is particularly useful for excavation of inclines, sumps, etc.

For face loading, the machine follows a V-shaped path with overall cycle times between 0.7 and 1.0 min (Fig. 13.3.19). For excavating inclines, etc., a fixed time of 0.65 min is normal, and an overall haul speed of 2.4 mph (4 km/h) can be used to calculate the variable time.

The cost estimation method used for rubber-tired FELs may be used for calculating ownership costs, except that no deduction is made for tire costs. Fuel and lubricant costs are about 1.2 times those shown in Table 13.3.29. Maintenance repair parts and labor are about 1.1 times greater than those for rubber-tired FELs. There are, of course, no tire costs.

13.3.6.3 Hydraulic Excavators

Initially introduced in the United States as a back-hoe ("hydraulic dragline"), the hydraulic excavator is largely a European development. Line production in the 200- to 240-ton (180- to 220-t) class is currently available from three manufacturers, and 500-ton (450-t) face shovel machines are operative in the United Kingdom. A large machine is currently successfully operating in difficult conditions in a large open pit in Nambia. Development of the 200-ton (180-t) plus machine with its effective working mass and flexible front-end geometry makes the hydraulic excavator a serious competitor to the rope-type shovel in metal mining.

The face shovel machine is usually fitted with an opening dipper bullclam. Because of its flexible geometry, the hydraulic excavator develops high-crowd, prying, and breakout force characteristics. It can mine selectively and follow the contours of irregular beds much better than a rope-type shovel.

In the back-hoe mode, the hydraulic excavator can stand on top of a bench and load a truck spotted on the bench below. Due to the small swing angle in this position, the cycle time is short, resulting in high production. Spotting the bullclam over the truck body is facilitated as the excavator operator looks directly into the truck body.

The back-hoe mode may also be used to load trucks on the same bench when it is difficult to operate trucks on the bench below.

Most hydraulic excavators are diesel powered, but electrical versions with cable reels for the trailing cable are available. The electric drive does not cause the wide fluctuations in system load characteristic of rope-type shovels with solid-state rectification systems, and generally has lower operating costs than diesel drives.

COST ESTIMATION. The usual cost estimating procedure is used with hydraulic excavators (see 13.3.2.2). In calculating ownership costs, the hydraulic excavator is considered a medium-life machine (8 to 10 years), the larger machines tending to have longer life. In areas where maintenance personnel have little previous experience with hydraulic systems, machines may have a shorter life (7 to 8 years) until experience is gained with their operation.

In estimating operating costs, availabilities of 90% or more are generally assured. The operating costs of the electric-drive hydraulic excavator are about 70% of the diesel drive, while the capital costs of the electric drive are about 20% higher than for the diesel drive.

13.3.6.4 Tractor Scrapers

Many medium to strong rocks tend toward blocky fracture after ground preparation and are unsuitable for excavation by tractor scrapers. Their open pit mining applications are mainly confined to overburden stripping, where their productive capacity and mobility are of considerable value, although they are also used to some extent in limestones and other medium rocks that fragment well after blasting or ripping.

Many models are available, some of 90-yd^3 (70-m^3) capacity with 120-yd^3 (92-m^3) designs proposed, but the most popular sizes in mining are in the 20- to 44-yd^3 (15- to 34-m^3) struck capacity range. Although here they are being considered as excavators, they are load-haul-dump machines, and their complete cycle is amenable to investigation by simulation techniques. Of the four main types available, the *single-engine conventional scraper* has the widest range of economic application. Except in very easy, downhill conditions, a pusher tractor is needed for loading. The tractor then boosts the scraper to haul speed. The conventional scraper does not perform well where the duty cycle includes steep adverse grades, high rolling resistances, poor floor conditions, or where the haul distances are short.

The *tandem-powered machine* has an all-wheel drive from front and rear engines, which provides a high power/weight ratio and tractive effort suitable for conditions with high rolling resistances and adverse grades. It can, therefore, work where other scrapers cannot, especially in mud or when dumping in wet conditions on uncompacted spoil heaps. Normally, the tandem-powered scraper operates with the assistance of a pusher tractor, but it can partially self-load or even fully load on downhill runs.

Elevating scrapers are self-loading and are ideally suited for short to medium haul distances where a queue can form while waiting for the pusher tractor. They do not perform well on adverse grades or with high rolling resistances.

The *elevating scraper* cannot efficiently handle sticky materials or material containing rock or boulders larger than 8 in. (200 mm). Because of its ability to work alone, the elevating scraper is ideally suited for selective mining or for small-fleet operations. The action of the elevating flights also breaks down and helps to blend the material being loaded. It is easier to spot over dumping hoppers than other scrapers.

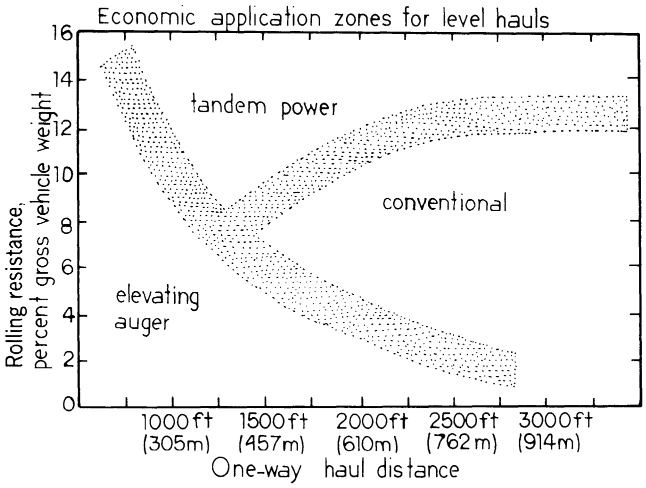


Fig. 13.3.20. Tractor scraper application zones: level haul (courtesy: Caterpillar Tractor Co., Ltd.).

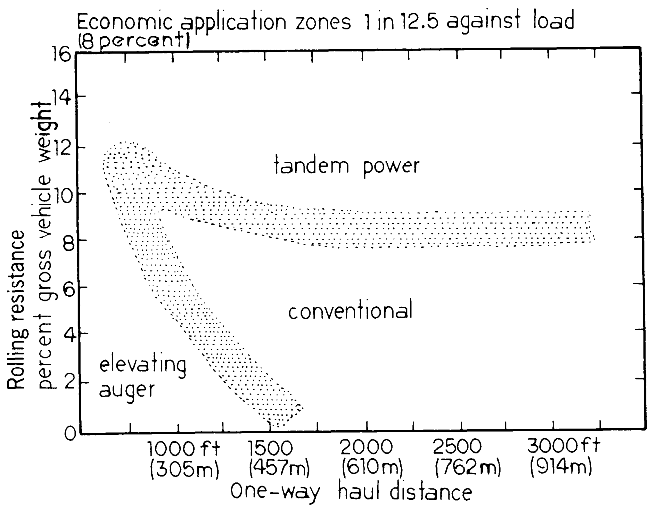


Fig. 13.3.21. Tractor scraper application zone: 1 in 12.5 against load (courtesy: Caterpillar Tractor Co., Ltd.).

The *auger scraper* is a self-loading machine and provides an alternative to conventional, push-pull elevating machines. The auger is placed at the center of the bowl, and as material flows over the scraper's cutting edge, it is fed to the rest of the bowl by the rotating motion of the auger.

The auger scraper requires a shorter cut distance, has excellent ejection characteristics, increased tire life, retains material well when hauling, and is suitable for a wide range of conditions. It cannot efficiently handle material containing large rock fragments or boulders larger than 12 in. (300 mm).

EQUIPMENT SELECTION. There is some overlap in the application of the different types of scraper, but Figs. 13.3.20 through 13.3.22 may be used to determine the economic application. These are based on the results of a computer simulation study by a leading manufacturer (Anon., 1972). Typical values of rolling resistance are shown in Table 13.3.32.

Scraper Cycle Times—The fixed time is made up of decelerating, spotting the scraper by the pusher vehicle, loading, accelerating ("boosting"), dumping, and spreading, plus the variable time,

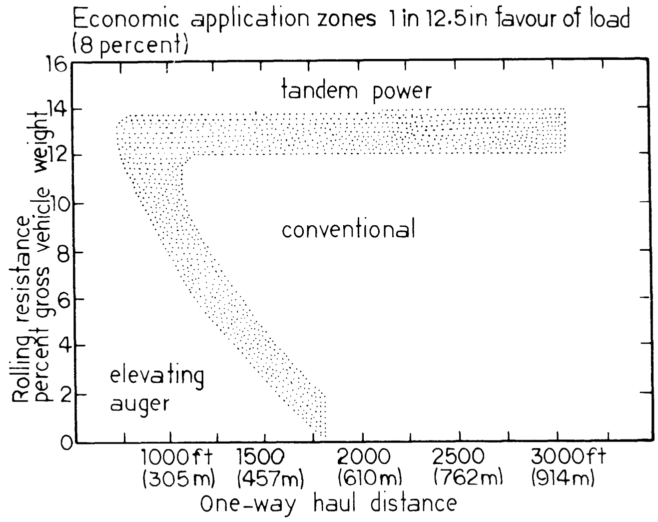


Fig. 13.3.22. Tractor scraper application zone: 1 in 12.5 in favor of load (courtesy: Caterpillar Tractor Co., Ltd.).

Table 13.3.32. Rolling Resistance for Rubber-tired Tractors (as a Percentage of Gross Vehicle Weight)

Description	% Gross vehicle weight
Hard pavement, no penetration (concrete or asphalt)	2
Flexible pavement (macadam or packed gravel construction)	3–3.5
Dirt road, flexing considerably (maintained)	5
Unmaintained dirt road (no compaction)	7.5
Loose sand or gravel	10
Soft muddy rutted road	15–20

Note: Ground preparation for and tractor scraper operations are described elsewhere (Caterpillar, 1970).

Table 13.3.33. Standard Fixed Cycle Time for Tractor Scrapers

Type	Time, min.	
Conventional	1.45	(push-loaded)
Tandem power	1.25	(push-loaded)
Elevating	1.55*	(self-loaded)

* No spotting time included

traveling between cut and dump. The fixed time is best determined by time studies, but in the absence of experience in similar conditions, the values given in Table 13.3.33 may be used.

The variable time is made up of the loaded and empty travel times:

$$\text{Variable time} = \frac{\text{Haul distance}}{\text{Haul speed (loaded)}} + \frac{\text{Haul distance}}{\text{Haul speed (empty)}} \tag{13.3.12}$$

Haul speeds, loaded and empty, are determined by time

Table 13.3.34. Fillability and Swell Factors: Tractor-Scrapers

Digging Conditions	Fillability	Swell Factor
Easy, dry	0.95–1.00	1.15–1.25
Medium, common earth	0.90–0.95	1.2–1.3
Medium-hard, clays,	0.85–0.9	1.35
weak rock	0.85–0.95	1.35–1.45
Hard, wet clay,	0.70–0.80	1.4
well broken rock	0.75–0.80	1.5
Extreme, broken basalt, etc.	0.50	1.5–1.6

studies or from manufacturers' performance charts, the gradient and rolling assistance of the haul road being taken into account. Therefore, the scraper cycle time t_c is

$$t_c = \text{fixed time} \div \text{variable time} \tag{13.3.13}$$

Scraper Production—The load hauled is usually expressed in bank volume and is based on scraper heaped capacity, fillability, and swell factor.

$$\text{Scraper production (bank volume/hr)} = \frac{\text{Scraper heaped capacity} \times \text{Fillability} \times 60 \times AO}{\text{Swell factor} \times t_c} \tag{13.3.14}$$

OA: Operational efficiency. Where no previous experience is available to determine *A* and *O,* their product *OA* can be obtained from Table 13.3.4. If the job conditions are bad owing to excessive rain, ice, and snow; dusty, bad haul roads; severe adverse grades; dense, abrasive, poorly fragmented rock; poor-quality operating labor; etc., then it should be given a *poor* rating. If management and supervision are poor, with bad maintenance, inefficient workshops, poor availability, bad job layout and organization, then it should also be given a *poor* rating. Conversely, with first-class management and very good job conditions *excellent* ratings are used.

Fillability: The fillability of scraper bowls is generally higher than that of other excavators because of the limited range of materials handled. Table 13.3.34 provides an approximate guide.

Swell factor: Because of its mode of loading, the swell factors of scraper-handled material can be different from those for other machines. Table 13.3.34 gives approximate figures.

When dense, broken rock is being loaded, it is essential to ensure that the maximum load of the scraper will not be exceeded.

Pusher Tractors: To avoid queuing of scrapers at the start of each cut, the number of scrapers must be correctly matched to the number of pushers. Because of variations in cycle time, a perfect match is never possible in an actual mining operation, and some waiting time is usually unavoidable.

The maximum number of scrapers served by each pusher tractor should not exceed the following:

$$\text{No. scrapers} = \frac{\text{Scraper cycle time}}{\text{Pusher cycle time}} \tag{13.3.15}$$

where pusher cycle time in min

$$= 1.4 \text{ (scraper load time)} + 0.25 \tag{13.3.16}$$

The pusher cycle time is usually in the range of 1.2 to 2.0 min.

Table 13.3.35. Economic Life: Tractor-Scrapers (Operating Hours)

Job Conditions	Management Conditions			
	Excellent	Good	Fair	Poor
Excellent	13,000	12,500	11,500	10,500
Good	12,500	11,500	10,500	9,500
Fair	11,000	10,500	9,500	8,500
Poor	10,000	9,000	8,500	8,000

Table 13.3.36. Maintenance Factor: Tractor-Scrapers

	Job Conditions			
Scraper Type	Excellent	Good	Fair	Poor
Conventional	0.65	0.80	1.00	1.2
Tandem Power	0.66	0.82	1.02	1.23
Elevating	0.68	0.84	1.05	1.28

In determining the number of pushers required, it is necessary to take the fleet size into account, especially with conventional scrapers, since the added costs of an extra pusher shared between a number of units may be justified by obtaining maximum production. Where haul distances are short, queuing also can become a serious problem, and an adequate number of pushers is essential. The problem with tandem-powered scrapers is not so acute as they can partially self-load.

Push-pull Operations: Pairs of tandem-powered scrapers are used in the push-pull method. The two machines link up through push blocks and a hook and bail assembly and enter the cut; the second scraper push-loads the first, the first then pull-loads the second; they disconnect and drive individually to the dump. Cycle times are slightly greater than for single push-loading, but, of course, pusher-tractor costs are not incurred.

COST ESTIMATION. The economic life of a tractor-scraper depends on job and management conditions. Where no previous records are available, values from Table 13.3.35 may be used as a guide. Table 13.3.36 lists maintenance factors.

Easy digging, long level hauls over good roads of non-abrasive materials with highly skilled operators would rate as *excellent* job conditions, whereas hard-digging, short hauls over steep adverse grades, with bad, rutted, muddy or abrasive road surfaces would rate as *poor* job conditions. Similarly, with good supervision and maintenance, the management conditions are *excellent;* whereas bad supervision, together with inadequate maintenance, would rate as *poor.* Tire costs are normally deducted from the machine cost and treated as an operating cost item.

Follow the procedure of 13.3.2.2 in estimating scraper ownership and operating costs.

13.3.6.5 Bulldozers

Although not usually employed as a primary excavator, the bulldozer has considerable application in surface mining. The crawler-mounted machine is more generally accepted, and 70-ton (63-t) units are at present in operation; but because of its mobility, the rubber-tired bulldozer is increasingly finding application. For short hauls in severe conditions where scraper operations would be difficult, bulldozing can usually be adopted.

Table 13.3.37. Swell Factor: Bulldozed Materials

Material	Swell Factor
Broken rock	1.65
Heavy clay (wet)	1.45
Earth with boulders	1.35
Earth	1.25
Sand and small gravel	1.1

Table 13.3.38. Maintenance Factor: Bulldozers

Type	Job Conditions			
	Excellent	Good	Fair	Poor
Crawler	0.8	0.95	1.1	1.3
Rubber Tired	0.7	0.85	1.0	1.2

EQUIPMENT SELECTION. The blade capacity (loose volume) can be obtained from manufacturers' literature. The swell factor for bulldozed material is not the same as for other forms of excavation, and the figures given in Table 13.3.37 may be used.

The cycle time of a bulldozing operation does not contain any fixed time element. The gears to be adopted for various materials, gradients, etc., and their speeds may be obtained from manufacturers' literature. First-gear dozing speeds of up to 3.6 fps (1.1 m/s) and third-gear reverse speeds up to 9.3 fps (2.8 m/s) are normal.

The cycle time t_c found as follows:

$$t_c = \frac{\text{Haul distance}}{\text{Dozing speed}} + \frac{\text{Haul distance}}{\text{Reverse speed}} \tag{13.3.17}$$

The production is then

$$\text{Bank volume/hr} = \frac{\text{Blade capacity} \times 60 \times AO}{t_c} \tag{13.3.18}$$

AO may be determined from Table 13.3.4 if previous experience is not available.

COST ESTIMATION. Ownership and operating costs for bulldozers may be obtained by use of the format for tractor-scrapers, with the following modifications. The economic life should be taken from Table 13.3.35. Tire costs do not apply to crawler-mounted machines. For maintenance, repair parts, and labor, see Table 13.3.38.

For pusher-tractor operations, multiply the maintenance factor by 0.8 to allow for absence of blade wear.

13.3.7 TRENDS IN EXCAVATION

There are considerable advantages to be derived from continuous excavation methods, but it is difficult to envisage any major breakthrough in the field of hard-rock excavation in the near future. A much better knowledge of rock properties, and particularly rock machining, must be gained before any revolutionary developments can be forecast. At present, no mechanical device can match the huge quantities of energy that can be so compactly stored in chemical form in an explosive charge, and the drill-blast-load cycle is likely to remain the major excavation method for some time in stronger rocks.

ACKNOWLEDGMENT

Acknowledgment is due to the Institution of Mining and Metallurgy, London, for permission to use material appearing in their publication (Atkinson, 1971).

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Chapter 13.4
DESIGN AND LAYOUT OF HAUL ROADS

THOMAS ATKINSON

13.4.1 INTRODUCTION

Over the past three decades, off-highway trucks have been developed for surface mines up to 400 tons (360 t) in capacity, although 200 tons (180 t) is generally the largest size in fleet operation. They represent a huge capital investment and a significant percentage of total costs (Table 13.4.1). If haul road design is inadequate, the trucks can be highly lethal in the confines of a surface mine.

Despite these facts, haul road design until recently has received little attention (Kaufman and Ault, 1977). The two major functions of haul roads are to promote (1) efficient transport and (2) safety. Haul road design factors must ensure:

1. Minimum costs on a net present value basis for the transport of mineral and waste throughout the life of the mine.
2. A minimum of traffic congestion and the maintenance of safe, ready access to the mining operations.
3. The avoidance of areas where slope stability problems could occur.
4. The use of long-life haul roads rather than short-life roads. This reduces haul road overall construction costs and operating costs as well as reducing the demand for haul road construction materials which may not be available in sufficient quantities from the overburden.

Other factors include the locations of mineral preparation plants, stock yards, external waste dumps, environmental constraints, etc. All these factors direct attention to:

1. Haul road layout.
2. Haul road geometry.
3. Haul road construction materials.

13.4.2 TYPICAL HAUL ROAD LAYOUT

13.4.2.1 Strip (Open Cast) Mines

As seen in Fig. 13.4.1, these operations are characterized by the T junctions formed by the intersection of the haul roads (access ramps) through the spoil with the pit. Because the presence of the access ramp eliminates some spoil space (and also involves rehandling of spoil), the spoil heaps at these junctions must be piled higher than elsewhere, often resulting in spoil heap failures due to incorrect dumping procedures. Generally, it is not practical to dump spoil directly to form these junctions; the side slopes are best cut from placed soil. This usually results in more stable slopes, since substructures may be formed within spoil heaps from consolidation due to self-weight; they can be excavated to stable angles steeper than the angle of repose of the spoil, thereby partially reducing the volume of rehandle required.

Table 13.4.1. Typical Truck Haulage Costs for Surface Mines

Operation	% of total costs
Strip	7–15
Bench	20–40*
Open pit	30–50

* These figures represent shovel/truck mines. Costs for bucket wheel excavator/conveyor mines are much less.

Because of the flat aspect of strip mines, the bottom-dump truck is the preferred haulage unit. For gradients greater than 6% (3.5°), however, rear-dump or alternatively unitized trucks may be essential. Careful attention to dips, pavement contours, etc., is therefore needed during layout.

Where sandstones and other competent materials are present in the overburden, there are usually few construction materials problems, but where mudstone, silts, clays, etc., predominate, it may be necessary to import materials for haul road construction. The pavement must be of suitable material for temporary haul roads within the strip. Where it is not, the haul road may be located on the top of the mineral bed and the boom length of the loading shovel (located on the floor) must be increased or a back-hoe excavator used from the top of the mineral bed.

The advent of cast blasting (explosives casting—see Chapter 13.2.3) provides the opportunity for locating the access ramps on the highwall side of the strip (Atkinson, 1983), eliminating the spoil stability problems at the junctions of the access ramps with the strip, as well as reducing interference to the rhythm of the prime stripping operation. To be successful, cast blasting usually requires relatively strong overburden that will stand almost vertically at the highwall. This also implies there will be an adequate supply of material suitable for haul road construction within and outside the overburden on the highwall side.

13.4.2.2 Terrace Mines

Haul roads are required in bucket wheel excavator/conveyor mines for access by service vehicles traveling at modest speeds. Apart from the need for bridges to cross conveyors at key points, haul road requirements are not a major consideration. Generally,

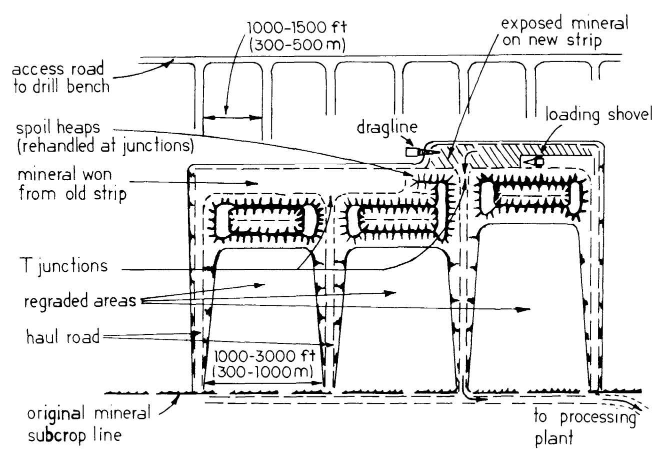


Fig. 13.4.1. Diagrammatic strip mine layout.

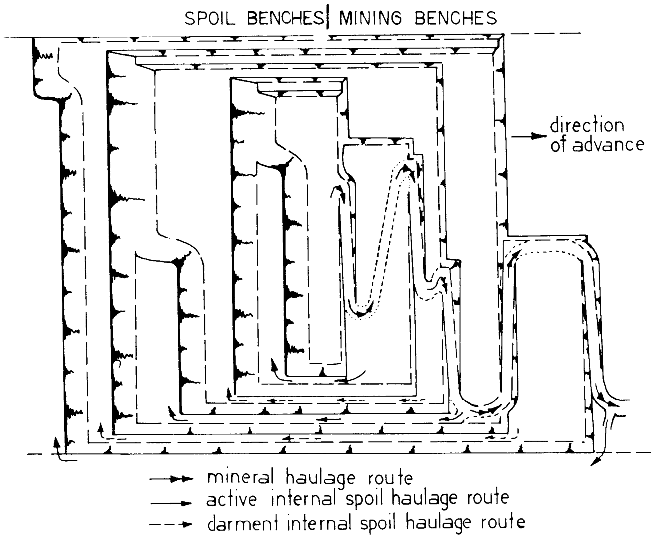


Fig. 13.4.2. Diagrammatic terrace mine layout.

such mines are excavated in relatively unconsolidated formations (e.g., lignite and bauxite mines) and materials suitable for haul road construction may not be available. Volumetric requirements are small, however, and in the case of lignite operations with a captive power station, can often be satisfied by the use of fly ash.

Shovel/truck terrace mines (Fig. 13.4.2) require major haul road construction, mostly of a temporary nature. Stratiform deposits with dips in excess of 14% (8°), where spoil stability problems could occur if down-dip strip mining is adopted, are increasingly being considered for terrace mining. In general, the strata dips are too steep for efficient truck haulage on the full dip, and it is necessary to form haul roads in the overlying strata.

Although haul roads in underlying strata may reduce haul distances for some spoil dumping, they can cause instability by undercutting the footwall bedding planes, as well as increasing stripping volumes (Walton and Atkinson, 1978).

Again, where suitable materials occur in the overburden, haul road construction need not be a problem, but in some geologic formations, a total lack of such materials, except for the mineral itself, may be experienced. This requires the import of large volumes of suitable materials and can have a significant impact on the visibility of such operations.

13.4.2.3 Open Pits

Massive ore bodies, steeply dipping stratiform deposits, and coal accumulations are generally mined by a benched, inverted, conically shaped open pit.

The facilities external to the pit (i.e., mineral preparation plant, waste dumps, etc.) are usually located by topographical, geotechnical, and environmental considerations. These locations may influence the alignment of the access roads to the crest of the pit.

The in-pit access haul road (Fig. 13.4.3) may be a clockwise or a counter-clockwise spiral, or a switchback or zig-zag (where the haul road turns back on itself). The main factors deciding the selection of layout are

1. It may be possible to locate permanent access haul roads on a "tight" side of a pit (e.g., the footwall of a stratiform deposit), in which case a switchback layout is adopted.

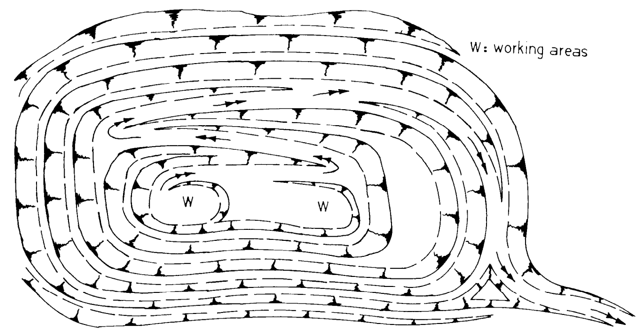


Fig. 13.4.3. Diagrammatic open pit layout with switchback haul road. W = working areas

2. In large pits, the haulage distance may be too great if a spiral layout is used.

3. Areas where potential slope stability hazards exist should be avoided, possibly eliminating a spiral layout.

4. The pit walls may be too steep to allow suitable bends to be formed for the switchback layout without greatly increasing the stripping ratio. In this case, the spiral layout would be adopted. Similarly, branching haul roads may exert an influence on the choice of layout.

Gradients of 8 to 10% (4.5 to 6°) are usually adopted, and rear-dump trucks are the preferred haulage unit, but a 12% grade may be adopted for trolley trucks.

Many metallic deposits are either igneous or metamorphosed, and haul road construction materials rarely present problems. Stratiform deposits, including accumulations, present the same materials problems as strip and terrace mines, but the bedding planes often dip into the pit, and the location of haul roads may cause difficulties.

Ideally, overall pit slopes are decided by geotechnical considerations. Bench heights are set by the operating requirements of the excavator, usually 30 to 50 ft (9 to 15 m). Loose material from minor falls must be contained on safety benches or berms from 30 to 40 ft (9 to 12 m) wide. These safety benches flatten the overall slope and are usually left after every 2 to 4 lifts.

Haul roads also flatten the overall slope (Fig. 13.4.4). In some cases, the geometry imposed by safety benches and haul roads result in overall slopes that are flatter than required by slope stability considerations alone, thereby increasing stripping costs.

The location of the haul road within or outside the ore body can also have a significant effect on mining costs. Fig. 13.4.4a shows diagrammatically the haul road located outside the ore body. The location increases the volume of minable ore but also the stripping ratio. Fig. 13.4.4b shows the haul road located within the ore body. The stripping ratio is reduced, but the volume of minable ore is also reduced in accommodating the haul road. The effects of the location of the haul roads must be fully considered. In general, the problem is site specific and must involve geotechnical as well as economic considerations.

13.4.3 HAUL ROAD GEOMETRY

There are a number of items that are best considered separately.

13.4.3.1 Number of Lanes

In-pit roads are usually constructed for single-lane, uni-directional traffic or two-lane, directional traffic (1) because traffic

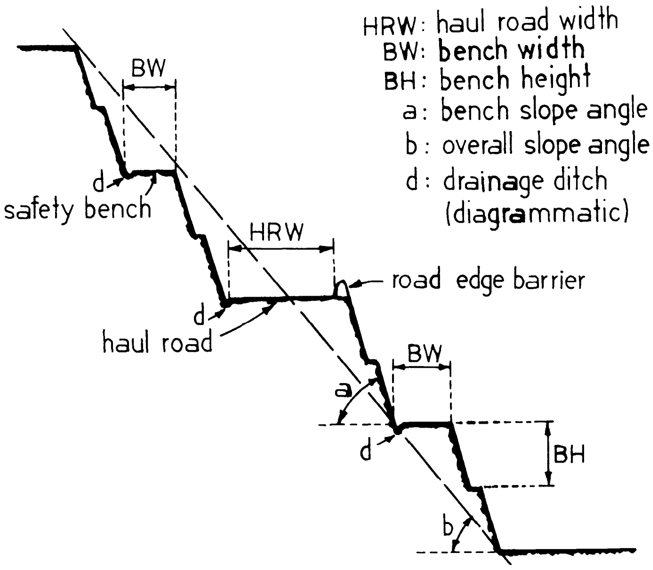


Table 13.4.2. Minimum Haul Road Width

No. of Lanes	Factor × Maximum Vehicle Width
1	2.0
2	3.5
3	5.0
4	6.5

Table 13.4.3. Additional Allowances

Vehicle	% of Table 13.4.2 (Inside Radius)* 20 ft (6 m)	150 ft (45 m)	200 ft (60 m)
Rear-dump and unitized Trucks	125	118	110
Articulated trucks	155	135	115

* Percentages for other radii may be interpolated.

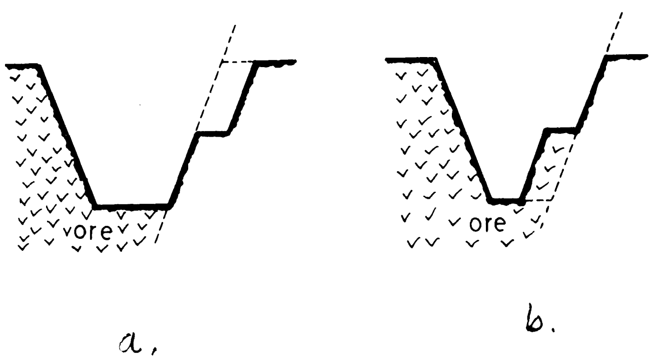


Fig. 13.4.4. Haul road geometric considerations.
a. Haul road outside ore body. b. Haul road in ore body.

density may not be high or (2) because of space problems. Haul roads from the pit to external waste dumps, preparation plants, etc., however, may require more than a single lane per direction. The number of lanes may be determined from the relation,

$$n = \frac{t\, d_b}{550\, v} \qquad (13.4.1)$$

where n is number of lanes for unidirectional travel, v is vehicle speed in mph, t is traffic density in vehicles/hr, and d_b is normal safe distance between trucks in ft. In SI units,

$$n = \frac{t\, d_b}{100\, v} \qquad (13.4.1a)$$

where v is vehicle speed in km/h, and d_b is normal safe distance between trucks in m.

13.4.3.2 Safe Distance Between Trucks

The safe distance between trucks depends upon driver reaction time (usually taken as 2.0s), the gradient, and the road surface plus an allowance (usually 16.5 ft or 5 m). The safe distance can be determined from

$$d_b = \frac{v}{1.08} + \frac{v^2}{91.5\,(C_t \pm i)} \qquad (13.4.2)$$

where C_t is coefficient of traction (less than unity), and i is steepest haul road gradient expressed as a fraction. In SI units, it becomes,

$$d_b = \frac{2.0\, v}{3.6} + \frac{v^2}{254\,(C_t \pm i)} + 5.0 \qquad (13.4.2a)$$

13.4.3.3 Road Width

The widest vehicles proposed determine the haul road width. For straight, regular grade roads, the rules of thumb given in Table 13.4.2 are adequate.

For sharp curves, additional width must be included, both on the curve and the tangent to the curve, to cover the front and rear overhangs of the vehicle and the difficulty of negotiating the curve. Minimum percentages of the figures for straight haul roads are given in Table 13.4.3.

Long tangents to curves assist drivers in negotiating curves.

13.4.3.4 Super Elevation

Trucks negotiating tight curves are subjected to an outward centrifugal force, which is opposed by the side friction between the tires and the road surface. Obviously, a good surfacing material is essential on sharp curves, and super elevation of the road surface is normally included in the haul road design. There are practical limitations to super elevation, since trucks driven at slow speeds on sharp curves could (1) overload the tires on the inside of the curve, and (2) in areas of ice, snow, and heavy rain,

Table 13.4.4a. Super Elevation Rates e Expressed in (in./yd)

Truck Speed (mph)	10	15	20	25	30	>35
Radius (yd)						
5	1.5	1.5	—	—	—	—
10	1.5	1.5	1.5	—	—	—
15	1.5	1.5	1.5	1.8	—	—
25	1.5	1.5	1.5	1.5	2.2	—
30	1.5	1.5	1.5	1.5	1.8	2.2
60	1.5	1.5	1.5	1.5	1.5	1.8
100	1.5	1.5	1.5	1.5	1.5	1.5

Table 13.4.4b. Super Elevation Rates e Expressed in (mm/m)

Truck Speed (km/h)	15	25	35	40	50	>60
Radius (m)						
15	40	40	—	—	—	—
30	40	40	40	—	—	—
50	40	40	40	50	—	—
75	40	40	40	40	60	—
100	40	40	40	40	50	60
200	40	40	40	40	40	50
300	40	40	40	40	40	40

Table 13.4.5a. Maximum Recommended Rates of Super Elevation Change on Tangents (in./yd)

Truck speed, mi/hr	15	20	25	30	<40
Super elevation change/ 100 yd of tangent (in./yd)	9.35	9.00	8.28	7.20	5.76

Table 13.4.5b. Maximum Recommended Rates of Super Elevation Change on Tangents (mm/m)

Truck speed, km/h	5	25	35	40	50	<60
Super elevation change/100 m of tangent (mm/m)	260	260	250	230	200	160

tend to slide towards the inside of the curve. Table 13.4.4a (or 13.4.4b) shows the super elevation/safe truck speed relationships for practical road construction.

Where possible, all the super elevation should be uniformly introduced in the tangent to the curve, the minimum amount being 70% of the total super elevation. Where sharp curves occur at the end of long downhill grades, maximum truck speeds must be restricted to those given in Table 13.4.4 and the maximum permissible travel speed read from Table 13.4.4 to prevent trucks sliding in towards the center of the curve.

Tangent lengths vary with truck speeds and total super elevation. Maximum recommended rates of change of super elevation are shown in Table 13.4.5a or 13.4.5b.

13.4.3.5 Gradients

Maximum gradients may be statutorily limited to between 8 to 15% (5 to 8.5°) for sustained gradients, but in general when considering the economics of uphill haulage, as well as downhill safety, the optimum gradient for most situations is about 8%

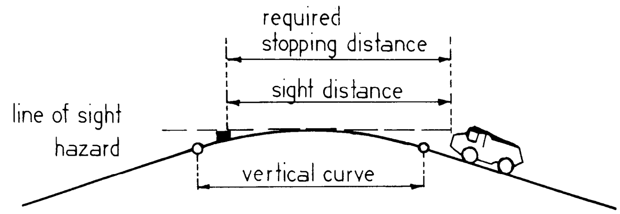


Fig. 13.4.5a. Vertical curve line of sight.

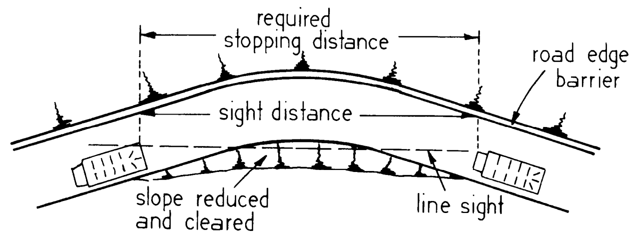
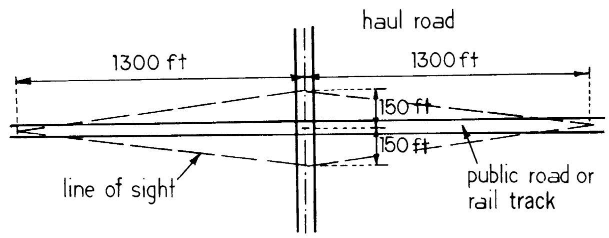


Fig. 13.4.5b. Horizontal curve line of sight.



Conversion factor: 1 ft = 0.3048 m.

Fig. 13.4.5c. Line of sight at route crossing.

(4.5°) but up to 12% (6.8°) for trolley-assist trucks. For safety and drainage reasons, long steep gradients should include 150-ft (50-m) long sections with a maximum gradient of 2% (1°) for every 1500 to 1800 ft (500 to 600 m) of severe gradient.

13.4.3.6 Safe Sight Distances

Sufficient sight distance must be possible to ensure that a truck can stop when traveling at its operational speed before reaching a hazard. Methods of determining suitable geometries related to safe stopping distances have been advocated using the following criteria (Kaufmann and Ault, 1977).

1. Vertical curves should provide smooth transitions from one grade to another and provide ample sight distance for the required braking distance at the operational speed. The sight distance should be taken from the lowest driver's eye height for vehicles in the fleet to a hazardous object 6 in. (150 mm) high as in Fig. 13.4.5a.

2. Similarly, horizontal curves must be laid out to provide ample sight distance (Fig. 13.4.5b). This may involve slope reduction at the inside curve.

3. Where a haul road crosses a public road or a rail track, a safe crossing geometry is shown in Fig. 13.4.5c. Where vision is obstructed (e.g., by a road located in a trench, trees, vegetation, etc.) 150 ft (50 m) either side of the crossing, a distance to 1300

ft (400 m) back along the public road or rail track must be cleared. Approach gradients should be as flat as practicable.

In some situations, it may not be possible to apply these criteria, and approach speeds must therefore be restricted.

13.4.3.7 Haul Road Signs

In general, totally inadequate road signs are used in surface mines since it is often considered that truck drivers become familiar with the route, but the pattern of traffic may be continually changing throughout the life of a mine. Large professionally produced signs with durable surfaces should be installed as needed throughout a mine haul road system. These signs can quickly become obscured by dirt and require periodic cleaning with a high-pressure water jet.

13.4.3.8 Lighting

Lighting is usually provided at crushers, dump points, etc., to improve efficiency, but the level of illumination must be gradually reduced from an illuminated area to a non-illuminated area to help drivers' eyes to adjust safely to these changes in illumination.

13.4.3.9 Runaway Precautions

Runaway trucks can be a serious hazard on steep downhill gradients, and safety provisions to guard against these hazards must be provided as part of haul road design. One well-tried method, originating in Australia, is the location of triangular piles of nonconsolidated fines along the centerline of the haul road. In the event of brake or retarder failure, the truck driver maneuvers into line with the pile so the truck straddles the pile and the truck is brought to a halt (Fig. 13.4.6a), with only minor damage to the equipment on the underside of the truck. The geometry of these "median berms" is fully described in the literature (Anon., 1978).

Escape lanes (Fig. 13.4.6b) are a further method available for arresting runaway trucks, but lack of space may prevent their application in many situations. Where switchback haul roads are employed, escape lanes may often be conveniently located at the end of long, steep grades where the direction of the haul road is reversed. Particular attention must be paid to the radius of entrance curves, haul road width, super-elevation, wearing materials, arresting materials, etc. (Anon., 1978).

13.4.3.10 Cross Slope

Where possible (e.g., dry situations, short-life roads, etc.), a level surface between road edges is preferable, since this provides more even tire loading and less driver fatigue. Where heavy rain is experienced, a cross slope is desirable. Any degree of cross slope must be a compromise that provides adequate drainage without incurring adverse tire loading conditions and driver fatigue. The normally accepted rate of cross slope is 0.75 to 1.5 in./yd (20 to 40 mm/m) depending on conditions.

In conditions of ice, frost, and snow, or on smooth permeable surfaces (e.g., crushed rock) with rock base and sub-base, a 0.75-in./yd (20-mm/m) slope is advisable.

For rough surfaces where ice, frost, and snow are not a problem, a cross slope of 1.5 in./yd (40 mm/m) should be adopted.

Single- and two-lane haul roads may have all the cross slope in one direction, while on benches, the cross slope should be applied inwards, but three- and four-lane haul roads may have a center high point with the cross slope applied in both directions.

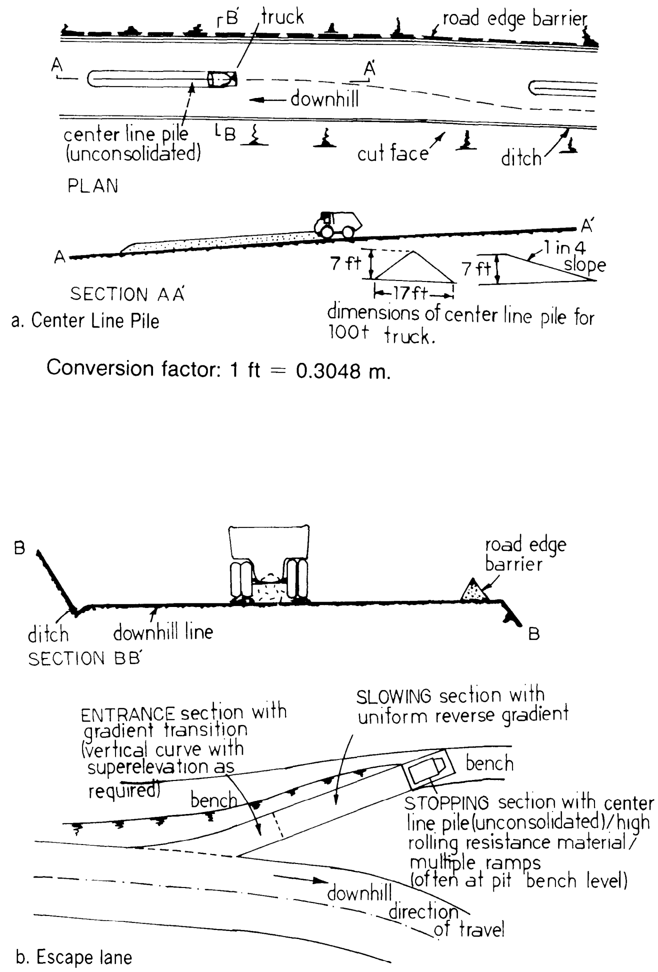


Fig. 13.4.6. Runaway precautions.

Road edge barriers (rock boulders, windrows of graded material, fines, etc.) should be located between roads and adjacent excavations (Kaufman and Ault, 1977).

13.4.3.11 Drains and Culverts

Run-off water can create major problems due to washouts, mud slides, and saturation, making provision of drains and culverts essential. The degree of drainage is dependent on rainfall, catchment area, ground conditions, depth of road base, storm water disposal requirements, etc.

V drains are generally more easily constructed and maintained, and the following features are desirable:

1. Drains should not be excavated in weak spoil (unless lined with flumes).

2. On benches, the cross slope should drain inward with drains excavated along the toe of the slope above. In cuts, drains should be excavated on both sides of the haul road.

3. Where the haul road is constructed on fill materials, drains should be excavated on each side of the embankment.

4. The ground between the edge of the haul road and the drain should be graded towards the drain and must not be obstructed by debris.

5. The sides of the V drains should have slopes of 4:1 where possible, with 2:1 as a minimum.

Table 13.4.6. Culvert Discharge Protection

Discharge Velocity		Slope Angle	Method
fps	m/s	%	
< 3.0	< 1.0	< 10	Divide outlet
< 3.0-10.0	1.0-3.0	> 10	Broken rock mat*
> 10.0	> 3.0	> 10	Divide discharge to broken rock fan

* Note that in some geologic settings, broken rock may not be available. In these circumstances, a velocity reducer may be economically constructed from oil drums spaced at 6-ft (2-m) intervals, manifolded to the culvert discharge. Small-diameter holes are drilled in the drums to provide a dissipated discharge.

6. Drain cross sections must be able to handle the predicted run-offs (Jackson and Walton, 1983). The materials in which the drains are excavated can affect the flow rates (e.g., low flow rates are necessary in weak erodable materials).

7. Where possible, the gradient of drains should be restricted to:

< 3% weak materials
3-5% strong clays, etc.
> 5% crushed rock lining required.

8. Long lengths of down-grade haul roads should be avoided. Sections of flat grade should be included. At these places, the drains should be diverted to the natural drainage system, or a pit sump, through drains or culverts.

Culverts are required to conduct run-off water beneath and away from haul roads. Culvert location, size, etc., is site specific but should include the following criteria:

1. Where possible, culverts should be governed by statutory requirements.
2. Culverts should be installed at all low points beneath haul roads where they intersect the natural drainage system.
3. All haul road intersections, construction changes, etc., require individual culvert design.
4. Drains should be diverted by culverts at regular intervals to the natural drainage system or a pit sump, to reduce flow in the haul road drains.
5. The culvert cross section must be sufficient to accept the maximum predicted run-off without causing excessive back-up at the entrance to the culvert. Several forms of culvert pipe are available, galvanized, corrugated steel, sectional pipe generally being most applicable due to its lightness, ease of handling, and recoverability. Manufacturers provide flow capacity vs. pipe diameter curves.
6. Culvert entries should be protected by a header wall of broken rock or concrete block masonry.
7. Culverts should not discharge on to spoil slopes and must be extended to a point beyond the toe of the spoil slope.

Table 13.4.6 should be used as a guide for avoidance of excessive erosion.

13.4.4 TROLLEY-ASSIST HAUL ROADS

The main advantages of trolley-assist haulage are cost savings through reduced fuel consumption; increased productivity due to increased speed and reduced cycle time; a reduction in electric wheel armature currents, allowing longer hauls out of deeper pits; and greatly increased engine life. At the Rossing Uranium operation, in Namibia, a 12% grade (6.8°) operation is highly successful with less than 2% initial (electrical) rejection on entry.

Driver skills, however, must be enhanced because of higher speeds and narrower steer paths, which demand greater skill and concentration. This can be facilitated, for example, by alignment lights and reflector targets fixed on support poles, etc. The most common accidents are line or pantograph/trolley pole damage caused by not steering on line. This fault is generally quickly corrected. The truck must be positioned under the trolley wire at the required entry speed while activating the pantograph/trolley pole, requiring a short driver training period.

Haul road geometry considerations are important, and the following points provide guidance:

1. A level section at the entry point to the trolley wire facilitates successful contact.
2. Curve radii of not less than 670 ft (200 m) assist the driver in maintaining contact.
3. Lanes should be wider than normal to accommodate higher speeds.
4. Three-lane ramps are necessary to allow trolley-assist trucks to overtake diesel trucks and service vehicles. Additional ramps for shovels are required between benches.
5. Reduced explosive charges and delay firing close to the trolley wire are essential to prevent trolley wire damage due to flyrock.
6. Because of higher speeds, haul roads must have well-graded surfaces, ± 18 in. (457 mm) control of pavement level being essential to ensure constant trolley wire contact; this is best achieved by laser beam indication of the bulldozer blade position during road surface formation.
7. Spillage must be constantly graded off the haul road to avoid trucks being steered off line.

13.4.5 HAUL ROAD CONSTRUCTION MATERIALS

Many mine operators, in the interests of capital cost, simply cut or fill haul roads with the materials existing at the location. This may present various problems since

1. In weak materials, the road surface will deteriorate and passage of trucks will be impeded.
2. Where competent rock is exposed, it is unlikely that an acceptable surface of suitable gradient and elevation will be exposed; irregular and jagged edges may form.

Both problems may result in reduced production and increased costs of fuel, maintenance, and tires.

The primary objective is to construct a roadbed or pavement that permits the transfer of wheel loads over the subgrade (i.e., foundation materials) so that the bearing capacity of the subgrade is not exceeded. It is necessary, therefore, to use guidelines for material selection that allow for the use of a wide range of construction materials including, ideally, those that arise onsite. Both subgrade and roadbed materials may be assessed using the empirical California Bearing Ratio (CBR) test (Anon., 1977a), which estimates the resistance to penetration of compacted materials.

13.4.5.1 Design Approach

Conventional design methods for flexible pavements (Anon., 1970), are generally inadequate for mine haul roads, since they are concerned with on-highway trucks having axle loadings typically less than 9 tons (8 t). Mine trucks may have axle loadings that exceed 60 tons (54 t). Airfield construction criteria have occasionally been adopted, but such methods remain difficult to interpret in the mining situation. A more recent approach recommended by Kaufman and Ault (1977) is preferred; this

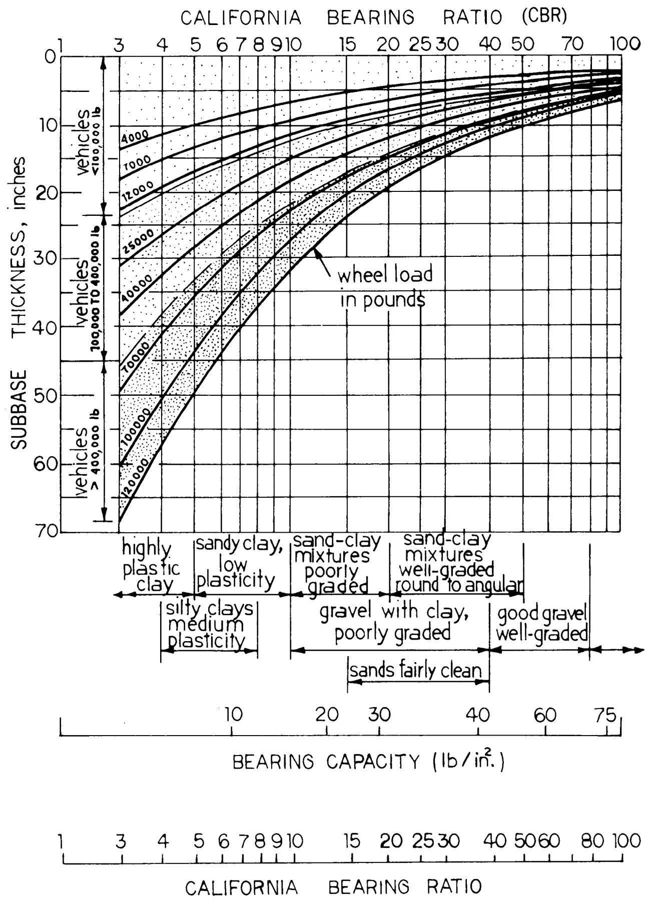


Fig. 13.4.7a. Classification and design curves for haul road construction materials, in English units (Kaufman and Ault, 1977).

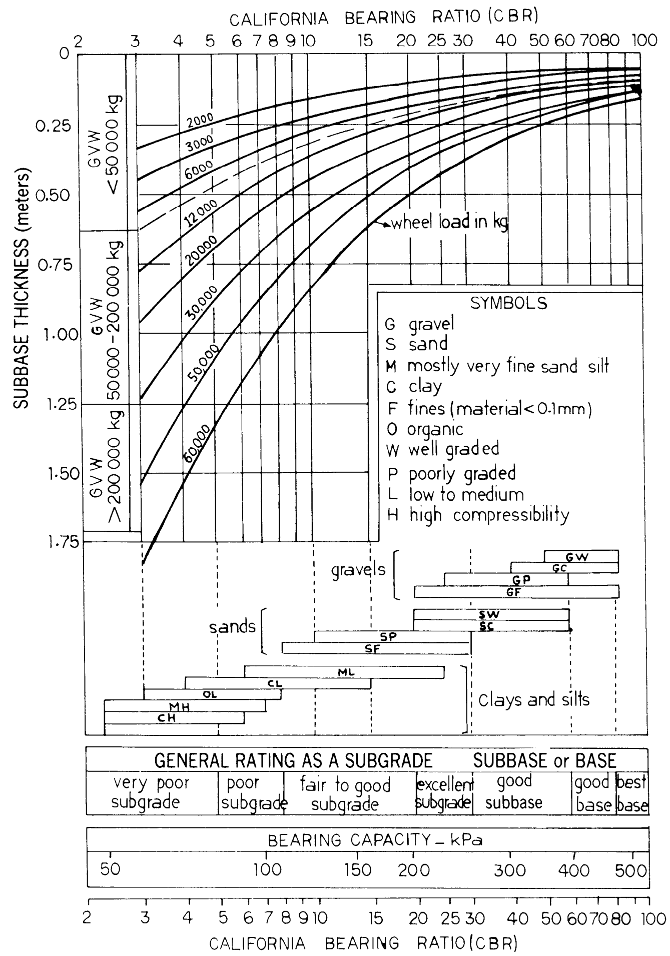


Fig. 13.4.7b. Classification and design curves for haul road construction materials, in SI units (Kaufman and Ault, 1977).

requires that CBR values and wheel loadings are first ascertained.

CBR values must be determined for likely subgrade and roadbed or pavement materials. The roadbed or pavement may comprise up to three distinct courses, a wearing or surface course, a base course, and a subbase; in certain circumstances, the subbase and even the base course may be omitted. Fig. 13.4.7 shows typical CBR values that should only be used as an initial guide. In subsequent design, CBR values must be determined from both soaked and unsoaked tests (Anon, 1977a); in areas with high rainfall or poor drainage, soaked results should be used.

Maximum wheel loadings may be calculated from manufacturers stated axle loadings divided by the number of tires per axle. Wheel loadings on tandem wheels are increased by 20%.

Fig. 13.4.8 illustrates the application of this approach with respect to a haul ramp in a proposed surface gypsum mine in which the constraints were as follows:

Maximum wheel loading 33,000 lb
Subgrade CBRs 5, 15, and 80%
Potential subbase CBR 15%
Required base course CBR 80%
Required running course CBR 80%

Considering the poorest CBR, the steps in calculating the roadbed thickness are as follows:

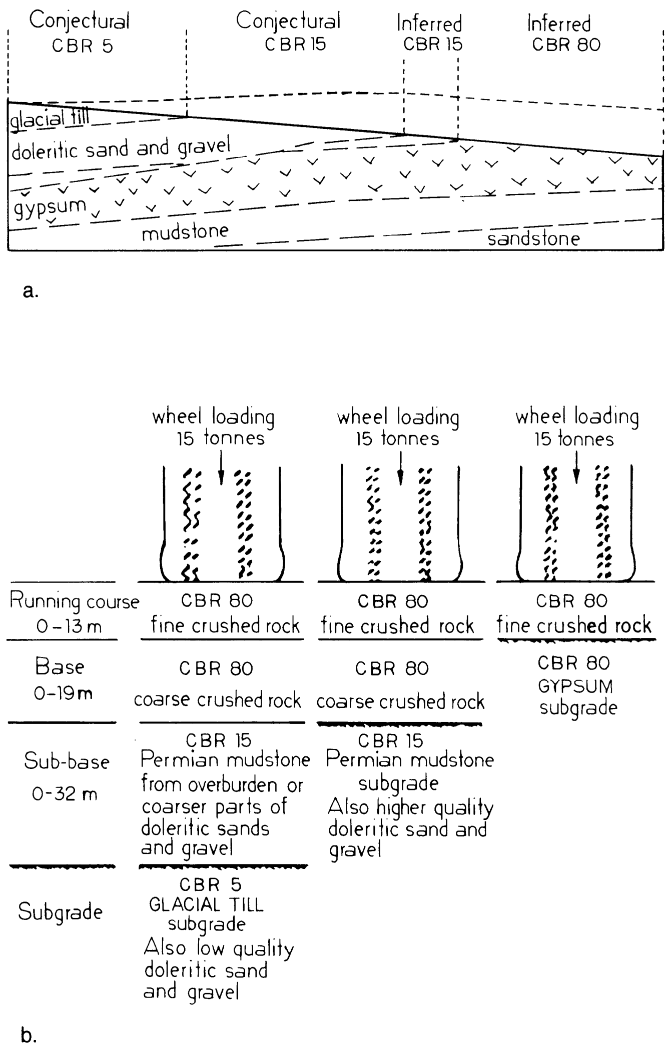
1. A subgrade CBR of 5% intersects as interpolated 33,000 lb (15,000 kg) a maximum wheel load line at 25 in. (635 mm). The top of the subgrade will be 25 in. (635 mm) below the road surface.

2. A subbase (or subgrade) CBR of 15% intersects the interpolated 33,000-lb (15,000-kg) maximum wheel load line at 12.5 in. (318 mm). The top of the subbase (or subgrade) will be 12.5 in. (318 mm) below the road surface. For conditions where the subgrade CBR has a value of 15%, only a base course need be found.

3. A running course CBR of 80% intersects the interpolated 33,000-lb (15,000-kg) maximum wheel load line at 5 in. (127 mm). The top of the base course will be 5 in. (127 mm) below the road surface, and the base course, therefore, will be 12.5 − 5 = 7.5 in. (190 mm) thick. A running course may be directly placed on naturally occurring base course material with a CBR of more than 80%; such materials must, however, be scarified, leveled, and graded.

13.4.5.2 Materials for Construction

It is apparent that both the materials over which haul roads must be routed and any onsite materials likely to be used during construction should be tested to give CBR values. Assessments should be made of the quantities required and the availability of materials within the scheduled mining operation; it is not uncommon to find the most suitable construction materials arise



Conversion factor: 1 ft = 0.3048 m, 1 ton = 0.9072 t.

Fig. 13.4.8. Use of construction material design chart. a. Section along haul road. b. Haul road pavement design.

in the later stages of working. Due regard must be given to roadbed thicknesses, widths, lengths, and future realignments in estimating quantities, particularly when materials may be imported onto the mine site. The various components of the roadbed may be considered separately.

SUBGRADE. As far as possible, this must be graded to an even surface; in many surface mines, this is the only step in road construction. While adequate for short-life haul roads in favorable materials and nonsevere climatic conditions given frequent regrading, such conditions are rarely adequate for longer-life haul roads. However, for very short-life lengths of haul road, the adequacy of naturally occurring materials with respect to trafficability may be assessed by reference to the recently developed moisture conditions test (Anon., 1982). A test value of greater than 13 indicates soils unlikely to give short-term problems with respect to rutting, etc., when the GVW does not greatly exceed 100 ton (90 t). Adverse subgrade materials appear to be increasingly encountered as more marginal mineral reserves are worked.

Excessive amounts of imported material are sometimes required. At one mine, the only economic solution was to construct the roadbed of coal; at another group of mines, peaty deposits had to be covered by soft-to-firm clay, which would rarely be considered as an ideal subbase. It is sometimes considered that synthetic membranes or fabrics reduce the quantity of subbase and base course materials in areas with poor subgrade. This is an over simplification. Membranes primarily restrict the loss of material punched down into subgrade materials (and may aid subsequent recovery for future road construction). The pumping or injection of weak subgrade into the roadbed is similarly inhibited.

The same approach to design thicknesses of the roadbed should, however, be used in all settings; membranes may allow the design to be achieved (Rankilor, 1981). Membranes also permit better compaction of the subbase and consolidation of the uppermost layers of the subgrade due to moisture loss occurring without mud pumping. It is often advisable to place sand and gravel-sized material over membranes if alternative subbase materials include large rock fragments that may excessively damage the membrane. Nonwoven fabrics are less prone to tearing. Whilst experience of the use of membranes is good, the economics of each case must be investigated and less expensive alternatives reviewed. These alternatives can include the use of brushwood and straw in extreme conditions such as peat bogs, and sand or fly ash over soft clays and silts, prior to forming the roadbed.

SUBBASE. The subbase width should exceed the surface and base course width by 24 in. (610 mm). Wherever possible, use should be made of on-site materials which should be compacted in layers of 6 to 8 in. (150 to 200 mm) in thickness. Compacted materials should be placed using rollers or with the controlled passage of mine trucks.

BASE COURSE. Most suitable materials for base courses are those having high CBR values such as crushed rock -4 in. (-100 mm) or high-quality gravels. Frequently, these are the least readily available materials, and offsite resource surveys are commonly required. It is essential that large lumps of rock are excluded, as well as organic material, clay, and silt. Materials should be compacted as for the subbase. The reuse of base course (or subbase) material is sometimes necessary, often justifying the use of synthetic permeable membranes.

Recent developments of high-water content (circa 90%) materials for underground mine packs (e.g., "Tekpack") show promise for use in base (or subbase) courses where the availability of other materials may present problems. Long-term stability is as yet undetermined, and a minimum temperature of 70°F (20°C) is required for the material to set.

Several chemical additives often known as binders or packers, including hydrated lime, calcium chloride, cement, and other stabilizers, are available for improving the characteristics of base course materials (and also subgrades). The effectiveness of these additives is heavily dependent on chemical compatibility. The economics for each site must be proven; materials other than cement, however, are often unsatisfactory and construction methods that rely on building a roadbed of adequate thickness in relation to the character of the subgrade materials may be preferred.

RUNNING OR SURFACE COURSE. The best surface courses are those that give hard, even surfaces and little dust with no penetration under load. High adhesion and low rolling resistance coefficients are preferred. The best surface materials that also have a low permeability are, therefore, concrete and asphalt (see Table 13.4.7), although construction costs are high. Where available, finer crushed rock –0.5 to +1.0 in. (–10 to +20 mm), coarse sand-size mill tailings, clean gravel, and clinker are

Table 13.4.7. Adhesion and Rolling Resistance Coefficients

Description	Coefficient of Adhesion	Rolling Resistance % GVW
Concrete of blacktop	0.9	2.0
Rock base, dry	0.7	3.0
Rock base, wet	0.65	3.5
Pit floor, rock	0.55	5.0
Partly compacted gravel	0.45	5.0
Gravel, unmaintained, wet	0.4	7.5
Weak materials, flexing considerably	0.35	8.0
Soft, muddy, rutted road	0.3	15.0–20.00
Ice	0.1	1.0

suitable low cost materials if compacted when moist. Fines (silt and clay) should essentially comprise between 5 and 10% of such running courses.

The coefficient of rolling resistance can be related to tire penetration by the equation (in SI units),

$$\text{Coefficient of rolling resistance} = 0.02 + 0.0007 \times \text{mm of tire penetration.} \tag{13.4.3}$$

Only carefully constructed haul roads will permit safe, high running speeds that will both improve production and reduce maintenance, tire, and fuel costs. Money spent on the investigation of materials and on haul road construction is usually a sound investment.

13.4.6 CONCLUSIONS

The construction of surface mine haul roads differs in many respects from regular highways in that the wheel loads in mining are far higher, yet the pavement life is often quite limited (Atkinson and Walton, 1983). The same general principles as those in highway engineering may apply, but the problems of economics, design life, continual change of layout, etc., in surface mine haul roads requires a different approach. The mine designer is required to use continuing ingenuity in layout by the use of cuts, spoil bridges, and ramps, etc., to provide the shortest, least steep and most easily negotiable haul roads.

The high investments tied up in truck fleets require high-quality haul roads to ensure:

1. Maximum production, by the use of high but safe truck speeds, less down time for maintenance, and increased efficiency because of reduced driver fatigue.
2. Reduced operating costs due to reduced fuel consumption, reduced repair costs, and longer tire life.
3. A safer working environment that will improve efficiency.

In the past there has perhaps been a failure to recognize that inadequate haul roads can be a major cause of reduced profitability. As the trend to larger pits, using large equipment, to achieve economies of scale continues, the need for improved haul road design and construction is readily apparent.

ACKNOWLEDGMENT

Acknowledgment is due to the Institution of Mining and Metallurgy, London, for permission to use material appearing in their publications (Walton and Atkinson, 1978; Atkinson and Walton, 1983).

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Chapter 13.5
IN-PIT CRUSHING AND CONVEYING

EARL M. FRIZZELL AND THOMAS W. MARTIN

In 1956, the first mobile crusher was installed in a limestone quarry in Hover, West Germany. The crusher enabled the quarry operator to take advantage of continuous belt conveyor haulage and eliminated a problem of high-cost road construction and maintenance in wet soft ground, with resultant cost savings. Since that time, the number of mobile in-pit crushing and conveying operations has increased to over 100 (Table 13.5.1). Most of these installations are limestone quarry operations of less than 1100 tph (1000 t/h) and are fed directly by a mine face excavator because of ground conditions in the quarry.

13.5.1 IN-PIT CRUSHER SYSTEMS

Crushing is done in-pit mainly to allow transport of the material out of the pit by a conveyor system. Therefore, the units used for primary crushing are jaw gyratory, gyratory, hammer, impact, roll, and jaw crushers. A summary of the capacity, type of crusher, and amount of material being processed world-wide is shown in Table 13.52.

In-pit crusher systems have mobility capabilities that range from fully mobile units in continuous use to permanently fixed units (Fig. 13.5.1). Once installed in the pit, fixed crushers normally require that haulage trucks travel a gradually increasing distance from the mining face.

The term "mobile" has been used to describe any type of crusher that can be moved. The term has become a generic one and should be used when referring to any crusher that is movable. The system (or sections of the system) is mounted on a frame base, allowing the complete unit to be moved by skids or a transporter, such as a crawler system or a walking mechanism, that may or may not be an integral part of the unit itself. An integral mobile crushing plant includes a hopper, feeder, crusher, and discharge conveyor, in addition to all the required auxiliary and electrical systems. The following terms are presented to help distinguish the range of mobility among mobile crusher systems now in use.

13.5.1.1 Mobile Crusher

This type of crusher works at the mine face, is directly fed by an excavator, and moves in unison with the excavator on its own transport mechanism as mining progresses. An example is the unit delivered in 1976 to Yamama Cement Co. in Reyadh, Saudi Arabia. Its capacity is 880 tph (800 t/h) in limestone using a double-shaft hammer crusher. Plant weight is 660 tons (600 t) (Fig. 13.5.2).

13.5.1.2 Semimobile Crusher

This unit works close to the mine face but is moved less frequently than a mobile crusher. The transport mechanism may be a permanent part of the crusher frame. Such a unit was delivered in 1971 to S.A. des Cements de Champagnole, Rochefort sur Nenon, France. A single-shaft hammer crusher provides the 496-ton (450-t) plant with a 660-tph (600-t/h) capacity in limestone (Fig. 13.5.3).

13.5.1.3 Movable Crusher

A movable crusher is centrally located in a mine near the same level as the mine's working face. It is relocated every 1 to 2 years, as required, to maintain the relationship between distance and elevation from the face. Fig. 13.5.4 shows the Dutex system at the Sierrita mine in Arizona. Copper ore is processed at rates up to 4000 tph (3628 t/h) by a gyratory crusher, which is part of a movable three-section system. The crusher can be completely relocated in 48 hours because no concrete structures or retaining walls are required.

13.5.1.4 Portable Crusher

This term is used in Europe for crushers with temporary support foundations. The crusher plant is moved in sections. In the United States, this term refers to units that can be moved on a highway with a minimum amount of dismantling. This term is ANSI-approved and is a national standard, with the number CAPPA-78.

13.5.1.5 Modular Crusher

In a modular crusher, the crusher station can be disassembled into modules and reassembled in a new location in about 30 days. Such moves normally would not be made more than every three to five years. An example is the 4- × 6-ft (1.2- × 1.8-m) primary gyratory crusher station designed for the Majdanpek copper mine in Yugoslavia, which can be dismantled into modules having a maximum weight of 58.7 tons (53.2 t) and a maximum size of 42 × 22 × 22 ft (12.8 × 6.7 × 6.7 m) (Fig. 13.5.5).

13.5.1.6 Semifixed Crusher

A semifixed crusher is mounted on a steel platform, which reduces the need for a concrete foundation. Any planned relocation would not be for less than 10 years. The two 4.5- × 6.6-ft (1.4- × 2-m) primary gyratory crushers at the Twin Buttes copper mine in Arizona were erected in part on heavy steel structures and can be moved. The original plan was to have the crushers remain in place for at least 10 years, after which they would be relocated down in the pit in a series of leap-frog moves to capitalize on the advantages of belt conveyors reaching to the bottom of the pit (Fig. 13.5.6).

13.5.2 TRANSPORTER TECHNOLOGY

Mobile crushers weighing up to 800 tons (725 t) have been moved by crawlers, tires, and walking mechanisms. Generally, the smaller and more mobile units are moved by skids, tires, or crawlers, and the larger semimobile units are relocated by crawlers or walking mechanisms. A transport system that should be considered is rail wheels, which carry extremely heavy loads with great efficiency because of their low rolling friction. Rail wheels create a severe braking problem on grades approaching 10%, however, which probably limits their use. Air flotation devices have moved more than 1500 tons (1360 t), but use of

Table 13.5.1. In-Pit Crushing Units 1956–1989

Date	Type	Transport	tph	Date	Type	Transport	tph
1956	Single-Shaft Hammer	Crawlers	250	1976	Impact	Walkers	850/1100
1959	Impact	Wheels	125		Roll	Pneumatic Tires	500
1961	Single-Shaft Hammer	Crawlers	250		Single-Shaft Hammer	Walkers	1000
1962	Single-Shaft Hammer	Crawlers	300	1977	Impact	Wheel	800
1963	Double-Shaft Hammer	Crawlers	200		Roll—2 stage	Pneumatic Tires	900
1965	Double-Shaft Hammer	Crawlers	400	1978	Gyratory	Pneumatic Tires	500
	Double-Shaft Hammer	Crawlers	450		Jaw-Type Gyratory	Crawlers	600
1966	Double-Shaft Hammer	Rails	335		Roll—3 stage	Pneumatic Tires	900
	Impact	Crawlers	650–750	1979	Gyratory	Walkers	1250
	Impact	Crawlers	700		Impact	Rails	1000
	Jaw-Type Gyratory	Walkers	300	1980	Gyratory	Walkers	2500
1967	Double-Shaft Hammer	Crawlers	400		Roll	Pneumatic Tires	500
	Jaw-Type Gyratory	Walkers	700	1981	Gyratory	Walkers	2000
1967/68	Single-Shaft Hammer	Crawlers	200		Jaw	Walkers	2000
1968	Double-Shaft Hammer	Crawlers	260	1982	Gyratory	Crawlers	6000
	Double-Shaft Hammer	Walkers	350		Jaw	Walkers	2000
1969	Double-Shaft Hammer	Crawlers	600		Roll	Pneumatic Tires	2500
	Double-Shaft Hammer	Walkers	500–800	1983	Double Roll	Crawlers	1000
	Double-Shaft Hammer	Walkers	500	1983	Gyratory	Crawlers	1600
	Double-Shaft Hammer	Walkers	700–1000		Gyratory	Walkers	1500
	Impact	Walkers	325		Gyratory	Crawlers	4000
1970	Double-Shaft Hammer	Walkers	700–1000		Gyratory	Walkers	3900
	Double-Shaft Hammer	Crawlers	400	1984	Double Roll	Crawlers	3600
	Double-Shaft Roll	Walkers	600		Double Roll	Crawlers	3600
	Jaw-Type Gyratory	Walkers	700		Double Roll	Crawlers	3600
1971	Double-Shaft Hammer	Crawlers	400/750		Gyratory	Pneumatic Tires	2500
	Double-Shaft Hammer	Crawlers	400/750		Gyratory		3600
	Double-Shaft Hammer	Crawlers	500		Impact	Crawlers	500
	Double-Shaft Hammer	Crawlers	450		Impact	Walkers	4500
	Impact	Walkers	400		Single-Shaft Hammer	Crawlers	850
	Impact	Walkers	325	1985	Feeder Breaker	Skids	2800
	Single Toggle Jaw	Walkers	1200–1500		Feeder Breaker	Skids	2800
	Single-Shaft Hammer	Walkers	600		Impact	Crawlers	800
1972	Double-Shaft Hammer	Crawlers	500		Impact	Pneumatic Tires	600
	Double-Shaft Hammer	Walkers	1000		Impact	Pneumatic Tires	600
	Impact	Walkers	1000		Impact	Pneumatic Tires	720
	Impact	Walkers	360/420		Impact	Pneumatic Tires	800
	Impact	Walkers	650		Impact	Crawlers	500
	Jaw-Type Gyratory	Walkers	700		Single-Shaft Hammer	Walkers	1250
1972/73	Impact	Walkers	500		Single-Shaft Hammer	Walkers	1000
1973	Single-Shaft Hammer	Walkers	600	1986	Double Roll	Pneumatic Tires	1200
	Single-Shaft Hammer	Walkers	800–850		Double Roll	Crawlers	1000
1974	Double-Shaft Hammer	Pneumatic Tires	500		Double Toggle Jaw	Crawlers	6000
	Double-Shaft Roll	Wheel	1200		Feeder Breaker	Crawlers	1350
	Gyratory	Walkers	1000		Impact	Pneumatic Tires	300
	Single-Shaft Hammer	Pneumatic Tires	600		Jaw	Crawlers	1000
	Single-Shaft Hammer	Walkers	700–770	1987	Gyratory	Crawlers	6600
1975	Double-Shaft Hammer	Crawlers	600		Gyratory	Crawlers	6600
	Double-Shaft Hammer	Crawlers	600		Gyratory	Crawlers	9000
	Double-Shaft Roll	Walkers	1200/1500		Single-Shaft Hammer	Tires	750
	Single Toggle Jaw	Walkers	1500/1700	1988	Feeder Breaker	Crawlers	800
	Single-Shaft Hammer	Walkers	1000	1988/89	Single-Shaft Hammer	Tires	750
1975/76	Double-Shaft Hammer	Crawlers	1000		Gyratory	Crawlers	6000
1976	Double-Shaft Hammer	Crawlers	800		Double Roll	Crawlers	2000

Conversion factor: 1 ton = 0.9072 t.

these devices is restricted to extremely level surfaces. Climbing a 10% grade, as required in a mine, would be impractical with current air flotation technology.

13.5.2.1 Crawlers

Crawlers are widely used in moving heavy equipment, such as mine shovels and dozers. Single sets of crawlers have moved up to 1000 tons (900 t) and can be teamed to move even heavier loads (Fig. 13.5.7). Crawlers are also able to move at relatively high speeds (up to 40 fpm, or 12 m/min) loaded. Crawler transporters are almost always diesel-powered and have hydraulic lift mechanisms. The weight of a 1000-ton (900-t) lift crawler transporter is approximately 200 tons (180 t).

Crawlers provide a stable platform and, with the tracks properly suspended, can move on uneven ground. They can negotiate steep slopes, although they are limited to grades of 10% or less when transporting a crusher. This is because traction forces are limited and the center of gravity and resultant moments on the crawler frame shift. Crawlers are very durable, but they are likely

Table 13.5.2. Mobile Crusher Distribution

	No.	%
Capacity, lb		
<500	30	27.8
501–1000	47	43.5
1001–2000	13	12.0
2001–3000	5	4.6
3001–4000	6	5.6
4001–5000	1	.9
5001–6000	3	2.8
6001–7000	2	1.9
7001–8000	—	—
8001–9000	1	.9
Crusher Type		
Jaw gyratory	7	6.5
Gyratory	15	13.9
Hammer	34	31.5
Impact	27	25.0
Roll	16	14.8
Jaw	5	4.6
Feeder breaker	4	3.7
Material		
Limestone	78	72.2
Waste rock	7	6.5
Copper ore	6	5.6
Bauxite	5	4.6
Aggregate	3	2.8
Coal	2	1.9
Tar sands	2	1.9
Phosphate	1	.9
Iron ore	1	.9
Oil shale	1	.9
Gypsum	1	.9
Serpentine	1	.9

Conversion factor: 1 lb = 0.4536 kg.

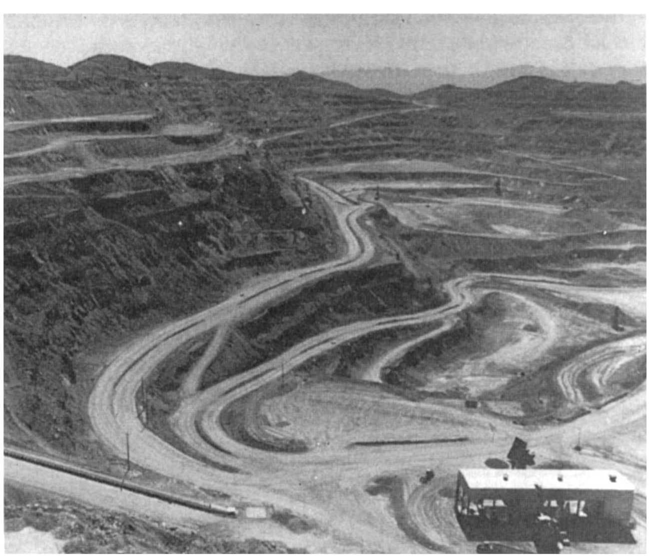

Fig. 13.5.1. Fixed crusher.

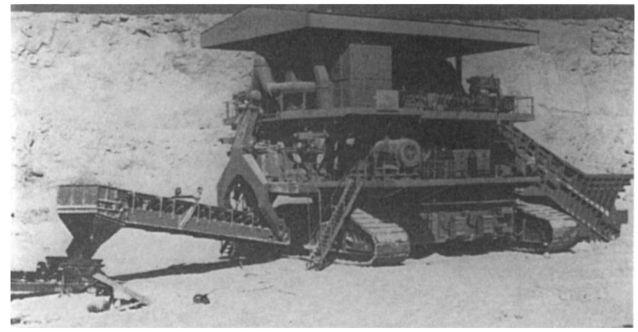

Fig. 13.5.2. Mobile crusher with cross-assembled crawlers (*courtesy:* Krupp International, Houston, TX).

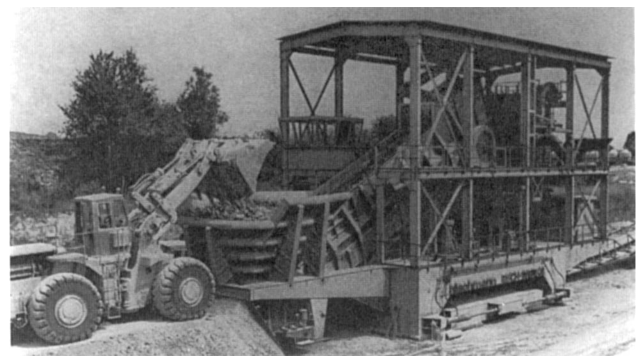

Fig. 13.5.3. Semimobile crusher on hydrowalkers (*courtesy:* Krupp International, Houston, TX).

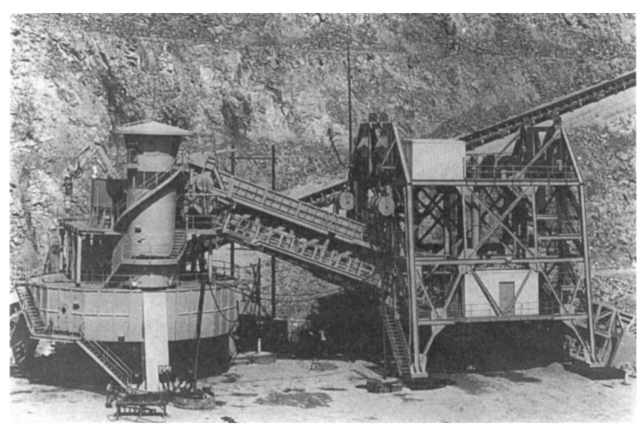

Fig. 13.5.4. Crawlers on movable crusher at a copper installation (*courtesy:* Dutex, Tucson, AZ).

to suffer damage from shock and vibration if they are left loaded under a working crusher for long periods of time. They can almost complete a turn in one spot by running one set of treads while locking the other. Under a heavy load, this maneuver is

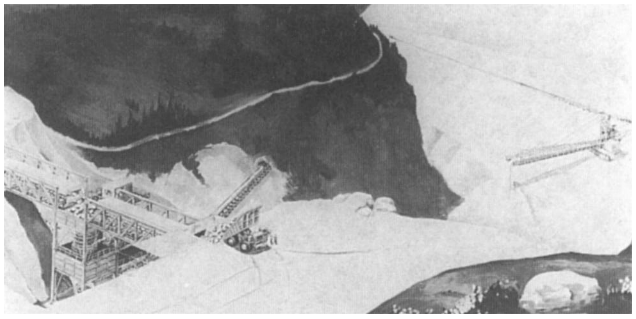

Fig. 13.5.5. Modular crusher (*courtesy:* Stephens-Adamson, Aurora, IL).

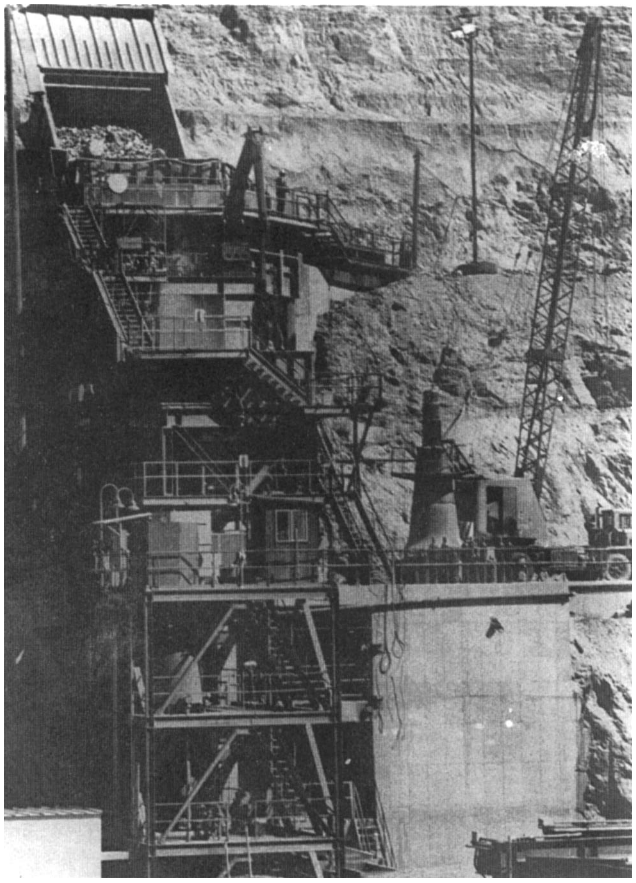

Fig. 13.5.6. Semifixed crusher (*courtesy:* Rexnard, Milwaukee, WI).

restricted for more than a few degrees of angular rotation because of ground damage or tread damage on the locked track.

13.5.2.2 Wheels

Wheels, either tired or rail, are the most widely used means for transporting heavy loads. Multiple wheels can carry very large loads, but speed is highly restricted. Drive mechanisms for tires includes electric motors like those used in haulage trucks and hydraulically operated wheels that can produce more torque.

Fig. 13.5.7. Movable crusher in transit on crawler transporter (*courtesy:* Krupp International, Houston, TX).

The cost of tires is relatively low, but the drive motors and controls can make tired transport cost more than a crawler or a walking mechanism.

Grades up to 10% are negotiable, and both electric and hydraulic motors will assist downhill braking. Rubber tires are quite durable and are relatively immune to shock and vibration. Wheels can be left under a crusher for long periods but tend to take a set if the carcass is made of nylon or steel cord. Cold temperatures also embrittle rubber tires, but the rubber can be compounded to reduce temperature-related problems and are, in fact, used in very cold climates. Ozone also attacks and embrittles rubber, but this can be reduced with chemical additives.

13.5.2.3 Walking Mechanisms

Walking mechanisms are the most widely used method of moving loads in excess of 1000 tons (900 t) (Fig. 13.5.3). Machines have been built that can lift up to 5000 tons (4500 t) and are capable of moving approximately 200 fph (60 m/h), a speed sufficient for modular or semifixed crushers. However, walking mechanisms are less desirable than crawlers where speed is necessary or constant hourly use is likely. These systems are generally less expensive than crawlers and have a good service record for reliability.

Walking mechanisms require careful ground preparation so they will not be impeded or the load redistributed by small bumps. Spillages around a crusher from dumping or from the conveyors have to be cleaned up before the units can be moved. The walking mechanism is very durable and can be left for long times under load (both static and dynamic) because when the hydraulic system is depressurized, the metal beam (foot) carries the load. In units up to 1000 tons (900 t), walking mechanisms have demonstrated the ability to turn 360°.

13.5.3. CONVEYOR SYSTEMS

A drawback of conventional conveyor haulage out of open pits is that the maximum vertical slope angle a conveyor can achieve is 17 or 18°, which does not coincide with the typical slope of an open pit wall, which ranges anywhere from 38 to 45°. This requires that either the conveyor must be run out of the pit

Fig. 13.5.8. Conveyor flights leaving a pit via switchbacks.

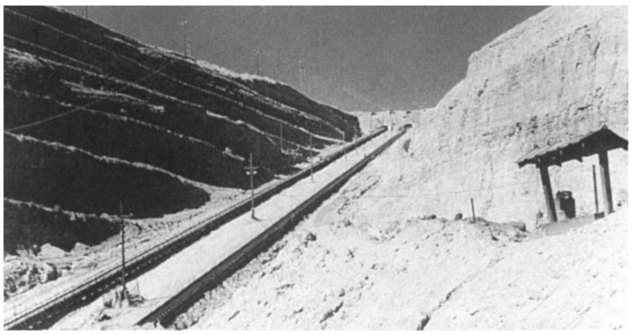

Fig. 13.5.9. Open pit conveyor trench.

in switchbacks similar to those used for truck haulage roads or that a trench must be cut through the side wall of the pit. Both of these approaches have disadvantages. Transfer systems must be installed to facilitate the change in direction of the conveyors at each switch point (Fig. 13.5.8) or a greater amount of waste material must be removed from the pit to create a lower slope angle (Fig. 13.5.9). These problems have led to the development of high-angle conveyors to remove material from a pit (see Chapters 9.3 and 13.3).

There are many types of belt systems that should be considered in developing a conveying system for a specific installation. These belt systems generally fall into the following categories:

1. Fabric belt systems.
2. Steel cable belt systems.
3. Cable belt systems.

13.5.3.1 Fabric Belt Systems

Conventional fabric (canvas) belt systems have been in operation for many years, but they have been greatly improved recently by the use of a synthetic carcass. The Rubber Manufacturer's Association has classes for belt carcasses from MP-35 through MP-240. Each classification number is an indication of the strength of the fabric used in manufacturing the belt carcass. Shorter belt conveyors that can use mechanical splices normally use belts up to a rating of MP-70, allowing for 55 lb/ply/in. of splice (0.98 kg/ply/mm) for maximum tension. The MP-240 rating, on the other hand, is for a vulcanized splice that will allow up to 240 lb/ply/in. of splice (4.3 kg/ply/mm) for maximum tension. Although conveyor belts can be made with more than 10 plies, it is a general rule that for a coordinated type installation, seven plies are a maximum. Therefore, a limitation is placed on the driver size and capabilities, or maximum belt tension that should be kept in mind with the use of fabric belts. For example, a 7-ply, MP-240, 60-in.- (1.5-m-) wide belt should carry only 100,889 lb (45,763 kg). The reason for the seven-ply limit is that pulley diameters are directly affected by the number of plys and the type of fabric construction in a belt. Other than limitations posed by pulley diameters and the capability of the belt to run smoothly in a trough, fabric belts can run on most conveyor systems.

13.5.3.2 Steel Cable Belts

A steel cable conveyor belt consists of multistrand steel cables embedded in a rubber or plastic compound. The steel cables are the load-carrying members of the conveying system. These units have the advantage of being able to take much greater tensions than fabric belts. The steel cable belts can be constructed to handle any desired tension ranging from 500 lb/in. (9 kg/mm) width of belt to 5000 lb/in. (89 kg/mm) width of belt. For example, a 60-in. (1.5-m) belt with a rating of 5000 lb/in. (89 kg/mm) width would allow a maximum tension of 300,000 lb (136,000 kg).

13.5.3.3 Cable-supported Belt Systems

A relatively new long conveyor system is the cable belt conveyor. These systems do not have cross roll idlers, and the belt is not the tension member. Rather, the belt is driven by friction on load-carrying steel cables running lengthwise in molded cleats or grooves near the outer sides of the belt on the upper run, and approximately a quarter of the belt width in from the sides on the return belt. The cables are in turn supported by cable sheaves and driven by specially designed drives and take-up systems. The driving cables for the cable belt system generally range in diameter from 1¼ to 2½ in. (32 to 64 mm). General alignment of this type of belt is simpler than the troughing belt idler system. Long-radius horizontal curves can even be accommodated with this system.

Capacities of the cable belt system have varied from 3000 to 5000 tph (2700 to 4500 t/h) with drives up to 10,000 hp (7450 kW). These cable belts are presently being manufactured and installed up to 60 in. (1.5 m) wide and have been included in installations for run of crusher materials. One such installation handles 1815 tph (1634 t/h) net of copper ore at 800 fpm (244 m/min), and uses a 2493-hp (1860-kW) drive. The conveyor is 42 in. (1.07 m) wide by 32,480 ft (9900 m) long with 50 ft (15.2 m) of lift. Another installation is a 3500-hp (2600-kW) cable system raising 800 tph (720 t/h) of iron ore 2432 ft (741 m) over a distance of 15,140 ft (4615 m).

13.5.4 AUXILIARY SYSTEMS

In addition to the crusher and main conveying system, other considerations when installing an in-pit system are the feeder to the main conveying system and rock transfer from one conveyor flight to the next.

13.5.4.1 Discharge Feeders from the Primary Crusher

There are three major types of feeders presently used under the primary crusher ore pocket.

OVERLAPPING FLIGHT APRON FEEDER. This feeder looks and functions much like a crawler tread where the crushed output falls directly on top of the tread. The ore is then carried out over the end of the feeder and dropped on to the conveyor. Depending upon the size and type of material handled, the flights are fabricated steel pans or heavy cast steel. This is an articulated type of feeder and normally runs less than 80 fpm (24 m/min). This is the most common output feed system used in large open pit mines because it is extremely rugged and can start up with a heavy load on it.

VIBRATING FEEDER. This type of feeder may have a flat pan up to 8 ft (2.4 m) wide and may be vibrated by more than one vibrating head. Up to 6000 tph (5400 t/h) of stone have been handled in stone quarries under primary crushers. The amplitude of this feeder is controlled by a DC controller and can be varied. Other types of eccentric weighted mechanical feeders are used. The variable speed arrangement on these latter units are not as easily controlled as one which has an electrical or mechanical impulse system.

BELT FEEDER. Because of the tendency of fabrics to rip when handling larger-sized material, the use of belts under primary crushers as feeders has only come about in recent years with the development of stronger materials. These materials are extremely heavy fabric or steel cable belts with nominal l-in. (25-mm) covers and breaker strips. The belts are usually thickened further with a layer of molded or vulcanized rubber (wear pads), so that the overall thickness of the belt cover could be 2 to 3 in. (50 to 75 mm). The rubber is added to improve abrasion, crushing, and tearing resistance of the belt. Belt feeders cost much less initially, can be made longer, and require less mechanical maintenance and cleanup than apron feeders. Because of these advantages, it can be assumed that belt feeders will be used more frequently in the future as belt technology improves.

13.5.4.2 Transfer Chutes

If the design and configuration of major conveyor systems calls for switchbacking conveyor flights to leave the pit, a major consideration will be the means of transferring material from one conveyor to another. The transfer height between one belt to another should be reduced as much as possible by almost laying the material on the receiving belt. This will not only minimize equipment wear, but will save energy by reducing the overall vertical height the material must be conveyed. While this is desirable, it may not readily be attainable where there are other than parallel transfers.

A transfer system stops the forward movement of the material moving in one direction, turns it, and starts it in a new direction. Transfer is usually accomplished by dumping the conveyed material into a rock box, dissipating much of the energy of the material. From the rock box, the material is transferred by chute or slide onto the next conveyor in the direction of movement of that conveyor. The back of the chute is sloped to impart velocity in the direction of the conveyor.

A major problem with transfer systems, especially when moving large, blocky types of stone or ore, is excessive wear on the belt surfaces and idlers directly below the loading area caused by the impact of the material. To alleviate wear, the back portion of the chute may also contain a grizzly, which allows the fine materials to fall through and provides a cushion on the belt for the large-sized materials that load at the end of the chute. Where grizzlies are not practical, a V-shaped chute can be constructed. A V-shaped chute restricts impingement of large pieces of ore or overburden directly on the center of the next conveyor belt and also tends to load finer materials onto the belt before larger materials.

13.5.4.3 Conveyor Hoods and Wind Guards

In moderate climates, conveyor hoods prevent the belt from blowing off the idlers when the belts are empty and prevent some dusting from the materials being handled. In more severe northern climates, the hoods greatly reduce the snow and ice problems. Hoods also retain dust at the approach and take-away areas of a transfer station.

As an alternative, wind loops of round steel bars can be arched over the top of the belt and attached to the idler support system. Spacing depends upon operational requirements. The supports for standard hoods can also be used, if hoods are a future consideration.

With deep trough idlers, the underside area of the belt exposed to wind is considerable. Permanent side shields have been used to prevent problems in these cases. These are adaptable also to tripper belts or trunk line stacker belts that cannot use hoods or loops.

13.5.5 SELECTION AND DESIGN GUIDELINES

Those material characteristics that affect the selection of an in-pit crushing and conveying system are the same as those affecting overland crushing and conveying equipment. The density, moisture content, maximum lump size, size consistency, abrasiveness, and degradation and dusting of the material are all factors to be considered.

13.5.5.1 Crushers

The size of the crusher is determined on the basis of the material to be crushed, the rate of production, feed size, and output size that is related to the selected belt width, which is a function also of the system capacity. Primary crushers reduce run-of-mine rock from the as-shot size to a size that can be carried on a conveyor or utilized in the rest of the processing plant. The requirements for a crusher include the following factors:

1. The crusher must be able to reduce the largest-sized feed normally received.
2. The maximum particle size for conveyed material should not exceed 30% of the belt width.
3. Ideally, the crusher should be able to break the rock to a size that reduces the amount of subsequent crushing required (for in-pit crushing and conveying, this is less important than sizing the rock to be carried on the conveyors without causing damage to the conveyor).
4. The crusher should use the minimum amount of power possible (however, the power consumed by the primary crusher is not really great compared to the power consumed in the following crushing and grinding stages).
5. The crusher should be labor efficient. It should not require a great deal of attention to remove oversized rock or to prevent clogging.
6. Its wearing components should last as long as feasibly possible (wear liner and mantle replacement ranges from 2 million tons (1.8 million t) of taconite before replacement to almost 10 million tons (9 million t) in copper ore).
7. The crusher should operate economically and dependably and should yield a long service life (current primary crusher installations in the mining industry are expected to operate from 20 to 30 years without needing replacement).
8. It must be backed by excellent maintenance services, as generally the entire throughput of a mine goes through one or two crushers, and the failure of these crushers can cause serious financial hardship to a mine.

Table 13.5.3. Advantages and Disadvantages of Specific Belt Fabrics and Construction

Advantages	Disadvantages
Synthetic Fabric	
1. Initial cost is generally lower than that of comparable belt systems. 2. The belt can be mechanically spliced when used at low tension. 3. Belts are stock units and are readily available. 4. Belts are adaptable to short conveyor systems.	1. Maximum strength is limited. 2. Large pulley diameters are required with more plies to handle the resulting stiffness of the belt. 3. Vulcanized splicing, which is required for heavier belts that cannot be spliced mechanically, cover large sections of the belt. 4. Thicker belts do not run easily in troughs. 5. Large amounts of take-up movement are required on long conveyors because fabrics tend to stretch. 6. Fabrics are susceptible to both crosswise and longitudinal ripping by foreign objects.
Steel Cable Belts	
1. Greater belt tensions are supported, permitting longer hauls with higher lifts. 2. Permanent steel cable splices that require no mechanical connection of cables and that can supplement a vulcanized rubber cover over the splice can be on site. 3. A very short take-up length is required as compared to fabric belts. 4. Belts are fabricated with steel cables under tension. 5. Cables are laid precisely in the same plane during fabrication and are controlled during curing, resulting in belt alignment superior to that of a fabric belt. 6. Steel cable belts have greater resistance to general impact and abuse.	1. Initial cost is generally much greater than that of a fabric belt where both have the same maximum tension rating. This generally applies only to the lower range of the steel cable belts. 2. Splicing a steel cable belt is more difficult and time consuming than splicing a fabric belt. 3. Smaller pulley diameters can be used with resulting savings in gear reductions and drives when compared to thick heavy fabric belts. 4. Belts are susceptible to lengthwise tearing.
Cable Belts	
1. Cables are easily inspected. 2. Because they are not connected to belts, off-center loading has little or no effect on the belt system. 3. The width of the belt is entirely determined by the capacity of the system. 4. End-drive cables are available in continuous lengths without separate splicing and so can be used in systems that are up to 10 miles (16 km) long. 5. Belts can go up and down over various landforms without major structural or conveying problems other than use of power. 6. There are few drive cable splices and a minimum of belt splices.	1. Cable belts are not readily movable and must be considered only for long fixed haulage distances.

13.5.5.2 Conveyors

The size, weight, and physical characteristics of the material, transport rate, and horizontal and vertical distances the material must be carried determine the type of conveying system to be used to handle the material. Table 13.5.3 lists the advantages and disadvantages of specific belt fabrics and construction.

13.5.6 ECONOMIC CONSIDERATIONS

In open pit metal and nonmetal mines, the cost of transporting materials out of the pit constitutes roughly 60% of the operating cost of mining. In many mines, this cost percentage is even greater. Consequently, haulage costs are the major factor influencing the economic viability of a mining operation and constitute a major opportunity to reduce costs and effect savings.

Because of their operating flexibility, relatively low capital cost, resale value, and mobility from operation to operation, trucks have historically been the favored method of moving both ore and waste from open pit mines. Technical developments, such as diesel engines and electric wheel drives, have led to larger and larger vehicles. Currently, several manufacturers produce 240-ton (216-t) trucks, and the largest truck in operation is a 350-ton (315-t) unit. Continuing this trend will probably result in standard units of 300 tons (270 t) within 15 or 20 years. The use of microprocessors to monitor, control, and dispatch vehicles has resulted in improved efficiency and increased productivity in recent years, a development that should continue as computer technology expands in the future.

Within the current state of the art, conveyors are the lowest-cost method of handling bulk materials. For truck haulage, 60% of the fuel energy goes to moving the truck weight and only 40% to moving the payload. For belt haulage, the corresponding relationship is 20% to belt weight and 80% to payload. Using diesel fuel costs of $1.14/gal ($0.30/L) and electricity costs of $0.05/kWh, energy costs favor conveyor haulage by a factor of 4 to 1.

In addition, conveyors provide automatic, instantaneous start-up and continuous operation. They are very reliable, achieving up to 90 to 95% availability. Their operation is not impaired by bad weather, which often halts truck haulage. Conveyors have lower manpower requirements; a 100-person crew operating and maintaining a truck fleet can be replaced by a 10-person crew handling the equivalent amount of materials via conveyor. Conveyors also have a significant potential to be automated in varying degrees.

Belt conveyors operate efficiently at grades up to 30%, while the maximum grades over which trucks and rail haulage can operate are 10 and 2%, respectively. The capability of using steeper grades lowers the need to remove overburden and establish haulage roads, thus improving the operating ore-to-overburden ratio and reducing costs.

Disadvantages of belt conveyors include the lack of operating flexibility at the loading area. Initial capital costs can be higher, but vary depending upon the size of the operation. A deeper, larger pit normally requires more capital for additional trucks relative to extending a conveyor system. For a 656-ft- (200-m-) deep pit, investment costs for trucks might be about the same as for conveyors. However, it has been reported in European surface mines and elsewhere that conveyor systems actually require lower initial costs than truck systems.

Within four or five years, however, higher operating costs cancel the initial cost advantage of truck haulage. The longer the life of the project, the more economical the conveyor system, especially in deep pits or pits that rapidly increase in depth. Thus, taking into account the necessity of purchasing additional haulage trucks to accommodate increasingly difficult haulage routes and to replace trucks as they wear out, conveyor systems will actually require lower capital costs over the life of a mine. Conveyor systems handling ore in numerous large crushing and port facilities, which have operated since the early 1950s, have clearly demonstrated a useful conveyor life of more than 25 years. In contrast, off-highway trucks have life spans of six to eight years.

Data generated from several large mines where in-pit crushing and conveying systems have been installed indicate lower operating, maintenance, and overall unit costs compared to the costs of conventional truck haulage. Productivity is increased with reduced truck fleet requirements and shorter truck cycle times. As pits deepen and hauls lengthen, in-pit crushing and conveying systems become even more economically attractive, since conventional truck haulage costs increase dramatically with pit enlargement. Consequently, the economics of belt conveying are particularly attractive in large-volume operations, in those where steep climbs are involved, and in those where haulage distances are longer than a few miles (kilometers).

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Chapter 13.6
FUTURE CONCEPTS IN SURFACE MINING

THOMAS ATKINSON

13.6.1 INTRODUCTION

Many surface mines have depleted reserves of shallow, high-grade ores in simple geologic settings. This will inevitably require mining companies to consider deeper deposits of greater complexity, with more critical groundwater conditions, increased geotechnical problems, greater stripping ratios and waste volumes, which are further from markets. While underground mining has made considerable progress in improving productivity over the past 10 years, however, surface mining has not matched these improvements (Atkinson, Burdon, and Longworth, 1985). During this period, two major factors have mitigated against surface mining:

1. The depletion of economic reserves near the centers of population and the large, extremely expensive infrastructures required for remotely located mines may cost many times that of establishing the mine itself. This dictates the need to achieve economies of scale, which have been pushed to the limits over the past three decades.

2. The activities of the environmentalist/conservationist lobbies.

In the past, as economic conditions deteriorated, the problems were reduced by (1) increasing equipment size to achieve economies of scale and (2) productivity gains achieved through increased efficiency. Neither of these solutions may remain available today since equipment sizes appear to have reached a plateau in the early 1970s, and only very marginal improvements in efficiency are available when full capacity using conventional systems is possible.

For smaller, short-life operations, major reductions in infrastructure costs may be achieved by the "fly in/fly out" of personnel, a well-established technique in the oil industry. The economics of this operation has yet to be tested for large-volume production mines.

This chapter explores possible concepts and trends for the future in surface mining. For other views, applicable to various forms of mining, see Section 22.

13.6.2 TOWARD CONTINUOUS MATERIALS HANDLING SYSTEMS

The disproportionate increases in price of petroleum-based products compared with the general rate of inflation since the OPEC action of 1973 have forced mine operators to consider continuous systems (usually with belt conveyor transport) to take advantage of cheaper electrical energy generated from other fuels, for example, coal. Truck transport systems are generally favored by mining engineers because of their inherent flexibility in operation, but in terms of energy efficiency, trucks utilize only about 40% of the energy consumed for payload, the remaining 60% being required to haul the tare weight of the truck. Belt conveyors use up to 80% of the energy consumed for transporting the payload.

Although oil prices were slashed in 1986, the energy costs for trucks in many parts of the world still remain in the region of 50% greater than for conveyors on level grades and 300% greater on steep grades. Belt conveyor systems are less labor intensive, relatively simple in design and operation, reducing the need for highly skilled labor, the cost of which tends to inflate at a rate greater than the general rate of inflation. Major operating cost savings are possible with belt conveyor systems compared with truck haulage, and the recent considerable interest in continuous systems is a result of these potential savings (Atkinson, 1985; Brealey and Atkinson, 1968) and may be summarized as follows:

1. Low operating costs, low manpower requirements, and long-life equipment (i.e., relatively inflation proof).

2. The need for buffer storage, etc., between individual operations can be almost eliminated.

3. Peaks of electrical demand, mechanical stress, materials flow, etc., are considerably reduced.

4. Lower specific energy requirements (hp/ton-mile or kWh/t-km).

Major disadvantages are

1. High capital cost.

2. If one component of the system is defective, the total system must stop (although availabilities of about 80% can be achieved).

3. Best suited to large-volume, long-haul operations.

13.6.2.1 Continuous Excavators

For many years, large, full-face tunneling machines have excavated medium-to-strong rocks using disk cutters. The first Robbins Mobile miner, fitted with disk cutters and now in early operation, has successfully cut extremely strong quartzite [40,000-psi (275-MPa) uniaxial compressive strength (UCS), with up to 93% free silica]. For most full-face tunneling machines, it is essential to provide rigid constraint to contain the reaction to cutting. In the surface mining situation, the cutting forces are transferred through the machine to its base. The machine must have the right combination of geometry and mass to contain these forces. This probably requires a different machine configuration (and probably a bench layout) from that of existing machines, with shorter booms, squatter profiles, increased crawler area, and possibly a limit on digging height. Additionally, the machine must handle larger volumes than tunneling machines. The bucket wheel excavator (BWE), and possibly the placer mining dredge, is the only machine currently excavating at the necessary rate. The BWE is the most effective machine for mining large outputs in weak ground. Its application is extending into stronger formations (e.g., Neyveli, India, and Athabasca, Canada), but the BWE's marginal performance near the limit of its application zone indicates that far more in-depth investigation is required in this area.

Surface mines encounter a range of weak-to-strong materials. The rotating bucket arrangement of the BWE, which can efficiently clear large volumes of cuttings for loading onto a belt conveyor, unfortunately does not perform well in more competent ground. Roadheader-type machines perform well in weak to medium-strength rocks (up to 14,500 psi or 100 MPa UCS) and give reasonable service in "soapy" materials, because the pick peripheral speeds are sufficiently high to keep the cut-

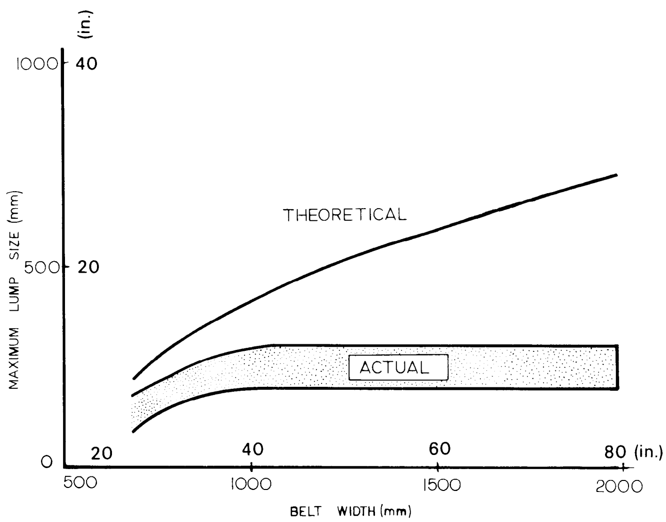

Fig. 13.6.1. Maximum lump size for belt conveyors.

terhead free. To meet the demands of surface mining duties, new designs are necessary to clear the large volume of cuttings from the cutterhead and load them onto a continuous transport system. Disk cutters may not excavate well in weak ground, such as engineering soils (particularly if clay-rich), but they give excellent performance in medium to strong rocks. Again new designs are needed to transfer the large volumes of cuttings to a continuous transport system.

These observations point towards a modified BWE configuration with driven precutters (e.g., disk cutters or roadheader-type cutters) in front of each bucket. Compensation for differing angles of attack due to changes in slew speed would probably be necessary but should not be an insurmountable problem. *It should be noted, however, that abrasion is of much greater importance for continuous excavations than for single-bucket excavators, and much more detailed study of the properties of rocks to be excavated is necessary* (Atkinson and Cassapi, 1984). The research and development of such a machine should be one of concerted effort over the next five to ten years.

13.6.2.2 In-Pit Crushing

Because of the absence of a suitable continuous excavator, in-pit crushing systems have attracted considerable attention in order to obtain the benefits of belt conveyor systems (Chapter 13.5) (Atkinson, 1985; Anon., 1982). A blasting/excavator/crusher/belt conveyor operation must work as a system, the excavator dipper (or bucket) being used so it can handle only lump sizes that will pass into the crusher. To avoid excessive secondary breaking, the blast design must ensure adequate fragmentation. Similarly, the crusher must reduce the feed to a lump size that can be handled by the conveyor belt widths. Fig. 13.6.1 shows theoretical and practical lump sizes for various conveyor belt widths.

The application range of various crusher types appears in Table 13.6.1.

Gyratory crushers can handle strong, abrasive rocks but are relatively heavy, incur high capital costs, and have a high feed height. At the other end of the range, roll crushers are relatively light, have a low feed height, a reduction ratio of 5:1 (usually adequate for feeding belt conveyors), can handle weak or sticky materials, and have low capital cost. They experience high wear rate for materials containing more than 4 to 5% free silica. In many situations, the ideal crusher must have a low feed height and low capital cost and, in general, be able to handle medium to strong, abrasive rocks; that is, it should combine the advantages of the gyratory and roll crushers but not have the disadvantages of either. The immediate thrust must be to reduce the size and cost of crushers (reduced size will also increase mobility). The MMD crusher has interesting possibilities (Potts, 1986).

The second approach must be directed towards improving the energy consumption of crushers, especially as large volumes of waste must be crushed, which represents an additional energy cost for which no additional revenue is generated. This requires a much better specification of the crushing requirements and an improved knowledge of rock material behavior during the crushing process. The use of ultrasonics in grinding provides spectacular reductions in energy requirements. A similar approach to crushing practice (including microwaves) may produce more economic operation. If this method is successful, it would have a major impact on the whole field of comminution. The long-term future may be in a new technique such as ultrasonics rather than in mechanical crushing.

13.6.2.3 Run-of-mine Rock Conveying

A recent US Bureau of Mines (USBM) study (Martin, Noble, and Derby, 1984) considered the use of belt conveyor systems for *blasted rock without crushing* (termed "mine-run rock"). It was proposed to use a crib and cable design very similar to early cable belt conveyor systems. The main factor that limits lump size on conventional belt conveyors is collision impact between the large lumps on the moving belt and the fixed idlers, the impacts damaging both idlers and belt. The rope-driven, crib supported design eliminates this problem (Fig. 13.6.2). A further major problem is due to impacts and blocked chutes at transfer points. The effect of impacts can be greatly reduced by the use of closely spaced, pneumatic-tired, impact idlers under the belt, a grizzly that allows fines to fall on the belt partially absorbing the impact of large lumps, and finally a chute that accelerates the large lumps to greater than belt speed, in the direction required. The life of the relatively short drive ropes that contain a "long splice" with a short profile is an unknown.

A disadvantage of the design is the belt speed limitation of 500 to 700 fpm (2.5 to 3.5 m/s), compared with up to 1000 fpm (5 m/s) of conventional high-speed belt conveyors.

Inclinations up to 20° are possible using the crib and cable design, compared with 16 to 18° with conventional belt conveyors for similar materials. The vibration of a material created by the moving belt over fixed idlers and the up-and-down motion of the belt as it passes over the idlers are eliminated. Initial tests carried out by the USBM appear promising.

The particle-size distribution of mine-run rock is not readily available; furthermore, it changes as rock properties and blasting effectiveness change. Generally, the maximum chord length of the largest lumps does not exceed 5 ft (1.5 m). In some mines, as little as 2% is this large, while generally it does not exceed 10%. By scalping off the large lumps prior to loading onto a mine run rock conveyor, the crushing operation can be eliminated. The oversize material can be dozed together and loaded into a truck, or broken by a pile hammer and loaded onto the conveyor.

The particle-size distribution of mine-run rock in varying conditions warrants further study. One possible method of estimating the maximum chord length of material in a rock pile is by miniature aerial photography carried out by model aircraft

Table 13.6.1. Application Ranges of Crusher Types

Rock/Soil Code[1]	Description	Compressive psi × 10³	Strength MPa	Crusher Type[2] G	J	H	RC
R7	Extremely strong rock	> 30	> 200.0	X[3]	M		
R6	Very strong rock	15.0–30.0	100.0–200.0	X	X	M	
R5	Strong rock	7.5–15.0	50.0–100.0	X	X	X	M
R4	Moderately strong rock	2.0– 7.5	12.5– 50.0	X	X	X	X
R3	Moderately weak rock	0.7– 2.0	5.0– 12.5		X	X	X
R2	Weak rock	0.2– 0.7	1.25– 5.0			X	X
R1	Very weak rock	0.09– 0.2	0.60– 1.25				X
C4(G4)	Stiff (weakly cemented)	0.02–0.09	0.15– 0.60				X

1. Geological Society Engineering Geology Group Classification.
   R—Rock
   C—Cohesive Soil
   G—Granular Materials
2. Crusher types G—Gyratory, J—Jaw, H—Hammer or Impactor, RC—Roll.
3. Application
   X —Suitable
   M—Marginal

Note that other parameters must be used in crusher selection, notably abrasive wear.
Source: Atkinson, 1985.

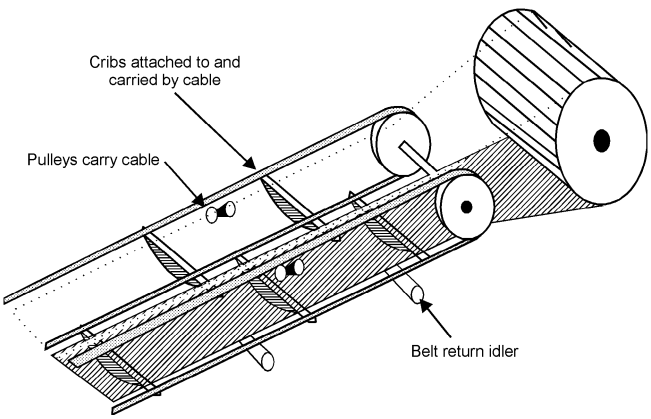

Fig. 13.6.2. Mine run rock conveyor: crib and cable design.

(Walton, 1985), the photographs being image analyzed so that blasting procedures can be modified as required.

13.6.2.4 High-Angle Conveyors

The elimination of truck haulage out of relatively deep pits, using conventional belt conveyor gradients, invariably requires conveyor ramps at much flatter angles than the pit slope angle. This results in additional excavations, often during the early life of the mine, which has an adverse effect on the discounted cash flow rate of return for the project (Atkinson, 1985). The conveyor ramp can also create sharp re-entrant angles, forming promontories into the pit and reducing pit wall stability in the conveyor ramp area. To overcome these problems, high-angle (HAC) (> 40°) and steep-angle conveyors (SAC) (18 to 40°) are in use and under development.

Several different belt systems have been proposed (Fig. 13.6.3), including (1) the "sandwich" belt, (2) corrugated "bellows" belt (e.g., "Flexowell"), and (3) the "pipe" belt. These may be installed on mobile or fixed installations. There are three basic mobile installations.

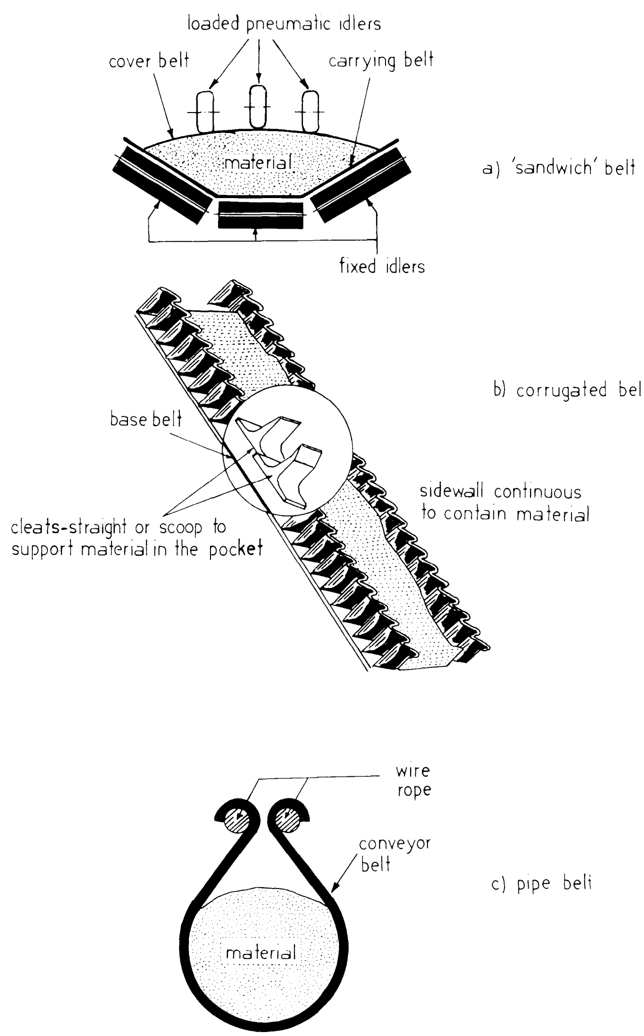

Fig. 13.6.3. Belt systems for high-angle conveying.

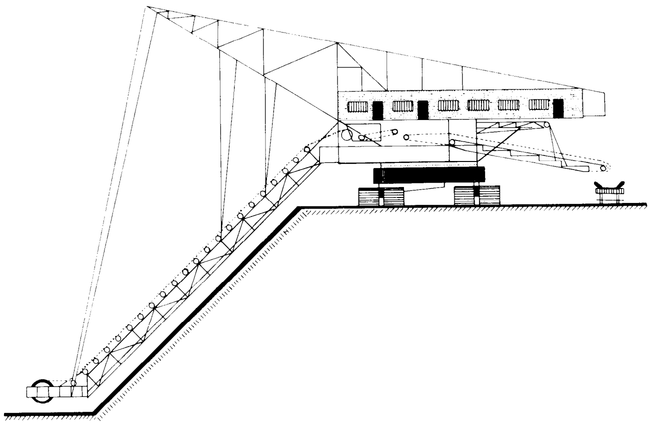

Fig. 13.6.4. High-angle conveyor: dredge type.

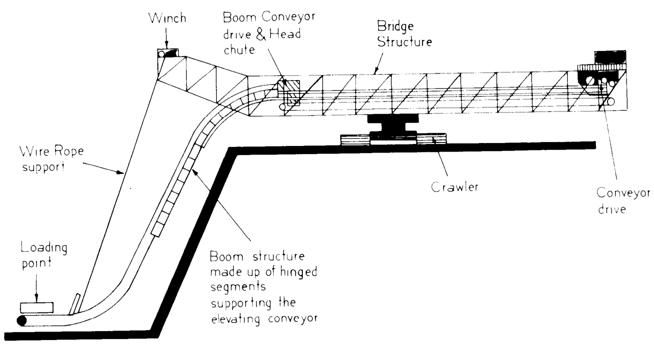

Fig. 13.6.5. High-angle conveyor: retractable boom type.

BENCH LIFT CONVEYORS. These conveyances are usually crawler-mounted bridges of triangular tubular section. Bench height adjustments are made by positioning the crawlers to change the inclination of the bridge. Special leveling devices are required to correct the inclination of the conveyor at the loading and discharge points. At least two manufacturers are developing this design probably for SAC, which is an extension of conventional bench lift conveyors (Brealey, 1963). The machine is relatively cheap to construct but is inflexible in operation.

DREDGE DESIGN. This is based on a bucket chain excavator configuration, that is, a crawler-mounted machine standing on top of the bench (Fig. 13.6.4), the boom being made of rigid linked sections. No slewing mechanism is being proposed to reduce cost. The boom can be raised to a horizontal position and the machine crawlers maneuvered to swing the boom over level ground for maintenance. It has the advantage of being more mobile than the bench lift conveyor; the boom angle can be altered to suit the bench slope, but it is of higher capital cost. This machine is being developed by a German manufacturer for vertical lifts up to 100 ft (30 m).

RETRACTABLE BOOM, MOBILE ELEVATOR. This is a recent Australian development using a segmented, hinged retractable boom (Fig. 13.6.5). It will be a crawler-mounted machine, standing on the top of the bench with the same advantages as the dredge design (James, 1985). It could be up to 50% lighter and of lower capital cost than other equivalent single-standing machines, with a much lower profile, reduced wind loads, greater stability, and faster relocation. The boom length is adjustable and is claimed to be capable of vertical lifts of 180 ft (55 m). Because the boom is retractable, maintenance work is greatly simplified. The boom length can be varied to suit the bench height and the boom angle varied to suit the bench angle.

The high capital costs of HAC and lack of experience in their use have been major deterrents to their adoption. HACs (up to 90°) are in use in other industries with vertical lifts of 300 ft (90 m). The newer developments should provide greater confidence in their use, for example, the "Hurise" conveyor.

13.6.2.5 Cross-pit Conveyors

Cross-pit conveyors can greatly reduce the haul distance of around-the-pit truck or belt conveyor routes in terrace mining operations. Depending on pit length, they can have lower capital cost than around-the-pit systems and invariably have lower operating costs. Bridge units (two sets of crawlers) are available from British and German manufacturers, while single-crawler, "spreader" types are available from German and US manufacturers. A spreader-type, cross-pit conveyor of 1000-ft (300-m) overall length and a 670-ft (200-m) discharge boom is scheduled to go into service in a Texas lignite mine.

Many mine operators are deterred by the presence of a conveyor structure crossing the benches of a mine since it requires more precise planning and scheduling, especially for bridge units that require a graded track on both sides of the pit. The advantages of the cross-pit conveyor are enumerated elsewhere (Collin and Waring, 1983).

13.6.3 DEVELOPMENTS AND TRENDS IN TRUCK HAULAGE

13.6.3.1 Conventional Trucks

From 1955 to 1975, substantial increases in mine truck size occurred due to the development of (1) economic, reliable, high-speed, lightweight diesel engines and (2) larger, long-life, high-load capacity tires. By the late 1970s, the haul-cost vs. truck-size curve had leveled out with an economic payload of 170 tons (150 t) or above (providing the mine could accommodate such truck sizes). The six major reasons inhibiting development of more economic, larger-capacity trucks are

1. Lack of suitable engines.
2. Cost of construction of haul roads (Atkinson and Walton, 1983), particularly on loading benches.
3. Truck maneuverability problems.
4. Maintenance problems due to access difficulties.
5. Poor potential cost savings, inhibiting development by manufacturers.
6. Problems in manufacturing, transporting, and handling larger tires.

The major routes for cost reduction are to achieve higher speeds with larger trucks fitted with higher power, GVW (gross vehicle weight) ratios (upgrade) and better braking, steering, and handling characteristics (downgrade), together with driverless (or semi-driverless operation), so that shift change, meal break, etc., delays are eliminated.

Developments beyond 170 tons (150 t) envisage the "power cube" engine, that is, a removable module engine of compact dimensions for easy removal and maintenance. Note that volume is far more important than mass as engine mass is only about 2% of GVW. Based on this concept (Atkinson, Burdon, and Longworth, 1985), Table 13.6.2 indicates future availability.

Mechanical transmissions will probably remain at 1600 hp (1190 kW) maximum, while two axle configurations with 2200-

Table 13.6.2. Future Availability of Truck Engines

Engine Power		Year	No. of
hp	kW	Available	Manufacturers
1343	1800	1985	4
1492	2000	1986	3
1615	2165	Available	1
1695	2272	1987	1
2014	2700	1987	1
2596	3480	?*	1

* Current turbo-charging technology applied to existing 16-cylinder engine.

hp (1640-kW) engines are available, and a three-axle, 3300-hp (2460-kW) system has been developed.

The largest tire in operation, 40.00 × 57, appears to be the upper size limit, and increased tire loadings are being sought. Radials are reported to be reducing fuel consumption, as well as increasing operating speeds, that is, conventional rather than off-highway development. Increased TMPH (ton-mile/hour) capacity is available from radials, but at increased first cost. One-piece, 15° rims, presently operating and now accepted in the United States are achieving cost reductions.

Disk brakes have virtually replaced drum brakes. Dry disks have the benefits of easy access, low cost, and cost-effective component life. Wet disks (oil-cooled) are fitted on one truck, using pump-circulated oil as the coolant, the rear brake pack operating as service brake and retarder. Long-term operating costs have yet to be established for comparison with the proven 20,000 to 50,000 hours achieved by dry disks.

Electric drives have the potential for adaption to monitoring and diagnostic assistance, minimizing down time and erroneous service calls. Electronic engine control is being developed to improve fuel efficiency, with reduced pollutants on starting and better emission control. For smaller trucks with mechanical transmissions, electronic control will produce a wider range of gear ratio combinations to maintain optimum engine outputs to suit haul road profile and conditions. Monitoring of payload and load distribution will possibly bring reduced operational problems.

13.6.3.2 Trolley-assist Truck Haulage

Although trolley-assist systems have been available for some years (Shevyakov, 1965; Atkinson, 1969), they have received most attention since the increase in oil prices in 1973. The main advantages of trolley-assist are

1. Reduced costs due to reduced fuel consumption. On adverse grades, electric power is used (except for the service loads on the truck).
2. Increased productivity due to increased speeds and reduced cycle time.
3. Reduced electric wheel armature currents allowing long hauls out of deeper pits.
4. Greatly increased engine life.

The South African operations—Sishen, Palabora, and Grootegeluk—employ trolley-assist systems. The full benefits of the system have probably yet to be achieved since full-capacity working has not been required, but other advantages, especially increased engine life, have been established.

Trolley-assisted haulage layout was discussed in Chapter 13.4.

13.6.3.3 Mammoth Trucks

A recent paper (Walker, 1985) advocated the development of even larger trucks. The paper analyzed the historical data of

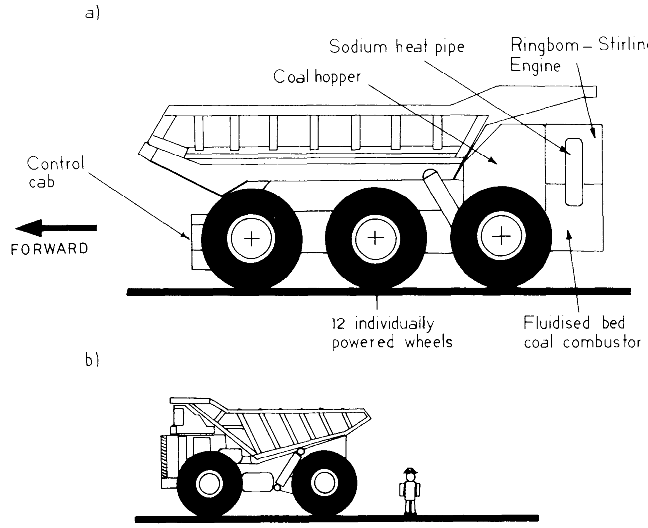

Fig. 13.6.6. Comparison between (a) projected 1000-ton (900-t) truck and (b) existing 200-ton (180-t) truck.

mine trucks to establish the power laws governing their past development, and using these laws, projected the geometry of 500-, 750-, and 1000-ton (450-, 670-, and 900-t) trucks (Fig. 13.6.6). The projections are not impractical. Diesel oil will again be expensive and in short supply in the future. The use of steam engines and coal-fired diesel engines has been advocated previously (Atkinson, 1982), but here the alternative of the external combustion engine (the Stirling engine) is suggested. Microprocessor control with driverless operation is considered feasible, the unmanned vehicles operating under the direction of a central computer. If the development from 30 to 300 tons (27 to 270 t) over 30 years is considered, the 1000-ton (900-t) truck could be a reality by the end of the century.

13.6.4 ROCK FRAGMENTATION

13.6.4.1 Blasthole Drilling

Rotary blasthole rigs have changed little since their introduction. Drillhole sizes up to 15 in. (381 mm) in diameter are most popular, and it is unlikely that hole diameters, say, up to 17½ in. (445 mm) will give adequate blast control or be cost effective. Deeper holes are attracting interest (200 to 250 ft, or 60 to 75 m) requiring longer masts, crawlers, and drill pipe. Angled drillholes can be profitable, and the ability to angle the crawlers to correctly orientate the rig to the bench face at each hole is essential. Greater inclination of the hole is desirable, but this creates problems due to cuttings packing at the lower side of the hole and some form of secondary flushing is indicated. Additionally, control of drill-pipe deviation is needed. Recovery of data from the drilling operation can be used to improve blasting effectiveness, reduce costs, and improve productivity. Tailored blasting, a more innovative approach to ground preparation, involving decked charges, emulsion explosive water stoppers, phased initiation, etc., is currently available but requires better information from borehole logging.

The rig of the future will have an automated drilling package (Waller, Ambrose, and Casappi, 1987) that will

1. Continuously perturb all the drilling parameters (i.e., thrust, peripheral speed, flushing rate) and select the best combi-

nation of parameters to maximize penetration rate, with minimum bit wear. The major obstacle is measurement of bit wear while in the hole.

2. These changes will be continuously monitored to provide blast design information.

3. Deviation of the drill from the desired direction will be monitored and the thrust altered to correct the deviation.

4. The crawlers will be programmed to locate and orientate the rig at each blasthole, using a gyro.

5. The hole locations, borehole inclinations, etc., will be logged so that each blast can be back analyzed.

13.6.4.2 Explosives

The effect of fragmentation and rock-pile profile on excavator performance and overall cost per ton have previously received inadequate attention (Winzer, Anderson, and Ritter, 1983). There are opportunities to use untapped, present-day technological advances. The present state of the art employs:

1. Bulk ANFO (plus water gels or emulsions in wet holes).
2. V1 sequencing with surface delays.
3. Equilateral triangle blasthole patterns, for optimal explosives distribution.

The following available techniques have the potential to significantly reduce costs:

1. Dense ANFO for increased energy and reduced drilling costs, particularly in stronger rock (Harries and Grieve, 1978).
2. Pre- and post-splitting for more economic blasting and improved slope stability.
3. In-hole delays for optimizing blasting, improving blast design, and tailoring blasts.
4. Cast blasting to exploit the cheap chemical energy of explosives compared with conventional excavation and transport (Chapter 13.2.3) (Atkinson, 1983).

The future indicates the need for further blasting research to incorporate the quality of the rock mass gained from better monitoring, blasting analysis [e.g., fragmentation and burden movement (Winzer, Anderson and Ritter, 1983)], and the use of computer modeling for the prediction of the effects of changes in blast parameters.

13.6.5 AUXILIARY AND SUPPORT DEVELOPMENTS

13.6.5.1 Electrical Power Systems

The economy of size and great flexibility of electrical drive systems has lead to their widespread use in the surface mine plant. They also have the advantages of reduced maintenance compared with other drives, and they are competitive in energy costs. Despite these apparent advantages, there is still room for improvement in their design and in their cost of operation.

Electricity costs have reached a level where they are a major component of the operating cost in electrically powered, mechanized mining, and it is therefore, well worth pursuing methods of reducing this cost. Whether electrical power is generated on site or obtained from a utility supply company, there are two cost components: a maximum demand cost and a unit cost. The unit cost is based on the quantity of electricity consumed, usually measured in kilowatt-hours (kWh). To minimize this cost, the consumer should be concerned with the efficiency of utilization of electrical energy and the potential advantages of using low-loss machines and controllers. The pay-back period of the difference in cost between normal and low-loss motors is often less than 1 year.

The maximum demand, measured in kilovolt-amperes (kVA), is used to determine the maximum capacity and, therefore, the cost of the supply system. Mine supplies experience large fluctuation in load, mainly as a consequence of cyclic operations that are typical of draglines and shovels. Devices are currently available to continuously monitor the maximum demand and, if necessary, limit it by shedding load by the switching off or the current limiting of plant. Automatic maximum demand control is not yet in widespread use, but there are many operations that could benefit from its application. In the future, individual large plant items should be fitted with maximum-demand monitors, data from which can be used to minimize peaks. Centralized control can de-synchronize operations avoiding simultaneous peak demands from a number of cyclic operations.

During design, it is important to adopt a systems approach to the whole of the drive, from the driven load right back to the point of supply. Dynamic models of induction motors have been designed, and complete computer-aided design (CAD) of electromechanical drives is used in other industries. It is also important to match the power ratings and speeds of multi-drive plants to ensure that the over- or under- capacity of one drive does not prevent the full utilization of the other drives.

For the overall design of the power distribution system, both the static and dynamic performance may be checked by using load flow models (Trutt and Thomas, 1981). Where static controlled variable speed drives or non-linear loads are used, then the effects of harmonic pollution and the desirability of filtering should be checked (Stratford, 1980). It is important that the operational data used to develop models are based as far as possible, on practical operating experience and not on idealized cycles that are only rarely achieved.

Variable speed drives are in common use in surface mines. In the future, most will be based on solid state or static control systems. The trend will be away from direct-current machines towards synchronous and induction machines driven via variable frequency and voltage invertors. For low speed or at very high power, the cycloconverter is preferred, but continued improvements in semiconductors—in particular high-speed, high-power switching transistors and gate turn-off thyristors—will extend the range of application of link converters. In addition to increasing power capacity, improvements in controller performance can also be anticipated with faster responses, improved reliability, extensive condition monitoring and self-diagnostics, reduced harmonic pollution, and maximum demand control. The greatest advances will come through the use of automatic control, most probably based on microprocessors and including closed loop, sequencing, and optimal and intelligent knowledge-based control systems.

As a consequence of the high cost of small-scale power generation, it has only been used for emergency power, peak lopping, supply stiffening, or in locations too remote for connection to utility supplies. The efficiency of the small-scale generation plant has improved, and this, together with waste heat unitization, is making local total energy programs attractive.

13.6.5.2 Electronics

Rapid advances in the capabilities and applications of electronic systems during the past three decades have already had a significant impact on the industry. As yet, there are no signs of any slowing in the rate of advance of electronic technology, and we can anticipate continuing improvements for some time to come. Despite its many successes, however, the rate of application of this technology in the mining industry has been retarded by some early failures.

The greatest advances made in the field of electronics are those resulting in simplification. Simplification reduces the burden of design work and improves reliability. Although systems may be complex, they are built up from a network of individual stages. Without simplicity of design at each stage, the feasibility of complex systems is destroyed. Greater emphasis can now be placed on the whole system performance and on its relationship with the application and the user.

One of the major fields of application for electronics in mining is in operational monitoring and control (Chapter 12.6). This involves the use of a measuring system to monitor critical operating parameters and relay this information to the user. The user may be an individual or an automatic control system. In either case, the measured parameter is compared with a desired value and the difference, or error, is used to modify the operation in order to minimize the error. This forms the basis of *control engineering,* another enabling technology, the full potential of which has not yet been realized. There are a number of reasons for this, some technical and some human.

The most severe technical limitation to our ability to control operations in mining is the availability of transducers to convert operating parameters into measurable signals. Often that transducer is required to operate at the "front end" of the process under severe environmental conditions and, as a consequence, suffers from poor reliability. These problems have long been recognized, and investment in research and development has yielded some suitable devices. The solution has often been to derive the parameter via an indirect measurement. This has enabled either the transducer to be moved to a more protected environment or for an unmeasurable parameter to be derived by its effect on some other measurable parameter(s). There is still plenty of scope for development in transducer technology, both in traditional techniques and in the utilization of optical, acoustic, chemical, and bacteriological techniques.

Since the majority of electronic systems into which these transducers will feed will be digital, then wherever possible, digital transducer techniques should be developed and utilized.

Much of current control engineering has evolved from the analysis and design of simple positional and speed control systems. In this situation, it is not difficult to formulate equations describing the precise response of the system and the control actions required to obtain the desired performance. In the real world of mining, the processes involved do not readily yield to rigorous analysis. Despite the opinion of some control engineers, it is not necessary to obtain a rigorous mathematical model of a process as a prerequisite to satisfactory control. There are many applications for which an approximation for the system is quite sufficient to get a control scheme started. Fine tuning and adaptation can then be used to optimize the response.

Often as a precursor to a full automatic control system, a scheme of data acquisition is used to gather operating data. The specification of the system must cover those parameters that are critical to the operation (Anon., 1983). There is a great temptation to collect data simply because the technology to do so is available. Data acquisition costs money and must be fully justified and followed up with an effective strategy for data utilization (Harrison, 1978).

Improvements in electronics technology have had impact in other areas, principally in digital electronics and computing, areas that are discussed separately in the following.

13.6.5.3 Image Analysis

Vision is one of the most advanced and complex of man's senses. Those of us who enjoy adequate vision are capable of a vast range of analytical observations. Some of the most basic functions of vision are to recognize shape, outline, color, and/or intensity and movement. Image analyzers are a rudimentary electronic equivalent to vision. They are developing rapidly and, in some situations, can replace human observers, frequently with dramatic improvement in performance.

Television images are familiar to most of us. The normal TV picture is made up of 625 horizontal lines with the picture refreshed at 25 times/sec. Image analysis systems further subdivide the horizontal lines so that the final picture is made up from a matrix of individual points called pixels. The intensity of each pixel is then measured and assigned a gray scale value. In color systems, the process is repeated for three separate primary color frames. Grouping of pixels with similar gray scale values, counting and determination of orientation enables the recognition of shapes and objects and the measurement of areas and a range of other tasks.

The technique has already been applied successfully to mineral identification tasks in both geologic and mineral processing applications. In the future, we can anticipate a wider spread of new applications for these systems in operations where vision and visual identification of features is important.

APPLICATIONS. Application of image analysis techniques to various aspects of surface mining operations will occur over the next few decades. The advent of this technology results in the need to define areas of possible application. A number of areas would appear to offer promise, all of which could broadly be defined as *automatic real-time surveying.*

This includes the use of multiple-scanning video cameras to provide images of the pit from a number of positions. These images could then be processed and the position of particular features identified from each of the video signals. Precise measurement of the camera orientation and magnification would provide data that would allow the calculation of three-dimensional coordinates of features of interest (using intersection techniques). "Targetless" surveying using similar principles but without image analysis and employing laser beams is already available (Anon., 1985).

Thus a single system would be capable of comparing current images with previously stored images to provide real-time information for the following:

1. Production volumetrics.
2. Slope stability monitoring.
3. Equipment positioning (possible links with automated dispatching).
4. Geologic data collection.
5. Blast performance monitoring.

Constraints on this type of system being developed in the near future are

1. Limitations of current recognition techniques to regular shapes.
2. Insufficient processing speed and capacity.
3. Recognition of the implication of image changes.

As images analysis techniques develop, however, and the benefits of parallel processing and artificial intelligence start to be applied, there would appear to be no reason why this type of system could not be introduced as a means of drastically reducing surveying costs while providing management with the most up-to-date operational information.

Some of the other potential operational applications of image analysis are the following.

Manless Operation—Positioning of the bucket of shovels, bucket wheel excavators, or draglines is controlled by the operator who relies entirely on visual identification of the desired digging position. Vision is also used to locate and maneuver the bucket to bring it to this position. Image analysis could be used to identify the broken material and blocks in the rock pile or to delineate the target material and waste and then guide the bucket to the optimum position.

A similar situation exists in truck haulage, with visual identification of the desired road position and visual feedback used to assist in maintaining the truck in that position. There are other potential truck guidance techniques that might prove easier to implement, but image analysis is one of the few techniques able to identify obstacles in the truck path.

Closed-circuit TV has already been used to provide blind-side vision for BWE operators and rear vision for truck drivers. Accidents during truck reversing are still too frequent, and in addition to TV, radar and ultrasonic image techniques have been used to give warning of obstacles.

Fly Rock Analysis—Conventional TV cameras do not respond well to fast-moving objects. Recently developed charge-coupled device (CCD) cameras have overcome this problem, and high-speed imaging is now possible. High-speed video used as input to an image analyzer could enable automatic fly-rock analysis.

The input to an image analyzer is not restricted to conventional or CCD cameras using visible light; ultraviolet and infra red imaging is also used. For precision work, scanning laser imaging could be used, this carries the additional benefit of distance measuring capability. The scope for real-time photogrammetry has yet to be explored.

13.6.5.4 Ultrasonics

Sonics or sound waves and vibrations can be used for detection and measurements in two main modes: in a passive or listening mode by detecting stress-induced noise, or in an active mode using an artificial source and detecting reflections and/or propagation velocities.

There are two main current applications for passive mode sonics. The first involves periodic monitoring of the vibrations induced in machinery: increasing levels are frequently a precursor to failure. By measurement of the frequency or spectral content, it is often possible to identify which component is failing. The technique has been known for more than two decades but is only now beginning to find acceptance in the industry. In the past, the time taken in measurement and data reduction has deterred from its general use. Modern microprocessor-based portable instruments will do much to alleviate this situation.

The second passive application is in the monitoring of microseismic activity (Chapter 10.3). Small acoustic or ultrasonic pulses (microseismic pulses) are emitted from materials during fracture propagation. Brittle materials such as rock are particularly good emitters of microseismic pulses. The production of microseismic events commences long before total failure, and by measurement of their rate of production, it is sometimes possible to predict failure. Much work has been done in South Africa and the United States on rockburst prediction from microseismic activity (Brink, 1985). The work involves not just the counting of event rates but also the location of each event, as well as measurement of its spectral content and the event's total energy. The technique has also been used with great success for monitoring the growth of hydraulic fractures in a dry-rock geothermal project (Baria, Hearn, and Batchelor, 1985). The technique could be adapted to monitor slope failure and might yet find application in surface mining.

In current applications, sonics have been used to investigate materials properties from the macroscopic, in global-scale seismology, down to the microscopic in detecting flaws and fractures of the order 10^{-5} ft (5×10^{-6} m) in engineering components. Its range of application is staggering, from measurements of rising magma chambers at Mount Washington to the measurement of blood flow in fetal arteries. With high-intensity sources, sonic energy can be used for agitation, homogenization, and cleaning, and at very high intensity, dislocation and material fracture can be induced. Ultrasonics are already in routine use for "stone" breaking—albeit gallstones, with logical progression to rock breaking. The energy consumed in rock breakers far exceeds the specific energy required to fracture the rock. For small particle sizes, in-slurry ultrasonic grinding might be used, while at larger sizes, ultrasonic aided mechanical crushing could be used to improve energy efficiency. In-pit crushing with continuous transport could benefit considerably.

High-energy ultrasonics could also be used as an aid in drilling operations by assisting in breaking out of chips, thereby increasing penetration rate. In conventional core drilling, one of the major problems, particularly when drilling heavily fractured material is core (barrel) blockage. This results in additional tripping and delay while blockages are cleared. By vibrating the barrel using an ultrasonic device, it should be possible to reduce blockages thereby reducing drilling cost.

Surface seismic survey are used routinely for the delineation of geologic structure. Surveys also may be made from borehole to surface or between boreholes. By measurement of the velocity of propagation, it is also possible to derive some of the mechanical properties of the target material, for example, the frequency and orientation of fractures and the elastic constants. More recently, in-seam seismics (Mason, Buchanan, and Booer, 1980) have been used to locate discontinuities in coal seams. The ability to predict changes in the structure of the target mineral is of great value in the planning of production. These techniques will find wider application in surface mining in the near future.

Ultrasonics are well established in the field of nondestructive testing (NDT) of engineering components. Routine testing gives an early indication of fatigue failure or of faulty manufacture. Adaptations of the acoustic camera, as used in medical scanning, can produce 3-D images of internal structure. Probing of the internal structure and microstructure of mineral deposits will be of great assistance in mine and processing plant design and in the assessment of abrasivity.

13.6.5.5 Laser

At this point, we enter the realm of speculation by exploring the potential of the laser. The laser is simply a device capable of producing narrow beams of short wave length, coherent radiation. Larger versions can be used to focus high power into small areas. For a long time, the laser has been dubbed as the solution looking for a problem, but it is now coming of age.

Low-power lasers have been used in surveying work for over a decade, in particular, for alignment and distance measuring (Chapter 8.2). Machine guidance systems have also been developed for underground and surface mining machines. Harrison and Thompson (Harrison and Thompson, 1982) have described a laser-based system for maintaining bench levels in open pit operations. A fixed laser rotating in the horizontal plane is set up to establish the required bench level. A vertical rod fixed to the BWE's boom is equipped with three arrays of sensors that detect whether the boom is above, below, or on the desired level. An indicator in the operators cab gives a clear indication of the position enabling the operator to take corrective action. The advantages claimed of the system include increased productivity, improved drainage, less operations wear and tear on the BWE and conveyors, and a payback period of only a few weeks. An underground system for laser-assisted alignment of roadheading machines has been developed, a system that includes automatic profile control (Anon., 1986).

In open pit operations, items of plant fitted with retro-reflectors could be located by a scanning laser. The ability to continuously locate plant is of great importance for effective scheduling and dispatching. The system could also be used to position drilling rigs and excavators. If sufficiently high resolution can be obtained, then the system could be used to monitor slope stability. Electronic total stations currently are capable of these functions but require manual operation; automation is the next step.

High-powered lasers have been used for two decades in cutting and welding operations, particularly with difficult materials. The feasibility of cutting rock has been demonstrated on numerous occasions. Debris removal remains a problem; fluid jets perhaps with the addition of ultrasonic energy might yet provide a solution. The very high temperatures reached during laser cutting glassifies the surrounding material, leaving a smooth and sealed profile, ideal in most mining applications. Laser-profiled holes, which do not even need to be round, would be of great help in maintaining dry holes open in water-bearing or weak strata.

13.6.5.6 Radiowave Techniques

Radiowaves can be used in two main modes: for the conveyance of information of the sensing of electrical characteristics of the surrounding media.

The first mode familiar to everyone is through television. In surface mining operations, the value of radio voice communications has long been appreciated (Chapter 12.6). The quality of voice communication equipment has improved considerably, mainly as a consequence of the development of military equipment. There is still room for improvement in its cost. Radiowave communication is not restricted to voice—digital data is ideal for transmission. Indeed, early radio communication was entirely digital and used Morse code. Data from shovels, trucks, and other mobile equipment can easily be sent over a radio network link and the data used for real-time adaptive scheduling, maintenance, and production monitoring.

Electromagnetic induction techniques may be used in both monitoring and communication. Voice communication using low-frequency induction transmitters and receivers has not yet realized its full potential in underground mining. Low frequency is used since the energy passes through the rock. Development in this field continues, and improvements in the performance and scope of new electronic devices will do much to accelerate the pace. Voice communication in underground, combined underground and surface, and surface mines with difficult high-frequency radio propagation characteristics could all benefit from the use of low-frequency radio communication.

Induction techniques are well established in geophysical exploration. Adaptations of the techniques can be used to detect moving objects or the proximity of metal objects. This has been utilized in the detection of the entry and exit of vehicles from zones within an open pit operation, and by fitting each vehicle with a coding device, individual vehicles can be identified. As an alternative, each vehicle could be fitted with a radio beam that could then be located using scanning direction finding. The same beacon could be used for voice and data transmission.

High-intensity radiowaves can also be used for rock breaking. There are two basic mechanisms involved. The conversion between electrical and mechanical energy can be made in a number of ways, one of which is through the piezoelectric effect. In materials exhibiting the piezoelectric effect, changes in mechanical stress result in an electrical stress or voltage, and vice versa. Many natural materials exhibit some piezoelectric effect, crystalline quartz being one of the best. Strange electromagnetic effects have been reported during the major stress relaxations from earthquakes. It may prove possible to monitor microseismic activity by monitoring the electrical pulses induced in the ground by the piezoelectric effect during microseismic events. The reverse process might be utilized; by the application of high energy electrical pulses, it may be possible to induce sufficient stress to cause failure. Rock failure may be also caused through thermal stress. Electrical impulse or induction heating causes both direct thermal stress and vaporization of interstitial moisture. The resulting weakening and failure could be of use in drilling and grinding operations.

13.6.5.7 Computer Technology

Many of the technologies described elsewhere in this chapter are dependent for their successful exploitation on the availability of cheap, rapid, and powerful computer processing. For discussion of computer methods in general, see Chapter 8.4.

The dramatic rate of growth in computing power per unit cost over the past 10 years seems certain to carry on and may even increase in the near future. The development of the so-called fifth generation of computers, linked to research in artificial intelligence (13.6.5.8), is well advanced.

The new generation computers will employ parallel-processing architecture instead of conventional serial-processing. Parallel-processing subdivides complex tasks into many simpler tasks, each of which is handled concurrently and independently, thus allowing the whole problem to be solved rapidly. Once this technology is developed, performance increases (in terms of processing speed and capacity) of the order of 10,000 times, are expected in the next decade. At the same time, computer hardware will become more compact and rugged. Hardware constraints are likely to be reduced to minimal levels in the next decade or so.

As hardware costs reduce, there is a tendency for computing power to decentralize. This will continue to occur as individual pieces of equipment are provided with the processing required to make intelligent decisions (e.g., each drill, excavator, truck, pump, etc. will have its own processor). Improvements in communications and networking and significant reductions in the real cost of data transfer have already contributed to this trend.

As the performance per unit cost increases dramatically, it seems probable that the limiting factor in the development of computer systems will continue to be software costs. As systems become more intelligent, they are able to program themselves; hence current shortages of applicable software may be a manifestation of “dumb” computing systems.

The computer will become easier to use as developments in the following areas occur:

1. Natural programming languages.
2. Voice recognition.
3. Graphical interaction (3-D holography).
4. Communications and networking.

The techniques that have improved our ability to measure the activities involved in a mining operation have already been discussed. The main feature of the increase in our capability to measure is that the volume and, hopefully, the quality, of the data at our disposal is greatly increased. However, in many instances, certain parameters have been monitored simply because we have the ability to do so and on the basis that the data might prove useful later.

Our current ability to collect, transmit, store, process, and display these volumes of data has been made possible by the digital computer. Current systems are based on a central computer to which all information is directed via outstations. The information is then available for display at a control console via visual displays and indicators. Control actions, which are generated manually at the control disk or automatically by software in the control computer, are then routed back to the appropriate plant again via the outstations. The trend now, and into the future, will be to make more of the control actions automatic and to distribute the computing power to the level of the outstation and the individual plant (Harrison, 1978). This will reduce the demands placed on the operator, central computer, and communication network and will distribute the software development burden.

Performance monitoring systems have already been applied to excavating and transport systems in surface mines. The Bucyrus-Erie BEPAL (Matuszak and Radomilovich, 1984), General Electric DIGMATE (Ivanco and Stobbelar, 1984), and the BHP Engineering DRAGLINE PERFORMANCE MONITOR (DPM) (Godfrey, 1984) are examples of on-board production and condition monitoring systems. These enable detailed analysis of digging cycles, plant, and operator performance, and can lead to reduced down-time, optimization of digging cycles, and maximization of power consumption. The DPM system also tracks the movement of a walking dragline, allowing automatic transfer of the path onto the mine plan, which enables rapid identification of inefficient digging positions.

Most of the effort to date appears to have been in the development of software and transducers. It is probable that these aspects of development will continue to dominate since many manufacturers of computer hardware produce off-the-shelf, ruggedized equipment quite suitable for mining use. It is unlikely that in-house development of computer hardware will be cost effective in the long term.

The level of microprocessor applications is now such that each individual stage of the operation can have its own dedicated controller. Each stage can then feature such facilities as

1. Signal conditioning and linearization.
2. Automatic closed-loop control.
3. Detection and action to alarm conditions.
4. Plant condition monitoring.
5. Self diagnostics.
6. Data collection and distribution.

It is difficult to identify any operation in surface mining that has not or will not in some way be influenced by the microcomputer. Any application must, however, be based on cost effectiveness. The evaluation of the potential benefit requires careful and detailed appraisal. Techniques of evaluation have been described by the British Coal Board (Anon., 1983), and a method based on the return on investment has been given by Harrison (1978).

Microprocessor-based survey instruments have increased both the speed and accuracy of field survey work. Many instruments allow data storage and direct transfer to automatic computer-aided mapping systems. The data can also be used for rapid volumetric calculations, of great value for stockpile or reserve estimation (Hodges and Alderson, 1985).

13.6.5.8 Artificial Intelligence

The objective of *artificial intelligence* (AI) is to extend the range of problems that can be solved with computers to include those that require decisions to be made (Chapters 8.3, 22.2).

The term AI is used to cover a broad range of relatively new computing techniques. The *"expert system"* or *"intelligent knowledge-based system"* falls in the category of AI.

Expert systems attempt to take information about a subject from a number of sources and to transform it into a knowledge base. This knowledge base can then be used to generate a set of rules concerning the subject. A major advantage of these types of systems is their ability to store vast amounts of expertise, to rapidly process this information, and to then make it available to non-specialists. The non-specialist can also feed in facts about a particular process, let the expert system interrogate the knowledge base, and provide recommendations about the process based on the experience loaded into the system (King, 1986).

Intelligent knowledge-based systems (IKBS) have enormous potential in the field of control engineering, particularly in the more difficult control situations typical of mining. They have the great advantage of being able to learn the rules for control of a process from a knowledge base. The knowledge base can be built up from a number of sources that include observation of manual operation or manual input of process rules, or the controller itself can explore the process by perturbation and observation, thereby deriving the process rules. In mining, both the processes and the raw materials suffer from enormous variability. The ability of an intelligent controller to continuously adapt to the changing conditions opens up new fields for automation. The principle has already been used for the control of feed to a grinding plant, and a system for automatic optimization of rotary drilling parameters is currently under development (Waller, Ambrose, and Casappi, 1987). IKBS control will, in the longer term, form the basis for automation of the excavating plant. The controller will learn typical digging cycles from manual operation; this combined with remote sensing of the optimum digging and loading position will lead to fully automatic operation. A similar scheme could be applied to manless trucks and a range of other manual operations.

The ability of IKBS to cope with large volumes of data makes it ideally suited to condition monitoring, maintenance, and diagnostic functions. As the size and cost of the work plant increases, rapid maintenance, fault diagnostics, and correction is vital.

Expert systems are already making an impact outside mining and appear to offer major benefits to the surface mining industry in a number of areas:

1. Geologic appraisal and reserve assessment.
2. Geotechnical hazard assessment.
3. Prediction of operational problems.
4. Enhancing computer-aided design (CAD) systems.

As improvements in data collection and processing occur, it would seem logical to further integrate real-time production data into the knowledge base. This would allow the IKBS to assess the efficiency of operations and to recommend ways of improving operations in the light of experience.

Constraints on widespread introduction of this type of technology are

1. Techniques not yet sufficiently advanced.
2. Processing power and capacity not yet cost effective.
3. Knowledge bases not wide enough (reluctance of experts to give up hard-earned information).

These systems are, however, currently available in relatively crude form and will undoubtedly find application in the surface mining industry in the very near future.

13.6.6 SURFACE MINE DESIGN AND OPERATIONAL PLANNING

The distinction between mine design and mine planning is often ill-defined (Chapters 13.1, 13.2). In the future, the two areas will tend to become even more fully integrated.

13.6.6.1 Mine Design

It is often stated that computers will never be able to fully cope with the complexities of mine design, or more commonly that they are unable to handle the constraints imposed by non-

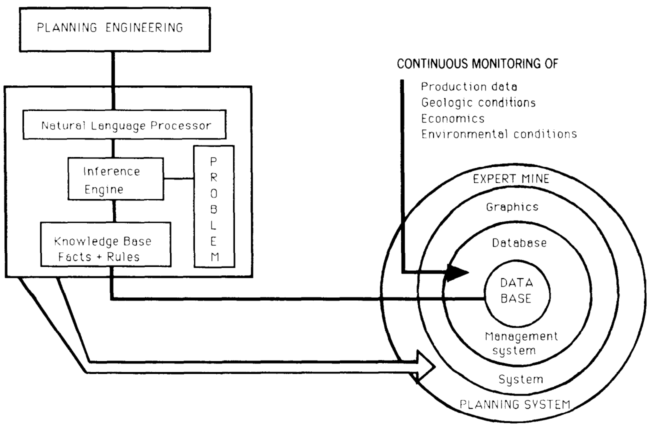

Fig. 13.6.7 Structure of expert mine planning system.

standard working methods (what is a "standard" working method?). Current CAD systems, for example, find it difficult to optimize haul-road positioning and usually rely on an engineer to modify an optimum pit limit design and an optimal mining schedule to produce a sub-optimal overall design.

More *integrated* mining CAD systems will become available that, when combined with increasing efficiency in operation, more powerful processing and greater machine intelligence will result in better design.

The mining CAD systems of today, while appearing to be sophisticated, are in fact no more than simple calculating tools, still relying on human experience and interaction to ensure that sensible mine designs are produced. This includes systems that attempt to optimize pit limit design or production scheduling, because these systems are currently incapable of handling or utilizing fully all the relevant information that is necessary for good design. Although this is likely to remain the case in the near future, in the long term, it seems likely that computers using AI will be capable of producing realistic mine designs, automatically and rapidly. Fig. 13.6.7 shows how this type of system may be implemented.

Computer applications have already resulted in surface mines being designed more rapidly, or resulted in better designs in the same time. As mine design rates increase and the design itself is more automated, it becomes easier and faster to redesign operations.

As the external constraints on optimal mining operations alter both in the short and long term (e.g., changes in market demand and mineral prices, exchange rate variation, etc.), and the data available from operations alter continuously, it becomes necessary to reexamine pit limits, production rates, areas, targets, grades, etc., more frequently to ensure that optimum mining strategies and tactics exist at all times (note the increasing importance of this as deposits become more marginal). Design systems will hence become much more dynamic and increasingly more sophisticated and powerful computing systems become available. The complete redesign of a surface mine in hours, minutes, or even seconds, while appearing ridiculous by today's standards, may not appear so in 10 years' time. Standards are also dynamic in nature.

13.6.6.2 Operational Planning

As increasingly marginal deposits are exploited, there is an obvious need to improve operational planning in order to remain competitive. Operational planning must therefore look to

1. Utilize the available information efficiently.
2. Employ the best analytical techniques.
3. Transfer the results of this analysis to the operations.

As operational planning now employs mathematical programming technology at a number of stages, it appears obvious that improvements in computing technology will benefit the planner. Operations research (OR) techniques will therefore, tend to be employed in production planning on a more intensive basis than is currently the case (Chapter 8.3).

As economies of scale reach their limits, the use of scientific management techniques is being seen as an area for increasing application (Atkinson, Burdon, and Longworth, 1985). The optimization of more complex production problems, using linear and dynamic programming, for example, will become possible as faster processing develops. This will allow larger scheduling problems with more complex constraints to be solved in real time. Once again, feedback of up-to-date production and exploration data will ensure that optimum extraction is taking place. This will become even more important if more flexible mining systems are employed to win minerals.

The boundaries between long- and short-term scheduling and daily production scheduling will become even more indistinct as more complex optimizations are carried out more rapidly.

It seems likely that more and more operational decisions will be made by computer. The introduction of automated truck dispatching systems, although meeting with varying degrees of success at the present time, is an example of the optimizing of operational processes. As teething problems are overcome, communications capabilities increase, and costs reduce sufficiently, it would appear that effective utilization of trucking capacity will become the norm rather than the exception.

Simulation techniques also will benefit significantly from increases in computer power and streamlining of the simulation process itself. More complex processes will be modeled and simulated, providing the planner with better information prior to the implementation of particular schemes. Simulation of parts of the surface mining system is already accepted practice.

The raw data for operational planning will be continuously collected as improvements in transducer and communications technology occur. Increases in the integration of planning and design systems with production data collection and processing will provide the opportunity to employ better input data. Hence, more realistic proposals will result from OR techniques as better input becomes available.

ACKNOWLEDGMENTS

Acknowledgment is due to the Institution of Mining and Metallurgy, London, for permission to use material appearing in their publications (Collin and Waring, 1983; Atkinson and Walton, 1983; Atkinson, Waller, and Denby, 1987), and to M.D. Waller and B. Denby, University of Nottingham, for contributions to the original document.

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Section 14 Surface Mining: Mechanical Extraction Methods

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Chapter 14.0
INTRODUCTION

M.K. MCCARTER

Surface mining applies to extraction of mineral resources from water and sediments of rivers, lakes, seas, and oceans; from solutions circulating through broken rock and soil; and from rock and soil excavated from the earth. Surface mining is responsible for the majority of mineral production in the United States and a substantial portion of the world's total production. This section deals specifically with extraction methods which employ mechanical processes in a relatively dry environment to free minerals from the earth. The following section, Section 15, deals exclusively with aqueous methods.

With recognition of the importance of reclamation in all activities which temporarily disturb the earth's ecosystems, it is appropriate to classify mechanical extraction methods as either *deferred reclamation* or *concurrent reclamation.* Deferred reclamation is necessary when the shape and logistics of the excavation require that it remain open throughout the productive life of the project. Upon exhaustion of economic reserves, the opening is then reclaimed in a manner consistent with public safety, environmental acceptability, and land use policy. Methods employing deferred reclamation are commonly referred to as *open pit* or *quarry mining.* Metallic ores, industrial minerals, and coal are all extracted using open pit methods. Typical examples are provided in Chapter 14.1.

Dimension stone is also produced in surface mines which, for the most part, remain open throughout the economic life of the deposit. *Quarry mining of dimension stone,* however, employs substantially different elements in the mining cycle, and for this reason is dealt with as a separate chapter (Chapter 14.2).

In the United States and in many other nations, concurrent reclamation is required for most coal and industrial mineral deposits mined by surface methods. These methods may be classified as *area mining* (open cast, strip, or furrow mining), *modi-*

fied open pit mining, contour mining, and *auger mining*. In all four cases, extraction and reclamation are contemporaneous and interrelated. Area mining, as the name suggests, covers large tracts of land and is usually the method of choice for nearly horizontal bedded deposits. In modified open pit and contour mining, the excavations are usually longer in the direction of advance compared to the width. Reclamation is accomplished by back-filling the rearward portion of the pit as the forward portion advances. There are many variations in area and contour mining. These variations depend primarily on deposit geometry and equipment used for excavation and haulage. Typical examples are presented in Chapter 14.3.

Auger mining consists of boring nearly parallel, horizontal openings into thin coal seams. It is employed to recover reserves beyond the economic limits of area or contour mines and where conventional underground methods are not economically viable or technically feasible. The distinguishing feature of this technique is the specialized excavator used to bore the holes and convey the coal to the surface. Because of the more or less specialized application of this technique, it is dealt with separately in Chapter 14.4.

In order to provide a broad overview of current practice, several case studies are presented covering typical operations and principal mineral products. Each presentation follows essentially the same format. A description of the mine and geologic setting provides background necessary to understand the conditions addressed by the selected method. This introduction is followed by a description of mine planning and development. The mining cycle (e.g., drilling, blasting, excavation, haulage, etc.) is then described with an emphasis on operational detail. Quality control of the mine product follows the discussion of unit operations, and the study concludes with a description of ancillary facilities and special processes typical of specific mineral commodities.

Chapter 14.1
OPEN PIT MINING

M.K. McCarter, Jeffery D. Tygesen, Gerald C. Smith, J.W. Villar, A.W. Swanson, Donald K. Winsor, David W. Farnsworth, George B. Whitman, and Bernard L. Grove

14.1.0 SURFACE MINING—DEFERRED RECLAMATION

M.K. McCarter

Open pit mining is a term properly applied to surface methods in which reclamation is deferred until all or nearly all of the deposit is removed within economic limits. The reason for deferring reclamation is usually a matter of access. If mining progresses sequentially to greater depths, backfilling the upper levels will either obstruct haulage routes or endanger work areas at the base of the slope.

Surface mining legislation distinguishes between "open pit mines" (usually metal) and "strip mines" (usually coal). In the western United States, mining permits for some metallic deposits, some industrial minerals, oil sands, and other commodities exclusive of coal may not require backfilling the pit. In this case, complete backfilling may never occur, and reclamation is usually deferred until the operator is certain that the economic limit has been reached.

Typically, overburden removed from an open pit is transported to an adjacent waste embankment or to a previously mined-out pit. Seldom is waste redeposited at its place of origin. In some operations, waste is placed on engineered surfaces and treated to recover values which are not high enough in grade to justify processing through normal milling operations. This activity, referred to as "leaching," is common for copper, gold, and uranium wastes (Dorey, van Zyl, and Kiel, 1988; Duncan and Smolik, 1977; Muhtadi, 1988) (see also Chapters 15.2 and 15.3).

In some open pit operations, typically western US uranium and phosphate deposits, the pits may be large enough or configured in such a way as to allow some backfilling before the entire pit is completed. This may be done to minimize haulage costs, reclamation costs, or as a matter of convenience. This handling of waste, however, is not required as part of the mining process as in other methods to be discussed later (Chapter 14.3). Mining procedures in which backfilling is done as a matter of choice should be described as open pit rather than area mining, contour mining, or other techniques in which excavations are filled and reclaimed contemporaneously with ore extraction.

There is a tendency to refer to all surface mining of coal as strip (area) mining. In some cases, particularly steeply dipping coal seams, the method is actually open pit mining. In such deposits, mining progresses through one or more seams. All overburden must be removed in order to gain access to the deepest economic volume of coal. Since the reclamation is deferred and all or most of the overburden is removed, the method satisfies the requirements for classification as an open pit method.

Mountaintop removal, as practiced in the coal regions of Appalachia and less frequently in bedded deposits other than coal, satisfies some of the requirements for open pit mining in that overburden is transported to a waste embankment located outside the area of mining. Reclamation of the exposed surface may or may not proceed contemporaneously with mining. The term "open pit," however, connotes a depression below the natural ground surface, which is not the case for removing the top of a mountain. Mountaintop removal, therefore, does not conveniently fit either definition. It may be considered "area mining" without concurrent reclamation, or as "open pit mining" without a pit.

Classic open pits such as Bingham Canyon, UT, can be thought of as inverted, truncated, circular cones where the radius of each circular bench decreases with depth. This shape is characteristic of large disseminated deposits such as copper. Open pits in deposits that are stratiform, shallow, and of substantial aerial extent often appear as footprints with steep sides and more or less flat bottoms.

Removal of material proceeds in one of two patterns referred to as *sequential pushbacks* or *pushbacks.* One or more pushbacks constitutes a phase or intermediate plan. The sequential pushback pattern is illustrated in Fig. 14.1.1. In this example, three zones are identified as A, B, and C, and one excavator (typically an electric shovel) would be assigned to each zone. Assuming zone C is ore and zones A and B are waste, it is necessary for cuts to be made through the areas identified as B1 through B4 before ore can be mined in areas identified as C5 through C8. Similarly, areas identified as Al through A4 must be removed before mining can proceed to areas B5 through B8. This pattern requires close coordination so that shovel A finishes its final level and begins to move up to the initial cut in zone A before shovel B moves up to initiate a new series of cuts in zone B. Through proper scheduling, waste stripping can be accomplished so as to provide the "mining room" needed to sustain the required level of ore production lower in the pit.

Zones in a sequential pushback pattern are separated by haulage ramps as can be clearly seen in Fig. 14.1.2. The elevation of the ramp is not constant but cycles through a range of elevations with a more or less constant number of benches between ramps. (The number of levels between ramps may decrease in the upper reaches of a large mine to accommodate the greater circumferential distance compared to benches at lower elevations. Alternatively, larger shovels may be employed at higher elevations to increase the rate of face advance to keep up with

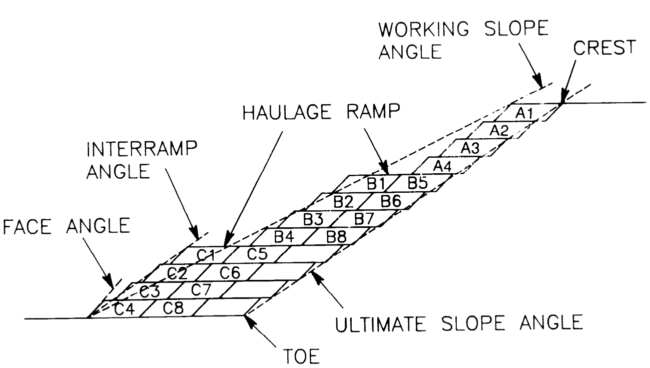

Fig. 14.1.1. Cut sequence for sequential pushbacks.

Fig. 14.1.2. Haulage ramp pattern in waste zone of a western US copper mine (1970).

the mining room requirements lower in the pit.) The ramps must be wide enough to accommodate the excavator and haulage equipment; therefore, the overall slope is less than the interramp slope which is less than the bench face angle (less by virtue of the catch bench between individual levels). This arrangement provides a "working slope" that is less than the design or ultimate slope.

Working slopes flatter than the ultimate have operational advantages. One advantage is additional production faces in both ore and waste. Additional faces provide greater flexibility and more uniform production. Dispersing production faces reduces the likelihood of incidental problems such as sector power outages or localized bench failures from having a major impact on production quotas. Additional production faces in ore also allow blending of ore types to optimize mill performance. Blending is often a difficult task in high-tonnage operations because of surge pile limitations. Finally, working slopes typically limit the size and consequences of major slope failure. At best, selection of an ultimate slope angle is based on geotechnical analysis of available information and engineering judgment. As new information is acquired, adjustments in the design angle may be needed. Approaching the pit limits using a shallower working slope provides an opportunity to obtain more geotechnical information and operating experience before committing to an ultimate slope angle. Selection of an inappropriate angle, either too large or too small, will have far greater consequences for pits approaching a depth of 1000 ft (300 m) than for pits with depths on the order of 100 to 300 ft (30 to 90 m). In the case of deep pits, massive instability caused by angles that are too steep may jeopardize the entire operation. If they are too shallow, the cost of stripping may be excessive resulting in lost revenue and low recovery of natural resources.

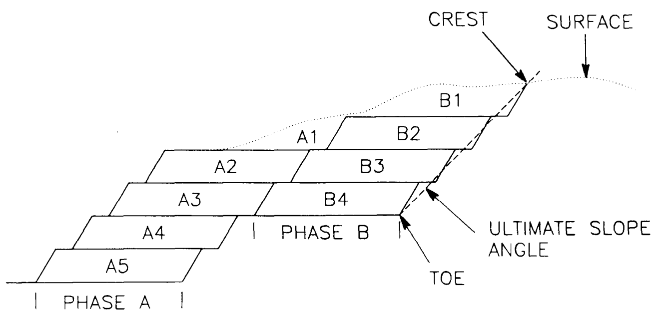

Fig. 14.1.3. Cut sequence for conventional pushbacks.

Conventional pushback, the more popular of the two mining patterns, is commonly employed in shallow pits and in some larger mines. As illustrated in Figs. 14.1.3 and 14.1.4, a pit sector of substantial plan area is mined in whole benches, one at a time, from the top down to perhaps the lowest ore level. One advantage of this method is less complicated scheduling compared to the sequential pushback method. The larger plan area also allows more working faces within a given sector of the mine, which reduces the number of supervisory personnel and requires less maintenance of mine utilities and haul roads. The disadvantages are less flexibility in scheduling operations, greater vulnerability to operational problems, and less opportunity to blend ores from other areas of the mine.

The backslope of a conventional pushback is often cut to the ultimate slope angle. This is an advantage in that the ultimate slope design is tested early in the mine life but at a scale that

Fig. 14.1.4. Pushback in progress at a western US copper mine (1979).

will not jeopardize the entire mine. The disadvantage is that if slope instability does occur, there may be a substantial but temporary loss in production.

The major steps in development of an open pit mine include (1) determining the three-dimensional distribution of mineralization and grade (Barnes, 1980; Barnes, 1989); (2) establishing the economic limits for the pit (Soderberg and Rausch, 1968; Koskiniemi, 1979; Robb, 1979); (3) selecting suitable sites for waste embankments and soil stockpiles (McCarter, 1985; McCarter 1990); (4) clearing vegetation from the land intended as sites for pit and waste embankments; (5) siting of processing, maintenance, office, and transport facilities close to the pit but outside potential pit limit expansion (Myntti, 1979); (6) selecting equipment; laying out haulage roads; and (7) initiating "pioneering" cuts. These initial cuts may be in the form of "box cuts" (Hartman, 1987) or access roads on steep hillsides that are enlarged to form mine benches.

Standard unit operations (drilling, blasting, excavating, and haulage) are generally employed in removing both waste and ore. The type of drills most commonly used include rotary and down-hole percussion rigs. If the material cannot be ripped, the explosive of choice is ANFO for dry blastholes and emulsion for wet blastholes. Some operations prefer to pump water from holes and install plastic liners to prevent degradation of the ANFO. After breakage, material is excavated using electric shovels, hydraulic shovels, backhoes, front-end loaders, scrapers, and, less commonly, draglines and bucket wheel excavators. The height of a bench is normally dictated by the reach of the prime excavators to be used. The height should be well within the maximum digging range so that the slope and the tendency toward caving of the face and highwall can be better controlled (Rumfelt, 1968). The width of individual benches is determined by the clearances required to conveniently load into the matched haul units, stability of the pit wall, and by the break-even stripping ratio. Haulage is accomplished with trucks, scrapers, conveyors (there is a current trend toward in-pit crushers and conveyors), and less frequently with rail, skip hoist, and hydraulic transport.

Open pit mining accounts for approximately 60% of the production from surface mines (Hartman, 1987). The principal metallic products and number of mines worldwide are shown on Fig. 14.1.5. The same information for nonmetallic products is shown on Fig. 14.1.6.

The following case studies provide examples of principal commodities and mining techniques for current open pit practice.

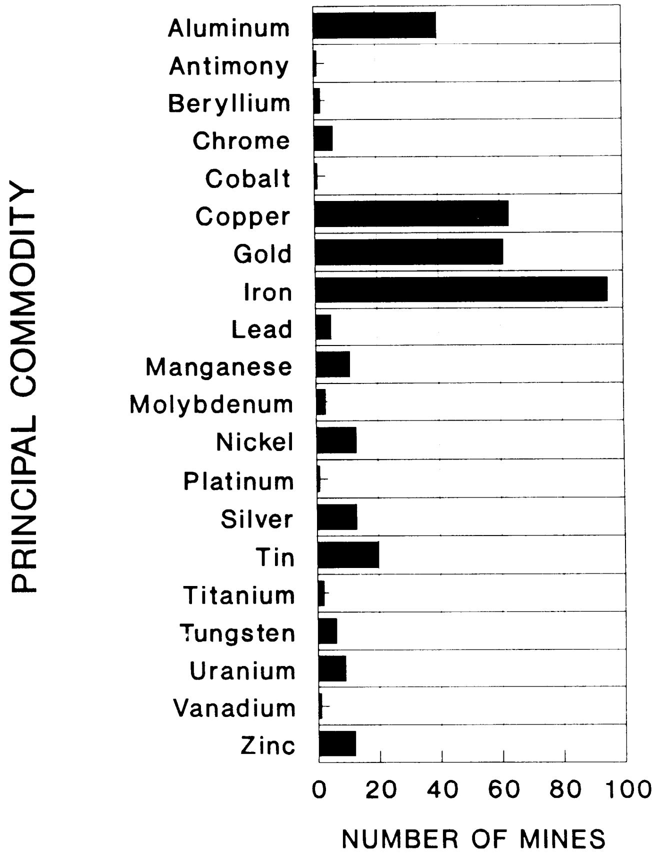

Fig. 14.1.5. Number of major metalliferous open pit mines in the Free World by commodity (Anon., 1988).

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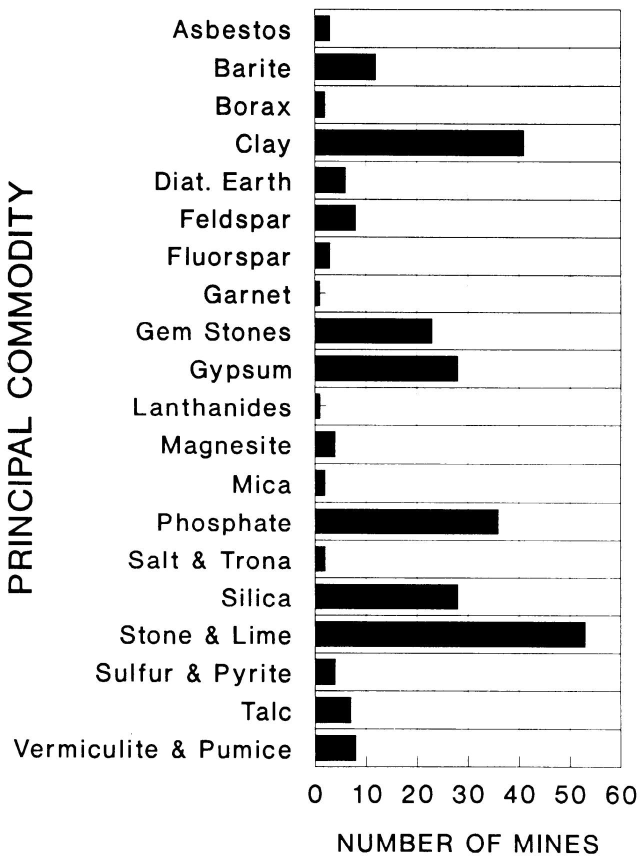

Fig. 14.1.6. Number of major nonmetallic open pit mines in the Free World by commodity (Anon., 1988).

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14.1.1 PORPHYRY COPPER: BINGHAM CANYON MINE

JEFFERY D. TYGESEN

14.1.1.1 Mine Description

The Bingham Canyon mine is the world's first low-grade, open pit copper mine and the largest man-made excavation. Since its inception in 1906, over 5 billion tons (4.5 billion t) of material have been removed and 13 million tons (11.8 million t) of copper have been produced. The mine's estimated life will extend until 2015, with an average stripping ratio of 0.4:1.

A new mill, completed in 1988, made Bingham Canyon one of the most cost-effective and productive mines in the copper industry. Following completion of this mill, mine production was at 107,000 tpd (97,000 t/d) of ore. In 1989, production increased to 111,000 tpd (101,000 t/d) of ore and 99,200 tpd (90,000 t/d) of waste[1]. The annual production of metals processed through smelting and refining was about 210,000 tons (191,000 t) of copper, 350,000 oz (11,000 kg) of gold, 2,500,000 oz (71,000 kg) of silver, and 14,000,000 lb (6350 t) of molybdenite. An additional grinding line will be added to boost concentrator throughput to 142,000 tpd (129,000 t/d).

The mine is located in the Oquirrh Mountains, 30 miles (42 km) southwest of Salt Lake City, UT (USA). The mine area including waste dumps encompasses approximately 9500 acres (38 km^2). The dimensions of the mine are roughly 2.5 miles (4.0 km) across at the rim and about 0.5 miles (0.8 km) from the rim at the 7740 level[2] to the bottom at the 5090 level (see Fig. 14.1.1.1).

Access to the mine is provided by truck-haulage ramps (width approximately 150 ft or 46 m) on the north and south sides of the mine. The north ramp exits the mine at the 6190 level and extends approximately 2.5 miles (4.0 km) to the Dry Fork dumps at an elevation of 6650 ft (2027 m). The south ramp exits the mine at the 6800 level, extending approximately 1 mile (1.6 km) to the East Side dumps at an elevation of 6800 ft (2073 m). A vehicular roadway through Bingham Canyon provides additional access to the northeast side of the pit at the 6190 level.

In addition to the roads, three tunnels enter the mine at the 6040, 5840, and 5490 levels in the northeast sector and exit into Bingham Canyon. These tunnels are 18 ft (5.5 m) wide and 24 ft (7.3 m) high. The lengths are 2500 ft (762 m), 5200 ft (1585 m), and 16,300 ft (4968 m) respectively. The two upper tunnels are equipped for rail haulage of ore from reload facilities within the pit; the lower tunnel is equipped with a 72-in (1.83-m) belt used to transport ore to the Copperton mill.

Operations are scheduled three 8-hour shifts/day, 365 days/year. The work force at the mine includes 70 supervisory and administrative personnel and 540 hourly personnel. The work force is generally divided into two categories, operations and maintenance. The operations group totals 336 people while the maintenance group accounts for 274 people.

14.1.1.2 Deposit Description and Geology

The Bingham Canyon ore body is a porphyry deposit centered in and around a complex monzonitic stock. This stock was intruded into older monzonite and surrounding Pennsylvanian sediments. Approximately 100 m.y. ago folding and faulting associated with mountain building processes resulted in a strong system of intersecting high-angle faults. This "structural crossroads" produced areas of weakness, which facilitated emplacement of magmatic material and, later, access to the host rock for hydrothermal solutions. Mineralization, dated at about 40 m.y., includes disseminated deposits, fracture or vein fillings, and replacement deposits in reactive limestone units.

Monzonite and monzonite porphyry are the most important host rocks for copper mineralization at Bingham Canyon. Quartz monzonite, a medium-grey, equigranular rock, makes up about 33% of the pit exposures. Porphyritic quartz monzonite, quartz monzonite porphyry, latite porphyry, and quartz latite porphyry are other intrusive materials accounting for about 12% of the pit exposures. Pennsylvanian quartzite occurs mostly in the east, north, and northwest sections of the mine making up about 40% of the pit exposures. The highly metamorphosed Jordan and Commercial limestones occur in beds 160 to 190 ft (49 to 58 m) thick in the south and southwestern sectors of the mine. These units account for 9% of pit exposures and are generally very blocky and difficult to mine. Moderately soft siltstone, intercalated with quartzite, makes up about 6% of pit exposures. Individual beds of siltstone average about 9 ft (3 m) in thickness and commonly are associated with low-angle thrust faults.

Overburden is divided into two categories. Iron-stained sediments occurring in the upper part of the mine have been oxidized

[1] The term "waste" is an economic classification referring to barren rock or rock with less contained copper than economically feasible to cover by conventional milling processes.

[2] Level designations are in terms of feet above mean sea level.

Fig. 14.1.1.1. Aerial View of the Bingham mine from the south.

and naturally leached of sulfide minerals. The remaining portion of the overburden consists of sediments and intrusives containing less copper than can be economically recovered through milling. This material is removed from the mine and deposited on dumps.

The primary copper-bearing minerals at Bingham Canyon are chalcopyrite, bornite, and chalcocite. Molybdenite, gold, and silver are the primary byproduct metals. Pyrite is the most common sulfide gangue mineral. Lead, silver, and zinc ores occur as a result of lode or replacement deposits in the peripheral areas outside the porphyry intrusion and the perimeter of the current pit. Copper oxides are negligible.

The metal distribution within the deposit is in the form of inverted shells or domes surrounding the Bingham stock (Fig. 14.1.1.2. The mineral zones are (from inside out) central barren core, molybdenite zone, bornite-chalcopyrite zone, chalcopyrite-pyrite zone, pyrite zone, and galena-sphalerite zone.

14.1.1.3 Mine Development

HISTORICAL PERSPECTIVE. The first ore was milled in nearby Copperton in August 1904. This 300-tpd (272-t/d) gravity pilot plant was constructed to demonstrate the feasibility and practicality of the milling and crushing equipment. In 1908, the Utah Copper Company built a 6000-tpd (5400-t/d) concentrator near Magna, some 15 miles (24 km) north of the mine. In 1909, Boston Consolidated completed a 3000-tpd (2700-t/d) mill in Garfield near the Magna concentrator. In 1910, the two companies merged as the Utah Copper Company, and within one year modified the mills and increased production to 13,500 tpd (12,200 t/d). Ore production had risen to 90,000 tpd (81,600 t/d) by 1940.

In 1966, the Bonneville facility was completed providing additional grinding capacity, and production increased to

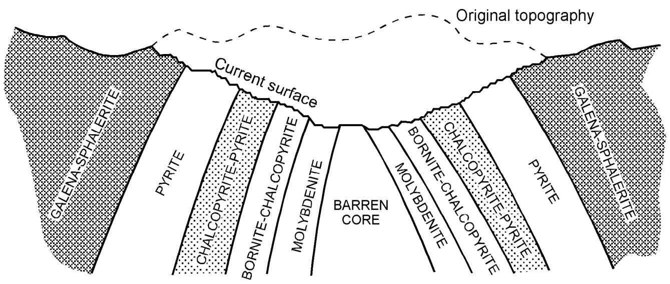

Fig. 14.1.1.2. Generalized shape of mineralized zone.

108,000 tpd (98,000 t/d). In part, the increased production was required to offset the declining ore grade (At the inception of open pit mining, ore grade averaged over 1.9% copper. In 1989, the average grade was about 0.6%). In the late 1970s, recognizing that the labor-intensive operation of the aging Magna and Garfield concentrators would not support future production requirements, plans were made to modernize milling and concentrating operations. In 1988, a new Copperton concentrator was completed. The plant was designed to process 77,000 tpd (70,000 t/d). Ore transport from mine to mill was accomplished through an in-pit crusher and 5-mile (8-km) conveyor.

By 1992, the new mill, with the addition of a fourth grinding line along with the Bonneville/Magna facility, will support a production of 142,000 tpd (129,000 t/d) for the next 25 years. The general layout of the mine, mills and smelter is shown in Fig. 14.1.1.3.

MINING PLAN. Mining progresses in the form of successive pushbacks. The pushbacks range from 100 ft (30 m) to 200 ft (61 m) in width and 50 ft (15 m) in height. The operating interramp pit slope, including bench face angles and catch benches, is 34° as shown on Fig. 14.1.1.4. Catch benches are 50 ft (15 m) wide and effectively collect spillage from blasting and loading.

In 1989, five ore shovel faces and five waste shovel faces were required to ensure planned production. Five ore shovels were required to meet grade and metallurgical blending requirements. The five waste shovels were required to meet long range stripping requirements ensuring future ore. The geometry of the deposit is such that the majority of the ore is located in the lower 900 ft (270 m) of the pit, roughly at the 5740 level and lower. The highest active waste cut is approximately 2000 ft (610 m) higher in elevation on the 7740 level. In the extreme case, mining room must be brought down nearly 40 benches before new ore is exposed; this process can take as long as seven years.

A recent reassessment of slope angles for ultimate pit designs resulted in subdividing the pit surface into 26 sectors based on rock type, structure, material characteristics, and strike and dip of geologic structure relative to pit slope orientation. The slope angles defined for each of these sectors range from 29 to 50°. Design slope angles in sediments, located in the upper benches, range from 30° on the east side and up to 50° on the northwest side of the pit. Design slope angles in intrusive materials range from 29 to 46°.

These slope angles will be achieved by double benching or single benching using a modified mining technique for cuts adjacent to the final slope. This technique includes controlled blasting, "digging to hard", and using dozers in soft material on final surfaces.

Slope dewatering also will be incorporated where necessary, using near-horizontal drains. Successful depressurization typically provides an average 3 to 5° steepening in ultimate slope angles for conditions at Bingham Canyon.

Long-term mine planning is implemented through the use of a model consisting of blocks measuring 100 by 100 ft (30 by 30 m) in plan by 50 ft (15 m) in height. Each block is assigned representative grades for copper, molybdenum, gold, and silver generated by Kriging techniques. In addition, the blocks are assigned values representing rock type and physical properties of the geologic formation encountered at the location of the block. Information for grade assignments, geologic data and physical properties is based on development drilling on 400- to 600-ft (122- to 183-m) centers. Material density varies only slightly with rock type; 2.18 tons/yd^3 (2.58 t/m^3) represents average conditions.

Mining plans are developed by defining the volume of ore and waste between a particular future pit surface and the existing topography. This material is then sequentially mined by a computerized mining simulator algorithm. This algorithm allows material with the highest relative profit margin to be mined first. The planning steps thus created are then modified to reflect haulage roads and operational constraints. The result is a series of annual plans for the first five years followed by five year plans to the end of mine life.

14.1.1.4 Unit Operations

DRILLING AND BLASTING. Rock breakage is accomplished by a crew of 13 working an average of 5 days/week, 2 shifts/day. Blastholes are drilled by eight Bucyrus-Erie 60R track-mounted drills. These drills are capable of exerting 120,000 lb (534,000 N) thrust and can drill 57 to 65 ft (17 to 20 m) in a single pass. Average production of 12.25-in. (311-mm) holes is about 12 per 8-hour shift. Rotary tricone bits with carbide inserts are used.

In addition to the 60R units, production blastholes also are drilled with two Driltech D75K track-mounted units and a secondary drill capable of drilling angled holes. The D75K drills employ carbide insert bits 9.875 in. (200 mm) in diameter and carry four 35-ft (10.7-m) drill rods. The mobility of these drills and the smaller-sized hole they produce make these drills useful in resilient formations where closer patterns are necessary for proper fragmentation. These drills also are used in pioneering shovel cuts where electric power and space are not available for the larger electric drills.

A secondary drill uses 2.5-in. (64-mm) bits and 12-ft (3.7-m) drill rods. The resulting holes are loaded with dynamite to break boulders and high spots in haul roads. Secondary breakage also is accomplished with a Drott, rubber-tired, backhoe chassis equipped with a Tramac rock breaker head. The rock breaker is used where blasting is not advisable because of proximity to equipment or facilities.

Drilling patterns vary with rock type but range from 30 × 30 ft (9 × 9 m) to 36 × 36 ft (11 × 11 m) for 12.25-in. (311-mm) holes and 25 × 25 ft (7.6 × 7.6 m) to 30 × 30 ft (9 × 9 m) for 9.875-in. (200-mm) holes. The hardest, least-fractured rock types are metamorphosed units of the Commercial and Jordan Limestones, locally known as metalimestone. This material produces blocky fragments and requires more closely spaced holes to improve fragmentation and material handling.

Explosives are loaded using two, front-discharge, one-man, bulk ANFO trucks. Blending of ammonium nitrate prills and fuel oil occurs when the bulk delivery trucks deliver these compo-

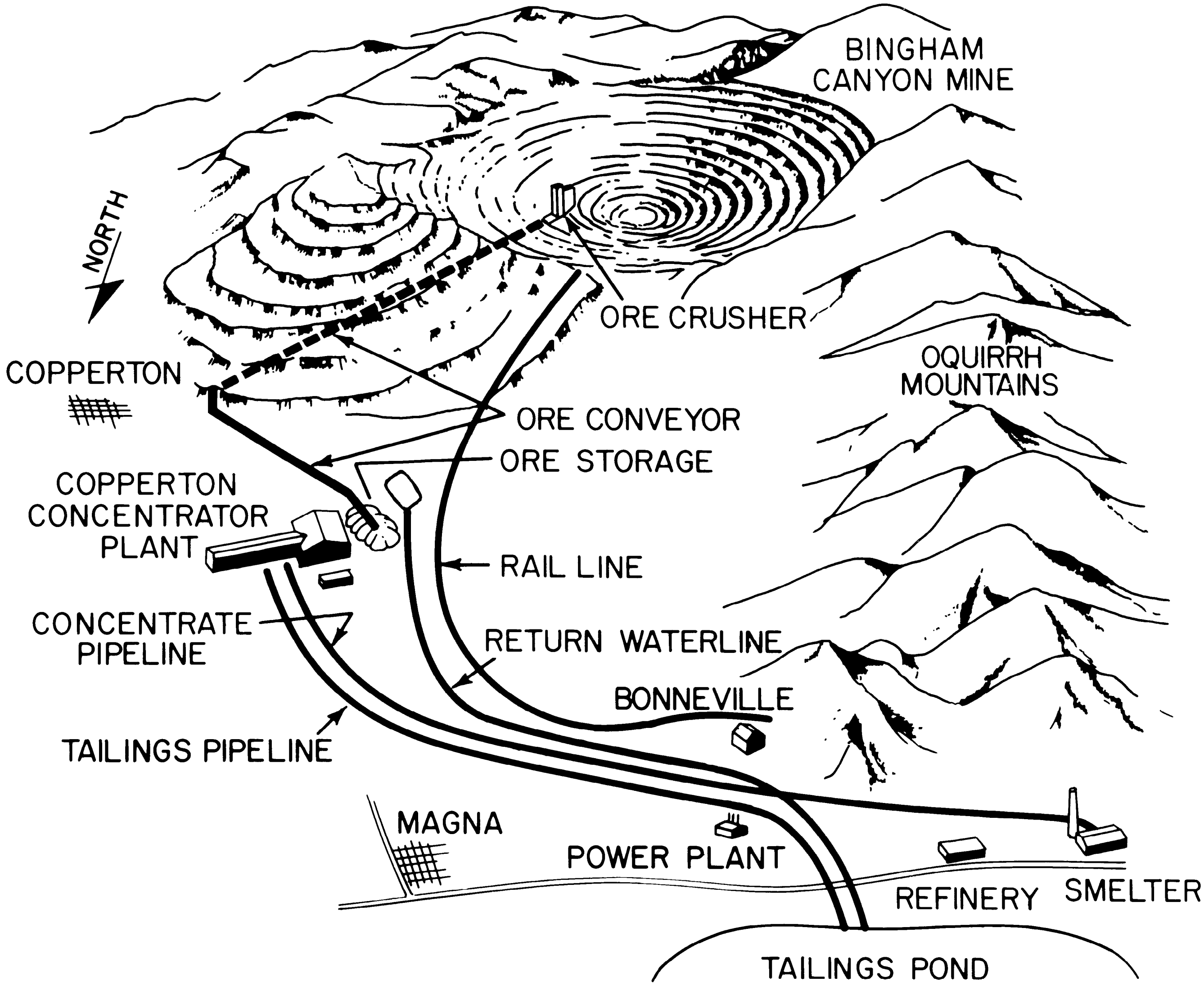

Fig. 14.1.1.3. General locations of mine, concentrators, smelter, and refinery.

nents to the mine-site storage tanks. ANFO is the preferred explosive, but commercial bulk emulsion-blend explosives are used in wet holes.

Typically, holes are primed with two 0.75-lb (0.34-kg) boosters. Both boosters are placed near the bottom of the explosive column. A 200-ms delay is inserted into each booster and connected to individual 7.5-grain Primaline down-lines. Twenty-five-grain detonating cord is used for trunk lines and cross ties. Surface delays of 17 ms are used between holes and 100 ms between rows. A single strand of detonating cord is extended from the pattern and initiated by a non-electric cap taped to the cord.

Drill cuttings are used for stemming. Each hole produces approximately 2.4 to 3.7 tons (2.2 to 3.4 t) of cuttings, depending on hole diameter. These cuttings are forced into loaded holes using two rubber-tired stemming machines developed at Bingham. The machine consists of a small front-end loader (FEL) chassis equipped with curved opposing arms that are closed by hydraulic cylinders.

Powder factors vary from 0.13 to 0.25 lb/ton (0.065 to 0.125 kg/t), depending on rock type and spacing of geologic discontinuities. The overall average is 0.16 lb/ton (0.08 kg/t). For purposes of blast design, geologic materials are classified as intrusive, quartzite, and limestone with the appropriate powder factors assigned.

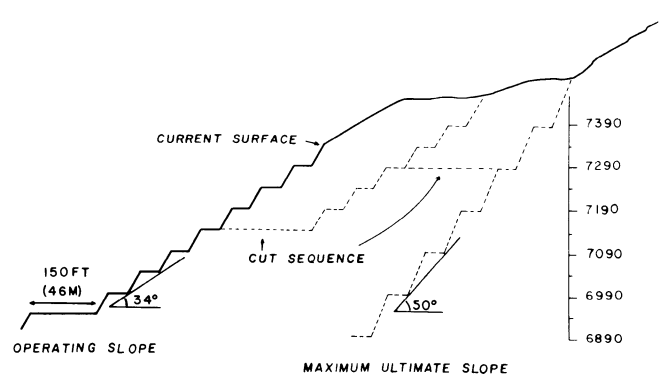

Fig. 14.1.1.4. Typical cut sequence (vertical section).

The objective in drilling and blasting is to loosen the material sufficiently to meet production goals without degrading the integrity of the final pit slopes. With this in mind, ground motion due to blasting is limited to 25 in./sec (640 mm/s) at planned final pit slopes.

LOADING. Primary production utilizes the following number and type of units:

2 15-yd^3 (11.5-m^3) P&H 2100 electric power shovels
4 27-yd^3 (20.6-m^3) P&H 2800 Mark II electric power shovels
3 30-yd^3 (22.9-m^3) P&H 2800 XP electric power shovels
2 34-yd^3 (26.0-m^3) P&H 2800 XPA electric power shovel

The 15-yd^3 (11.5-m^3) shovels are used primarily for ore reload into trains, narrow cuts, and construction cuts. The larger shovels are used exclusively for waste and ore production faces.

The availability for the 2100 shovels averages 78% while that of the larger shovels averages 88%. The smaller shovels are scheduled 2 to 3 shifts/day and shovels larger than 15-yd^3 (12.5-m^3) are scheduled 14 to 15 shifts/day. Average productivity for the 2100 is 10,000 tons (9100 t); for the Mark II and 2800 XP, 15,000 tons (13,600 t); and for the 2800 XPA, 20,000 tons (18,100 t) per shovel shift.

Power is provided by 44-kva distribution lines circling the pit. At appropriate points power is fed to 13.8-kva substations. Radial lines are then fed through smaller substations where the voltage is reduced to 5500 V ac to power shovels and drills. Electrical connections between the switch houses and shovels are made using trailing cables up to 2000 ft (607 m) long for shovels and 3000 ft (914 m) long for drills. However, cables have been run as long as 5000 to 6000 ft (1524 to 1828 m).

In addition to the main production fleet, two 8-yd^3 (6.1-m^3) International, one 12-yd^3 (9.2-m^3) Clark, and two 12-yd^3 (9.2-m^3) Caterpillar rubber-tired FELs are used for construction, clean-up, reload, and auxiliary production when needed.

HAULAGE. Originally, the mine was entirely rail haulage. Steam engines gradually were replaced by trolley locomotives, which, in turn, were replaced by diesel locomotives. Truck haulage was introduced in 1963, and by 1983, all primary production was accomplished by trucks. Rail reload operations are a vestige of the original haulage system and will continue to supplement ore production as long as they are profitable. In 1989, the rail haulage system accounted for approximately 30% of the ore production. The majority of the ore is hauled by truck to an in-pit crusher and transported to the mill by belt conveyor.

Rail reload utilizes 15-yd^3 (11.5-m^3) shovels and 12-yd^3 (9.2-m^3) loaders to fill 100-ton (91-t) rail cars. Each 25-car train is powered by two 135-ton (122-t) diesel locomotives. The total rail fleet consists of 11 locomotives and 500 ore cars. Ore is currently being transported approximately 12 miles (32 km) to the Bonneville facility for crushing and processing.

Truck haulage utilizes a fleet of 44 trucks. This fleet is composed of

28 190-ton (172-t) Caterpillar mechanical-drive trucks
8 170-ton (154-t) Unit Rig diesel-electric trucks
8 170-ton (154-t) Wabco diesel-electric trucks.

At the 1990 production level, approximately 34 truck-shifts/shift are scheduled, with an average availability of 94% for the new, larger trucks and 84% for the older, smaller trucks. All trucks are equipped with two-way mobile radios, which assist appropriate dispatching.

The in-pit, movable, 60- by 109-in. (1.5- by 2.7-m), 1000-hp (745-kW) Allis-Chalmers gyratory crusher can accommodate 120,000 tpd (109,000 t/d) on a continuous basis. The crusher can handle two trucks at a time and a dumping rate of one truck per minute. This rate is achievable only for short periods when the rock is highly fragmented. This facility is situated on the 5490 level, which is centrally located with respect to planned ore production over the next eight years. The entirely self-supporting crusher assembly is 100 ft (30 m) high (from the truck dump ramp on the 5590 level to conveyor discharge on the 5490 level). Currently, it is resting on a 4200-yd^3 (3200-m^3) concrete foundation. The crusher will be relocated using a 2000-ton (1814-t) crawler transporter. Three to four weeks will be required from the time a move begins until the crusher is brought back on-line at its new location.

Material processed through the facility is crushed to -10 in. (-250 mm) and fed directly to the first flight of a six-segment 72-in. (1.8-m) belt. The belt system is 5 miles (8 km) long and capable of 10,000 tph (9072 t/h) at 900 ft/min (4.5 m/s).

After the ore is processed at the Copperton concentrator, the copper concentrate is pumped through a 6-in. (152-mm) diameter pipeline 17 miles (27 km) to the smelter. The tails also are pumped through a 48-in. (1200-mm) diameter pipeline 12 miles (19 km) to the tailings pond near Magna.

14.1.1.5 Product Control

The Mine Planning Department defines cuts to meet production budgets for short-term and long-term schedules. Once a cut sequence is defined, a computer model of the deposit is used to project whether the material is ore or waste. When the actual cut begins, blasthole cuttings are sampled. If the area is within a projected ore zone, all samples are assayed using atomic absorption spectrometry. If the area is within a projected waste zone, only every fourth hole is assayed. Each hole defines the grade of the surrounding material equal to the drill pattern spacing.

Each mining block is identified by survey stakes and marked with flagging. This procedure allows the area supervisor and shovel operator to distinguish between ore and waste.

The mine also is mapped geologically for structure, rock type, and material characteristics. These properties are associated with individual ore blocks to allow projection of milling performance in terms of recovery, fines production, and Bond work index/SAG work index.

14.1.1.6 Ancillary Facilities

LEACHING OPERATIONS. Since 1923, leaching and precipitation processes have been used to treat rock containing very low copper values. The copper-bearing solutions resulting from this operation are collected in clay or concrete lined ditches or pipe routed to the precipitation plant. The current precipitation plant consists of one module housing 13 cone units. The 13 cones are designed to process copper-bearing solutions through shredded scrap iron on a continuous basis and discharge a copper precipitate slurry at preset intervals into a thickener. Precipitate slurry is pumped from a surge-mixing tank to a filter press for dewatering and drying. This material, containing 80% copper, is then shipped to the smelter for further processing.

Barren solution from the cones passes through a 140-ft (43-m) settling basin with the overflow going to the sump of the central pump station. The distribution system delivers leaching solution from the central pumping station 2.5 miles (4.0 km) to a height of 1270 ft (524 m) on the east-side waste embankments. The west-side water is pumped 3.0 miles (4.8 km) to heights over 1400 ft (430 m). The water is applied to the dump surfaces dissolving soluble copper as it percolates through the dumps. The copper-bearing solution flows from the bottom of the embankment to a collection system that carries it back to the precipitation plant.

The entire leaching complex, including solution distribution, collection, makeup and primary water systems, and various specialized auxiliary lines, requires more than 45 miles (72 km) of pipe. Water is stored in two reservoirs totaling 520 million gallons (2 million m^3) of storage capacity. The copper produced from these facilities is approximately 3 to 4% of the total copper production at Bingham Canyon.

OVERBURDEN EMBANKMENTS. Active embankments are located to the northwest and east of the pit. The most common construction is end-dump, side-hill fills; however, valley fills and heaps also are employed. The older, inactive embankments were built using 100- to 200-ft (30- to 60-m) lifts. Current embankments are constructed in lifts up to 900 ft (270 m).

MINE MAINTENANCE. Mine operations are supported by a maintenance group of about 274 people, approximately one-half of the total mine work force. Maintenance personnel include mechanics, electricians, welders, lubemen, warehousemen, field repairmen, boilermakers, water servicemen, carpenters, and millwrights. Facilities to maintain operations comprise an eight-bay haulage-truck shop and a three-bay haulage-truck tire shop, both located at the 6 190 level at the head of Bingham Canyon. In addition, the field repairmen and field electricians are headquartered adjacent to the mine to service shovel, drill, and crusher/conveyor equipment. Additional facilities are located approximately 2 miles (3.2 km) down-canyon. These facilities include two warehouses and shops for electrical motor repair, welding, machining, small equipment repair, railroad car repair, and diesel locomotive repair.

ROAD DEPARTMENT. At the mine site, 28 miles (45 km) of haulage roads and approximately 40 miles (64 km) of support roads are maintained. It is well known that high-quality roads result in higher productivity and lower maintenance costs. The road department is staffed with 37 people who operate the following equipment:

20 dozers (Caterpillar D-9H, D-9L, D-10L)
11 graders (Caterpillar 16G)
2 scrapers (Caterpillar 631)
4 salt trucks (6-ton or 5.4-t capacity)
6 water trucks (converted 65-ton or 59-t haulage trucks, 10,000- to 30,000-gallons or 38- to 114-m^3 capacity)

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14.1.2 WESTERN US DISSEMINATED GOLD: GOLD QUARRY MINE

GERALD C. SMITH

14.1.2.1 Mine Description

The Gold Quarry mine (Fig. 14.1.2.1) is the largest of six deposits being mined by Newmont Gold Co. (90.1% owned by the Newmont Mining Corporation) along the Carlin Trend in Northeast Nevada (Fig. 14.1.2.2). The mine is located in the Maggie Creek Mining District approximately 7 miles (11 km) north of the town of Carlin in Eureka County, NV, at an elevation of 5400 ft (1646 m).

Of the 100.3 million tons (91 Mt) of ore and waste mined in 1988 by Newmont Gold Co. from the ore bodies along the Carlin Trend, the Gold Quarry mine accounted for 57% or 56.9 million tons (51.6 Mt) of ore and waste. Of this total, 4 million tons (3.6 Mt) were mill-grade ore, 18 million tons (16.3 Mt) were leach-grade ore, and 34.9 million tons (31.7 Mt) were waste. The overall waste-to-ore ratio was 1.59:1.

The Gold Quarry mine operates 3 shifts/day, 7 days/week and is scheduled to mine 205,000 tpd (186,000 t/d) of ore and waste for a total of 74.8 million tons (67.9 Mt) during 1989.

14.1.2.2 Deposit Description and Geology

The complex geologic history of Nevada has been described by Stewart (1980) and by Rota and Ekburg (1988). During early Paleozoic, the Carlin Trend area lay along the western margin of the North American continent. A westward thickening wedge of sedimentary rocks, varying from carbonate in the east to siliceous clastic and cherty units to the west, was deposited offshore on the slope and in a marine basin.

Tectonic activity related to the Antler erogenic event in late Devonian to possible earliest Triassic time resulted in the transport of thick sequences of Ordovician deep-water siliceous and volcanic rock eastward along the Roberts Mountains Thrust over Silurian and Devonian shallow-water carbonate rocks. Subsequent doming, faulting, and erosion have exposed the lower plate Devonian and Silurian carbonates through windows in the upper-plate Ordovician rocks. The various mines along the Carlin Trend are located within or adjacent to these windows.

According to Rota and Ekburg (1988), the Gold Quarry mine is located along the southern edge of the Carlin Window, a large structural feature similar to the Lynn Window that hosts the Carlin deposit 7 miles (11 km) to the north. The deposit is hosted by siltstone, shale, sandstone, silty limestone, and chert assigned to the Vinini Formation. These thin-bedded sediments have been locally argillized and highly silicified, particularly along high-angled structures.

Gold Quarry is divided into two ore bodies: the upper, structurally controlled Main ore zone and the lower, stratiform-replacement Deep West ore zone. Together, these ore bodies form a northwest-trending deposit 4000 ft (1219 m) long, 2500 ft (762 m) wide, and 1200 ft (397 m) deep.

The main ore body at Gold Quarry is a stockwork system along a series of north-northwest trending faults. Arsenic, antimony, copper, lead, nickel, and mercury all occur in elevated geochemical abundance throughout the deposit. Silver occurs in variable concentrations, with an overall gold/silver ratio of about 2:1.

Preexisting, high-angle, normal faults and associated fractures are the major controlling features of Gold Quarry mineralization. Mechanically induced fracture systems formed channels that directed the mineralizing hydrothermal fluids. This pro-

Fig 14.1.2.1. Newmont Gold Company. Gold Quarry mine.

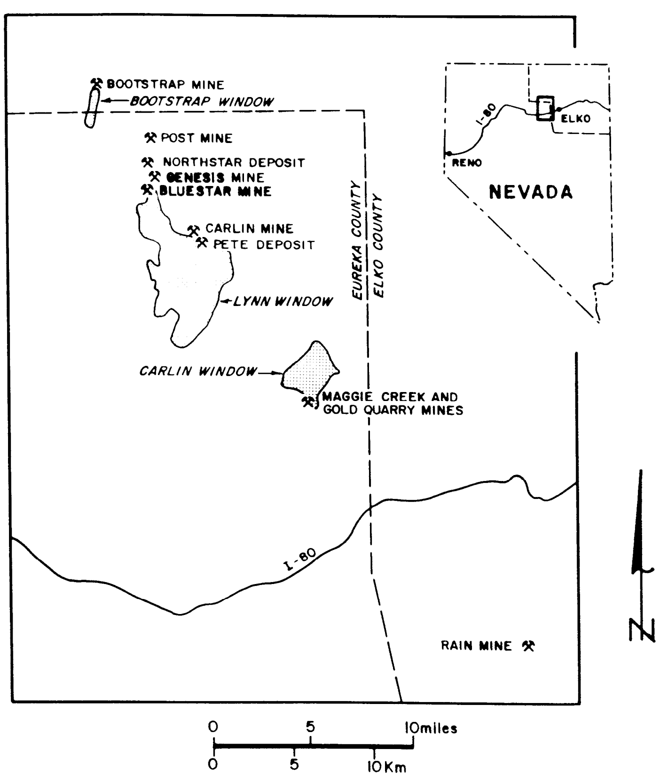

Fig. 14.1.2.2. Newmont Gold Company. Major gold deposits of the Carlin Trend.

duced the observed pattern of high-grade ore zones, concentrated along major structures, surrounded by overlapping lower-grade zones.

Submicroscopic-sized gold is disseminated throughout the shale, chert, and silty limestone of the Vinini Formation. The bulk of the near-surface waste material within the present pit is composed of lacustrine tuffs, siltstone, and sandstones of the Tertiary Carlin Formation. These sediments contain abundant montmorillonite and bentonite, which creates difficult mining conditions during inclement weather.

Hypogene alteration is so extreme that ore types are distinguished more by intensity of alteration than by primary lithology, which is often not recognizable. At Gold Quarry, silicification is the most pervasive type of alteration. Intense argillization followed and locally overprinted the earlier silicification.

Supergene oxidation followed formation of the deposit. Weathering removed most of the carbonaceous material from the upper ore body. Carbon occurring in the lower ore body is associated with calcareous lithologies.

The proven and probable ore reserves for the Gold Quarry mine, as of Dec. 31, 1988, were 246.1 million tons (223.3 Mt) at 0.042 oz/ton (1.4 g/t) and contain 10.3 million oz (320 t) of gold.

14.1.2.3 Mine Development

Several deposit models were developed by a joint effort of the Newmont Gold staff and the Technical Scientific Services group of Newmont Mining Corporation, using varying kriging techniques and parameters. The current kriged block model was developed then compared with 11 benches of production data to verify its accuracy. The model parameters used were 20-ft (6-m) assay composites compiled from atomic absorption (AA) and fire assays at 5-ft (1.5-m) intervals, 50- by 50- by 20-ft (15.2- by 15.2- by 6-m) blocks and the updated surface.

Several geologic zones were included in the model. The Main Zone contains disseminated stockwork mineralization controlled

Fig. 14.1.2.3. Gold Quarry ultimate pit based on cone Z130 and $400 gold.

by structure and the Deep West Zone contains disseminated replacement mineralization controlled by lithology and structure. The alluvium bedrock contact also is included in the model.

Fig. 14.1.2.3 is a plan of the Gold Quarry ultimate pit, which was produced using a computerized floating cone program. Parameters used to produce the ultimate pit outline include mining production unit costs, milling costs, leaching costs, and refining costs. Revenue factors included were the gold price, mill recovery, leach recovery, and royalty payments. Breakeven grades were included for cone definition on mill ore and leach ore. Pit physical design parameters included tonnage factors of 16.0 ft^3/ton (0.5 m^3/t) for alluvium and 13.2 ft^3 /ton (0.4 m^3/t) for rock. Slope angles are 45° (inter-ramp for bedrock), 35° for alluvium less than 300 ft (91.4 m) thick, and 30° where the alluvium is more than 300 ft (91.4 m) thick. Haul roads were designed 80 ft (24.4 m) wide with a maximum 8% grade.

A large portion of the prestripped waste at Gold Quarry was used to construct a tailings dam 3000 ft (914.4 m) long and 200 ft (61 m) high. The dam also serves as a connecting road from the mine to the mill, truck shop, and administration building.

Phased laybacks 300 ft (91 m) wide are planned to keep ore material exposed until the ultimate pit limit is reached. The larger shovels work a bench 30 ft (9 m) high in these phased layback areas of waste, while the ore benches are held to 20 ft (6 m) because of ore control concerns.

Around the ultimate pit perimeter, the alluvium waste varies from zero to more than 300 ft (91 m) thick. The pit slopes will be flattened as this unstable alluvium becomes thicker. An ongoing slope stability program monitors pit slopes, waste dumps, and dump leach pads for stability problems.

The number of working faces available for mining each shift varies from day-to-day, but on the average, six areas in the ore and leach grade material are kept ready for mining. Several areas in the waste material also are available. Each shift operates seven loading units.

14.1.2.4 Unit Operations.

DRILLING. There are eight rotary drills available for drilling blastholes at Gold Quarry. All of the drills are track-mounted, equipped with dust collectors that are a no-visible-emission type with pleated paper filter elements, pneumatic backflushing, and have heated water injection systems for positive dust control. The three oldest drills are Driltech D60KII, equipped with low-pressure Sullair 1200 screw compressors, 3408 Caterpillar engines, and have a maximum pulldown of 60,000 lb (267 kN). Four Ingersoll-Rand DM-M rotary drills are at Gold Quarry. Three of these drills are equipped with Ingersoll-Rand single-stage, oil-flooded, asymmetrical rotary-screw compressors of 1200 ft^3/min (40 m^3/min), and one drill has a 1400-cfm (47-m^3/min) capacity. Cummins KTA19C engines provide power for these units. These drills also have a pulldown capacity of 60,000 lb (267 kN). The eighth drill is a Reedrill SK-60II, equipped with a Sullair series 32 asymmetrical, screw-type, 1200 cfm (39.9 m^3/min) compressor, a Detroit Diesel 12v-71T engine for power, and has a pulldown capacity of 70,000 lb (3 11 kN).

The blastholes are drilled using a 7 7/8-in. (200-mm) tricone rotary carbide-button bit to a depth of 25 ft (7.6 m), which allows about 5 ft (1.5 m) of subgrade drilling. All of the drills utilize an integral drill steel, 30 ft (9.1 m) long, with either an API or Beco threaded box on each end. When the bit end becomes worn, the steel is turned end-for-end, which doubles the life of one piece of steel.

Each drill is operated by one operator drilling the 25 ft (7.6 m) without a steel change. A 14- by 14-ft (4.3- by 4.3-m) drill pattern is standard.

The drillhole spacing and the 20-ft (6-m) bench height is a function of ore grade control. A study is underway to determine the hole spacing and bench height that will optimize drilling, fragmentation, loading height, and ore control.

During 1988, the drills averaged 90 fph (27.4 m/hr).

BLASTING. Presently, two Amerind-McKissic 7-ton (6.4-t) pneumatic prill trucks are used for loading holes. Blasting-grade ammonium nitrate prills are loaded into the trucks from bulk storage bins. Diesel oil is sprayed onto the prills at the required 6% by weight as the prills are being pneumatically placed in the blastholes at a rate of 550 lb/min (204 kg/min) through a 35-ft by 3-in. (11-m by 76.5-mm) hose.

All holes are bottom primed with a 1.0-lb (0.453-kg) cast primer. Ensign-Bickford 25-grain E-cord is used as downlines to detonate the cast primers. Ensign-Bickford 18-grain Detacord is used for the trunk lines and cross ties. Displacement is minimized by using millisecond delays between the rows. The blast is initiated with two no. 6 caps and fuse that are attached just prior to blasting.

Wet holes are loaded with ANFO packaged in waterproof bags measuring 5.0 in. (127 mm) in diameter. Flat cast primers are used with 50-grain detonating cord. Bagged slurry also is used in some of the wet holes. A slurry loading truck will be required when mining reaches the water table.

Drill cuttings are used for stemming whenever possible. During the winter, stemming may be transported to the shot if the

Table 14.1.2.1. Production Loading Equipment

Make, Model and Type	Dipper Capacity	No. of Units	Productivity per Operating Hour
Demag H-285 Hydraulic Shovel	23 yd³ (17.6 m³)	2	1700 tph (1542 t/h)
DeMag H-185 Hydraulic Shovel	16 yd³ (12 m³)	1	1100 tph (1000 t/h)
Letourneau L-1100 Front-End Loader	22 yd³ (16.8 m³)	3	1150 tph (1043 t/h)
Letourneau L-1000 Front-End Loader	17 yd³ (13 m³)	1	1000 tph (907 t/h)
Caterpillar 992-C Front-End Loader	12 yd³ (9.2 m³)	5	825 tph (748 t/h)

Table 14.1.2.2. Mine Equipment List—Gold Quarry Mine (1989)

2	DeMag H-285 Hydraulic Shovels	23 yd³ (18 m³)
1	DeMag H-185 Hydraulic Shovel	17 yd³ (13 m³)
3	Letourneau L-1100 Front-End Loaders	22 yd³ (17 m³)
1	Letourneau L-1000 Front-End Loader	16 yd³ (12 m³)
5	Caterpillar 992-C Front-End Loaders	12 yd³ (9 m³)
3	Driltech 60K Rotary Drills Track Mounted	
4	Ingersoll-Rand DM-M Rotary Drills, Track-Mounted	
1	Reedrill SK60 Rotary Drill, Track-Mounted	
2	Dresser Wabco Haulpak Trucks Mechanical	100 ton (91 t)
18	Dresser Wabco Haulpak Trucks Electric	120 ton (109 t)
5	Dresser Wabco Haulpak Trucks Electric	140 ton (127 t)
11	Caterpillar 785 Trucks Mechanical	140 ton (127 t)
1	Caterpillar D10N Dozer	
2	Caterpillar D9L Dozers	
2	Caterpillar D8H Dozers	
1	Komatsu D375A-1 Dozer	
2	Caterpillar 834C Rubber-Tired Dozers	
2	Caterpillar 16G Motor Graders	
1	Caterpillar 14G Motor Grader	
2	Water Trucks	15,000 gal (57 m³)
1	Water Truck	12,000 gal (45 m³)

cuttings are frozen. Blasting is performed every day shift at lunch time.

Good fragmentation is obtained, and very little secondary blasting is required. The overall powder factor averaged 0.30 lb/ton (0.15 kg/t) during 1988.

LOADING. The mine plan calls for loading 205,000 tpd (186,000 t/d) of material. This is accomplished by using seven or eight of the loading units shown in Table 14.1.2.1 per shift. The remaining loading units are used as backup. Three of the front-end loaders (FELs) are used to feed the three primary crushers, in order to smooth out the feed between trucks and to blend ore from the stockpile at the crushers.

Two Demag H-285 shovels are used to strip alluvium in 30-ft (9.1-m) benches. Hydraulic shovels have been very successful working in the extremely muddy conditions which prevail during inclement weather.

All loading equipment is diesel powered and mechanically driven except for the four Letourneau FELs, which have electric wheel motors in all wheels. The FELs are very advantageous at Gold Quarry because of their mobility.

HAULAGE. There are 36 trucks available for haulage: two 100-ton (91-t) Dresser-Wabco mechanical drive Haulpak, eighteen 120-ton (109-t) Dresser-Wabco electric drive Haulpak, five 140-ton (127-t) Dresser-Wabco electric drive Haulpak, and eleven 140-ton (127-t) Caterpillar mechanical drive 785 trucks.

A computerized dispatch system added in April 1989 should help optimize truck haulage.

Everyday, haulage is required to deliver 25,000 tons (22,700 t) to either of two primary crushers supplying mill feed, 50,000 tons (45,400 t) to one crusher for leach material, and 130,000 tons (118,000 t) of waste to the waste dumps.

The haulage and loading units are equipped with two-way radios to facilitate communication between the dispatcher and pit supervision.

Haul roads are maintained with three motor graders and two rubber-tired dozers. Track dozers maintain the dumps and stockpiles.

A complete mining equipment list is provided in Table 14.1.2.2.

14.1.2.5 Product Control

While drilling blastholes, a 20-ft (6-m) composite sample of the blasthole drill cuttings is taken by the driller. A cylinder containing an open sample bag is placed in a holder and lowered through a hole cut in the drill deck 18 in. (460 mm) from the drill steel. The cylinder is lowered to the ground into the cuttings stream, and a representative sample of the cuttings is collected from every blasthole. The driller removes the sample before drilling the subgrade.

The sample bags were developed to be compatible with a new automated and computerized assay lab, which utilizes robotics throughout the assaying process. The bags are provided with a bar-coded label for easier and faster identification, and they are able to withstand temperatures of more than 250°F (121°C) encountered in the automatic sample drying process.

The samples taken from the blasthole drilling are logged into the assay lab where they are analyzed by fire assay methods to indicate total gold values and by atomic absorption methods of cyanide agitated solutions. A ratio of the atomic absorption assay to the fire assay indicates the approximate recovery that can be expected from this ore through the oxide mill circuit. Low-recovery ores are considered refractory. Further assaying is performed on this material to determine if the refractory nature is caused by carbonaceous material that adsorbs gold from the pregnant solutions (“preg-robbing”) or from the fine gold particles being encapsulated or locked-up by coarse-grained quartz. A solution with a known gold value is added to the sample from which the original atomic absorption assay was derived. The sample is reassayed and the results compared to the first assay. If the second assay does not equal the first assay plus the amount of gold added for the second assay, within established criteria, the sample is considered to be preg-robbing and will be identified as such in the kriging program and flagged in the field. The results of the fire assay, the atomic absorption, the recovery ratio, and the preg-robbing assay are automatically entered into the computer blasthole datafile after verification.

Blasthole locations, which are supplied by the mine engineering surveyors using total station surveying instruments, together with the blasthole assays, that are supplied by the computerized assay lab, are automatically entered into the blasthole datafile within the company computer system. A kriging program was developed by Newmont’s Technical Scientific Service group with preset kriging parameters and the blasthole datafile to produce a computer-plotted map that defines the ore, leach, and waste zones. The blast patterns are numbered, as are the different zones within the pattern for identification during mining. Coordinates outlining the different zones within the pattern are provided to the surveyors, who stake the various boundaries with color-coded lath after blasting to identify the field location of ore, leach, and waste. Also, a sign identifying each zone is placed in the field so that a loading unit can identify the zone being mined to the computerized dispatch system. As a zone within the staked boundaries is being mined, the loading unit signals the haulage unit either by a horn signal or by two-way radio to identify the destination for each load. The computerized dispatch system stores the number of tons and grade loaded by each loading unit and will track the haulage unit to ensure that the material is taken to the correct destination.

A five-year mine plan is generated each year by the mine engineering department. The first year of the plan is detailed by months, and the subsequent years are annualized. This five-year plan is used by all departments to develop a detailed cost and production budget, which is compiled into a five-year business plan.

An engineer using a graphics work station develops a rolling two-month detailed mine plan. This plan is complete with tabulated forecasted production quantities and maps showing the mining cuts and haulage access for each month.

The graphics work station contains the block model, the geological surfaces, the ultimate pit outline, and the current mining surfaces, based on end-of-month surveys. The monthly plans are the subject of weekly meetings held to discuss production vs. the mining plan with representatives from the mine engineering, mine operations, geology, maintenance, milling, and leaching departments. Adjustments are made to the monthly plan to compensate for higher or lower mine production and for changes in milling requirements.

14.1.2.5 Ancillary Facilities

WASTE DISPOSAL. The waste dumping area is approximately 2 miles (3.2 km) east of the Gold Quarry pit. Most of the waste material in the present pit consists of water-deposited sediments containing abundant amounts of montmorillonite and bentonite. This material is very unstable, and major failures occur when a dump height of 80 ft (24 m) is exceeded. The material deposited near the edge of the dump will slump vertically and run at the toe like wet concrete. A dumping sequence was developed to minimize these failures. This sequence generates a stair-step configuration where the lower lifts are advanced first, to keep the dumping height less than 80 ft (24 m).

Topsoil is stripped from the dumping areas and stockpiled for reclamation purposes. The final dump plan allows for an overall 3:1 slope. The edges of the dump lifts will be rounded, and the dump area will be reseeded.

MAINTENANCE. Maintenance facilities are located approximately 2 miles (3.2 km) from the mine. These facilities include an eight-bay truck repair shop, a tire repair and change area with a heated jacking pad, an enclosed truck washing building (high enough to allow the trucks to raise their beds), a drive-through lubrication bay, and a separate light-vehicle wash, repair, and lubrication shop.

Roving field mechanics cover every shift in the mine and repair minor items on all production equipment. These roving mechanics troubleshoot problems, repair small failures, and refer larger failures to the shop, which dispatches crews to the disabled unit. Haulage trucks are sent to the shop for repair, if possible.

Engines, transmissions, wheel motors, and other larger components are exchanged with vendor rebuilt units.

Consultants were used to implement a preventive maintenance (PM) program called "assembly-line preventive maintenance." The maintenance planner issues a maintenance schedule for all equipment based on operating hours. Every two hours, mobile equipment is brought to the wash bay and cleaned with high pressure water. After washing and cab cleaning, the equipment is moved to the lube bay for two hours of lubrication, oil changes, and filter exchange. The equipment is then moved to a PM bay for a third two-hour period, and a group of mechanics inspect, repair, check pressures, and install backlog items. At the end of this two-hour period the equipment is released back to operations, or if a major problem is found, the equipment is sidelined for repair.

A maximum of three units is in the preventive maintenance line at any one time. Shovels and drills are covered by a field preventive maintenance crew. Utilizing this system of PM, the mechanical availability, which includes the PM time, of the mining equipment has remained at an average of 85%.

When the front tires of the haulage units are about 60% worn, they are replaced with new tires, and the worn tires are rotated to the dual position for wear out.

A lubrication crew works with a lube truck and a fuel truck to fuel and lube the shovels and loaders when they shut down for a staggered lunch period. Haulage trucks report to a fuel station near the pit for fuel and daily lube.

A computerized warehouse is attached to the truck shop to facilitate checkout of replacement items.

Maintenance records, component histories, tire records, and maintenance scheduling are all kept on the main computer system.

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14.1.3 LOW-GRADE IRON ORE: EMPIRE MINE

J.W. VILLAR AND A.W. SWANSON

14.1.3.1 Mine Description

The Empire mine (Fig. 14.1.3.1), operated by the Cleveland-Cliffs Iron Co., has the capacity to produce 8 million long tons/year (8.1 million tpy) of high-grade iron ore pellets from a low-grade, magnetic ore. The pellets, which contain over 65% iron, are a desirable feed for blast furnaces located throughout the Great Lakes steel producing areas. The mine is located on the Marquette Iron Range, 2½ miles (4 km) south of Negaunee in the Michigan Upper Peninsula, not far from where iron mining first began in the US in 1847.

The Cleveland-Cliffs Iron Co. is a wholly owned subsidiary of Cleveland-Cliffs Inc., headquartered in Cleveland, OH. The Empire mine is owned by Inland Steel Co., Wheeling-Pittsburgh Steel Corp., and Cleveland-Cliffs. As manager of eight iron ore mines in North America and Australia for the ownership interests of Cliffs and various steel companies, Cleveland-Cliffs is the world's largest producer of iron-ore pellets.

The mine area (Fig. 14.1.3.2), including open pits, waste stockpiles, concentrator, tailing basins, water reservoirs, and pellet plant, covers 13,500 acres (55 km^2) and extends over 7 miles

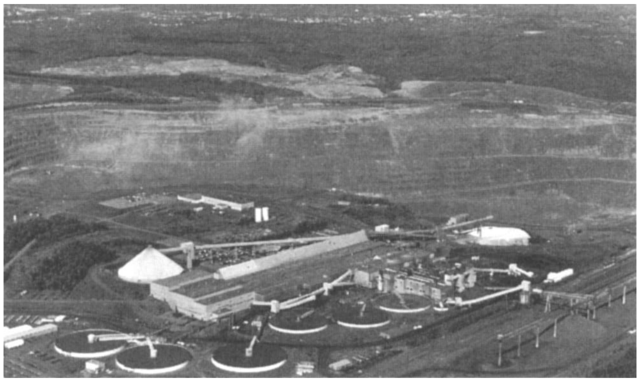

Fig. 14.1.3.1. Overview of plant and mine.

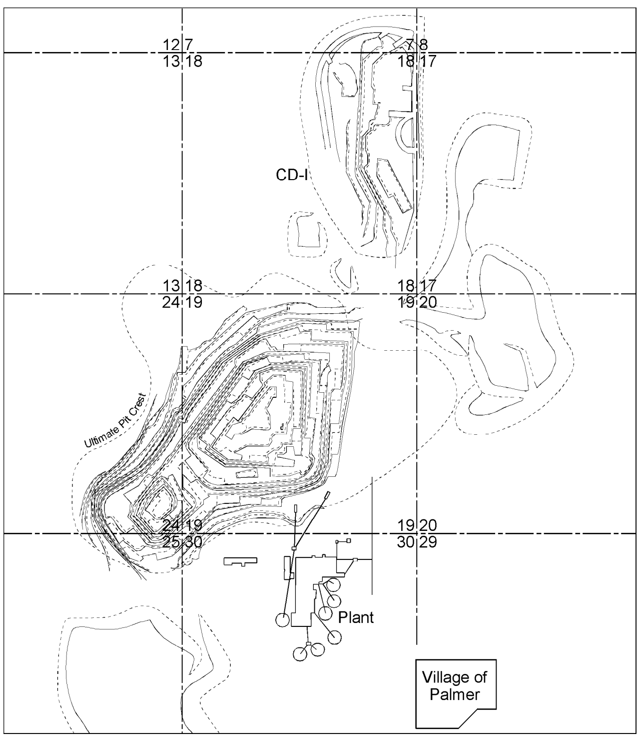

Fig. 14.1.3.2. General layout of Main pit, CD-I pit, and plant for the Empire mine (1990).

(11 km) north and south and 8 miles (13 km) east and west. The Main pit will ultimately have a surface area of 850 acres (3.4 km^2) and crest-to-crest dimensions of 5000 by 10,000 ft (1500 by 3000 m). The adjacent CD-I pit will cover 165 acres (0.7 km^2) with crest dimensions of 3000 by 5000 ft (1100 by 1500 m).

During 1989, over 61 million long tons (62 Mt) of total material was removed from the active mining areas. This consisted of 360,000 long tons (366,000 t) of unconsolidated surface overburden hauled to stockpiles, 35.2 million long tons (35.8 Mt) of rock stripping hauled to waste stockpiles, and 25.6 million long tons (26.0 Mt) of crude ore hauled to the crusher. A total of 8 million long tons (8.1 Mt) of pellets was produced from this ore. Of this production, 4.5 million long tons (4.6 Mt) were standard pellets assaying 65.0% iron, and 3.5 million long tons (3.6 Mt) were fluxed pellets containing 60.2% iron and 6.0% lime.

Operations at the mine and mill complex are scheduled to run continuously (365 d/yr). Total employment at the complex at year end in 1989 was 1214 persons, with 178 salaried and 1036 hourly. Of these, 28 salaried and 291 hourly were charged to the pit operating group. Pit maintenance included 21 salaried and 172 hourly, for a grand total of 512 persons.

14.1.3.2 Deposit Description and Geology

The Marquette Iron Range is located in Marquette and Baraga counties of Michigan. The topography is rugged, heavily forested, with numerous rock outcrops interspaced with lakes and swamps. The highest points are over 1200 ft (365 m) above nearby Lake Superior, which is at an elevation of 602 ft (183 m) above sea level. General elevations range between 1400 and 1600 ft (425 and 490 m).

The Empire ore deposit is in the lower portion of the Negaunee Iron Formation, which is part of the Menominee Group of Middle Precambrian Age. This member, which has a total stratigraphic thickness in excess of 3500 ft (1000 m), is and has been the major site of iron mining on the Marquette Iron Range. The ore body consists primarily of banded, magnetic, cherty carbonate with varying amounts of iron silicate and small amounts of hematite. The banding is due to alternating layers rich in iron oxide and silica.

Structurally, the major feature of the Marquette Range is a westward pitching synclinorium, open at the west end. The Empire mine is located in the southeastern part of the structure where locally the formation strike varies from northeasterly in the south end of the pit to northerly in the north. The beds dip to the northwest at about 30 to 40°. Local variations in strike and dip are common, with numerous drag folds plunging generally to the northwest. The ore zone is separated into a number of individual blocks by a series of dike-filled transverse faults. The ore body is overlain by glacial drift that in some areas reaches up to 150 ft (45 m) in thickness but generally averages 30 to 40 ft (9 to 12 m) thick. The lower portion of the minable zone grades into a predominantly silicate iron formation, which cannot be upgraded economically. The hanging wall is a lean subeconomic iron formation that contains occasional zones of ore-grade material. The footwall is composed of the Siamo slate, the member that lies conformably below the Negaunee and is exposed near the southeast crest of the Main pit. The iron minerals in the ore are magnetite, iron carbonates, iron silicates, martite, and earthy hematite with minor geothite and pyrite. In the Empire mine area, the basic iron formation has been altered by oxidation, enrichment, and low-grade metamorphism. This has increased the acid-soluble iron content to 33%, from an average of 26% in unaltered areas. The economic iron mineral is very fine-grained magnetite, requiring grinding to 90% minus 500 mesh (0.028 mm) for liberation. Enrichment continues to below economic mining depths, as indicated by diamond drilling showing good ore grades to depths of 1740 ft (530 m).

The ore grades within the minable zone vary from a low of 13% iron as magnetite (cutoff grade) to a high of 32%. The average mill head in 1989 was 23.36% iron as magnetite and a total soluble iron of 33.99%. It is necessary to carefully schedule the number and location of the shovels to be operated each shift so that a uniform grade of crude ore is presented to the mill. In addition to the magnetite content, grain size, and liberation characteristics of the crude ore, scheduling daily production also depends on analysis of the iron concentrate that will be produced and amenability of the crude ore to autogenous grinding.

A portion of the annual waste rock movement consists of the overlying, low-grade iron-formation hanging wall, although approximately two-thirds of the total waste rock is produced from interbedded waste zones within the ore horizons. The cutoff grade for ore is greater than 20% concentrate weight recovery, greater than 65% iron, and less than 7.5% silica in the concentrate, as determined by analysis of diamond drill core from exploration drilling and production blasthole rotary drill cuttings. Zones of waste or ore less than 20 ft (6 m) in thickness are not mined separately and are included with the adjacent material.

Exploration drilling, totaling in excess of 250,000 ft (76,000 m), has been done on a 300- by 300-ft (91- by 91-m) grid. The majority of these holes are vertical with about 20% drilled at an angle to better define the geology.

As of Jan. 1, 1990, the Empire mine had an estimated reserve of 775 million long tons (787 Mt) of crude ore remaining in the present design of the pits currently being mined. To mine this tonnage of crude ore, an estimated 448 million long tons (455 Mt) of waste rock and 34 million long tons (34.5 Mt) of surface overburden must be removed from the pit. A total of 226.5 million long tons (230.1 Mt) of standard pellets with an average analysis of 64.9% iron can be produced from this reserve for an estimated mine life of 28 years at the current production rate.

The specific gravity of the ore and waste rock varies with the soluble iron content. To estimate reserve tonnages, a regression equation based on extensive test work is used to calculate the conversion from volume to weight. It is $13.45 - (0.0792 \times \%\ \mathrm{Fe}) = \mathrm{ft}^3/\mathrm{long\ ton}$. For surface overburden, tonnages are based on a specific gravity of 2.0, or 1.5 long tons/yd^3 (2.7 or 2.0 t/m^3).

Ultimate overall pit slopes for the Main pit are designed to be 49° on the west hanging wall and 37° on the east footwall. Slopes for the north and south walls are to be 55°. Ultimate depth of this pit will be 1500 ft (460 m).

As of Jan. 1, 1990, the Empire mine had produced a total of 393.7 million long tons (400.0 Mt) of crude ore and 349.7 million long tons (355.3 Mt) of waste rock and overburden stripping. From this, a total of 122.8 million long tons (124.9 Mt) of pellets has been produced.

In addition to the ore reserves in the pits presently being mined, there are several other ore bodies in the immediate area that are held as long term reserves. In total, they contain in excess of 250 million long tons (254 Mt) of crude ore, from which over 72 million long tons (73.2 Mt) of additional pellets could be produced.

14.1.3.3 Mine Development

Initial pellet production at a rate of 1.2 million long tons/year (1.22 million tpy) began in late 1963 after an almost two-year construction period. Expansions to 3.2 million long tons (3.25 Mt) of annual capacity occurred in 1966, to 5.2 million long tons (5.28 Mt) in 1974, and to 8 million long tons (8.13 Mt) in 1980. Total capital investment for the mine and plant was $408 million.

Original production from the ore body occurred during the period 1907 to 1950 when a total of 6,364,133 long tons (6,466,258 t) of direct-shipping, low-grade siliceous ore was mined from several small pits by other operators. In the 1940s, the Cleveland-Cliffs Iron Co. began assembling the present property and, in the mid 1950s, started exploration and metallurgical testing of the ore. Bulk grinding tests using both conventional rod and ball mill grinding and autogenous milling were run at the Cleveland-Cliffs pilot plant. In 1961, a pilot plant using full-size autogenous grinding equipment was erected on the property. Over a five-month period, 148,000 long tons (150,000 t) of crude ore were treated, showing it was possible to liberate the fine-grained iron mineral by grinding to 90% minus 500 mesh (0.028 mm). With confirmation that the process was commercially viable, construction of the concentrator and pelletizing facilities began in early 1962. Preproduction stripping, conducted in 1962 and 1963, removed 2.5 million long tons (2.5 Mt) of overburden before mining of crude ore began. Initially, mining at the rate of about 11,500 long tons/day (11,700 tpd) on a single-shift, five days/week basis, provided the 3 million long tons/year (3 million tpy) of crude ore necessary for the first stage of plant capacity. Currently, all material movement is budgeted at 57,000 long tons/shift (57,900 t/shift), 21 shifts/week. This results from 24,000 long tons (24,400 t) of crude ore and 33,000 long tons (33,500 t) of waste rock. On an annual average, an additional 400 long tons/shift (410 t/shift) of surface overburden also are moved.

Bench heights are 45 ft (13.7 m) in both ore and waste, with 15 benches presently active. Ore is being produced from almost all benches, with major waste movement concentrated on the upper benches. The ultimate pit crest has been developed in portions of the southwest extension and on the footwall side of the Main pit, with further expansion of the pit crest planned to the north, northwest, and east. Working benches are generally maintained at a minimum width of 180 ft (55 m), but they are being expanded to a width of 300 ft (91 m) in stripping areas to accommodate a new 33-yd^3 (24-m^3) shovel that will soon be placed in operation.

14.1.3.4 Unit Operations

DRILLING AND BLASTING. Drilling is conducted 21 shifts/week on the same schedule as the balance of the mine. Drilling is performed with five large track-mounted rotary machines using 9-7/8- and 16-in. (251- and 406-mm) tricone bits with carbide inserts. The drills are electrically powered at 4160 V through trailing cables. A 1000-ft (300-m) reel is mounted on each machine; although up to 5000 ft (1500 m) of cable may be used.

The majority of holes are drilled 50 ft (15 m) deep, providing 5 ft (1.5 m) of subdrilling below the 45-ft (13.7-m) bench height. Machines are equipped with 60-ft (18-m) masts, allowing single-pass drilling of all holes. Average penetration rate with 16-in. (406-mm) bits is 50 fph (15 m/h), with an actual average production per shift of 210 ft (64 m). The 16-in. (406-mm) bits are used for production blastholes, and the 9-7/8-in. (251-mm) bits are used for drilling final pit walls. Average shift production with the smaller bits is 345 ft (105 m). A single operator per machine is employed. Bit life for 16-in. (406-mm) bits averages 5100 ft (1550 m) and 3200 ft (9800 m) for 9-7/8-in. (251-mm) bits.

Approximately 60% of the holes are loaded with bulk ANFO and the balance with aluminized emulsions. Emulsions are used primarily in wet holes. The average powder factor is 0.37 lb/ton (0.17 kg/t) broken. Loading of holes is performed by a contract explosive supplier with separate bulk trucks for ANFO and emulsion under the direct supervision of mine employees.

Two 16-oz (0.45-kg) boosters are used to prime each hole, with the bottom booster located at the bench grade and the upper booster midway between the grade line and top of the charge. Blasts are fired in echelon or staggered patterns with 17-ms delays between holes and 50 ms between echelons. In-hole delays of 400 ms also are used. Holes are fired using a noiseless nonelectric system.

Waste rock, crushed to –7/8 in. (22 mm) for use as haul-road surfacing, is used for stemming with 26 to 28 ft (8 to 9 m) used in each hole for production blasting. Presplit 9-7/8-in. (251-mm) holes, which are loaded with about 135 lb (61 kg) of powder, are blocked off with an expansion bladder 12 ft (4 m) below the collar of the hole, and then the balance of the hole is stemmed with crushed rock.

Drill patterns in ore and most waste rock are typically 40 ft (12.2 m) burden and 42 ft (12.8 m) spacing when using 16-in. bits (406-mm). In certain diabase-sill waste-rock areas, patterns are reduced to 38 by 40 ft (11.6 by 12.2 m). Individual blast sizes vary widely, depending on pit location and production requirements, with over 1 million long tons (1 million t) broken in the larger ones and 300,000 long tons (305,000 t) in an average size production shot. Blast patterns are generally composed of four or five rows parallel to the length of the bench containing a total of 80 to 100 holes.

At the final pit wall, a single row of presplit 9-7/8-in. (251-mm) holes are drilled on 15-ft (4.6-m) centers. These lines are shot, and then buffer patterns of 9-7/8-in. (251-mm) holes, three to four rows wide, are drilled. These patterns are generally 22 by 30 ft (6.7 by 9.1 m) with the final row 17 ft (5.2 m) behind the next row and 10 ft (3 m) in front of the presplit holes.

Blasting is normally scheduled for 12:10 pm each weekday to take advantage of the general shutdown for lunch break. An average of 3.9 blasts is fired each week.

Seismograph readings at one or two locations are taken for each blast to confirm that ground movement is held below levels that could cause damage beyond property boundaries. As a general rule, weather conditions have not been a concern with regard to air-blast damage and have not been a factor for blast scheduling.

The drilling and blasting program at the Empire mine is designed to minimize the total mining cost by balancing increased breakage costs against savings in loading and haulage. This practice results in well-broken ore, but care is taken not to overblast as it is necessary to ensure sufficient coarse material in the mill feed to provide grinding media for the autogenous mills. Secondary breakage is minimal, and no additional drilling or blasting is done. Secondary breakage is performed with a drop ball with one 50-ton (45-t) rubber-tired crane scheduled one shift on weekdays.

LOADING. The Empire mine shovel-loading fleet is made up of nine 11-yd^3 (8.4-m^3) machines and four 9-yd^3 (6.9-m^3) machines. The units are powered through trailing cables up to 5000 ft (1500 m) long at 4160 V. Essentially all production loading is performed by these machines, with only a minor amount of pit clean up being done with front-end loaders (FELs).

The productivity of 11-yd^3 (8.4-m^3) machines averages 7300 long tons (7400 t) per operating shift, while that for the 9-yd^3 (6.9-m^3) units is 4700 long tons/shift (4800 t/shift). Nine or ten of the machines are scheduled to operate 21 shifts/week for a utilization of almost 75%. The age of this fleet varies from 8 to 27 years and averages about 14.

A single operator is assigned to each shovel with all maintenance and lubrication done with mobile lube units. When bench width exceeds 120 ft (37 m), double-side loading is normally practiced.

A fleet of seven rubber-tired dozers is utilized to maintain the loading areas around the shovels by cleaning up spills and throw rock. Normally four units are scheduled on each shift, with each machine assigned to maintain loading around two or three active shovels.

HAULAGE. The production truck haulage fleet consists of thirty 120-ton (109-t) units and ten 170-ton (154-t) units. In addition, there are six 85-ton (77-t) trucks which are used as utility units to handle road dressing, flux stone, concentrates, and pellets. All are diesel-electric, single-drive-axle units.

The 120-ton (109-t) units move an average of 1600 long tons/shift (1630 t/shift), and the 170-ton (154-t) trucks move 2700 long tons (2740 t). About 75% of the trucks are scheduled for operation on each shift.

Haul distances average 1.5 miles (2.4 km) for ore and waste. Haul roads are 110 ft (34 m) wide with a maximum grade of 8%. Haulage roads are constructed by cutting to the desired elevation with the primary loading shovels. High spots or poorly broken areas are brought to grade by ripping with a dozer. Haulage roads are then surfaced with graded crushed rock. Approximately 500,000 long tons (510,000 t) of waste rock are used for this purpose annually. This material is produced by a contractor who crushes 100% through a 7/8-in. (22-mm) screen. Tire life averages 4.6 tires/million long tons of all material moved.

A fleet of four graders is utilized in maintaining the 41 miles (66 km) of haulage roads within the pit and serving the crushers and waste stockpiles. On average, three units are scheduled on day shift and two on the off shifts.

All truck movement is directed by a radio dispatch system using a computerized program to maximize truck utilization. Radios are installed in all haulage units, shovels, and dozers and at locations along the haulage roads to report actual unit movements. The dispatch system also is used to notify truck drivers when their units require fueling by keeping track of operating time. When 16 hours of operating time have been accumulated, the driver is instructed to refuel at either the fueling station near the crusher or one adjacent to a waste haul road.

Waste rock is hauled to one of three stockpile areas, dependent on material type and haul distances. Waste rock is dumped short of the stockpile crest and pushed over the edge with a dozer to maximize both safety and haulage output. The crude ore is hauled to one of two 60-in. by 89-in. (1.52-m by 2.26-m) gyratory crushers where it is reduced to –9 in. (229 mm). From the crusher, the broken ore is conveyed to ore storage facilities that have a total capacity of 610,000 long tons (620,000 t), or enough for about 8 days of operation.

14.1.3.5 Ancillary Facilities

MILLING. Crushed ore from the surge piles is withdrawn by under-pile-feeders and conveyed to 24 lines of two-stage, fully autogenous milling where the final concentrate is ground to 90% – 500 mesh (0.028 mm) (Fig. 14.1.3.3). The primary mills include

3 8500-hp (6300-kW), 32-ft (9.8-m) diam., 16.5-ft (5.0-m) long
5 3500-hp (2600-kW), 24-ft (7.3-m) diam., 12.5-ft (3.8-m) long
16 2200-hp (1600-kW), 24-ft (7.3-m) diam., 8.0-ft (2.4-m) long

The primary autogenous mills are closed circuited with double deck-vibrating screens. Undersize material is magnetically cobbed, with the magnetic portion going as feed for the pebble grinding circuit and 48% of nonmagnetic material discharged to tailing. The top deck provides pebble media for the secondary circuit and the intermediate product is recirculated.

The pebble mills include

16 1500-hp (1100-kW), 12.5-ft (3.8-m) diam., 24-ft (7.3-m) long
5 2200-hp (1600-kW), 15.5-ft (4.7-m) diam., 24-ft (7.3-m) long
6 2500-hp (1900-kw), 15.5-ft (4.7-m) diam., 32-ft (9.75-m) long

The pebble mill grind is controlled by cyclones with the underflow going to siphonsizers where another 13 to 15% is rejected to tailing. The siphonsizer underflow is further upgraded with magnetic finishers and flotation to 66.9% iron and 5.3% silicon.

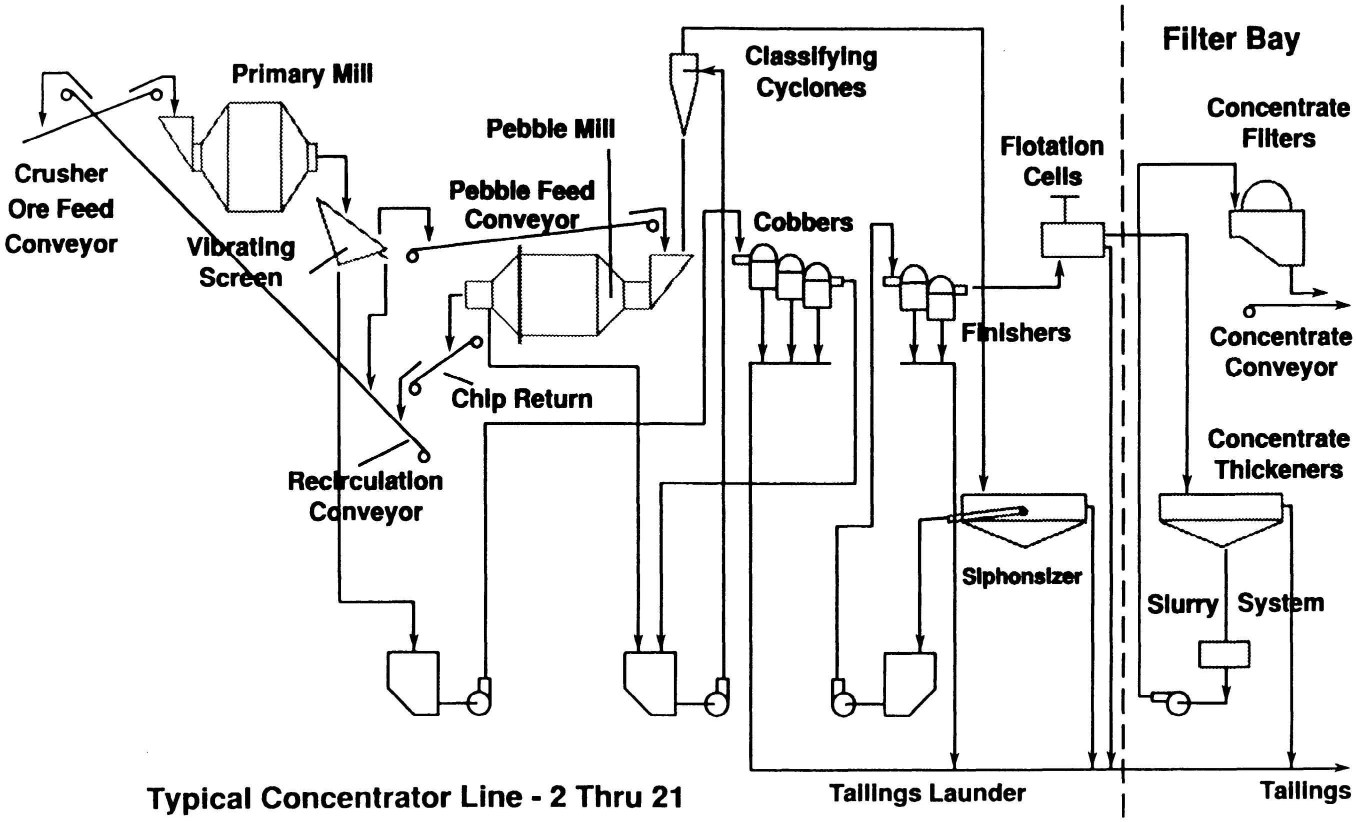

Fig. 14.1.3.3. Simplified flow diagram for concentrator and filter bay.

The concentrate is thickened, filtered, and conveyed to the pelletizing plant where it is balled and indurated in one of the four grate kiln pelletizing units. The fired pellets, which are heated to temperatures of 2330°F (1275°C), are cooled and conveyed to rail car loading bins or stockpiles for later shipping.

The plant is equipped to produce up to 3.2 million long tons/year (3.3 million tpy) of fluxed pellets with facilities to receive and grind 500,000 long tons/year (510,000 tpy) of limestone and dolomite as additives to the green balls.

TAILING DISPOSAL. The combined tailings discharged from the plant are thickened to about 45% solids and pumped at the rate of 12,000 gpm (0.76 m^3/s) through 18,000 ft (5500 m) of 26-in. (660-mm) pipe to a tailing basin located 4 miles (6 km) southwest of the plant site. This basin, which is adequate for the disposal of tailing until the year 2002, covers 3050 acres (12.3 km^2). At completion, the tailings will be almost 100 ft (30 m) in average thickness.

The initial basin site has little natural relief, requiring dikes to be constructed around essentially 100% of the perimeter by the time ultimate height is reached. These dikes were initially built using locally available glacial till borrow. In areas where a long tailing beach is developed and the pond is a long distance from the dike, dikes are raised by the upstream method, using tailings. In areas where the pond is adjacent to the dikes, dikes are raised downstream with borrow material.

Decant water from the tailing basin is further clarified in secondary and tertiary settling basins before return to the plant for reuse. Additional new water at the rate of approximately 10,000 gpm (0.63 m^3/s), to make up losses in the process, tailing disposal, and evaporation, is obtained from a separate reservoir that impounds spring runoff and freshet flows.

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14.1.4 SOUTH TEXAS URANIUM: RHODE RANCH MINE

DONALD K. WINSOR

14.1.4.1 Mine Description

The Rhode Ranch mine is a joint-venture mining operation between Chevron Resources Co. and Total Minerals Corp., a subsidiary of Total Compagnie Francaise des Petroles, with Chevron being the operator. A total of 1,500,000 lb (680,000 kg) of uranium oxide will be recovered annually after a 1,000,000-lb (454,000-kg) first-year objective. To produce 1,500,000 lb (680,000 kg), 4.2 million bank yd^3 (3.2 million m^3) of overburden, 220,000 tons (200,000 t) of cap rock, 1.3 million tons (1.2 Mt) of internal waste, and 300,000 tons (272,000 t) of ore will be moved.

The mine is located in McMullen County, 50 miles (80 km) northwest of Corpus Christi, TX, some 30 miles (48 km) west of George West. Overburden removal began in late 1987 and ore production began in January 1988. The uranium-bearing sand is processed at the Chevron mill in Panna Maria, TX, 85 miles (137 km) northeast of the mine site.

The permitted area covers 4400 acres (17.8 km^2). There are 63 people employed at the mine. The overburden crews work four 10-hour shifts/week, two shifts/day. The two mining crews alternate working seven days on and seven days off. They work 80 hours during the seven-day period. Maintenance crews are set up to cover the overburden and mining shifts.

14.1.4.2 Deposit Description and Geology

The Rhode Ranch deposit is one of the approximately 100 known uranium occurrences that define the South Texas Uranium Region. This area consists of an arcuate belt measuring 250 miles (400 km) long and 40 miles (64 km) wide. Total production from both open pit and in situ leaching operations and remaining reserves amounts to approximately 100 million lb (45.4 million kg) of uranium oxide. Economic concentrations of uranium ore are hosted mainly in the Eocene Age Jackson Group. The Oligocene Age Catahoula Formation, the Miocene Age Oakville Formation, and the Goliad Formation of Pliocene Age form only a modest interval of the more than 50,000 ft (15,240 m) of interbedded Tertiary marine and non-marine sediments that compose one of the largest accumulations of clastic Cenozoic sediments in the world. The constant loading of sediments into the Gulf Coast Basin produced instabilities that led to the development of local growth faulting. In many cases, displacements on the growth faults arranged sands and shales in juxtaposition and formed ideal traps for hydrocarbon accumulations. The fault plane subsequently provided a pathway for the upward migration of hydrogen sulfide-bearing gases and solutions into overlying sediments. This invasion of sulfide-rich solutions into shallow aquifers is a critical aspect of the localization and concentration of uranium in the South Texas Uranium Region.

Uranium at Rhode Ranch is localized into ore rolls hosted within the Manuel and Rincon Members of the Miocene Age Oakville Formation. The mineralized sediments are fine- to medium-grained quartz arenites to arkosic arenites with interstitial digenetic calcium carbonate cement. The Oakville sediments are interpreted to have been deposited as bed-load and mixed-load channel fill and associated sheet flow splay sands that are bounded by flood plain muds and silts.

Hydrogen sulfide-bearing solutions emanating from a fault, located roughly parallel to the roll fronts and about 0.5 miles (0.8 km) to the southeast, are thought to be responsible for engendering the plume of pyritic sediments capable of reducing and precipitating the uranium and causing subsequent rereduction of all geochemical cell-related oxidation. Coffinite is identified as the dominate uranium phase with subordinate amounts of a brannerite-like mineral. Pyrite and/or marcasite are believed to encapsulate much of the uranium. The lack of recent oxidation of the ore body has allowed radiometric equilibrium conditions to be established. Structurally the sediments are flat-lying with a minor southeastward dip toward the Gulf of Mexico. Aside from the growth faults, the sediments are structurally undisturbed.

Ore reserves are estimated to be 7 million lb (3.2 million kg) of recoverable uranium oxide. At the projected rate of 1,000,000 lb (454,000 kg) in 1988 and 1,500,000 lb (680,000 kg) annually thereafter, the ore body has a mine life of five years. There are other uranium occurrences in the vicinity that may be mined, depending on the outcome of current grade evaluations and future uranium prices.

14.1.4.3 Mine Development

Initial mining operations began by removing the topsoil and placing it along the pit perimeter in a continuous berm for runoff control and reclamation. A box cut was made, starting at the extreme north end of the main ore body (see Figs. 14.1.4.1 and 14.1.4.2). The overburden was placed in a stockpile to be used for final reclamation. A topsoil berm was placed around the stockpile area to collect run-off water for dust control use. The waste embankments can be seen to the right of the pit in Fig. 14.1.4.1. The dark areas are shadows cast by the highwalls.

Fig. 14.1.4.1. Aerial photograph, Rhode Ranch mine.

Overburden was ripped, and scrapers were push-loaded to heaped capacity. Sloper blades on two dozers kept the highwall smooth for ground control as the overburden was removed.

The cap rock in the initial box cut was thin enough to be ripped and pushed off the ore sand by dozers. After mining was completed, the cap rock was pushed back into the mined-out area. The cap rock thickness normally varies between 4 and 20 ft (1.2 and 6.1 m) and averages between 8 and 12 ft (2.4 and 3.7 m). This material must be drilled and blasted.

When the cap rock was removed, the ore was mined by a backhoe that made cuts 2- to 3-ft (0.6- to 0.9-m) deep across the uncovered area until the bottom of the ore sand was reached. Ore control personnel probed the sand and backhoe buckets using hand-held scintillometers as mining took place. Trucks hauled either to the ore pad at the top of the pit or to mined-out areas, depending on the grade of material.

As stripping and mining progressed, it was possible to begin backfilling mined-out areas. This practice substantially limited the amount of material that needed to be handled more than once. After only one year of operation, 90% of the overburden was being replaced as permanent backfill.

14.1.4.4 Unit Operations

OVERBURDEN REMOVAL. Stripping operations (Fig. 14.1.4.3) utilize Caterpillar (Cat) 651 scrapers push-loaded by two Cat D-9 push dozers and one Cat D-10 dozer (see equipment

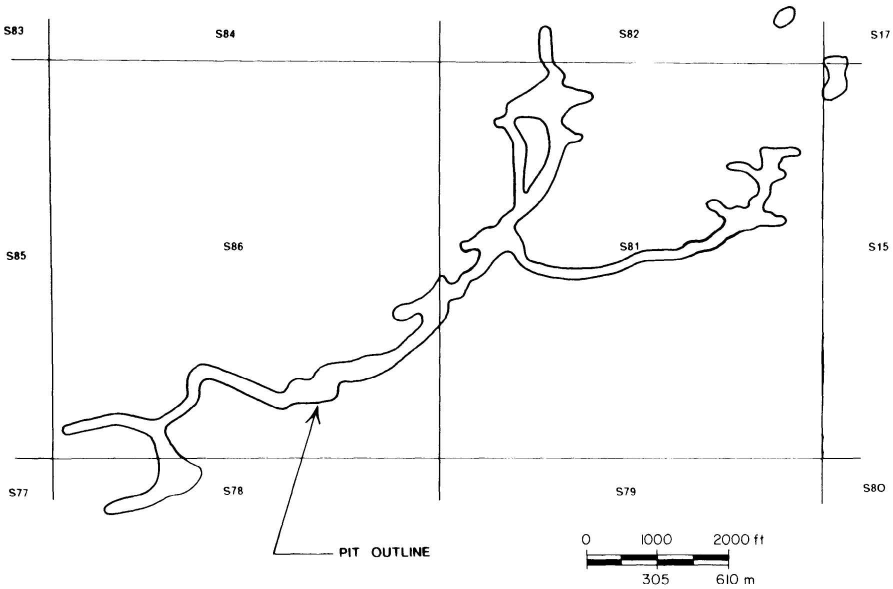

Fig. 14.1.4.2. Pit outline.

Fig. 14.1.4.3. Stripping operations—overburden removed and rock exposed for drilling.

list, Table 14.1.4.1). The scrapers haul to either the waste stockpile or to backfill. The overburden removal ramp serves as one of the accesses to the mine and is kept at a maximum 10% grade to facilitate ore hauling to the ore pad. The overburden is ripped by D-9 or D-10 dozer/rippers prior to removal. The highwall is groomed using sloper blades mounted on the D-9s. The stripping crew also operates a motor grader to keep roads and ramps in good driving condition and a water wagon as needed for dust control.

Table 14.1.4.1. Mine Equipment List (1989)

Make, Model and Type	Number	Usage
Cat 651 scrapers	11	Stripping, reclamation
Cat D-9 dozer	11	Stripping, mining
Cat D-10 dozer	1	Stripping, mining
Cat 16G grader	2	Stripping, mining, and reclamation
Water wagons, 8000 gal	2	All roads and pit
Cat 245 backhoe	2	Mining
Cat 769 truck	3	Mining
G-D 3100 track drill	1	Drilling caprock
I-R ECM 360 track drill	1	Drilling caprock

Each scraper moves an average of 2500 bank yd^3 (1900 m^3) for each 10-hour operating shift and operates for approximately 190 days/year. Rain-outs were projected at 15% but were less than 5% during the first year of operation. Day shift normally operates three scrapers with six on night shift. The highwall varies from 50 to 120 ft (15 to 37 m) to the top of the cap rock.

CAP ROCK REMOVAL. Drilling the cap rock is accomplished by utilizing one of two drills and occasionally both drills operating together. The drills include a Gardner-Denver Model 3100 track drill pulling a G-D Model 800 portable air compressor and an Ingersoll-Rand ECM 360 self-contained drill. The ECM 360 is preferred because of its faster penetration rate and because the air compressor is a part of the machine.

Hole size is 3.5 in. (90 mm), and the depth varies with the thickness of the cap rock. Hole burden is 7.5 ft (2.3 m), and hole spacing is 7.5 ft (2.3 m). The optimum blast is 300 holes (25 rows with 12 holes/row). Blasts have been made with as few as 80 holes and as many as 700 holes.

Cap rock removal is the responsibility of the mining crew, so the whole crew participates in loading each blast. The optimum size blast takes less than five hours to load and shoot.

Each hole is primed with a 1 1/8- by 6-in. (29- by 150-mm) stick of a non-nitroglycerin emulsion cartridge and detonated by a nonelectric delay cap. ANFO is used as a blasting agent, with 3 to 5 ft (0.9 to 1.5 m) of 3/4-in. (20-mm) gravel stemming, depending on the depth of the hole. The holes in each row are initiated by detonating cord placed along the row. The first row is detonated by an instantaneous electric cap. All subsequent rows are initiated by a nonelectric delay connecting the end of each row to each end of the next row. Approximately 24 lb (11 kg) of explosive is loaded in each hole, resulting in an average powder factor of 2 tons/lb (4 t/kg).

There are high-pressure natural gas pipelines, shallow oil wells and their pipelines, and a gas well either within the pit limits or in close proximity to the pit. In order to meet regulations limiting ground motion due to blasting, initiation was changed from whole row, using detonating cord, to nonelectric delays in each hole. Fragmentation also was improved by this change.

When two mining faces are available, D-9 dozers push the cap rock into the mined-out area adjacent to the blast while the second face is being mined. When only one mining face is available, the backhoe and trucks may have to help move the cap rock due to space limitations. Probers follow the cap rock removal down to the ore sand to determine when the ore is reached. Occasionally, there is ore in the bottom of the cap rock, which is mined if above cutoff grade.

ORE REMOVAL. Mining is accomplished using a Cat 245 backhoe loading two Cat 769 trucks. As the ore sand is very loosely consolidated, blasting is not required. Mining of ore and waste takes place approximately 300 days/year. This schedule includes days lost due to holidays, training, loading and blasting, and rain-outs. With the ratio of 4.5 tons of waste/ton of ore, the mining crew removes 4300 tons/day (3900 tpd) of waste and 1000 tons/day (907 tpd) of ore per day of mining operations.

A 2- to 3-ft (0.6- to 0.9-m) cut is taken by the backhoe for better grade control and more selective mining.

ORE HAULAGE. Ore is taken from the mine to the crest of the pit and dumped on an ore pad for transshipment to the mill at Panna Maria, some 85 miles (137 km) away. Transportation of the ore is accomplished using a contract hauler. The equipment used consists of single and tandem belly-dump trailers. The contractor furnishes and operates the loader to load the trucks.

RECLAMATION. Enough of the pit had been backfilled to the original contour by the end of 1988 to begin reclamation. The reclamation plan requires systematic sampling of the backfill for radiation. The topsoil is to be replaced to the original approximate thickness using the Cat 651 scrapers and the Cat 16G motor grader for final smoothing. Coastal Bermuda grass will be planted in the drainage areas to help control erosion. A mixture of Kleingrass, Buffel grass, and Bell Rhodes grass will be seeded on the rest of the area. Cattle grazing is the primary use of the land. A pond will be left at the extreme northeast end and near the southeast end of the ore body in existing drainages for run-off control and livestock watering.

14.1.4.5 Product Control

From drillhole data, the geology department makes l-ft (0.3-m) contour maps of the ore. These maps are called "slice maps" and are used by the shift foreman and prober to assist in grade control.

Fig. 14.1.4.4. Probing material in backhoe bucket for grade control.

Grade control is accomplished by a prober who follows the backhoe and monitors buckets and the sand being removed (Fig. 14.1.4.4). The cutoff grade for ore/waste separation was 0.08% U_2O_3 until late 1989, when it was changed to 0.10% U_2O_3 for economic reasons. Averaging the prober's results gives the daily grade of ore produced. Another prober drills 2-ft (0.6-m) holes with an auger ahead of the backhoe. These holes are probed for grade and marked by colored flags as to ore/waste grade. Samples are taken of the cuttings and checked in an X-ray analyzer as additional control.

14.1.4.6 Ancillary Facilities

Milling of the ore is performed at the Chevron mill in Panna Maria utilizing a two-stage acid leach and solvent extraction. Recoveries are in the 96 to 97% range. At a milling rate of 70 tph (64 t/h), the mill operates 14 to 15 days/month yielding a total of 24,000 tpm (21,800 t/m).

A 10-acre (40,500-m^2) site was established outside the mining area for all maintenance, personnel, and office facilities. A 120- by 190-ft (36.7- by 57.9-m) building was constructed to house the shop, warehouse, tool room, lunch room, men's and women's restrooms/change rooms, and offices for staff, engineering, geology, safety/first aid, and maintenance. Accounting and major purchasing are performed at the Panna Maria offices.

The shop has four drive-through bays and is equipped with two 2-ton (1.8-t) and two 10-ton (9.1-t) overhead cranes, air compressor, and a welding area. A field maintenance crew takes care of minor repairs and fueling of the mobile equipment. The shop crew performs major repairs in the shop. A night crew works with the night stripping crew for minor repairs in the pit and in the shop for major work as time permits. A fueling station for scrapers is located at the shop/office site. All other mobile equipment is fueled in the pit area.

14.1.5 WESTERN PHOSPHATE: HENRY MINE

DAVID W. FARNSWORTH

14.1.5.1 Mine Description

The Henry mine is located in the heart of the Western Phosphate Field, just east of the small town of Henry, ID (Figs. 14.1.5.1 and 14.1.5.2). The mine provides the necessary phosphate ore for Monsanto Company's elemental phosphorus furnace plant located 18.5 miles (29.8 km) south, near Soda Springs.

One challenging aspect of Monsanto's operations is the complicated surface and mineral ownership illustrated on Fig. 14.1.5.3. Two federal phosphate leases (ID-011451 and ID-

Fig. 14.1.5.1. North Henry mine area showing backfilled pit in the center (dark area). Aerial photo.

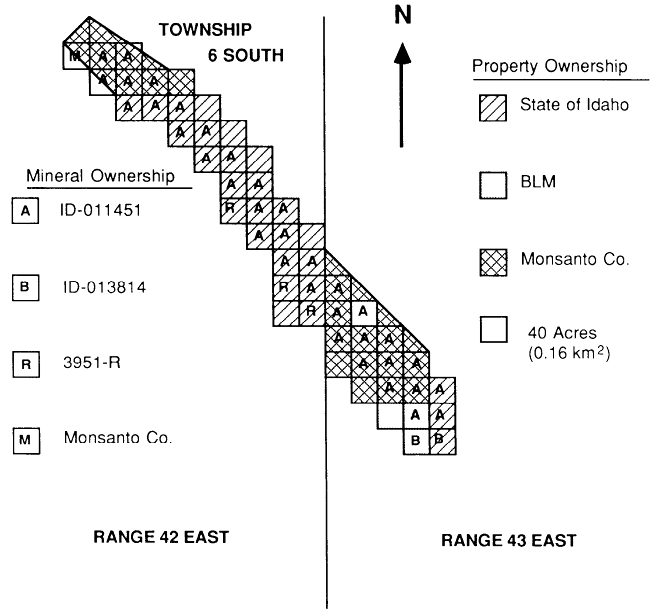

Fig. 14.1.5.3. Henry mine, mineral and property ownership.

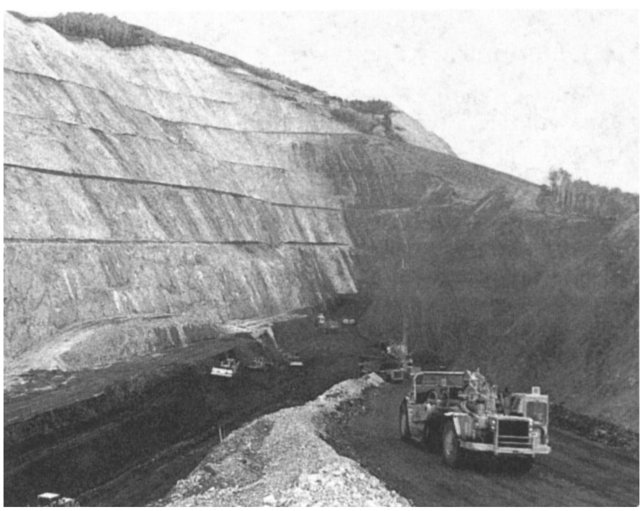

Fig. 14.1.5.2. General pit workings, 1989.

013814) and one State of Idaho phosphate lease (3951-R) comprise the mineral ownership of most of the deposit. In addition to the lease areas, a small unit of private mineral under lease to Monsanto, occupies the extreme northern end of the mine area. In total, some 1642 acres (6.6 km^2) are under lease or control.

Surface ownership is divided among the Bureau of Land Management (BLM), the State of Idaho, and the Monsanto Co. A surface easement from the State of Idaho Department of Lands covers 1080 acres (4.4 km^2). The federal phosphate lease allows for use of 120 acres (0.5 km^2) during mining, and a "phosphate use permit" allows for use of an additional 40 acres (0.2 km^2). Monsanto Co. owns the remainder of the surface estate.

Monsanto's mining operations in southeast Idaho began in 1951 at the Ballard mine. As that deposit was nearing depletion, operations were begun at the Henry mine in 1969. Operations have continued there at an approximate production rate of 1,000,000 tpy (907,000 t/a) of ore. Annual waste volumes vary greatly from year to year. Waste requirements vary, depending upon topography, dip of the ore structure, haul lengths, required ore volumes, and other factors. Total ore and waste volumes range from a low of only 2.5 million yd^3 (1.9 million m^3) to a high of about 5 million yd^3 (3.8 million m^3).

Monsanto's personnel are responsible for all aspects of leasing, exploration, environmental monitoring, permitting, mine design, contractor oversight, and reclamation. Actual mine operations and maintenance are under the direction of Monsanto's mining contractor, Dravo-Soda Springs. The contractor is responsible for overburden removal, ore mining, and delivery of ore to Monsanto's elemental phosphorus plant.

Waste removal is scheduled for two 10-hour shifts each day, four days each week. Ore is normally mined only during the day shift. Ore haulage to Monsanto's plant is limited to the warm-weather months, typically beginning early in May and ending by mid-October. To complete the haul within this period, two 10-hour shifts, five days/week are scheduled. In order to cover both mine operations and the ore haul, maintenance personnel are split into three crews. These crews also work 10-hour shifts, with the third crew overlapping portions of the other groups' hours.

14.1.5.2 Deposit Description and Geology

The Henry mine is located along a 5-mile (8-km) long outcrop of the Phosphoria Formation in Caribou County, ID. The Phosphoria outcrops near the base of a northwest-trending ridge formed of Wells Limestone. To the north and east, the ground slopes gently into Enoch Valley. The deposit is terminated by the Henry fault on the north and east sides. To the south, the deposit is terminated and offset to the east by another smaller fault system. The ore strikes approximately N35°W and dips to the northeast at 40 to 80°.

The Phosphoria Formation is composed of Permian marine sediments. In the mine area, the formation is composed of two members, a lower phosphatic shale member and an upper Rex Chert Member. The phosphatic shale member is composed of interbedded black mudstone, phosphatic mudstone, limestone,

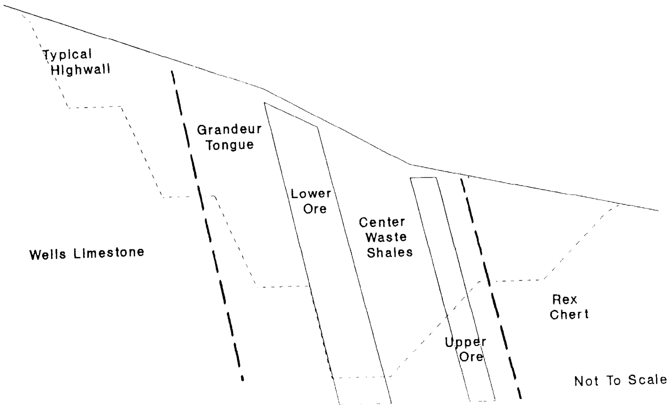

Fig. 14.1.5.4. Typical cross section for the Henry mine area.

and oolitic phosphate rock. Usually, the shale member is not exposed in outcrops but is eroded to form a characteristic swale between the underlying Wells and Grandeur Tongue Formations and the overlying Rex Chert Member. A typical cross section in the mine area is shown on Fig. 14.1.5.4.

At the Henry mine, two major ore horizons are recognized within the phosphatic shale member. The lower ore zone extends from about 5 ft (1.5 m) above the Grandeur Tongue/Phosphoria contact upward for about 40 to 50 ft (12.2 to 15.2 m). The other zone, termed the upper ore, is typically some 80 ft (24 m) above the top of the lower ore zone and is approximately 20 ft (6 m) thick. The strata between the ore zones consist of extremely low-grade phosphatic shale, mudstone, siltstone, and limestone beds. These units are collectively referred to as the center shales or center waste shales.

The other member of the Phosphoria Formation, the Rex Chert Member, consists of bedded dark gray and black chert and cherty mudstone. Much of the chert weathers to brown and reddish brown. Locally, the chert is highly weathered and fractured.

Underlying the Phosphoria Formation is the Permian Grandeur Tongue Limestone and Permian/Pennsylvanian Wells Formation. The Grandeur is composed of limestone, dolomite, and some chert beds. It ranges from 80 to 100 ft (24 to 30 m) thick. In some locations it is a very hard, consistent, and competent formation. The Upper Wells, which also is encountered in the footwall, is composed of a calcareous quartz sandstone with scattered limestone and chert beds.

14.1.5.3 Mine Development

The initial step in developing an area prior to mining is the removal of vegetation. Any timber is offered for sale to a private logging operator. In most areas, insufficient volumes of merchantable timber are present to make logging economical. Douglas fir and quaking aspen, along with most of the other smaller underbrush, are knocked down with dozers, piled, and burned.

Following removal of vegetation, the recoverable topsoil is removed. In most areas of the Western Phosphate Field, the true topsoil is only about 12 to 18 in. (0.3 to 0.5 m) thick. No differentiation of prime topsoil and subsoil is attempted as the thickness of individual units varies greatly over a mine area. When other nearby areas are at the stage in reclamation where they can be covered with planting medium, the topsoil from a new mining area is loaded in scrapers and direct-placed on the reshaped land. However, in most cases the topsoil is stockpiled for use following completion of mining in the area.

With the vegetation and topsoil removed, the area is ready for the start of normal mining operations. Waste removal begins at the high point of the limestone footwall. Lifts of 25 ft (7.5 m) are excavated in the waste until the mining floor encounters the lower ore section. At this time, operations shift from the limestone to the center waste shales and a 20-ft (6-m) lift of shale material is removed, leaving a lift of ore ready for mining. The ore is mined and the process of developing lifts of lower ore repeated until the upper ore zone is encountered. The upper ore zone is mined in a similar manner. The waste material between the upper ore and the toe of the hanging wall is removed first to expose a lift of ore.

During the period from May to October, removal of the limestone and some chert is minimized, and up to 75 vertical ft (23 m) of waste material is left in place for later removal. Operations during this period concentrate on ore mining to maximize ore quality and minimize rehandle on the stockpile. Removal of the limestone and chert backlog is then scheduled for the winter months when the blocky material can be moved more efficiently.

Individual ore beds, or in some cases, a series of similar grade ore beds, are mined separately. The lower ore zone, which ranges from 40 to 50 ft (12 to 15 m) in thickness, is mined in as many as nine different units. Individual splits are made upon such factors as chemistry, grade, and hardness. The minimum minable thickness is considered to be 0.75 ft (0.23 m); although some units of 0.5 ft (0.15 m) may be handled separately. This extreme level of selectivity is achieved by utilizing a slope board or side arm attached to the side of a dozer blade. The slope board is hydraulically controlled by the dozer operator who can manipulate the arm to match the dip of the bedding. Thin beds are knocked down in order to allow the scrapers to load the material. Thicker beds are pushed down sufficiently to yield the width necessary to operate the scraper fleet on top of the section.

Slope design is very critical in achieving design depths in the highly weathered materials encountered in the Western Phosphate Field. Just as the ore material is highly leached and altered, so too are the adjacent limestone and chert wall rocks. Monsanto Co. conducts an extensive drilling and mapping program to sample wall materials and ascertain bedding structures. Bedding in the limestone is inconsistent, and often weathering of the strata has eliminated the actual bedding structure. Leaching, coupled with the stresses induced during uplift, has produced a very broken chert mass. Since mining is limited primarily to the altered shales, the shale material is of very low strength.

Intact rock strengths of the limestone and dolomite are extremely high, as high as 24,000 psi (155 MPa). However, analysis of core recovered from the footwall indicates few if any areas with significant amounts of intact rock. Therefore, fracture strengths are used in slope designs. Measured fracture strengths in the limestone vary from a low of about 5 psi (34 kPa) to a high of 18 psi (124 kPa). Because of the highly fractured nature of the chert, recovery of suitable core samples for measurement has been limited. Analysis of the recovered core indicates a cohesive strength of only about 3 to 5 psi (21 to 34 kPa). The ore and center waste shales are very highly weathered, with little or no compressive strength.

The footwall is designed utilizing the measured fracture strength of the limestone and by mirroring the bedding structure of the lower ore zone. A trial design, based on past designs and actual construction, is proposed and tested utilizing various computer simulations. These programs yield probability of failure and factor of safety estimates for the proposed design. The wall design is tested for both overall integrity and bench-sized failures. A typical footwall design calls for a bench 40- to 45-ft

(12- to 14-m) wide spaced 60 to 80 ft (18 to 24 m) vertically. Slope angles between benches are a maximum of 60°. The slope is laid back to the angle of repose through the highly weathered, near-surface material. A typical highwall design is shown in Fig. 14.1.5.4.

The hanging wall design is typically based upon the measured joint angles in the chert section. In most areas of the Henry mine, the primary joint angle is at 70 to 90° to the bedding plane. The wall angle must provide for removal of most of the material along the joint plane while minimizing the total volume of waste required to be moved. In most of the Henry mine pits, the hanging wall angle is about 45°. The hanging wall is not benched because of its lower overall height and physical characteristics that make stable benches difficult to maintain. In some pits, an access roadway of 50 to 70 ft (15 to 21 m) in width has been included in the hanging wall design. Minor failures during mining may reduce the safe operating width of these roadways to 30 ft (9 m) and one-way traffic.

The foregoing is a general discussion of average wall design conditions. Specific conditions have allowed for areas of much steeper designs. In one area, a pit included a section of the Grandeur Formation where the detrimental effects of alteration were nearly absent. By increasing the slope of the footwall from an average of about 40° to an overall angle in excess of 50°, the entire design was incorporated into the material of the Grandeur Tongue Formation. On the other hand, in another area, the chert had been weathered away and replaced by alluvium to a depth of about 150 ft (46 m). Included in the alluvium were several basalt seams 2 to 10 ft (0.6 to 3 m) thick. The extent and depth of alluvium were missed by exploratory drilling. During mining, failure of some 50,000 yd^3 (38,000 m^3) of the alluvium occurred. Because mining had progressed to a sufficient depth, and the adjacent location of ore stockpile areas limited "recut" operations, an alternate solution had to be developed to allow mining to continue down-dip. The failure and adjacent intact areas were extensively modeled and analyzed to determine the cause of the original failure. Based upon the results of this analysis, a new design for the remaining hanging wall was developed that required no additional removal of waste material but still recovered the majority of the ore present. In addition, a rock fence was designed and implemented to capture pieces of basalt that were loosened by minor failures in the underlying alluvium.

14.1.5.4 Unit Operations

The Henry mine employs both scraper loading and haulage as well as truck and shovel operations.

DRILLING AND BLASTING. Drill and blast operations are conducted on an "as needed" basis. The limestone and chert material both require blasting. At depth and for shovel loading, portions of the center waste shale section are lightly blasted. Equipment consists of two Reich blasthole drills. One is mounted on a crawler chassis, and the other on a rubber-tired truck chassis. Drillholes are normally spaced on a 16 by 16 ft (4.9 by 4.9 m) pattern in both the limestone and chert. Holes are 7-7/8 in. (200 mm) in diameter. In the shales, the pattern is adjusted to conditions and material hardness. Both drills have steel measuring 30 ft (9.1 m) long and can drill holes 25 to 28 ft (7.0 to 8.5 m) deep.

Approximately 50 to 65% of all waste materials require blasting prior to loading. Most blasting is performed during the fall and winter months. Bagged ANFO is used almost exclusively, as few if any holes are wet. Powder factors range from about 0.3 lb/yd^3 (9.2 kg/m^3) in the altered shales to 1.0 lb/yd^3 (0.5 kg/m^3) in some areas of the Grandeur Limestone. The altered shales are "bumped" (lightly blasted) to increase shovel productivity. Patterns and powder factors are adjusted frequently based upon the degree of alteration in the rock. The ore is not blasted prior to loading.

Fig. 14.1.5.5. Slope board used to recover thin, steeply dipping layers of ore.

As a supplement to blasting in waste and at all times in the ore, four Cat D-9H, two D-9L, and one D-10N crawler tractors are used. Most of these units are equipped with double-shank rippers. Following blasting, all scraper material is ripped to further aid productivity. In most cases, ripping of the well-altered waste shales is sufficient. Individual ore beds are separated utilizing one of the D-9Ls or the D-10N. These dozers have hydraulically controlled side-arm blades or slope boards (Fig. 14.1.5.5) that are used to knock down the individual beds creating a zone of sufficient width to allow scraper operations. During mining of the ore, it is frequently ripped to aid scraper loading.

LOADING AND HAULAGE. Prior to 1987, all loading and hauling of both ore and waste was accomplished by a fleet of Cat tractor scrapers. In March 1987, a Hitachi 801 hydraulic shovel and a fleet of Cat 777B trucks were added. A used Cat 992 front-end loader (FEL) was added as backup to the shovel. Subsequently, blocky waste will be scheduled for the shovel; the scraper will be used in the shales and ore. During the summer months, some shale and minor amounts of ore are moved with the shovel to maximize its utilization and productivity. Shovel production averages about 600 yd^3/hr (460 m^3/h). For the first two years of operation, the shovel averaged over 96% availability and over 97% utilization of available time. An integrated shovel/scraper operation is illustrated in Fig. 14.1.5.6.

Loading of the scrapers is accomplished using either a Cat Quad D-9 unit or the Cat D-10N series tractor as a pusher. Three Quad D-9s are available for use. Typically, the scrapers are divided into two spreads with a quad for loading and a D-9L dozer for support. Nearly all loading is done with the push dozers. This action maximizes loading efficiency while minimizing tire damage. Since quad units are no longer available, a Cat D-10N tractor is being evaluated as a pusher for eventual replacement of the quad.

The scraper fleet consists of eight Cat 651B single-engine scrapers and four Cat 657B dual-engine units. The 651 units were originally put to work in 1976 and by 1989 averaged in excess of 31,000 operating hours. The current fleet of 657 units was purchased in 1981 and 1983 to replace four older 657s which had been in continuous service since 1970. These units average over 16,500 operating hours each. Because of the age and number of available scrapers, availability ranges from 60 to 90% and utilization from 35 to 70%.

Five Cat 777B trucks were selected for use with the hydraulic shovel. These are 85-ton (77-t) units and have 10-in. (0.3-m) side

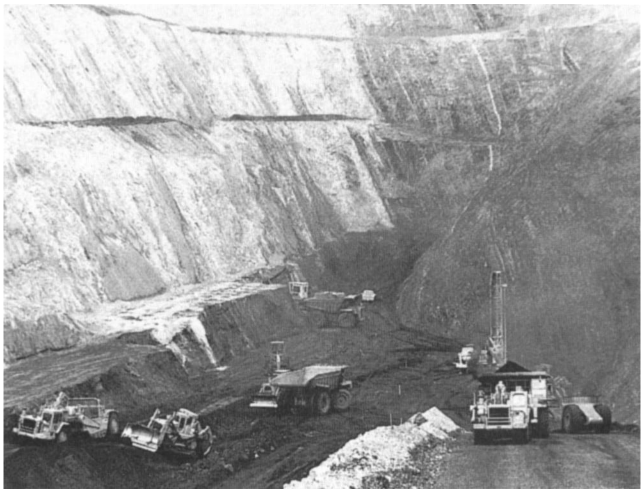

Fig. 14.1.5.6. Integrated shovel and scraper operations.

Fig. 14.1.5.7. Henry tipple and ore transport truck and trailers.

boards. Weight studies show a typical load to average about 80 tons (73 t). These units were put to work in 1987 and averaged about 4200 operating hours by 1989. Availability averages about 94% and 67% utilization of available time.

The equipment used on any particular day is dependent upon the available materials, haul distances, and weather conditions. The shovel is matched with sufficient trucks to maximize its output; typically this requires three trucks. Because of the extreme weather conditions encountered, truck operation in winter storms may be shut down and the operators transferred to scrapers. The scraper fleet also is adjusted to reflect operating conditions. During winter storms and pioneering operations, a larger number of all-wheel-drive 657s must be used. As grades flatten and hauls lengthen, more 651s are utilized. Two scraper spreads are worked on a typical day. Each consists of two 657s and five 651s.

The usual assortment of patrols, water trucks, dozers, portable light plants, and other equipment support pit operations. In addition to the dozers used in scraper support, one or two units/shift are scheduled to cover waste dumps, shovel cleanup, and scaling of the highwalls. During the warm weather months, an 8000-gallon (30-m^3) water truck is used for dust control on all pit roadways. A Cat 16G patrol is used for maintenance of haul roads.

Ore mining is conducted only during daylight hours to maximize ore recovery and quality control. As noted previously, the individual ore beds are knocked down utilizing the side-arm-equipped dozers. The ore is loaded into the scrapers and hauled to the adjacent tipple stockpile area. There it is put in three graded ore piles. During the period of May through October, the ore may be placed in stockpiles or placed directly over the loadout hoppers, depending upon the grade of material being shipped to the plant. The volume of ore mined during the off season is minimized because of quality problems encountered during winter mining and the need to minimize the volume of ore requiring double-handling.

The final operation at the Henry mine is screening and loading ore into the haul trucks for transportation to the plant (Fig. 14.1.5.7). Ore from the stockpile area is dozed into two hoppers buried in the stockpile pad fill. From there it is conveyed out and over two single-deck screens. Oversized material is directed to a rotating trommel with the undersized product falling directly into loadout hoppers. Some ore accompanies the oversized material in the form of high moisture balls or as a surface coating on limestone seams that are too thin to be economically separated during the mining process. The rotating action of the trommel is usually sufficient to break down the balls of ore and clean the ore from the waste material. The fine ore falls through the sides of the trommel and is conveyed back to the screens. Oversize material coming from the trommel is backfilled with the normal pit waste.

The haulage fleet for the haul to the plant consists of seven specifically designed tractors and 14 belly-dump trailers. Each trailer has a 70-ton (64-t) payload. Three of the tractors are newer units equipped with 750-hp (559-kW) Cat engines and specially designed Clark transmissions. These units were built by Kenworth and Pacific specifically for this operation. Each of these units pulls three trailers. The remainder of the tractors were built in 1970 to pull three trailers down the haul profile to the plant from the original tipple site. Each of these units, however, is capable of pulling only two trailers from the present tipple. In current practice, two older tractors pulling two trailers are used along with the newer units. The final trailer is coupled to one of the older tractors and maintained on standby for use when repairs to one of the main trucks extend beyond one shift.

14.1.5.5 Product Control

As with many sedimentary deposits where the ore zone is restricted to specific lithologic units, quality control is simplified greatly. However, production of a quality product is still of the highest priority. Most ore/waste designations are obvious. Analysis of the core obtained during exploratory drilling is used to predict each area's overall chemistry and general mining scheme. Ore unit designations are made based upon the chemistry and hardness characteristics of the core and verified during operation.

Pit sampling during mining is used to update and monitor the original chemistry forecast. Detailed trench sampling of both major ore units is conducted on about 300-ft (90-m) centers for each mining lift. The samples on the next lift will be offset by 100 ft (30 m). In this way, trends in chemistry can be monitored. Additionally, faulted zones are sampled to determine if normal lithological grade correlations can be maintained or if downgrading of one or more units is needed to maintain quality limits of each stockpile. Periodic grab samples are taken on the ore face and stockpile to ensure that proper cleanup of waste seams has taken place.

In order to track mining progress, a minimum of five complete field surveys of the active working area is conducted each year. Progress against final wall designs is surveyed each time the pit floor drops 10 to 15 ft (3 to 4.6 m) vertically. Each lift of ore is surveyed as to the location of the top and bottom of each major bed. At the end of one year, a detailed map is prepared and submitted to the BLM showing all disturbed and reclaimed areas.

The surveys are monitored by mine planning personnel for deviations in geology and progress. Problems noted are evaluated for possible adverse effects on wall stability and reconciliation of ore and waste volumes. Typically, during the third quarter of each year, an operating plan, which produces sufficient ore for the coming year's anticipated elemental phosphorus production, is developed. This plan is reviewed and updated at year-end based upon the actual surveys.

14.1.5.6 Ancillary Facilities

Placement of overburden materials is an important part of the planning and mining process. It is Monsanto's goal to provide for the return of disturbed lands to premining uses as soon and as efficiently as practical. In order to achieve this, the planning of material movement is critically important. Every effort is made to maximize backfilling of each pit area without the rehandling of waste materials.

Each area is designed to minimize the volume of waste rock placed in outside waste piles. These outside waste areas are prepared by removing vegetation and topsoil. This provides for the most stable base practical. Coarse limestone and chert are placed in the core of the waste pile. The altered center waste shales are layered on the sides and top. A minimum covering of 3 to 5 ft (0.9 to 1.5 m) of altered shale material is placed over all areas. Research efforts by Monsanto, the US Forest Service Intermountain Research Station, and other western phosphate producers have shown the altered waste shales to be an effective growing medium for reclamation plantings. In this way, the small amount of topsoil recovered from the pit and outside waste piles is saved for use after the cessation of operations on roads, stockpiles, service areas, and other disturbed lands. The waste piles are designed to be rounded in shape with a maximum side slope of 3 ft (0.9 m) horizontal to 1 ft (0.3 m) vertical. These areas blend well with the surrounding natural features and are easily reclaimed.

The adjacent mined-out areas are targeted for most of the waste material. Once again, the coarse limestone and chert material is dumped into the bottom of the pit and the altered shales placed over it. The available material is strategically placed to minimize the visual effects of mining. Material is placed higher along the highwall faces leaving a shallow swale in the middle of the now partially backfilled pit. Slopes are targeted at 18°, but some areas are allowed to go as steep as 24°.

All disturbed areas are reclaimed as soon as practical after mining. Placement of altered shales and final shaping is an integral part of the mining process. Reclamation crews complete only minor shaping and seedbed preparation. Although artificial fertilization is not a necessity, almost all areas are fertilized utilizing a tractor-drawn spreader. The addition of artificial fertilizer aids in rapid plant growth and the prompt control of runoff. The area is typically seeded with a Brillion seeder-packer. On flat or gently sloping terrain, the fertilizer spreader and seeder are pulled by a farm type tractor. On steeper slopes or where the seedbed may be rocky, a Cat D-5 tractor is used. Seeding is done along the contour on slopes of up to 18°, and up and down the slope on steeper terrain. The seed mix used consists of a mixture of grasses, legumes, and forbs.

Two maintenance work areas are utilized at the Henry mine. The mining contractor's office and major shop facilities are located at the Ballard mine. Since this site is now some 6 miles (9.7 km) from the active mining area, a separate yard and outdoor service area are located adjacent to the active pit. During the summer months, almost all maintenance is conducted at the mine, with the Ballard shop handling only the mine-to-plant ore trucks and major overhauls. During other periods, more maintenance work is shifted to Ballard. The Ballard shop has three drive-through bays (each of which is sufficient to handle an ore haul truck with two trailers), a small two-story warehouse, and an outside wash area. These facilities were originally constructed in the mid-1960s and are no longer adequate to service the current mining fleet. Height and width considerations preclude using the Ballard shop for servicing the Cat 777Bs, and distance constraints will not allow servicing of the hydraulic shovel at this facility. For these reasons a new shop will be constructed near the active mining areas in the near future.

14.1.6 STEEPLY DIPPING BITUMINOUS COAL: KEMMERER MINE

GEORGE B. WHITMAN

14.1.6.1 Mine Description

The Kemmerer mine (Fig. 14.1.6.1), owned and operated by The Pittsburg & Midway Coal Mining Co. (P&M), is located 3 miles (5 km) southwest of the town of Kemmerer, WY (Fig. 14.1.6.2). Coals produced are high-quality subbituminous B with the following characteristics: 9900 Btu/lb (23,027 kJ/kg), 0.6% sulfur, 5.0% ash, 39.0% fixed carbon, 36.0% volatile matter, and 20.0% moisture. Underground mining at this property commenced on a small scale as early as 1881 and continued intermittently until 1964. Underground production was replaced entirely by the current surface operations, which began in 1951.

Surface coal reserves are approximately 120 million tons (109 Mt), while the probable underground reserves are at least 500 million tons (454 Mt) (as of 1989).

The 10,000 acres (40 km^2) under permit consist of a series of active and proposed pits that extend in a north-south direction for approximately 7 miles (11 km), from the FMC mine on the south to US Highway 30 on the north. Annual production is in the range of 3 to 4 million tons (3 to 3.6 Mt). Approximately two-thirds of the production is shipped by conveyor to a mine-mouth power plant; the balance is shipped by train to smaller industrial customers. The mine uses shovels and trucks to move 15 to 20 million bank yd^3 (11.5 to 15.3 million bank m^3) of waste rock on an eight-hour shift, three-shift day, five-day week, and approximately 250 days/year.

There are three mining areas that include 1-UD started in 1971, 2-UD in 1983, and I-Area in 1988. Each of these areas is

Fig. 14.1.6.1. Aerial photograph of the Kemmerer mine (1-UD pit) looking south.

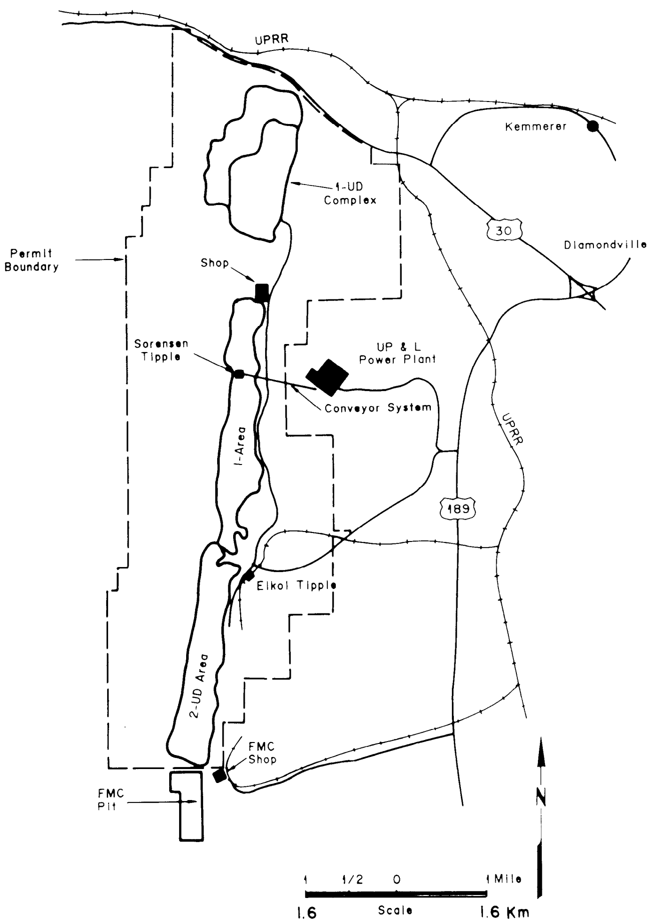

Fig. 14.1.6.2. Map of the Kemmerer mine and surrounding facilities.

separated from other areas by either geologic features or conditions left by earlier mining activities.

14.1.6.2 Deposit Description and Geology

Two coal-bearing formations exist in the immediate area (Fig. 14.1.6.3), the Frontier and the Adaville. The Frontier Formation has three major seams ranging from 5 to 15 ft (1.5 to 4.6 m) thick. The Adaville has up to 30 different seams ranging from 2 to over 100 ft (0.6 to over 30 m) thick. These two formations are separated by the Hilliard Formation, which is about 5000 ft (1500 m) thick.

The Kemmerer mine is situated within the Adaville Formation. The Adaville is approximately 2900 ft (880 m) thick and is Late Cretaceous in age. The youngest sediments in this formation are possibly of Tertiary Age. The Adaville consists of gray interbedded sandstone, siltstone, and clay with carbonaceous clay zones and coal seams. The sediments strike generally north-south and dip to the west from 18 to 28°.

During Late Cretaceous time, this portion of the North American Continent was located at 60° latitude. At this time, the area was a large delta-prodelta-swamp environment. A vast sea existed to the east, and volcanically active mountains existed to the west. The complex geology within the Adaville Formation was caused by the constant shifting of the delta and the transgression and regression of the sea.

In the Early Tertiary age, major structural changes occurred in this area, creating what is known today as the Overthrust Belt. Located approximately 1 mile (1.6 km) west of the mine, the Absaroka Thrust Fault displaced up to 15,000 ft (4600 m) of sediments and is one of several faults of this nature. At the same time, the faults were active, and considerable folding also occurred. As a result, the Adaville Formation was tilted into its present position.

14.1.6.3 Mine Development

The entire upper horizon of topsoil is removed prior to mining and overburden disposal. The topsoil is strategically stockpiled for later reapplication to reclaimed areas, or topsoil is replaced immediately if adjacent reclaimed sites are available.

Within each mining area are a series of pits, and within each pit are a number of horseshoe-shaped cuts open to the east. The cuts, as defined at Kemmerer, extend vertically from the surface to the lowermost seam. Since most of the coal is concentrated in the three lowest seams (i.e., No. 1 through No. 3), these seams must be available when developing less-productive portions of adjacent cuts.

Scheduling of adjacent cuts within the individual areas and combining area productions results in desired coal flow and uniform costs. Each cut must be scheduled carefully to achieve balanced coal production by combining high-ratio benches with low-ratio benches that intersect the bottom three seams. This combination of cut ratios for all mining areas is referred to as the overall phase ratio.

1-UD AREA. The 1-UD area (Fig. 14.1.6.1) is permitted as a special bituminous operation, allowing total out-of-pit spoil and minimum in-pit reclamation. As permitted, the entire pit was intended to be left open. However, an acquisition of the adjoining reserves to the north will result in a revised mine plan that will employ the original 1-UD pit for disposal.

Production benches are at 33-ft (10-m) intervals with 50-ft (15-m) safety benches at 100-ft (30-m) vertical intervals. The overall pit wall is designed at 45° (Fig. 14.1.6.4).

The 1-UD area encompasses nine distinct coal seams (Table 14.1.6.1), numbered in sequence from bottom to top. These thicknesses are taken at mid-pit on an east-west cross section.

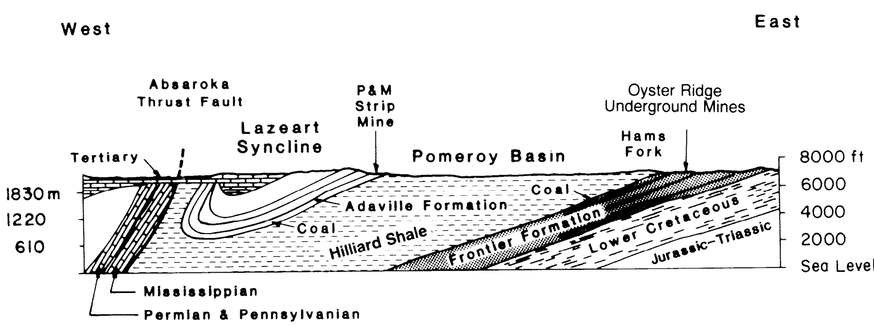

Fig. 14.1.6.3. Geologic cross section.

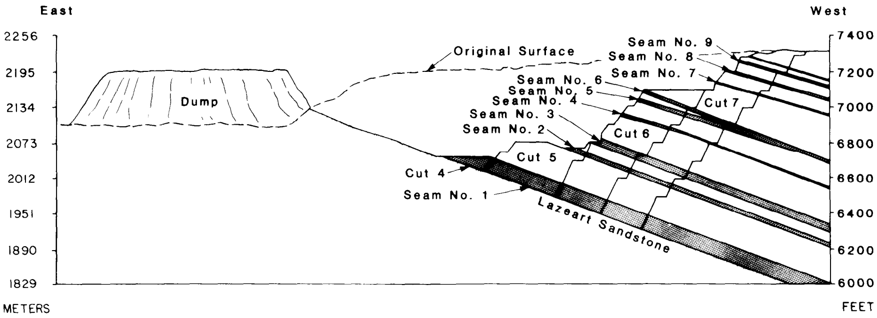

Fig. 14.1.6.4. Typical cross section through the 1-UD pit.

Table 14.1.6.1. Coal Seams in 1-UD Pit

Seam No.	True Thickness	
1	87 ft	26.5 m
2	28 ft	8.5 m
3	55 ft	16.8 m
4	12 ft	3.7 m
5	25 ft	7.6 m
6	15 ft	4.6 m
7	6 ft	1.8 m
8	9 ft	2.7 m
9	12 ft	3.7 m
Total	249 ft	75.9 m

There is wide variation in individual seam thickness along strike. Local faulting, particularly in the south wingwall of 1-UD, makes it difficult to forecast available coal. As a consequence, sequence changes involving other areas have been necessary to maintain the desired overall mining ratio. This condition is expected to persist in future mining areas.

Overburden waste is disposed of entirely out-of-pit in the 1-UD operation (Fig. 14.1.6.1). The dump development plan includes the initial construction of a three-terrace outer ring with 50-ft (15-m) benches, variable vertical intervals, and 17° slopes. These maximum slopes are stipulated in the approved reclamation plan and are necessary for stability and to enhance revegetation.

Fig. 14.1.6.5 illustrates a typical mining sequence in the south wingwall or end section. The north wingwall development is similar, except for variations in the number and location of seams. The bench width varies from 350 to 400 ft (107 to 122 m); in this case, a section line is drawn to split the bench in halves.

The inner half of the bench serves as the haulage road for coal and overburden exiting the area to the east. There are five seams of coal to be removed, with the No. 4 Seam missing in this illustration.

Because the seams dip to the west, all development must progress from west to east, which alternately removes overburden and coal. The sequence to uncover the seams varies, depending on the relative location of each seam.

Access ramps are limited to 8% Pit planning, therefore, requires sufficient horizontal distance between adjacent seams to drop 33 ft (10 m), the designed bench interval. Seams 6, 2, and 3 can be removed in this manner. Because of space limitations, seams 5 and 1 are accessed horizontally, as shown in Fig. 14.1.6.5.

After all seams have been removed, the ribs that remain are taken, progressing from east to west. During this process, the

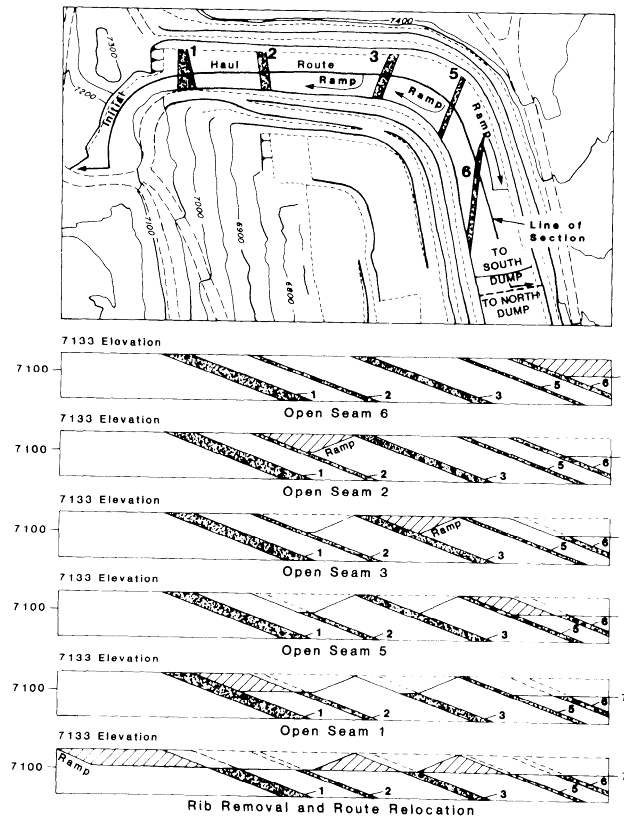

Fig. 14.1.6.5. Bench development for the south wingwall, 1-UD pit.

overburden haulage route is relocated to the outside of the bench and extended to the east pit limit. Finally, the inner portion of the bench is removed in a sequence similar to the process identified on Fig. 14.1.6.5. In this case, however, all development is horizontal, and work progresses from the outside inward.

2-UD AREA. The first pit in the 2-UD area was designed with three vertical cuts containing 40 million yd^3 (30.6 million m^3) of overburden and 8 million tons (7.3 Mt) of coal. There will be a series of seven additional pits developed, in succession, to the south—each one fully or partially fitting into the previous pit.

The lowest minable coal in 2-UD is the No. 2 Seam. The seam identified as No. 1 was mined earlier by underground methods and terminates just beyond the limits of the first pit development. The final pit must be backfilled to the extent that positive drainage is achieved. To accomplish this, some material from previous pits will be stockpiled at the perimeter for final haulback.

Table 14.1.6.2. Drilling Summary

Unit	Application	Pattern	Hole Diameter	Production Rate
45R	Overburden	37 × 39 ft (11.3 × 11.9 m)	10 5/8 in. (270 mm)	75 ft/hr (22.9 m/h)
DM50	Overburden	30 × 39 ft (9.1 × 11.9 m)	9 in. (229 mm)	68 ft/hr (20.7 m/h)
DM50	Coal	20 × 25 ft (6.1 × 7.6 m)	9 in. (229 mm)	83 ft/hr (25.3 m/h)

Table 14.1.6.3. Primary Loading Equipment

Unit	Size	Number	Util.	Production
201M Electric Shovel	25 yd^3 (19 m^3)	2	97%	1001 yd^3 OB[a] (765 m^3)
2100 P&H Electric Shovel	15 yd^3 (11 m^3)	1	93%	774 yd^3 OB (592 m^3)
UH-801 Backhoe	12 yd^3 (9 m^3)	1	79%	301 yd^3 OB (230 m^3)
600E Front-end Loader	12 yd^3 (9 m^3)	1	60%	OB and Coal
600E Front-end Loader	22 yd^3 (17 m^3)	2	87%	OB and Coal

[a]Overburden

The 2-UD production benches are at 50-ft (15-m) intervals, and safety benches are 100 ft (30 m) wide. An oversized bench is designed into the highwall for eventual haulback operations. The overall highwall angle is 45°.

I AREA. Development of I area started on the south end, with the eventual removal of 36 million yd^3 (27.5 million m^3) of out-of-pit spoil. A succession of pits follows with the north limit being just short of the shop/office complex. The final pit will not be backfilled, but the floor will be topsoiled and seeded.

Production benches will be 50 ft (15 m) high with safety benches 50 ft (15 m) wide at 100-ft (30-m) intervals. The highwall designed slope angle is uniformly 45°.

14.1.6.4 Unit Operations

DRILLING AND BLASTING. Four units are used for blasthole drilling. These include two Bucyrus Erie 45R electric, and two Ingersoll Rand DM-50 diesel-powered drills (Table 14.1.6.2). The highly flexible diesel drills are necessary in confined areas and on the steep slopes resulting from the 20° dip of the formation.

In dry overburden holes, heavy ANFO is used as the bottom load and standard ANFO as top load. Emulsions are used in wet holes. The density is adjusted depending on energy requirements.

LOADING. Two 25-yd^3 (19.1-m^3) and one 15-yd^3 (11.5-m^3) electric shovels comprise the principal overburden loading equipment (Table 14.1.6.3). Auxiliary equipment includes a 12-yd^3 (9.2-m^3) backhoe for use in confined areas adjacent to the highwalls, and a 12-yd^3 (9.2-m^3) front-end loader (FEL) employing an ejection-type bucket. Coal is loaded using two 22-yd^3 (16.8-m^3) FELs, also with ejection buckets.

HAULAGE. Fifteen 170-ton (154-t) Euclid rear-dump trucks are used for overburden haulage. These units also are used occasionally for coal haulage when demand increases or there is a temporary shortage of coal trucks. Coal is hauled in six 120-ton (109-t) Wabco rear dump trucks and one 150-ton (136-t) rear dump truck.

Coal is transported the entire distance to the tipples by truck from the three active pits. Typical coal haul distances are shown in Table 14.1.6.4. The pit/tipple combination is determined by the quality and availability of coal.

Table 14.1.6.4. Coal Haulage Distances—One Way

Origin	Destination	Distance
1-UD Pit	Sorensen (Power Plant) Tipple	2.8 miles (4.5 km)
1-UD Pit	Elkol (Rail/Industrial) Tipple	5.2 miles (8.4 km)
2-UD/I-Area	Sorensen Tipple	3.5 miles (5.6 km)
2-UD/I-Area	Elkol Tipple	1.2 miles (1.9 km)

14.1.6.5 Quality Control and Production Coordination

All area design limits are based on an exploration program consisting mainly of structure holes on wide centers and a sufficient number of core holes for long-range quality scheduling. An annual summer development drilling program is conducted to assure greater quality detail and to identify unusual localized structural conditions that might impact coal flow.

Fortunately, there is a high degree of quality uniformity within each of the nine minable seams. As the quality of each seam is established through grab samples or actual experience, future control is assured with only slight variations. With critical limits on ash, care must be exercised to thoroughly clean the top of coal (often a difficult operation on 20° slopes). During coal cleaning (using dozers), it is estimated that 0.5 ft (0.15 m) is lost on both the top and bottom of the seam. This 1-ft (0.3-m) loss also is one of the parameters used in the computer reserve model.

An ASTM sampler is located at both tipples. There also is an on-line ash analyzer on the conveyor system to the power plant. This device permits controlling "as received ash" to ± 0.5% on a shift-by-shift basis.

Broad, long-range mine plans are reviewed on an annual basis by top management and in greater detail at closer intervals as required.

An engineer is assigned to each mine area. This individual, in consultation with production personnel, develops a three-week plan based on the performance of all primary excavating units. This entire plan is provided on a detailed map.

14.1.6.6 Ancillary Facilities

The shop is located near the north end of the I area. It contains large equipment and light-fleet maintenance shops, as well as the warehouse. The large equipment shop includes 10 bays with an area of 16,000 ft^2 (1500 m^2). The steam-cleaning bay, electrical shop, and unheated storage are separate and detached.

14.1.7 AGGREGATE: FREDERICK QUARRY

BERNARD L. GROVE

14.1.7.1 Mine Description

The Frederick Maryland quarry of Genstar Stone Products is located on the eastern edge of Frederick, MD. Frederick city itself is located 40 miles (64 km) west of Baltimore and 40 miles (64 km) northwest of Washington, DC. The area is traversed by two interstate highways, I-70 to Baltimore and I-270 to Washington. Additionally, the CSX railroad serves Frederick, thereby providing an alternate means of transporting crushed stone. The proximity of two major cities and their attendant growth peripheries provide a strong and growing market for the limestone products produced at the Frederick location.

The mine site covers some 800 acres (3.2 km^2), an assemblage of smaller tracts aggregated over the course of 100 years. The current quarry (December 1988) occupies about 120 acres (0.5 km^2), while the disturbed area covers about 200 acres (0.8 km^2). The mine is approximately 2500 ft (760 m) long by 1500 ft (460 m) wide and 220 ft (67 m) deep at the lowest level.

A predecessor of Genstar Stone Products, the M.J. Grove Lime Co., began quarrying limestone at the site in the late 1880s. At the time, most of the stone production was converted to lime through shaft and pot kilns. The lime was used for agricultural purposes and in the masonry trades. Because the chemical content of the limestone deposit was not homogeneous, the trend over the life of the quarry has been away from lime and into crushed commercial aggregate. Lime burning was abandoned completely in 1956.

In 1988, the quarry produced and sold 4.6 million tons (4.2 Mt) of commercial aggregate over a wide range of products. Available products range from armor stone in sizes up to 10,000 lb (4500 kg) down to mineral filler with top size of 120 mesh (0.12 mm). Of the majority of products sold, approximately 95% lies between –2 in. (50.8 mm) and +200 mesh (0.07 mm) in size. Waste product generated at Frederick is minimal. The demand for fill dirt in the growing metropolitan area provides a market for most of the earth waste. In recent years, the overburden rock and soil has been processed, producing a substandard base material. This product is highly salable. Any waste generated is used to build boundary berms (Fig. 14.1.7.1). These berms create a buffer between the quarrying operation and the growing commercial area around the quarry.

Fig. 14.1.7.1. View of boundary berm construction.

Fig. 14.1.7.2. Preparation of top bench.

The quarry and crushing plant are operated 10 months/year. Generally the months of January and February are utilized for maintenance. The quarry and primary crushing plant are operated on two-shift, 18-hour days. The secondary crushing plants are run three shifts a day, five days a week, with some Saturday crushing for special high demand sizes.

14.1.7.2 Deposit Description and Geology

The Frederick quarry sits astride two massive limestone formations. These beds lie on a north-south-trending fold. The eastern deposit in the quarry is the Frederick Limestone. This is a blue, slabby, thin-bedded limestone with minor shale of Cambrian Age. The deposit is approximately 500 ft (150 m) thick. The western deposit is the Grove Limestone. This is a light-to-dark gray, thick-bedded limestone containing dolomite beds in the lower measures. The deposit is of Cambrian Age and is approximately 600 ft (180 m) thick.

Both the Grove and Frederick Limestones have good-to-excellent properties for commercial aggregate. Los Angeles abrasion test results are in the 25 to 30 range. There are occasional problems with the shape of the Frederick Limestone particles, but this problem is overcome through controlled reduction at the tertiary crushers and blending of the Frederick and Grove materials.

The ratio of waste to stone is very low at the present time, measuring about 14:1 stone to waste. The ratio will increase as the quarry is deepened, and local demand for waste, fill dirt, and substandard base should also increase. The proven permitted reserves at the Frederick quarry at the end of 1988 were 500 million tons (454 Mt), but development drilling indicates an additional 250 million tons (227 Mt) lower in the stratigraphic sections.

14.1.7.3 Mine Development

Due to the increasing demand for materials previously considered as waste products, site preparation at Frederick has changed substantially in the last 40 years. The quarry is presently on a five-year mining plan that is updated annually. The plan considers expansion of the quarry westerly, southwesterly, and vertically over the five-year period. One major objective of the plan is to optimize the blending of the two limestones and minimize the haul to the primary crusher.

Initial removal of soil and subsoil is performed by outside contractors seeking select fill. The properties of the overburden, a combination of soil and weathered limestone sand, make an ideal fill material. Once the overburden has been removed to a predetermined elevation (Fig. 14.1.7.2), a decision is made as to whether or not the uncovered area is to be crushed for its value as substandard base or treated as waste. This area also represents the top bench or level in the quarry. As the top level is pushed back, a series of faces, ranging from 40 to 100 ft (12 to 30 m) in

Fig. 14.1.7.3. Overview of mine benches.

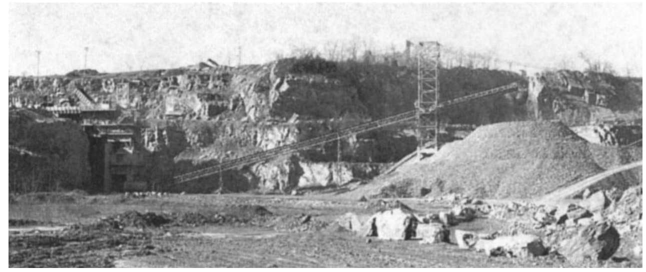

Fig. 14.1.7.4. In-pit crusher.

height, are developed (Fig. 14.1.7.3). Bench heights are based on rock quality, stability, and quarry development stage. At the present time, there are five benches on the deepest side of the operation.

14.1.7.4 Unit Operations

DRILLING. Contract drillers provide all holes for blasting using air track and well drills. Air tracks are utilized in the top or substandard levels because of their greater maneuverability and stability. Small holes also are more practical in the seamy upper levels where spacings must vary widely due to sand pockets. A typical drill pattern in the upper level would be 9 by 9 ft (2.7 by 2.7 m) drilled to a 40-ft (12-m) depth using bits 2.5-in. (64-mm) in diameter. Once the substandard or top level is removed, all drilling, other than toe holes or secondary blasting is accomplished with rotary drills. The contractor performing this work has 10 drills and can provide drilling equipment as required. The primary drills used for production are IRD-4s. The holes are 6 3/4 in. (171 mm) in diameter, drilled on an average pattern of 14 by 18 ft (4.3 by 5.5 m). The pattern, however, varies based on bench heights, site conditions, and production requirements. The hole depth varies from 40 to 100 ft (12 to 30 m). Primary production drilling averages 25 holes per 10-hour day, but the schedule varies widely based on demand. All drilling and blasting are performed under the direction of a Genstar quarry supervisor. Seismographs are utilized with every shot, and all records are analyzed by an independent consultant. Records also are maintained at the quarry site and are available to various state agencies and any concerned citizens.

BLASTING. The evolution of explosives technology industry-wide is mirrored at the Federick quarry. Dynamite and all stick material, along with fuse and electric caps, have been replaced with ANFO, slurries, and nonelectric detonating systems. Water is not generally a problem, but where water occurs, holes are dewatered and, whenever possible, bulk loaded with a combination of slurry in the bottom and ANFO higher in the column. If water is persistent and the ground highly permeable, sleeves may be used to protect the explosive. All shots are double primed. A booster is placed at the top and bottom as a precautionary safety measure. Each hole is stemmed with 12 to 14 ft (3.7 to 4.3 m) of granular material. Again, as with drilling, all explosives work is performed by contractors under the supervision of Genstar's quarry supervisor. Very little explosive is stored on site, as most of the bulk explosives are brought in daily from a bulk storage facility at another Genstar location.

Powder factors vary widely, depending on operating experience in the particular area of the quarry. A good average is 2.5 to 1 ton/lb (5.0 to 2.0 t/kg).

LOADING. Once a shot is fired, loading is performed by two 11-yd^3 (8.4-m^3) hydraulic shovels. The loading operation at the Frederick location mirrors the evolution of loading within the quarry industry. Until the late 1960s, all primary quarry loading was performed by 2.5- and 3.5-yd^3 (1.9- to 2.7-m^3), cable-hoist shovels. Beginning in the late 1960s and continuing through the early 1980s the primary loading machine was the rubber-tired loader. Loaders, while very versatile, are high-cost machines for large, well-developed quarries such as Frederick. In 1983, the first hydraulic shovel was acquired. This machine, with some 20,000 hours of operation as of December 1988, was the first large hydraulic shovel placed in a quarry operation in the eastern United States and has proved to be a highly reliable and efficient loading machine.

In 1988, to meet increased production demands, a second 11-yd^3 (8.4-m^3) shovel was added. The machines provide a combined loading capacity of 1500 tph (1360 t/h) and to date have an availability in excess of 90%. An 8-yd^3 (6.1-m^3) rubber-tired loader is available for backup, cleanup, and loading rip rap and other stone sizes.

HAULAGE. Hauling from quarry face to primary crusher is performed by off-highway trucks. As with the quarry loading equipment, the size of the haul units has increased over the years. Prior to 1973, haulage units were 30 to 35 tons (27 to 32 t) in capacity. In 1973, 50-ton (45-t) trucks were placed in service. In 1987, two 85-ton (77-t) trucks were purchased, and an additional truck was added in 1989. The present quarry fleet consists of three 85-ton (77-t) and three 50-ton (45-t) trucks with a 50-ton (45-t) truck in reserve. As a general mode of operation, one shovel loads two 85-ton (77-t) trucks, while the second shovel loads 50-ton (45-t) trucks. The average haul to the primary crusher is approximately 2000 ft (600 m) round trip.

CRUSHING. Primary crushing is accomplished in-pit, (Fig. 14.1.7.4). Rock is hauled from the face to the primary crushing station and is dumped into a 250-ton (225-t) rock box above a 7- by 24-ft (2.1- by 7.3-m) reciprocating plate feeder. The use of this feeder has proven invaluable, both in optimizing truck utilization and in reducing lost time at the primary by providing a controlled feed to the crusher.

The primary crusher is a 42- by 65-in. (1.1- by 1.65-m) gyratory. Crushed rock with nominal 15-in. (0.38 l-mm) top size is fed by 54-in. (1.4-m) belt to an 80,000-ton (72,575-t) surge pile located over a vault. The vault contains a 5- by 10-ft (1.53- by 3.05-m) feeder. At the feed point, the crushing system becomes automated and is under the control of a microprocessor. Feed from the surge pile is carried to the ground elevation. This requires a lift of approximately 100 ft (30 m) via an 850-ft (259-m), 48-in. (1.2-m) conveyor. The feed is split through a 200-ton (18l-t) splitter bin. The splitter bin feeds a new secondary crushing plant with a nominal capacity of 750 tph (680 t/h) (Fig. 14.1.7.5) and an old secondary crushing plant with a nominal

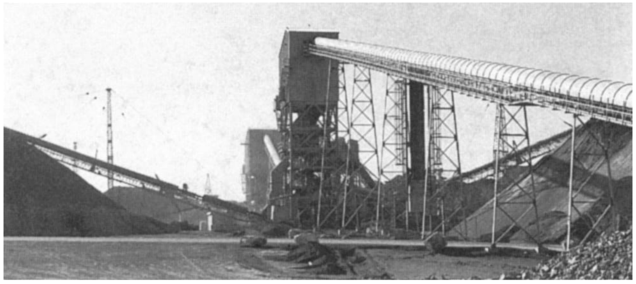

Fig. 14.1.7.5. Secondary crushing plant.

capacity of 500 tph (454 t/h). Through a variety of tertiary crushing stations, it is possible to produce as many as 20 different products simultaneously upon demand. This unique ability to produce a wide variety of sizes has played a key role in the growth of the Frederick operation.

PRODUCT TRANSPORTATION. Of more than passing interest is the fact that due to the quarry's strategic location, more than one-third of the commercial aggregate produced is converted or incorporated into other construction materials. Five bituminous concrete plants are located at or adjacent to the quarry. Included in these is the largest bituminous concrete plant in the state of Maryland. There are four ready-mixed concrete plants at or adjacent to the quarry. Finally, a large modern concrete block plant is in close proximity. The location of these end users provides significant economic advantages. Raw materials can be provided to the user plants by large mobile equipment not subject to highway weight restrictions. A second benefit is the elimination of double hauling; that is, raw material is delivered directly to the finishing plant and finished product is hauled out. With finishing plants at or adjacent to the quarry site, finished product goes directly to the point of use. This fact has a significant effect on reducing volumes of traffic over increasingly congested urban highways.

14.1.7.5 Product Control

Commercial aggregate sold from the Frederick quarry is under strict product control, both as to quality and quantity. Federal and state specifications are becoming more rigid each year. Product samples are taken from operating faces on a daily basis to assure conformity with specifications. By utilizing frequent sampling techniques, trouble spots can be isolated and corrected. Additionally, as there are size overlaps within products, frequent gradation checks assure the operations management that optimum production of critical sizes can be assured.

14.1.7.6 Ancillary Facilities

The sale and billing of all aggregate are fully automated. All vehicles, whether customer or contract hauler, transporting stone from the Frederick quarry must stop on the inspatch side of the Material Distribution Center (MDC). At the MDC, all information relative to the sale involving the individual vehicle is recorded. The truck is then directed to the proper loading point where material is loaded. The vehicle then returns to the MDC on the dispatch side. As the truck crosses the scale at the MDC, a ticket is created. Once the ticket is created via scale, the information from the ticket is handled electronically and is not seen again until the invoice is received and opened by the customer.

All phases of product and material handling and dispatch from mine planning to weighing of the finished product are under the direct control of the plant superintendent. This concentration of control minimizes jurisdictional disputes and makes problems easy to find and correct.

One area of paramount concern with both residential and nearby commercial neighbors is maintaining acceptable environmental conditions. Frederick's first step is to buffer the operation on all sides. Buffering is accomplished with berms created from waste. The berms are planted with grass and legumes and eventually with fast growing trees. As a result of the buffering, few people are aware of the large-scale operations at the site. In addition to buffers, dust and noise control are essential. The Frederick quarry has embarked on a long-range paving program that will, over a five-year period, provide a paved surface for all plant haul roads. A sweeper will be provided to remove dust from the paved surface. The new secondary crushing plant is a wet-process plant, thereby eliminating process dust. Plans are currently underway to incorporate wet process into the entire secondary crushing facility to reduce dust emissions. Noise emissions are monitored by regular boundary-level readings by dosimeter. Any violations can be pinpointed and corrected. Seismograph readings are taken with every shot, and historical data provide limiting loads for blastholes, the number and design of delays, and overall size for shots.

Maintenance is another major concern. All major equipment, mobile and stationary, is placed in a preventive maintenance program. The mobile equipment shop is capable of accommodating any piece of equipment presently used, including the hydraulic shovel and 85-ton (77-t) trucks. This facility ensures an environment for safe and clean maintenance of equipment free from weather hazards.

Well-trained and educated employees, sound environmental practices, good preventive maintenance, and adherence to maintaining these programs are the keys to survival in a modern aggregate operation. Being a good neighbor with responsible attitudes also is essential for acceptance in the community.

Chapter 14.2
QUARRYING

M.K. McCarter and L.P. Meade

14.2.0 QUARRY MINING

M.K. McCarter

Quarry mining is a term often applied to either mining of dimension stone or aggregate (Morrison and Russell, 1973). It is preferable, however, to restrict this term to production of dimension stone only (Hartman, 1987).

The products of quarries are prismatic blocks of rock such as marble, granite, limestone, sandstone, slate, etc., that are useful for primary construction of buildings and monuments or decorative facing materials for the exterior or interior of buildings. Quarries normally have benches with vertical faces ranging from a few feet (meters) to 200 ft (60 m) in height. The overall depth of a quarry may be as much as 1000 ft (300 m). Material is removed in a highly selective manner using time-consuming and expensive methods for freeing the blocks from the surrounding formation. Production from individual quarries is relatively small compared to other surface mining methods but has a high average product value of $114/ton ($126/t) (Hartman, 1987).

From the mid-1800s through the mid-1900s, the dimension stone industry was an essential resource for construction of North American cities, railroads, and major public buildings. It reached a zenith in the 1960s and then entered a slump which continued through 1985. The current high demand for natural stone is bringing about a resurgence in the industry. This demand is forcing higher product values along with sharply increasing US imports, principally from Italy. Production in the United States, however, is lagging the demand because of environmental restrictions along with the required investment and lead time necessary to develop deposits or restore existing quarries. In addition, the demand for cubic stone, a more abundant resource, is declining in favor of large panels of exterior and interior cladding which is more difficult to produce. As of the late 1980s the number of active quarries in North America was approximately 300 with only about six integrated quarries and mills of major importance to the local economy (Meade, 1986a).

The characteristics that make a deposit commercially viable for dimension stone include color, appearance, competence, uniformity, and freedom from defects. Location of suitable zones within a formation is accomplished normally through diamond core drilling. Planning the excavation is based primarily on geologic factors such as direction and attitude of bedding and on suitable disposal sites for overburden and unsuitable material from the quarry.

There are four major operations in quarrying: overburden removal and cutting, splitting, and handling stone. Overburden is loosened by ripping or blasting and then removed by dragline, scraper, or front-end loaders. Occasionally, the deposit is cleaned by hydraulic monitors (Chapter 15.1.6). Once the surface is cleared, an opening cut or slot is produced across the quarry width. This slot provides an additional free face which facilitates removal of subsequent blocks until the first lift, extending across the entire width and length of the quarry, has been removed or until sufficient room has been produced to remove another key block initiating the second lift (Morrison and Russell, 1973; Power, 1975).

Each stone quarry is unique and requires a customized cycle of unit operations. The upper surface and key slot provide two free faces (the top and front of a prismatic block). Three additional faces are produced, which define the ends and the back. These faces are normally produced by line drilling with pneumatic hammers mounted on a "quarry bar." Soft formations may be cut with wire saws, carbide tipped chain saws, or circular saw blades impregnated with diamond or other abrasive materials. Hard formations with minerals exhibiting contrasting thermal expansion characteristics can be cut with a flame jet known as the Browning torch. This method is particularly suitable for granite. Other formations are amenable to cutting by high-pressure water jet. Channel cutting for the torch ranges from 11 to 28 ft/hr (3.4 to 8.5 m/h). Slot drilling in granite ranges from 8 to 14 ft/hr (2.4 to 4.3 m/h) (Meade, 1986a).

Once the block is defined on five sides, the bottom is freed by wedging. A common practice in years past was to drill short holes at the base of the block and to insert "feathers and plugs." Hand hammering of these closely spaced wedges propagated a horizontal fracture (usually along a bedding plane) that detached the block at the base. Alternatively, the wedges can be used to produce a fracture along the back of the prism if the bottom is defined by a bedding plane or joint. Recent advancements include replacing the feathers and plugs with hydraulic wedges, air bags, or partially stemmed explosives for line splitting. Blocks are often tipped 90° and split along the bedding to form slabs close to the desired size before removal from the quarry.

Derricks are still widely used for hoisting of blocks. However, mobile equipment is being used more frequently because of greater flexibility and versatility. The use of mobile equipment has facilitated a substantial reduction in the number of employee hours expended per quarry block removed resulting in an increase of 30 to 50% in productivity over the past decade. Productivity ranges from about 5 to 18 ft^3 (0.1 to 0.5 m^3)/employee-hour in thinly bedded, easily cut deposits. Productivity in marble ranges from about 5 to 14 ft^3 (0.1 to 0.4m^3)/employee-hour (Meade, 1986a).

Large operations maintain mill and finishing shop facilities adjacent to the mine. In the United States and Italy, the trend is to upgrade the finishing facilities by installing automated equipment. Until the unit cost of domestically produced materials can compete with imports, European and South American products are likely to increase. North American producers are able to compete to some extent by selling a superior product. The American Society of Testing Materials has established world standards for dimension-stone test procedures, and close coordination needed between producers and customers to maintain quality control provides a slight edge for domestic producers. In addition, some producers have succeeded in developing markets for in-pit waste for use as construction aggregates, tennis court topping, whiting filler, and agricultural lime, thus reducing the overall mining cost. In some cases, however, it is now economically advantageous to produce the stone in the United States, ship it overseas for finishing, and then return the finished product to domestic customers (Meade, 1986b).

The following case study for the Friendsville quarry describes a typical operation involving both quarry and finishing plant in the eastern United States.

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14.2.1 DIMENSION STONE: FRIENDSVILLE QUARRY

LANCE P. MEADE

14.2.1.1 Mine Description

Cost savings resulting from new quarrying methods are partly responsible for reactivating previously closed quarries in the Tennessee Marble District near Knoxville, TN. The Tennessee marble quarries are located in a limestone known as the Holston Formation. This material is a buff-to-pink fossiliferous limestone that, because it takes a good polish and can be cut into slabs, is classified as marble. Numerous buildings in Washington, DC (notably the National Gallery and Air and Space Museum), as well as older post offices and railroad stations in other parts of the country, have used this marble. The district was relatively active through the 1950s and into the 1960s. By the end of the 1970s, only one small quarry, saw plant, and finishing shop remained active.

In the early 1980s a small marble finishing shop (MSI) in Knoxville purchased a quarry property located off Vinegar Valley Road in the town of Friendsville, Blount County, TN. This

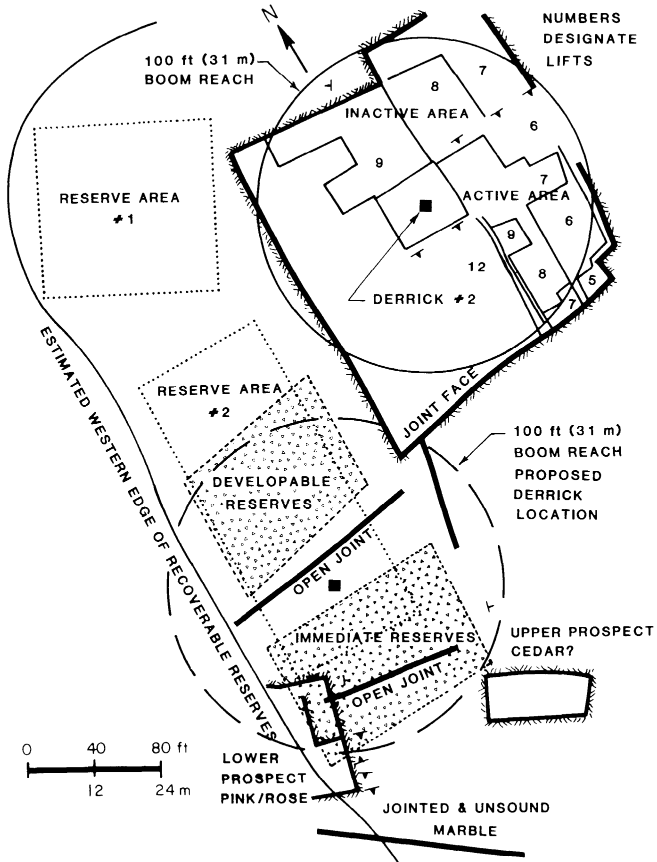


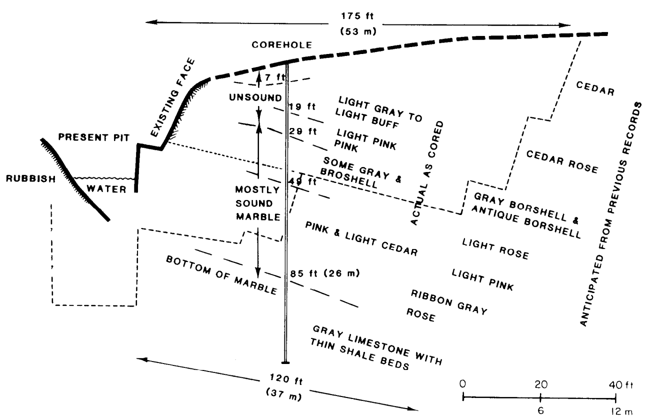
Fig. 14.2.1.1. Plan view of quarry operations.

property was previously operated by the Vermont Marble Co. and the Grey Knox Marble Co. under the name of the Brown quarry. The marble was known for its pink color and uniform texture. It was a desirable marble, but the depth of the quarry, limited development work, and associated high production costs resulted in its abandonment in the 1950s.

The reopening of this quarry in 1985 and subsequent purchase of MSI by the Tennessee Marble Division of the Luck Stone Corp. reactivated the property. Since 1987, new equipment, overburden stripping, waste rock removal, and demand for the stone have contributed to the on going development of the quarry.

Approximately 39 acres (0.16 km^2) of the 52-acre (0.21-km^2) property cover acceptable marble beds and have the potential for development into quarry space (Fig. 14.2.1.1). Along with the existing 40,000-ft^2 (3700-m^2) Brown quarry opening, three smaller openings are being developed into a continuous working face of multiple benches accessible by rubber-tired loaders. The balance of the property is used for block and slab inventory, finished material, storage, and a diamond gang-saw plant with an adjacent finishing shop that cuts and polishes thin veneer marble panels, tile, and specialty items.

Current operations are emphasizing removal of overburden and waste rock in the new development area. When this phase is complete, approximately 150,000 yd^3 (115,000 m^3) of material will be removed. Since reopening of the quarry, annual production totals approximately 60,000 ft^3 (1700 m^3) of marble.



Conversion factor: 1 ft = 0.3048 m.

Fig. 14.2.1.2. Geologic cross section through quarry.

14.2.1.2 Deposit Description and Geology

The marble deposit occurs in easterly dipping (10 to 15°) layers of a homoclinal structure (Fig. 14.2.1.2). The Holston limestone is composed of partially recrystallized calcite that ranges from light buff or dark pink to dark mahogany in color. A minor amount of the deposit is a light-gray, Fleuri-type limestone of limited usage. Underlying the marble is a gray, shaley limestone.

High-angle, east-west-trending joints allow for easy extraction of blocks. In some portions of the property, diagonally intersecting joints minimize the recovery of usable blocks. Portions of the joint system exhibit deep weathering with subsequent solution cavities and mud seams. These areas are either avoided or removed and wasted in routine quarry development. Similarly, small areas of structurally and stratigraphically controlled cutters and pyrite zones also are avoided or removed as quarry waste. A marble recovery factor of 50 to 60% of quarry space has been the historical experience. Modular-sized panels and tile production should allow greater stock utilization and an improved recovery factor in the future.

The strike of the marble layers extends across the property in a north-south direction. Overburden is variable but averages 7 ft (2 m) in thickness. A 20-ft (6-m) underlying zone of unsound cap rock is wasted due to unacceptable mechanical properties and poor color. The commercial layer is 50 to 70 ft (15 to 21 m) in thickness, and within this horizon, six to eight varieties of marble normally occur that are classified by color. The present operating additional quarry (Quarry No. 4, Derrick No. 2) is approximately 80 ft (24 m) deep with working benches 30 to 40 ft (9 to 12 m) above the quarry floor.

A core drilling campaign verified by test block removal, has confirmed the existence and continuity of the marble layers, and subsequent proven reserves have been established. The reserves of a dimension stone deposit are usually not limited by the available stone but rather by the whims of architectural preference and profitable extraction. A 20-year supply (considered adequate by industry standards) is projected based on these reserves and current and projected demand.

14.2.1.3 Mine Development

The first three years of operation (1985–1988) consisted of predevelopment activity. During this time, the existing Brown quarry was dewatered and the derrick, hoist, and compressor recommissioned. Access to the working area was by a series of stairs and ladders for the men and a boom-fall derrick for the equipment and supplies. After cleaning up debris with a small front-end loader and removing it with the derrick, old working benches were evaluated for their potential yield of stone. Typical Tennessee-style, skid-mounted quarry bars with chain-feed air hammers were utilized for vertical drilling and broaching. Small pneumatic layer drills on horizontal quarry bars were used for breakline cuts.

Slabs from sawn blocks were graded, classified, inventoried, and marketed on small projects. Based on this experience, a contract was obtained for supplying marble to the Library Square project in Los Angeles, CA. This enabled scaling-up operations by purchasing additional quarry equipment, a second Bennetti wire saw, and lightweight drills that do not require a derrick for in-quarry moves.

Various alternatives for opening the existing quarry for direct access by mobile equipment were evaluated. The final decision was to develop the area adjacent and south of the Brown quarry for this innovative approach.

Cap rock is being removed by a combination of wire sawing, shooting, and wedging. The loosened rock is removed with the use of a dozer, excavator, and 988B loader. The waste is dumped in a grout pile or used for ramps and roadways. The careful preparation and maintenance of all road and haulageways has kept vehicle maintenance costs to a minimum.

With the old (pre-1970s) Tennessee quarry methods of working under a guyed boom-fall derrick and using large quarry bars and pneumatic hammers, 1.5 to 2.0 ft^3 (0.04 to 0.06 m^3) of stone could be removed per employee-hour of operation (with a 50% recovery factor). Currently, utilizing rubber-tired loaders, small wire saws, and lightweight pneumatic drills, this figure has been increased to 5 ft^3 (0.14 m^3)/employee-hour with a crew of eight. With present sales forecasts and production demands, the quarry crew size is expected to increase to 10 or 12.

14.2.1.4 Unit Operations

Currently, a crew of eight utilizes three Bennetti line drills for channel and line cutting, one penetrating drill for wire-saw pilot-hole development, and three Bennetti wire saws. Coupled with a layer hammer for horizontal drilling, a 988B loader, and a hydraulic jack system for raising and tipping blocks, the crew of eight produces what a crew of 18 would produce with the older system.

The confined space in the Friendsville quarry has necessitated retention of the horizontal floors and benches. Lifts of 4 to 6 ft (1.2 to 1.8 m) are made depending upon the soundness, color variations, or required block sizes. The initial key-way cut, made with the wire saw, is wide enough (8 ft or 2.4 m) to allow access for the layer hammer, if horizontal break lines are needed. Once the key blocks are removed, additional strips are cut. Depending on the height of the strip, they are tipped into the keyway for splitting. Splitting can also be performed in the vertical position (Figs. 14.2.1.3 and 14.2.1.4).

The newly developed quarry area will have benches of greater height (15 to 25 ft, or 4.6 to 7.6 m), which will allow tipping the strips of cut marble for splitting into required block sizes. Full use of the wire saws and mobile equipment allows for more efficient continuous cutting. The older Tennessee system resulted in more time moving and setting up equipment rather than actual cutting of stone. As the new area develops, a diamond chain saw is being considered for vertical cuts. These units can cut to depths of 13 ft (4 m) but are more efficient if the cuts are limited to 6 to 8 ft (1.8 to 2.4 m). The greater efficiency of the

Fig. 14.2.1.3. Line drilling in the Friendsville quarry.

chainsaw units will more than offset the need to revert back to lower working benches.

14.2.1.5 Product Control

One criterion that distinguishes dimension stone from ordinary rock is the consistency of color, texture, and soundness. In earlier times, very stringent requirements were placed on the particular variety names for various marbles. For instance, Tennessee cedar would be recognizable and consistent with all other Tennessee cedar. Job specifications and architectural requirements usually demand close product control and continuous coordination between the quarry, mill, finishing shop, and sales. The range of allowable variations usually is greater for nonpolished exterior applications than for highly polished and matched interior installations. The tightness or looseness of the job specifications has a very dramatic impact on the amount of quarry, mill, and shop waste that will result from each cubic foot (cubic meter) of quarry space excavated.

During the 1920s when the demand for dimension stone was at its peak, marble graders and stock specialists played a very significant role in the rating and classification of stone. This system generally was abandoned to some extent as interior usage of stone diminished in the 1970s. The current resurgence in demand for decorative interior marble has resulted in the need for closer stock selection and utilization.

14.2.1.6 Ancillary Facilities

One major difference between operating a dimension stone quarry in the 1990s vs. the 1920s is the type and amount of support services needed. In the 1920s the economy of major quarry localities, such as Knoxville, TN, Bedford, IN, and Rutland, VT, was centered around the quarries, mills, and shops. Local machine shops, foundries, and equipment manufacturing establishments supplied the quarries with the needed equipment, tools, and supplies.

In the 1990s, most of the quarry equipment is manufactured in Italy, Germany, and Finland. Inventory of replacement parts must be kept on hand or air-freighted to the job site in emergency situations. Repair of the newer hydraulic drills necessitates highly trained technicians rather than general maintenance personnel who repaired older pneumatic drills. In other words, factory repair personnel who speak a language other than English are a frequent part of a modern dimension stone operation.

Onsite blacksmith shops for hand-forging drill steel and bits are virtually obsolete. Likewise, the forge at the Brown quarry, used during the early stages of start-up, has now been abandoned. The use of slip-on and threaded bits has virtually eliminated the need for forged tools.

ACKNOWLEDGMENTS

The Luck Stone Corporation and Mr. Claude Ledgerwood have been most helpful by making information on the Friendsville quarry development available for publication.

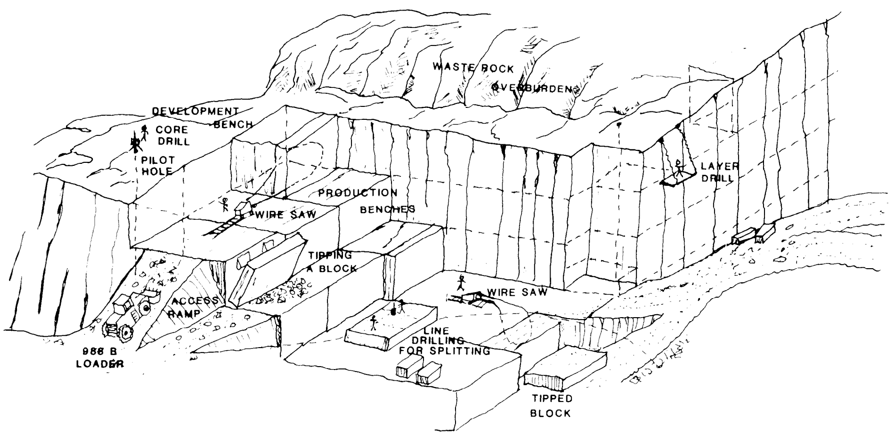


Fig. 14.2.1.4. Schematic representation of Friendsville quarry development.

Chapter 14.3
OPEN CAST (STRIP) MINING

M.K. McCarter, Mark L. Bricker, G. Paul Anderson, Steven J. Kirk, David B. McDonald, Gary M. Stubblefield, Roger W. Fish, Ira J. Tussey, Gene Rand, Wayne N. McKee, and Steven I. Olson

14.3.0 SURFACE MINING—CONCURRENT RECLAMATION

M.K. McCarter

Surface mining in which reclamation is carried out contemporaneously with extraction may be classified as either area mining or contour mining. *Area mining* is often referred to as *open cast* or *strip* (furrow) *mining.* It is frequently practiced on a large scale, which results in low unit cost, high productivity, high recovery, and greater safety compared to underground mining. *Contour* (or collar) *mining* progresses in a narrow zone following the outcrop of a coal seam in mountainous terrain. The specific methods used in contour mining are referred to as haulback, box-cut, and block-cut mining (Anon., 1979). The unique aspect of both area and contour mining is that overburden, removed to gain access to the mineral commodity, is immediately placed in the previously mined area.

Coal of bituminous rank as well as lignite are the principal products of area mining. Phosphate and most oil sands are also mined in this manner. To a lesser extent anthracite, bauxite, and bentonite are also mined by area methods (Hartman, 1987). Contour mining is reserved almost exclusively for removing the thin coal beds of the Appalachian region.

In area mining, removal of overburden is accomplished with large draglines, stripping shovels, or bucket wheel excavators (BWEs). These machines excavate and deposit material in one continuous operation. The same function can be accomplished, in part, through the use of explosives (cast blasting). Often removal of soil and overburden is accomplished using a combination of casting machines to maximize the width of the mining area. Conventional excavation and haulage techniques (truck and shovels, scrapers, or dozers) are also used by themselves, or in conjunction with one or more casting machines. Conventional excavation and haulage may be dedicated to topsoil removal only, or it may be employed in removing both topsoil and overburden.

Area mining usually progresses in a series of parallel, deep trenches referred to as furrows or strips that may exceed several thousand feet (1 km or more) in length (Fig. 14.3.0.1). Material is removed from the undisturbed area on the highwall side forming a deep trench. It is then placed in the open pit at the angle of repose forming a spoil ridge. The width of the open pit is typically 75 to 150 ft (23 to 46 m), the height of the highwall may be up to 200 ft (61 m), and the height of the spoil ridge may be up to 250 ft (76 m). Typical highwall angles range from 50° to vertical. Typical spoilpile angles range from 30 to 45° measured from the horizontal. Angles steeper than 37°, which is the angle of repose for most materials, can be obtained by allowing the spoil to compact under its own weight then excavating the material a second time to an oversteepened condition. Experience has shown that oversteepened spoil, in some locations, is capable of standing long enough to extract the mineral and begin backfilling the pit before major instability occurs.

Haulback mining proceeds in a series of nearly square rectangular pits following the outcrop for as much as several miles (kilometers) (Fig. 14.3.0.2). The pit width is dictated by the economic stripping ratio and steepness of topography. Since mining roughly follows a contour, the path of mining is generally sinuous with both outside and inside curves. Mining progresses by first removing a box cut and hauling the overburden to a suitable disposal site. Thereafter, overburden is broken, excavated, and hauled by truck, scraper, or conveyor to fill previously mined-out pits with little or no waste deposited on the outslope. In this manner waste moves in a direction opposite to the direction of mining and hence the name "haulback."

Box-cut contour mining (Fig. 14.3.0.3) is practiced in moderately sloping terrain. It is essentially the same as area mining where the total number of strips is limited to two or three. The strips are oriented parallel to the outcrop line and progress uphill into the highwall. The first strip is a consecutive series of box cuts, which provides an open pit for spoil cast from the second strip.

A dozer is normally employed to clear vegetation over the box cuts and down-slope from the outcrop line. Overburden is then pushed from the box cuts to the outslope area. Once the coal is uncovered, it is removed, and the dozer establishes a level bench upslope from the box-cut line. From this bench, a dragline then excavates the overburden and deposits it into the adjacent box cut. In place of the dragline, a shovel can be used to remove overburden from the second cut thus eliminating the need to create an upslope bench. Upon reaching the economic limit of mining, dozers push the spoil uphill to cover the highwall before revegetation commences.

Block-cut mining can be applied to either area or contour mining. This method differs from conventional area and contour mining in that blocks of overburden in excess of 70 ft (20 m) wide are removed as a unit and deposited in mined-out areas either perpendicular to the direction of advance (area mining) or along the outcrop direction (contour mining). If the mineral commodity is relatively shallow with more or less uniform cover, the overburden can be removed by front-end loader (FEL) or pushed by dozer to the disposal site in one operation.

If a coal seam is close to flat lying and occurs near a mountain top, the deposit can be mined from outcrop to outcrop in series of strips as practiced in conventional area mining. In some cases, the spoil is leveled rather than contoured to approximate the original topography. This method is referred to as mountaintop removal (Anon., 1979). This modification, however, differs from mountaintop removal as described in Chapter 14.1. In the previous section, mountaintop removal was identified as a "deferred reclamation" technique because overburden was hauled to a disposal site away from the mining area. In the above case, overburden is deposited in the open pit adjacent to the seam being uncovered and reclaimed as mining progresses.

The cycle of operations includes clearing vegetation, soil removal, drilling and blasting overburden, stripping, removal of coal or other mineral commodity, and reclamation. In the case of area and contour mining, reclamation should be considered as a unit operation since it is repetitive and concurrent with mining.

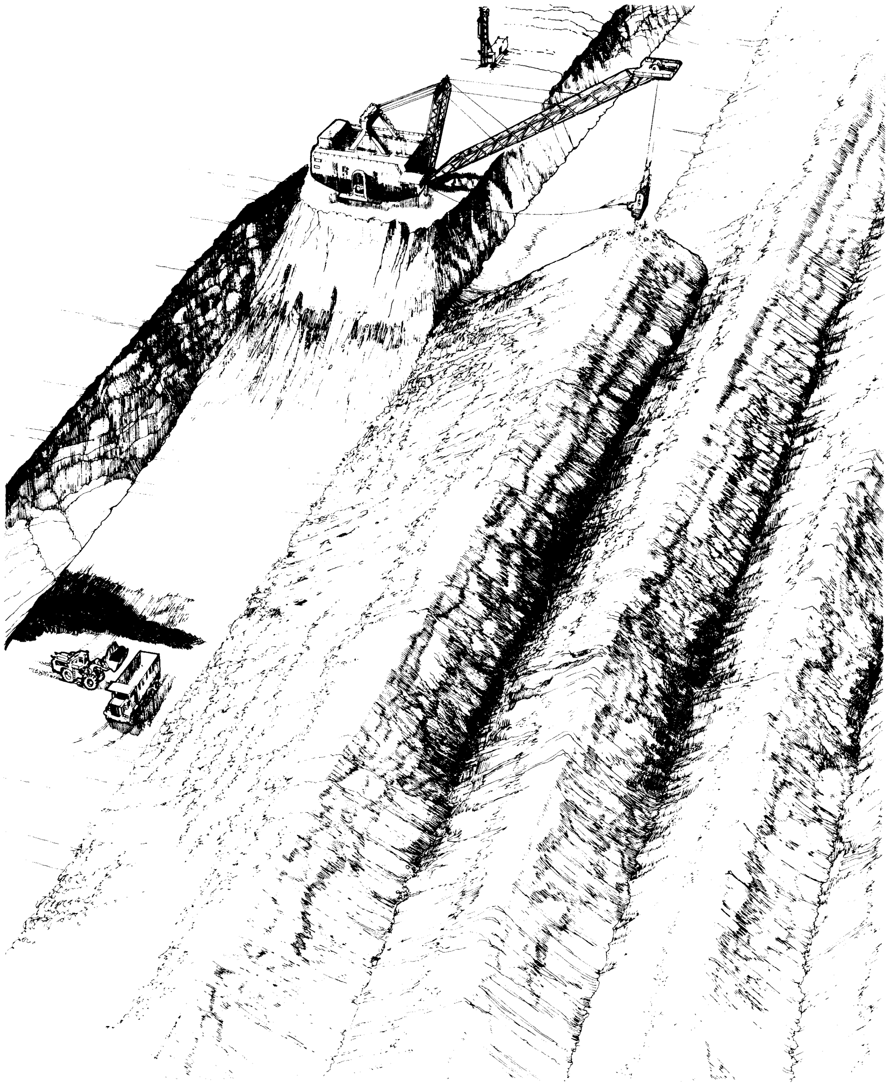


Fig. 14.3.0.1. Conventional area mining utilizing a dragline. (Anon.,1979; illustration by Frank Kulczak. By permission from Skelly and Loy, Harrisburg, PA.)

Clearing vegetation is nearly always accomplished with track-mounted dozers. Soil removal is often assisted by dozers but more commonly achieved with scrapers and dozers, or trucks and FELs. Overburden can be broken with dozers equipped with rippers if the rock is weak or drilled and shot if it is more competent. Drilling may be done with drag bits, rotary bits, or percussion. Holes are usually vertical, but occasionally inclined holes are used to achieve better highwall stability and more uniform fragmentation especially if the highwall is not vertical. The explosive of choice is ANFO unless water precludes its use.

Selection of an appropriate excavator is usually dictated by the properties of the overburden and mineral commodity as well as the deposit geometry. Stripping shovels are able to exert maximum digging effort and are the best choice for blocky material. They also have a shorter cycle time compared to draglines. The main disadvantage is that they must work on top of the seam to be mined. If this material is friable, fines are created by movement of the excavator, and the fines may contribute to poor recovery or poor product specifications. In addition, shovels must reach from the top of the seam to the top of the spoilpile rather than from the original ground surface to the top of the spoil pile as is the case for draglines.

Effective use of draglines requires well-broken and disaggregated material. The main advantage of the dragline over stripping shovels is flexibility. It usually has greater reach for a given set of circumstances and can operate on either the highwall side or on the spoil side. In addition, it can cope better with adverse pit conditions such as slides and flooding that would be difficult for a shovel. It can also dig a deeper box cut and can dig both above and below its position on the overburden. The cost per cubic yard (cubic meter) is slightly higher for bucket (dipper) sizes less than 70 yd^3 (54 m^3) compared to shovels but about the same for the larger sizes (Weimer, 1968).

Bucket wheel excavators perform best in material which is well fragmented and void of large boulders and buried vegetation. They are particularly well adapted to hard pottery clays, phosphates, tar sands, and bauxite. Often they are able to handle these materials without blasting. The advantage of BWEs is a more even power demand compared to either shovels or draglines. They also tend to provide more stable pit slopes, more uniform spoil piles, and greater control in selective mining of interbedded deposits. BWEs are able to excavate both above and below the elevation of the base of the machine. The major

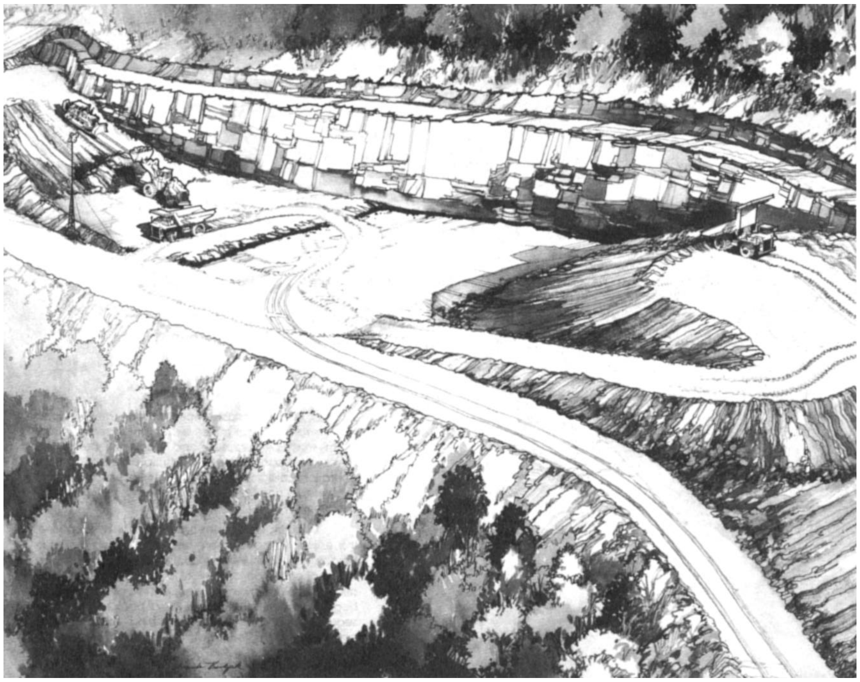


Fig. 14.3.0.2. Conventional haulback mining utilizing trucks, dozers, and front-end loaders. (Anon., 1979; illustration by Frank Kulczak. By permission from Skelly and Loy, Harrisburg, PA.)

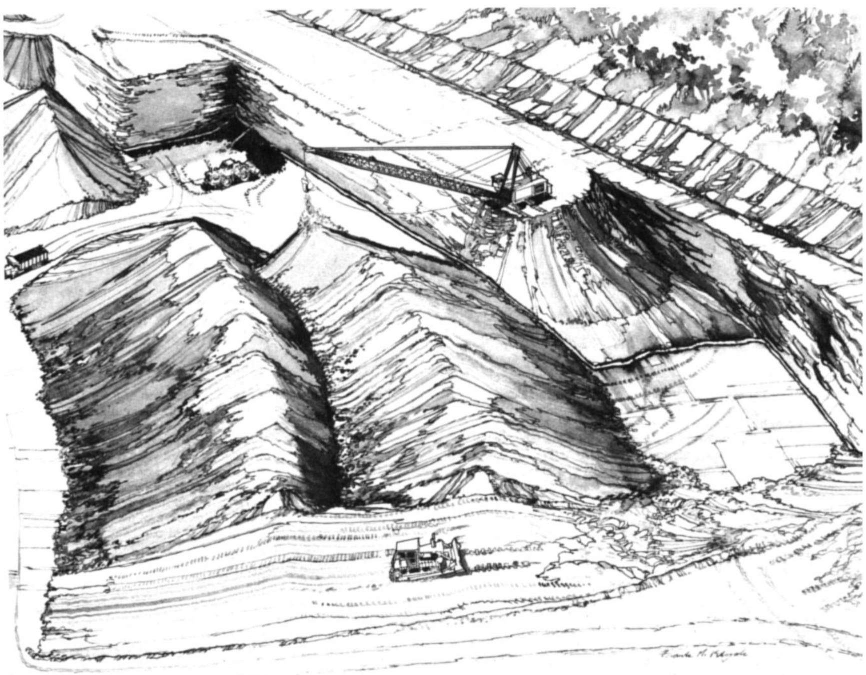


Fig. 14.3.0.3. Box-cut contour mining employing dozers and dragline. (Anon., 1979; illustration by Frank Kulczak. By permission from Skelly and Loy, Harrisburg, PA.)

disadvantages are the higher capital cost and lower maneuverability (Aiken and Wohlbier, 1968).

Proper selection of a casting machine (or combination of casting machines) is determined through use of a range diagram. This diagram represents the configuration of highwall and spoil pile in cross section. It allows graphical estimation of overburden "bank" volume and the equivalent volume of broken waste (swell is commonly 25 to 30%). This diagram allows planning of cuts and excavator moves. Careful planning is necessary to ensure that a spoil pile of given width, height, and slope angle can accommodate the volume produced by a highwall excavation of given width and depth of cover (Phelps, 1973).

Once the commodity is exposed by stripping, loading of coal or other materials may be done directly with BWEs, FELs, electric shovels, hydraulic shovels, or backhoes. Stronger material may require ripping or blasting for efficient loading. In the case of Florida-type phosphate deposits, high-pressure jets of water can be used to slurry the ore and transport it to a sump where it can be pumped to processing facilities.

Common applications for surface mining with concurrent reclamation can be subdivided on the basis of the deposit geometry and type. Area mining may be applied to deposits where (1) the overburden is thick and uniform down to a single, near-horizontal coal seam or down to several coal seams separated by thin parting; (2) the overburden is shallow and the coal seam is thick and near horizontal; (3) the layers of overburden are interspersed with numerous nearly horizontal coal seams; and (4) the overburden is of variable thickness covering coal seams which are dipping. Contour mining is applicable where nearly-horizontal, thin seams are exposed in mountainous terrain. Area mining can also be applied to lignite, tar sands, and phosphate. Case studies of each of these situations are presented in Chapters 14.3.1 through 14.3.8.

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14.3.1 MONOLITHIC OVERBURDEN, HORIZONTAL COAL SEAMS: BRIDGER MINE

MARK L. BRICKER

14.3.1.1 Mine Description

The Jim Bridger mine is operated by Bridger Coal Co. and is located in Sweetwater County, 35 miles (56 km) northeast of Rock Springs, WY. The general location of the mine site is shown on Fig. 14.3.1.1 and an aerial overview on Fig. 14.3.1.2.

Bridger Coal is a joint venture of the operator, NERCO Coal Corp., and Idaho Energy Resources Co. The permit boundary encompasses in excess of 20,000 acres (81 km^2) and is nestled against the western side of the Continental Divide.

Production started on July 3, 1974, and by the end of 1989, the mine had produced 83 million tons (75 Mt). Annual production averages 5 to 7 million tons (4.5 to 6.4 Mt) of run-of-mine coal. Coal is consumed on site by the Jim Bridger Power Plant, a 2000-MW generating station, and the electricity is transmitted to the Pacific Northwest power grid (Oregon, Idaho, Northern California, etc.).

Dragline stripping operations are scheduled around the clock, two 12-hr shifts/day, seven days/week, 356 days/year (nine holidays are observed). A dragline crew works a 12-hr shift each day for four days and then has the next four days off. The remaining operations are scheduled for three 8-hr shifts/day, five days/week, 251 days/year. Some maintenance and warehouse personnel work the same schedule as the draglines.

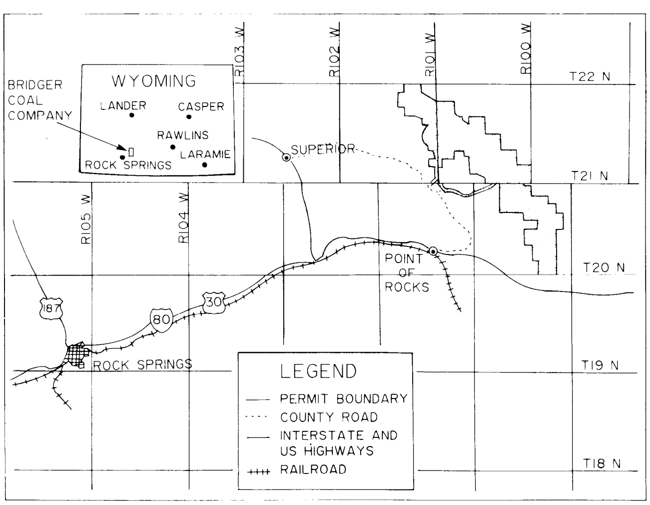

Fig. 14.3.1.1. General location map.

Fig. 14.3.1.2. Aerial overview of mine site and power plant.

The mine employs four draglines and one shovel for casting operations and two front-end loaders (FELs), one shovel, and one hydraulic backhoe for five to six active production faces. The total number of employees at the mine is 422. Of this number approximately 109 are in supervisory or administrative positions.

14.3.1.2 Deposit Description and Geology

The Jim Bridger mine is located on the northeastern flank of the Rock Springs uplift, a broad, asymmetrical anticlinal feature about 60 miles (97 km) long and 35 miles (56 km) wide with a north-trending axis. This uplift separates the Green River Basin to the west and the Great Divide and Washakie Basins to the east.

Precambrian rocks on the apex of the uplift are estimated to be 17,000 ft (5200 m) above the Precambrian rocks in the Green River and Washakie Basins. Normal faults with generally less than 100-ft (30-m) vertical displacement cut through the uplift. The dip of strata on the flanks is generally between 3 to 15°.

The coal seams occur in the Deadman Coal Zone of the Fort Union Formation, which is Paleocene in age. This zone is about 60 to 80 ft (18 to 24 m) above the contact with the underlying Lance Formation. The Fort Union Formation is about 1500 ft (460 m) thick, and the Lance Formation is about 900 ft (270 m) thick. There are five coal seams exposed in the mine; they are designated D5 through D1 from top to bottom. Various combinations of these seams exist throughout the field forming single-seam, two-seam, three-seam, and four-seam areas. The seams generally dip 2 to 5° to the northeast.

Sedimentary rocks of the Fort Union Formation represent depositional processes of a fine-grained, fluvial-flood-basin complex with extensive swampy conditions and minor lacustrine influence. Recognized depositional environments within the three-dimensional framework include poorly drained swamps, well-drained swamps, crevasse splays, and fine-grained channel sandstone deposits.

Overburden and interburden materials consist of interstratified sandstones, siltstones, claystones, and minor shale and thin, discontinuous limestone stringers. These stratigraphic units exhibit a high degree of rock variability, both laterally and vertically.

Table 14.3.1.1. Coal Reserves

Lease	Acres (km^2)	Tons $\times$ 10^6 (t $\times$ 10^6)
Federal Coal	8595.68 (34.79)	91.5 (83.0)
State Coal	1280.00 (5.18)	0.4 (0.36)
Union Pacific Coal	8579.46 (34.71)	103.8 (94.2)
Totals	18,455.14 (74.69)	195.7 (177.5)

Stripping ratio: 8 bank yd^3/ton (6.7 m^3/t)

Table 14.3.1.1 is a summary of reserves and stripping ratio for the mine. Coal leases are held by the federal government, the State of Wyoming, and the Union Pacific Railroad.

The coal resource is classified as subbituminous. The energy content averages 9400 Btu/lb (21,864 kJ/kg) with 18% moisture, 9.5% ash, and 0.59% sulfur.

14.3.1.3 Mine Development

Soil is removed from areas prior to stripping operations. In the early years of the mine, reclaimed areas were not available for direct application of soil. Therefore, soil was placed in stockpiles at various locations. This stockpile practice is not preferred because of the loss of soil fertility and the microbiological community. Use of stockpiled soil on reclaimed areas requires fertilization, while direct-applied soil has reduced fertilization costs over 50%. Direct-applied soil is hauled from the highwall side of the pit across or around the pit and distributed on reclaimed areas. Soil depths at Bridger range from 0 to 60 in. (0 to 1524 mm) with a field average of 15 in. (381 mm). A buffer of 600 ft (180 m) is maintained from the highwall for operational flexibility.

Once soil is removed, the overburden material is drilled and blasted. Outcrop development is generally accomplished by a scraper or truck and shovel fleet. The outcrop overburden material is placed to minimize out-of-pit spoil. After outcrop is developed, the draglines remove overburden by simple side casting or extended bench methods up to approximately 80 ft (24 m) in depth. Depths greater than 80 ft (24 m) require a multiple-pass, spoil-side, dragline stripping method.

Highwall angles generally range from 50 to 70°, with an average of 65°. The angle of repose for the spoil material is 36°.

General pit development is northwest to southeast along the strike of the coal beds. Each successive pit is farther down the dip of the coal seams. Pit widths vary greatly throughout the mine, but generally outcrop pits are 200 to 250 ft (61 to 76 m) in width. Where the overburden is less than 150 ft (46 m), the pit width varies from 150 to 200 ft (46 to 61 m). For overburden depths greater than 150 ft (46 m) in depth, pit width ranges from 120 to 150 ft (37 to 46 m) in width. An overall average pit width is 150 ft (46 m).

The active pit is approximately 9 miles (14.5 km) in length. Spread across this area are four draglines, a truck fleet and shovel operation, and a scraper fleet operation. Therefore, as many as six faces are operative at any time.

Pit access is available through a series of ramps or entries spaced at approximately 4000-ft (1220-m) intervals. Since the mine is located to the northeast of the power plant, the ramps start at the pit floor and advance through the spoil side of the pit up a 3 to 5% grade to a haul road network ending at the power plant truck dump.

Table 14.3.1.2. Drilling Equipment

Drill Name	No.	Bit Size in. (mm)	Purpose
Bucyrus-Erie 60R	1	12¼ (311)	Overburden
Drilltech D60K	1	10⅝ (270)	Overburden/ Parting
Schroeder Twin Mast	1	5¼ (133)	Coal
Drilltech D50K	1	9⅞ (251)	Overburden/ Parting
Marion M-3	1	12¼ (311)	Overburden
Drilltech D25K	1	6 (152)	Parting/ Coal

Future mine development will consist of opening the remaining southern portions within the permit boundary, which will extend the pit approximately 6 miles (9.7 km). As the southern expansion is started, final reclamation will begin in the currently active area. This development philosophy minimizes the total area disturbed and allows portions of the mine to return to its original livestock and wildlife land use as soon as possible.

14.3.1.4 Unit Operations

DRILLING AND BLASTING. Drilling is accomplished by six different makes and models of drills. Table 14.3.1.2 identifies the drill fleet at the mine.

Blast patterns are rectangular or square and vary with depth and type of material. An emulsion-product mix is used, which is a blend of 33% emulsion and 67% ANFO. In extremely wet areas, the product mix is raised to 50% emulsion and 50% ANFO. Powder factors average between 0.6 to 0.8 lb/yd^3 (0.36 to 0.47 kg/m^3) for overburden, 0.4 lb/yd^3 (0.24 kg/m^3) for coal, and 0.8 lb/yd^3 (0.47 kg/m^3) for parting.

OVERBURDEN REMOVAL. After outcrop development, draglines remove overburden by simple side casting or extended bench methods up to approximately 80 ft (24 m) in depth. These methods are illustrated in Fig. 14.3.1.3 for simple side casting and Fig. 14.3.1.4 for extended bench methods.

Depths greater than 80 ft (24 m) require a multiple-pass, spoil-side, stripping method. The overburden is split into two lifts of approximate equal depth and the first or upper lift is drilled and stripped by simply sidecasting the material into the empty pit (Figs. 14.3.1.5 and 14.3.1.6).

Next the dragline maneuvers from the highwall elevation down a ramp (cut out of the upper lift) to the lower lift elevation. Prior to cutting the ramp with the dragline, the lower lift material is drilled and blasted. The dragline strips a key cut that establishes the lower portion of the highwall and places that material behind the upper lift spoil.

Finally, the entire upper lift and lower lift key-cut spoils are leveled, and the dragline maneuvers onto the spoil-side pad. From this position, the dragline strips the remaining lower lift material and spoils the material in its final position.

Since mining operations occur in multiple-seam areas, supplemental stripping operations usually remove the interburden materials. In areas where the parting and overburden materials are approximately equal, the multiple-pass dragline operation described above is altered to allow the dragline to remove interburden materials. These interburden materials must be tested prior to mining to ensure they are not toxic. Only nontoxic materials can be deposited with dragline spoil.

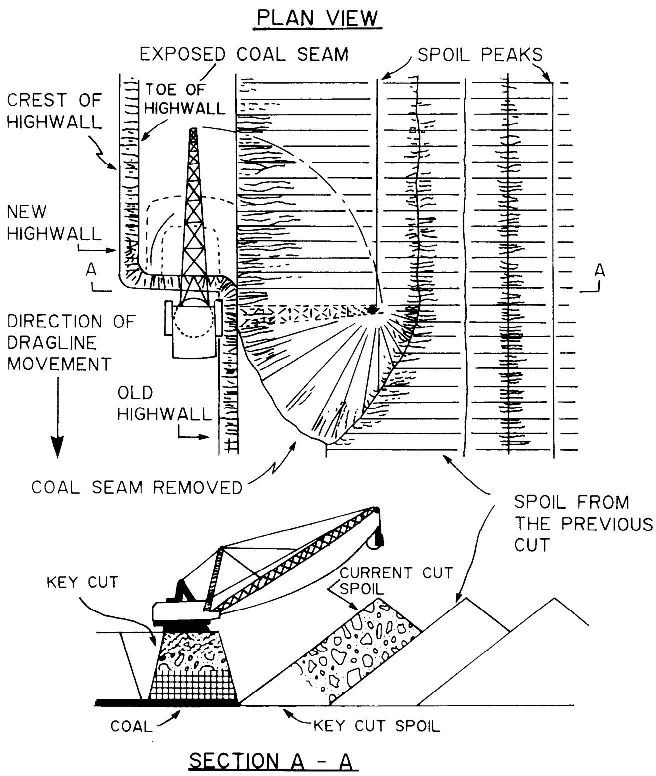

Fig. 14.3.1.3. Simple side casting.

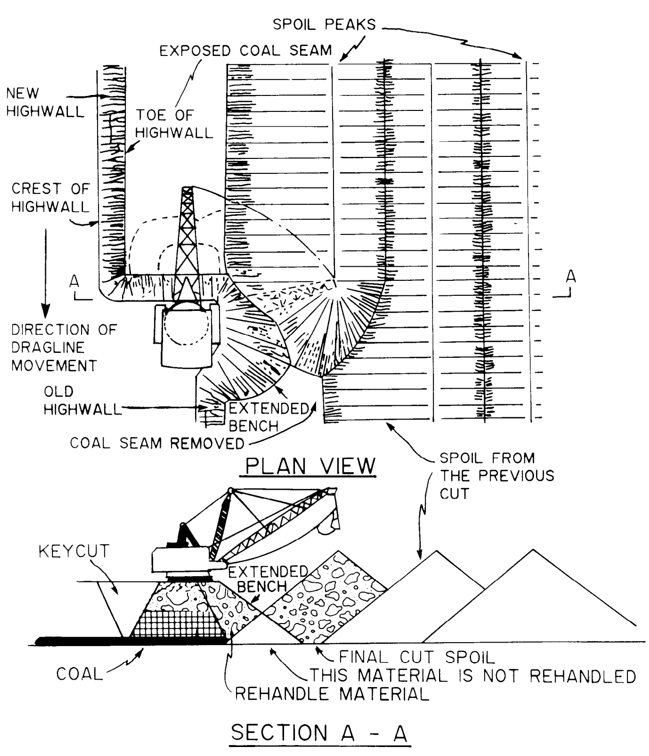

Fig. 14.3.1.4. Extended bench stripping.

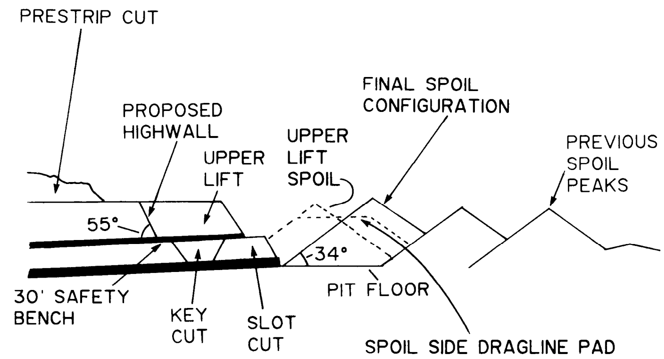

Fig. 14.3.1.5. Multiple-pass, spoil-side stripping. Conversion factor: 1 ft = 0.3048 m.

Fig. 14.3.1.6. Dragline on the spoil side.

Table 14.3.1.3. Stripping Equipment

Description	No.	Size yd^3 (m^3)	Rate yd^3/shift (m^3/shift)
Marion 8200 dragline	2	75 (57.3)	16,000 (12,234)
Page 757 dragline	1	60 (45.9)	13,000 (9940)
Page 732 dragline	1	20 (15.3)	5000 (3823)
Bucyrus-Erie 195B shovel	1	12 (9.2)	3000 (2294)
777B Caterpillar 90-ton truck	3	130 (99)	N/A
657 Caterpillar scraper	4	32 (24.5)	4500 (3441)

Note: All production rates are expressed in terms of 8-hr shifts.

Supplemental stripping operations consist of a truck-and-shovel fleet or a scraper fleet. These operations provide added flexibility to the mine by selectively handling toxic materials, performing advance benching of the highwall for the dragline, or removing interburden materials in multiple-seam areas.

Table 14.3.1.3 summarizes the stripping equipment as of 1989.

Fig. 14.3.1.7. Coal loading operations.

Table 14.3.1.4. Coal Loading and Support Equipment

Description	No.	Size yd^3 (m^3)	Rate yd^3/shift (m^3/shift)
Dart 600 C FEL	1	23 (17.6)	7100 (5429)
Caterpillar 992 FEL	1	13 (9.9)	4100 (3135)
Koehring 1266E hydraulic backhoe	1	15 (11.5)	6600 (5046)
Bucyrus-Erie 195B shovel	1	18 (13.8)	7800 (5964)
Michigan 380 RT dozer	3	N/A	N/A
Caterpillar 16G blade	3	N/A	N/A
Caterpillar 633 scraper	1	34 (26)	N/A
Caterpillar 777B truck	5	130 (99)	N/A

COAL PRODUCTION. Once the overburden and/or interburden has been removed and the top surface of the coal seam exposed, the coal surface is cleaned by rubber-tired dozers, blades, or scrapers. The coal is then drilled using twin-mast auger drills employing 5 1/4-in. (133-mm) bits. The material is lightly blasted to facilitate loading operations.

Coal loading is accomplished by FELs, electric shovels, or hydraulic backhoes (Fig. 14.3.1.7). Table 14.3.1.4 summarizes the coal loading equipment as well as the major pieces of support equipment.

HAULAGE. Prior to 1989, coal was hauled 11 miles (17.7 km) from the mine to the power plant by a fleet of thirteen 120-ton (109 t) bottom-dump trucks. Due to length of haul, a 42-in. (1067-mm) conveyor system was installed in 1989 to reduce the distance for truck haulage. Five 90-ton (82-t) end-dump trucks replaced the bottom-dump trucks. These new trucks shuttle coal a distance of approximately 4 miles (6.4 km) round trip from the pits to two truck dump sites feeding the conveyor system. Each truck dump site uses a feeder-breaker to size the coal to – 6 in. (– 147 mm) prior to loading the belt.

The conveyor system consists of a 13,000-ft (3962-m) main belt and an 8000-ft (2438-m) northern wing. The belt runs at 825 ft/min (244 m/s) and conveys coal at a rate of 1500 tph (1361 t/h). Coal is discharged at the power plant's newly constructed transfer building.

RECLAMATION. Reclamation consists of spoil pile leveling and grading, soil application, fertilization, and mulching. Spoilpile leveling is initiated by dozers and completed by the scraper fleet. The regraded area is deep-ripped with a dozer prior to soil application. Generally, in order to meet approximately original contour (AOC) criteria, as defined by the Wyoming Department of Environmental Quality (WDEQ), spoil material must be moved several hundred feet (meters), thereby requiring use of the scraper fleet.

Once an area has been regraded, soil is applied directly from the highwall or from soil stockpiles. Soil is applied to regraded areas at an average thickness of 15 in. (381 mm). A survey of elevations and results of chemical analyses of samples from the regraded area are submitted to the WDEQ for evaluation prior to soil application.

After soil is applied, the area is lightly ripped with a blade or a chisel plow along the contour, and native seed mixes are drill-seeded. Finally, most areas are mulched with hay to provide additional protection from erosion. Presently, there are over 1130 acres (4.6 km^2) of reclaimed land on the mine site, and the WDEQ has rated 90% of Bridger's reclamation as fair to good.

14.3.1.5 Product Control

Coal quality is monitored by an in-pit sampling program of cuttings from production drillholes in coal. Short proximate analyses and sodium oxide content in the ash analyses are performed at the mine utilizing the power plant's coal lab. These data are used for daily and weekly forecasts as well as actual daily quality delivery estimates. Minimal in-pit blending is required, as the coal generally exceeds the contractual quality restrictions. Long-term coal quality projections are based upon core drilling on 500- to 1000-ft (152- to 301-m) spacings. These coreholes are gridded and modeled on the computer, resulting in values for the life-of-mine plan.

14.3.1.6 Maintenance Facilities

Maintenance is performed onsite in a large diesel shop and small gas vehicle shop. These facilities are located adjacent to the power plant and main mine office. The diesel shop is 30,000 ft^2 (2787 m^2) and consists of four repair bays, one lube bay, one wash bay, a warehouse, three welding bays for dragline buckets, and offices for maintenance and warehouse personnel. Additional satellite facilities are located closer to the pit for large pieces of equipment, such as shovels and drills.

14.3.2 MONOLITHIC OVERBURDEN, THICK HORIZONTAL COAL SEAMS: JACOBS RANCH MINE

G. PAUL ANDERSON AND STEVEN J. KIRK

14.3.2.1 Mine Description

Kerr-McGee Coal Corporation's Jacobs Ranch mine is a two-pit operation located approximately 52 miles (84 km) southeast of Gillette, WY, and 11 miles (18 km) east of Wright, WY. It is one of 15 coal mines currently (1989) operating in the Powder River Basin of Wyoming (Fig. 14.3.2.1). The mines in the Powder River Basin are typified by thick coal seams and relatively thin overburden. This mine, along with the majority of surrounding mines, employs the truck-shovel method as the sole means of overburden stripping and coal mining. Two pits

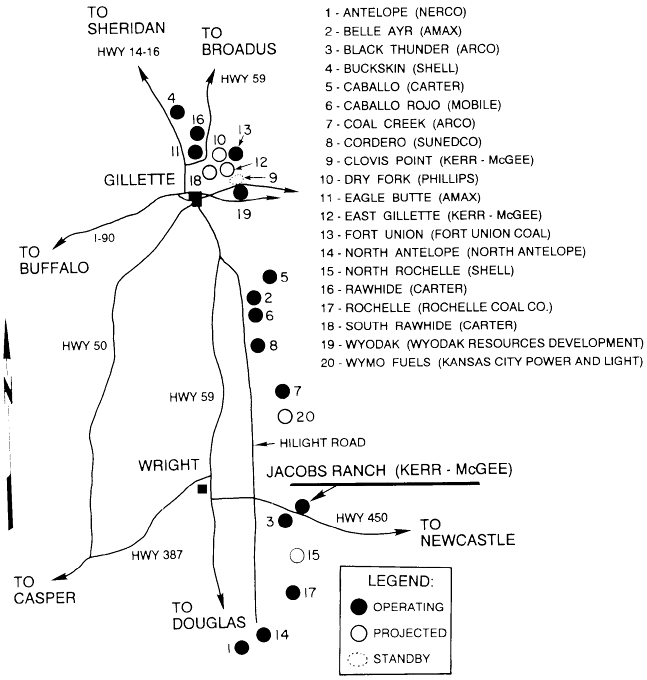

Fig. 14.3.2.1. Campbell County coal mine locations.

Fig. 14.3.2.2. Aerial photograph of Jacobs Ranch mine.

are mined so that coal can be blended to meet coal quality needs of customers (Fig. 14.3.2.2).

Construction began at Jacobs Ranch mine in 1975. Kerr-McGee erected the first shovel at Jacobs Ranch mine and began stripping overburden in August 1976. Coal shipments began in February, 1978 and are currently scheduled through the year 2009.

Annually, the mine produces 15 million tons (13 Mt) of subbituminous, low-sulfur coal that is used to generate electricity in Texas, Louisiana, Arkansas, and Oklahoma. Stripping and mining are currently conducted during three 8-hr shifts, 7 days/week, 355 days/year. Trains are loaded every day of the year, at any hour.

14.3.2.2 Deposit Description and Geology

The 12,000-mile^2 (31,000-km^2) Powder River Basin contains an estimated 22.8 billion tons (20.7 billion t) of subbituminous

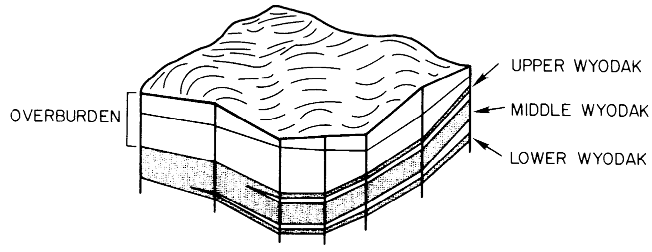

Fig. 14.3.2.3. Geologic cross section of Jacobs Ranch mine.

coal that can be economically recovered using surface mining methods. In general, the coal increases in heating value from north to south. The coal north of Gillette averages 7900 Btu/lb (18,375 kJ/kg). Coal in the southernmost portion of the Powder River Basin has an average heating value of 8900 Btu/lb (20,700 kJ/kg) (Glass, 1982).

Bounded by the Big Horn Mountains on the west, the Black Hills on the east, and the Laramie Range and accompanying highlands on the south, the Powder River Basin stretches across northeast Wyoming in the form of a syncline. The sediments that fill the basin were deposited over a 500-m.y. period, but the structure itself is a product of Laramide folding that began about 70 m.y. ago. At that time, the land folded to create a system of rivers, flood plains, and swamps (Ayers, 1986). Streams originating in the highlands to the south and the southwest carried sand and mud into the basin, where they were deposited in channels and on flood plains. The swampy areas between the channels supported a dense growth of grasses. The remains of these plants constitute some of the thickest coal deposits in the world (Ayers, 1986).

The deposit at Jacobs Ranch mine consists of 375 million tons (340 Mt) of recoverable subbituminous coal. The coal averages 8600 Btu/lb (20,000 kJ/kg), 0.46% sulfur, and 5.58% ash. Over 125 million tons (113.4 Mt) have been shipped since 1978. The 5000-acre (20.23-km^2) coal deposit is located on the 15,000-acre (60.70-km^2) Jacobs Ranch, an active cattle property.

The formation being mined at Jacobs Ranch mine is the Wyodak coalbed. This horizontal seam is continuous throughout the lease area except near the eastern and southern boundaries and in Burning Coal Draw, where it has naturally burned in the geologic past. The coalbed is comprised of up to three separate coal seams referred to as the Upper, Middle, and Lower Wyodak. Only in the southwestern part of the lease do the three seams merge to form a coal unit 50 to 60 ft (15 to 18 m) thick (Fig. 14.3.2.3). The bed is contained within the Fort Union Formation.

The Upper Wyodak coal is the uppermost coal being mined. It is present as a separate unit over all the lease, except where it merges with the Middle Wyodak Seam. The Upper Wyodak Seam ranges from 0 to 7.5 ft (0 to 2.2 m) in thickness and shows no discernible trends.

Separating the Middle Wyodak Seams from the Upper Wyodak Seam is Split A, a carbonaceous shale parting 0 to 38 ft (0 to 12 m) thick. Below Split A is the Middle Wyodak coal. The seam is approximately 40 to 55 ft (12 to 17 m) thick over much of the lease and shows no trends, except for a gradual decrease in thickness that begins about 1000 to 1500 ft (305 to 457 m) from the geologic burnline.

The Lower Wyodak coal ranges in thickness from 0 to 8.5 ft (0 to 2.6 m) over the lease. It merges with the other two coal seams in the southwest and is separated from the Middle Wyodak toward the northeast by a shaley parting. This parting, referred to as Split B, varies from 0 to 73 ft (0 to 22 m) in thickness,

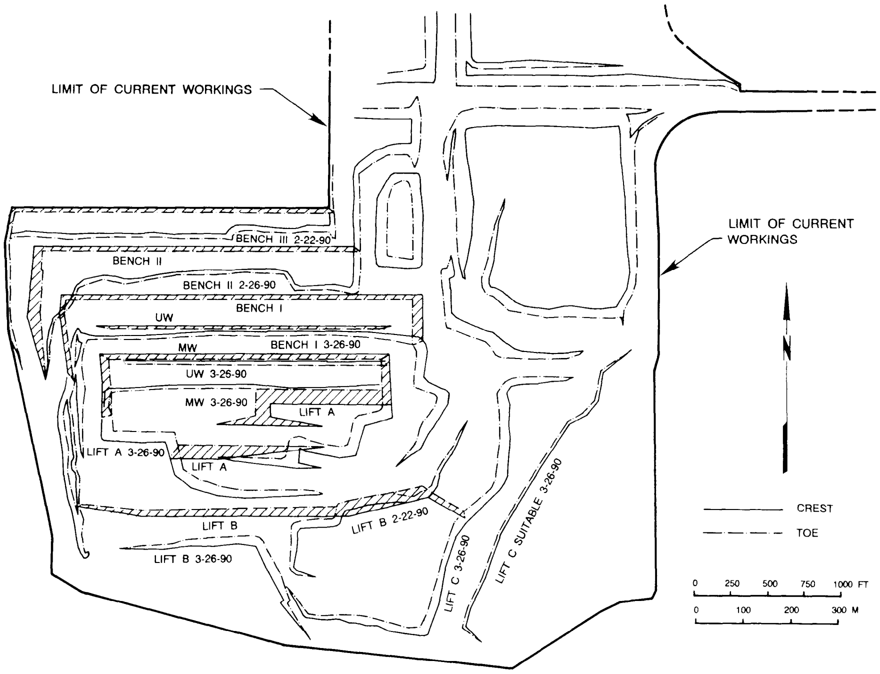

Fig. 14.3.2.4. Monthly mine plan.

averaging between 5 to 10 ft (1.5 to 3.0 m) thick over most of the lease.

About 70% of the Jacobs Ranch coal lease is rolling upland. This upland consists principally of shale and sandstone of the Wasatch Formation. Playas in scattered depressions occur on the plateau. These intermittent lakes, varying in size from less than a few yards (meters) to about 0.5 miles (0.8 km) in diameter, may be caused by local structure of the underlying strata. The remainder of the lease area is either shale slope or alluvial terrace.

The overburden-to-coal ratio varies throughout the Powder River Basin. Ratios range from 0.80:1 to almost 3.00:1. In general, overburden is thinner in the northern part of the Basin; however, stripping ratios vary from mine to mine and throughout individual deposits.

14.3.2.3 Mine Development

As is typical with surface coal mines in the United States, no surface disturbance can occur without the proper permits. Jacobs Ranch mine is regulated by several federal and state agencies. The Office of Surface Mining, the Bureau of Land Management, and the Forest Service all oversee activities at the mine. In addition, the Wyoming Department of Environmental Quality (WDEQ) requires that a mining permit be renewed every five years. An annual report also is required for WDEQ.

In order to reclaim the land successfully after mining, topsoil must be categorized, removed, stockpiled, and planted. The categorization of topsoil types and quantities is accomplished by augering on a grid with a maximum spacing of 200 ft (60 m). An environmental technician analyzes the topsoil and recommends the depth to which it should be removed. Scrapers then remove the topsoil and transport it to strategically located stockpiles or place the topsoil directly on recontoured overburden surfaces. If the topsoil is placed in a stockpile, it is seeded with a quick growing, perennial crop to hinder wind and water erosion.

The initial stripping at Jacobs Ranch mine was made in an area of low-overburden depth. The "box-cut" material was placed in permanent overburden piles adjacent to the initial openings. Due to marketing demands, the mine soon became a two-pit, blended-coal operation.

The pit and backfill design must be performed concurrently. Optimal pit design minimizes haul distance, while taking into account the after-mining contours. The overburden highwall stands at an angle of 1:1, horizontal to vertical. The backfill angle of repose is 1.5:1.

Because of the complex, three-seam coal deposit at Jacobs Ranch mine, the pit geometries are "stair-stepped" in both lateral directions from the pit floor. As mentioned previously, the coal thickness averages between 50 and 60 ft (15 to 18 m). The pit width averages approximately 2000 feet (610 m). The total disturbed length, at any time, varies between 2400 and 4000 ft (730 to 1200 m) along the axis of advance.

Haulage for overburden occurs on roads built into overburden benches (Fig. 14.3.2.4). Coal haulage to the preparation plant is on semi-permanent haul roads.

14.3.2.4 Unit Operations

DRILLING AND BLASTING. Virtually all the coal and overburden is drilled and blasted to maintain high truck and shovel productivities and low operating costs. Drillholes are spaced on approximate 50-ft (15-m) centers and are loaded with ANFO explosives if the holes are dry, and ANFO-based slurries if the holes are wet.

LOADING. The unit weight of overburden at Jacobs Ranch mine averages 111 lb/ft^3 (1778 kg/m^3) and is loaded with stripping shovels that have bucket capacities up to 36 yd^3 (27.5 m^3) (Fig. 14.3.2.5). In addition, front-end loaders (FELs) perform waste removal on an as-needed basis. The diesel/electric loaders can load 15 yd^3 (11.5 m^3) per pass.

The stripping equipment is used on an around-the-clock basis. (A shovel operates up to 6000 hr/yr after allowing for mechanical breakdowns and operating inefficiencies. FELs are used one-half that time.) The thick coal seams and homogeneous overburden contribute to high productivities and economies of scale.

Fig. 14.3.2.5. Stripping operations: 36-yd^3 (27.5m^3) shovel loading 170-ton (154-t) end-dump truck.

Fig. 14.3.2.7. Continuous-miner loading a 170-ton (154-t) bottom-dump truck.

Fig. 14.3.2.6. Production face: 40-yd^3 (30.6-m^3) shovel loading 170-ton (154-t) bottom-dump truck.

Fig. 14.3.2.8. Preparation of topsoil for reclamation.

Coal mining is normally conducted during 2 shifts/day, 5 days/week. One coal shovel is assigned to each of the two pits. Bucket capacities for the coal shovels range up to 40 yd^3 (31 m^3) (Fig. 14.3.2.6). The output from each of these shovels is regulated, on a shift-by-shift basis, in order to blend the coal properly.

In addition to the shovels and FELs, Jacobs Ranch mine recently purchased a continuous loader (Fig. 14.3.2.7). The loader employs a rotating drum that cuts coal in horizontal slices. The coal is discharged directly into trucks by a movable conveyor belt. A typical coal crew consists of a shovel operator, an oiler, and truck drivers. Support equipment includes rubber-tired dozers, crawler tractors, road graders, and water trucks. It is commonplace to have overburden and coal shovels operating simultaneously.

Productivity at Powder River Basin mines ranges between 100 and 300 tons of coal (76 and 272 t)/employee-shift. Productivity also is measured by total units produced. A unit is considered to be either a cubic yard (0.76 m^3) of overburden or a ton (0.91 t) of coal. The unit productivity in the basin ranges between 300 and 700 units/employee-shift.

HAULAGE. All material transported at Jacobs Ranch mine is moved by trucks. The overburden is hauled mainly by 170-ton (154-t) end-dump trucks that have a capacity of 104 yd^3 (80 m^3) of overburden per trip (Fig 14.3.2.5). The mine is also using four 240-ton (218-t) trucks in an attempt to further increase production efficiency. Coal is hauled primarily by 170-ton (154-t) bottom-dump trucks (Fig 14.3.2.6).

RECLAMATION. Kerr-McGee Coal's philosophy toward reclamation is straightforward: make the land more productive than it was when mining began. This simple goal translates into many hours spent planning, testing, and restoring the Wyoming range lands in an efficient and effective program, at the lowest possible cost. Recognition of these efforts includes the National Reclamation Award given by the Department of the Interior in 1988.

At Jacobs Ranch mine, reclamation is continuous. The mining process is similar to a "moving hole," as overburden removed ahead of mining is placed behind the advancing pit. Once the backfilled overburden has been contoured to match the surrounding terrain, topsoil is spread on the site. The ground is planted with blends of up to 14 types of grasses and shrubs (Fig. 14.3.2.8). The blend ensures vegetation hardy enough to withstand harsh Wyoming winters. Fences protect the lands until the grasses are healthy and established.

In addition, employees erect natural rock habitats that provide cover for small animals and vantage points for birds of prey.

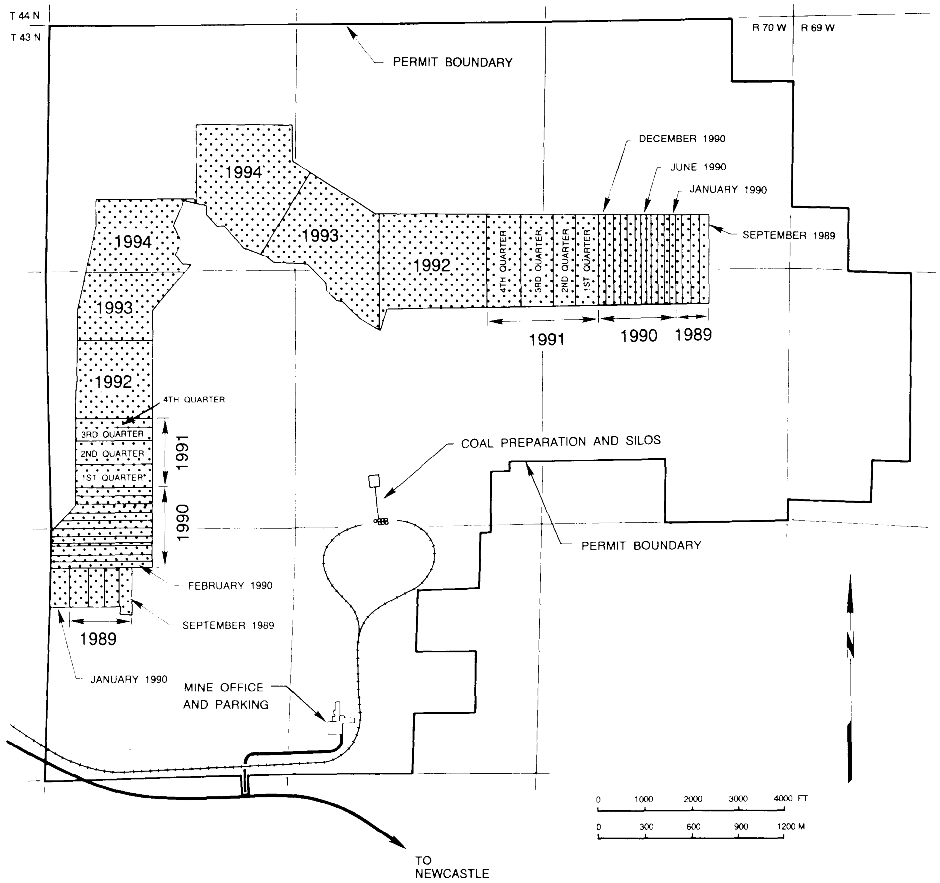

Fig. 14.3.2.9. Five-year mine plan.

Before completing reclamation, crews dig ditches and ponds to prevent run-off water from eroding topsoil.

The best testimonies to Kerr-McGee Coal's success in reclamation are the deer and antelope grazing on the protein-rich grasses growing on acreage that has yielded its coal. Studies carried out at Jacobs Ranch mine document that reclaimed areas are more productive than surrounding undisturbed range land.

14.3.2.5 Product Control

As mentioned previously, coal quality at Jacobs Ranch mine varies throughout the deposit. The challenge for the engineers at Kerr-McGee Coal Corporation was to determine an ultimate mine plan that would ensure quality coal over the life of the mine.

Every year an updated five-year mine plan is completed (Fig 14.3.2.9). Monthly mine plans shown in Fig. 14.3.2.4, are prepared in accordance with the five-year outline.

On a daily basis, a quality control engineer inspects the pit and ensures that proper grade control is maintained for the shovels and loaders. The quality control engineer is responsible for blending the coal into the silos and, eventually, into the coal trains. Onsite proximate analyses are performed for all coal shipments.

14.3.2.6 Ancillary Facilities

MAINTENANCE. Jacobs Ranch mine's ongoing maintenance program emphasizes inspections and repairs of mining equipment before a major failure hampers coal production. This effort requires two teams of diesel mechanics, welders, and maintenance electricians. The maintenance shop is large enough to accommodate 24-ft (7.3-m) tall haul trucks. Equipment too large and slow to be brought to the shop is serviced in the field. This equipment includes bulldozers, drills, and shovels.

COAL PREPARATION. The preparation plant at Jacobs Ranch mine has two parallel circuits that are each capable of producing coal at 2000 tph (1815 t/h). The coal delivered to the preparation plant hopper ranges up to 15 in. (381 mm) in diameter. A dual stage crusher in each circuit reduces the coal to its final 2-in. (51-mm) size.

Coal is sampled as it is placed in one of seven, 14,000-ton (12,700-t) silos. Coal is loaded into two unit trains using automatic train loading and top-off systems. The 115-car unit trains are loaded to within ±0.2% of the desired weight.

Individuals with supporting skills such as accounting, engineering, and computer science help ensure that Kerr-McGee Coal meets its goals. The company strives to provide the safest working environment, to deliver high-quality coal, and to continue operating as a responsible corporate citizen in the Powder River Basin.

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14.3.3 VARIABLE INTERBURDEN, HORIZONTAL MULTIPLE COAL SEAMS: COLOWYO COAL COMPANY

DAVID B. MCDONALD

14.3.3.1 Mine Description

The Colowyo Coal Co., a 50-50 partnership of W.R. Grace & Co. and M.A. Hanna Co., started coal production on March 11, 1977. The mine (Fig. 14.3.3.1) is located at Axial, CO, which is 26 miles (42 km) southwest of Craig, in Moffat County, CO. The elevation of the mine site is 7000 ft (2100 m). Colowyo operates on federal and state coal leases with some private ownership of the surface. The logical mining unit (LMU) is 11,405 acres (46 km^2).

Colowyo's 1989 production rate is 4.1 million tpy (3.7 Mt/a) of coal that requires the removal of 27 million yd^3 (21.4 million m^3) of virgin overburden and 3.8 million yd^3 (2.9 million m^3) of rehandle. The mine is designed and permitted to operate at a maximum production rate of 4.4 million tpy (4 Mt/a). In order to produce at this rate, Colowyo operates 323$\frac{2}{3}$ days/year, 3 shifts/day. Twenty shifts/week are scheduled in operations, with the 21st shift being a maintenance shift. The remaining days not worked are 14 days for a vacation shutdown, 10 holidays, and 17$\frac{1}{3}$ days that comprise the 21st shift.

14.3.3.2 Deposit Description and Geology

GEOLOGIC SETTING.

Formation—The coal measures mined at Colowyo are contained in the Williams Fork Formation, part of the Late Cretaceous Mesa-Verde Group deposited approximately 70 m.y. ago. In the western United States, the Mesa-Verde Group contains several coal-bearing formations. The Williams Fork Formation consists of alternating beds of sandstone, siltstone, shale, and coal, typical of deposits along a linear clastic shoreline. The Williams Fork Formation is approximately 1600 ft (490 m) thick in the vicinity of the Colowyo mine.

Structure—The predominate regional structure is influenced by the White River/Sierra Madre uplift to the east, the Uinta Uplift to the west, and the Sand Wash Basin to the north. Within

1 - Primary coal crusher
2 - 37 yd^3 (28.5 m^3) dragline
3 - 27 yd^3 (20.5 m^3) dragline
4 - 25 yd^3 (19.2 m^3) electric shovel
5 - 28 yd^3 (21.6 m^3) electric shovel
6 - Overburden drill
7 - Equipment parking lot
8 - Explosive storage
9 - In-pit rock crusher
10 - Coal cleaning
11 - Overburden shot-rock inventory
12 - Topsoil removal
13 - Topsoil replacement - final reclamation
14 - Active waste dumps - truck and shovel

Fig. 14.3.3.1. Aerial view of the Colowyo mine looking eastward.

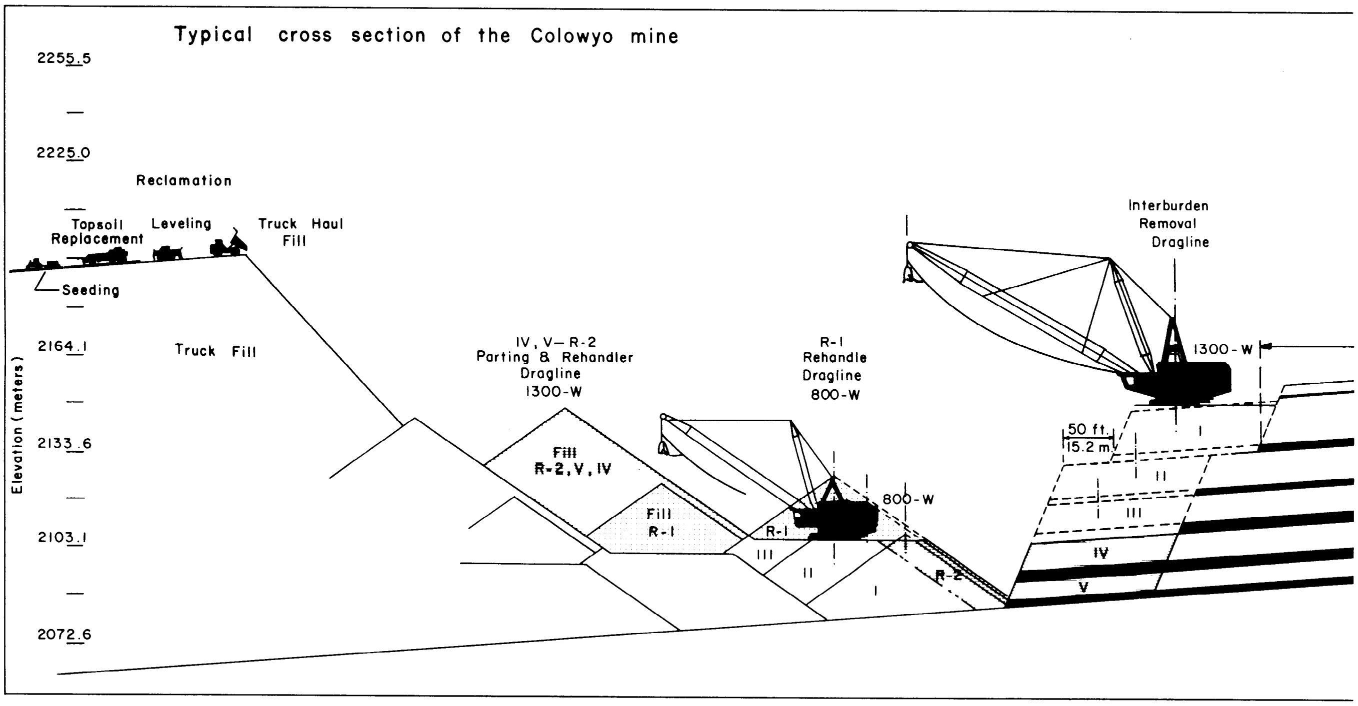

Fig. 14.3.3.2. Typical cross section of Colowyo mine.

the mine area, the geologic attitude of the overburden and coal seams strike at an orientation of N70°W and dip 4 to 6° north. Minor interruptions in the local attitude are seen through the effects of differential compaction linked to changes in interburden material and thickness. Local structural anomalies surrounding the Colowyo leases cause abrupt changes in attitude beyond the Colowyo lease boundaries.

Maximum overburden depth, including the thickness of the coal seams, is about 400 ft (120 m). Near outcrop locations, where the upper coal seams have eroded, the minimum depth of the pit will be about 275 ft (85 m). Interburden depths range from 12 ft (4 m) to 90 ft (30 m).

NUMBER AND THICKNESS OF SEAMS. Colowyo mines coal from eight major seams. In addition to the eight-seam operation, two of the major seams split within the area of the pit, bringing the total number of coal seams currently recovered to ten. Fig. 14.3.3.2 is a typical cross section showing the geometry and distribution of the coal seams mined at Colowyo.

PROPERTIES OF MATERIALS. The coal rank is subbituminous ‘A’ and ‘B’. The coal averages 10,700 Btu/lb (24,888 kJ/kg), 0.2 to 0.4% sulfur, 4 to 6% ash, and 14 to 16% moisture. The overburden and interburden consists of sandstone, siltstone, shale, and claystone.

STRIPPING RATIO. The in-place stripping ratio is 6.5 yd^3/ton (5.5 m^3/t) and will increase to 7.5 yd^3/ton (6.3 m^3/t) due to coal splits and subsequent loss in recovery.

ESTIMATED RESERVES. The recoverable reserves amenable to surface mining include 138 million tons (125 million t). The recoverable underground reserves are estimated at 74 million tons (67 Mt) as of 1989.

14.3.3.3 Mine Development

The eight major seams can be mined by either of two different scenarios: (1) all truck and shovel or (2) a combination of truck and shovel and draglines. The latter system was used in the mine design due to the inherent cost advantage of draglines. The mine is separated into two areas using different types of mining equipment but still integrally tied together. The interburden for the first three coal seams is mined by standard truck and shovel mining methods and has a stripping ratio of approximately 10 yd^3/ton (8.4 m^3/t). The remaining five coal seams are mined with draglines, with a stripping ratio of approximately 4:1. The mining cuts are 150 ft (46 m) wide in both mining areas, and the two mining areas are separated by at least two mining cuts (Fig. 14.3.3.2). This mining configuration of five dragline seams and three truck-and-shovel seams is designed for a nominal 4 million tpy (3.6 Mt/a) of coal production. Increasing or decreasing annual production requires lowering or increasing the vertical height of the dragline highwall and a corresponding addition or reduction in the truck and shovel area. In all production scenarios, the two mining systems must move through the deposit at the same rate. The constant rate keeps the truck haulage cycle distance approximately the same, which, in turn, stabilizes the required haulage truck fleet. In this scenario, the mine is a moving V, which removes approximately three mining cuts per year.

The mine design incorporates an in-pit or stockpile coal reserve of two-months production. The dragline area of the mine has no significant in-pit coal reserves, as sequencing of the mining process does not allow coal to be left behind the draglines without limiting dragline production. The coal must be removed and the next lower bench shot before the dragline can be moved to the next bench. The mining sequence of removing interburden—cleaning, drilling and blasting coal, mining and hauling coal, drilling and blasting overburden for the next bench, and then reconstructing the coal haul road without delaying the dragline—requires careful pit planning and engineering. Two draglines working within a pit less than 1 mile (1.6 km) in length also increases the critical nature of dragline scheduling.

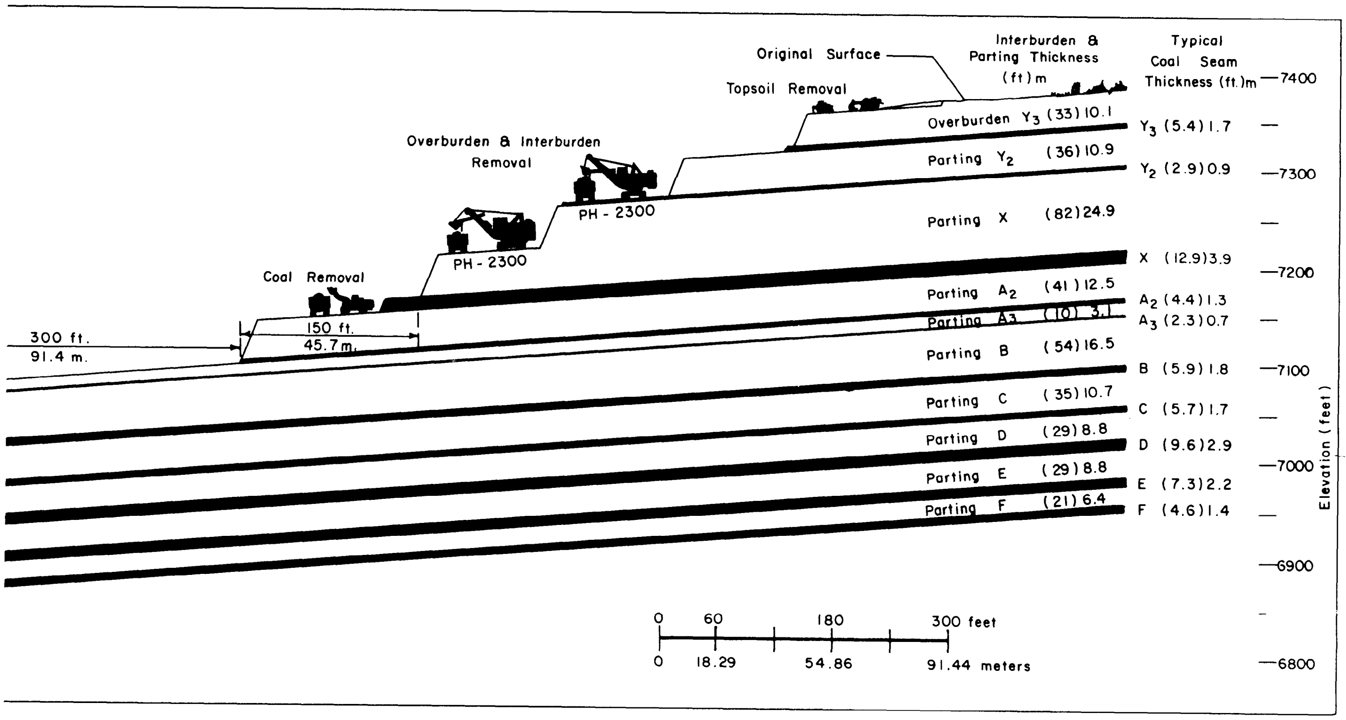

Fig. 14.3.3.2. (cont.)

In-pit reserves are maintained on the ‘X’ coal seam in the truck and shovel area.

The coal seams dip about 6%. The mature pit is 4600 ft (1400 m) long and 2000 ft (610 m) wide, with highwalls at an average angle of 63.5°. The swell factor for the dragline dump is approximately 25% and only 14 to 15% for the truck-and-shovel dump. The average truck haulage distance is 8400 ft (2560 m).

14.3.3.4 Unit Operations

DRILLING. All overburden is drilled and blasted. Overburden drilling is accomplished with three rotary drills. One electric drill is capable of drilling 10 5/8-in. (270-mm) holes and is used mainly in the truck and shovel area. The other two drills are used in the dragline area of the mine, where maneuvering room is limited and numerous holes are required due to the limited thickness of overburden. The two diesel machines drill 9 7/8-in. (250-mm) holes. All drills use 8 5/8-in. (219-mm) steel, medium formation tricone button bits and are capable of angle-drilling a 15° hole. Scheduling of drills in the dragline area of the mine requires advance planning due to the limited time available to keep the dragline supplied with broken material. Drill availability and utilization is critical for the cast blasting of ‘B’ level and requires maximum utilization of all drills. The hole size, drill pattern, and average depth of each level is shown on Table 14.3.3.1.

Coal drilling is accomplished with one Schroeder Brothers 4-in. (102-mm), twin-auger drill mounted on a John Deere tractor. The drill averages about 1500 ft (457 m) per shift and is used 5 shifts/week.

BLASTING. Blasting is accomplished daily using either ANFO or heavy ANFO. Heavy ANFO is a mixture of 25% emulsion to ANFO by weight in dry holes, while a 35% emulsion is used in dewatered wet holes. The 25% blended heavy ANFO has a specific gravity of 1.18, while the 35% blended heavy ANFO has a specific gravity of 1.28. Regular ANFO has a specific gravity of 0.85.

The ‘Y,’ ‘X,’ ‘A,’ and ‘B’ overburden benches have about 50% wet holes; therefore, these areas use the majority of the heavy ANFO. The lower overburden area, ‘C,’ ‘D,’ ‘E,’ and ‘F’ benches, rarely have wet holes and use the regular ANFO for blasting.

Table 14.3.3.1 lists the drill patterns, powder factors, tie-in and direction, and type of explosives used for each bench. Cast blasting is utilized only on ‘B’ level. Fig. 14.3.3.3 shows a typical cast blasting pattern.

LOADING. From the original surface to the ‘A_3’ seam (Fig. 14.3.3.2), overburden and interburden are removed by two 25-yd^3 (19.2-m^3) shovels and one 15-yd^3 (11.5-m^3) shovel. The two 25-yd^3 (19.2-m^3) shovels are scheduled 20 shifts/week. The 15-yd^3 (11.5-m^3) shovel is a spare. The mining cut is 150 ft (45 m) wide, which allows enough working area for double spotting of haulage trucks.

Interburden removal below the ‘A_3’ seam is accomplished with Bucyrus-Erie walking draglines (one 800W and one 1300W) with capacities of 27 yd^3 (20.8 m^3) and 37 yd^3 (28.5 m^3), respectively. Both units work from either the highwall or spoilpile locations. Beginning with the interburdens over the ‘B’ seam, the 800W dragline, located on the east end of the pit, mines toward the west; the 1300W (37-yd^3) (28.5-m^3) dragline, located on the west end of the pit, mines toward the east.

After ‘B’ interburden is removed, the 800W moves down the ramp to remove the ‘C’ interburden, mining to the east. The 1300W then mines the ‘B’ ramp and removes the remaining ‘C’ interburden from west to east.

In the ‘D’ seam interburden, the 800W moves down the ramp and mines a small portion of the ‘D’ interburden on the east end of the pit, while the 1300W completes removal of the ‘C’ interburden. The 1300W then replaces the 800W and com-

Table 14.3.3.1. Drilling and Blasting

Bench or Area	Hole Size in. (mm)	Drill Pattern ft (m)	Average Hole Depth ft (m)	Tie-in and Direction	Explosive	Powder Factor lb/yd^3 (kg/m^3)
'X' and 'Y'[a]	$10\frac{5}{8}$ (270 mm)	37 × 42 staggered (11.3 × 12.8)	50 (15.2)	30° echelon to outside	Heavy ANFO	0.63 (0.37)
'A'	$9\frac{7}{8}$ (251 mm)	30 × 35 staggered (9.1 × 10.7)	45 (13.7)	30° echelon to outside	Heavy ANFO	0.63 (0.37)
'B'	$10\frac{5}{8}$ (271 mm)	variable burden staggered	64 (19.5)	Echelon to outside	Heavy ANFO	1.25 (0.74)
'C' West	$9\frac{7}{8}$ (251 mm)	18 × 24 staggered (5.5 × 7.3)	15 (4.6)	Rows to outside	ANFO	0.55 (0.33)
'C' East	$9\frac{7}{8}$ (251 mm)	25 × 30 staggered (7.6 × 9.1)	35 (10.7)	Rows to outside	Heavy ANFO	0.75 (0.44)
'D' West	$9\frac{7}{8}$ (251 mm)	25 × 30 staggered (7.6 × 9.1)	30 (9.1)	Rows to outside	Heavy ANFO	0.65 (0.39)
'D' East	$9\frac{7}{8}$ (251 mm)	18 × 24 staggered (5.5 × 7.3)	20 (6.1)	Rows to center	ANFO	0.65 (0.39)
'E'	$9\frac{7}{8}$ (251 mm)	21 × 26 staggered (6.4 × 7.9)	30 (9.1)	Rows to center	ANFO	0.72 (0.43)
'F'	$9\frac{7}{8}$ (251 mm)	14 × 20 rectangular (4.3 × 6.1)	10 (3.0)	Rows to center	ANFO	0.50 (0.30)
All coal seams	4 (102 mm)	12 × 12 square (3.7 × 3.7)	6 (1.8)	No delays	Packaged gel	0.14[b] (0.06)[c]

[a] Upper bench has 6 ft (1.8 m) of subdrilling and is drilled to a grade. Lower bench is drilled to coal.
[b] lb/ton of coal.
[c] kg/t of coal.

pletes the removal of the 'D' interburden, moving once more from west to east. At this point, the 800W rehandles a portion of the 'B,' 'C,' and 'D' interburdens, preparing the spoilpile from west to east so the 1300W can remove 'E' and 'F' interburdens. The 1300W digs a coal ramp for 'E' and 'F' coal removal on the west end of the pit and then proceeds west to east again, removing 'E' interburden from the spoil side and casting spoil 180° against the previous cut spoil. Meanwhile, the 800W removes a necessary portion of the previous cut spoil, starting in the center of the pit and proceeds from west to east to make room for the 1300W to spoil 'E' interburden.

Once the 'E' interburden is removed, the 1300W begins on the west end, removing 'F' interburden from the spoil side, advancing from west to east. The dragline exits the pit via a 10% ramp on the east side through the spoilpile and walks to the west end of 'B' level to start its sequence over again. During this period, the 800W pulls 'F' rehandle and incurs some idle time waiting for the east 'B' bench to be blasted. Table 14.3.3.2 shows the major equipment and corresponding productivity.

A 22-yd^3 (16.9-m^3) front-end loader (FEL) is used to load coal and to remove the overburden when dressing a coal face. This overburden comes from blasting the material directly over coal. Smaller 4- to 7.5yd^3 (3.1- to 5.8-m^3) FELs are used to load topsoil and clean the top of the coal.

HAULAGE.

Overburden—Two 25-yd^3 (19.2-m^3) class electric shovels are scheduled to remove overburden. One shovel is assigned to each side of the pit and each works to a point that is equal distance between the working face and the centroid of the truck dump. Each shovel is scheduled with six haulage trucks. The haulage profile is balanced by working one shovel on the higher benches on a long haul and the other on the lower benches with a short haul. The haulage truck fleet consists of fourteen 170-ton (154.2-t) trucks, three 120-ton (109-t) rock trucks, and four 120-ton (109-t) end-dump coal trucks.

The haulage roads for overburden have a maximum 8% grade with a preferred grade of 6%. Coal hauls from the dragline area have a maximum grade of 10%, with a preferred grade of 8%. The coal haul ramp is located on the west end of the pit.

Coal—Coal is mined with a 22-yd^3 (16.9-m^3) FEL. From the pit, coal moves in 120-ton (108.8-t) end-dump trucks to a primary crusher on the pit rim. The primary crusher is a feeder breaker that reduces pit-run coal to a 6 × 0-in. (152 × 0-mm) product. The coal is then stored in a 150-ton (136-t) surge bin. Fifty-ton (45-t) bottom-dump coal haulers are loaded from the surge bin and haul the coal 3.7 miles (5.92 km) to the rail loadout facility. These coal haulers are heavy-duty, on-highway trucks that travel loaded down a 7.5% grade to the secondary crusher at the rail loadout. The trucks' primary braking system is an electric magnetic retarder that is located in two of the three rear axles on the trailer. The trucks also have a normal braking system, and the tractors are equipped with Allison transmissions for more retarding power.

Topsoil—An average of 18 in. (457 mm) of topsoil over 50 acres (0.2 km^2) is removed each year in front of the advancing mine faces. The topsoil is dozed into piles and loaded into three 50-ton (45-t) haulage trucks by a 7.5-yd^3 (5.8-m^3) loader. The topsoil is then placed on the dump surface that has been prepared for reclamation. During the time this fleet is not moving topsoil, it is used to build haulage roads and to haul mine-made crushed rock for road surfacing.

RECLAMATION. During the years of exploration, and prior to active surface mining, Colowyo conducted extensive baseline environmental studies. In addition, a 3-acre (12,141-m^2) area was disturbed to simulate mining, and many reclamation studies (including runoff plots, shrub establishment, mulching, and planned species trials) were conducted for use in developing the Colowyo reclamation plan.

When the Colowyo mine was permitted by the State of Colorado under the permanent program of the Surface Mining Control and Reclamation Act, Colowyo submitted a complete application through the year 2017 and thereby obtained a life-of-mine approval. By obtaining a life-of-mine approval, Colowyo has the

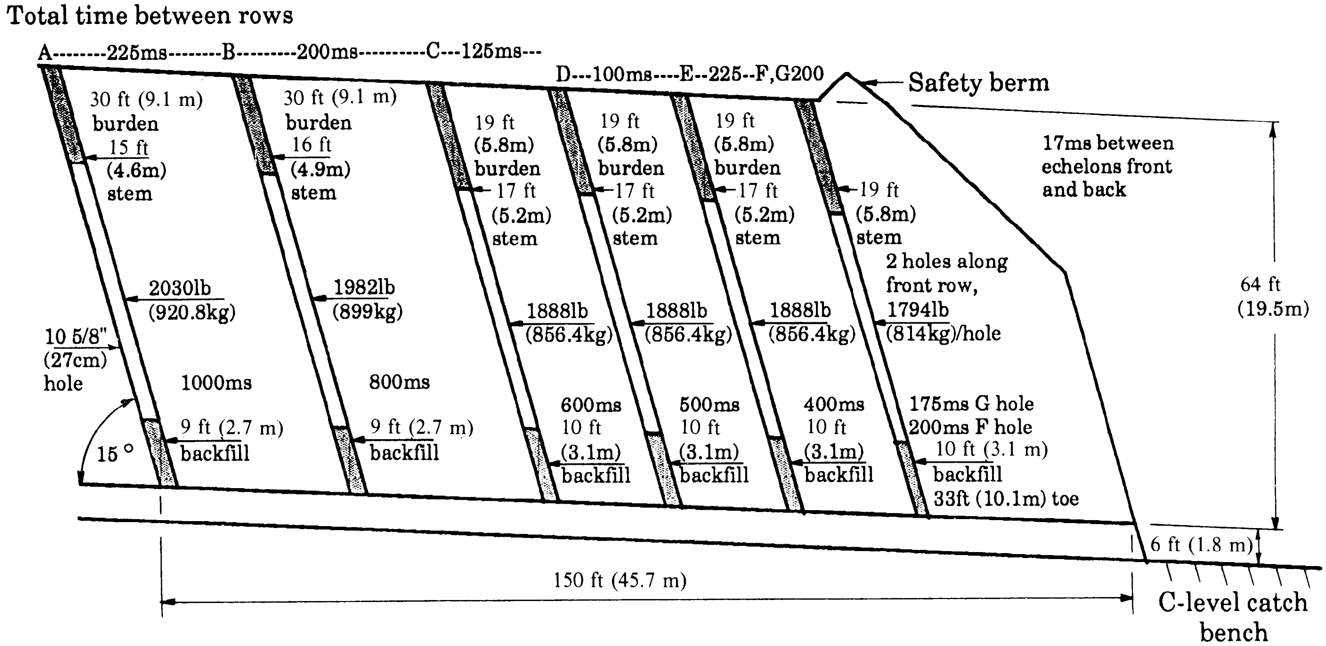

Fig. 14.3.3.3. Typical B-level cast blast pattern.

Table 14.3.3.2. Major Equipment

Equipment	No. Units	Capacity yd³/hr (m³/h)	Capacity tons/hr (t/h)	Annual Volume yd³/yr × 10⁶ (m³/yr × 10⁶)	Annual Volume tons/yr × 10⁶ (t/yr × 10⁶)
37 yd³(28.3 m³) dragline 285 ft (87 m) boom	1	1700 1292		9.33 7.09	
27 yd³ (20.6 m³) dragline 195 ft (59 m) boom	1	1070 813		4.47 3.40	
25 yd³ (19.1 m³) shovel	2	1400 1064		15.01 11.41	
15 yd³ (11.5 m³) shovel	1	900 604		1.87 1.42	
22 yd³ (16.8 m³) loader	2	615 467	660 599	0.72 0.55	4.10 3.72
170 ton (154 t) rock truck	14	300 228		15.37 11.68	
120 ton (109 t) rock truck	3	200 146		2.33 1.77	
120 ton (109 t) coal truck	4		255 231		4.10 3.72
50 ton (45 t) coal truck	6		125 113		4.10 3.72

"right of successive renewal" for each five-year permit term through 2017.

Reclamation activities are conducted in accordance with the approved reclamation plan which includes a "reclamation topography map" representing the approximate original contour. By survey control, overburden is dumped by trucks into the final configuration of the reclamation topography map and graded to final configuration by dozers. Where practical, the uppermost weathered overburden is placed on the surface to provide the most acceptable rooting material to enhance revegetation.

After final grading of the overburden is completed, topsoil is end-dumped with 50-ton (46-t) and 120-ton (109-t) trucks. Since the mine is in a mature configuration, the regraded area is prepared prior to topsoil removal for the next year, thereby allowing the topsoil to be hauled directly from the mining advance. The replaced topsoil is spread to a depth of 12 to 18 in. (305 to 457 mm) by dozers. Since the initial 6 in. (152 mm) of

Table 14.3.3.3. Coal Quality History

Year	Btu/lb	kJ/kg	Heat contents Ash (%)
1980	10,555	24,551	6.50
1981	10,585	24,621	6.02
1982	10,666	24,809	5.54
1983	10,714	24,921	5.38
1984	10,737	24,974	5.07
1985	10,722	24,974	4.91
1986	10,737	24,974	4.63
1987	10,713	24,918	4.73
1988	10,704	24,898	4.96

topsoil are an important source of native plant materials, that material is segregated and spread on top when practical.

The regraded topsoil is then checked for the required depth and sampled for plant nutrients. If necessary, appropriate amounts of nitrate-phosphate fertilizer are applied to the surface. After fertilizer is applied, all areas are tilled with a chisel plow to a depth of approximately 8 to 12 in. (203 to 305 mm) to reduce compaction.

A substantial amount of the Colowyo reclaimed area consists of 3:1 slopes. Newly reclaimed slopes require special attention to control erosion. Colowyo's D7 dozer has been equipped to pull farm implements, which allows the 3:1 slopes to be worked on the contour to increase erosion control. After the area has been chisel plowed, it is seeded by a rangeland drill using the approved seed mixture with approximately 30 different species of plants.

While seeding, contour furrows approximately 8 in. (203 mm) deep and 14 in. (356 mm) wide are cut into the topsoil by plowshares attached to the two outside ripper shanks on the D7 dozer. The use of contour furrows and the chisel plow are the primary reasons that Colowyo has been able to successfully control erosion on 3:1 slopes.

In addition, for every 100 ft (30 m) in elevation, a 2% ditch approximately 8 ft (2.4 m) wide and 2 ft (0.6 m) deep is constructed to carry any runoff that accumulates in the permanent drainages. The remaining reclaimed areas on flatter slopes are prepared in a similar manner by utilizing a farm tractor to pull the chisel plow, a rear blade for contour furrows, and the rangeland drill.

Most of the 3:1 slopes are outslopes of a 54,000,000-yd^3 (41,300,000-m^3) excess-spoil valley fill. The fill is a critical element of the Colowyo mine, and was constructed as an experimental practice. Existing regulations required, among other things, 4-ft (1.2-m) lifts. Colowyo's fill has been constructed in lifts that vary from 75 to 200 ft (23 to 61 m). The primary regulatory concerns have been over stability, and after 10 years of monitoring, no significant movement has been detected.

To meet permit requirements, approximately 15% of the reclaimed area on 10% or flatter slopes is revegetated to forbs and shrubs only. Shrubs are established from seed, seedlings, and by transplanting mature shrub clumps taken from the pit advance areas prior to topsoil stripping.

Reclaimed areas are monitored the first year after seeding and every three years thereafter to determine the reclamation success during the ten-year reclamation liability period.

14.3.3.5 Product Control

Quality of the Colowyo coal has been optimized since the mine achieved a mature pit configuration. Table 14.3.3.3 illustrates this in terms of the "as shipped" Btu/lb (kJ/kg) and ash percentage. Sulfur has been maintained at a relatively constant 0.35% during this period.

Quality is evaluated through application of the principles of statistical process control. The Quality Control Department is responsible for inspection, sampling, testing, statistical evaluation, and customer feedback.

INSPECTION. Prior to drilling and blasting, the coal is cleaned by a FEL or rubber-tired dozer. This progress of cleaning is monitored and reported by the Quality Control Department. As the coal is mined, periodic inspection is conducted to observe the extent of product dilution. Also the coal is inspected at other stages of transport and stockpiling. Observations of any inferior product are reported to the coal foremen.

SAMPLING. Sampling is done before mining, during production, and upon loading of rail cars. Pre-production sampling involves the collection of drill cuttings or channel samples just prior to mining of the coal. Production samples are taken from truck tops after loading at the primary crusher for transport to the rail loadout. As rail cars are loaded, a three-stage mechanical system collects the shipment samples. A backup system is in place for car-top sampling in the event of mechanical system failure. These systems are continuously monitored for performance through statistical testing.

TESTING. The Quality Control Laboratory performs short proximate analyses, ash fusibility, Hardgrove grindability, ash elemental analysis, and other tests as needed. A mine-site lab is maintained in order to furnish timely results.

STATISTICAL EVALUATION. Production and preproduction sample data are used to define production goals. Data from each dragline and each shovel mining cut are used to predict the potential qualities for subsequent mining cuts. Causes for deviations are normally easy to track and correct; monthly production averages then are normally close to quality goals.

Control of shipment quality is based on past performance. The Quality Control Department constructs control charts to define the expected limits of variability for each customer. Two standard deviations are used for warning limits and three for rejection limits. When a shipment is in the warning region, a search for assignable cause is made. If a shipment is in the rejection area, corrective action is taken immediately by management, and the customer is informed.

CUSTOMER FEEDBACK. Customers are willing to share with Colowyo their changing needs, especially with respect to quality parameters. The most common request from the customer is for product consistency. Consistency is primarily achieved through blending.

BLENDING. A blended product with consistent qualities is achieved by tracking certain quality parameters and recommending mining, stockpiling, and coal removal in a sequence that is compatible with the mining and customer shipment schedules. The main variations in properties are a function of coal age (depth). In-pit coal reserves are not maintained in the dragline area of the mine; therefore, the five lower coal seams are always being mined. The upper three coal seams in the truck and shovel area of the mine are naturally blended into the coal flow on an "as needed" basis. Natural blending through mining meets most requirements. The use of one- or two-layered stockpiles allows this process to be fine tuned. Table 14.3.3.4 illustrates the impact of increased blending on reducing the standard deviation (sigma) in "as received" Btu/lb (kJ/kg) for one of Colowyo's customers.

14.3.3.6 Ancillary Facilities

MAINTENANCE FACILITIES. Maintenance facilities are located at the rim of the pit. The maintenance department is

Table 14.3.3.4. Blending Deviation

Year	Standard Deviation (Btu/lb)	Standard Deviation (kJ/kg)
1984	149	347
1985	133	309
1986	110	256
1987	109	253
1988	95	221

Table 14.3.3.5. Manpower

Operations	No.
Drill operator/helper	13
Powdermen	8
Dragline operator/oiler	16
Shovel operator/oiler	16
170-ton (154-t) Haulage/truck driver	66
150-ton (136-t) On-highway Coal haulage/truck driver	22
Water/truck driver	4
Equipment operator	34
Primary crusher operator	4
Train loadout operator/helper	9
Utilityman	5
Subtotal	197
Maintenance	**No.**
Mechanics	69
Electricians	20
Welders	17
Servicemen	4
Tireman	2
Steamcleaner	2
Utilityman	2
Subtotal	116
Staff and Administrative	**No.**
Administration	9
Engineering	21
Human relations	12
Finance & accounting	29
Maintenance & operation	36
Subtotal	107
TOTAL	420

divided into three areas: (1) mobile equipment, (2) field equipment, and (3) electrical. The truck shop has eight truck bays, a tire shop, and a machine shop. Near the shop is a wash building for haulage trucks and pickups. In addition, Colowyo has a welding shop, light vehicle repair bays, electrical shop, and a dozer repair bay. The warehouse, along with its outside storage area, is attached to the truck shop. The total enclosed maintenance area is approximately 67,800 ft^2 (6300 m^2).

The 21st shift is a maintenance shift and is scheduled normally on Friday during the day. One dragline is scheduled for a 16-hour preventive maintenance every other week. On the off-week, it receives electrical maintenance for eight hours. Shovels and drills are scheduled for 16 hours of preventive maintenance every two weeks. Haulage trucks receive an assembly-line, six-hour preventive maintenance check every 150 operating hours. Every 150 hours, the maintenance cycle is increased in depth and type of maintenance accomplished.

At the annual 4.1-million ton (3.7-Mt) coal production rate, the maintenance department is staffed with 116 employees. The operating department has 197 employees. Table 14.3.3.5 shows a breakdown of each department by classification, along with staff and administration.

RAIL LOADOUT. Colowyo ships its coal by rail on the Denver & Rio Grande Western Railroad. The primary product is 1.5 × 0-in. (38 × 0-mm) crushed coal for the thermal market. The 50-ton (45.4-t) bottom-dump coal trucks deliver 6 × 0-in. (152 × 0-mm) coal to the secondary crusher located at the rail loadout. The crusher is a ring hammer mill that reduces the coal to the primary product size.

The rail loadout utilizes an open storage concept. The facility consists of a 125-ft (38-m) high stacking tube capable of stockpiling 120,000 tons (108,840 t) of coal, of which 17,000 tons (15,419 t) is live storage for direct loading into unit trains. Unit trains move through a tunnel under the stockpile and are loaded at an average rate of 5000 tph (4535 t/h), using one or two 5-ft (1.5-m) square chutes. The facility is equipped with a unit train sampling and rail weighing "in-motion" system. Stoker coal is loaded into railroad cars by a FEL on a side track and shipped weekly.

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14.3.4 MONOLITHIC OVERBURDEN, INCLINED COAL SEAMS: TRAPPER MINE

GARY M. STUBBLEFIELD AND ROGER W. FISH

14.3.4.1 Mine Description

The Trapper mine (Fig. 14.3.4.1), operated by Trapper Mining Inc., is located 6 miles (9.7 km) south of Craig in the northwest corner of Colorado. With an annual coal production of 2.1 million tons (1.9 Mt) delivered from a property encompassing 9600 acres (39 km^2), Trapper ranks as the third largest producing coal mine in the state (1991). Three electric draglines strip waste material from five dipping coal seams. These units work three 8-hour shifts/day, 5 days/week, except during periods when the strip ratio is greater than 10:1 yd^3/ton (8.4:1 m^3/t). In addition, stripping can occur 6 or 7 days/week when box-cutting is necessary. Coal extraction activities are performed 2 shifts/day, 5 days/week.

14.3.4.2 Deposit Description and Geology

Trapper is situated at the southern edge of the Green River Coal Field in a geologic setting of gently rolling, east-west-striking anticlines and synclines. The mine is on the southern flank of the Big Bottom Syncline where the coal dips at a fairly consist-

Fig. 14.3.4.1. Aerial view of Trapper mine looking southward.

Table 14.3.4.1. Average Seam Thickness and Run-Of-Mine Coal Quality

Pit/Seam	Thickness ft (m)	Heat Content Btu/lb (kJ/kg)	Sulfur (%)	Ash (%)	Moisture (%)
Ashmore:					
H Seam	5.4 (1.6)	9,900 (23,030)	0.47	5.1	18.4
I Seam	7.8 (2.4)	10,150 (23,610)	0.39	4.2	17.1
Colt:					
L Seam	5.4 (1.6)	9,650 (23,030)	0.64	10.0	17.1
Q Seam	7.0 (2.1)	10,100 (23,490)	0.32	7.1	17.9
Derringer:					
L Seam	3.3 (1.0)	10,150 (23,610)	0.37	6.0	17.3
Q Seam	11.6 (3.5)	9,850 (22,910)	0.33	7.2	18.2
R Seam	4.1 (1.2)	10,050 (23,380)	0.45	5.8	18.4
1986 Avg	—	9,900 (23,030)	0.39	7.5	16.6
1987 Avg	—	10,000 (23,260)	0.39	7.7	17.1

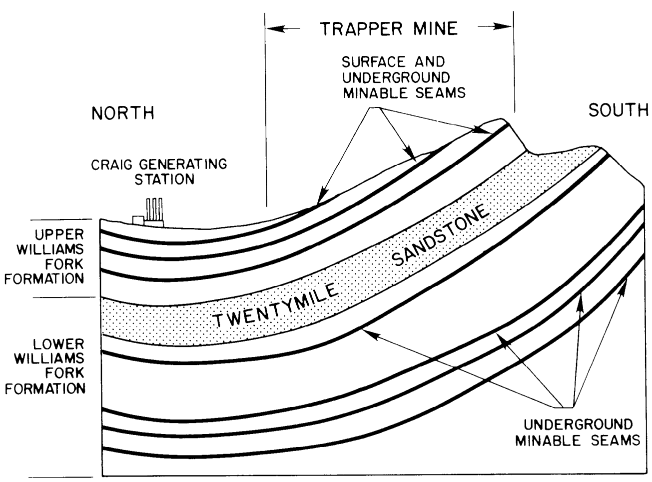


Fig. 14.3.4.2. Typical Trapper mine cross section.

ent 16% to the north, about 2% steeper than the surface of the land. There are few faults on the property. Overburden ranges from 0 to 140 ft (0 to 43 m) at the economic stripping limit. Fig. 14.3.4.2 presents a typical cross section of the mining area.

Seven major surface-minable coal seams exist within the permit boundary, with five of these seams mined from three active pits. Coal at Trapper is ranked as high-volatile subbituminous and is dedicated to steam coal application. Currently, delivered coal averages 9900 Btu/lb (23,028 kJ/kg), 0.4% sulfur, 7.5% ash, 44% fixed carbon, and 17% moisture, with a coal recovery of roughly 90%. Table 14.3.4.1 exhibits the average thickness and run-of-mine quality of the seams mined at Trapper.

Waste material (overburden and interburden) consists primarily of interbedded shale, siltstone, and sandstone layers. The waste is generally soft with an average in-place density of 3400 lb/yd^3 (2017 kg/m^3). Trapper's minable strip ratio (1991) averages 9:1 yd^3/ton (7.6:1 m^3/t).

An estimated total of 67 million tons (61 Mt) recoverable, strippable reserves remain within the permitted boundary at a minable strip ratio of 8.5:1 yd^3/ton or 7.2:1 (m^3/t). Coal recovery is expected to average 89% overall. Of the 9600 surface acres (39 km^3) of reserves controlled by Trapper, 51% is leased from the State of Colorado, 44% from the federal government, and the remaining 5% from Moffat County or private individuals. Underground reserves controlled by Trapper above the Twentymile Sandstone (Fig. 14.3.4.2) total an estimated 270 million minable tons (245 Mt) or 130 million recoverable tons (117 Mt) from five deep coal seams. An additional 300 million minable tons (272 Mt) exist below the Twentymile Sandstone.

14.3.4.3 Mine Development

TOPSOIL REMOVAL. Trapper's mining sequence begins with topsoil removal operations during the summer months of June through October. Caterpillar (Cat) D9 dozers first windrow surface vegetation onto an adjacent area where the topsoil has previously been removed. Six to eight Cat 627E scrapers are then used to remove an average of 17 in. (432 mm) of topsoil (range is 6 to 24 in., or 152 to 610 mm) from an area large enough to allow mining to continue until the following summer. After the topsoil is loaded, it is either hauled directly to previously mined and recontoured areas or stockpiled for later deposition. Topsoil is transported directly to regraded areas approximately 40% of the time by way of either across-pit ramps or haulage routes around the end of the pits.

Topsoiling productivities are 150 and 200 yd^3/hr (115 and 153 m^3/h) for direct respread and stockpiling, respectively. Over 600,000 yd^3 (459,000 m^3) are handled each year.

LAYOUT AND ACCESS TO PITS. Trapper presently has four active pits: Ashmore, Browning, Colt, and Derringer. Each extracts two major coal seams. As the Colt pit progresses to the west, it will be mined in conjunction with the Browning pit. A recently idled pit, Enfield, was a single-seam operation that employed cast blasting. Operations at Enfield were moved to Colt in order to recover coal before a neighboring underground longwall mined below the reserves. Three additional pits, Flintlock, Gatlin, and Hawkin, are projected to be activated in the future as reserves are exhausted in current pit areas.

The pits are accessed by 60-ft (18-m) wide main haul roads. Ramps into the pit are placed approximately 1000 ft (304 m) apart and may be situated through the spoil or highwall, depending largely on which side the main haul road accesses the pit.

OVERBURDEN REMOVAL. Given the dipping seams and sloping topography at Trapper, the three theoretical techniques available for a dragline to remove overburden are strikeline, dipline, and oblique stripping. Fig. 14.3.4.3 present the general layout of the strikeline and dipline methods. *Oblique stripping* is a combination of the first two methods where the pits are oriented between the strike and dip of a seam.

With the strikeline method, the dragline advances the cut along the strike of the coal. Strikeline pits can be laid out either beginning at the outcrop or at the point of maximum economic overburden depth. It is obvious that the latter option will incur the highest initial stripping costs, since stripping operations begin at the point of maximum cover. Other problems with beginning at the deepest overburden include highwall instability, possible coal reserve sterilization down-dip from the boxcut, and poor utilization of equipment in later years. Alternatively, starting at the outcrop eliminates high initial production costs. However, as the pits get deeper, smaller areas are available to spoil the increasing quantities of waste. This problem, referred to as being spoil-bound, requires substantial waste rehandling, a process that is both inefficient and costly.

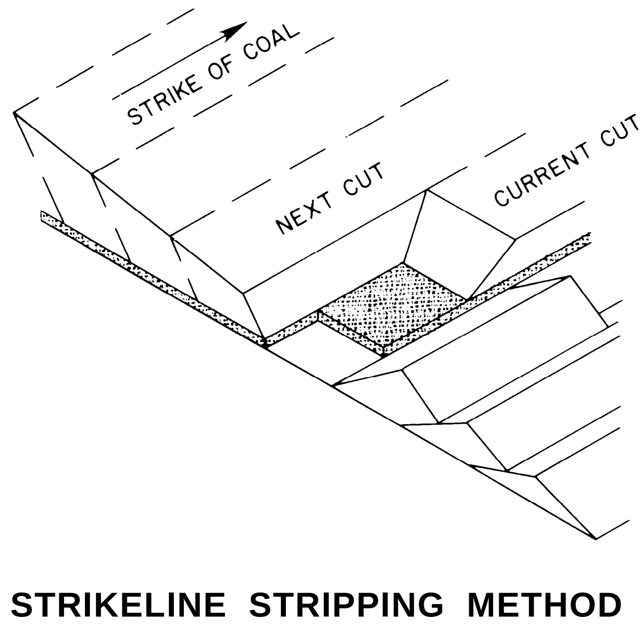
Oblique stripping is defined as aligning the pits at an angle to the strike of the coal. The technique assumes the same advantages and disadvantages as the dipline method as the orientation of the pit approaches the dip direction. Conversely, it assumes the same advantages and disadvantages as the strikeline method as the orientation approaches the strike direction.

Trapper selected the dipline method because of the greater highwall stability and lower rehandle quantities relative to the strikeline method. Another advantage of the dipline method is the uniform stripping costs and equipment requirements realized from similar amounts of overburden taken in each cut. To make this stripping plan work, Trapper had to develop an efficient method of moving the draglines up and down the dip slope.

Dragline Movement—The procedure for moving along the pit length has evolved from a switchback ramping system to a benching or "see-saw" method. In the switchback method shown in Fig. 14.3.4.4, the dragline digs a ramp to an intermediate bench approximately 45° away from the pit and then makes a switchback to the next bench above or below the original bench. Since this method requires considerable walk-road construction and rehandle, the see-saw method was developed to reduce the rehandle and provide the dragline a longer digging time on the same bench.

The sequence for the see-saw method moving uphill on the spoil begins with the dragline at position 1 on pad 1 as presented in Fig. 14.3.4.5. Using interburden material from the pit, the dragline builds a bench extension, area A, and moves onto the extension to position 2. While stripping from this position, the dragline spoils waste into area B and a dozer builds an 8% ramp and another bench 20 ft (6 m) higher than pad 1. When complete, the dragline walks up the ramp to pad 2 at position 3 and begins the sequence again. The reverse procedure is utilized for moving downhill if required.

Stripping Sequence—Several plans have been implemented in the process of developing a method that maximizes dragline productivity. It has been a learning process that has included input from many sources, especially the dragline crews themselves. Through the years, Trapper has developed a method that allows the draglines to remove overburden from the highwall as they move downhill and to extract interburden as they dig back uphill on the spoil side. Since the four active pits have fairly similar sequences, the following is a general discussion of the method employed at Trapper.



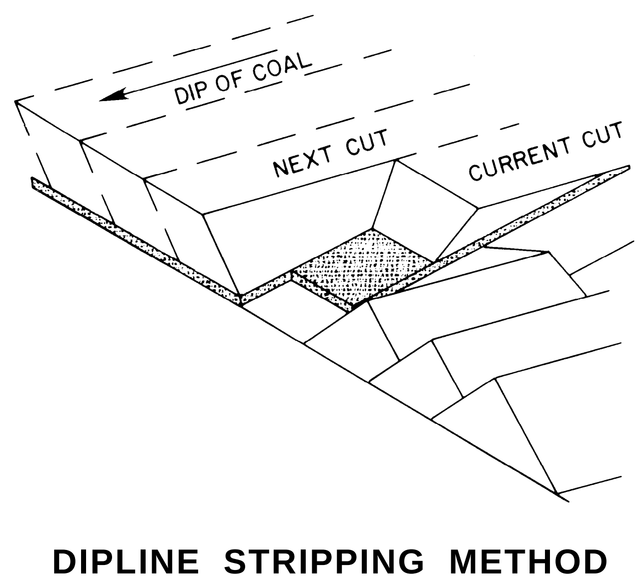


Fig. 14.3.4.3. Stripping methods for dipping seams.

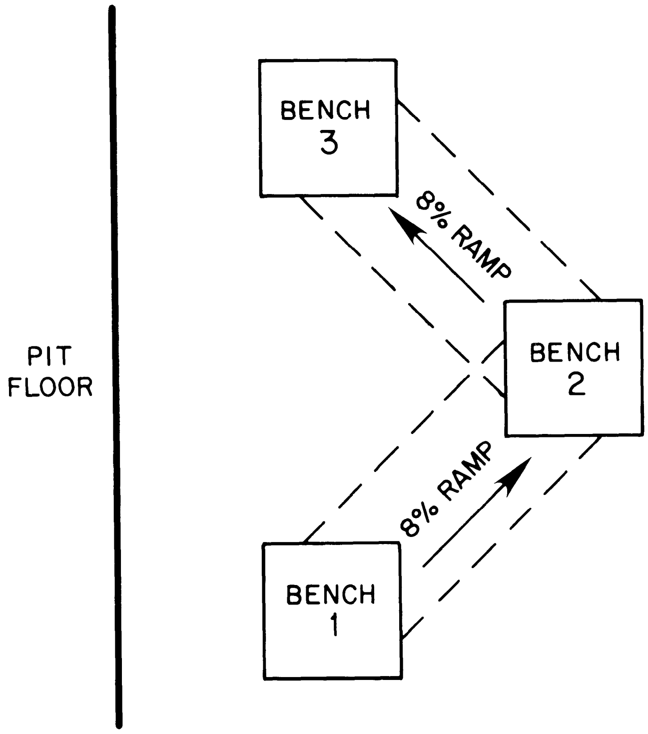


Fig. 14.3.4.4. Switchback method on the spoil side. Plan view showing dragline movement up the hill.

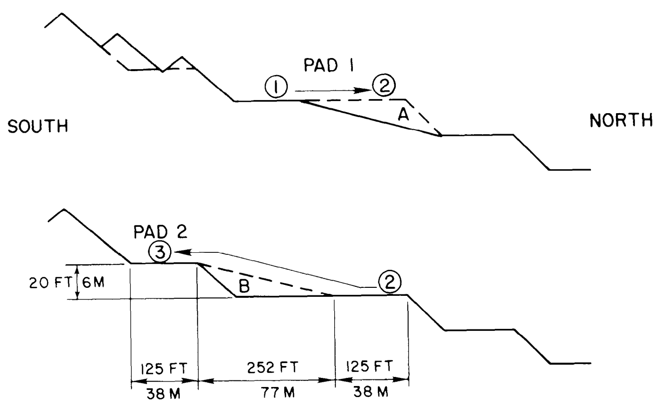


Fig. 14.3.4.5. See-saw method on the spoil side. Plan view showing dragline movement up the hill.

Typically, the dragline begins at the top of the hill near the outcrop and digs overburden to the bottom seam on the highwall side. As the dragline progresses down the hill, it encounters the upper seam outcrop and switches to removing overburden above the upper seam until it reaches maximum economic cover at the end of the pit. Using an in-pit spoil bridge, the dragline crosses over to the spoil side of the pit and waits until the upper seam is mined out and the interburden to the lower seam is drilled and blasted. Once blasted, the interburden is removed by the dragline, exposing the lower seam as it continues back up the hill using the see-saw method shown in Fig. 14.3.4.5. As the dragline reaches the point where the upper seam originally outcropped, it bridges back over to the highwall side to begin the sequence again. Thus the dragline strips down and then back up the hill to complete one cut.

Table 14.3.4.2. Major Equipment Productivities

Machine Type	Productivity
Draglines, 30 yd^3 ($22.9\ m^3$)	1400 yd^3/digging hr ($1070\ m^3$/digging h)
Overburden drills	120 ft/drilling hr (36.6 m/drilling h)
Coal drill	720 ft/drilling hr (219 m/drilling h)
Coal backhoes, 15 yd^3 ($11.5\ m^3$)	550 tons/operating hr (499 t/operating h)
Coal trucks, 50 tons (45.4 t)	130 tons/hauling hr (117.9 t/hauling h)

Ashmore pit uses this method to recover the H and I seams, as does Colt to mine the L and Q seams. However, a variation of the preceding general case is used in Derringer. Here three seams are recovered (L, Q, and R), and no across-pit ramping in the middle of the pit is necessary. In portions of the pit where all three seams are recovered, the interburden is generally thin (less than 10 ft or 3 m). This allows one of the parting intervals to be pushed by a Cat D9L dozer or loaded by other equipment.

PIT DIMENSIONS. Because the coal dips more steeply than the topography, the maximum strip limit of approximately 140 ft (43 m) is reached quickly at the bottom of the hill. This situation sometimes requires the pits to be short, from 800 to 6000 ft (240 to 1830 m). Scheduling becomes difficult in short pits; thus 175-ft (53-m) pit widths are purposely designed to keep the dragline on the same working bench as long as possible before moving up or down to the next digging level.

Highwall angles vary widely from 26 to 63°, depending upon overburden competency and existence of ancient stream channels situated in the highwall. An average slope of 37° for the spoils provides adequate stability for the broken material.

14.3.4.4 Unit Operations

Productivities for major production equipment are summarized in Table 14.3.4.2. Trapper has continued to improve equipment performance, partly due to the high level of experience and dedication of its union operators. Since the mine's inception in 1976, the average hourly labor employee has over 13 years of service at the mine.

DRILLING. Waste materials are prepared by three rotary drills using 10⅝-in. (270-mm) tricone bits. The drills are positioned on a series of level benches cut into the hill slope. Patterns for blastholes vary depending on the type of material but commonly average 20 by 30 ft (6.1 by 9.1 m) for overburden and 22 by 25 ft (6.7 to 7.6 m) for interburden. A twin-masted auger drill is used for drilling all 4½-in. (114-mm) blastholes in coal. Coal patterns are drilled on a 12- by 15-ft (3.7- by 4.6-m) grid.

BLASTING. A mixture of ammonium nitrate and 6% fuel oil (ANFO) is used almost exclusively for overburden and interburden blasts. The gas-generating characteristics of ANFO provides better digging conditions relative to the more expensive, high-velocity slurry explosives. Bagged emulsion and bulk slurries are used occasionally when blastholes are wet. However, when only small quantities of water are encountered in the blastholes or when the shot will proceed in a short period of time, a plastic hole liner is used to sleeve the bulk ANFO. Blasting operations utilize the Detaline system with down-hole and surface delays for blast initiation and sequencing.

In general, all waste materials are shot. Powder factors average 0.7 lb/yd^3 (0.42 kg/m^3) for overburden buffer blasting, 1.0 lb/yd^3 (0.59 kg/m^3) for overburden cast blasting, and 0.6 lb/yd^3 (0.36 kg/m^3) for interburden materials. All three figures are based on weight of explosive per volume of material blasted. These powder factors provide optimum fragmentation to allow efficient digging with the 30-yd^3 (23-m^3) buckets. When designing cast blast shots, a breakeven cast benefit of 15% must be attained. The single-seam Enfield pit extensively used cast blasting with a resulting average of 22% of blasted material in final position.

ANFO was initially used for blasting the coal seams, but due to occasional water problems and fairly high labor requirements, higher costs for thin seams resulted. A packaged slurry explosive and a powder factor of 0.16 lb/ton (0.08 kg/t) to loosen the already highly cleated coal has provided lower costs and acceptable loading characteristics. The low powder factor presents no problem for high break-out force of the hydraulic backhoes used to load the coal.

OVERBURDEN REMOVAL. Surface mining (open cast or strip mining) is the only mining method used at the Trapper mine. Three Page 752 long-range draglines remove overburden to expose the coal. Arched 30-yd^3 (23-m^3) buckets are utilized in areas, of dense and poorly fractured overburden and where chopping is required. This bucket is well suited for these purposes, due to its heavier construction. Archless buckets with a capacity of 32 yd^3 (24.5 m^3) have been used with good productivity in overburden which is lighter and well fractured. High-density plastic bucket liners minimize the quantity of overburden material carried back between swings. The Page 752LR has the ability to negotiate 8% uphill and 10% or greater downhill grades, which is essential for the steep slopes at the Trapper mine.

Dragline productivities and utilization have been high at Trapper due to aggressive training and safety programs. Another important reason for the high production and cost efficiency of Trapper's draglines is the use of computerized monitoring and control systems installed on each machine. These systems have allowed for automated lubrication pumping, tightline prevention, and instantaneous operation feedback on data related to motor and bearing temperatures, tightline warnings, and productivity parameters, such as load counts, digging rates, and cycle times. The goal of these systems is to assist the operator to improve performance and reduce costs through better preventive maintenance. Radio links are used to transmit data from each dragline to the engineering department to facilitate reporting daily statistics and tracking overburden volumes and delay times.

Scrapers and dozers remove thin horizons of separable waste material between seam splits. Dozers typically rip these parting layers and assist by push-loading the scrapers. Scrapers are desirable because they allow greater recovery of the top 1 to 2 in. (25 to 50 mm) of the coal seam instead of dozers that tend to break the coal surface. In addition, Trapper has recently begun its first small scraper pit in an area of low cover to supplement coal production from the three dragline pits.

COAL LOADING. Coal loading operations are performed by three Demag H-111 hydraulic backhoes with 15-yd^3 (11.5-m^3) buckets. One backhoe is generally stationed at each pit and loads from atop the coal seam. The crawler-mounted backhoes were selected over rubber-tired loaders because of the backhoe's excellent performance in a pitching environment and possible wet floor conditions. Loading is performed from 25-ft (7.6-m) wide slots directed down dip to provide coal trucks a path following the dip of the coal.

COAL HAULAGE. Coal from the mine is delivered under a long-term fuel supply contract to a steam generating utility located adjacent to the mine. Typically, six trucks from a fleet of nine 50-ton (45-t) Euclid end-dump coal haulers are employed to haul the coal an average of 2.5 miles (4 km) one way to a crusher that feeds the power plant. Since the crusher is located at the bottom of the hill, the haul profile is favorable for loaded trucks. Coal is dumped onto an adjacent stockpile if the crusher is down or surge capacity is reached. A Cat 988B front-end loader trams the stockpiled coal to the crusher when required.

As mining operations begin to advance away from the crusher, analysis of alternative coal haulage methods will ensue. Two options Trapper will investigate are (1) upgrading its 50-ton (45-t) haulage fleet to 85-ton (77-t) or greater capacity and (2) utilizing an overland conveyor to directly feed the plant crusher.

RECLAMATION. After all coal is mined, overburden from the next adjacent pit is spoiled by the dragline into the just completed cut. After the overburden is graded and contoured by Cat D9L and D9N dozers and Cat 16G graders, the scraper fleet redeposits the topsoil; the area is seeded in the fall of the year. A mixture of grasses, forbs, and shrubs is planted to assure proper forage for wildlife and domestic livestock. Wheat or an alfalfa grass mix is used on reclaimed croplands found on the lower slopes.

Where possible, surface water above the mining area is diverted around the mine pits and directed into settling basins to control suspended solids in water being discharged from the mine area. Groundwater encountered in the mine pits is pumped out and allowed to continue through the surface drainage system. Due to Trapper's pitching topography, drainage maintenance and reconstruction are important aspects of reclamation activities.

In 1981, 1987, and 1989, Trapper received state reclamation awards for outstanding achievement. These awards exhibit that a mine can be a profitable venture while maintaining the necessary balance between resource development and environmental integrity.

14.3.4.5 Product Control

Daily coal samples are taken from each seam being mined to determine in situ quality. An onsite coal laboratory runs a modified short proximate analysis for tracking current seam qualities and forecasting future seam quality trends.

Samples also are captured from the crusher stream for analysis by the customer to determine the "payable" quality of coal delivered to the power plant. Trapper is paid on the basis of total heat content delivered.

The seams present at Trapper usually need no blending to achieve the minimum daily requirement of 9500 Btu/lb (22,100 kJ/kg). However, when blending is required, it is performed by alternating loads of high- and low-Btu coal from different seams. No stockpile blending is required.

14.3.4.6 Ancillary Facilities

Trapper's mine maintenance system (MMS) has progressed significantly since its first formal program in 1978. Since that time, the dedication of mine management and maintenance personnel to improve maintenance performance has resulted in achieving significant improvements in equipment availabilities, minimum staffing levels, and machine component lives.

The priority of the MMS focuses keenly on maintenance planning and control systems. The program contains the following major elements:

1. Work Order System—captures repairs needed, repairs completed, employee-hours consumed, and parts used.

2. Repair and Cost History—kept on all major equipment units and components in order to reveal long-term repair problems requiring special attention.

3. Preventive Maintenance (PM)—strictly scheduled routine maintenance.

- Oil Analysis—done in conjunction with the PM program.
- Component Failure Analysis—identification of the causes for component failures so remedies can be implemented.
- Labor Utilization Reporting—tracks labor performance to spot inefficiencies in maintenance methods.
- Mechanic Training and Safety Incentives.
- Maintenance Planning—daily and weekly plans to coordinate maintenance objectives and parts inventories.

The maintenance facilities are located near the main haul road to the power plant and adjacent to the mine administration building. The maintenance complex includes eight large equipment bays, two bucket bays, wash and lube bay, welding shop, light duty shop, and electrical shop.

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14.3.5 APPALACHIAN CONTOUR MINING, HORIZONTAL COAL SEAMS: MARTIN COUNTY COAL CORPORATION

IRA J. TUSSEY

14.3.5.1 Mine Description

Martin County Coal Corporation's mine is located in the Appalachian coalfields, Martin County, KY, near the West Virginia border. The leasehold extends from the southern portion of Martin County into northern Pike and Floyd Counties (Fig. 14.3.5.1). The property is bordered by Brushy Fork of John's Creek to the South and Middle Fork of Rockcastle Creek to the north. Wolf Creek, the other major watershed within the leasehold, bisects it. The total leasehold covers more than 33,000 acres (134 km^2).

Total annual production at Martin County Coal Corp. is currently 2.5 million tons (2.3 Mt) but has exceeded 3 million tons (2.7 Mt). Approximately 45% of the annual production comes from two surface mines, one of which is dedicated wholly to mountaintop removal mining. The remaining balance is produced from five underground mines. Martin County Coal Corporation's work schedule consists of 240 days/year, two 9-hour shifts/day.

The topography of the area is characterized by rugged hills and ridges and sharp V-shaped valleys demonstrating a cliff and slope topography. This topography consists of alternating benches of outcrop sandstone and slopes. The deep valleys have been carved by the actions of many small streams in the area. These streams are contained within the steep slopes of narrow, twisted mountain ridges. The ridges are underlain by massive sandstones, causing them to be sharp and rocky. The height of the hills above the valley floors ranges from 400 to 800 ft (120 to 240 m).

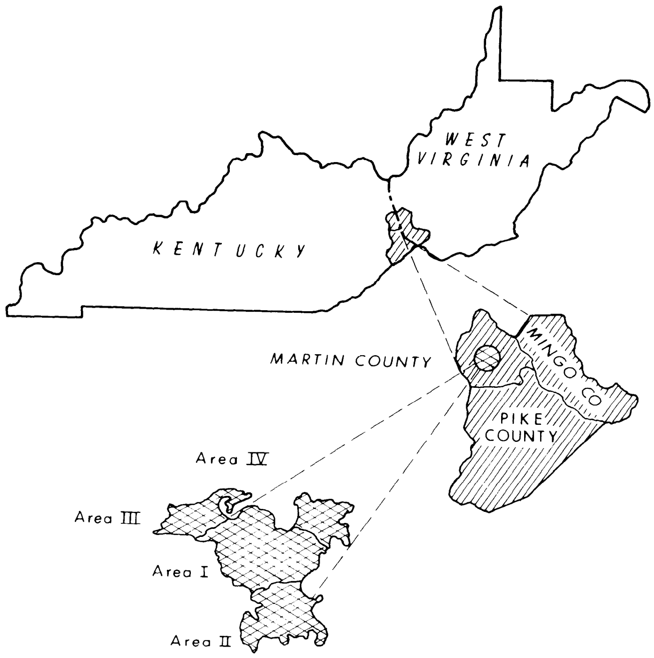

Fig. 14.3.5.1. Regional location map.

14.3.5.2 Deposit Description and Geology

GENERAL GEOLOGY AND STRATIGRAPHY. The stratigraphic sequence of the property consists principally of sandstones, siltstones, shales, coals, and underclays of the Upper Breathitt Formation of the Pennsylvanian Age and demonstrates a pattern of cyclic deposition. Approximately 60% of the strata overlying the coals is sandstone. The marine-fossil Magoffin Shale Bed is present throughout the leasehold at an elevation of 800 to 850 ft (240 to 260 m) and ranges in thickness from a few inches (tens of millimeters) to several feet (meters). Its unique lithologic characteristics and areal consistency have made it an excellent marker bed for coal-seam correlation. Structural dip across the property is generally to the northwest at a magnitude of 0.75 to 1.5%; however, local flexures have modified this somewhat in certain areas (Fig. 14.3.5.2)

COAL SEAMS OF IMPORTANCE. The major coal seams presently being mined by Martin County Coal Corp. are in the following descending stratigraphic order: No. 5-Block, Clarion, Lower Clarion, Stockton, Coalburg Rider, and Coalburg. All of these seams are high-quality bituminous steam coals. Other coal seams occurring below the Coalburg are of lesser economic consequence and are not being mined presently. These include the Little Coalburg, Winifrede/Haddix, and Taylor (Fig. 14.3.5.3).

No. 5-Block—The No. 5-Block seam is present only near the crests of the higher knobs on the property and ranges in thickness from 6 to 11 ft (1.8 to 3.4 m). This seam is mined exclusively by the mountaintop removal method. Because of its proximity to the crests of the mountains, its aerial distribution and recoverable reserves are restricted. The No. 5-Block consists of many benches of varying thickness and quality and contains numerous partings.

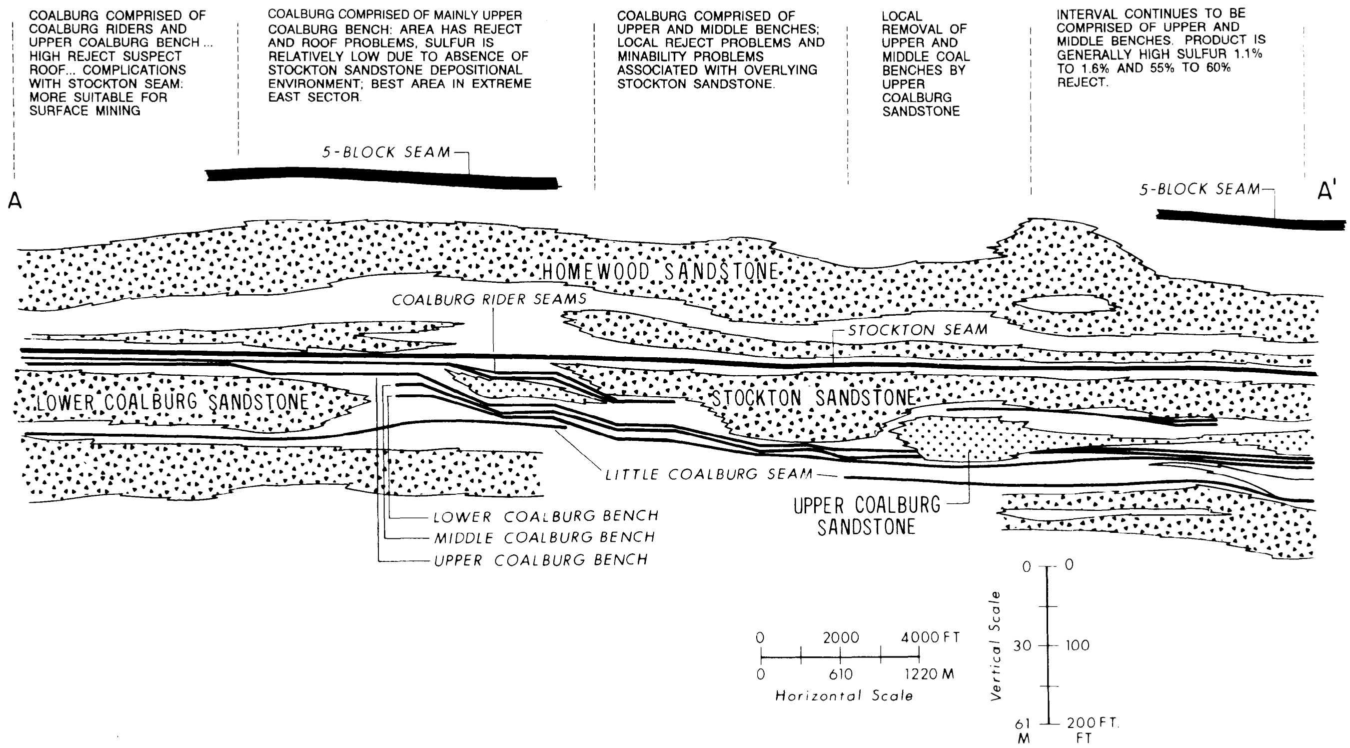

Fig. 14.3.5.2. Geologic cross section.

Clarion—The Clarion seam underlies the No. 5-Block by a depth of 10 to 40 ft (3 to 12 m) across the property and ranges in thickness from 6 to 60 in. (152 to 1524 mm). The seam is a relatively clean coal with few or no partings. Like the No. 5-Block, the Clarion seam lends itself exclusively to mountaintop removal.

Lower Clarion—The Lower Clarion occurs only locally in certain areas and is surface mined in conjunction with the No. 5-Block and Clarion. It is also a relatively clean coal with few impurities.

Stockton—The Stockton seam underlies the Clarion seam by a depth ranging from 180 to 210 ft (55 to 64 m). Its thickness ranges from 30 to 86 in. (762 to 2184 mm) across the property, but is typically 50 to 60 in. (1270 to 1524 mm). This coal is mined by a combination of mountaintop removal, contouring, and underground mining. The seam displays excellent consistency, uniformity, and quality with few associated geological complexities. It consists of a single bench with a small bone parting in the upper portion of the seam.

Coalburg Rider—The Coalburg Rider seam lies from 2 to 40 ft (0.6 to 12 m) above the Coalburg and is mined in conjunction with surface mining of the Coalburg seam. It is typically less than 24 in. (610 mm) in thickness and is a high-grade coal with few impurities.

Coalburg—The Coalburg seam underlies the Stockton by 20 to 130 ft (6 to 40 m) across the property and is stratigraphically the most complex seam of the leasehold. It consists of three individual, distinct benches or splits: the Upper Coalburg, the Middle Coalburg, and the Lower Coalburg. The Coalburg seam displays a high degree of lateral stratigraphic change across the property and has been completely replaced with sandstone in a number of areas. Total seam height ranges from 20 in. (508 mm) to several feet (meters). However, the seam contains numerous partings of fireclay and shale with abundant plant material. All three splits are mined in one or more forms by both surface and underground mining methods (Fig. 14.3.5.4).

Little Coalburg—Approximately 20 to 80 ft (6 to 24 m) below the Coalburg lies the Little Coalburg seam. The Little Coalburg is generally thin throughout the property and is not presently considered a potentially minable reserve.

Winifrede/Haddix— The Winifrede/Haddix lies approximately 90 ft (27 m) below the Little Coalburg. It is also a relatively thin seam. However, potential deep mining reserves exist for this seam in localized areas.

Taylor— The Taylor seam lies directly below the Magoffin Shale Bed. It, too, is a thin coal with only limited reserves.

Total Reserves—A recent reserve study revealed that Martin County Coal Corporation's total reserve base is approximately 100 million tons (90 Mt) of clean recoverable coal, extending the life of mining capabilities well into the next century based upon current production levels.

14.3.5.3 Mine Development

General Description of Contour Surface Mining. Contour surface mining is a technique of mining where the coal is extracted by removing the soil and rock (overburden or spoil) immediately overlying the coal with bulldozers, trucks, shovels, and other earthmoving equipment. The mined location is then reclaimed to its approximate original contour (AOC) by depositing overburden from the next successive cut in the area and then completing by grading and seeding. This procedure is followed along the outcrop of the coal seam as successive cuts are taken. Martin County Coal's Lee Construction Co. surface mine operation is presently engaged in contour surface mining (Fig. 14.3.5.5). Monthly production at this surface mine is in excess of 40,000 clean tons (36,000 t), and drilled yardage exceeds 400,000 yd^3 (306,000 m^3) per month. The employees of this mine

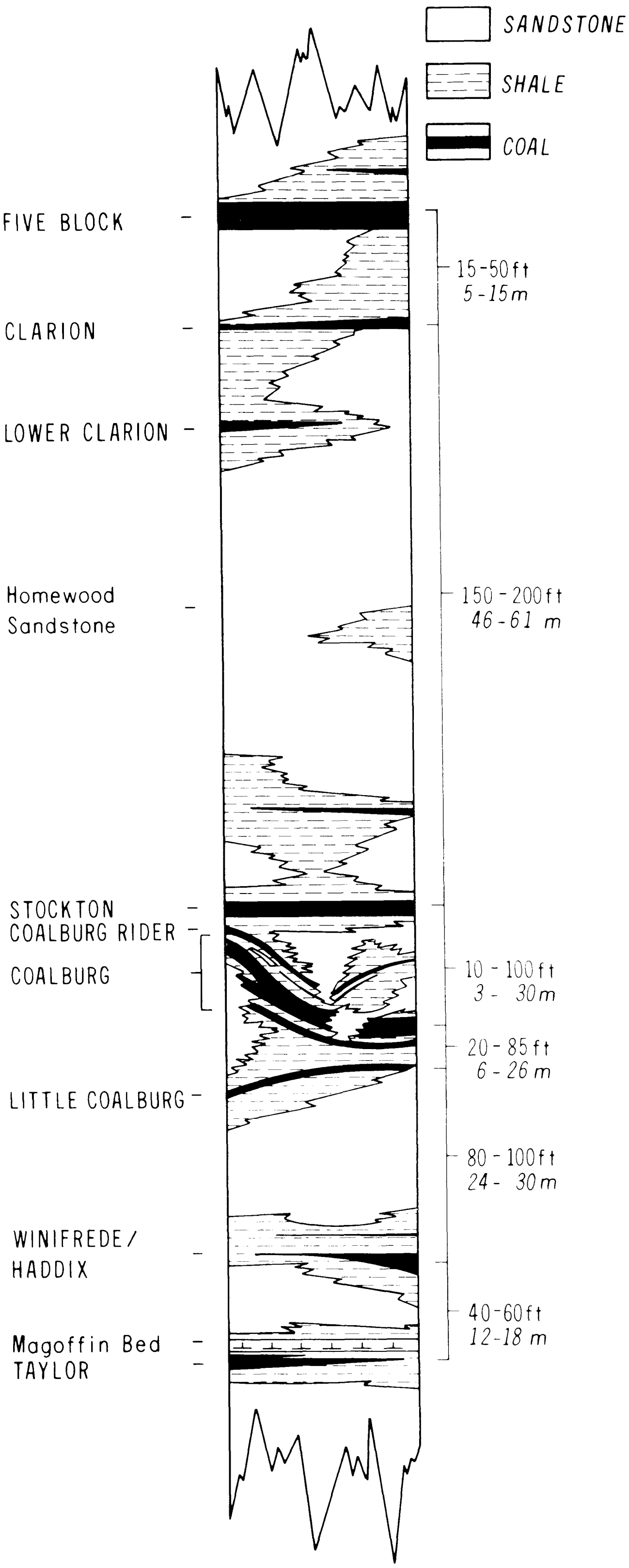

Fig. 14.3.5.3. Generalized stratigraphic column.

have won several safety awards from the National Safety Council and the Holmes Safety Association.

MINE PLANNING. Prior to mining activities, a mine plan is devised in which projections for surface and underground mines are developed with consideration to the safety and optimum efficiency of each operation, least interference with each other, and maximum resource recovery at the lowest possible cost. Topographic maps are prepared by Martin County Coal Corp. engineering personnel showing coal seam outcrops, proposed highwalls, coal pits, hollowfills, roads, sedimentation structures, and underground portal areas. These projections are then reviewed with production personnel and revisions and adjustments are made if indicated. After this mine plan is developed, a surface disturbance permit is obtained from Kentucky's Department for Surface Mining Reclamation and Enforcement (DSMRE). The permit application is prepared by an outside consulting firm. The designs contained within the application are reviewed and approved by Martin County Coal Corporation's Engineering Department prior to submission of the application.

STRIPPING RATIO. Martin County Coal's contour operations attempt to maintain a surface mining ratio of 10:1 yd^3/clean ton (8.4:1 m^3/t). Variances may occur in some local areas. The height of the highwall is dictated by the thickness of the coal seam, topography, and equipment size.

POND CONSTRUCTION. Mine development is initiated with the construction of sedimentation ponds in accordance with designs approved by DSMRE. The purpose of these structures is to trap any runoff from the area disturbed by mining activities and to maintain water quality. Since the structures remain intact during mining and reclamation activities, proper construction is crucial. These sedimentation ponds are constructed in smaller streams near their confluence with the major streams. After all trees and other organic material are cut, grubbed, and removed from the site, bulldozers push earth from the valley slopes into the streams and compact the material in lifts to the elevation of the principal spillway. The principal spillway, a corrugated metal pipe 36 to 60 in. (914 to 1524 mm) in diameter, is placed at the proper elevation. Additional earth is then placed on the dam and compacted to the proper elevation. The width at the top of the dam is a minimum of 12 ft (3.7 m), and side slopes are 2.5:1.0 horizontal to vertical configuration. The emergency spillway is then constructed by cutting into solid rock on one side of the pond with a bulldozer and drilling the rock with a pneumatic drill. This rock is blasted, with the shot rock used to rip-rap the slope below the principal spillway.

ACCESS ROAD CONSTRUCTION. After the sedimentation pond is completed, road construction to the area begins. These roads are constructed with the intention of retaining them as permanent facilities after release of the reclamation bond. With this concept in mind, the roads are constructed on as moderate a gradient as possible and are composed of high-grade, shot-rock that does not weather easily. The design and construction of these roads also are approved by DSMRE. The roads intersect an existing haulage road and are constructed to a head-of-hollow fill where mining begins. Road width averages 100 ft (30 m).

14.3.5.4 Mining Operations

Mining commences by cutting the timber overlying the proposed cut on the Coalburg coal seam. This timber is dragged by bulldozers to the outcrop of the coal seam. A "catch basin" or ditch is simultaneously constructed to prevent any rock or spoil from falling or rolling below the outcrop line. Timber in the head-of-hollow fill is also cut, piled, and burned, or dragged to the perimeter of the planned head of hollow fill. Because of the steep slopes and timbering activities at the turn of the century, little or no topsoil is present on the property. Consequently,

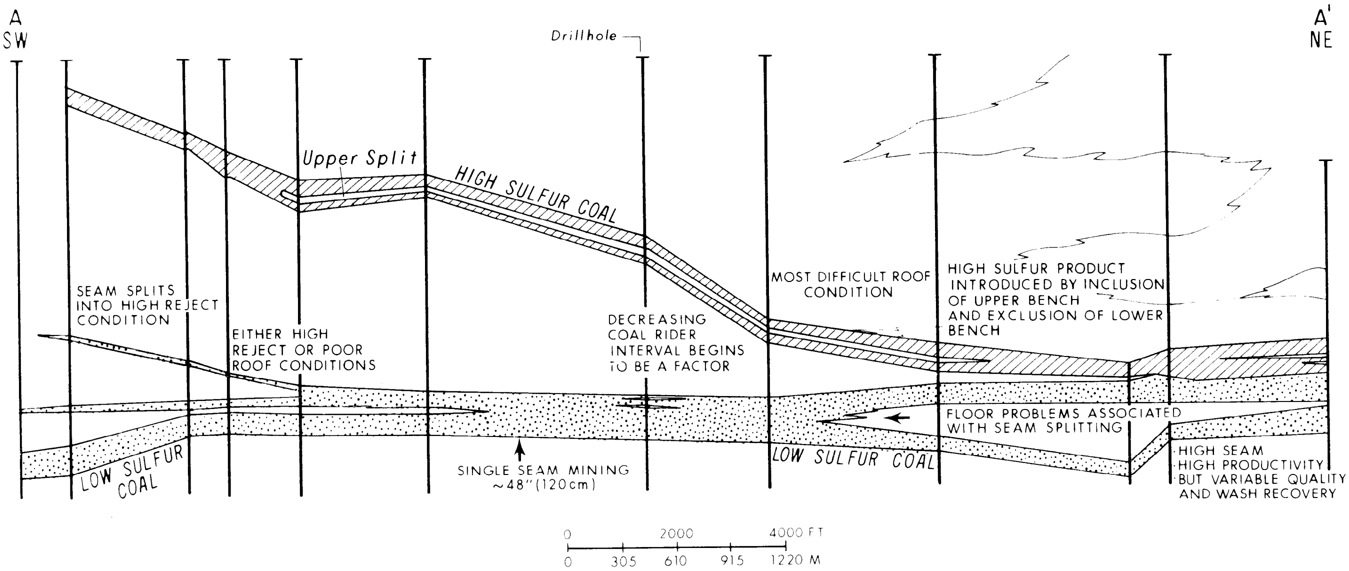

Fig. 14.3.5.4. Coalburg seam showing lateral physical and quality changes.

Fig. 14.3.5.5. Typical mining area.

removal and subsequent redistribution of topsoil is not required. Alternate topsoil material exists in abundance and is located in close proximity to the coal seam, generally 1 to 5 ft (0.3 to 1.5 m) in depth. This material consists of sandstones and shales and is usually the last stratum to be encountered and removed. As a result, it is the last material to be used in backfilling operations and is distributed in quantities necessary to promote plant growth. Fertilizers and other soil nutrients are applied as needed to the alternate topsoil material to enhance the chemical and physical properties and to promote growth of grasses, legumes, and trees specified by DSMRE.

OVERBURDEN REMOVAL. The Stockton outcrop is utilized as a marker for constructing the drill bench for the Coalburg highwall. A bulldozer prepares the drill bench by following this outcrop. After the drill bench is established, a crawler-mounted rotary drill begins drilling presplit holes. These holes are 40 to 44 ft (12.2 to 13.4 m) deep, 9 in. (229 mm) in diameter, and are drilled on 18- to 20-ft (5.5- to 6.0-m) centers. Presplitting is accomplished by utilizing approximately 5 ft (1.5 m) of ANFO (ammonium nitrate and fuel oil, S.G. of 0.85), 2-lb (0.9-kg) cast primers, and non-electric caps and lead line. Hill seams and other geological discontinuities are localized and have been found to follow no definite trend or pattern. Consequently, presplitting the highwall is necessary to minimize potential adverse effects on safety and productivity. All presplitting is accomplished vertically. This results in a vertical or near vertical highwall angle. Attempts made to bench the highwall at underground "face-up" areas have resulted in fracturing, backbreak, and large rocks being disjointed above the coal seam. As a result, benching the wall is not attempted, and a cleaner, smoother, and safer wall is obtained. Highwall height and subsequent pit widths vary and are dictated by seam thickness, existing topography, equipment size, and other factors. Highwall height ranges between 80 to 100 ft (24 to 30.5 m) and pit width between 160 to 180 ft (49 to 61 m).

After the presplit shot is detonated, the overburden is drilled with a crawler-mounted rotary drill. These drillholes are 9 in. (229 mm) in diameter and 21 ft (6.4 m) in depth. Burden and spacing range from 15 to 18 ft (4.6 to 5.5 m). The drillholes are loaded with approximately 12 ft (3.7 m) of ANFO and are detonated using 2-lb (0.9-kg) cast primers and non-electric blasting caps. ANFO is delivered in bulk to the mine site where it is stored in 40-ton (36.3-t) storage bins.

After the overburden shot is detonated, bulldozers push the spoil into the head-of-hollow fill and simultaneously construct another drill pad. The crawler-mounted drill is again moved back to the site and drilling is resumed, this time to a depth of 20 ft (6.4 m). This shot is drilled on an 18- by 18-ft (5.5- by 5.5-m) pattern. Bulldozers move a large portion of this shot into the head of hollow fill. They are assisted by rubber-tired loaders with a bucket capacity of 13.5-yd^3 (10.3-m^3) and 85-ton (0.91-t) trucks that haul the remaining spoil into the head-of-hollow fill.

Fig. 14.3.5.6. Beginning of final highwall grading operations.

Fig. 14.3.5.7. Final highwall slope configuration.

The highwall is "scaled" or cleaned of any small rocks by the loader at this time.

After this spoil is removed, the crawler-mounted drill again prepares another presplit shot to the coal. Following detonation of this shot, drilling the remaining overburden resumes with drillholes going down to the coal seam. A powder factor of 1.10 lb/yd^3 (0.59 kg/m^3) of overburden is maintained on production blasts. This powder factor is economical and produces excellent fragmentation of the overburden. After detonating this shot, the loader and trucks move the blasted material to the head-of-hollow fill. Because most of the sandstones and shales do not readily degrade, durable rock fills are constructed by dumping the spoil over the edge of the hollow-fill in a single lift. This results in an excellent underdrain system composed of large boulders that is constructed simultaneously with excess spoil placement.

COAL REMOVAL. After the spoil is completely removed, the coal is cleaned and piled with small rubber-tired loaders 7.0 yd^3 (5.4 m^3). Because the coal is friable, blasting is not required. Dozers rip or break the coal, when needed, to facilitate loading. Any "binders" (in-seam impurities) greater than 4 in. (100 mm) are wasted and not loaded.

BACKFILLING. After the coal is removed, backfilling operations begin. The pit is immediately backfilled with overburden from the next adjacent pit to be mined. This spoil material is moved into the pit by a combination of bulldozers, loaders, and trucks. As the equipment removes the overburden from the coal, it is placed into the previous pit, thereby restoring the mined land to AOC. Once a sufficient quantity of spoil is placed in the pit to eliminate the highwall, bulldozers begin final grading operations (Fig. 14.3.5.6). The spoil is pushed to the top of the wall and "walked in" to the toe of the backfill. A road is retained at the bottom of the backfill to provide access to the mine. This activity is repeated as the operation moves along the periphery of the coal seam. The final slope configuration is approximately 30° (Fig. 14.3.5.7).

Because overburden removal and backfilling operations follow a sequential order, it is difficult to maintain more than one pit in either coal seam. At times, it is possible to access the coal from two directions. When this situation exists, two or more pits may be open at one time.

A calculated spoil balance is prepared to determine the volumetric requirements of the head-of-hollow fills. Hollow-fills are sized as small as possible to minimize disturbed acreage and contain only the volume of rock originated by the swell of the blasted rock and the volume contained in retention of "face-up" areas for underground mines. The "face-ups" are constructed whenever possible near the head-of-hollow fills so as to take advantage of the additional area for coal stockpiles, mine management areas, etc.

Because current regulations require elimination of the highwall, contour mining must end on a point of a ridge to avoid rehandle of spoil. A higher cut on the point is taken, and the excess spoil is used to eliminate the wall from the previous cut. By ending mining operations on a point, rehandle of spoil material is eliminated, and the last remaining highwall is reclaimed as it is mined. After the Coalburg seam is mined, the overlying Stockton seam is mined in the same manner. Mining is initiated in a head-of-hollow fill and progresses along the coal seam outcrop. Underground "face ups" are retained where projections indicate. Contouring the Stockton also ends on a point.

Exceptions to this sequence occur when the No. 5-Block and Clarion coals are present in sufficient quantities to "spoil over" on the Stockton bench. The Coalburg overburden is placed into the head-of-hollow fill. The Coalburg highwall is eliminated by covering it with Stockton overburden. The Stockton highwall is then eliminated by covering it with overburden from the No, 5-Block and Clarion coals, as they are mined by mountaintop removal. On other occasions the topography will lend the Coalburg and Stockton coals to point removal. When this situation exists, the spoil overlying these seams is placed in head-of-hollow fills, and highwall elimination is not required. This method leaves a long, narrow, flat area on a ridge that previously was rocky and steep (Fig. 14.3.5.8).

These two methods are safer and more economical. They have the added advantage of leaving large flat areas on what previously was mountainous terrain. However, they are not practiced a great deal due to the No. 5-Block and Clarion seams having been eroded or the topography not lending itself to point removal.

AUGER MINING. Auger mining (Chapter 14.4) follows contouring in a few isolated cases but, in general, is not an integral part of the mining operations. Augering is performed only when mountaintop removal cannot be performed or where underground mining cannot be accomplished because of inaccessibility. This inaccessibility may be caused by narrowness of the ridges, property lines, gas well barriers, or previous mining operations.

RECLAMATION. Reclamation is an ongoing operation and begins in the strictest sense when the pit is backfilled with spoil

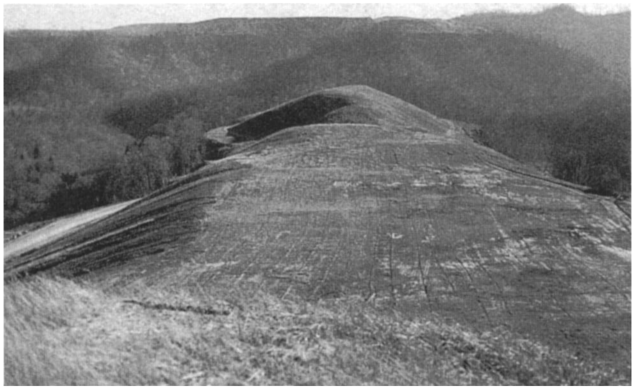

Fig. 14.3.5.8. Reclaimed mountaintop removal area.

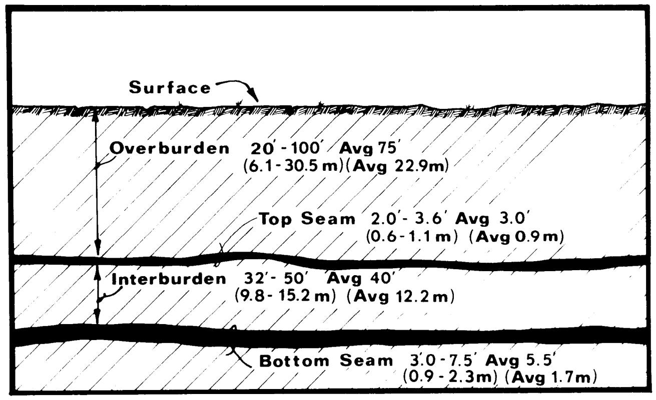

Fig. 14.3.6.1. Typical cross section B area.

from the next adjacent pit. This procedure is continued until the area is reclaimed to its approximate original contour. Bulldozers then begin to prepare slopes to final grade. Following final grade work, the slopes are sprayed by a hydroseeder with a mixture of grasses, legumes, fertilizers, and mulch. Seedlings of approved tree species are then planted. Reseeding or spot seeding is accomplished on an "as needed" basis.

The head-of-hollow fills are reclaimed as soon as practicable after they have reached their designed capacity. The outslope of the hollow-fill is graded to 2:1 (horizontal to vertical). Benches or terraces. are then cut by bulldozers after the face of the fill is graded. These terraces are graded to a 3% slope toward the fill and do not exceed 50 ft (15 m) vertically. Surface runoff is carried across these terraces to designed drainage channels in natural ground along the periphery of the fill. The top of the fill is graded to 5% toward these drainage channels.

14.3.5.5 Product Control

Martin County Coal Corp. maintains close supervision over the quality of coal in the pit. Previous core drilling (usually on 1/4-mile or 400-m spacing) provides a good indication of the coal quality. These holes, however, are usually drilled in the center of the ridges and are sometimes far removed from the surface mine pits. Regional trends relating to coal quality can be accurately predicted. Additionally, in order to maintain close supervision of the coal quality, a number of samples are taken from each pit prior to loading. If indicated, blending is accomplished by combining coal from the underground mines and from the other surface mine.

14.3.5.6 Ancillary Facilities

A four-bay shop area is maintained adjacent to the mine. It is equipped for major repair of trucks and other mobile equipment. Field crews service drills and other large equipment that cannot be easily moved to the shop area.

14.3.6 LIGNITE: BIG BROWN MINE

GENE RAND

14.3.6.1 Mine Description

The Big Brown mine is a mine-mouth operation owned and operated by Texas Utilities Mining Co. The mine produces 6 million tons (5.4 Mt) of lignite annually as fuel for the two nearby 575-MW units of the Big Brown Steam Electric Station owned by TU Electric. The mine is located 90 miles (144 km) south of Dallas, TX, in Freestone County near the town of Fairfield. Initial lignite production began in 1971. The permitted mining area covers some 15,000 acres (60.7 km^2). The mine employs 410 and operates 24 hours/day, 7 days/week, 365 days/year.

14.3.6.2 Deposit Description and Geology

The lignite at the Big Brown mine is recovered from two minable seams within the Calvert Bluff Formation, which is the uppermost member of the Wilcox Group, Early Eocene. The reserves trend southwest to northeast and dip to the southeast at approximately 1°. The overburden and interburden material consists of unconsolidated fine-grained sand and laminated silts and clays that do not require drilling or blasting prior to removal. The sand in the overburden locally contains groundwater that must be dewatered in advance of stripping operations.

Lignite is recovered from two mining areas located about 1 mile (1.6 km) apart. The southernmost area, the B area, generally contains two minable seams, as shown in Fig. 14.3.6.1.

Stripping ratios of 9.5 to 14.3 yd^3/ton (8.0 to 12.1 m^3/t) exist over the remaining 15 years of life in this area. Approximately 28 million tons (25.5 Mt) of lignite will be recovered during this period.

The northernmost mining area, the C area, is underlain by two lignite seams in the southern two-thirds of the deposit and a single seam in the northern third of the area. The general profile of this area is shown in Fig. 14.3.6.2.

The C area contains approximately 90 million tons (82 Mt) of recoverable lignite that will be removed over the next 15 years at ratios of 6.9 to 12.1 yd^3/ton (2.7 to 4.0 m^3/t).

The average lignite quality for the total deposit on an as-received basis is as follows:

Heating Value	6800 BTU/lb (15,817 kJ/kg)
Moisture	31.5%
Ash	15.3%
Sulfur	0.76%

14.3.6.3 Mine Development

Initial mining operations began with a box cut along the sub-crop at the extreme southwestern edge of the deposit and

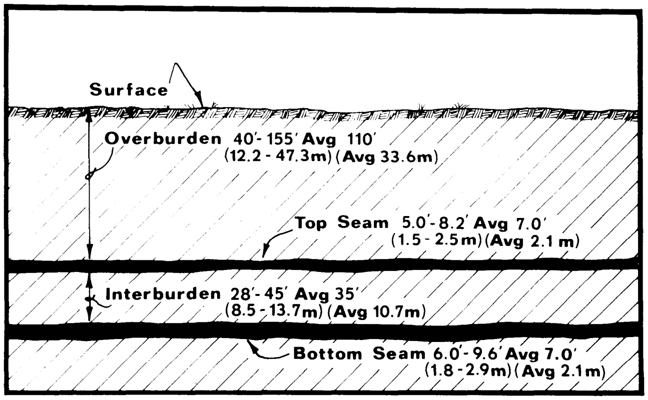

Fig. 14.3.6.2. Typical cross section C area.

progressed down-dip. As the overburden became deeper, sand channels containing groundwater were encountered, causing highwall and spoil instability. Since 1981, dewatering wells have been installed to remove most of the groundwater in advance of stripping operations. Most of the wells required are in the C area.

Surface water run-off from haul roads and other disturbed areas is contained and processed by a specially designed network of sedimentation ponds prior to discharge into the natural drainage.

In the B area, all overburden and interburden materials are removed with a dragline working in a pit 10,500 ft (3200 m) in length. Haulage ramps enter the pit at each end and in the middle from the highwall side.

The C area contains one pit 27,000 ft (8235 m) long in which two draglines and a cross-pit spreader are used to remove overburden and interburden. Haulage ramps in this pit are maintained at each end and at approximately 4000-ft (1220-m) intervals along the highwall.

In both mining areas, highwall angles are cut at 70° while spoil angles average 32°. Pit widths are maintained at 120 ft (36.6 m). Slopes on haulage ramps are held to a maximum of 8°. From time-to-time, 637D scrapers are used to remove overburden and interburden material when additional capacity is required.

14.3.6.4 Unit Operations

OVERBURDEN REMOVAL. Stripping operations in the B area utilize a Bucyrus-Erie 1500-W dragline with a 70-yd^3 (54-m^3) bucket. The dragline uncovers the upper lignite seam by conventional side-casting from the highwall.

The interburden is removed by the dragline working from the spoil side of the pit. A range diagram of this operation is shown in Fig. 14.3.6.3.

This dragline moves 18 million bank yd^3 (13.8 million bank m^3) of material per year. Rehandle averages 30% while availability averages 80%. A Caterpillar (Cat) D9N dozer is used around the dragline to prepare the walkway, move the power cable, and clean the top of the lignite seam prior to loading.

The C area pit utilizes a cross-pit spreader (XPS) system designed by Mannesmann DeMag of what was formerly West Germany to remove up to 100 ft (30 m) of overburden. The remaining overburden and interburden is removed by a Bucyrus-Erie 1500W dragline with 70-yd^3 (54-m^3) bucket and a Bucyrus-Erie 1350W dragline with a 60-yd^3 (45-m^3) bucket.

The XPS is a unique pre-stripping system that is designed to assist in the recovery of deeper lignite seams. The XPS system is designed to remove 24.5 million bank yd^3 (18.7 million bank m^3) per year of generally unstable unconsolidated overburden. The 656-ft (200-m) discharge boom transports the material across the pit and deposits it behind the second dragline spoil pile. This system creates a bench for a dragline to remove the remainder of the overburden above the top lignite seam from the highwall side of the pit. After the upper lignite seam has been loaded, the interburden is then removed by a dragline sitting on the lower bench. An overview of the XPS system and dragline combination is shown on Fig. 14.3.6.4.

The XPS system, commissioned in December 1985, consists of two bucket wheel excavators (BWEs), each rated at 2000 bank yd^3/hr (1530 bank m^3/hr), discharging onto a mobile across-the-pit conveyor (Fig. 14.3.6.5). Overburden material travels nearly 1000 ft (305 m) from the digging face to the spoilpile. The design availability of the XPS is 80%. The 1500-W dragline moves 18 million bank yd^3/yr (13.8 million bank m^3/yr), while the 1350-W dragline moves 16 million bank yd^3/yr (12.2 million bank m^3/yr). Both draglines operate at 80% availability. Overall productivity for the stripping operations at the Big Brown mine is 560 yd^3 (428 m^3)/employee-shift.

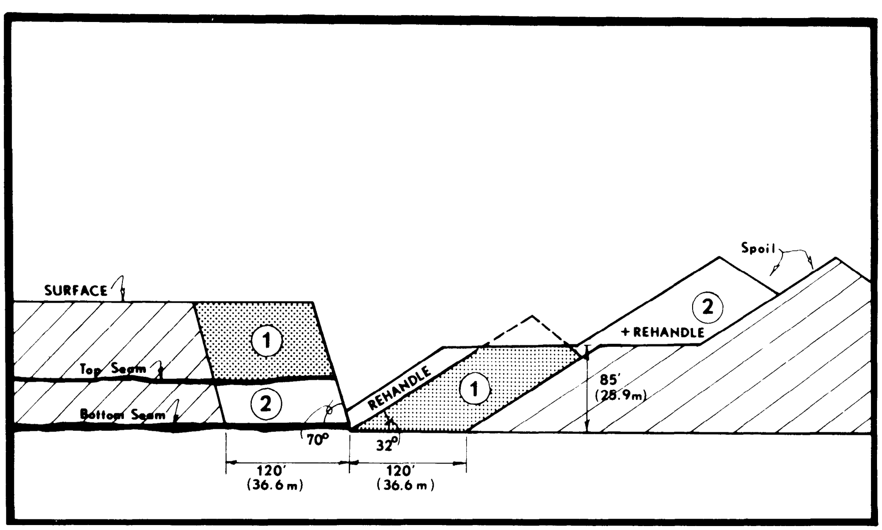

Fig. 14.3.6.3. Range diagram B area.

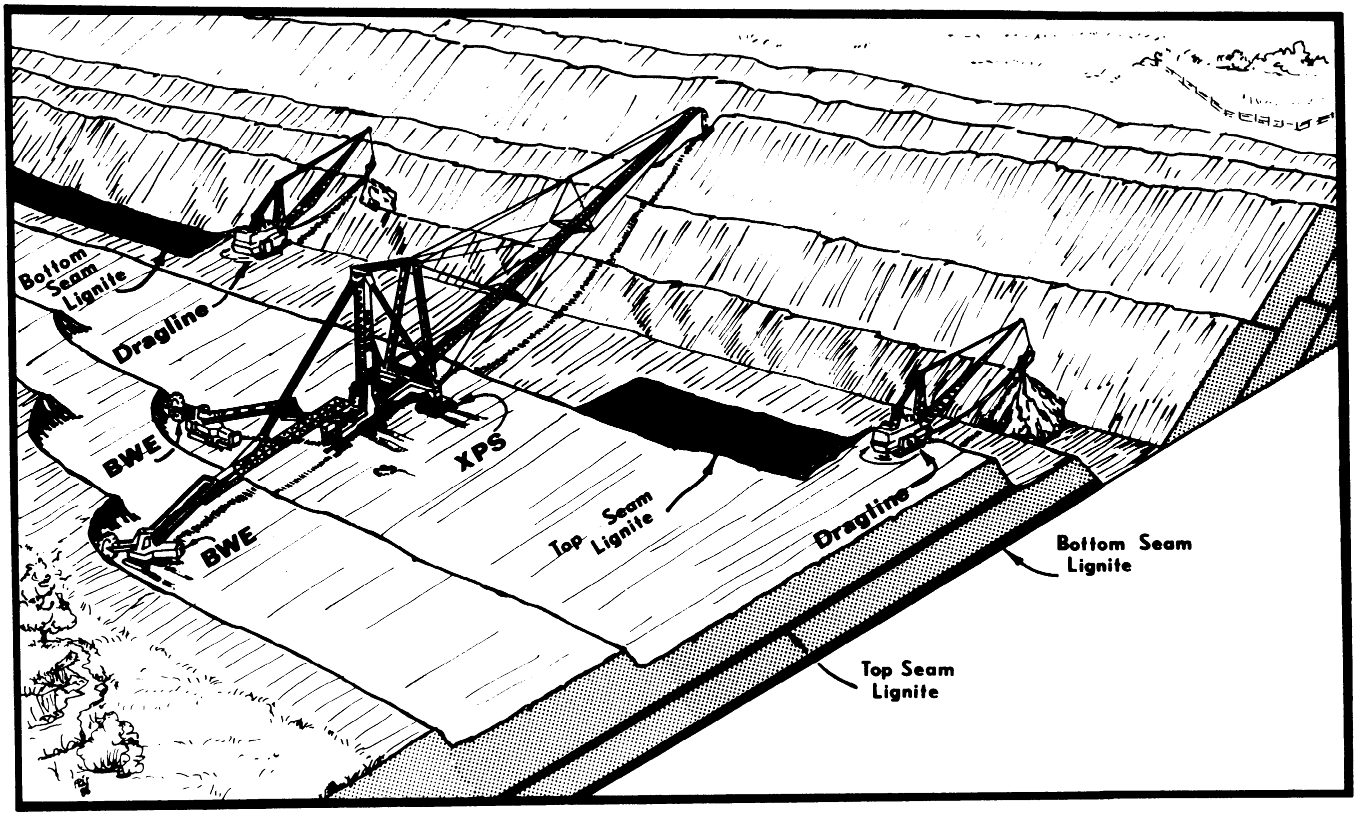

Fig. 14.3.6.4. Cross-pit spreader and dragline combination C area.

Fig. 14.3.6.5. Cross-pit spreader and BWE combination.

LIGNITE LOADING. Lignite loading in the B area is accomplished by a Cat 992-C front-end loader (FEL) with a 16-yd^3 (12.2-m^3) bucket.

In the C area, lignite is loaded with a Hitachi EX 1800 backhoe with a 13.5 yd^3 (10.3 m^3) bucket. A Cat 992 is also used.

The backhoe and FELs operate at 92% availability. Overall productivity at the Big Brown mine is 46 tons (41.8 t)/employee-shift.

LIGNITE HAULAGE. Lignite is hauled from the pits using 11 Dart 100-ton (91-t) bottom-dump coal haulers. These trucks were originally purchased when the mine began operations in 1971 and have been rebuilt as necessary. Approximately 17.5 miles (28.2 km) of haul road are maintained with five Cat 14G motor-graders and four 8000-gal (30.3-m^3) water trucks. Haul road width is 50 ft (15.3 m) to allow safe passage of haulage trucks and service equipment. Water used for dust suppression on haul roads is taken from holding ponds used for surface water control.

LIGNITE PROCESSING. Lignite is dumped from the haulage trucks into a 600-ton (545-t) double-compartment hopper. The lignite flows from the hopper through two reciprocating feeders into two McNally-Pittsburg 30- by 60-in. (762- by 1524-mm) double-roll crushers rated at 550 tph (500 t/h). The lignite is crushed to –6 in. (152 mm) and dropped onto a 60-in. (1524-mm) wide belt conveyor traveling at 520 ft/min (2.6 m/s) for transfer to the power plant stockpile.

RECLAMATION. Since the beginning of mining and reclamation activities in 1971, in excess of 8400 acres (34 km^2) have been reclaimed to a permanent cover of grasses, trees, and ponds. The mined area is leveled to the approximate original contour, as required by law, soon after the stripping process is completed. Three Cat D10N dozers are used to level the spoil areas. Planting of the various cover crops takes place according to a previously-approved reclamation plan.

Perennial grasses comprise approximately 75% of the reclaimed area. This is consistent with premine vegetation since the majority of the land-use is for livestock grazing. Coastal Bermuda grass is the principal grass species utilized for livestock forage and to maximize erosion control.

Ponds extend over 2% of the reclaimed area, which is consistent with premine conditions. The ponds facilitate pasture management and provide the required environment for certain shrub and tree species.

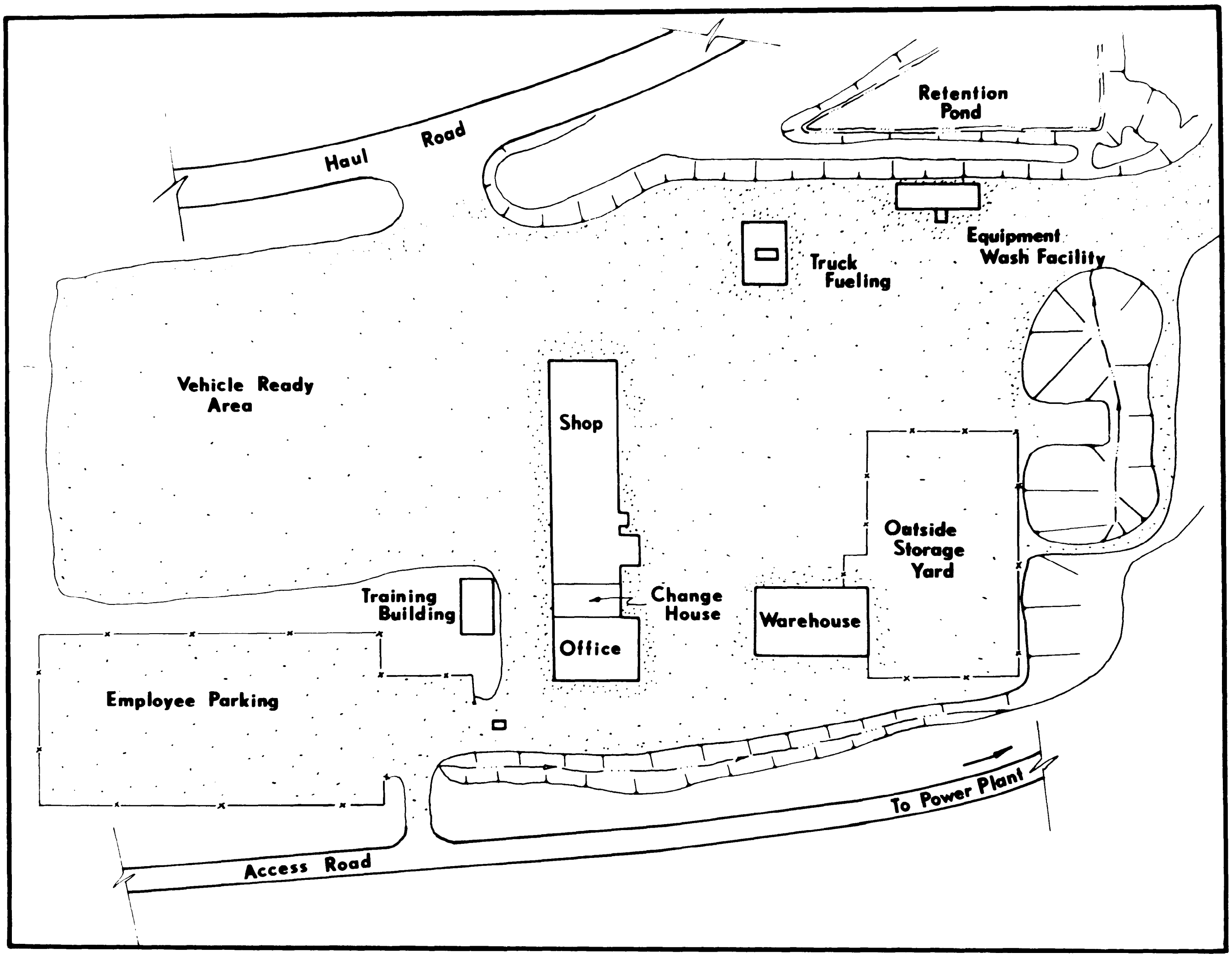

Fig. 14.3.6.6. Plan view of ancillary facilities.

The establishment of approximately 23% of the postmine acreage with hardwood and conifer tree species, shrub, and native grass species also is part of Big Brown's program. This is done to provide proper pasture-land management in order to enhance wildlife habitat, to increase species diversity, and to provide for distribution and cover for livestock. Tree stocking rates range from 200 to 700 trees/acre (50,000 to 173,000/km^2). To date, over 500,000 trees and shrubs have been planted at the Big Brown mine.

14.3.6.5 Product Control

Lignite samples are taken daily from the conveyor belt that delivers to the plant stockpile. These samples are analyzed in the quality control laboratory at the power plant. Results of the lignite analysis are returned to mine management for future use in mine planning.

14.3.6.6 Ancillary Facilities

A plot plan of the ancillary facilities at the Big Brown mine is shown in Fig. 14.3.6.6. The shop contains 21,440 ft^2 (1,994 m^2) and is equipped with a tool room, four welding machines, two air compressors, a bulk lubrication tank, six drive-through bays, and two 15-ton (13.6-t) overhead cranes. Repair and maintenance of mine vehicles are performed in the shop. Routine maintenance and small repairs on mobile equipment working in and around the pit area are performed in the field. When vehicle repairs have been completed in the shop, vehicles are moved to the ready area. Other usable equipment and vehicles are parked in the ready area when not being operated. Lignite haulage trucks, water trucks, and motor-graders are refueled and serviced at various intervals throughout the day. This eliminates much congestion around the shop area when trying to service many vehicles in the short period of time at shift change.

The changehouse area contains 3840 ft^2 (357 m^2). It features separate shower and locker space for male and female employees. This area is conveniently located next to the parking lot and shop area. Employees working in the pit area are picked-up near the changehouse for transportation to their workplace.

The office area, 6420 ft^2 (597 m^2), contains space for management offices, a conference room, engineering offices, accounting and personnel offices, and storage space for file cabinets. The training building located near the office and changehouse provides 1100 ft^2 (102 m^2) of space for employee training in safety, first-aid, and technical skills.

The warehouse, 10,880 ft^2 (1012 m^2) in area, is located in a separate building to reduce congestion around the shop and office area. The outside storage area provides space for tires, bucket parts, wire rope, large shafts and gears, and other items that need no protection from the weather.

Over the years, the Big Brown mine has been a consistent reliable producer of lignite fuel for the Texas Utilities System. In the future, Texas Utilities Mining Co. will continue to rely on this mine to provide a low-cost supply of lignite for use in electric power generation.

14.3.7 ATHABASCA OILSANDS: SYNCRUDE MINE

WAYNE N. MCKEE

14.3.7.1 Mine Description

The Syncrude mine, located 26 miles (42 km) north of the city of Fort McMurray, Alberta (Canada), is an integrated operation that has produced synthetic crude oil from the Athabasca oilsand deposit since 1978. The operation owned by Syncrude Canada Ltd., consists of a surface mine, a bitumen extraction plant, a bitumen upgrader, and a utilities plant, produced 60.3 million bbl (9.6 million m^3) of synthetic crude oil in 1991. Production from the Syncrude operation has increased steadily since start-up and is expected to peak over 70 million bbl (11.1 million m^3) per year within the next few years.

The Syncrude surface mine covers a total area of 8545 acres (34.6 km^2), shown as the Base mine in Fig. 14.3.7.1, with the mined-out portion at the end of 1991 amounting to 3885 acres (15.7 km^2) of this total. Significant additional mining areas exist in the remaining portions of the Syncrude oilsand leases. The strip mining operation involves two production highwalls 3 miles (5 km) long advancing to the east and west from a common box cut. A total of 420 acres (1.7 km^2) of ore body is mined out each year.

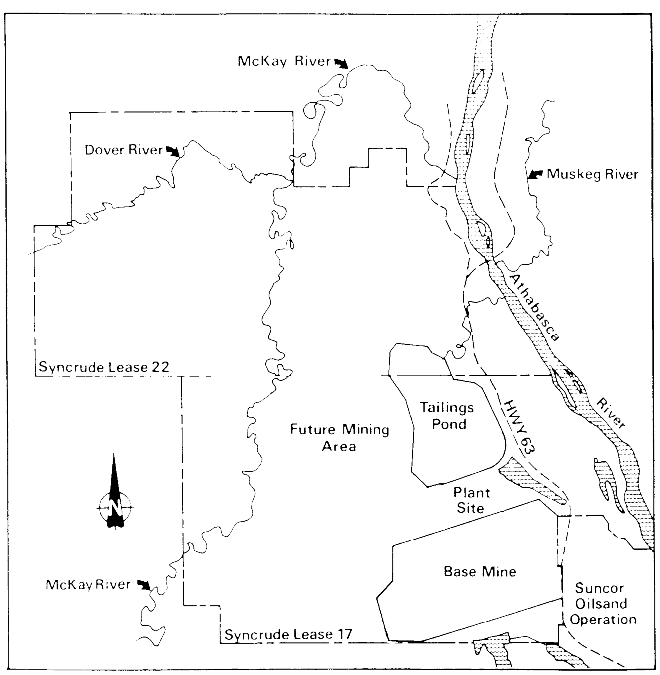

Fig. 14.3.7.1. Syncrude oilsand lease plan.

Ore production was in the 120 to 137 million tpy (109,000,000 to 124,300,000 t/a) range for the 1988 to 1991 period, with 140 million tons (127.5 Mt), or 79 million bank yd^3 (61 million bank m^3), projected for 1992. Production normally averages 400,000 tpd (364,000 t/d) but can peak at 500,000 tpd (455,000 t/d). Production is limited to 50% of normal during the annual maintenance shutdowns of the upgrader, lasting 30 days.

Waste handling by the shovel-and-truck fleet has increased steadily since 1985, with the volume rising from 28.9 million to 67.1 million bank yd^3 (22.1 million to 51.3 million bank m^3) in 1991. The 1992 waste handling volume is projected to be 72.2 million bank yd^3 (55.2 million bank m^3), of which 62.1 million bank yd^3 (47.5 million m^3) is in situ overburden and muskeg, 5.5 million bank yd^3 (4.2 million bank m^3) is oversized rehandle from the extraction plant, with the remaining 4.6 million bank yd^3 (3.5 million bank m^3) covering miscellaneous activities.

The Syncrude mine is operated on a continuous basis with two 12-hour shifts/day, 365 days/year. Four complete operating crews are required to maintain the 12-hour shift cycle. Maintenance crews operate on a modified cycle, with most workers permanently assigned to day shift. All technical, planning, and clerical staff work normal 8-hour/day shifts. The total mining workforce of 1400 consists of 975 occupationals and 425 staff. In functional terms, the total workforce is divided into 485 assigned to ore production and waste removal, 600 to maintenance, 140 supervision and management, and 175 technical and administrative.

14.3.7.2 Deposit Description and Geology

GEOLOGICAL SEQUENCE. The geology of the Syncrude mine is illustrated on the east-west cross section shown in Fig. 14.3.7.2. The oil-bearing McMurray Formation is of sedimentary origin, deposited unconformably in an extensive drainage basin eroded into the Devonian Limestone. The lower layer of the McMurray Formation, consisting of saturated silts and muds from a marsh environment, created a highly pressurized basal aquifer when overlain by the remainder of the deposit. The upper sedimentary layers are complicated by depositional and erosional patterns on a local and regional scale that create a structure of irregularly dipping bands within the ore body. The overburden deposit lying above the McMurray Formation consists of marine silts and clays, glacial tills, and muskeg. The lowest-lying and thickest unit of overburden is the Clearwater Formation, consisting of highly plastic and saturated silts and clays. The Clearwater Formation is overlain by a variety of material types including sand, gravel, clay, glacial till, and muskeg.

ORE ZONE. The upper layers of the McMurray Formation constitute the main ore zone, containing oilsand with a bitumen content varying in both horizontal and vertical directions from near zero to 15% by weight. Oilsand is an average composition of 86.8% quartzose sand, 11.0% bitumen, and 2.2% water. The extraction and upgrading processes produce an average of 0.74 bbl (0.12 m^3) of synthetic crude oil from each barrel (0.16 m^3) of bitumen in the mine feed.

The main ore zone varies from 115 to 164 ft (35 to 50 m) in depth, with approximately 15% of the zone grading as waste below the cutoff grade of 6% bitumen. Waste occurs in bands throughout the ore zone with variable thickness and irregular lateral extent (Fig. 14.3.7.2). Closely spaced core drilling (approximate 300-ft or 100-m square grid) and continuous highwall mapping are required to accurately locate waste bands for accurate separation. Thin clay bands up to 2 ft (0.6 m) in thickness are often found in the ore zone, usually associated with waste and ore contacts. These clay bands can create failure surfaces

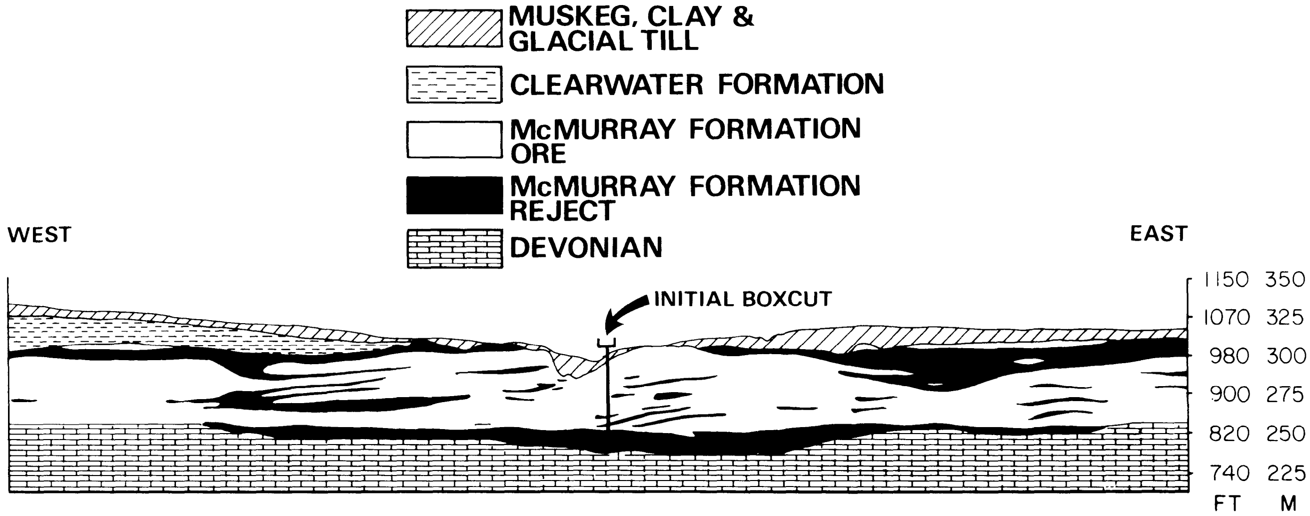

Fig. 14.3.7.2. Typical geologic cross section of Syncrude mine.

when dipping 10° or more out of the highwall. Siltstone lenses also occur in the main ore zone, predominantly within or along the waste bands. The lenses, normally less than 2 ft (0.6 m) thick, are horizontally discontinuous and fracture into large rectangular slabs when excavated.

The sand, bitumen, and water composition of oilsand cause the material characteristics to vary greatly with temperature. Oilsand is soft and sticky in summer and stiff or frozen during winter. Oilsand is very abrasive at all temperatures, due to the composition of angular sand grains suspended in the bitumen.

OVERBURDEN. Overburden material varies in thickness (Fig. 14.3.7.2) and generally becomes thicker as the mine progresses east and west from the box cut. Overburden thickness has increased from near zero in 1977 to 61 ft (18.3 m) in 1992, with a maximum of 98 ft (30 m) to be encountered in the future. Overburden is composed of loosely packed granular materials and uncemented clays with numerous lenses of siltstone. Overburden siltstone is generally thicker and more continuous than that found in the ore zone, with some lenses as thick as 3 ft (0.9 m). Gravel deposits occur within the overburden in quantities much lower than required for the ongoing needs of the plant.

The base elevation of the overburden removal operation is normally designed within the upper layers of the McMurray Formation. Low-grade oilsand and occasional bands of ore are encountered in conjunction with the overburden materials.

RESERVES. The in situ minable reserves remaining within the Syncrude mine after 1991 include a total of 739 million bank yd^3 (565 million bank m^3) of waste and 790 million bank yd^3 (604 million bank m^3) of ore (1.4 billion tons or 1.27 billion t) grading 11.1% bitumen in situ. The waste to ore ratio of the remaining mine reserves is 0.94:1.

The total potential reserves contained within the Syncrude oilsand leases, excluding the existing mine, are 10.2 billion bank yd^3 (7.8 billion bank m^3) of waste and 7.2 billion bank yd^3 (5.5 billion bank m^3) of ore grading 10.7% bitumen in situ. The waste-to-ore ratio rises significantly in these areas, averaging 1.43:1 due to greater overburden thickness.

14.3.7.3 Mining Methods

ORE PRODUCTION METHOD. In principle, the ore mining method shown in Fig. 14.3.7.3 is relatively simple. Draglines excavate the entire ore zone and deposit the ore in windrows on the operating bench. Waste is separated from the ore by the draglines and it is cast in to the mined-out pit. Bucket wheel reclaimers are paired with the draglines to reclaim ore windrows onto a conveyor system for transport to the extraction plant. Positioning of the major equipment is such that four independent production areas or quadrants are created, each with a dragline, bucket wheel reclaimer, and conveyor system (Fig. 14.3.7.4).

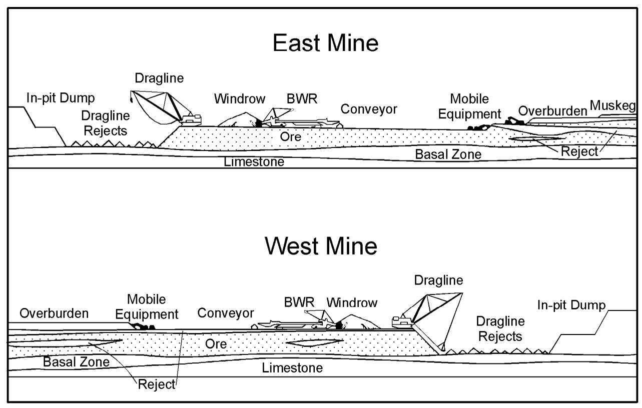

Fig. 14.3.7.3. Typical mining bench arrangement.

The mining method, employing both excavating and reclaiming equipment, was selected after extensive technical and economic evaluation of specific mine geology, production requirements, and capital cost. The use of a conveyor system was considered essential to handle long-distance transport of high ore tonnages at low operating cost. The mining method has proven successful in the first 14 years of operation, with numerous advantages apparent. These include

1. Blasting of ore is minimized because of dragline excavating capabilities.

2. Waste material is efficiently rejected from ore zone to mined-out pit.

3. Conveyor system complexities, necessary for the handling of ore and waste, are avoided because of the ability of draglines to separate ore and waste.

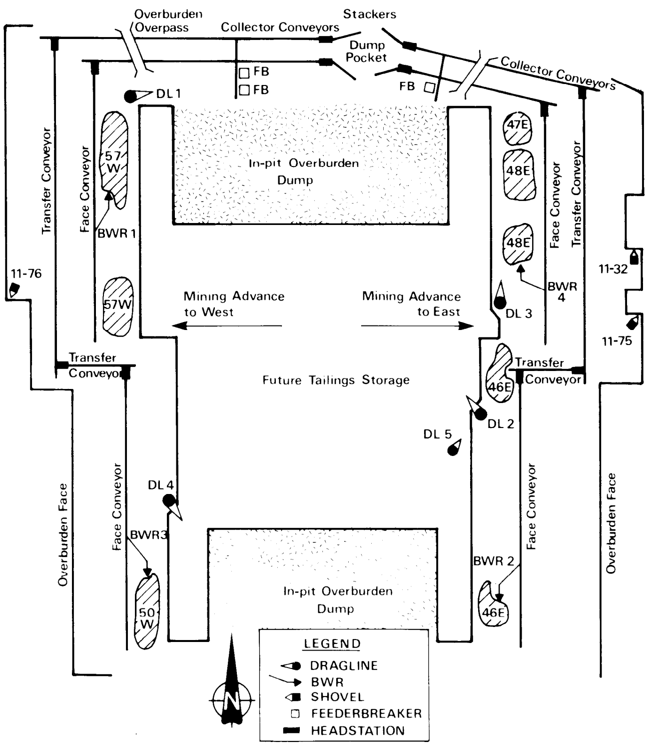

Fig. 14.3.7.4. Schematic mine layout.

4. Bench preparation is minimized by the draglines excavating the entire ore zone from one bench.

5. Trafficability is improved by avoiding operations on weak saturated basal sediments.

6. Lifting of ore by conveyor is minimized because draglines hoist all ore above the in situ ore zone.

7. Preproduction development costs are minimized by establishing one ore bench only.

The major disadvantages of the method are

1. Highwall stability must be continuously monitored to ensure dragline safety.

2. Inter-equipment delays are encountered due to the close proximity of equipment and the required mining sequence.

3. Power supply must be designed to handle high peak demands of draglines.

While simple in principle, the mining method actually requires significant technical support to ensure efficient, continuous production at the high rates required. The most notable areas of complexity are geologic interpretation, equipment positioning, material placement, highwall stability, and production scheduling.

OVERBURDEN REMOVAL METHOD. The overburden removal operation utilizes a shovel and truck fleet in a conventional benching method, with the waste material hauled to the mined-out pit for storage. The fleet excavates north-south strips of overburden in advance of the conveyor systems (Figs. 14.3.7.3 and 14.3.7.4), creating an operating surface for the production equipment. Timing of overburden strip removal is a critical balance between adequate preparation for the advancing ore production equipment and minimal preinvestment in overburden removal.

Preparation of the original ground surface is carried out well in advance of the overburden removal operation, due to the saturated condition of most upper overburden layers. Trees are cleared approximately five years in advance, allowing adequate time for subsequent muskeg drainage, muskeg removal, and overburden drainage. Muskeg is a loosely-compacted, saturated, organic material located in isolated surface channels. Muskeg is hauled directly to land reclamation sites or to separate storage areas for future use.

The overburden removal method has proven successful under Syncrude conditions, providing a high-degree of flexibility in acquisition of additional capacity, flexibility of operation, and relatively low capital cost. The Syncrude-owned fleet is supplemented by contractor equipment to handle long-term peak demands. Disadvantages of the shovel-and-truck method include relatively high operating cost, large operating and maintenance work force, and road building with less than ideal materials.

Numerous other overburden removal methods have been considered. The most notable was a bucket wheel excavator-conveyor-stacker method. High capital cost and questionable ability to handle high-strength siltstone lenses were the major reasons for rejecting this alternative.

14.3.7.4 Unit Operations

OVERBURDEN REMOVAL. The overburden removal operation is conducted by a mobile equipment fleet consisting of electric and hydraulic shovels, front-end loaders, and haul trucks, as described in Table 14.3.7.1. The utilization and productivity levels listed are sustainable averages, inclusive of seasonal variations in performance. In addition to the major equipment listed, the overburden removal operation also utilizes support equipment, including 9 Cat D10/D11 dozers, 3 Komatsu 475 dozers, 3 Cat 824/834 rubber-tired dozers, 6 Cat 16G graders, plus packers and service vehicles. Mobile equipment numbers have increased steadily to match increasing volume requirements. The initial fleet in January 1981 was composed of two Demag shovels and 10 Titan trucks. Options for handling further increases in overburden volume over the next few years include contracting, equipment rental, and Syncrude fleet expansion, with the final choice based on comparative costs, work force levels, and the availability of maintenance shop space.

Overburden removal is carried out in each mine quadrant, in north-south strips to accommodate the 490 to 660 ft (150 to 200 m) yearly advance of the draglines. Fig. 14.3.7.5 shows an active overburden strip located well in advance of the highwall and separated by the prepared bench. Bench space available for conveyor movement varies throughout the year, depending on completion of the overburden strip and rate of advance of the conveyor. The maximum bench space or "lead time" established in the south quadrants allows for nine months of conveyor shifting, while the maximum in the north quadrants allows for 12 months. The lead time criteria were established after considering capabilities of the removal equipment, conveyor move cycles, and seasonal limitations on bench surface preparation. The difference in lead time criteria between north and south quadrants is related to the different conveyor arrangement of each.

The overburden removal operation takes place in one or two lifts, depending on material thickness and type, and on the type of loading equipment being used. Other factors involved in bench design decisions include:

1. Trafficability of middle geologic units.

2. Recovery of construction-quality materials in the overburden face.

3. Recovery of ore bands in the overburden face.

4. Required strip completion date.

Overburden is hauled an average of 2 miles (3 km) one way to waste dump sites situated at the north and south ends of the mined-out pit (Fig. 14.3.7.4). Haulage routes are necessarily located around the ends of the ore conveyors. Bridges are used

Table 14.3.7.1. Major Equipment for Overburden Removal and Ore Production

Area	Equipment Type	Make and Model	Size	Fleet Size	Historical Utilization	Historical Productivity
Overburden Removal	Electric Shovel	Bucyrus-Erie 395B	34–36 yd^3 (26–27.5 m^3)	3	50%	2680 yd^3/hr[a] (2050 m^3/h)[b]
Overburden Removal	Hydraulic Shovel	Demag H241	19 yd^3 (14.5 m^3)	2	50%	1014 yd^3/hr[a] (775 m^3/h)
Overburden Removal	Front-end Loader	Hough 580	22 yd^3 (16.8 m^3)	1	50%	536 yd^3/hr[a] (410 m^3/h)
Overburden Removal	Hydraulic Shovel	O&K 200	21 yd^3 (16 m^3)	1	60%	1960 yd^3/hr (1500 m^3/h)
Overburden Removal	Haul Truck	Cat 789	200 ton (181 t)	6	68%	327 yd^3/hr (250 m^3/h)
Overburden Removal	Haul Truck	Titan T2000	200 ton (181 t)	4	68%	327 yd^3/hr 250 m^3/h)
Overburden Removal	Haul Truck	Titan T2200	240 ton (218 t)	2	NA	NA
Overburden Removal	Haul Truck	Titan 33-15B	170 ton (154 t)	16	68%	267 yd^3/hr[a] (204 m^3/h)
Ore Production	Dragline[b]	Bucyrus-Erie 2570W	80 yd^3 (61 m^3)	2	63%	4186 yd^3/hr[a] (3200 m^3/h)
Ore Production	Dragline[b]	Marion 8750	90 yd^3 (69 m^3)	2	63%	4186 yd^3/hr[a] (3200 m^3/h)
Ore Production	Bucket wheel Reclaimer[c]	O & K/Krupp	Schrs 2375 × 21 1.5	4	53%	6724 tph (6100 t/h)
Ore Production	Conveyors[c]	Krupp	6-ft belt (1.8-m) belt	18	53%	6724 tph (6100 t/h)
Ore Production	Feeder-breakers	Stamler/Syncrude	353-ton bin (320-t) bin	3	20%	3660 tph (2800 t/h)

[a] Volume is in bank units
[b] Indicated productivity is for all draglines combined
[c] Indicated productivity is for combined output of bucket wheels and conveyors
NA = Not available yet.

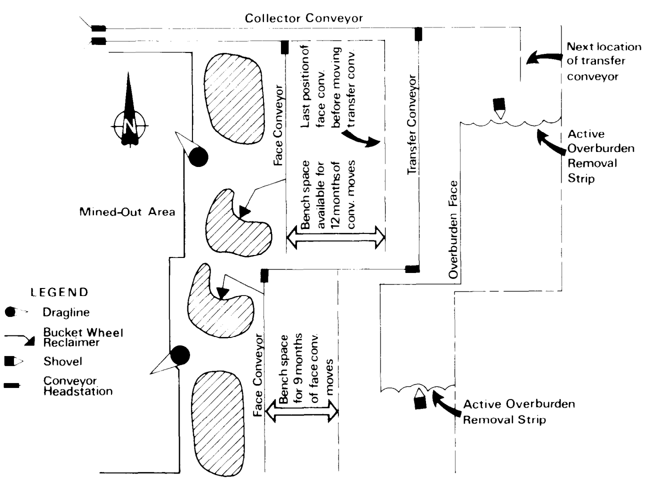

Fig. 14.3.7.5. Overburden strip removal and conveyor move sequence.

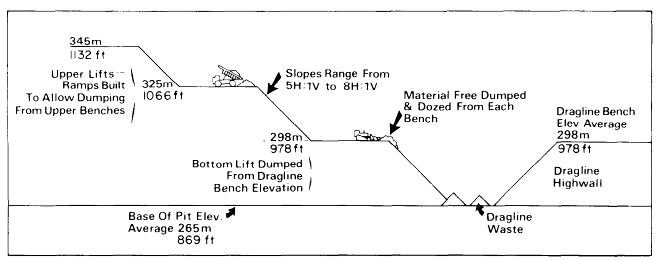

Fig. 14.3.7.6. Typical in-pit overburden dump cross section.

to cross collector conveyors situated along the north end of the mine.

Dump construction to a maximum height of 260 ft (80 m) is carried out in three free-dumping lifts (Fig. 14.3.7.6), with very little controlled placement of material required. Controlled placement is necessary only when hauling construction-quality material for capping of dump operating areas. Capping is necessary to ensure haul trucks can travel on dumps in the frost-free season.

The dump space available between ore highwall and dump toe is adequate to allow overburden to flow to an angle of repose as it is pushed over the edge of the dump. The types of overburden material and variable pit bottom contours lead to dump slopes as low as 8:1 (horizontal to vertical). The concept of building the dump at the angle of repose of the material avoids most stability concerns, except for localized slumping on the edge of each lift. Slumping does not threaten equipment safety because of the slow rate of movement.

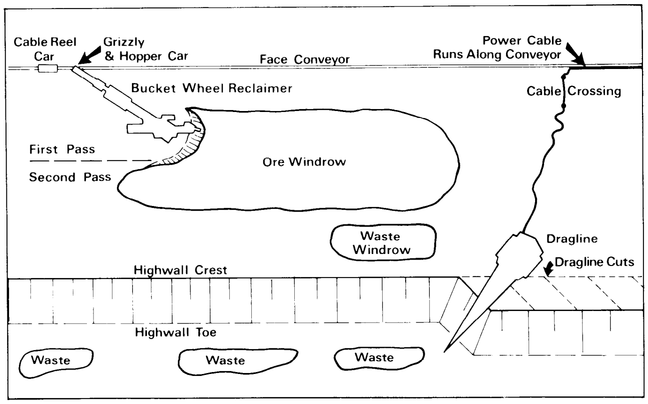

Fig. 14.3.7.7. Typical dragline and bucket wheel reclaimer configuration.

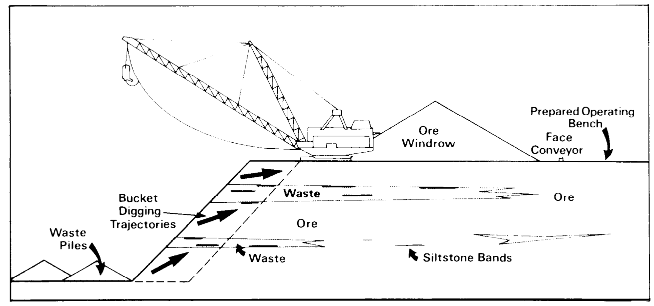

Fig. 14.3.7.8. Typical dragline highwall cross section.

Final bench preparation after overburden strip completion is a time-consuming and costly process, due to the tight specifications necessary to ensure efficient operation of ore production equipment. The preparation process involves subcutting and backfilling soft areas and leveling the entire bench surface within an acceptable tolerance of the original design. Bench preparation is complicated by a chronic shortage of quality backfill material, wet weather, and the low productivity of the work. The completed bench is quality tested by running a loaded haul truck over the surface and comparing actual rutting depth to a standard developed by trial and error.

Seasonal variation in overburden removal productivity is small. The winter operation is slightly more productive, due to a significant reduction in rolling resistance in the excavation and dump areas. Main haul roads are constructed to a high standard, with little variation in rolling resistance between seasons. An offsetting factor in winter is the 10 to 15% reduction in loading rate for shovels due to frost penetration. Blasting of overburden has proved to be economically unattractive.

OILSAND REMOVAL. Mining of in situ oilsand is carried out by four draglines (Table 14.3.7.1). The 360-ft (110-m) boom length of the four draglines is less than optimum for Syncrude highwall conditions, and this has a significant impact on material handling capabilities. With highwall depths averaging 148 ft (45 m) and highwall angles varying from 40 to 50°, situations frequently arise where dragline reach is insufficient.

Support equipment utilized in the dragline operation includes four Cat D10 dozers, one D9 dozer, one Komatsu 375 dozer, one 350 dozer, and one Cat 980 loader. In addition, two Cat 988 loaders have been modified to handle electrical cable spools, and two John Deere 300B loaders modified for electrical cable handling on the bench near the draglines.

The draglines mine the oilsand in 66- to 82-ft (20- to 25-m) wide panels over the 7550-ft (2300-m) length of each quadrant. The panel mining sequence is variable. Three main patterns are used, depending on highwall stability, production demands, and windrow inventory. Each panel is divided into 82-ft (25-m) long cuts (Fig. 14.3.7.7) providing a convenient increment of advance. Each dragline averages two cuts/day. Ore from each cut is placed in a windrow parallel to the receiving conveyor and at a predetermined distance from it. Careful windrow placement is necessary to minimize the dragline swing angle while ensuring the bucket wheel reclaimer can reach the entire windrow.

Dragline mining involves separation of oilsand into ore and waste as defined by the 6% bitumen cutoff grade. Grade bands are carefully delineated by core drilling and highwall mapping and communicated to the dragline operator by mine planners. Visual differentiation of grade bands is not always possible, and the operator must be assisted by a continuous indication of bucket depth by an on-board computer. The capability of the draglines to separate the relatively flat-lying bands in the McMurray Formation is restricted by the normal trajectory of the bucket (Fig. 14.3.7.8). Separation near the top of the highwall is good, and bands as thin as 10 ft (3 m) can be separated. Progressing down the highwall, the trajectory becomes closer to vertical, and efficient separation of bands can only occur if greater than 33 ft (10 m) thick.

Waste volumes vary considerably from cut to cut, and in some cases are high enough to allow large ore body blocks to be by-passed as uneconomical. Where waste volumes are high but the block still economic, rehandle is frequently necessary. Waste handling is carried out in various ways:

1. Placed direct to pit as it is excavated.

2. Placed on the bench beside the ore windrow until cut completion, and then rehandled into the resulting void.

3. Placed on the bench and left for later rehandle by mobile equipment .

Rehandle is minimized for economic reasons, and is normally carried out by the dragline. Mobile-equipment rehandle has a significantly higher cost, as well as logistical problems in getting the equipment in and out efficiently. Dragline waste is frequently required as construction or backfill material for the overburden operation, in which case, the waste is deliberately left in appropriate locations along the panel for later recovery by mobile equipment.

The amount of siltstone in each dragline cut is variable but predictable based on geologic interpretation and historical trends. Individual siltstone layers are usually impossible to reject unless associated with a thick waste band. The siltstone does not create a digging problem for the draglines, but once placed in the windrow, it does create a potential for damage to subsequent production equipment.

Stability of the highwall and operating pad is an important factor in efficient operation of the draglines. Considerable effort is put into preparing the pad by ripping and leveling prior to walking a dragline to a new cut. Tub deterioration has been reduced significantly by careful pad preparation work. Highwall stability is a major consideration due to the high risk of failure existing over approximately 15% of the highwall length. Numerous failure modes have been experienced to date, but very few situations have occurred where a dragline has been in jeopardy. An extensive geotechnical program, utilizing continuous instrumentation and visual monitoring, has been developed to virtually guarantee dragline safety.

In situations where bedding planes in the top 50 ft (15 m) of the highwall dip steeply toward the face, the geotechnical monitoring program alone is not adequate to ensure safe dragline mining of the affected area. Remedial work must be carried out to disrupt the bedding planes by either subcutting and backfilling, or blasting. Subcutting and backfilling are conducted by either dragline or mobile equipment.

The winter season affects the dragline operation by reducing the production rate by some 10%. Frost penetration up to 13 ft (4 m) can cause difficult digging conditions in faces that have been left unmined for extended periods. Blasting is used for all stable areas of the highwall, amounting to approximately 90% of the total length.

Bucket Wheel Reclaimer Operation. The windrows placed by the draglines, averaging 72 ft (22 m) high and 328 ft (100 m) wide, are reclaimed by the bucket wheel reclaimers (BWR) (see Fig. 14.3.7.7). The four reclaimers described in Table 14.3.7.1 are each capable of producing 30.8 million tpy (28 Mt/a), with production expected to peak at 31.4 million tpy (28.5 Mt/a) as downstream constraints are removed. Hour-by-hour production targets are met by utilizing three of the four reclaimers, allowing the fourth to be scheduled for preventive maintenance, component change-out, or conveyor shifting. In situations where only two reclaimers are available, production can be supplemented by mobile equipment.

Support equipment utilized in the reclaimer and conveyor operations includes two Cat 988 loaders, four Komatsu WA400/WA600 loaders, six Cat D10/D11 dozers, four Komatsu D355/D375 dozers, two Komatsu D355 sideboom dozers, two Cat 641 scrapers, three Cat 16G graders, and two 50 ton HaulPac trucks.

Reclaiming a window is accomplished in either a single pass or two approximately equal passes depending on the width of the window. The reclaimers are capable of reclaiming over a total range of 50 to 425 ft (15 to 130 m) from the conveyor centerline. Reclaimer reach is adjusted by changing the discharge angle between bridge and conveyor and by changing the length of the bridge discharge boom. The windrow is placed carefully to ensure that ore is kept within reclaiming range.

The sequence for reclaiming a windrow over a complete panel length is dependent on the relative position of the dragline and reclaimer and the amount of windrow existing in the quadrant. A complete panel over one quadrant contains approximately 3.3 million tons (3.0 Mt) of in situ ore, with the windrow inventory maintained between 550,000 and 1,650,000 tons (500,000 and 1,500,000 t). An ideal inventory of 1.1 to 1.6 million tons (1.0 to 1.5 Mt) in each quadrant allows flexibility in scheduling reclaimer production while providing a buffer in case of a dragline outage, either planned or unplanned. The reclaiming sequence is carefully planned to free the dragline when it becomes windrow-bound at the end of each panel. By ensuring the reclaimer is correctly positioned as the dragline completes a panel, the dragline can be freed in three days to start mining the next panel in the opposite direction. With the dragline starting a new panel, the reclaimer repositions to handle windrow in other parts of the quadrant, completing reclamation of the previous panel.

Operation of reclaimers in the winter season has been modified to solve the problems associated with frozen ore. Frost can penetrate a windrow pile to a depth of 6.5 ft (2 m), depending on the severity of the weather and the water content of the ore. Frost penetration is less in a windrow than in situ due to the "fluffed" state of the windrow. Small windrows, such as a second pass that is left unmined for an extended period, can freeze completely. Reclaiming during winter is accomplished at a 10% slower rate than in summer. The first and second passes are scheduled carefully to minimize frost penetration and to reduce the risk of damage to reclaimers and downstream production equipment. In addition, close coupling of dragline and reclaimer operations is carried out to reduce the exposure time of the windrow and to allow less frost penetration. Windrow inventory is maintained at lower levels. This lower level, however, increases the risk of windrow shortage during unplanned dragline outages.

Efforts to reduce the amount of frozen lumps and siltstone pieces being picked up by reclaimers have been partially successful, and damage to ring chute, slope sheet, conveyor belt, and baffle walls occurs on a much reduced frequency. Each bucket on the reclaiming wheel has a modified opening to restrict the size of lumps picked up. This modification was carried out by trial and error to find a suitable balance between production rate and lump rejection.

Conveyor Operation. Ore flow from each reclaimer is discharged through an A-frame grizzly, with one-directional bars, into a hopper located over a conveyor. Large frozen lumps and siltstone pieces are deflected to the ground and trammed by front-end loader to the highwall or into piles for later rehandle by dragline. The grizzlies are successful at removing most of the oversize material. Some tabular pieces, however, pass between the bars and onto the conveyor. Development of grizzlies has been by trial and error to find a balance between high throughput and maximum lump rejection.

The layout of conveyors from reclaimer to dump pocket appears in Fig. 14.3.7.4. The north quadrants each have a receiving conveyor, a collector conveyor, and a stacker conveyor. The south quadrants have a more complex system, each with an additional transfer conveyor to route the ore flow around the north quadrant. Ore is discharged from each stacker conveyor onto a surge pile at the north end of the mine. Total surge capacity is 303,000 tons (275,000 t), of which 27,600 to 33,100 tons (25,000 to 30,000 t) can be fed directly to the extraction plant through 16 conical drawpoints and four feeder conveyors. The remaining surge capacity is situated close to the drawpoints and can be dozed in when the feed rate from reclaimers is restricted.

Each conveyor has a headstation to provide belt drive power and tension, and to transfer ore to the next conveyor. Power is supplied by four 1250-hp (932-kW) electric motors in each headstation and supplemented by an additional motor on the tailstation, if required. Tailstation anchoring is accomplished with large concrete deadweights that can be added or subtracted to attain the necessary belt tension.

The conveyor system is expanded to the east and west to maintain correct positioning with respect to the highwall as the mine expands. The receiving and transfer conveyors are shifted over the bench surface by specially equipped dozers that traverse the length of the conveyor, pulling the conveyor sections over 1.5 ft (0.5 m) in each pass. Movable headstations are walked to a new location by three hydraulic "walking legs" that are detachable for common use. Collector conveyors remain in a fixed position and are extended east and west to match the headstation position of the receiving and transfer conveyors.

The sticky nature of oilsand causes conveyor operating problems, such as buildup on components and spillage within and under the structure. Build-up on conveyor idlers, causing premature shell and bearing failure, was resolved by changing from steel-shelled to rubber-lagged idlers. Build-up on the belt and pulleys has been resolved to some extent by belt wetting and mechanical cleaners. Research is continuing in this problem area. Spillage, occurring mainly from oilsand falling from the return belt, creates an ongoing need for cleanup. Specially equipped front-end loaders are used to clean under conveyor sections and around headstations.

AUXILIARY PRODUCTION SYSTEM. The main production system of reclaimers and conveyors is at times incapable of meeting production demands and must be supplemented by the auxiliary production system. The system, consisting of the overburden haulage fleet and three feeder-breaker units, is utilized in situations where the production shortfall is expected to last two hours or longer. Shorter periods of production shortfall can be supplemented by dozing on the surge pile. Various sources of ore exist for the auxiliary system, with the choice dependent on the necessary production rate and the inventory level of sources at the time.

Design of the Syncrude feeder-breaker units is a size-up from similar units operating at other materials handling operations. Consisting of a 350-ton (320-t) hopper, a single breaker drum, and a pan feeder, the units can break and transfer ore from haul truck to collector conveyor. The feeder-breakers have achieved average rates of 2755 tph (2500 t/h) each while handling ore containing frost lumps. The units have not been as successful at breaking siltstone, necessitating either a lower production rate or careful selection of feed sources to avoid high siltstone content. The units are installed against a Reinforced Earth retaining wall and ramp system, providing an efficient facility for haul trucks dumping into the hoppers.

RECLAMATION. Reclamation of disturbed land surfaces has been underway since 1985, with work carried out on both temporary and permanent surfaces. Final land surfaces have been available for reclamation since 1991.

Research is underway to find the optimum soil-building technique that will allow efficient reforestation of disturbed land areas. Current reclamation work is carried out by selectively stripping surface materials in advance of the overburden removal operation and creating a 2.3- to 3.3-ft (0.7- to 1.0-m) thick soil layer on disturbed surfaces. Fertilizer is incorporated into the new soil prior to planting indigenous grass or trees.

14.3.7.5 Ancillary Operations

DRILLING AND BLASTING. Blasting is utilized in the Syncrude mine for winter dragline digging. Blasting of the highwall prior to dragline digging is carried out with a cratering technique during the December to March period. Holes are drilled 4 in. (100 mm) in diameter to the depth of the frost, using auger and percussion drills. The grid pattern is approximately 10 by 13 ft (3 by 4 m). Amex explosives are used at a rate of 18 lb (8.3 kg) in the bottom of each hole with a Forcite primer, Primacord, and electrical detonation.

Blasting of auxiliary production sources for winter use utilizes the same technology as described for highwall frost blasting. Auxiliary production blasting is performed to loosen the oilsand to reduce frost penetration, and, therefore, a smaller explosive charge is necessary.

Blasting also is carried out to improve highwall stability by disrupting undesirable bedding planes. Approximately 2.2 million tons (2 Mt) of oilsand are blasted each year with a cratering technique that utilizes 31.5-in. (800-mm) diameter holes on a 66-ft (20-m) diagonal pattern. The holes are drilled by an auger drill to a maximum depth of 92 ft (28 m). Heavy ANFO (HANFO) explosives are loaded in the bottom of each hole to a maximum charge of 5070 lb (2300 kg), with l-lb (0.45-kg) primers electrically detonated at one hole per delay.

MAINTENANCE. Mine maintenance activity is divided between shop and field, with most mobile equipment maintenance carried out in the permanent shop facilities and most production equipment maintenance carried out in the field. A total of 187,300 ft^2 (17,400 m^2) of shop space is available at the north end of the mine for mobile equipment bays, fabrication work, dragline bucket rebuilding, conveyor belt splicing, electrical equipment rebuilding, and major component construction. In addition, a 60,300-ft^2 (5600-m^2) machine shop is available in the upgrader complex for use by all operating areas of Syncrude.

Field maintenance is carried out on the larger equipment utilizing trailer facilities to increase efficiency through reduced travel time and improved working conditions. Individual trailers containing worker facilities, tool cribs, parts stores, fabrication shops, and lubrication materials are set up adjacent to the maintenance work.

The total maintenance work-force of 600 is organized with 350 workers assigned to permanent day shift and 250 covering the night shift. Maintenance work during major equipment outages can be scheduled on a continuous basis depending on production need. A total of 40 inspectors is utilized to monitor equipment condition through visual and non-destructive testing techniques.

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14.3.8 FLORIDA PHOSPHATE: FT. GREEN MINE

STEVEN I. OLSON

14.3.8.1 Mine Description

The Florida phosphate industry produces approximately 75% of the domestic output and nearly 25% of the world's supply of phosphate rock (Anon., 1988a). Agrico Chemical Co., which is a division of Freeport-McMoRan Resource Partners, a vertically-integrated entity which mines phosphate and sulfur and produces chemicals and fertilizer. Its roots can be traced back to the very beginnings of the phosphate mining industry (1880) and the sulfur mining industry (1912). Agrico currently has two operating mines: Ft. Green and Payne Creek located, respectively, in Imperial Polk County and Hardee County, FL.

Agrico's Ft. Green mine (Fig. 14.3.8.1), commissioned in 1975, produces about 4 million tpy (3.6 Mt/a) of phosphate rock. The Payne Creek mine, started in 1966, produces about 2.5 million tpy (2.3 Mt/a). The Ft. Green mine is typical of the newer and larger operations in Florida.

About 700 acres (2.8 km^2) per year are mined at the Ft. Green mine, which involves moving about 28 million yd^3/yr (23 million m^3/yr) of overburden and about 12 million yd^3/yr (10 million m^3/yr) of matrix.

Operating schedules in the mining and beneficiation of phosphate vary substantially from one operation to another, depending on the specifics of the operation (current supply/demand situation, mine/plant limitations, etc.). Production schedules range in the industry from a full seven-day schedule (24 hr/day and 7 days/week) to variations of five-day schedules that may run from 1 to 3 shifts/day. Some schedules take advantage of lower electric rates during off-peak demand periods.

The Ft. Green operation is on a full 7-day schedule with 1 shift/week scheduled for repairs to the beneficiation operation. Each dragline is scheduled for a one-shift repair period every third week.

14.3.8.2 Deposit Description and Geology

GEOLOGIC SECTION. The phosphate ore (matrix) of Central Florida is a sedimentary deposit located in the west-central part of the Florida peninsula in west-central Imperial Polk County, east-central Hillsborough County, and northern Manatee and Hardee Counties. This area is known as the "Bone Valley" and is the source of phosphate ore for present mining activities. A southern extension exists in southeastern Manatee, southwestern Hardee, and northern DeSoto Counties. As the land pebble district deposits become depleted, mining will continue into the southern extension. The industry in Florida has proven reserves for several decades, and resources are known (at current production rates) that would satisfy needs forward for over 100 years.

Fig. 14.3.8.1. Overview of the mine site.

Agrico's reserves lie in southwestern Imperial Polk and northwestern Hardee County. Recently acquired lands extend farther south into Hardee and into Manatee County. Currently about 98,000 acres (324 km^2) are controlled by Agrico.

The phosphatic, clayey sands, known as phosphorites, occur in the Hawthorn and Bone Valley Formations. The phosphate occurs as carbonate-fluorapatite (francolite). Varying degrees of weathering produce alteration minerals, such as wavellite or crandallite. The matrix consists of the phosphorites, silica sand, and montmorillonite and palygorskite clays.

The Hawthorn Formation is the source of primary phosphate, while the Bone Valley Formation phosphate is the result of reworking the underlying Hawthorn material. The primary phosphate is thought to be the product of biological and/or chemical precipitation that took place in a shallow marine environment during the middle Miocene Epoch (12 to 18 m.y. ago). Upwelling along the southeastern United States continental shelf provided cold nutrient- and phosphorus-rich waters to proliferate plant and animal life. Precipitated phosphate and marine skeletal remains were deposited, along with sands and clays, to form the youngest portion of the Hawthorn Formation. Dropping of the sea level exposed the Hawthorn to erosion and weathering during the Plio-Pleistocene Epochs (1 to 7 m.y. ago). The reworking of the top portion of the phosphate-bearing Hawthorn and redeposition produced the Bone Valley Formation, a sandy phosphate bearing material with various stages of weathering and cycles of reprecipitation of phosphate grains.

DEPOSIT GEOMETRY. At the Ft. Green mine, the overburden can vary from 5 to 45 ft (1.5 to 13.7 m) thick and averages 25 ft (7.6 m). It typically varies gradually over a local region. The matrix section consists of up to several horizontal strata that can vary considerably in components. Total matrix thickness varies from 0 to 25 ft (0 to 7.6 m) and averages about 10 ft (3 m). The phosphate can vary in product grade, diluent levels, and size distributions. The sand may vary in amount and occasionally the size distribution. The clay will vary in mineralogical composition and proportion. The top strata are frequently a "leach zone" that will tend to be higher in uranium values. These strata are usually mined for their phosphate values, but will occasionally be spoiled because of unacceptable product grades. The matrix will typically bottom out in a yellow, clayey, dolomitic horizon.

PROPERTIES OF ORE AND WASTE. The overburden in the Bone Valley is primarily a loosely-aggregated, clayey sand. The underlying ore, or matrix, is a composite of sand, clay, and phosphate. Historically, the industry has encountered matrix that averaged 1/3 sand, 1/3 clay, and 1/3 phosphate. Southern deposits tend to have more sand, less clay, and less phosphate. Agrico's is a southern deposit.

The sand is normal silica sand that ranges from about 14 to 150 mesh (Tyler) (1.2 to about 0.01 mm). The clay is a complex mixture of montmorillonite and palygorskite. The clay fraction (defined as – 150 mesh or –0.01 mm) has an average particle size of 1 μm and presents a challenge for disposal.

The phosphate grain sizes range from pebble-sized gravel to very fine sands. The color of the grains varies from an opaque black, brown, and tan to cream or a translucent amber.

GRADE AND SIZE DISTRIBUTION. During mining and beneficiation, purity is measured in BPL (bone phosphate of lime). In the chemical processing area, P_2O_5 is the standard unit. Ft. Green produces three phosphate products: pebble, intermediate pebble (I.P.), and concentrate. The pebble is the coarsest product and is –3/4 in. to +14 mesh (–19 mm to +1.2 mm). The I.P. is –14 to +20 mesh (–1.2 mm to +0.75 mm), and the concentrate is the product from a flotation process and is –20 to +150 mesh (–0.75 mm to +0.01 mm). Typical product grades are: 65 BPL for pebble, 60 BPL for I.P., and 72 BPL for the concentrate. Diluents include calcium, silica, magnesium, iron, and alumina. Each impacts the chemical processing requirements differently.

Over the years, Florida phosphate mining has produced primarily pebble. Before the use of the flotation process, the pebble-sized phosphate was mined and the sand-sized phosphate discarded with the silica sand and clay as tailings. Some operations still have pebble:concentrate ratios of 3:1. Pebble is a cheaper product that requires less processing to produce. Agrico's operations are not so fortunate. Typical ratios are 1:1 or occasionally less.

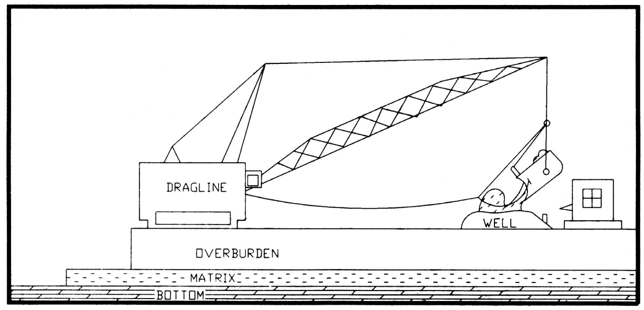

Fig. 14.3.8.2. Typical mining cross section.

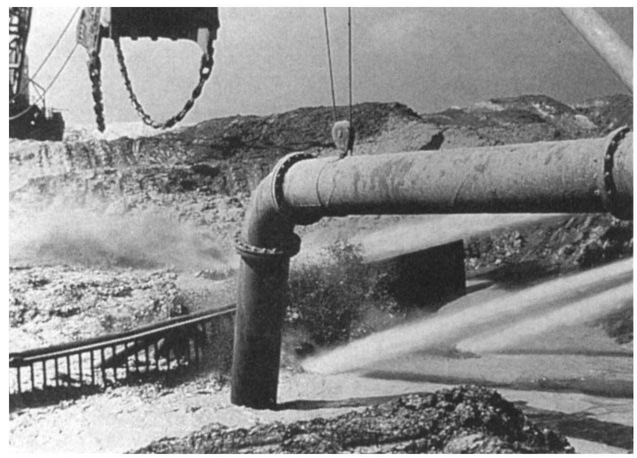

Fig. 14.3.8.3. Well and ore pumping station.

14.3.8.3 Mine Development

DEPOSIT DELINEATION. Prospect drilling is completed to initially delineate the deposit. This type of drilling is performed on a broad spacing of one 4-in. (100-mm) drill core to bedrock every 40 acres (0.16 km^2). The prospect cores are logged and cut by a staff geologist. Each stratum is then processed through a laboratory simulation of the actual process. Each core is measured and processed to produce metallurgical, analytical, and processing data.

Followup drilling is completed on 330-ft (100-m) centers (one hole every 2.4 acres or 9712 m^2). These prospect samples are handled identically to the preliminary drilling samples and are used to expand the resource database.

This drilling density enables more definitive planning and forecasting. From this level of detail, annual budgets and monthly forecasts are made. Some short-term "surprises" (richer or leaner areas encountered in a given period) still occur, but the drilling density provides a good representation in the long run for monthly and annual forecasts.

LONG-RANGE MINE PLANNING. A long-range mine plan is developed for the mine that is intended to optimize the reserves. This long-range plan includes not only mining specifics but also waste product disposal plans, reclamation plans, primary power distribution networks, and permit requirements. This plan is updated frequently to reflect recent reserve acquisitions, current demand situations, and anticipated time periods to secure the required permits. The long-range mine plan is refined prior to the beginning of each year. Pipeline and pump locations, hydraulic stations, dewatering schemes, permits, and local area power service needs are included in the plans.

PERMITTING. Permitting for phosphate mining has become more involved in recent years. The uncertainty of successfully receiving all required permits is a substantial disincentive for new mines. The permitting process for a new mine may take as long as 10 years and cost over $10 million (Anon., 1984). New mines (or major expansions) are required to prepare a State Development of Regional Impact Study. They may also be required to complete a Federal Environmental Impact Study.

Current mining operations must have permits from federal, state, regional, and county governments. Some of the agencies involved include the Environmental Protection Agency, the Florida Department of Natural Resources, the Florida Department of Environmental Regulation, the Army Corps of Engineers, the Southwest Florida Water Management District, State Department of Community Affairs, the regional planning councils, and various county agencies. Duplication and overlap exist in several areas. The FDNR requires mining and reclamation permits (reclamation plans must be approved prior to commencement of any mining activities). Several of the others are involved in wetland permitting. It is estimated that 20% of the Florida phosphate reserves are under wetlands (Anon., 1984). Wetlands are a scarce and unique part of the environment. They serve a role in water purification, recharging of aquifers, and wild life habitat.

14.3.8.4 Unit Operations

Stripping is accomplished with electric walking draglines, most typically in the 35- to 45-yd^3 (28- to 34-m^3) category. The draglines strip the overburden and spoil it into the voids left from the previous mining activity. After the overburden is removed, the ore or matrix is exposed. It is excavated and placed in a shallow well (Figs. 14.3.8.2 and 14.3.8.3). It is slurried with high-pressure water, washed through a grizzly, and pumped away for beneficiation.

Ft. Green has three walking draglines: two model 1260-W Bucyrus-Erie machines, each with 42-yd^3 (32.1-m^3) buckets, and a model 1250-W with a 36-yd^3 (27.5-m^3) bucket. Each has a boom 225 ft (68.6 m) long. Power at 4160 Vac is fed to the machine through a trailing cable and motor/generator sets that

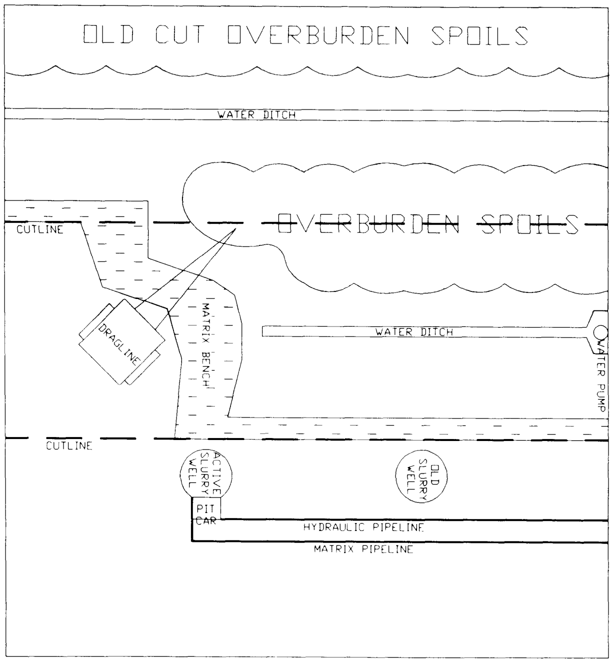

Fig. 14.3.8.4. Typical mining plan view.

produce direct-current power. Typical digging rates are 2400 yd^3/hr (1840 m^3/h) per dragline.

CLEARING. Area preparation for central Florida phosphate mining is relatively simple. The land is generally flat and is typically scrub brush with palmetto bushes and pine trees. These areas are prepared for mining by removing all the vegetation. Wetlands (swamps, bayheads, isolated ponds, etc.) also are common. Preparation of these areas following receipt of all permits can become quite involved.

MINING. The strip mining that occurs in central Florida is not complex. Mine planners lay out long cuts (3000 to 4000 ft, or 914 to 1218 m, if possible) that are 225 to 330 ft (68 to 100 m) wide (depending upon deposit geometry). The draglines remove the sandy overburden. The overburden from the first activity in the area must be spoiled on unmined ground. Subsequent overburden is placed in the void left from the previous cut. The phosphate ore, or matrix, is then exposed. In this mining geometry, the pumping system is moved parallel to the advancing dragline and is placed on the edge of the next cut to be mined by the dragline. A ditch is maintained by the dragline in the bottom of the cut to collect all seepage (Fig. 14.3.8.4). A submersible pump or "waterjack" is placed in the ditch and pumps the water to a drainage system on the surface that routes the water to the area hydraulic system for re-use. The ditch may be up to 15 ft (4.6 m) deep. The spoil from the ditch (Hawthorn Formation material) is placed at the bottom of the cut. This material, like the "leach zone," is slightly elevated in uranium values. Uranium progeny includes radon, which is a subject of concern for health in dozens of areas in the United States. Both are bottom spoiled when encountered to facilitate better-quality reclaimed land.

As the dragline advances (typically, 100 to 150 ft/day (30 to 46 m/day), the pumping systems must remain within the reach of the machine. They generally are moved 300 to 400 ft (91 to 122 m) at a time. A few rare situations, such as mining a corridor, call for "windshield wiper" cuts that are about double the normal width. In these cases, the pumping systems are centered behind the dragline. The dragline mines in a horseshoe pattern around the well (see Fig. 14.3.8.3) starting at the open cut. To facilitate dewatering, the dragline maintains a waterjack ditch that is tied to the one in the old cut. As the dragline mines and retreats down the cut, the pumping system is cut back directly behind the dragline.

ORE HANDLING. The initial or "pit" pump is a 20-in. (508-mm) centrifugal slurry pump fitted with a long suction pipe that extends into a depression behind a grizzly. It is slurried with water from three high-pressure (up to 275 lb/in.^{2} or 1900 kPa) water cannons, similar to those used in placer mining (Chapter 15.1). The material passes through a grizzly with 6-in. (152-mm) openings. The pumping systems from the mining area to the processing plant can be several miles (km) long in older mines. They average 4 miles (6440 m) at Ft. Green. Each system employs a 20-in. (508-mm) seamless, abrasion-resistant steel pipeline, with several booster pumps spaced from 3500 to 5000 ft (1070 to 1520 m) apart, depending upon ore characteristics, terrain, and system design. The systems are designed to handle 13,000 to 15,000 gal/min (0.8 to 0.96 m^3/s) of the matrix slurry at a density of 30 to 45% solids. One or more of the boosters in a given system has a variable speed control to allow for pipeline extensions or reductions coincident with pit moves. The pumps are powered by 1250-hp (932-kW) motors and are controlled remotely. All systems are radio-controlled. Mass flow systems are used extensively to optimize matrix pumping performance.

RECLAMATION. Mining is a temporary land use that disrupts the landscape (Anon., 1986). The Florida phosphate industry has been acknowledged as producing a reclamation result unequaled by mining industries in other states (Anon., 1988b). Various land forms have been created and are being used for agriculture, housing, industrial tracts, recreational areas, and wildlife preserves (Anon., 1986).

Deposition of tailings from the beneficiation plant is the beginning of reclamation. Tailings are pumped back into the mined area between the overburden spoil piles. An elevation is achieved that permits covering the surface with the overburden and sculpturing the landscape as articulated in the reclamation plans.

Examples of successful reclamation projects include a 366-acre (1.48-km^2) upland section that recharges a 150-acre (0.61-km^2) wetland area known as Morrow Swamp. The area has diverse vegetation and wildlife and is representative of a naturally occurring swamp. Agrico also has established pasture lands, orange groves, and several lakes. These efforts have received national awards.

Waste-clay disposal presents a challenge for reclamation. Historically, the clay ponds have been abandoned, and after two decades or so, the surface has adequately dewatered to permit limited grazing by cattle. The disposal of these poorly consolidating clays has been the subject of millions of dollars in research by operating companies, universities, private research groups, and various arms of the federal and state governments.

The clay disposal problem is a volume problem and is influenced by the composition of clay types. The clay in the matrix (after millions of years of consolidation) is typically in the range of 60 to 65% solids (by weight). The components, montmorillonite and palygorskite, have very poor settling characteristics, depending upon the specific mineralogy. A sample of clay could require from a few weeks to several years to consolidate from 3 to 5% solids initially to 15% solids during the desliming process. Generally speaking, the clay fraction must achieve about 30 to 35% solids to fit into the void left from mining (excluding the volume of the sand tailings).

Dozens of disposal schemes, settling enhancement methods, and consolidation improvement techniques have been investigated. It was discovered that the clays proceed through two phases: a settling phase (typically from 3 to 15% solids range), and then a consolidation phase (from 15% solids upwards). Flocculation studies by the dozens were completed. However, they generally only addressed the settling phase of the problem.

Disposal methods evaluated included freezing and thawing the clay slurry to produce enhanced consolidation. Potential uses for the clay were investigated. Sand was admixed to increase the unit weight to improve consolidation. Elaborate techniques were attempted to allow placement of a "cap" of sand upon partially consolidated clays to enhance consolidation. Some methods were proven to be technically successful, but none were proven to be commercially viable.

Currently, Agrico is employing one method that does show promise for rapid reclamation of these large clay disposal areas. Unique low-ground-pressure vehicles (tires are 6 ft or 1.8 m wide, and 5 ft or 1.5 m tall) are being used to facilitate surface drainage. They have a hydraulically driven attachment that cuts ditches in the surface of the pond. This surface is continuously reworked to enhance drying of the surface. With this method, clay ponds have been converted to functional land forms within a matter of five to seven years. They are being used as pastures and for farming research. The Institute of Food and Agricultural Sciences of the University of Florida, in conjunction with the Florida Institute of Phosphate Research, is conducting agricultural studies of these areas. The goal is to produce a landform that will be functional and useful for the local economy. (Agrico received a national award in 1987 for this innovative technique.)

14.3.8.5 Ancillary Operations

PRODUCT PROCESSING. The initial separation step in phosphate beneficiation is washing (Fig. 14.3.8.5). There the phosphate slurry is received from the transportation system and passed through several stages of disaggregation (log washers) and screening, with the purpose of separating the +14 mesh (+1.2 mm) pebble from the balance of the matrix.

At Ft. Green, the slurry from any two pumping systems is received in a tank where it is transferred to parallel trommels that reject the +¾ in. (+19 mm) material. The reject is termed "debris" and is primarily dolomite. The –0.75 in. (–19 mm) material then passes flume or "flat" screens, primary vibrating screens, primary log washers, secondary vibrating screens, secondary log washers, and tertiary or finish vibrating screens. All are fitted with plastic screen media and sized at 14 mesh (1.2 mm). The +14 mesh (+1.2 mm) or pebble is taken, via conveyor belt, to temporary storage, awaiting shipment by Agrico's own railroad to Agrico's South Pierce Chemical plant in Imperial Polk County, or to Agrico's Big Bend Terminal, which is a deep-water port in Tampa. From there, rock may be shipped to Agrico's Faustina or Uncle Sam chemical facilities in Louisiana.

As in other regions of this country, water consumption is a sensitive topic. The phosphate industry is obviously a large water user, but, in fact, it is not a large water consumer. The industry ranks third in the Southwest Florida Water Management District in water consumption, behind agriculture and municipalities. The industry has made major strides in water conservation. In 1970, 238 million gal/day (10.427 m^3/s) were used to produce 24 million tons (22.7 Mt) of phosphate rock. In 1980, 153 million gal/day (6.703 m^3/s) were needed to produce 43 million tons (39 Mt) (Anon., 1983).

Ft. Green uses about 100,000 gpm (6.39 m^3/s) of water. Between 90 to 95% is recovered and reused. The small amount of fresh water consumed is used in the second portion of the flotation process and for potable purposes.

Feed sizing follows the washing stage and includes desliming and classifying the feed into appropriate sizes for separation by flotation. The –14 mesh (–1.2 mm) material (sand, sand-sized phosphate, and clay) is processed through two stages of hydrocyclones to deslime the material. The size of separation is a nominal 150 mesh (0.1 mm). The waste clay (3 to 5% solids) is collected and pumped to large above-ground disposal areas called "ponds." The clays in the Florida phosphate matrix are montmorillonite and palygorskite, which have an average particle diameter of 1 µm and poor dewatering characteristics.

The –14 to +150 mesh (–1.2 to +0.01 mm) material is sized into three fractions: –14 to +20 mesh (–1.2 to +0.75 mm), –20 to +35 mesh (–0.75 to +0.45 mm), and –35 to +150 mesh (–0.45 to +0.01 mm). This sizing is achieved with a combination of hydraulic classification and screens. At Ft. Green, the twice-deslimed material is called "unsized" feed. It is processed through parallel hydrosizers that make a split at 35 mesh (0.45 mm). The hydrosizers achieve that separation with a combination of elutriation and hindered settling. The –35 mesh (–0.45 mm) portion is put through yet another stage of desliming and becomes the "fine feed." The +35 mesh (+0.45 mm) fraction is fed to vibrating screens that size at 20 mesh (0.75 mm). The oversize from these screens is called "I.P. feed." The fine fraction (–20 to +35 mesh or –0.75 mm to +0.45 mm) is "coarse feed". The processing of the coarsest of these fractions (I.P.) is an area of difference in most Florida phosphate beneficiation operations. Some will upgrade it using a fatty acid flotation step and spiral classifiers or belts. Others will basically ignore it and back-blend it with the pebble product. Agrico passes it through a single flotation stage using a strong amine in small flotation cells that have been supercharged. Fine sand is removed, and the product is used in the company's own chemical plants.

Separation of the sand-sized phosphate from the silica sand is achieved with a double-float "Crago" process that was developed about four decades ago (Anon., 1986). The process uses a fatty acid rougher flotation step to float the phosphate from the sand. The rougher concentrate is then de-reagentized, and a reverse, or cleaner, step follows that uses amine to float the remaining sand from the phosphate. In this manner, the minor constituents are reagentized and removed. The coarse and fine feeds are treated similarly. Prior to rougher conditioning, the pulp is deslimed again to remove residual –150 mesh (–0.1 mm) clays, and dewatered to 70 to 75% solids (by weight). The pH is adjusted to about 9.0 with ammonia. A mixture of fatty acid and fuel oil is added in the conditioners to facilitate separation. Fine flotation is completed in 500-ft^3 (14.2-m^3) Wemco flotation cells, retrofitted with operating mechanism from 1000-ft^3 (28.3-m^3) cells. Coarse flotation occurs in 300-ft^3 (8.5-m^3) Wemco units fitted with 500-ft^3 (14.2-m^3) mechanisms. The rougher concentrate from both fine and coarse feeds is combined, dewatered to +70% solids, and scrubbed with a small amount of sulfuric acid to de-reagentize the pulp. It is then rinsed to wash away the spent reagents and any clays that may have been generated. The pulp is conditioned with an amine, and most of the remaining silica is removed in a cleaner stage, conducted in 300-ft^3 (8.5-m^3) Wemco cells. Final concentrate acid insoluble levels (primarily silica) typically are 4.0 to 5.0%.

WASTE DISPOSAL. Flotation tailings are pumped to a mined-out area according to reclamation plans. Tailings pipelines are high-density plastic. Deposition of the sand tailings in the voids between the overburden spoil piles is the initial phase of reclamation. The water is recovered and added to the recycle system.

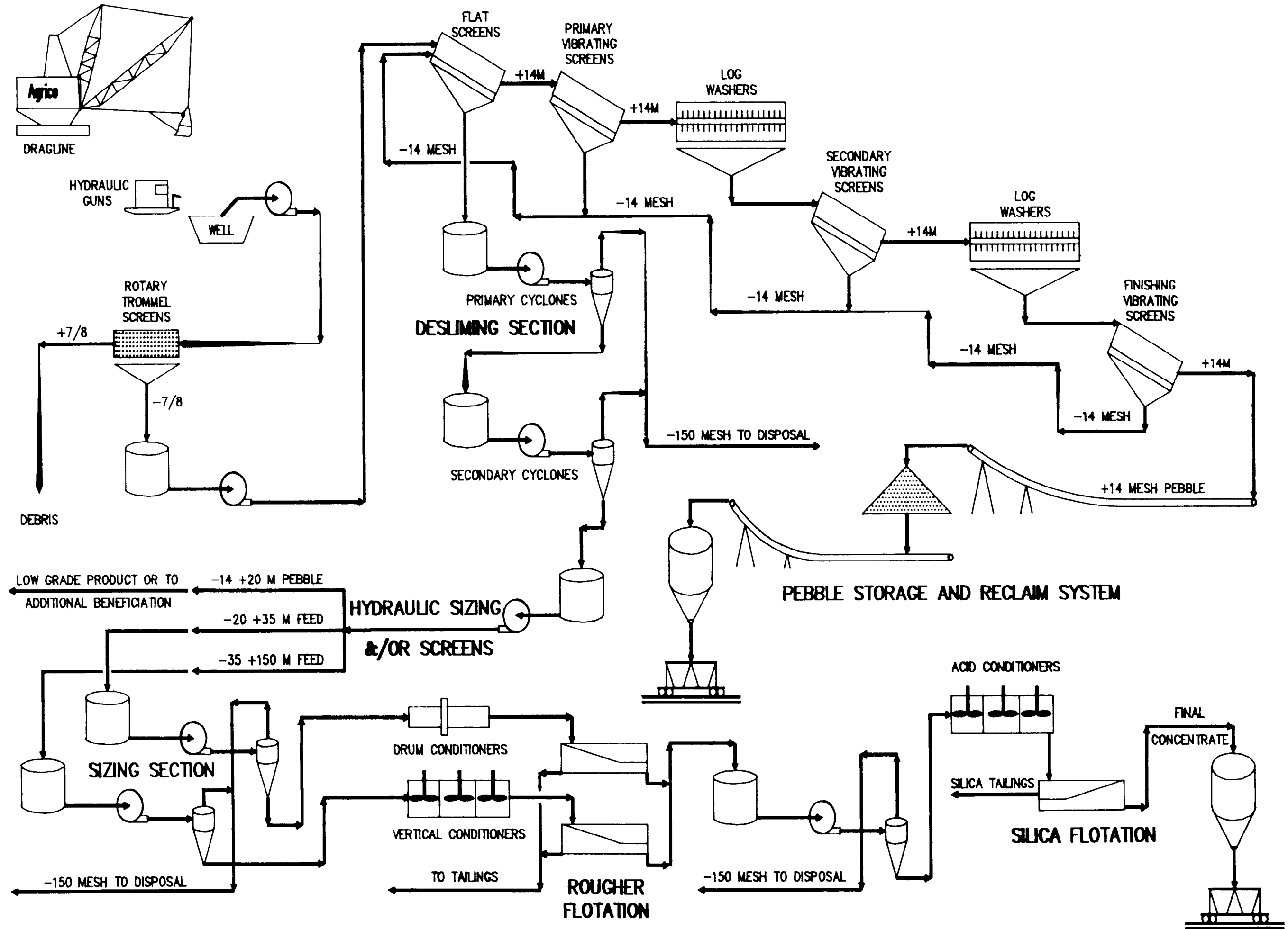

Fig. 14.3.8.5. Mining and beneficiation flowsheet.

The clay fraction is pumped to aboveground disposal areas that have been constructed on mined lands. The clay "ponds" are often 1 mile^2 (2.6 km^2) in size and can be 25 ft (7.6 m) to the crest. These impoundment areas are designed by specially trained soils engineers and are inspected by professionals on a regular basis. They are typically constructed adjacent to one another to permit "flow-through" from one to the next. This construction maximizes clay deposition, minimizes cost, and simplifies water recovery. These clay ponds act not only as clay disposal areas but are used in the management of the recycle water.

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Chapter 14.4
AUGER MINING

M.K. MCCARTER AND HAROLD M. SMOLNIKAR

14.4.0 CONTINUOUS SURFACE MINING WITH AUGERS

M.K. MCCARTER

Auger mining refers to a method for recovering coal from thin seams, 2 to 8 ft (0.6 to 2.4 m) thick, beyond a highwall produced by conventional open cast or strip mining (area mining or contour mining), or from seams accessed by deep trenches. Coal is produced by boring openings into the seam beneath the overburden. It is used primarily where conventional surface or underground methods are not economically or technically feasible. Often this is the only means available for recovering coal that would otherwise be lost (Treuhaft, 1984).

The first augers were placed in service in 1945 in the bituminous coal fields of West Virginia. The earliest augers were adaptations of horizontal rock drills, which were used to produce holes measuring 6 to 9 in. (150 to 220 mm) in diameter (Anon., 1975).

An auger typically consists of three components: (1) cutterhead, (2) auger flight(s), and (3) prime mover. They are mechanically connected and function as an integral power train. The cutterhead is a barrel-shaped device that cuts and fractures the coal. The auger flight is a spiral wound about a drill stem, forming a screw conveyor that serves to transport the coal to the surface. In the case of a single auger, the coal fragments, confined by the sides of the hole, are forced towards the open end of the hole by the turning action of the drill stem.

The cutterhead often consists of more than one bit assembly. Single heads range in diameter from 1.5 to 8 ft (0.5 to 2.5 m), double heads from 1.5 to 3 ft (0.5 to 0.9 m), and triple heads from 1.5 to 3 ft (0.5 to 0.6 m) (Hartman, 1987). The multiple heads cut overlapping patterns, producing an elongated cross section with less coal left at the web. If two cutterheads are employed, each turns in opposite directions forcing the coal towards the center of the flight. This counterrotation stabilizes the assembly and assists boring of straighter holes with less out-of-seam deviation.

The prime mover is usually skid mounted or pad mounted. It is able to traverse uneven train and employs jack-legs to level the carriage. The carriage provides a mounting for the engine, drive head, and controls. The carriage is able to move towards the face in the driving mode and can retreat far enough to insert another flight to extend the hole to greater depth. Individual flights are stored in racks and are handled by hoists attached to the frame. As coal arrives at the surface, it is elevated by conveyor or front-end loader and deposited in waiting trucks.

Recent evolution of the auger has resulted in the thin seam miner (TSM), a form of surface continuous miner. This equipment incorporates a cutterdrum that cuts a rectangular hole 8 ft (2.4 m) wide and up to 5 ft (1.5 m) high. Power to drive the cutterdrum is supplied by hydraulic hoses rather than auger flights. The entire assembly is forced into the coal seam by push beams and rigid box sections which contain a screw conveyor to bring the coal to the surface. The cutterdrum is also equipped with sensors which allow detection of the bottom and the top of the seam.

The mining cycle for the TSM is automated and consists of a sump cut at the base of the seam. The cutterdrum then pivots upward, shearing the coal until it approaches the top of the seam. The cutterdrum is then advanced forward for another cycle. Ten cycles are completed before another 20-ft (6-m) push beam is added. Ten push beams can be used, providing a total depth of 240 ft (73 m). The length of the mining cycle is about three hours (2.5 hr mining, 0.5 hr retracting). Production can be as high as 300 tph (272 t/h) (Chironis, 1984).

At present, auger mining is most commonly practiced in the hilly coal country of Appalachia, primarily Kentucky and West Virginia. This area typically has steep slopes where the overburden increases rapidly and thin seams are common. This method, however, is now being considered for recovering additional reserves at the final highwall of large western mines employing conventional area mining. Future application may also include seam outcrops in rugged terrain where high stripping ratios or environmental concerns would make conventional surface mining unacceptable (Hartman, 1987).

Reserves amenable to augering in the United States have been estimated at about 24 billion tons (22 billion t) for augering depths up to 200 ft (60 m) and about 100 billion tons (90 billion t) for extended depth techniques to 600 ft (180 m) (Hartman, 1987). Optimum recovery will require reducing the amount of top or bottom coal left in place by matching cutterheads to the seam thickness. Techniques will also be needed to reduce or eliminate web (pillar) thickness between holes and to extend augering depth beyond conventional practice.

Web thickness is commonly about 1 ft (0.3 m) for a 2-ft (0.6-m) diameter hole. This spacing results in a 64% loss in coal recovery compared to leaving no web. The loss can be reduced by using corner reamers or expandable backreamers to produce a more nearly square hole (Chironis, 1985). When these devices are used, the web is weakened and provides less support for the highwall. It is a common practice to retract backreamers as the cutting action approaches the surface to provide more support at the immediate highwall. This results in greater stability of the highwall and less likelihood of sloughing or “raveling,” which would endanger personnel and equipment working at the base of the slope.

Development of an augering operation is relatively straightforward. Initial development drilling and experience derived through conventional mining is usually sufficient for establishing the continuity of the seam. Continuity, uniform seam thickness, and near-horizontal orientation (less than 10°) are essential for this technique. Once the continuity is established, the face is prepared by grooming the highwall to remove loose rocks and other hazards which would compromise the safety of operations. The pit floor is then leveled and cleaned to facilitate alignment of the prime mover and auger flights. A roadway parallel to the highwall is also needed for access, beltways, and/or truck transportation of the product. Generally, a clear area 50 to 75 ft (15 to 25 m) wide is all that is needed for efficient operation (Hartman, 1987).

Ohio and Kentucky mining regulations require specific procedures for reclaiming land mined by augers. In Ohio, holes are to be sealed with clay or other impervious material and covered

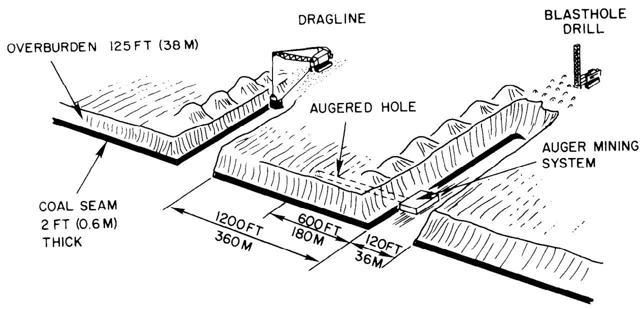

Fig. 14.4.0.1. Auger hole patterns and effect of highwall curvature (Treuhaft, 1984).

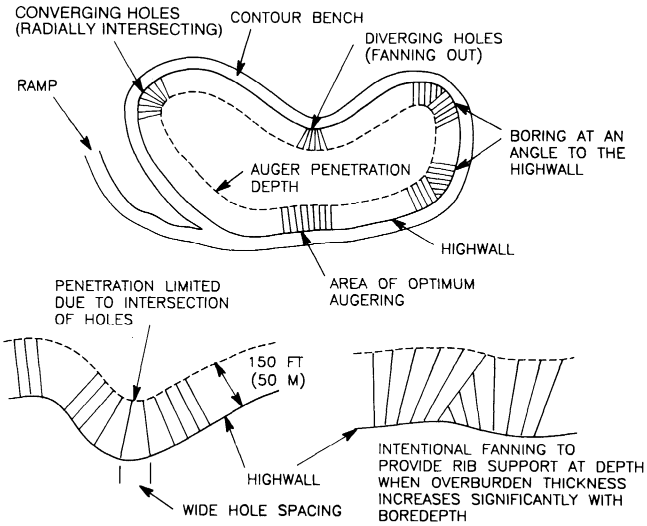

Fig. 14.4.0.2. Trench-strip mining with the extended depth highwall mining system (Treuhaft, 1984).

within 48 hours after removing the auger. In Kentucky, the highwall must be reduced to no more than 45°, and backfilling and grading must be completed within 15 days of augering (Anon., 1975). In addition, Kentucky requires that bench width on steep mountain slopes be no wider than 35 ft (11 m) (Anon., 1974).

Operations are usually low cost and highly productive, averaging 25 to 500 tons (22 to 450 t)/employee-shift. The size of the crew can be as few as three or four individuals producing 100 to 2500 tons (90 to 2200 t)/shift. Coal recovery, however, may be as low as 40 to 60% (Reilly, 1968).

Recovery can be further compromised by the shape of the highwall. If the holes are spaced far apart, coal will be lost between the holes. If they are close together, the ribs may be too thin to support the overburden. Maximum recovery is achieved with straight highwalls while inside and outside curves require fanning of the holes resulting in sterilization of reserves between holes. Clearly, careful mine planning is needed to optimize the shape of the final highwall before augering begins, and appropriate patterns for the auger holes are required to provide maximum resource recovery. Fig. 14.4.0.1 illustrates the effect of highwall curvature on the spacing of holes and strategy for various hole patterns.

A method which may be applicable in relatively flat topography and deep seams is referred to as trench-strip auger mining (Treuhaft, 1984). This concept requires excavation of a series of parallel trenches as shown in Fig. 14.4.0.2. The coal between the trenches can then be removed with augers working at right angles to the trench. The straight configuration of the highwall should maximize recovery and minimize the surface area requiring reclamation. It is estimated that less than 10% of the mined land will be disturbed if this method is employed, but the amount of waste to be handled will be reduced only by about 50% compared to conventional area mining. At this time, the method has not been proven but may become attractive as extended depth augering is perfected and demand requires exploitation of deeper reserves in more environmentally sensitive areas.

Auger mining can be implemented with relatively low capital investments, it is safer than underground mining, and more tons per employee-day can be produced than for either surface or underground methods. Productivity decreases as the depth of auger hole increases. Augered coal often contains less ash than surface-mined coal from the same seam. However, once the cutterhead enters the seam, the operator is not able to observe the cutting action and must rely on the "feel" of the machine to sense problems at depth. Because of this, augering is a highly specialized skill, and experience is essential for successful operation.

Reclamation laws are having an impact on the application of auger mining. The requirement to immediately cover exposed highwalls encourages abandonment of reserves beyond the current economic limit of conventional mining if auger or TSMs are not immediately available. The requirement to restore highwalls to the approximate original topography has also made contour mining less attractive; consequently, fewer sites are available for augering. In addition, it is easier to obtain a variance from the approximate contour requirement if the entire mountain top is removed. In the mountainous areas of Appalachia, this method is preferred, especially if the resulting flat area can be used for agricultural, industrial, commercial, residential, or recreational purposes. As a result, few new augers have been built recently, and the trend is to lease older units as the need arises (Chironis, 1985).

The description of auger mining presented in Chapter 14.4.1, describes current operations at the Winifrede mine.

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14.4.1 SURFACE CONTINUOUS MINING, HORIZONTAL COAL SEAMS: WINIFREDE MINE

HAROLD M. SMOLNIKAR

14.4.1.1 Mine Description

Arch of Kentucky's Winifrede mine is located in Harlan and Letcher Counties, approximately one mile (1.6 km) north of the city of Lynch, KY. The mine lies within a 40,000-acre (162-km^2) tract, formerly owned by US Steel Corp. The bed being auger mined is named the Winifrede Seam. Extensive underground mining has taken place in the seam, and considerable surface contour mining was performed prior to enactment of the Surface Mining and Reclamation Control Act of 1978. Approximately 70,000 ft (21,340 m) of old highwall were available for augering, with an estimated in-place reserve of over 1,200,000 tons (1,090,000 t).

The annual production is 110,000 tons/year (99,790 tpy) of coal. The mine is scheduled to run 1 shift/day, 220 days/year at a production rate of 5090 tons/shift (4618 t/shift).

14.4.1.2 Deposit Description and Geology

The Benham and Appalachia USGS Geologic Quadrangle Map identifies the Winifrede Seam as being in the Hignite Formation of the Pennsylvanian Breathitt Group. The overall dip of the bed is about 1° to the southeast, but locally there are small rolls and folds that make local dips highly variable. The seam

Fig. 14.4.1.1. Typical sediment control structure on an auger bench.

elevation ranges from 2640 to 2907 ft (805 to 886 m) AMSL. The maximum topographic elevation in the area being mined is approximately 3600 ft (1097 m) AMSL. The area is typical of the Appalachian Mountain Region with steep, high, densely timbered ridges and narrow meandering valleys.

The seam thickness ranges from 43 to 70 in. (1092 to 1778 mm). The coal is classified as high-volatile A bituminous. The quality attributes of the coal are as follows: 13,000 BTU/lb (30,238 kJ/kg), 7.0% ash, 0.9% sulfur, and 6.0% moisture. The Hargrove Grindability Index is 53. There are 500,000 tons (453,590 t) of coal recoverable from the seam by auger mining.

The predominant highwall material is a hard, massive, gray sandstone. In some areas, the overburden immediately over the coal is a gray calcareous shale about 2 ft (0.6 m) thick. Where this is absent, the immediate roof material is sandstone. The floor material of the Winifrede Seam is a dark gray, fissile shale that grades into a siltstone and, finally, a fine-grained sandstone.

14.4.1.3 Mine Development

The initial preparation work for augering the Winifrede Seam consists of cleaning a few hundred feet (meters) of the old bench at a time, grading the haul road to an acceptable condition, and constructing sediment control structures. The cleanup of spoil material on the old bench is required to obtain sufficient bench width for operating room, and to remove any loose overburden that had previously been pushed over the seam. The preferred minimum bench width is 50 ft (15 m), with 42 ft (13 m) being an absolute minimum. Sediment control ponds, berms on the outside of the old bench, and ditches must be constructed prior to mining to catch any surface run-off from the auger area.

The sediment control ponds are built on the outside edge of the bench so that any clarified water overflow can be diverted down the slope of the mountain. Shallow ditches are constructed on the inside of the haul road as required to convey any surface water to the ponds. Culverts measuring 12 in. (305 mm) in diameter are placed wherever necessary to carry water under the haul road to the ponds. Fig. 14.4.1.1 shows a typical sediment control structure on the auger bench.

The loose spoil material cleaned up during the development work is either stockpiled along the outer edge of the bench, if there is room, or it is hauled to a wider bench area for storage. A D-7 bulldozer is used to clean the coal face, and a Caterpillar 988 front-end loader (FEL) hauls the material to the edge of the bench or loads it into a 35-ton Cat 769 haul truck.

Fig. 14.4.1.2. Coal auger.

After a few hundred feet (meters) of bench have been cleaned and properly developed, the auger is hauled to the site, assembled, and initially maneuvered into location by the bulldozer. When positioned approximately 5 ft (1.5 m) from and perpendicular to the highwall, the auger is ready to begin operating. Fig. 14.4.1.2 shows the auger set up and ready for operation.

Table 14.4.1.1. Specifications for Coal Auger

Front highwall guards	None	
Carriage overhang	29.5 ft	(9.0 m)
Operating length—Minimum	31.0 ft	(9.4 m)
Maximum	16.3 ft	(5.0 m)
Width (less auger racks, pan, conveyor, and skids)	18.0 ft	(5.5 m)
Width (overall)	31.5 ft	(9.6 m)
Height (with stiff-leg lowered)	16.3 ft	(5.0 m)
Height (with hoist removed)	11.3 ft	(3.4 m)
Auger length	12.0 ft	(3.7 m)
Auger diameters (dual MCD)	18-30 in.	(457–762 mm)
Primary power (maximum hp @2000 rpm)	510 hp	(380 kW)
Auxiliary power (maximum brake hp 2200 rpm)	200 hp	(149 kW)
Carriage thrust—push	32,500 lb	(144,560 N)
pull	15,900 lb	(70,723 N)
Distance between bottom of auger and bottom of skid	5 in.	(127 mm)
Leveling jacks—attached to skids	4-ft lift	(1.2-m lift)
Auger guide extension	30 in	(762 mm)
Skid cylinder stroke (maximum lateral movement/stroke)	7 ft	(2.1 m)
Estimate weight of machine (less augers and cutterheads)	112,000 lb	(498,176 N)

Notes: Gear train—combination 3-speed transmission (2 forward, 1 reverse) divider box driving adjustable swing type drop boxes for all auger sizes (single, dual, or tri-head) 18 in. (457 mm) and larger. Stiff-legged Mul-T-Action hoist handles one, two or three augers at once. Pendant control. Pan conveyor attached. Easily adaptable to discharge from either side.

14.4.1.4 Operations

AUGERING. Two Salem augers with different sized cutterheads and auger flights are used intermittently, depending on seam height and quality problems. One machine has a dual 22-in. (559-mm) cutterhead; the other has a dual 27-in. (686-mm) cutterhead. Each machine is equipped identically, except for the size of cutting equipment. Table 14.4.1.1 contains a list of the basic machine specifications.

A three-man crew operates the auger. One man controls the auger position along the highwall and the auger advance or retrieval in the seam. The second man is responsible for handling of the auger flights. A third man, a mechanic, is available as needed to provide fuel, lubrication, and maintenance.

Vertical control of the auger is accomplished through the use of four hydraulic jacks, one located at each corner of the machine on the main frame. The machine is raised to position, with the cutterhead approximately 6 in. (152 mm) below the top of the zone to be mined. Usually, this is 6 in. (152 mm) below the top of the seam. Care is taken to differentially level the machine to try to parallel the plane of the coal seam in order to avoid out-of-seam dilution. The dual cutterhead consists of the cutting unit, which is equipped with tungsten carbide-tipped bits, and the auger scroll to convey the coal away. The auger scroll is banded to minimize deflection of the steel. Raker bits between cutterheads cut the interior rib left between the rotating heads. Counter rotation of the individual auger tool strings assists in minimizing hole deflection. Rotation of the auger flights drags the coal out of the hole to an elevated chain stacking conveyor.

The auger is advanced into the coal by a combination of cutting action by the cutterhead and forward thrust by the hydraulically-powered thrust ram. When the 12-ft (3.7-m) flight section is completely advanced, the hydraulic push ram is disengaged and pulled back. Then another pair of auger flights is inserted, locking pins are hydraulically activated, and augering resumes again. Insertion of auger flights is completely mechanized, with one man using an overhead-mounted hoist to grip, lift, and spot two 12-ft (3.7-m) auger flights simultaneously. The rigid hoist mechanism has tongs that clamp each auger flight at each end, allowing exact positioning of the flights. This cycle of auger boring, advance, and insertion of auger flights continues until the auger reaches its maximum depth, encounters old mine workings, or cuts into the roof or floor resulting in excessive out-of-seam dilution. At this point, retrieval of the string of auger flights commences.

On retrieval of the stem of auger flights, the auger operator pulls the auger sections back and actuates a hydraulic uncoupler to unlock a set of flights from the tool string still in the hole. The pair of 12-ft (3.7-m) long flights are then picked up by the overhead hoist and stacked on a rack located on each side of the machine.

Upon retrieval of all auger flights from the hole, the operator engages the hydraulic jacks located on the corners of the machine to lower the machine into proper position for augering a second pair of holes. After proper leveling is completed, the augering cycle is repeated. After the upper and lower pair of auger holes are finished and all the steel withdrawn from the holes, the operator lowers the machine and prepares to move the auger parallel to the highwall to the next augering location.

Movement to the next location is achieved by activation of the hydraulically powered moving skid. Under each pair of leveling jacks, there is a moving skid operated by a double-acting hydraulic cylinder. When the leveling jacks are extended, the moving skid is advanced in the intended direction of travel. The leveling jacks are then retracted with the machine coming to rest on the moving skid; it is then hydraulically advanced on the skid. The maximum lateral machine movement per cylinder stroke is 7 ft (2.1 m). The machine is thus repositioned on the highwall so that a pillar or “web” of coal 6 to 12 in. (152 to 304 mm) wide is left between sets of holes. Fig. 14.4.1.3 shows the hole configuration and remnant web or coal pillar.

Fig. 14.4.1.3. Hole configuration and remnant web.

The maximum penetration the machine is capable of achieving is 150 ft (46 m). The average penetration experienced is 120 ft (37 m). The most common problem causing a set of auger holes to fall short of the maximum depth is augering into the roof or floor. This is generally caused by improper machine leveling, the coal rolling, or drifting of the auger flights. The average production rate achieved is 490 tons/shift (445 t/shift).

GROUND CONTROL. The pillar size or web between the auger holes varies throughout the operation and is largely determined by the judgment of the auger operator. The factors taken into consideration by the operator when determining the width of pillar are the following.

Change in Seam Thickness—As the seam thickens, the pillar height increases and subsequent life decreases unless pillar width is increased.

Increase in Overburden—The higher the highwall, the thicker the pillar must be.

Fractures or Jointing in the Highwall—If these features are noticeable in the highwall, the operator may try to change the angle of augering to allow the auger to penetrate in a perpendicular direction to the features, or if conditions are hazardous, he may choose to skip the area entirely. If the highwall is not fractured but the coal is, the operator may increase the width of pillars.

Soft Floor—The problem of a weak floor can result in pillars "punching" into the floor. This causes a redistribution of pressure along the highwall and into the augering area. Failure of the highwall or trapping of the auger flights can result from this pressure redistribution. By increasing pillar width, the time until failure occurs can be increased to allow for the safe recovery of coal.

Intersecting Underground Works—The intersecting of old works results in loss of support or abutment pillars to support the highwall or old mine entries. The result can be highwall failure or a trapped auger. It should be noted that whenever the auger removes coal in areas that are less than 50 ft (15 m) from old underground mine works, there is an increased incidence of highwall failure. This mandates an increase in web thickness.

Intersection of Auger Holes—This results in a reduction in size of the supporting pillar and may cause failure of the highwall. This most frequently occurs when working on a point at the end of a ridge.

HAULAGE. The coal discharged from the auger holes is dropped onto a short chain conveyor on the auger machine. This chain conveyor discharges the coal onto an elevating chain stacking conveyor that drops the coal onto a stockpile next to the auger machine. The coal is loaded from the stockpile with a Cat 988 FEL into dual-axle coal haul trucks with modified beds that accommodate 38 tons (34.4 t). Two trucks are normally required to haul the coal to the stockpile. Occasionally, the haul distance requires the addition of a third truck.

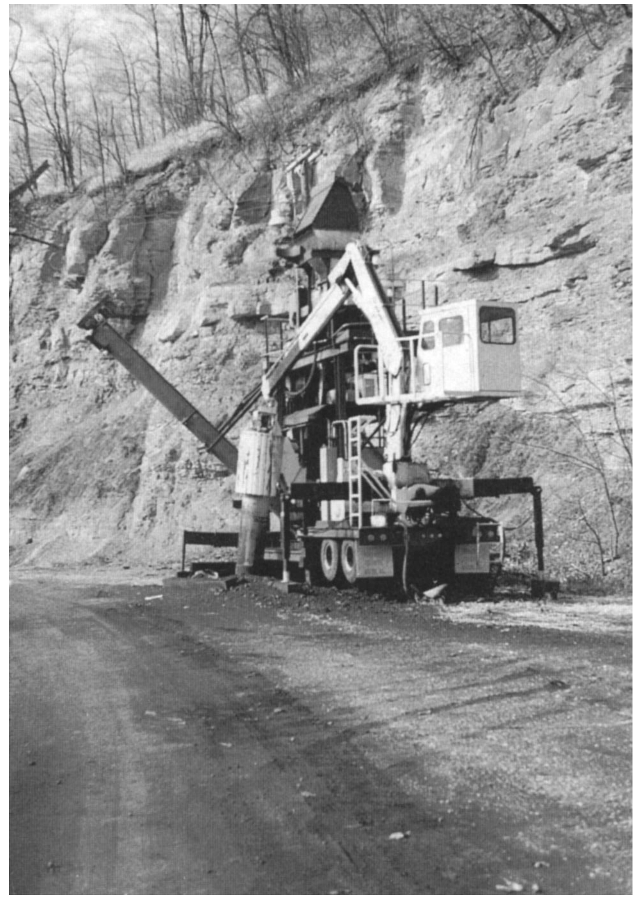

Fig. 14.4.1.4. Truck sampler.

The auger coal is pushed onto a stockpile that has a capacity of 12,000 tons (10,900 t). When coal is withdrawn from the stockpile, it is pushed by a dozer to a chute that discharges onto a belt measuring 42 in. (1067 mm) wide. This belt transports the coal approximately 120 ft (37 m) to a hammer crusher, which crushes the coal to a maximum size of 2 in. (50 mm). The crushed coal is discharged onto another 42-in. (1067-mm) belt that carries it to a 1000-ton (907-t) surge bin. The coal is then conveyed to a 500-ton (454-t) surge bin on top of a 5-ft (1.5-m) diameter, 740-ft (226-m) vertical borehole. The coal drops through the borehole to an old mine entry where a vibratory feeder meters the coal onto a 36-in. (914-mm) conveyor. The coal is then transported to a 2000-ton (1814-t) silo. From there, the coal is transported by a 60-in. (1524-mm) conveyor to the Lynch No. 1 loadout, where it is loaded into rail cars.

PRODUCT CONTROL. When haulage trucks arrive at the stockpile area on the Winifrede bench, each load is sampled by a truck sampler. The coal samples are analyzed daily by a commercial laboratory. If any problems are encountered, the augering method can be changed immediately. Fig. 14.4.1.4 is a photograph of the truck sampler.

14.4.1.5 Reclamation

Initial reclamation during auger mining has to be kept within 1500 ft (457 m) of the auger's location. The initial reclamation consists of covering the coal outcrop with a minimum of 4 ft (1.2 m) of spoil material, which must be non-combustible, non-acid forming, and non-toxic. Any permanent roads, ditches, or ponds also are graded to final configuration. The backfill material is graded to a 2:1 slope, and soil samples are taken to determine what amounts, if any, of fertilizer are required to support plant growth.

Seeding can be conducted in either the fall or the spring. Spring, however, is the preferred planting time. A fertilizer and grass seed mixture are hydroseeded with a wood-cellulose mulch material. The seed mixture used is 25 lb/acre (2800 kg/ km^2) Kentucky 31 Fescue, 20 lb/acre (2200 kg/km^2) of Perennial Rye, and 10 lb/acre (1100 kg/km^2) of Bermuda Grass. Fig. 14.4.1.5 illustrates a typical "pre-law" reclaimed auger bench.

After initial regrading and revegetation are completed, the areas are monitored for erosion and vegetation growth. In the event that erosion results in rills and gullies deeper than 9 in. (229 mm), the areas must be filled, regraded, and replanted.

After an area has been regraded and revegetated according to the mining permit, the area is inspected by the regulatory agency. If the area is approved, a partial bond release on the regrading can be requested. Bond money for revegetation is held for five years to ensure that good vegetative growth has been achieved.

Fig. 14.4.1.5. Typical reclaimed auger bench.

Chapter 15.2
SOLUTION MINING: SURFACE TECHNIQUES

W.J. SCHLITT

15.2.1 INTRODUCTION

Although commonly referred to as a type of solution mining, *surface leaching* techniques are really hydrometallurgical recovery operations practiced at the mine. The basis for this statement is the fact that these leaching systems must be used in conjunction with conventional open pit or underground mining operations. Mining provides the ore feed. This broken material is then leached in either run-of-mine or crushed form to extract the values. Surface solution mining practices thus represent alternatives to conventional ore beneficiation.

This is in sharp contrast to in situ operations that represent a true mining technique. Note that *in situ leaching* removes values from in-place deposits that are essentially undisturbed by conventional mining technology. Thus in situ leaching is an alternative to mechanical mining operations, while surface leaching is an adjunct to them. As a result of this distinction, in situ leaching is applicable to a wider variety of resources, including soluble salt and sulfur deposits, as well as hard-rock ores, such as those of copper and uranium. On the other hand, surface leaching systems have only been applied to hard-rock ores, mainly copper and gold/silver, and to a lesser extent, uranium.

Another distinction between the in situ and surface techniques lies in the relative importance that each has to the overall operation. In virtually all cases, in situ leaching is the *primary method of production.* However, surface leaching may be either the primary production option or only a *secondary,* or supplemental, *form of production.* A good example of the primary option is the mine-heap leach type of operation that has become so popular in the gold industry.

Adjunct or secondary leaching operations are most prevalent in the copper industry. Here total exploitation of the resource requires that leaching be applied to low-grade materials that must be mined in order to expose the mill-grade ore. In these cases, leaching will provide some additional production that would not be achieved otherwise. Such supplemental production often has a low incremental cost so that it lowers the overall average cost of production.

The wide variations in ore deposits make leaching a rather site-specific endeavor. To be successful, a *heap* or *dump leach* operation must satisfy two general sets of criteria. One relates to the characteristics of the ore or waste to be leached. The other involves the establishment of effective operating practices.

Clearly, first consideration must be given to the characteristics of the deposit and the leach material itself. These parameters include the location of the ore body, the extent of the mineral inventory, the mineralogy and mode of occurrence of the metal values, and the chemical and physical nature of the host rock. Obviously, nature must provide a sufficient tonnage of material with adequate recoverable values per ton. Otherwise, the leach project will never reach the point where operating practices are of any consequence. For a proposed operation, the same type of exploration and development program will be required as for any other mining project. These programs are covered elsewhere in this *Handbook.* The difference is that the metallurgical testing will be directed toward leaching instead of ore comminution and beneficiation.

The remaining portions of this chapter deal with the establishment of effective surface leaching operations. The first is a rather fundamental segment that covers ore–lixiviant systems. This is followed by two more practical parts dealing with materials handling and solution management. Next are two short segments that address commodity recovery systems and ancillary and infrastructure requirements. The final segment includes two case studies, one for copper and one for gold.

15.2.2 ORE LIXIVIANT SYSTEMS

The first consideration in any proposed leaching operation is the selection of the *lixiviant.* If a suitable lixiviant cannot be identified, then the project is not viable and no further effort need be expended on it. Ideally, the lixiviant should be highly selective and should attack the value(s) rapidly and completely. The chemical kinetics should be so fast that diffusion to or from the mineral surface is always the rate-controlling step in the process. Various actions can then be taken to enhance diffusion. Other characteristics of the ideal lixiviant include a low price, low toxicity, ease of handling, and a capacity for regeneration or recycling. The lixiviant should allow for ready recovery of the value(s) in concentrated form or even in the refined state.

Unfortunately, ideal lixiviants do not exist; practical considerations then dictate that the lixiviant be fast and selective in solubilizing the values. In addition, the cost should be low, net of any credits for recycling or regeneration of the reagents. Then the necessary environmental controls and handling and recovery systems can be designed to suit the operation.

There are a surprisingly small number of practical lixiviants for the metal values recovered in heap and dump leach operations These are reviewed in the following.

15.2.2.1 Gold/Silver

An alkaline sodium cyanide solution is virtually the only commercial lixiviant presently used in gold/silver heap leach operations. Studies on the mechanism of dissolution indicate that there are two reactions involved in gold dissolution (Dorey et al., 1988). Most of the gold dissolves according to the reaction,

$$2\ Au + 4\ CN^- + O_2 + 2\ H_2O \rightarrow 2\ Au(CN)_2^- + H_2O_2 + 2\ OH^- \qquad (15.2.1)$$

However, a small but still significant portion of the gold dissolves according to the Elsner's reaction:

$$4\ Au + 8\ CN^- + O_2 + H_2O \rightarrow 4\ Au(CN)_2^- + 4\ OH^- \qquad (15.2.2)$$

In both cases, the balancing cation is sodium, and oxygen is required. For any given ore, the gold dissolution rate is dependent on the cyanide concentration, the oxygen content, and the alkalinity of the solution. The optimum pH is about 10.3 (Dorey et al., 1988), but many operators tend to run at a somewhat higher value as a margin of safety. Since most ores are not so basic, an alkaline reagent must also be added to the system to raise the pH. This is generally done by blending lime (CaO) with the ore, or adding milk of lime [(Ca $(OH)_2$] or caustic (NaOH) to the solution.

In theory, there is nothing to prevent dissolution of virtually all of the gold in a heap or dump. In actual operations, extraction seldom exceeds 85%, and recoveries as low as 40 to 50% are not unheard of. The problem is accessibility of the gold values to the lixiviant. This can be caused by poor heap construction, which leaves portions of the ore blinded off. As a result, solutions do not wet the entire rock mass. Another cause is gold encapsulated in silica or locked in another refractory mineral such as pyrite or arsenopyrite. The processing of these so-called refractory ores is an area of technology that is undergoing significant development at the time of this writing. This is due to gradual exhaustion of accessible oxidized ores that are easily processed.

Although commonly extracted along with the gold, silver is not recovered as effectively. The dissolution reactions with cyanide are analogous, but there is not as great a chemical driving force involved. In addition, silver is much less noble than gold and tends to associate with various refractory minerals, in addition to pyrite and arsenopyrite.

Particularly troublesome are the manganiferrous silver ores that are virtually insoluble in cyanide. These ores can be processed in two stages. The first involves an aqueous sulfur dioxide leach to break down the manganese structure by converting it to soluble manganese sulfate. This liberates the silver so that it can be extracted in a cyanide leach. Unfortunately, the first step is actually a sulfurous acid leach that would necessitate a thorough neutralizing wash step prior to cyanidation. Hence, little silver has even been recovered from such ores.

A number of other precious metal lixiviants have been investigated as alternatives to sodium cyanide. The impetus is to find a lixiviant that overcomes some of the problems inherent with cyanide. These include the following:

1. High potential toxicity.
2. Environmental concerns such as aquifer contamination.
3. Relatively slow kinetics for dissolution of both gold and silver.
4. Generally poor extraction of silver.
5. Requirement for alkaline reagents when leaching acidic ores.
6. High reagent consumption due to poor selectivity when leaching complex ores containing various base metals. Particularly troublesome is copper, which often leaches well as it forms a highly soluble cyanide complex. In addition, the copper usually reports with average gold and silver as they are stripped from the leach solution. The presence of copper then complicates the final parting and refining of the precious metals.

Some progress has been made on alternatives to a straightforward cyanide leach. However, success has been largely confined to the treatment of high-grade ores and concentrates, rather than the low-grade materials likely to be leached in heap or dumps. One of these routes involves the extraction of silver using a brine leach on the concentrates. Another reagent with more promise is thiourea [$(CS(NH_2)_2$]. Hiskey (1981, 1984) among others has provided comprehensive reviews of this subject, and Charley (1984) has described development of a thiourea process for treating high-grade materials. The chemistry involved in thiourea leaching requires that the gold be oxidized from the free-metal zero state to the +1 oxidation state. In an acid solution, this can be done with an external oxidant like ferric iron or by oxidation of thiourea to produce formamidine disulfide (RSSR) by the reaction,

$$2\ CS(NH_2)_2 \rightarrow RSSR + 2\ H^+ + 2\ e^- \qquad (15.2.3)$$

Thus the overall reaction for gold becomes

$$2\ Au + 2\ CS(NH_2)_2 + RSSR + 2H^+ \rightarrow 2\ Au\ [CS(NH_2)_2]^+_2 \qquad (15.2.4)$$

Silver undergoes analogous reactions, except the final species involves three thiourea molecules: $Ag[CS(NH_2)_2]_3+$.

Care must be taken to avoid too high an oxidation potential in the leach circuit as this will cause irreversible oxidation of RSSR to elemental sulfur. This not only causes loss of reagent but generates sulfur that can coat the mineral surfaces and slow further dissolution.

The continued interest in thiourea leaching is generated by the following factors:

1. Thiourea leaches gold and silver 10 to 12 times faster than cyanide, implying a leach cycle of hours rather than days or months.
2. Thiourea is functional in an acidic rather than an alkaline medium.
3. Thiourea has a better selectivity for gold and silver than cyanide.
4. Thiourea has a lower toxicity than cyanide.

Ferric-sulfate-stabilized thiourea represents a potential improvement over use of thiourea alone for gold leaching. Huyhua et al. (1989) showed that leach times for gold and silver with ferric sulfate as an oxidant were up to four times faster than thiourea alone in air. Thiourea consumption increased only slightly due to the presence of the ferric oxidant; the increased rates and ultimate extractions obtained offset this cost. McInnes et al. (1989) reported the use of thiourea with ferric ion in conjunction with an acid ferric preleach to remove copper from a gold-bearing copper oxide/sulfide ore. This is a compatible system for gold recovery in an acidic, sulfide environment (pH = 1.5) and showed faster leach times than other types of oxidants (H_2O_2,O_2, or air). There is the promise that this approach could provide an acidic gold leaching system to be used with bio-oxidation or other chemical pretreatments in heap leaching ores with sulfides or cyanide consumers.

In spite of these advantages, thiourea has only been used commercially on concentrates and has not yet been applied to low-grade ores in heap or dump leaching. The main problems are high unit costs and consumption of thiourea.

15.2.2.2 Copper

The choice of a lixiviant for copper is dictated by the mineralization to be leached. The oxidized ores, including oxides, silicates, carbonates, hydroxides, and chlorides, can be dissolved effectively with nonoxidizing acids or a base like ammonia that forms soluble copper complexes. However, the unoxidized ores, mainly the various sulfide minerals and native copper, do require an oxidant. The oxidant may be either dissolved oxygen or an ionic species such as ferric iron in a sulfate-based system or ferric or cupric ions in a chloride system. Regardless of the lixiviant, the leach solution must be either acid enough or basic enough to carry copper in solution as copper salts tend to precipitate from near neutral solutions.

Sulfuric acid is the overwhelming choice as the lixiviant for oxide ores. The chemistry involves a straightforward dissolution reaction, such as,

$$CuCO_3Cu(OH)_2 + 2\ H_2SO_4 \rightarrow 2\ CuSO_4 + 3\ H_2O + CO_2 \qquad (15.2.5)$$

for malachite. The advantages of using sulfuric acid include the following:

1. Relatively benign with regard to materials of construction.
2. Compatible with recovery of copper by conventional electrowinning from concentrated sulfuric acid-copper sulfate (electrolyte) solutions.
3. Inexpensive and commonly available as a byproduct from copper smelters that are often nearby.

The only problem with sulfuric acid is that it must be applied in dilute form (pH 1 or higher). This means that heaps must be relatively shallow in order to prevent complete consumption (neutralization) of acid and reprecipitation of copper within the rock mass. Use of high acid strengths in short-term vat or agitation leaches is not a problem and does promote rapid and complete copper dissolution. However, in heaps and dumps, a high acid content would exacerbate acid attack on the gangue constituents. This will raise acid consumption per unit of copper recovered. In addition, the dissolved gangue species such as silica and calcium can lead to serious problems like gypsum precipitation in the copper recovery process. Acid decrepitation of the rock may also generate clays and fines that cause permeability or copper absorption problems during long-term dump or heap leaching operations.

Other mineral acids such as hydrochloric and nitric are more aggressive in terms of copper dissolution and have been proposed for hydrometallurgical treatment of concentrates. However, such acids are not likely to be used on heaps or dumps due to their higher costs and corrosiveness, and to their poorer compatibility with common methods of copper recovery.

The heap or dump leaching of sulfide or mixed oxide-sulfide ores is much more complex than the leaching of oxide ores due to the need for an oxidant. Virtually all operations rely on natural oxidation processes to accomplish the leaching. In such a system, ferric iron is the actual lixiviant for copper. It will attack all the common copper sulfide minerals, including chalcopyrite. However, in a sulfate-based leach solution, chalcopyrite is much more refractory than the secondary minerals such as chalcocite, covelite, or bornite. A typical ferric leach is represented by the reaction with chalcopyrite:

$$4\ Fe^{3+} + CuFeS_2 \rightarrow Cu^{+2} + 5\ Fe^{+2} + 2\ S^{\circ} \qquad (15.2.6)$$

Here the balancing anion is sulfate. Although small amounts of elemental sulfur are detected on partially leached mineral surfaces, there is certainly no elemental sulfur accumulation observed in actual operations. This shows that any elemental sulfur is transitory and is progressively oxidized all the way to sulfate in the presence of bacterial catalysts and oxygen in the air within the rock pile. The overall reaction is

$$S^{\circ} + H_2O + 3/2\ O_2 \rightarrow H_2SO_4 \qquad (15.2.7)$$

The iron that acts as the oxidant is itself derived from the oxidation of pyrite, which is typically three to ten times more abundant than the copper minerals. Pyrite oxidation (Hiskey and Schlitt, 1982) is analogous to chalcopyrite oxidation. However, pyrite oxidation serves to do more than provide iron in solution. It is also the net in situ source of the sulfuric acid needed to maintain a low enough pH to keep the copper and iron in solution. Finally, pyrite oxidation is strongly exothermic and causes the temperature of the rock mass to rise. This enhances the intrinsic copper leach rates. But more importantly, the heat provides the driving force for the convective flow of air through the heap or dump. Thus pyrite oxidation stimulates air convection and provides the oxygen that is ultimate oxidant in the system.

The leaching of sulfide copper is clearly complex and cannot be described in detail here. For this, the interested reader is referred to the papers of Cathles and Apps (1975) and Cathles and Schlitt (1980).

Native copper can be leached in a ferric sulfate solution like the sulfides. However, ammonia leaching also has been used commercially on native copper ores and tailings. According to Chase (1980), the preferred reagent is cupric ammonium carbonate. The reaction requires the presence of excess ammonia; the cupric complex is then regenerated in the presence of oxygen. These reactions are

$$Cu^{\circ} + Cu(NH_3)_4CO_3 \rightarrow Cu_2(NH_3)_4CO_3 \qquad (15.2.8)$$

$$Cu_2(NH_3)_4CO_3 + 1/2\ O_2 + (NH_4)_2CO_3 \rightarrow 2\ Cu(NH_3)_4CO_3 + 3\ H_2O \qquad (15.2.9)$$

Copper can then be recovered by steaming the pregnant leach solution to precipitate the copper as a basic salt while recovering the ammonia. Due to the scarcity of native copper ores, further information is not presented here (but see Chase, 1980, for details).

15.2.2.3 Uranium

Commercial heap or dump leach operations treating primary uranium ores are not common. One example is the Los Gigantes sulfuric acid heap leach operation in Cordoba, Argentina. Byproduct uranium has also been recovered from the copper sulfide dump leaching operation at Bingham Canyon, UT, and the Anamax copper oxide leaching plant in Arizona. However, substantial amounts of uranium are recovered by in situ leaching of sandstone deposits (see 15.3.3 in the following chapter, Solution Mining: In Situ Techniques).

Regardless of the method of leaching, the same lixiviant chemistry is applicable to the uraninite (UO_2) and coffinite [U(SiO$_4$)l-X(OH)$_4$X] ores that are commonly treated. As discussed by Grunig (1981), there is a choice of lixiviants on both the acidic and basic side. In order to minimize reagent losses in solution mining, these lixiviants have been tailored to avoid excessive gangue reactions. Thus a sulfuric acid-ferric sulfate system was developed for sandstone ore and a sodium bicarbonate or ammonium carbonate leach for uraninite in calcerous sandstone. The oxidant in the latter system is either hydrogen peroxide or oxygen dissolved in the leach solution.

In both cases, dissolution requires the oxidation of quadrivalent uranium to the hexavalent state. In dilute sulfuric acid (pH 1.6 to 2.0), uranium is oxidized by ferric ion and goes into solution as the very stable uranyl sulfate complex. Similarly, sodium or ammonium bicarbonate lixiviants form uranyl carbonate. Based on information presented by Merritt (1971), the overall reactions in an acid circuit are

$$UO_2 + Fe_2(SO_4)_3 \rightarrow UO_2SO_4 + 2\ FeSO_4 \qquad (15.2.10)$$

$$UO_2SO_4 + 2H_2SO_4 \rightarrow [UO_2(SO_4)_3]^{4-} + 4H^+ \qquad (15.2.11)$$

and in an alkaline circuit the reactions are

$$2UO_2 + 2H_2O_2 \rightarrow 2UO_3 + 2H_2O \quad (15.2.12)$$

$$UO_3 + Na_2CO_3 + 2NaHCO_3 \rightarrow Na_4UO_2(CO_3)_3 + H_2O \quad (15.2.13)$$

The acid and alkaline circuits are both well-established processes. They have seen extensive commercial applications in both solution mining and conventional mills employing agitation leaching.

15.2.2.4 Other Metal Systems

Potter et a1.(1982) have reviewed the potential for applying solution mining techniques to hard-rock ores containing values other than copper and uranium. In addition to gold and silver, the most likely candidates are aluminum and manganese. Neither has been recovered in commercial leach operations, but the potential is clearly evident.

For example, sulfuric acid leach liquors produced in copper leach operations often contain a higher aluminum content than copper. In fact, some of these solutions may be nearly saturated with soluble aluminum salts. Thus recovery of byproduct aluminum may be possible. In addition, sulfuric acid leaching of rock containing weathered alunite or various aluminum-bearing clays will generate aluminum sulfate solutions. The US Bureau of Mines solvent extraction process could then be used to recover alumina.

As suggested above, low-grade manganese ores would also be susceptible to solution mining. The lixiviant would be aqueous SO_2, which would react with MnO_2 according to the reaction

$$MnO_2 + SO_2\,(aq.) \rightarrow MnSO_4\,(aq.) \quad (15.2.14)$$

Less oxidized forms of manganese would require an oxidant such as dissolved oxygen, hydrogen peroxide, or perhaps ferric sulfate. Tests reported by Potter et al. (1982) suggest pregnant leach solution grades ranging upward of 10 to 12% $MnSO_4$, implying that manganese can be recovered as the sulfate by evaporation or as the carbonate or hydroxide by neutralization and precipitation.

15.2.2.5 Bio-leaching*

As suggested above, there is a microbial or bacterial aspect to sulfide leaching. The natural bacterial oxidation of sulfide minerals and its role in the leaching of copper heaps and dumps has been well-known for years. The chemoautotrophic bacterial strains *thiobacillus ferrooxidans* and *thiobacillus thiooxidans* play the predominant roles. The exact mechanisms for the bacterial action are still in dispute, but the accepted theory is that the bacteria obtain energy for growth from the enzymatic oxidation of ferrous to ferric iron and elemental sulfur to sulfate. With the addition of nitrogen, carbon, oxygen, phosphorous, potassium, and other elements as nutrients, the bacterial metabolism can be maintained to sustain a stable population.

The action of the bacteria is that of a catalytic agent in accelerating the chemical oxidation reactions. As presented by Brierly (1978) and LeRoux et al. (1973), the *t. ferrooxidans* bacteria catalyze the oxidation of ferrous to ferric iron:

*Segment 15.2.2.5 was contributed by Wayne C. Henderson, Brown & Root Braun, Houston, TX.

$$2\,Fe^{2+} + \tfrac{1}{2}\,O_2 + 2\,H^+ \rightarrow 2\,Fe^{3+} + H_2O \quad (15.2.15)$$

and the *t. thiooxidans* bacteria act to oxidize sulfur to sulfate:

$$S^\circ + 3/2\,O_2 + H_2O \rightarrow 2\,H^+ + SO_4^{2-} \quad (15.2.16)$$

The bacteria increase the rates of reaction by a factor of as much as 10^6 times those of the chemical, non-biological reactions alone.

There is little evidence for direct bacterial attack on the sulfide minerals (e.g., chalcopyrite or pyrite). The bacteria do, however, regenerate the chemical leachates (ferric iron and acid) and augment the oxygen transfer into the leach system. They also help keep the mineral surfaces free of solid reaction products that could retard the chemical leaching.

Surface leaching operations using bacterial catalysis have been confined predominately to copper and uranium systems in which these metal values are directly leached in the acidic environment favored by the bacteria (pH = 1.0 to 2.0). However, the use of a bacterially assisted pretreatment to oxidize refractory sulfide minerals containing gold or silver is just beginning to be developed. The gold and silver, once liberated by sulfide oxidation, would be extracted into solution by a separate leaching step.

A preleach washing and neutralization step would be required to convert the acidic bio-heap system (pH < 2.0) to that favored for conventional cyanide leaching (pH > 10.0). The other option would be to use an acidic lixiviant system to extract gold and silver after the bio-pretreatment (e.g., thiourea or halides).

The bio-pretreatment process actually serves a dual role. The acidic leach removes various base metals and other components that have a negative impact on the subsequent cyanide leach. Their removal prior to cyanidation substantially reduces cyanide consumption and preg-robbing potential. The pretreatment also liberates the gold or silver values from the sulfide mineral matrix. The liberation occurs not only by the oxidation and destruction of the sulfide matrix, but also by creation of additional porosity and permeability in the sulfide grains. The latter improves lixiviant access to the values, boosting extraction rates.

An attribute of the *bio-leaching process* is that the bio-oxidation reactions seem to favor the sulfide phases that contain high levels of dissolved or associated metals. These include arsenic, antimony, bismuth, and copper, among others. These phases are also commonly enriched in gold and silver. This virtue often allows bio-oxidation to achieve high levels of gold extraction in the subsequent cyanidation step with low levels of total sulfide oxidation. This selectivity reduces the time and oxygen transfer requirements for the pretreatment process to be effective.

Process testing and development for possible commercial implementation of bio-oxidation pretreatment for heap and dump leaching of gold ores are underway. Emphasis is being placed on overcoming a number of the major problem areas:

1. Establishing and sustaining an active, viable bacterial culture in the heaps.
2. Supplying sufficient oxygen to maintain bacterial growth and sulfide oxidation.
3. Developing washing and neutralization schemes.
4. Evaluating the feasibility of non-cyanide, acidic leaching systems.
5. Isolating and evolving sulfide oxidizing bacterial strains that are effective at higher pH levels (> 2.0).
6. Developing techniques and appropriate binders for agglomeration that can be used under acidic conditions.

In addition to the ferric sulfate/thiourea leaching system for gold or silver, there are other possible lixiviants that can be used effectively without neutralization following the acidic bio-oxidation pretreatment. The K-Process using a bromine-based reagent is reported by Soper (1988), and the use of brominated hydantoins (Sergent, 1988; Dadger, 1989) show some promise. Iodine-based and mixed halides (chloride, bromide and iodine) also are technically feasible lixiviants for gold under heap leaching conditions.

Environmental considerations and potential toxicity effects on the bio-oxidation pretreatment system would play an important role in selection of either cyanidation or an alternative lixiviant. The significantly faster leaching rates, higher possible extractions, and cost savings due to elimination of neutralization could offset the generally higher costs of the non-cyanide reagents. Economics would have to be defined for each specific system.

15.2.2.6 Metallurgical Testing

Regardless of the values to be recovered, a proper metallurgical test program will be needed for any new leach project. The results will establish expected operating parameters and set the design criteria needed to engineer and construct the leaching and recovery facilities. The most important single factor in any test program is the selection of the samples to be tested. Great care must be taken to assure that the samples are representative of the material to be leached. In fact, if the deposit contains more than one type of ore, either in terms of mineralogy or host rock characteristics, then each type must be thoroughly evaluated.

As discussed by McClelland (1988), there are at least two and generally three levels of testing to be undertaken:

1. Preliminary —small column percolation or bottle roll tests.
2. Detailed—larger column tests to optimize feed size.
3. Pilot scale—large column or field tests.

The preliminary tests provide good insight into the amenability of the material for leaching and permit selection of the preferred lixiviant, should the choice not be obvious. The test results also provide information on the chemical aspects of the proposed ore lixiviant system. These chemical parameters include reagent consumption and the rate and ultimate level of extraction of the value(s). Finally, the small-scale tests would indicate whether or not agglomeration is needed in order to avoid permeability problems with clays or fines.

Care must be exercised in trying to extrapolate the preliminary test results to predict parameters in the eventual operating system. Unless a factor is applied to the preliminary results, they will always indicate higher leach rates and reagent consumption than will occur in the commercial operation. The reason for this is that the material is crushed much finer in the small-scale tests than in the full-sized system. This will maximize liberation of the values and minimize diffusion distances to any mineralization that is not exposed. Thus the values will be solubilized rapidly and completely, assuming that the results are not impacted by complete reagent consumption in the typical batch-type tests.

Reagent consumption will also be maximized. In part, this reflects the high extraction of values but also the high host-rock surface area. In particular, the fine rock size will maximize reagent consumption in cases where the lixiviant is not highly selective for the values and attacks the host-rock (gangue) minerals as well. Such behavior is particularly common with acidic lixiviants since these tend to be less selective than the alkaline reagents. For example, in the acid leaching of copper, there is generally much more reaction with gangue constituents than copper minerals on a mass basis.

The detailed tests are run in larger columns and are generally intended to determine whether or not the ore should be leached in run-of-mine condition, crushed to some smaller size, or crushed and then agglomerated. In such a program, the extraction rate curves and reagent consumption figures are determined as a function of the particle size and degree of agglomeration. These tests can be run in one of two ways—single or multiple feed sizing. In the former, a medium-sized material is selected and column leached until the extraction rate levels off to near zero. Then the contents of the column are screened and assayed to determine recovery as a function of particle size. Generally, a second confirmatory test is then run on material crushed to the selected size. This approach is less costly but more time consuming than running parallel tests on multiple samples crushed to cover a range of sizes.

With either approach, the last step is to balance projected recoveries and projected revenues against the cost of progressively finer crushing and/or agglomeration. Then the most economic operating scenario can be selected.

The pilot scale tests are not always run due to the time and costs required to get meaningful results. However, if run, they greatly reduce the risk of a metallurgical failure when advancing to a commercial operation. Typical large-scale column tests are run with 20 to 40 tons (18 to 36 t) of gold/silver ore (McClelland, 1988) and as much as 200 tons (180 t) of copper ore (Murr, 1980). In many cases the large column tests will be more meaningful than small field heaps due to edge effects caused by the large percentage of material contained under the sloping heap faces. On the other hand, percolation characteristics in the columns probably will not duplicate those in heaps and dumps since material emplacement is different in each system.

Large-scale field tests are generally required only when the material will be leached at a coarse size in large piles for extended periods. The test heaps should be full height when compared to the planned commercial operation, or at least as high as the first lift. Costs can be controlled by making the pilot heaps the first commercial heaps so that the revenue from mineral production offsets the costs of the tests. If the tests suggest that modifications are required, these can be made on succeeding heaps or dumps.

McClelland (1988) compares the advantages and disadvantages of the large-scale field tests as follows:

The advantages for conducting large pilot heap tests include

1. A more representative bulk sample of the ore deposit can be obtained.
2. Commercial operation is simulated.
3. Accurate design data is obtained.
4. Actual recovery, recovery rate, and reagent usage requirements and data are determined.
5. Information concerning heap stability, heap blinding, solution channeling, and fines migration is obtained.

The principal disadvantages of large pilot heap tests include

1. Costs are higher than those for pilot column tests.
2. Environmental and/or operating permits may be required.
3. Obtaining large tonnage bulk samples may be difficult.
4. Preparation of leach pads and solution ponds is involved.
5. A solution application system must be designed and assembled.
6. A small solution recovery system must be constructed, but will probably not be used in the commercial operation.
7. Monitoring is difficult and the test is labor intensive.
8. Shutdown and removal (reclamation) of the test heap may be necessary.

In addition to the general test criteria outlined above, there are many commodity-specific aspects to metallurgical testing. These cannot be addressed in detail here but do require that a well-planned research program be conducted by knowledgable metallurgists and engineers in order to get meaningful results. To cite just one commodity specific example, bottle roll tests on cyanide extraction of gold should be done in open bottles. If the bottles are sealed, poor results may be due to oxygen deficiency rather than the presence of refractory mineralization.

15.2.3 MATERIALS HANDLING

15.2.3.1 Mining

Surface heap and dump leaching operations are almost invariably associated with open pit mining operations. These require essentially the same planning functions and employ the same unit operations as those described in Sections 13 and 14 of this *Handbook.* There are, however, two areas that may be impacted somewhat by leaching operations—mine planning, and drilling and blasting.

For operations where heap or dump leaching supplements production from conventional milling, the mine planning function can become more complex. One of these complexities lies in the need for more segregation of mined materials. Instead of dividing material into just two categories, mill ore and waste, three will be required: mill ore, leach material, and waste. Both leach and waste dumps will need to be planned, and the movement of all three types of material will have to be scheduled. In addition, water distribution and collection systems will have to be designed and installed for the leach system.

Supplemental production of leach copper also impacts on the calculation of the ore cutoff grade and the ultimate pit limits. Since leaching permits partial recovery of copper from material that would otherwise be wasted, the effect of a leaching operation is always to raise the cutoff grade on ore going to the mill. This, in turn, lowers the unit costs on the mill output. Also, since leaching boosts total recovery of values from the mineral resource, it has the effect of increasing the tons of material that can be profitably mined. Stated another way, a supplemental leach operation causes the ultimate pit limits to expand. The exact impact of leaching will be site-specific and will depend on the relative costs and output of leach production vs. mill production.

Drilling and blasting is another area that may be impacted by leaching—particularly when a mine-to-leach operation utilizes run-of-mine ore. Under this scenario, the drilling and blasting operation will be designed to give a well-fragmented product with minimal quantities of oversized rock, say, plus 8 to 10 in. (200 to 250 mm) across (see Chapter 9.2). The well-fragmented product should stack and leach well, while providing the good mineral exposure needed for effective leaching. As rock characteristics are very site specific, costs for the drilling and blasting program that gives an optimum as-mined leach product may need to be compared with costs for a low-cost drilling and blasting approach, plus primary crushing to produce material for leaching.

15.2.3.2 Ore Preparation

Material scheduled to go to heaps or dumps may be handled in run-of-mine condition or prepared for leaching in one of two ways: crushing or agglomeration. Crushing will be undertaken when the increase in production achieved with the finer product generates sufficient incremental revenue to cover the capital and operating costs for crushing. In general, low-grade wastes that must be mined to expose mill-grade ore will not support much material handling or preparation. Such material is often dumped in run-of-mine condition using dump locations that minimize haulage costs rather than maximizing leach output. Then the dump will be leached to recover some of the values that would otherwise be lost.

Agglomeration is undertaken to provide the opposite effect of a crushing step, that is, to provide a strong, porous and coarsely sized material that otherwise would be unleachable due to poor percolation characteristics when placed in heaps and dumps. This is particularly true when material finer than 50 mesh is present. Such conditions usually lead to low metal extraction because of slow and uneven percolation (channeling) of solution and the development of impermeable (dead) zones within the heap or dump. Clays are probably the leading source of fines, although fines produced by poor crushing practices are undesirable as well. Agglomeration is also required when heap leaching is selected as the route for reprocessing tailings or other very fine materials.

At present, agglomeration is widely applied only in precious metal operations. However, uranium-vanadium tailings have been agglomerated and heap leached in Colorado and chrysocolla is being agglomerated with a polymer binder and heap leached at the Ray mine in Arizona. For gold and silver, important parameters include the quantity of binder (typically, 10 lb/ton or 5 kg/t), the moisture content (10 to 20% total), and the curing period for the agglomerates (at least 8 hr). Since the subsequent cyanide leach requires alkaline conditions, binders are typically lime or Portland cement. These binders generally have two important properties. First, clay permeability is improved by the exchange of sodium ions in the clay with calcium ions in the binder. Second, the binders have a cementing or pozzolanic effect that strengthens the agglomerates. Leach extraction rates can also be accelerated by using a barren cyanide solution as the source of moisture.

Agglomeration follows one of two pathways. One causes the adherence of fines onto the coarser particles. The other involves agglomeration of fines into stable balls. As discussed in detail by Milligan (1983) and reported by McClelland and van Zyl (1988), design factors in agglomeration include feed characteristics (particle size and presence of fractures), process control, and equipment selection. For gold and silver materials, belt, drum, and pan agglomerators are the most popular types of equipment. Graphical design data on both agglomerator sizing and costs are reported by McClelland and van Zyl (1988).

Fine tailings are agglomerated much like ore, but optimum conditions and equipment are different. More binder is often required (10 to 15 lb/ton, or 5 to 7.5 kg/t), considerably more water is needed (16 to 22%), and curing times are longer (typically 72 hr). In addition, the agglomerator should roll, rather than bounce the particles. Generally, drums, disks, and pug mills provide good action, while belt agglomerators tend to bounce the particles.

Crushing is not covered in detail here, and the reader is referred to Chapter 25.3 and standard works such as the *SME Mineral Processing Handbook* (Weiss, 1985) for details. For dump leaching, there is generally no size reduction unless primary (in-pit) crushing is practiced in order to transport and stack the material by conveyors. For heap leaching, primary, secondary, and tertiary crushing are often used to produce a final product that is nominally –1.0 in. (–25 mm) to –0.5 in. (–10 mm). Occasionally, some grinding may also be needed to liberate encapsulated values. This material will then require agglomeration prior to heap leaching.

15.2.3.3 Ore Emplacement

The stacking of the material for leaching is a major factor in the success of a surface leaching operation. The approach to emplacement represents one of the principal differences between waste dump leaching and heap leaching. The former is characterized by dumping practices that reflect topographic constraints and minimization of haulage costs. Any consideration of leach recovery is usually limited to such practices as waste segregation and dump surface preparation. Segregation usually involves efforts to avoid mixing barren or mineralogically unleachable materials into the leach dumps. Surface preparation typically is limited to leveling and deep ripping to breakup the compacted zone associated with the truck traffic pattern.

Such waste dumps are often high; elevation differences of 300 to 750 ft (90 to 230 m) between toe and crest are not uncommon. This profile exacerbates the size segregation that occurs when run-of-mine material is dumped over the crest. This allows the coarsest material to slide to the bottom while the fines tend to remain at or near the crest. As a result, the least permeable and most easily compacted zone of material is found at the top. Application of leach solution then causes slow and uneven percolation at the top of the dump, with a great tendency for solutions to channel and short circuit through this layer. These poor percolation patterns normally persist at depth so that a significant volume of material in the dump is leached poorly, if at all.

There is also another form of segregation that inevitably occurs in waste dumps. This is the development of alternate coarse and fine layers of material that lie parallel to the angle of repose. The coarse zones act as conduits and carry the bulk of the lixiviant flow. Hence they are usually well leached in spite of their large particle size. By contrast, the fine material has a low permeability so it undergoes little solution penetration or leaching.

These factors in waste emplacement are responsible for the generally poor extractions observed in such dumps. For run-of-mine material, recovery of metal values seldom exceeds 50% after many years of leaching. Only 10 to 15% extraction during the first one or two years is not uncommon, at least for copper sulfide dumps.

As opposed to dump leaching, heap leaching involves careful emplacement of rock in order to maximize extraction of the values. In general, this will involve construction of much shallower lifts, typically 10 to 35 ft (3 to 10 m) high. This tends to minimize size segregation, as does the use of crushed or agglomerated ore. In turn, solution percolation is more uniform and solution-rock contact is improved, resulting in faster and more complete metal recovery.

As described by Muhtadi (1988), there are several approaches to heap construction that warrant consideration due to their widespread use.

TRUCK DUMPING AND DOZING. This method of construction is essentially the same as that described above for waste dumps. However, more care is given to emplacement of material. Typically, a ramp of waste is constructed to the height of the first lift. Then ore is dumped from the end of the ramp down onto the pad or foundation of the heap. As the ore builds up, the heap extends away from the ramp. To minimize compaction, an elevated roadway is often extended as the heap expands, with haulage trucks restricted to this path. Once the heap is completed, the roadway is dozed off and the surface of the heap is graded and ripped in an effort to assure good permeability and uniform percolation of solution.

In general, this method of heap construction is restricted to ores that do not undergo compaction or generate fines. This requires a strong, competent material handled either in run-of-mine or crushed condition. Agglomerated materials cannot generally be handled by truck dumping as they are too soft and friable. In addition, truck dumping can still cause the coarse-fine segregation noted above.

PLUG DUMPING. This method of emplacement also uses trucks, but is not characterized by the rough handling noted above. Hence it is suitable for softer ores and agglomerates. In plug dumping, the first step is to bed the liner or foundation with as much as 1.5 ft (0.5 m) of crushed rock as a protective measure to insure liner integrity. Then each haul truck dumps its load as closely as possible to the previously dumped pile until the entire pad is covered with overlapping mounds. This gives a maximum heap height of about 7 ft (2 m), slightly less if a dozer is used to level the tops of the mounds.

Another adaptation is to replace the trucks with large rubber-tired front-end loaders. These are very efficient and stack material to heights approaching 16 ft (5 m) without the need for bedding the liner first. Good operators can even produce a nearly level surface that is suitable for installation of the leach solution distribution system without the need for dozing.

CONVEYOR SYSTEMS. Conveyor stacking is gaining widespread acceptance in gold heap leaching. This concept is not new, however. It started at least as early as the beginning of the 20th century when high-grade copper ores were bedded and removed from leach vats using a clamshell bucket on a traveling gantry crane or another mechanical system. Indeed, one of the first stacking systems in gold heap leaching (Gold Fields Mining Company's Ortiz, NM, operation) utilized a traveling bridge that spanned a reusable pad and used a moving tripper to continuously add ore to the face of the heap. In this operation, spent ore was removed from the asphalt pad by front-end loaders.

Now conveyor hauling and stacking of ore on heaps has evolved into a highly mobile operation. The front end of the system usually consists of a short fixed conveyor that receives prepared ore from a crusher, agglomerator, or a stockpile via a feed hopper. This conveyor transfers material to a mobile, radial arm conveyor/stacker using a system of movable intermediate conveyor sections. The stacker has great flexibility and can bed the heap uniformly and without compaction to almost any height desired. Thus conveyor systems can handle anything from primary crushed ore to agglomerated tailings.

15.2.3.4 In-place Systems

In-place systems represent a special case of material handling that is intermediate between surface heap and dump leaching and in situ mining. This case involves broken or rubblized material that is leached more or less in-place. In-place leaching, clearly involves no loading or hauling of broken ore to heaps or dumps. However, unlike in situ mining, in-place leaching still relies on the downward percolation of solution through unsaturated leach material. (By contrast, in situ mining is characterized by pressure-driven solution flow in a saturated medium).

In-place systems are so site specific that discussion is limited to brief descriptions of a few commercial operations. Although broken uranium ore has been leached in underground settings, copper appears to have been the main target to date. Probably the largest-scale operation was undertaken by Ranchers Exploration and Development Co. when they drilled and blasted the entire upper portion of the Old Reliable ore body and then acid leached the rubblized zone in-place. Technically, Ranchers was able to fracture and rubblize the ore, but leaching yielded less copper and lower solution tenors than expected. Thus the project was not the commercial success that was hoped for.

Greater success has been achieved in programs that involve progressive rubblization and leaching of abandoned pit walls. Since most porphyry ore bodies contain a halo of low-grade mineralization around the economically minable zone, pit-wall leaching often recovers a portion of the copper that is otherwise left behind. Current examples include pit-wall leaching at Mineral Park and the Lavender pit near Bisbee, both in Arizona. Another good example is leaching of broken, remnant ore left behind in block caving operations. Current examples of this system include El Teniente in Chile and San Manuel in Arizona.

The pit-wall system uses a conventional solution distribution system (see the following segment) to apply lixiviant to the upper surface of the broken ore. Pregnant solution is then collected in the pit bottom or where solution breaks out at the bottom of rubblized material. However, leaching of underground material may pose problems in getting uniform distribution of leach solution throughout broken rock mass. In extreme cases, injection wells may have to be drilled from the surface, much like in situ mining. Solution is then recovered in one or more sumps at the bottom of the ore body.

15.2.4 SOLUTION MANAGEMENT

Materials handling and solution management are interactive, with rock preparation and stacking having a substantial effect on solution flow. For example, maximum recovery of metal values can only be achieved if the heap/dump design permits good distribution and collection of leach solution. The design must also provide for movement of air into the piled rock since oxygen is needed as a reactant in solubilizing many values. Likewise, the rock must be broken and stacked so that the entire rock mass is accessible to solution percolating down from the surface of the heap or dump. Nevertheless, once the rock has been prepared and put in place, there is little an operator can do except to optimize the way the leach solution is handled. This will accomplish two things. One is to transport the lixiviant to the mineralized values throughout the piled rock. The other is to wash the solubilized values from the heap or dump to the recovery plant.

15.2.4.1 Fluid Flow Phenomena

STATIC CONSIDERATIONS. From a theoretical point of view, solution flow in heaps and dumps involves the gravity-driven, downward percolation of water in an unsaturated, semiconsolidated (porous) medium. However, as reviewed by Schlitt (1984), this is a regime that remains too complex to address on completely theoretical grounds.

The first step in understanding solution flow through the rock mass is to categorize the space within the dump or heap. As shown in Fig. 15.2.1, there are at least four distinct regions. By far the largest is the solid rock (including isolated pores), indicated by v_s. This represents completely stagnant space and typically involves about 60% of the total volume. Blasting and handling the rock expose pores and create fractures or cracks on the faces of the rock fragments. The space associated with these openings ϵ is generally only 2 to 4% of the total volume. Nevertheless, this space is quite important because it greatly increases the exposed surface area and makes the contained mineral values more accessible to the lixiviant. The remaining volume within the rock pile is the void space. This results from the swelling that is a consequence of fracturing and stacking the rock during mining and handling. For newly prepared heaps, the void space associated with swelling normally constitutes about 40% of the volume of the rock pile. This will decrease as weathering and leaching consolidate the material. The void space will probably drop to less than 30% if a mine dump is leached over a period of years.

Obviously, the void space can either be filled with solution or with air. However, because the water/rock interface has a

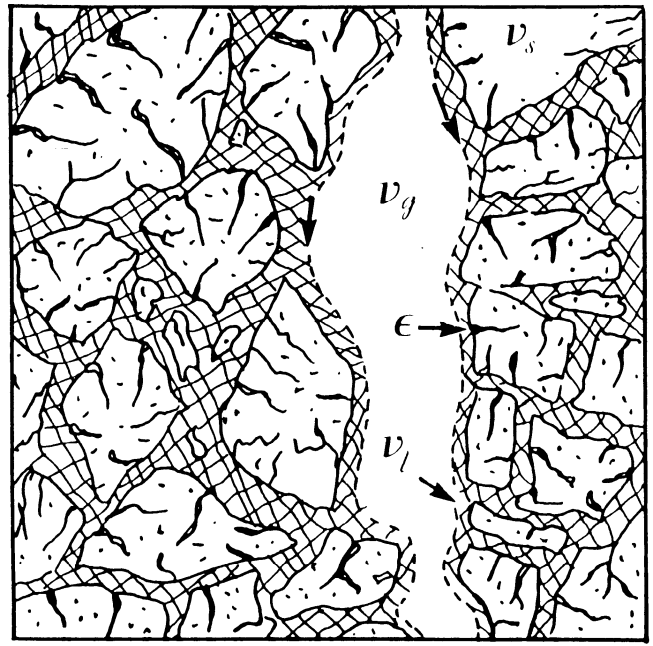

Fig. 15.2.1. The categories of space found within a heap or dump: solid rock (v_s), open space within rock fragments (ϵ) water-filled void space (v_l), air-filled void space (v_g), (after Schlitt, 1984).

lower surface energy than the air/rock interface, all the openings in the rock fragments will be filled with solution, and the rocks themselves will be covered with water under conditions of percolation. The water-filled void space v_1 represents the area for active water flow. On the other hand, mass transport within the ϵ-type space will occur only by diffusion. One other consequence of the difference in the interfacial energies is the location of the air-filled void space v_g. Under conditions of percolation, only the larger voids and channels will contain air, as the smaller openings are flooded.

Except under conditions of flooding, the competition between air and water for the available void space will be almost independent of the rate at which leach solution is applied. Instead, the controlling factor will be the effective particle size of the rocks in the pile. This is a consequence of the limitations imposed by capillary forces.

As shown in Fig. 15.2.2, capillary rise or drain-down height is related to effective rock size. The same figure shows the correlation between rock size and the percentage of the total void space that remains water-filled after leaching ceases and drain-down is complete. At this point, capillary rise is balanced by the force of gravity, and only evaporation will further reduce the water-filled void volume.

The experimental results shown in Fig. 15.2.2 are significant when ingress of air is needed to aid in solubilizing the metal values. The data demonstrate that if the material is finer than about 48 mesh, it will not drain down at all, and little or no air will be able to penetrate into the pile. On the other hand, if the material is coarser than about 10 mesh, the pile will drain down almost completely.

DYNAMIC CONSIDERATIONS. When leaching begins at a low application rate on a thoroughly drained dump, some additional air-filled void space becomes water-filled. This occurs at air-water interfaces, and flow begins. As the application rate increases, the only change will be to thicken the film of flowing water in the larger voids and channels where there is a solution-

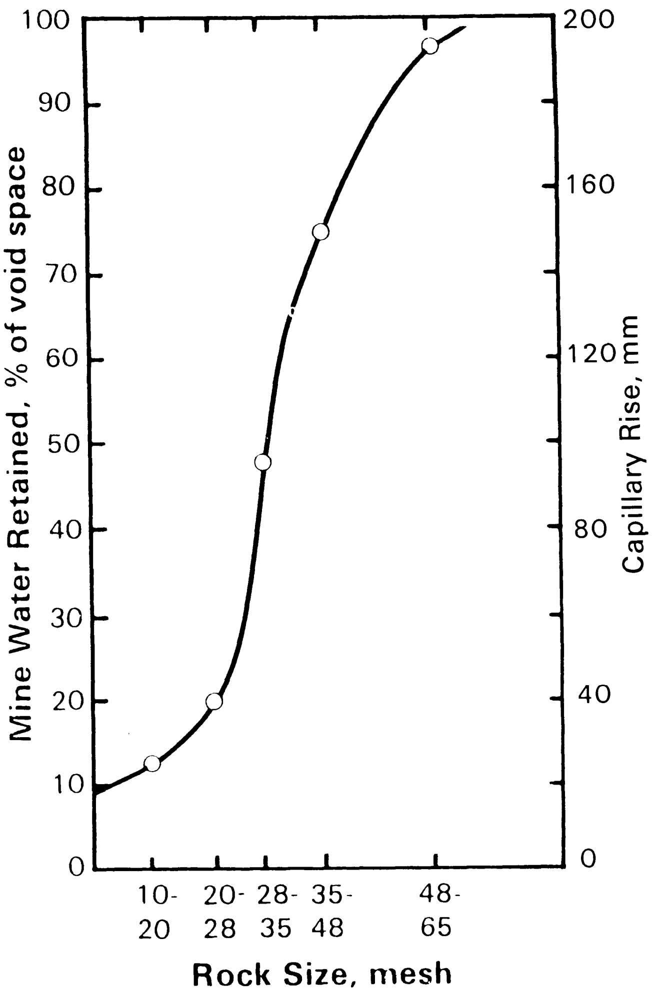

Fig. 15.2.2. Solution retention and the corresponding capillary rise as a function of rock size (after Schlitt, 1984).

air interface. No substantial change in the volumes occupied by air and liquid will occur until the application rate increases to the point where the air-filled voids begin pinching off. When such bottlenecks occur, the open voids will flood upward from the pinch points and the air-filled void spaces will begin disappearing. Experimental column data demonstrating this phenomenon are shown in Table 15.2.1. Note that at typical application rates (0.1 to 1.0 gal/ft^2/hr, or 4 to 40 mm/h), there is considerable air-filled void space. Thus adequate aeration should not be a problem unless the material being leached is very fine, for example, flotation mill tailings.

Regardless of the application rate, the velocity of solution flowing through a water-filled void is controlled by the pressure gradient associated with the hydrostatic head and the permeability at that location. This relationship is expressed by Darcy's law as given by Eq. 15.2.17:

$$V_l^\circ = \frac{k}{\mu}(dp/dx) \qquad (15.2.17)$$

where V_l° is superficial velocity of the leach solution, k is permeability, μ is fluid viscosity, and dp/dx is pressure gradient or hydrostatic head of the leach solution.

Table 15.2.1. Effect of Solution Application Rate on Water-filled Void Space

Application Rate		Water-filled	
mm/h	gal/ft²/hr	Void Space, %*	Comments
0	0	8.0	Drained
1.2	0.03	9.6	Capillaries filled
6.0	0.15	9.7	Typical range
24.0	0.60	10.1	of application
120.0	3.0	10.3	rates
3050.0	75.0	10.5	
4070.0	110.0	27.6	Flooded

*Expressed as a percentage of the total space within the rockpile.

Optimum Application Rate. Substituting appropriate values into Eq. 15.2.18 defines an equivalent maximum application rate q_{max}. This is the rate which can be used without experiencing shortcircuiting through the larger voids and channels that are partially air-filled. Using operator-oriented units, this expression is

$$q_{max}\ (\text{gal/ft}^2/\text{hr}) = 0.9k\ (\text{Darcy}) \qquad (15.2.18)$$

Although heaps and dumps typically contain a broad rock size distribution, the permeability and thus q_{max} will be dominated by the smaller particles that fill the interstices between the larger rocks. Since capillarity also confines the leach solution to the smaller voids, flow is limited almost entirely to these capillaries when the application rate is less than q_{max}. Once the water application rate exceeds this value, the excess floods laterally and substantial flow occurs in the films along the walls of the larger air-filled voids. Further increases in the application rate simply shunt an increasing volume of water to the large channels. This leads to significant short-circuiting of the leach solution and dilution of values in the pregnant solution.

Clearly, the average permeability of the heap or dump matrix will limit the application rate that can be used to flush the solubilized values to the recovery plant. When the rate of metal extraction is controlled by washing the soluble values from the water-filled void space, use of application rates in excess of q_{max} is detrimental. Such high flows mean that higher than necessary pumping costs are incurred. The recovery plant must also handle the excessive solution flow. In addition to higher operating costs, this means that capital costs must be increased to provide the oversized facilities.

The actual flow rates involved are high even when there is no short circuiting. For a matrix permeability of 5 darcies (equivalent to a particle size of 35 mesh), V_l° will be 4.5 m/day. However, the actual water velocity will be five times as high, or about 75 ft/day (22.5 m/day) assuming that the water-filled voids occupy 20% of total volume of the heap or dump.

Such a high flow rate within the interior of the rock pile does more than efficiently wash dissolved values out of the heap or dump. The flow will also rapidly distribute the lixiviant throughout the liquid-filled void space. However, this will not carry the lixiviant to the undissolved mineral values unless these are exposed on the outer surfaces of the rock particles.

As discussed above, when the values are contained within the rocks, the required lixiviant transport can only occur by diffusion. This will be orders of magnitude slower than solution flow. The penetration distance of the lixiviant will depend on the rate lixiviant consumption as well as on the diffusion rate.

Effective diffusional penetration will typically be about 4 in. (100 mm) before the lixiviant is consumed.

From the standpoint of the leach operator, the main point of the foregoing discussion is that the maximum effective application rate will be limited by the zone of minimum permeability in the system under leach. This may be the bulk permeability of the rock pile. However, care should be taken so that the solution application rate does not exceed the surface acceptance rate of the heap or dump. This is an important consideration because there are several factors which can combine to make the surface permeability lower than the bulk value. These factors include the following:

1. Accelerated decrepitation or weathering of the surface layer.
2. Breakup and compaction of the surface rock due to equipment movement, particularly haulage truck traffic.
3. Segregation of fine material at or near the top of the heap where dumping occurs.
4. Formation or accumulation of chemical precipitates such as gypsum or hydrated iron oxides.

Where surface permeability is lower than the bulk value, the application rate should be reduced to avoid ponding. Such surface flooding will prevent air circulation into or through the rock. Ponding will also provide ideal conditions for continued chemical precipitation that seals the surface of the rock pile.

To select the maximum effective application rate, the leach operator will need some knowledge of the range permeabilities found in his system. This information is probably best obtained by using one of the standard water permeability tests. Unfortunately, there is little published data on heap and dump permeability to serve as a guide. Limited field measurements on copper leach dumps suggest a value of q_{max} between 0.5 and 1.0 gal/ft^2/hr (20 and 40 mm/h). Higher flows simply cause increasing amounts of water to short-circuit through the rock without recovering additional values.

The importance of controlling the application rates is further demonstrated in Kennecott data published by Jackson and Ream (1980) for copper dump leaching. These results compare sprinkler leaching at 0.5 gal/ft^2/hr (20 mm/h) with trickle leaching at 2 gal/ft^2/hr (80 mm/h). Sprinkler leaching produced 11% more copper and a 15% higher copper content in the pregnant solution. Overall, 13% less water was pumped per pound of copper produced when using the lower application rate. This provided a significant reduction in power costs per unit of copper produced.

LEACH/REST CYCLES. Leach/rest cycles are a factor in solution management. Once a volume of rock has been thoroughly wetted with solution, the lixiviant will be distributed throughout the rock mass. From that point on, dissolution of the values will continue without further solution application unless the lixiviant is consumed. Thus rest cycles should not affect overall extraction of the metal values.

However, rest cycles can extend the economic life of older heaps and dumps, and this permits maximum recovery of values from the material that was mined. Use of rest cycles will also permit an operator to reduce costs by maintaining a limited flow of high-grade pregnant liquor to his plant. In both cases, the key factor is switching the application of water from one leach area to another. While standing fallow, chemical and diffusional processes dissolve the values and cause their concentration to build up in the water-filled void space. Then a relatively short period of active solution application will wash the solubilized values from the heap or dump. On older material a longer rest cycle may be required to build up an economic concentration prior to washing. In a similar vein, recycling barren solution to the oldest material first, then advancing the outflow to fresher material will reduce the volume and increase the grade of the pregnant solution being treated in the plant. For these reasons, leach/rest cycles should not be overlooked as a way of optimizing a leach operation and minimizing facility costs.

15.2.4.2 Production Aspects

PUMPING SYSTEMS. Selection of a pumping system depends on several factors: the hydrostatic head on the system, the line pressure needed to drive the solution distribution equipment on the heap or dump surface, the flow volume, and the corrosion potential of the lixiviant. Because there are so many site-specific factors involved, only general guidelines are provided here.

In many operations, the recovery plant is located at a lower elevation than the heaps or dumps. Thus pregnant leach solution generally flows to the plant by gravity in lined ditches or low-pressure pipelines. Any pumping from a heap or dump outflow or from a plant feed pond would generally require only a low-pressure horizontal pump. Such a pump may be mounted beside the pond with the pump intake suspended from a float in the pond. This readily accommodates a rising or falling water level.

With the situation noted above, the recycling of barren solution back to the heaps or dumps requires a high-head pumping system. Typically, this involves multistage vertical pumps. These may be mounted on a fixed structure, as in a pump house where pumping is from a sump. Alternatively, the pump may be barge-mounted and float in the barren solution pond. If the system pressure is not too great, flexible pipe or couplings can be used to tie the pump into the shore-based pipeline system. This allows the pump to rise and fall with the pond level.

The type of pipe will be dictated by the elevation difference between the plant and leach surface, plus the line pressure needed to drive the solution distribution equipment on the heap or dump surface. The elevation difference may be only 100 ft (30 m) or so, or may be as much as 2000 ft (600 m). The latter will require sophisticated design starting with heavy-walled metal pipe, shifting gradually to lighter schedules. The entire pipe system must be well anchored, yet allow for expansion and contraction due to temperature changes. In addition, the entire header system will almost certainly be above the 10- to 50-psi (70- to 345-kPa) pressure needed on the leach areas. This will require the installation of outlet stations on the main header line that will serve as pressure reduction points as well. Once the pipeline profile and other flow parameters are defined, sophisticated computer programs can be used to aid in the pipeline design.

Pipe materials must be selected with the chemical nature of the lixiviant in mind. When line pressure dictates metal pipe, 316L stainless is a frequent choice for acidic copper leach liquors that also contain ferric and cupric ions. Polyvinal chloride (PVC) or other liners have been tried on black iron or steel pipe, but the linings tend to deteriorate with time, particularly at joints. For alkaline solutions, black iron may be satisfactory.

For lower pressures, high-density polyethylene (HDPE) pipe is becoming the standard of choice. This is rugged, easy to cut and fuse in the field, resists scaling, and is essentially corrosion-proof. PVC and other rigid plastic pipe may be less expensive but are not as rugged and tend to shatter. One of the few problems with HPDE pipe is its lack of rigidity so that it twists and turns; it may require staking or anchor blocks to keep it in place. However, this flexibility can also be an asset when routing the pipe during initial installation.

SOLUTION APPLICATION SYSTEMS. As pointed out in 15.2.4.1, careful distribution of solution on the heap or dump surface is required to optimize the leach operation. Development of such a solution management system requires a clear understanding of three concepts:

1. *Application rate* is calculated by dividing the leach solution volumetric flow rate by the surface area to which the solution is actually being applied.

2. *Surface acceptance rate* is the maximum application rate that can be employed without encountering noticeable ponding.

3. *Irrigation rate* is a measure of the intensity of leaching. It is calculated by dividing the total volumetric flow rate by the total area planned for leaching. This area includes both areas under active leach and areas that are being allowed to rest. For sulfidic copper materials, typical irrigation rates are only 20 or 25% of the application rate, and may be only 10% on old areas needing a long rest cycle. Irrigation rates would be much higher when leaching gold, silver, and oxide copper since the extraction rates are higher.

The desired application rate should be based on q_{max} as defined in Eq. 15.2.18. However, not all leach systems are capable of delivering this flowrate. The most rudimentary system is pond leaching where a berm is dozed around an area that is then flooded with leach solution. Here the application rate may be as high as 5 gal/ft^2/hr (200 mm/h), assuming good surface acceptance. Such a high rate will almost certainly exceed q_{max}, leading to extensive short circuiting and dilution of the values being leached.

Trickle systems using drilled or perforated pipe are fed off main header lines. Typical trickle lines are spaced 20 to 25 ft (6 to 8 m) apart. Holes about 0.5 in. (12 mm) in diameter are drilled every 8 to 12 ft (2.4 to 3.6 m) at an angle of 15 to 20° from vertical in order to get wide coverage. The hole spacing decreases from the header line to the opposite end of the feeder line. This is done to compensate for the pressure drop along the line. Trickle systems typically provide an application rate of 2 gal/ft^2/hr (80 mm/h). As shown by Jackson and Ream (1980), this rate still exceeds q_{max}, at least for copper leaching.

Sprinkler leaching utilizes several types of equipment including rotating impulse sprinklers, Bagdad wigglers, and Senninger Wobblers. The first is much like the typical “Rain Bird” agricultural sprinklers except for materials of construction. These include acid resistant plastic or stainless steel for all wetted parts. The wiggler utilizes short pieces of surgical tubing attached to nipples on a feeder line. The tubing then flops randomly, giving fairly uniform solution coverage. Wobblers are off-center rotary action sprinklers that provide a coarse spray pattern which minimizes evaporation. However, the off-center design creates thrust problems, and Wobblers must be staked down.

All three designs are capable of providing a pattern of reasonably uniform solution coverage at an application rate of 0.5 gal/ft^2/hr (20 mm/h). This is safely below q_{max}. With their own pressure regulators, the Wobblers are sophisticated enough to provide even lower rates (e.g., 0.3 gal/ft^2/hr, or 12 mm/h), the level typically cited as the optimum for gold. Ranges of operating parameters for several types of sprinklers are shown graphically by Schlitt (1984) and are available in vendor literature.

Another type of leaching device, the pressure emitter, is seeing growing acceptance. This is equivalent to the drip irrigation devices used in agricultural applications. The emitters operate at low gage pressure, 15 to 20 psi (100 to 140 kPa), and produce a steady seepage of water that gives about the same application rate as sprinklers.

Muhtadi (1988) provides details on the design and installation of emitter systems. These have the following advantages over sprinklers:

1. The continuous drip minimizes droplet impact, reducing rock decrepitation, fines migration, and solution channeling.

2. The system works well either on the surface or when buried. The latter permits leaching to continue even under freezing conditions.

3. Evaporation losses are minimized, especially when the system is buried.

The major drawback to emitters is the likelihood of blocking the small flow channels with precipitated salts. Therefore, conditioning of the recycle solutions to prevent scale formation or remove suspended solids must receive careful attention if emitter plugging is to be avoided. Conditioning may involve use of settling ponds, filtration systems, or chemical additives.

15.2.4.3 Environmental Aspects

At least in the United States, obtaining the necessary environmental permits is probably the single most time-consuming activity in the development of a heap or dump leaching project. Permitting requirements will vary from state to state and may differ depending on whether the project is to be sited on private, state, or federally owned land. Because of these differences and constantly changing regulatory requirements, a detailed guide to the appropriate permitting process cannot be presented here. (For a general discussion of environmental law and practice, the reader is referred to Chapters 3.4 and 7.3.) The best approach is usually to hire knowledgeable environmental consultants who practice in the area where the project will be located. These people should be quite familiar with applicable regulations and procedures and will often know the local and state regulatory personnel.

Although specifics may differ from one site to another, the regulatory agencies generally have two major concerns regarding surface leaching projects. One is assuring the safe handling of toxic chemicals and reagents. The other is the prevention of contaminated run-off or seepage to ground water. Detoxification of the heaps or dumps may also be a concern in all future operations and is already having an impact on the gold heap-leaching segment of the industry. However, groundwater restoration, a major factor in in situ operations, will have little impact on surface operations unless contaminating leaks have been a problem.

The other environmental concerns are those common to most mining operations. These include control of particulate emissions (dust) from the mining and any crushing or processing operations. Another concern is topographic restoration to preexisting profiles and revegetation of the mine and leach areas. The restoration activities usually carry with them some sort of bonding requirements. In terms of cost, solution containment and heap or dump stability are the major considerations. Each is discussed below.

GEOTECHNICAL DESIGN. The design of piled rock structures that are stable under conditions of water flow, especially in seismically active areas, is a complex undertaking. Entire books have been written that address the necessary design features for both heaps (van Zyl, 1987) and dumps (McCarter, 1985). Because of this complexity, a detailed analysis of the problem cannot be presented here.

The first consideration is the method of heap or dump construction. The alternatives are reusable pads, expanding pads, and valley dumping. The major requirements and considerations in each case are provided by Dorey et al. (1988). In terms of both containment and stability, the reusable pad is the best. This system utilizes a sloping asphalt or concrete base that is highly impermeable and will easily support the single shallow lift of ore. This is bedded on the base with front-end loaders or an automated stacking system. Material is loaded with little or no compaction. The leach solution will usually percolate readily through the pile with the pregnant leach solution draining from the heap at controlled points. Solution is conveyed to the recovery plant in lined ditches or pipelines with little likelihood of loss. From an environmental point of view, the biggest concern is the integrity of the final impoundment area for the leached tailings. This will require a properly designed retaining dam and will also require a liner(s) to prevent seepage losses of residual reagents. A monitoring system will also be required to detect any leakage.

The other two types of heaps, the expanding system and the valley system, do not entail the rehandling required with the reusable pad. The expanding pad system involves continued extension of the prepared heap foundation, which is then loaded with a single lift of ore. This system is characterized by a gently sloping base, often with intermediate drains. Eventually, topography and property constraints may require that the original lift be overdumped with one or more lifts. Overdumping must be done with care as the upper lifts often leach less effectively than the original one.

The valley system utilizes a natural or constructed retaining structure such as a ridge or dam located at the bottom of a canyon or valley. Ore is then stacked behind the structure in progressive lifts to fill the canyon or valley. Total land area required is usually less than that required for an expanding system, but lining the underlying surface may be difficult due to slope and terrain features.

Solution containment in such one-time structures is addressed in the following segment, while stability is covered here. Stability depends on a number of factors including the strength and slope of the foundation, the load imposed by the stacked ore, and the build-up of a solution saturated zone at the foundation or elsewhere in the rock mass. The latter is critical as more heap and dump failures have been caused by the build-up of excessive pore pressure than any other cause (Caldwell and Moss, 1985). Without going into the mathematical details, the reason for this is that the pore pressure within a saturated zone will reduce the shear strength and frictional resistance of stacked material. As pore pressure builds, especially near a free face, it can cause the shear strength to drop below the loading imposed by the rock mass. At this point, the structure fails, often catastrophically.

The build-up of a saturated zone depends on several factors such as the permeability of the rock mass, the slope of the foundation, the length of the drain path along the foundation, and the application rate on the heap or dump surface. The latter illustrates another reason for avoiding excessive application rates, particularly those above q_{max}.

Obviously, the foundation design can be changed to mitigate against the development of a large saturated zone. This could be something as simple as the installation of drain lines along the foundation. These would serve as draw-down points for the saturated zone. In rare cases, a hole could be drilled into the saturated zone and then cased, with the water pumped out. This has actually been done on large waste dumps where water was building up behind an internal barrier and could not drain normally.

SOLUTION CONTAINMENT. An effective liner is the principal barrier to seepage losses. This applies not only to heap or dump foundations but also to catchment basins, flow channels (including ditches and trenches), and holding ponds. In addition, the point should be made that no containment system is perfect. Some seepage will occur and can be measured by Darcy's Law. As indicated in Eq. 15.2.17, the permeability of the liner and the hydrostatic head are the factors that control seepage. Thus the engineer must design the system so that the expected seepage level is within environmentally acceptable limits.

Strachan and van Zyl (1988) cover liners in detail, including liner selection, system design, and installation. Selection is based on the type of application, nature of the solution to be contained, physical loading, and exposure to the elements. Liner materials include clays, clay or chemically amended soils, and geomembranes (PVC, Hypalon, HDPE, asphalt, etc.). These may be used alone as single liners or in combination. When used in combination, the upper liner is usually a geomembrane while the lower one is a soil material. On double or triple liners, a leak-detection system is often installed in a porous zone between the layers.

A final point should be made about the use of systems that minimize seepage losses. These not only meet regulatory requirements, but they also make good sense from an operational perspective. Even a small leak, if undetected, can cause a major loss in metal values and reagents. Such losses are equivalent to lowering the ore grade or getting poorer metallurgical recovery. In many cases, the dollar value of the lost production equals or exceed the cost of a good solution-containment system.

15.2.5 COMMODITY RECOVERY SYSTEMS

The systems used in connection with recovery of values from heap and dump leaching operations are the province of the hydrometallurgist. The recovery systems themselves do depend on the lixiviant but are essentially independent of ore emplacement or solution management practices. Thus the unit operations are commodity specific and are covered in standard metallurgical texts and reference volumes. (See, for example, Ross and Mackiw, 1985). For completeness, current recovery systems are briefly described as follows.

COPPER. Historically, copper has been recovered as an impure metallic mud or "cement" by cementation (precipitation) on scrap iron. The product typically grades 70 to 85% copper (dry basis) and requires smelting and refining prior to sale. Now scrap iron cementation is rapidly being replaced by solvent extraction-electrowinning, a process that produces directly salable cathode copper at the mine site (see Chapter 25.4).

GOLD/SILVER. Pregnant solutions from heaps and dumps are currently treated in one of two ways. One is absorption on activated carbon granules packed in columns. The gold and silver cyanide complexes load on carbon that is periodically stripped into a concentrated cyanide solution. The gold and silver are then recovered by electrolytic deposition from the strip liquor. As an alternative, the gold and silver may be stripped from solution as a sludge using zinc precipitation (the Merrill-Crowe process; see Chapter 25.4). The precipitate is then dried and smelted to produce doré metal for refining. Since carbon is less effective for silver recovery than gold, high silver solutions are usually treated by the Merrill-Crowe process.

URANIUM. The relatively low-grade leach liquors from heaps and dumps are usually concentrated by ion exchange, then purified and concentrated further by solvent extraction. Finally, yellow cake (U_3O_8) is precipitated from the solvent extraction strip liquor.

The only potentially unusual feature of the various recovery systems is conditioning of the barren solution prior to recycling it to the heaps or dumps. Since most of these facilities must operate under conditions of zero discharge, use of a bleed stream to control the build-up of undesirable impurities may not be possible. Thus the conditioning may be done to control scale-forming constituents, remove potentially harmful impurities, replenish reagents that have been consumed, adjust solution Eh or pH, or simply make up the water lost to evaporation. Conditioning will be highly site specific and will depend on the ore-lixiviant system, gangue reactivity, and even the chemical makeup of the available process water. Overall conditioning requirements for heap and dump leaching are less than those needed for in situ mining. This comes about because of the sensitivity of the in situ formation to loss of permeability caused by suspended or precipitated solids building up and blocking the flow paths between the injection and recovery wells.

15.2.6 ANCILLARY AND INFRASTRUCTURE REQUIREMENTS

Ancillary and infrastructure requirements are much the same as those for any open pit mining operation. This is because heap and dump leaching are merely the ore processing operation associated with mining. The main considerations are electric power, water, a transportation system, and access to a sufficient labor force. Fuel (e.g., natural gas) is not a concern since there are generally no large furnacing or other heating operations. Building heat, small precious metal furnaces, and driers can easily be fired electrically or with bottled gas if there is no natural gas line available.

In general, ancillary and infrastructure requirements are lower for primary heap and dump leaching operations than they are for mine-mill operations. Thus leaching plants are well suited to small operations and those in remote locations. A good example is in site access. A mine and mill complex requires a rail line or a heavy-duty road, or both in order to move ore from the mine to the mill and concentrate from the mill to a smelter. However, the leach operation merely requires an all-weather access road since shipping is limited to receipt of reagents and other supplies and shipment of small tonnages of crude or refined product. Acid trucks or similar vehicles are likely to be the largest units entering or leaving the plant on a regular basis.

Labor requirements are also low in leaching operations, with most of the people actually needed for the mining activities. Exclusive of mining, labor may represent no more than 5 to 15% of the direct operating costs. The low labor requirements also mean that much of the work force can be recruited locally, even in rural areas. However, a camp arrangement may still be required at very remote sites. Specialized services such as instrument and computer system maintenance may also have to be done on a contract basis.

Power requirements will vary widely, depending on the type of operation. Gold plants generally have low requirements that go mainly for pumping relatively small solution flows and running small heaters and furnaces. As a result, many remote gold plants use diesel generator sets or other types of onsite generation as the power is cheaper and more dependable. On the other hand, a large copper leach operation is power intensive. With flows as high as 40,000 to 50,000 gpm (2.5 to 3.2 m^3/s), water distribution systems may include multistage pumps with 500- to 1500-hp (37.5- to 110-kW) motors. Copper electrowinning is also power intensive, requiring at least 1 kWh/lb Cu. Thus power is usually purchased from a utility company. This power will be a big factor in the operating costs and may incur a high capital cost if an extensive pole line with transformers, rectifiers, and substations is required.

Water is another requirement that must be considered. There are two separate aspects: startup requirements and steady state needs. Startup requirements will be the higher of the two. The main reason is that the various catchment basins and recovery plant feed and tail water reservoirs must be filled. The heaps and dumps must also be saturated with water before outflow will occur. This represents a significant solution inventory since the rock mass will retain 8 to 12% moisture by weight. This is about 0.1 gal/ft^3 (13L/m^3) of heap or dump material.

Steady state requirements will be less because these cover only evaporation and other miscellaneous losses. Since most plants will have to operate under zero discharge permits, no net makeup will be required to compensate for bleed off. The exact make-up requirement will generally vary seasonally with the local evaporation rate. For copper in the southwestern United States, a typical annual average figure is 10 to 15 gal/min/ton (35 to 50 L/min/t) of daily production.

15.2.7 CASE HISTORIES

Two case histories are presented, one for copper and one for gold. Any individual facility will embody many site-specific design criteria. The histories presented here are developed by combining information from several actual or proposed operations. The resulting composites are representative of the medium-sized leach plants that are being developed today.

As with any mining and ore processing operation, the operating costs for surface leaching plants are driven by the mining strip ratio, ore grade, and the percentage metal extraction. This is because the mining cost per total ton of material moved is not particularly site specific; that is, the cost probably does not vary by more than 10 or 15%. However, the strip ratio, ore grade, and metallurgical recovery will vary widely, depending on the site and nature of the ore. Thus the cost per pound or ounce of product recovered varies by a much greater percentage than the mining cost per ton of material.

15.2.7.1 Copper Leach Project

PROJECT DESCRIPTION. The copper leach project involves exploitation of a modestly sized copper oxide deposit in the southwestern United States. The terrain is fairly rugged, with elevations ranging from 5800 to 6300 ft (1900 to 2065 m). The climate is dry and mild with about 10 in. (250 mm) of precipitation a year. This comes in the form of short but intense summer thunderstorms, plus occasional winter storms (rain or snow). Freezing conditions are generally limited to winter nights.

A thorough exploration drilling program has shown that the ore body can be mined as an open pit and contains 13 million tons (11.8 Mt) of minable ore with an average grade of 0.6% acid soluble copper. The cutoff grade is taken as 0.3% Cu. The mine plan calls for a strip ratio of 2.0:1, waste to ore. However, this includes 7 million tons (6.4 Mt) of low-grade leach material, averaging 0.25% acid soluble copper at a cutoff grade of 0.1% Cu. The balance of the waste is considered barren.

Metallurgical test work shows that the high-grade ore warrants crushing from ROM to –2 in. (–50 mm) as this increases copper extraction from 60 to 80%. The extraction rate also increases so that 60% of the copper is recovered in a 45-day leach cycle. Then 50% of the remaining copper is extracted in a long-term leach. Only 60% of the copper in the low-grade fraction is extracted from the ROM material when leached in permanent leach dumps. Overall acid consumption is 7.65 lb acid/lb Cu.

In order to balance capital requirements against revenues and to fully amortize the process equipment, a 10-year mine life was selected. This provides a steady state cathode output of 7300 tpy (6600 t/a).

PROCESS DESCRIPTION. The contract mining operation segregates the mined material into three categories: high grade, low grade, and barren. The respective mining rates for each are 1.3, 0.7, and 1.9 million tpy (1.2, 0.6, and 1.7 million t/a). Mining is done in two 10-hr shifts/day, 4 days/week, year round.

Crushing and heap stacking operate on the same schedule as the mine. High grade is hauled to a 30- by 40-in. (760- by 1000-mm) single toggle, skid-mounted jaw crusher set. The crusher has a 380 tph (345 t/a) capacity. This is comfortably oversized and provides the flexibility to match variations in mine output and make up production lost during downtime.

With a closed side setting of 8 in. (200 mm), the crusher produces a nominal 312 tph (284 t/h) of ore with 80% passing 7.5 in. (190 mm). This material is moved by overland conveyor 1800 ft (550 m) to the permanent pad area. Here the crushed ore is screened at 1.5 in. (38 mm) using a 6- by 10-ft (1800- by 3000-mm) double-deck vibrating screen. Screen oversize (approximately 250 tph or 225 t/h) is fed to a 5.5-ft (1700-mm) diameter standard cone crusher with a l-in. (32-mm) closed side setting. Screen undersize and crusher discharge are combined, conveyed to the leach pad, tripped onto a traveling stacker with a 10 ft (3 m) stacking arm, and loaded onto the pad.

The permanent pad area is 365 by 840 ft (110 by 255 m). It is constructed with a layered composite of asphalt and geotextile fabric laid over a prepared base. The latter contains a network of perforated pipe to serve as a leak detection system. The asphalt is overlaid with 2 ft (600 mm) of crushed and screened rock (2 by 8 in. or 50 by 200 mm). This layer serves as a permanent protective liner and also provides good solution drainage from the heap.

Ore is stacked 20 ft (6.1 m) high and leached in a single lift. This provides more effective solution-rock contact than the multiple-lift approach (see following case study). Ore is leached for 45 days. The pad is operated in a continuous cycle, with some material under leach, while other zones are being loaded with fresh ore or unloaded. The partially leached ore is drained and then reclaimed and sent by conveyor 1800 ft (550 m) to the leach dump area where it is co-mingled with the ROM low grade.

The dump leach area is in an isolated canyon. This requires about 500,000 ft^2 (46,450 m^2) of area that was cleared and compacted. Where necessary, a chemical soil amender or a spray-on asphalt coating was used to reduce dump foundation permeability. As an additional precaution against solution loss, a grout curtain from the surface to bedrock was installed across the mouth of the canyon. A solution recovery well was installed on the canyon side to recover any subsurface seepage, and monitor wells were installed down gradient from the grout curtain. These can be pumped back into the PLS pond if a leak occurs in the grout curtain.

Both the high- and low-grade material are leached with a network of drip emitters installed 30 in. (760 mm) apart and fed from HDPE headers. Emitter operating pressure is maintained at 20 psi (140 kPa). Both the tops and sides of the leach areas are irrigated at a flow rate of 0.007 gal/min/ft^2 (17 mm/hr). The lixiviant is SX plant raffinate acidified to 10 g/L with sulfuric acid.

Both leach areas have separate PLS ponds, each with 330,000-gal (1250-m^3) capacity. These are constructed by a cut-and-fill technique, then compacted and lined with 40-mi1 (l-mm) HDPE sheeting. Each pond has a vertical barge-mounted pump equipped with a 100-hp (75-kW) motor and can deliver 1100 gpm (70 L/s) against a head of 200 ft (60 m). The dump leach pond has the capacity to recirculate solution to build up the copper tenor.

The pad leach delivers 840 gpm (53 L/s) of PLS at a heading of 2.36 gpl copper. The dump leach area contributes 966 gpm (60 L/s) of PLS at a heading of 1.68 g/L copper. These flows are combined in the 650,000 gal (2460 m^3) SX plant feed pond constructed like the other ponds. This provides a 4-hr operating capacity and delivers 1800 gpm (110 L/s) of PLS at an average copper tenor of 2.0 g/L. Flow from the feed pond to the SX units is by gravity.

Both the SX and EW facilities are over-designed by 10% to accommodate the normal variations in leach extraction. The SX plant consists of two stages of extraction and one stage of stripping (2 x 1 circuit), all operated in series. In the first two stages, copper is extracted from the PLS and loaded onto an organic extractant (12% by volume) carried in a kerosene-type diluent (88% by volume). Each extraction unit has a 3-stage mixer with a total retention time of 3 min. The organic to aqueous (O/A) ratios are 1.0/1.0, and the settlers are designed for a settling rate of 2.0 gal/min/ft^2 (4.9 $m^3/m^2/h$). Settlers are 48 ft (15 m) wide, 56 ft (17 m) long, and 40 in. (1000 mm) deep. Copper recovery from the PLS is 95% with a 95% plant availability.

In the stripping unit, the loaded organic (4.3 g/L Cu) is contacted with 345 gal/min (22 L/s) of lean electrolyte (35 g/L Cu and 160 g/L sulfuric acid). This strips copper from the organic, generating a rich electrolyte containing 55 g/L Cu and 146 g/L acid. This unit is similar to the extraction units, except that mixing is only 2-stage, and the overall O/A ratio is much higher (5.3:1).

From solvent extraction, the rich electrolyte is returned to the EW tankhouse via the electrolyte filter and recirculation tank. Barren leach solution returns to the 500,000-gal (1900-m^3) raffinate pond. There the leach solution is reacidified, and about 300 gpm (19 L/s) of make-up water is added to offset losses in leaching that are due to evaporation and entrainment in the ore. The raffinate pond has two operating vertical pumps, each with 150-hp (110-kW) motors. These can deliver 1100 gpm (70 L/s) against a 300 ft (90 m) head. One pump normally feeds each leach area.

The rich electrolyte is processed in the tankhouse to produce high-quality cathode. The ISA process system of permanent stainless steel cathode blanks is used instead of having a separate electrolyte circuit for production of copper starter sheets. The corresponding anodes are lead, alloyed with tin and calcium for corrosion control.

The tankhouse contains 36 cells, each holding 39 cathodes and 40 anodes connected by copper buss work. Cells are constructed of concrete with wooden floors and PVC para-liners and buffer sheets. Electrolyte is heated to 115°F (46°C) and flows through the cells at a rate of 0.75 gpm/cathode (0.003 L/s/cathode). A rectifier supplies 22,500 amps at 80 v dc. Primary voltage is 4160 v. The operating current density is 23.2-amps/ft^2 (250-amps/m^2). Current efficiency is 95%.

Operationally, one-third of the cathodes in a cell is pulled at a time. These are washed free of electrolyte, then manually stripped from the stainless steel blanks. The normal pulling cycle per cathode is 7 days, giving 190 to 200 lb (85 to 90 kg) cathodes for bundling and shipment. Slightly over 200 cathodes are normally stripped one shift per day to meet the 20-tpd (18-t/d) output.

The tankhouse itself has only a roof and partial siding to improve the ventilation needed to control the sulfuric acid mist arising from the cells. An overhead crane is provided to move cathodes (and anodes) as appropriate.

In addition to normal utilities and ancillaries, the copper leach project includes a tank farm. This contains the storage and circulating tanks for the SX and EW operations. The tank farm area is located below the SX and EW operations. This permits gravity flow to the tank farm and low head pumping back to the other areas. The SX area of the tank farm includes the barren organic surge tank, diluent storage tanks, a holding tank, a gunk tank, and the backwash bleed tank. The EW area contains the electrolyte filter and feed tank, the electrolyte heat exchanger and package boiler, the electrolyte recirculation tank, and the lean electrolyte tanks. The filter is a combined media (anthracite and garnet) automatic backwash unit. This unit is essential for maintaining electrolyte purity and ensuring production of high-grade cathodes.

CAPITAL COSTS. The capital cost (total installed basis) for the facility described above is developed in March 1990 dollars. No capital costs for mining equipment or mine support facilities

Table 15.2.2. 1990 Capital Cost Breakdown for the Copper Leach Project

Code of Accounts	Cost, $1000s
Site Preparation	$ 357
Site Improvements	692
Underground Electrical	23
Underground Piping	92
Concrete and Excavation	622
Specialty Items	3
Structural Steel	204
Building	256
Aboveground Piping	930
Aboveground Electrical	1,429
Instrumentation	94
Insulation and Painting	15
Major Equipment	4,649
Proratables	120
Total Construction Activities	$ 9,486
Escalation and Premium Pay	488
Total Direct Cost	$ 9,974
Construction Equipment	502
Field Indirects and Fee	1,235
Total Field Cost	$11,711
Engineering	800
Gross Receipts Tax	600
Water Wells/Supply	675
Initial Reagent Fill	630
Capitalized Spares	85
Working Capital (3 mo. operating cost)	1,740
Licensing Fee	120
Project Capital Cost	$16,361

Table 15.2.3. Staffing Requirements for the Copper Leach Project

Plant Area	Number of People: Supervisory	Labor
Administration	3	1
Mining and Geology	2	3[a]
Leaching	1	4
SX-EW	5	11
Maintenance	1	5
	12	24

a. Exclusive of contractor.

are included as the contract miner has responsibility for these. There is no allowance for housing or a construction camp. Any of the owner's costs incurred before the start of engineering are also excluded. This exclusion applies to permitting, although the facility was designed and constructed in accordance with regulations in effect at the time.

The total capital cost is $16,361,000. The cost breakdown is given in Table 15.2.2. This is based on a typical construction code of accounts. (Compare with the next case study where a process-area breakdown is used.) These costs exclude any sale or ad valorem taxes, the environmental monitoring system, and the power line upgrade. The latter is provided by the local utility with cost recovery through a rate adjustment.

OPERATING COSTS. The direct operating costs for the copper leach project are developed in March 1990 dollars. These include supervision, labor, supplies, and utilities. Staffing requirements are shown in Table 15.2.3.

Table 15.2.4. 1990 Operating Costs for the Copper Leach Project

Area	Annual Costs, $1000s	Cost $/lb Copper
General and Administrative	$ 454	$0.031
Mining[a]	3691	.253
Crushing and Pad Loading	353	.024
Leaching[b]	1793	.123
SX-EW	2796	.191
Maintenance	780	.050
Total Direct Operating Costs	$9867	$0.675

a. Includes transfer of ore from the pad to the leach dump.
b. Includes purchase of 55,845 tpy (50,770 t/a) acid @ $25/ton($28/t).
Conversion factor: 1 lb = 0.4536 kg.

Operating costs are presented in Table 15.2.4. Labor costs include a payroll burden of 32%. General and administrative costs include local taxes and insurance, but federal taxes are excluded. Also excluded are depletion, depreciation, capital financing charges, and any corporate allocations.

Mining is the single most expensive component of the operation. Contractor rates include $0.65/ton ($0.72/t) to mine and haul ore to the crusher, $0.85/ton ($0.94/t) to mine and haul barren and low grade, and $0.30/ton ($0.33/t) to move leached ore from the pad to the leach dump. These are substantially less than the unit costs in the following case study. The differences reflect shorter truck hauls, use of conveyors for ore movement, a longer-lived operation, and continuous (year-round) operation. Costs in the following case reflect the three-month idle period and the need to mobilize and demobilize annually.

Originally, the copper leach project was planned as a conventional heap leach operation. This called for the ROM ore (+0.3% Cu) to be leached on a single-use pad. Ultimately, this pad would have been covered with three 30-ft (9-m) lifts. Overall copper recovery would have been only 70%; no low-grade leaching was planned. This approach saved about $1 million in capital costs over the scenario described above. However, less copper was produced per ton mined, so more ore would have to be mined per year to reach the desired production level. This reduced mine life to seven years and increased operating costs by $0.11/lb ($.24/kg) of copper. Thus it was a less attractive option and was not pursued.

15.2.7.2 Gold Leach Project*

PROJECT DESCRIPTION. The gold leach project is located in northern Nevada. The setting is typical basin and range, with rolling hills and an average elevation of 6500 ft (2000 m). Subfreezing temperatures are prevalent during the winter so that mining and ore processing are conducted in a 9-mo/yr schedule (March through November). However, use of drip emitters allows for a year-round leach operation. Water is available from wells on the property, and power is available from the local utility.

The ore body is mined by open pit techniques and contains 7 million tons (6.4 Mt) of minable ore at an average grade of 0.05 oz/ton (opt) (1.7 g/t) gold and 0.12 opt (4.1 g/t) silver. For

*Case study contributed by J.H. (Jack) Templeton, Brown & Root Braun, Houston, TX.

purposes of illustration, two different waste-to-ore strip ratios are used: 0.75:1 and 2.0:1.

Column leach tests on agglomerated ore samples from throughout the deposit show average gold and silver extractions of 85 and 40% respectively. Similar recoveries are achieved in the operation, but leach times are extended to 60 days.

PROCESS DESCRIPTION. Contract mining is employed and uses conventional drilling, blasting, loading, and hauling operations described elsewhere (see Section 9). ROM ore is delivered to the crushing plant or the adjacent 10,000-ton (9100-t) stockpile. Both mining and primary crushing operate 40 wk/yr, 4 days/wk, two 10-hr shifts/day, giving 160 operating days/yr. Production is 8750 tpd (7950 t/d), equivalent to 1.4 million tpy (1.3 Mt/a) of ore.

The ore is normally dumped directly into a 100-ton (90-t) live-capacity hopper equipped with a fixed grizzly with 16-in. (400-mm) bar spacing. Any oversized rock is broken with an air-driven rock breaker mounted on a hydraulic boom. Ore is withdrawn from the hopper with an apron feeder. This feeds a single toggle jaw crusher (36 by 48 in., or 910 by 1200 mm) with a 9-in. (230-mm) closed-side setting. The primary crusher discharge feeds a 20,000-ton (18,200-t) live-capacity coarse ore stockpile.

This stockpile, nominally –8 in. (–200 mm), feeds the fine crushing plant. A single conveyor belt is fed by two under-pile vibrating feeders. The ore is delivered to two stages of fine crushing that are operated 6 days/wk, two 10-hr shifts/day, on the 40-wk/yr schedule. Production through this circuit is 5830 tpd (5300 t/d). Fine crushing is done in open circuit with an 8- by 20-ft (2400- by 6100-mm) double-deck vibrating screen equipped with 0.5-in. (13-mm) slots. The first crusher is a 7-ft (2130-mm) diameter standard cone crusher with a l-in. (25-mm) closed-side setting. This unit discharges to the screen. The screen oversize feeds a 7-ft (2130-mm) diameter short-head cone crusher. This uses a closed side setting of 3/8 in. (9.5 mm). Short-head crusher product is 80% –0.5 in. (–13 mm). When combined with the screen undersize, the agglomeration feed is 86% minus 0.5 in. (13 mm).

Water sprays at crusher and conveyor discharge points control dust emissions and provide an agglomeration feed moisture of 8%. The agglomeration immobilizes fines (–65 mesh or –0.2 mm) and improves heap permeability during leaching. The ore contains less than 5% clay and can be agglomerated by adding 10 lb/ton (5 kg/t) Portland cement and 3 lb/ton (1.5 kg/t) hydrated lime to the fine ore. The constituents are intimately mixed on three conveyor belts operating in series. Strong cyanide solution (1%) is sprayed onto the mixture at each free-fall point. This increases the moisture to 13% and provides about 50% of the sodium cyanide needed for leaching. This practice also starts gold dissolution before the agglomerates ever reach the heaps. To maximize the agglomerate strength, a minimum 72-hr curing period is used. This minimizes decrepitation of the agglomerates during leaching.

The agglomerated ore is transferred to overland conveyors and moved 1800 ft (550 m) to the leach area. This consists of three single-use leach pads or cells, each 600 by 100 ft (180 by 30 m). Ultimately, each cell will contain four 18-ft (5.5-m) lifts of agglomerated ore. The pads are sloped at a grade of 3% to facilitate drainage of PLS from the heaps. To meet environmental restrictions, the pads consist of a graded, compacted clay liner 6 in. (150 mm) thick. This is overlaid with a 40-mil (l-mm) HDPE liner. Liner protection is provided by a geotextile overlay bedded with 8 in. (200 mm) of –0.5 in. (–13 mm) crushed low-grade ore. Collection of PLS is enhanced by bedding 3-in. (76-mm) perforated plastic pipe in the crushed material.

For purposes of control, each cell is divided into four equal subcells. As soon as a subcell is available, it is leached with solution from the barren solution pond. This contains 1.0 lb NaCN/ton solution (0.5 g/L) at pH 10.5. Both the top and sides of the subcell are irrigated with a drip emitter system at a flow rate of 0.005 gal/min/ft^2 (12 mm/h). Leaching is continued for 60 days, with the PLS draining to the PLS pond. Average gold and silver contents in the PLS are 0.018 and 0.022 opt of solution (0.62 and 0.75 g/L), respectively. Following the 60 days of leaching, the subcell is allowed to drain and dry out for another 60 days. Then another lift of ore is added, and the leach cycle is repeated.

To avoid needless dilution of values, no rinsing will be done until the final lift has been leached. Following rinsing, the rinse water will be neutralized with an excess of dilute hypochlorite solution to destroy the residual cyanide. The neutralized solution will then be used to detoxify the heap.

Both the PLS and barren solution ponds have 2.4 million gal (9100 m^3) capacities and are lined with 40-mil (1-mm) HDPE. There is a third pond of the same capacity that holds the fresh water supply and contains any overflows or runoff. A fourth much smaller pond (35,000 gal or 130 m^3) is located at the process plant. This is designed to contain waste water and slurry streams from the carbon handling circuits. All ponds are fenced to exclude livestock and wildlife.

PLS is advanced continuously from the PLS pond to the carbon adsorption section at a flow rate of 1580 gpm (100 L/s). This section contains five 10-ft (3000-mm) high by 10-ft (3000-mm) diameter columns that are operated in series. Each column contains 2.6 dry tons (2.4 t) of 6- by 16-mesh (3.36- by 1.00-mm) activated carbon.

Solution advances from column to column by gravity due to the 3.5-ft (1100-mm) elevation difference between columns. Nominal solution retention time is 2 min/column. Carbon is advanced counter-currently using recessed impeller centrifugal pumps. Final carbon loading is 135 opt (4620 g/t) gold and 155 opt (5300 g/t) silver, with 95% gold adsorption and 90% silver adsorption from the PLS. The stripped solution is discharged back to the barren solution pond. The gold and silver contents of the barren solution are 0.0009 and 0.0022 opt (0.031 and 0.075 g/t) of solution, respectively.

Carbon is advanced from column to column at a rate of 1.3 tpd (1.2 t/d). Loaded carbon exiting the last column is dewatered and rinsed on a 3- by 6-ft (900- by 1800-mm) vibrating screen and is discharged into a conical bottom storage tank (3.5-ton or 3.2-t capacity). This allows for one 3.5-ton (3.2-t) batch to be processed 3 days/wk.

Processing utilizes a modified Zadra stripping technique. It is performed in a pressure stripping vessel operated at 35 psi gage (240 kPa) and 280°F (138°C). After being pumped into the vessel, the carbon batch is contacted with caustic cyanide solution (1% NaOH, 0.1% NaCN), which solubilizes the precious metals adsorbed on the carbon. Ten bed volumes (18,000 gal or 68 m^3) of strip solution are passed through the vessel to dissolve the gold and silver, a process requiring about 20 hr. The pregnant strip solution (PSS) is held in an insulated storage tank before being advanced to electrowinning.

The PSS exiting the stripping column passes through a plate and frame heat exchanger and warms the barren strip solution to 165°F (74°C). The warmed solution then passes through a second heat exchanger where live steam heats the strip liquor to 290°F (143°C).

Gold tenors in the PSS are increased and lower residual gold concentration on the stripped carbon are achieved by splitting the PSS. The first two-thirds of the PSS is sent directly to the electrowinning storage tank. However, the remainder is stored and used as the initial strip solution for the next batch of carbon. This also reduces the flow to electrowinning by one-third. Recovery of precious metals from the carbon averages 93% for gold and 90% for silver. The stripped carbon contains about 9.5 opt (325 g/t) gold and 15.5 opt (530 g/t) silver. This carbon is advanced to the reactivation circuit.

When a batch of carbon has been stripped, the PSS storage tank will contain 12,000 gal (45 m^3) of solution containing 428 opt (14,640 g/t) gold and 484 opt (16,600 g/t) silver. This solution is first pumped to the primary electrolytic cell at 10 gal/min (0.6 L/s). About 80% of the gold and silver are recovered here. The partially stripped PSS is then recirculated through a secondary cell at twice the flow rate until the gold content is reduced to a minimum of 10 ppm. This raises overall recovery to 89% for both gold and silver. At this point, barren strip solution is drained to the barren strip solution tank.

The primary and secondary cells are identical. Each has a volume of 20 ft^3 (0.6 m^3), with nine stainless steel wool cathodes and stainless steel plate anodes. Cell buss work is copper. Direct current is supplied to the cells at 168 amps and 6 v. Current density is 14 amps/ft^3 (490 amps/m^3) of cathode, and current efficiency is 30%. Solution residence time in the primary cell is 15 min. Total time to process a batch of PSS through the primary cell is 20 hr.

About 80% of the recovered precious metals deposit on the steel wool. The remaining 20% collects as a sludge on the cell bottoms. After each electrowinning cycle, half the loaded cathodes are removed from the primary cell and the other half are advanced. Sludge is collected once a week. The same operations are performed once every two weeks in the secondary cell.

The cathodes and dewatered cell sludge are dried and mixed with appropriate fluxes (sodium nitrate, sodium borate, borate, silica, and sodium carbonate). Then the mixture is charged to a l-ft^3 (0.03-m^3) induction furnace. When the charge is fully molten, the slag is poured off, and then the remaining doré metal is cast into a button. This doré is remelted in a smaller furnace (0.2 ft^3 or 0.006 m^3) and cast into bullion bars containing 47% gold and 53% silver. (This assumes a lack of contaminants, such as copper.) Total physical metal production is 59,500 oz/yr (1850 kg/yr) gold and 67,200 oz/yr (2090 kg/yr) silver.

The stripped carbon is either regenerated or returned directly to the carbon adsorption circuit, depending on its reactivity. However, every batch of carbon is washed with 5% nitric acid before regeneration. The acid wash dissolves the calcium carbonate that has deposited in the pores of the carbon. Washing with deionized water and then a 10% caustic solution removes any residual acidity. This is always a concern when sodium cyanide is one of the principal reagents used in the operation.

The washed carbon is dewatered and then reactivated. This requires heating the carbon to 1200°F (650°C) in a horizontal rotary kiln in an atmosphere devoid of oxygen. Carbon is treated at a rate of 200 lb/hr (90 kg/h). Retention time at temperature is 20 to 30 min. Great care must be exercised to avoid overheating or infiltration of air. These can greatly reduce carbon reactivity.

As an alternate to carbon adsorption, stripping and electrowinning, the gold and silver in the leach solution could be recovered by the Merril-Crowe zinc precipitation process. Here the addition of zinc precipitates both gold and silver from the clarified and deaerated PLS. The precipitate is then smelted to produce doré metal. Zinc precipitation is generally favored when silver contents are high. Since silver loadings on carbon are much lower than gold loadings, high silver contents mean that a very large amount of carbon must be handled. Also the cost for carbon stripping per ounce of gold recovered is increased. This creates materials handling problems and increases precious metals in inventory.

Table 15.2.5. 1990 Capital Cost Breakdown for the Gold Heap Leach Project

Area	Cost, $1000s
Crushing Plant	$ 2,019
Agglomeration and Pad Loading	2,431
Heap Leaching and Solution Ponds	2,283
Carbon Adsorption, Stripping, and Regeneration	1,442
Electrowinning and Refining	856
Reagent Handling	106
Buildings	510
Utilities	80
Total Direct Cost	$ 9,727
Indirects (supplies, construction equipment, general expenses)	486
Contractor's Fee	306
Engineering	778
Working Capital (3 mo. operating cost at a 0.75:1 strip ratio)	4,263
Project Capital Cost	$15,560

Table 15.2.6. Staffing Requirements for the Gold Heap Leach Project

Plant Area	Number of People Supervisory	Labor
Administration	3	6
Mining and Geology	4	4[a]
Operations	8	40
Maintenance	1	7
Total	16	53

a. Exclusive of contractor

In addition to the process facilities themselves, the normal types of infrastructure will be required. These include reagent receiving, storage, and mixing. The principal reagents are fresh carbon (delivered in drums), sodium cyanide (delivered as prills or briquets), cement and lime (delivered in bulk), caustic soda (delivered as flake or pellet, or as a 50% solution), nitric acid (64.2% solution), and an anti-sealant to keep the emitters and leach lines clear.

CAPITAL COSTS. The capital cost (total installed basis) for the facility described above is developed in June 1990 dollars. No capital costs for mining equipment or mine support facilities are included as the contract miner has responsibility for these. There is also no allowance for housing or camp costs. Owner's predevelopment costs are excluded from the estimate. These include land acquisition, exploration and in-fill drilling, assaying and test work, geotechnical and hydrologic studies, environmental baseline studies, and permitting. However, the capital costs do reflect a facility that is designed and constructed to meet all environmental regulations in effect in Nevada at the time of construction (early 1990). The leach pads required for the first year of operation are capitalized. Pads for subsequent years are expensed in the year incurred.

The total capital cost is $15,560,000 for the case where the strip ratio is 0.75. The cost breakdown is presented in Table 15.2.5. The higher strip ratio will increase needed working capital.

OPERATING COSTS. The direct operating costs for the gold heap leach facility are developed in June 1990 dollars. These include supervision, labor, supplies, and utilities. Staffing requirements are shown in Table 15.2.6. Labor costs include a

Table 15.2.7. 1990 Operating Costs for the Gold Heap Leach Project at a Strip Ratio of 0.75:1

Area	Annual Costs, $1000s	Costs, $/oz Gold
General and Administration	$ 1,198	$ 19.83
Mining	4,386	72.61
General Plant Operation	912	15.09
Crushing, Agglomeration, and Pad Loading	1,840	30.46
Leaching	1,792	29.66
Carbon Loading and Regeneration	455	7.54
Stripping, Electrowinning, and Refining	420	6.95
Maintenance	626	10.36
Total Direct Operating Costs	$11,629	$192.50

Conversion cost: 1 oz = 28.35 g.

Table 15.2.8. 1990 Operating Costs for the Gold Heap Leach Project at a Strip Ratio of 2.0:1

Area	Annual Costs, $1000s	Costs, $/oz Gold
General and Administration	$ 1,198	$ 19.83
Mining	6,486	107.38
General Plant Operation	912	15.09
Crushing, Agglomeration, and Pad Loading	1,840	30.46
Leaching	1,792	29.66
Carbon Loading and Regeneration	455	7.54
Stripping, Electrowinning, and Refining	420	6.95
Maintenance	626	10.36
Total Direct Operating Costs	$13,729	$227.27

Conversion factor: 1 oz = 28.35 g.

35% payroll burden, a 10% overtime allowance for hourly employees, and a 5% production bonus for supervisory positions. Local taxes and insurance are included, but federal income taxes are excluded. Depletion, depreciation, capital financing charges, and corporate (offsite) overhead allocations are also excluded. However, a 10% contingency is included.

Operating costs for the two strip ratios (0.75 and 2.0) are shown in Tables 15.2.7 and 15.2.8. These costs are based on a contract mining price of $1.20/ton ($1.32/t) of ore and waste mined. This figure includes the contractor's direct costs, general overhead, depreciation, and profit. The impact of mining is obvious from the tables. Even at the low strip ratio, mining accounts for over one-third of the operating costs or $73/oz ($2/g) of gold. Increasing the strip ratio to 2.0 adds about $35/oz ($1/g) to this cost.

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Chapter 15.3
SOLUTION MINING: IN SITU TECHNIQUES

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In situ solution mining is a true mining technique. It represents an alternative to mechanical excavation and is applicable to a wide range of commodities that are soluble in water or an aqueous lixiviant. Since most waste components are not soluble, the gangue constituents tend to be left in place as the values are solubilized and removed to a surface processing facility. As a result, in situ operations are characterized by a singular absence of mine waste piles and tailings impoundments. However, in situ mining does carry a significant potential for contamination of subsurface water. As a result, flow control is a major factor in the design of in situ well fields.

Chapter 15.3 is divided into three subchapters on the basis of the characteristics of the minerals being recovered by in situ techniques. Water soluble salts are addressed in Chapter 15.3.1. These include such minerals as potash, trona, nahcolite and, particularly, common salt (NaCl). In addition to a high degree of water solubility, these salts are typically found in extensive beds where the salt itself is the matrix.

Particularly for common salt, the cavern produced by solution mining may have greater value than the sodium chloride that was extracted. When such caverns are created in salt domes or beds, they have an invariant temperature and a very high degree of inherent stability. Thus the caverns are widely used to store hydrocarbon products and a variety of materials including hazardous wastes. Due to the removal of large contiguous quantities of material, surface subsidence can be a problem in such operations.

The second segment of the chapter, 15.3.2, focuses on Frasch sulfur mining. Frasch minable deposits do not occur in as massive a form as the soluble salts, even though the sulfur is often associated with salt domes. However, the sulfur occurrences are continuous even if they do not form the matrix of the deposit.

Unlike other forms of in situ mining, the water used in Frasch mining is not a true lixiviant. Instead, the water is merely the medium for transferring heat into the deposit. The heat then melts the sulfur so that it will drain to subterranean pools for pumping to the surface.

Hard-rock in situ mining is the final subject, 15.3.3, covered in Chapter 15.3. In this type of operation, the values are often disseminated and discontinuous when compared to soluble salts and sulfur. Another characteristic of hard-rock in situ mining is its similarity to surface leaching (Chapter 15.2), particularly in terms of ore-lixiviant systems.

A final point is that the soluble salts and sulfur can be mined from undisturbed in-place deposits. While hard-rock in situ operations also treat in-place deposits, some disturbance is often required. This takes the form of formation stimulation using such techniques as hydrofracturing which are borrowed from the oil industry. These are used to enhance the permeability of the formation so that the lixiviant can penetrate to the disseminated mineral values more easily.

15.3.1 IN SITU MINING OF SOLUBLE SALTS

DONALD R. RICHNER

15.3.1.1 Introduction

Extraction projects involving the solution mining of water-soluble evaporites have continued to expand rapidly during the past decade.

The selective dissolution, recovery, and processing of different kinds and/or admixtures of evaporite mineral deposits are being studied and field tested in many areas of the world. Commercial operations involving the dissolution of trona (sodium carbonate/sodium bicarbonate), potash (potassium chloride, potassium carbonate), and magnesium chloride have been established in North America and Europe, and are anticipated within South America, Asia, and Africa before 1992.

Problems relating to the selective dissolution and recovery of evaporite minerals (ores) are being approached from many fronts. Well drilling and completion technology now embraces a variety of nonvertical drilling techniques, low-density cements, special drilling fluid compositions, and diagnostic well-testing procedures. A variety of versatile geophysical well-logging tools has been developed and the identification and quantification of many different evaporite minerals and mineral assemblages are now possible. Core laboratory studies now frequently include a combined mineral identification/dissolution rate/rock mechanics investigation that can be directly integrated with a sophisticated geophysical well-logging program. After proper calibration of the two investigative systems, considerable detailed exploration can then be conducted using the more rapid, less-expensive well logging approach.

The dissolution process for mineral wells embraces a number of single-well concepts involving multiple concentric or independent tubing strings, various fluid injection/production depth combinations, as well as several different cavern roof-protection schemes involving "blankets" of liquid hydrocarbons (crude oil, LPG), gases (nitrogen, carbon dioxide), and solids (floatable "cements", hollow spheres, etc.).

Multiple-well interconnections can also be accomplished (when geologic conditions are appropriate) through roof padding, hole deviation, and hydraulic fracturing. Hole deviation technology, particularly horizontal drilling, is being driven by new opportunities for the development of thin petroleum reservoirs. Hydraulic fracturing concepts, fueled by recent rock mechanics investigations related to hydrocarbon production, now envision well interconnection through vertical or inclined fractures, as well as improved success in "horizontal" fracturing.

Underground storage technology (Chapter 24.2) has developed rapidly as a result of large-scale storage operations and

Table 15.3.1.1. Commodities Stored in Solution-Mined Salt Caverns (PB-KBB Inc.)

Solids	Liquids	Gases
Grains	Crude Oil	Natural Gas
Wastes	LPG[a]	Helium
	Ethylene Dichloride (FDC)	Chlorine
	Special Brines	Compressed Air
		Nitrogen
		Carbon Dioxide

[a] LPG (liquified petroleum gas) such as (1) propane, (2) butane, (3) LP mix, (4) ethylene.

research and development programs conducted by various governmental agencies, research institutions, and large petroleum operators. Within the United States, these include the Strategic Petroleum Reserve (690 million bbl or 110 million m^3 of crude oil), the Louisiana Offshore Oil Port (LOOP) (32 million bbl or 5 million m^3 of crude oil), and more than 1200 caverns in salt containing more than 450 million bbl (70 million m^3) equivalents of liquid and gaseous hydrocarbons. These storage facilities have created opportunities for the development of new materials of construction and new techniques for testing and evaluating materials, equipment, and procedures. There are also increasingly more stringent requirements for safety and the protection of the environment from hazards related to the development of such subsurface facilities. Table 15.3.1.1 contains a partial listing of commodities storable within solution-mined caverns.

The strong trend against disposal of industrial and hazardous wastes within landfills has generated considerable pressure toward incineration and burial underground within impermeable strata (Chapter 24.3). Under tightly controlled conditions, certain salt deposits appear to hold promise for the development of waste repositories. Preliminary designs for salt cavern repositories located within Texas have been completed, and permitting (licensing) efforts are presently underway for disposal within several salt domes.

The development and expansion of governmental regulation and control are two of the most rapidly growing forces leading to innovation and change in the subsurface technologies of solution mining, underground storage, and underground disposal.

The business slowdown of the mid-1980s also had a profound effect upon many down-hole technologies. Income and profits were severely squeezed, leading to the liquidation of numerous small, high-cost, and inefficient companies, and to the consolidation of many former competitors. Most services and some materials were heavily discounted. Several organizations used the general malaise in down-hole activity to press forward in research and development, an effort that has now yielded an explosion of new and/or improved equipment, materials, and services.

Still on the horizon: the potential advent of "superconductivity" could lead to a rapid increase in the development of cryogenic underground storage in salt caverns.

15.3.1.2 Types of Evaporite Deposits

The evaporite professional characterizes evaporite deposits as to their *chemical nature* and their *physical form.* Chemistry provides information concerning the products that can be manufactured and the processes that will be involved. Form—the physical attributes of the deposit and enclosing strata—indicate criteria concerning the solution mining system(s) and its geotechnical and economic limitations for producing ore.

Evaporite deposits being explored or developed by solution mining techniques include salt (halite), potash (sylvite, sylvinite, carnallite), magnesia (tachydrite, bischofite), trona (sodium carbonate and sodium bicarbonate), nahcolite (sodium bicarbonate), borates (borax, ulexite, colemanite), and sodium sulfate (mirabilite or glaubers salt, thenardite, etc.).

The form (shape, attitude, extent) of these deposits varies widely—from flat lying to vertical, in thin-to-thick extensive sheets to small discontinuous lenses, and at shallow to ultra-deep depths. Deposits such as the salt domes of the US Gulf Coast and the Middle East, or the highly contorted strata within the US interior, or the Zechstein of Europe require special solution mining techniques.

For discussion of the basic rock mechanics properties of evaporites, see Chapter 10.7.

15.3.1.3 Mineral Exploration/Evaluation

The exploration and evaluation of evaporite mineral deposits have been significantly improved through modified geophysical surveying equipment, procedures, and diagnostic capabilities.

Ground surface and down-hole seismic surveys may be combined to yield more useful stratigraphic information:

1. Both within and *below* an exploration borehole.
2. Lateral correlations beyond a borehole or between boreholes.
3. Vertical and horizontal boreholes within evaporite beds.

Down-hole geophysical well-logging equipment and procedures have improved the definition of thin (0.5 ft or 0.2 m) evaporite beds. In particular, gamma ray spectrometry has increased the potential for the identification and quantification of potash mineralization within vertical and inclined boreholes.

Geophysical well-log combinations readily provide geomechanical information relating to the in situ stress field: direction and magnitude of principal stress, improved hydraulic fracture diagnostics, probable direction of maximum cavern enlargement, etc.

Cavern sonar surveys, now offered by several companies, include both horizontal and inclined distance measurements from the borehole to cavern roof, walls, and floors. Digital recording and processing of all signals is now commonplace. At least one company (Prakla-Seismos AG) has developed the ability to survey within multi-fluid cavern environments (water, oil, and gas), and through one string of casing (Fig. 15.3.1.1).

Newly developed cavern radar surveys are also offered by Prakla Seismos. In 1988, this technique was introduced into the United States during the monitoring of the CAES (compressed air energy storage) cavern being developed by the Fenix & Scisson division of PB-KBB Inc. within the McIntosh Salt Dome near Mobile, AL.

15.3.1.4. Drilling Technologies

New drilling technologies have continued to evolve at a rapid pace. Their main goal is to improve the economics of drilling—to decrease costs per foot or meter—or to permit drilling under conditions that were formally prohibitive. Much of the industry's success is due to down-hole innovations such as measurement-while-drilling systems, steerable down-hole motors, and improved drillbits. Top-drive drilling systems are playing an important role in reducing hole problems and improving safety. Slanthole drilling is useful in the exploration of steeply dipping evaporite beds; horizontal and sub-horizontal directional drilling techniques are beginning to offer practical solutions to multi-cavern interconnection and operation.

DIRECTIONAL DRILLING TECHNOLOGY. Directional drilling technology presently provides controls to reach thousands of feet (meters) horizontally, and to build angles ranging

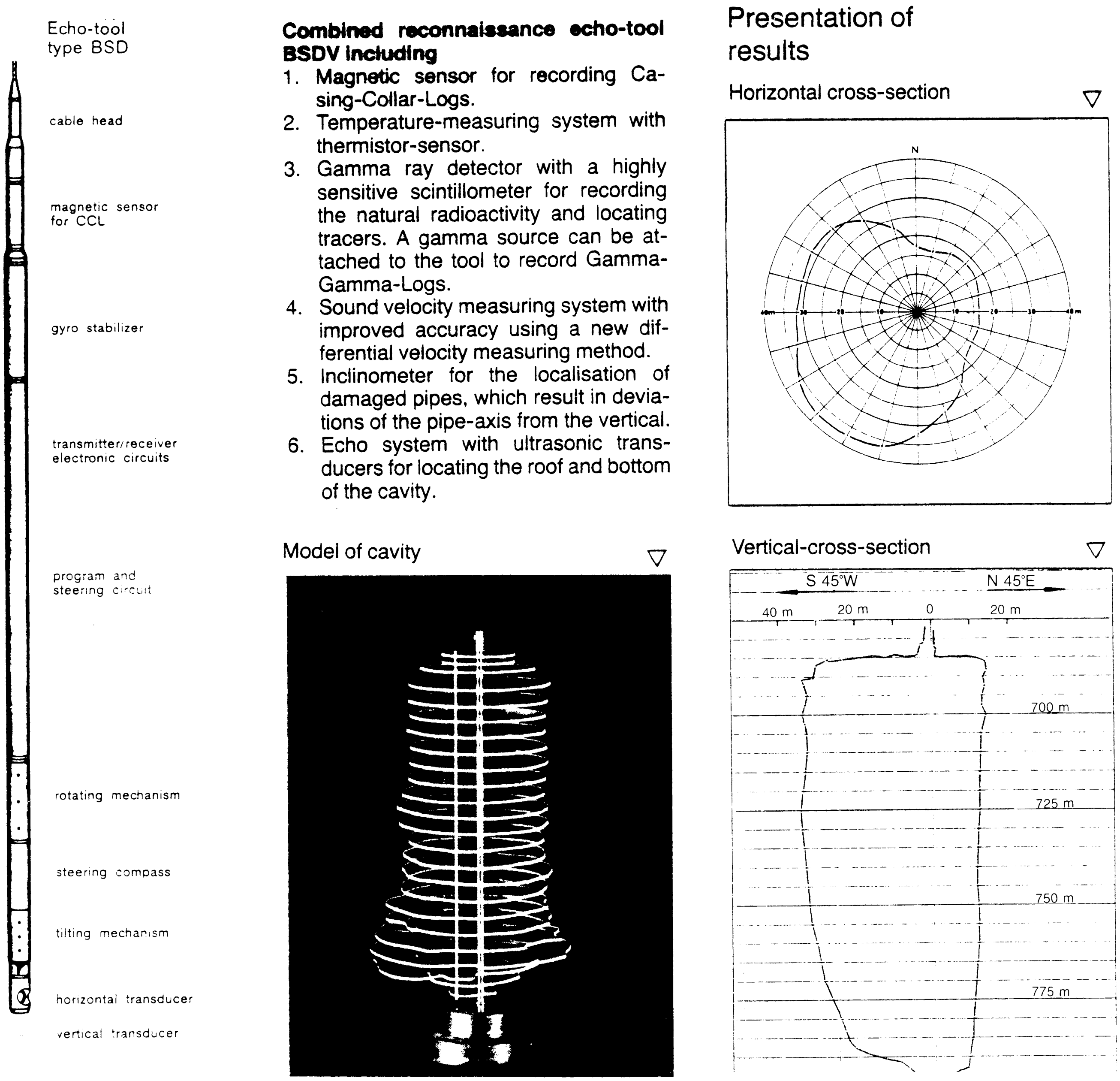


Fig. 15.3.1.1. Prakla-Seismos type BSD sonar surveying presentation (*courtesy:* PB-KBB/Prakla Seismos Joint Venture). Conversion factor: 1 ft = 0.3048 m.

from 3 to 35°/100 ft (30 m). Most drilling systems include a slow-speed, high-torque variable-bend, positive-displacement, down-hole motor, with special stabilizing features. Bottom-hole assemblies may contain a tri-cone or polycrystalline bit; a high-torque, slow-speed motor; a measurement-while-drilling (MWD) tool; and key positioning stabilizers of various designs and gages. Examples of directional drilling profiles are shown in Fig. 15.3.1.2.

Special problems related to high-angle drilling programs include (1) cuttings removal from the downside of the drillhole, (2) annulus in the cement sheath on the top side of the casing, (3) specialized equipment and procedures needed for geophysical well logging, and (4) difficulty in maintaining directional control within evaporite mudstones. Despite these problems, both the International Salt Co. (Akron, OH) and the Morton Salt Co. (Rittman, OH) have undertaken horizontal drilling in order to achieve an early connection between salt wells where hydraulic fracturing was unsuccessful and roof-pad interconnections by leaching were time-consuming. Additional installations are being designed for salt wells in Ontario, Canada, and nahcolite wells in Colorado.

Extended-reach horizontal wells are drilled as directional wells before becoming horizontal.

Drainhole drilling techniques usually involve entering an existing vertical well, cutting a section through the casing, and drilling a horizontal hole. The radius of the "drainhole" is typically only 30 to 40 ft (9 to 12 m), and the horizontal section of the borehole frequently extends only 300 to 400 ft (90 to 120 m).

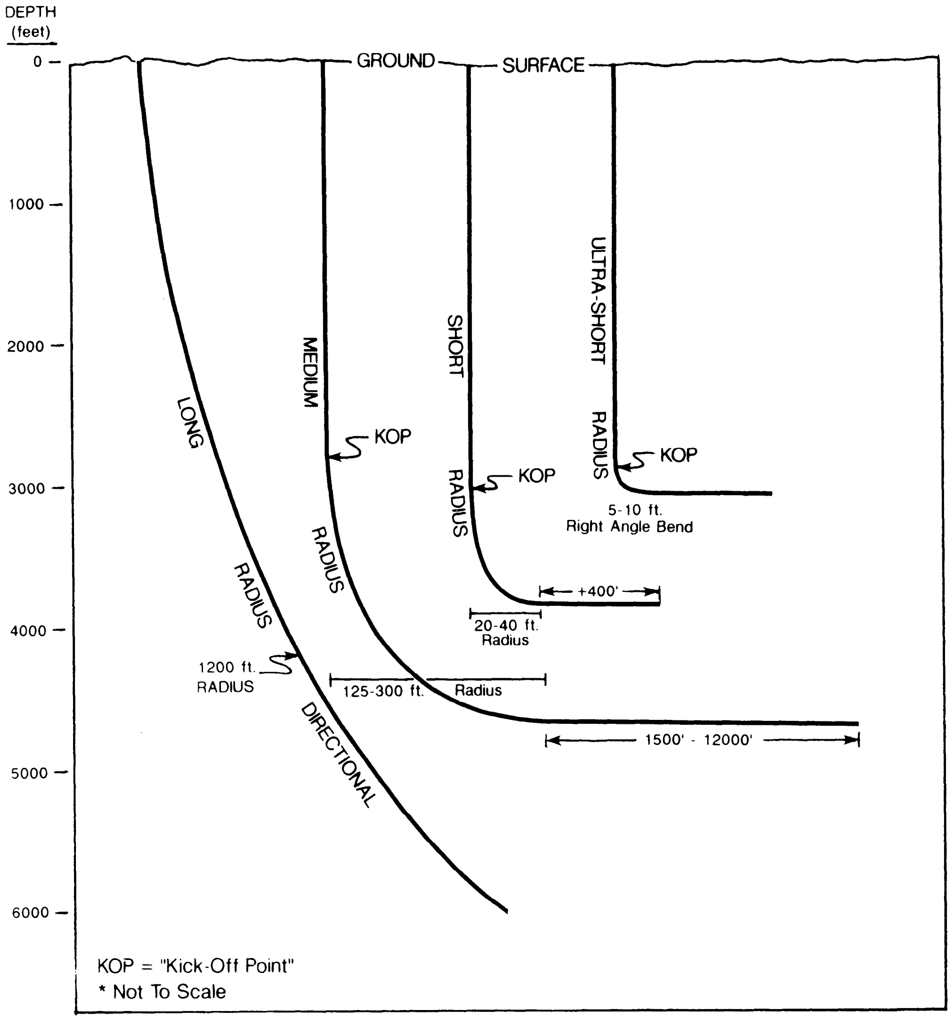


Fig. 15.3.1.2. Examples of directional drilling well profiles (*courtesy:* PB-KBB, Inc.). Conversion factor: 1 ft = 0.3048 m.

There is presently no good technique for logging or casing the horizontal section, and future applications to solution mining of evaporites may be limited to interconnections with existing brine caverns. The drainhole drilling technique is illustrated in Fig. 15.3.1.2.

Slanthole drilling requires a drilling rig with a mast that can be slanted up to 45°. Pipe drag during lifting and lowering operations is considerable. Lifting capacities required to pull off-bottom with the drill string or liner and push-down capacity to obtain force on the drill bit or push a liner to bottom can be significant. Such "push-down" capability is required because only 70% of the weight of the drill collars and heavyweight drill pipe is applied down the drill string at 45°. This effective weight rapidly declines as the wellbore angle from vertical increases.

Slanthole drilling techniques can be particularly useful during exploratory drilling of steeply dipping evaporite strata, such as potash or magnesium chloride beds within a salt diapir (Fig. 15.3.1.3).

Slanthole drilling has terminated in a horizontal drilling program within the shallow Athabasca oil sands. Modifications of this technique may someday be applied to selective solution mining or underground storage problems.

MEASUREMENT-WHILE-DRILLING (MWD). Measurement-while-drilling is another rapidly developing technology that provides additional geologic control and better directional control during the drilling process. Anadrill, a division of Schlumberger, provides equipment that can send down-hole measurements to the surface at rates exceeding 3 bits/sec. The service provides gamma ray and resistivity measurements every foot (0.3 meter) of the hole, as well as gravity tool face updates every 6 sec and a magnetic tool update every 4 sec. Down-hole information also includes down-hole weight on bit (DWOB) and down-hole torque (DTOR) updates. These data, used with a steerable drilling system, can allow continued drilling mechanics and formation logging without interruption of the directional drilling program.

15.3.1.5 Drilling Fluids

Drilling fluid technology within evaporites has advanced rapidly along two fronts, composition and density.

Water-based, salt (NaCl)-saturated drilling muds are still commonly used, but water-soluble calcium, potassium, and magnesium compounds find increasing use. Their appropriate addition reduces hole enlargement through soluble mineral beds (halite, trona, nahcolite, potash, etc.) and increases the stability of the borehole through intervening shale, mudstone, and marlstone layers. Such hole conditioning provides improved drilling conditions and a much better down-hole environment for geo-

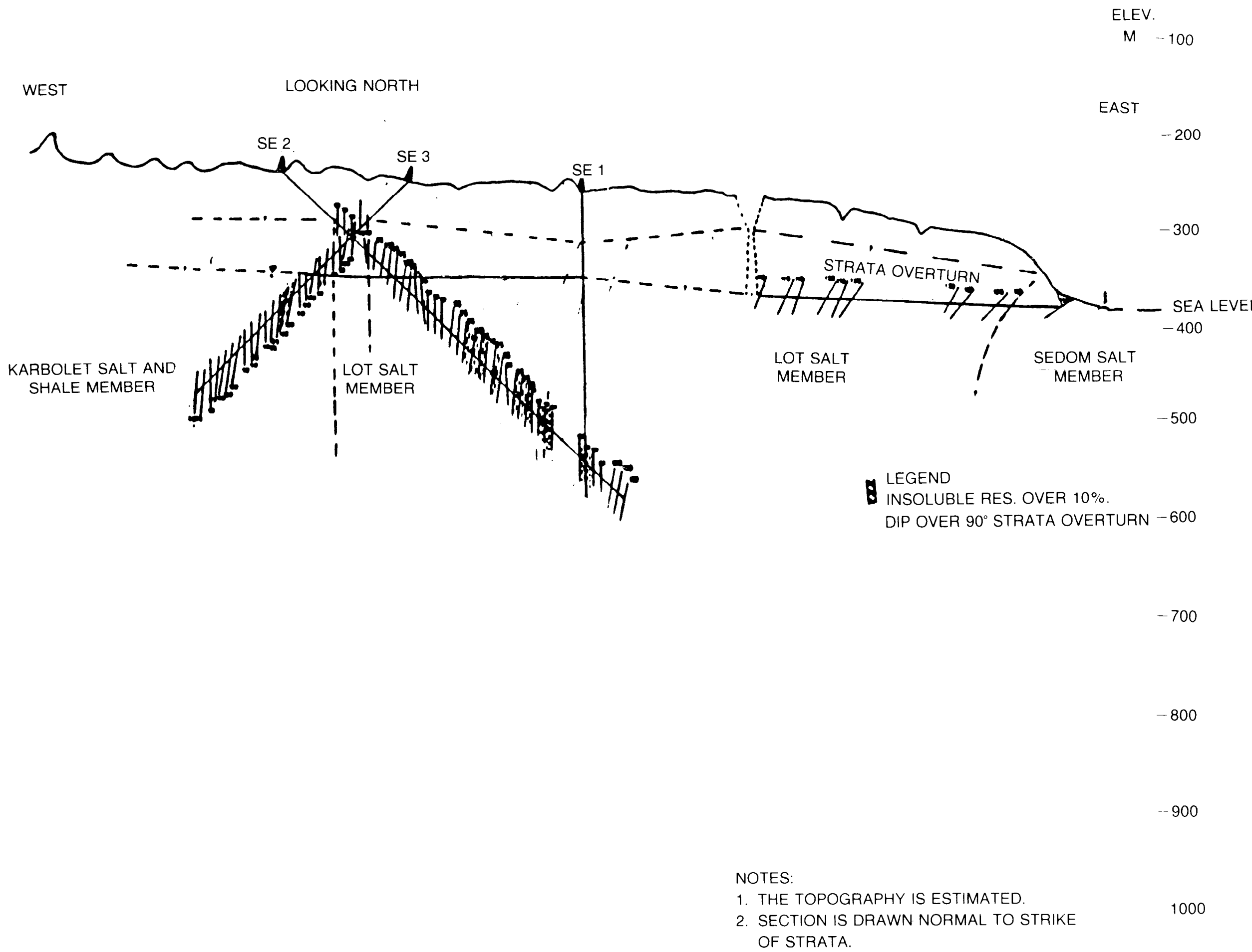


Fig. 15.3.1.3. Slanthole drilling at Mt. Sedom, Israel, during 1978 exploration program (*courtesy:* Petroleum Services Ltd.). Conversion factor: 1 ft = 0.3048 m.

physical well logging, running and cementing casing, and subsequent well/cavern development.

It is now well established that evaporite strata within certain regions of the United States (and within other countries) presently exist under conditions of high in situ lateral stress. In such instances, the use of high-density drilling fluids may be essential to prevent or minimize hole closure prior to running casing. Local and regional examples of high lateral stress areas can be found within the Appalachian and Michigan evaporites, and within the Williston, Denver, and Green River Basins.

15.3.1.6 Tubulars

Tubulars (casing, tubing, and liners) are readily available in an increased range of sizes (diameter) and strengths (tensile, compressive, and burst), with a variety of threads and from numerous foreign sources.

Small-diameter casings (B-Q, N-Q, and H-Q) are available for the continuous casing of slanted wireline coreholes. Medium-diameter articulated casings (4 to 8 in., or 100 to 200 mm) are now available for highly deviated and horizontal walls. Large-diameter casings (24 to 60 in., or 610 to 1524 mm, and larger) are available for special well completions, such as for CAES caverns or crude oil storage requiring high delivery schedules. Increased grades of seamless casing are now available with higher minimum internal yield, collapse, and joint strength properties. Such grades include US95, P-110, Q-125, and V-150. The minimum yield strength factors for casing, in psi (MPa), are:

Grade	Yield Strength, in psi (MPa)
H-40	40,000 (276)
J-55 and K-55	55,000 (380)
C-75 and E	75,000 (518)
L-80 and N-80	80,000 (552)
C-90	90,000 (621)
C-95	95,000 (656)
P-105	105,000 (725)
P-110	110,000 (759)
Q-125	125,000 (863)
S-135	135,000 (932)
V-150	150,000 (1035)

The round thread and buttress thread casings popular within the United States now include a variety of other casing threads developed for special purposes or produced by foreign manufacturers. Asian and European suppliers are rapidly expanding their share of the US and worldwide market for tubulars, in part because of the high cost of producing steel within the United States.

15.3.1.7 Cements and Cementing

The limits in physical and chemical composition and properties of cementitious materials, as well as their range of potential applications, have expanded rapidly during the past decade.

Considerable research effort has been focused upon increasing the long-term stability of portland cements. Reduction of porosity and permeability and delay of natural decomposition of cement materials have been achieved through fine grinding (to 1000 mesh) and through more extensive use of API Class G and H (high-temperature, sulfate-resistant) cement slurries. Special control of cement slurry properties emplaced within an evaporite environment is commonly achieved through the use of additives—particularly salt (to reduce dissolution of halite wallrock), bentonite (to induce expansion/reduce shrinkage of slurry during setting), and calcium chloride (to decrease slurry setting time and increase early cement strength).

A variety of unconventional cements and/or cementitious materials designed to solve special down-hole problems is now readily available. Such materials include:

1. *Epoxies.* Epoxy resins are most frequently used where very high bonding strength is required through short intervals, or when portland cement slurries might react rapidly with certain magnesium or calcium-containing evaporite wallrocks or strong brines. Usage is restricted to very special problems because of very high material and installation costs.

2. *Lightweight Cements.* Special lightweight cement slurries have frequently been used to reduce induced fracturing within incompetent formations having low parting pressures, or flow into such formations that are naturally highly fractured. Low-density additives such as gilsonite, expanded perlite, and pozzolans have long been favored.

New low-density cement slurries are now available through the addition of hollow organic and inorganic spheres and microspheres. High-strength cements can be developed with densities under 11 and 24-hr compressive strength of more than 1700 psi (11.7 MPa), cured at 300°F (149°C).

3. *Foamed Cements.* Ultra-light water cement slurries can be formed through the foaming action of gaseous additives. Air, carbon dioxide, and nitrogen have been most commonly used, depending largely upon cost and availability.

Foamed cements may be useful where formation fracture gradients are abnormally low. The material is also a good insulator and has potential for both arctic and geothermal operations.

Foamed cement slurries can be made with densities as low as 4 lb/gal (500 g/L). Although having little strength, the slurry will float on brine within a salt cavern, and can thus be used as a “bridging material” behind a cemented casing shoe that has been “washed out.”

4. *Asphalt Resin Cements.* Solid asphaltic resins occur naturally within various locations and types of mineral deposits within North America. The Athabasca tar sands (Canada) and the Brea tar pits (California) are well known examples.

Black asphaltic resins also occur as veins within country rock in Utah. One material, called gilsonite, is mined and processed by the American Gilsonite Co. Much of this finely powdered material is used as an additive to cement slurries used behind well casings, providing a plastic pore-space filler to an otherwise brittle cemented annulus. This usage has potential application within easily deformable evaporite formations. Under special conditions, where plasticity of the cement sheath is more important than its strength, the use of asphalt as the primary cementing ingredient may be economically justified within evaporite wells.

15.3.1.8 Geophysical Well Logging

Geophysical well-logging measurements are functions of the chemico-physical properties of the rock matrix, the fluid(s) within the formation, conditions within the borehole, the volume of rock investigated by the logging tool, the tool’s vertical resolution, and the design characteristics of each individual tool. Measurements are also affected by idiosyncrasies related to the logging cable, logging truck equipment and instrumentation, and operating personnel. Thus the response *measured* with a logging tool must always be considered an *apparent* rather than a *real* geotechnical property value.

The conversion of apparent-to-real properties frequently requires the expertise of a professional well-logging analyst. Inasmuch as fewer than 1/2% of all wells drilled are for the purpose of the evaluation or exploitation of evaporite deposits, the geotechnology of such log analysis is highly dependent upon adaptation from petroleum and water-well-logging technologies, and experienced evaporite well-logging analysts are a small select group.

Logging constants for various evaporite minerals and rocks are shown in Table 15.3.1.2. Log responses within a potash-bearing sequence are provided in Figs. 15.3.1.4 and 15.3.1.5. Trona beds within the Green River Formation in Wyoming can be identified by a gamma ray/sonic/neutron log suite while nahcolite beds within the Piceance Creek Formation of Colorado have similar log responses.

Geophysical well log “cross plots” are frequently used by log analysts to assist in the identification and evaluation of evaporite mineral deposits. Examples of sonic/neutron, density/neutron, and sonic/density crossplots (charts) are illustrated in Fig. 15.3.1.6.

The proliferation of small, local, independent logging companies was sharply curtailed during the recession of the mid-1980s. Strong competition for the dwindling down-hole dollar led to a consolidation of service companies, abandonment of antiquated equipment, and significant reduction in work force.

Several of the major well service companies used this period of temporary retrenchment to expand their research and development effort, mainly along four lines:

1. Improvements of existing logging tools, particularly their reliability, sensitivity, selectivity, and quantitative usefulness.

2. Synergistic integration of different types of well-log data, together with information derived from other down-hole services such as coring and core analysis, and down-hole seismic surveying.

3. Consolidation of service companies to provide a broad range of integrated down-hole services.

4. Improved global communication systems and computer networks to permit rapid evaluation and dissemination of down-hole data.

Three rapidly developing geotechnologies are of special interest to the evaporite professional:

1. Natural gamma ray spectral logging devices measure natural gamma rays through “energy windows” that permit determination of K (potassium) concentrations. The logs can be run in both open and cased wellbores and in useful combinations with various neutron devices. Following calibration against chemical analyses of cores, the log combinations can provide a rapid, less costly method of potash exploration.

2. Knowledge of the elastic properties and in situ stresses of the strata enveloping an existing or planned salt cavern is becoming increasingly important for both economic and environmental reasons. The elastic constants—Young’s modulus, bulk modulus, shear modulus, and Poisson’s ratio—are related to the density and elastic wave velocities of the rocks. The advent of special-

Table 15.3.1.2. Logging Constants

Symbol	Name	Composition	Sp. Gr.	Log P_B (g/cm^3)	Average Δ†	(GNT) Neutron ϕN	Deflection γ-ray (API) (d = 8 in.)	% K_2O
		Evaporite Minerals (ϕ = 0)						
	Anhydrite	$CaSO_4$	2.960	2.977	50	0	0	0
	Carnallite	$KCl \cdot MgCl_2 \cdot 6H_2O$	1.61	1.57	78	65	200	17
	Gypsum	$CaSO_4 \cdot 2H_2O$	2.320	2.351	52.5	49	0	0
	Halite	NaCl	2.165	2.032	67	0	0	0
	Kainite	$MgSO_4 \cdot KCl \cdot 3H_2O$	2.13	2.12	—	45	225	18.9
	Langbeinite	$K_2SO_4 \cdot 2MgSO_4$	2.83	2.82	52	0	275	22.6
	Polyhalite	$K_2SO_4 \cdot MgSO_4 \cdot 2CaSO_4 \cdot 2H_2O$	2.78	2.79	57.5	15	180	15.5
	Sulphur	S	2.07	2.03	122	−3	0	0
	Sylvite	KCl	1.984	1.863	74	0	~500	63
	Trona	$NaCO_3 \cdot NaHCO_3 \cdot 2H_2O$	2.12	2.10	65	40	0	0
		Sedimentary Minerals (ϕ = 0)						
	Calcite	$CaCO_3$	2.710	2.710	47.5	0	0	0
	Dolomite	$CaMg(CO_3)_2$	2.870	2.876	43.5	4	0	0
	Quartz	SiO_2	2.654	2.648	51.5	−4	0	0
		Sedimentary Formations						
	Limestone	(e.g., when ϕ = 10)	2.540	2.540	62	10	5–10	0
	Dolomite	(e.g., when ϕ = 10)	2.680	2.683	58	13.5	10–20	0
	Sandstone	(e.g., when ϕ = 10)	2.489	2.485	65.3	3	10–30	0
	Shale			2.2–2.75	70–150	25–60	80–140	2–10

Source: Northern Ohio Geological Society, 1966.
Conversion factor: 1 lb/ft^3 = 0.016 g/cm^3.

ized digital data handling permits the determination of both the compressional wave and shear wave velocities needed to define all four elastic constants.

3. Vertical seismic profiling (VSP) has attracted considerable attention within the geophysical community during the past decade. Its major potential uses by evaporite professionals include (1) providing geologic structural imaging near the borehole, (2) providing time-to-depth conversions for use with conventional surface seismics, and (3) predicting formation tops ahead of (below) the drillbit, or below hole TD. Examples of VSP are presented in Fig. 15.3.1.7.

15.3.1.9 Solution Mining Systems

Most single-well and multiple-well solution mining systems have been described in considerable detail by Jacoby (1973). Only a few modifications or innovations need be reported at this time.

General characteristics now common to most well systems are

1. Solution mining is initiated near the bottom of a selected water-soluble evaporite sequence. Subsequent vertical and lateral cavern development then proceeds in stages, governed by local geologic conditions, engineering requirements, and safety/environmental considerations.

2. Cavern dimensions are time-dependent, strongly influenced by the physicochemical properties of the host strata and the characteristics of cavern fluids and solids. Cavern heights have ranged between 20 and 2000 ft (6 and 600 m), but generally are 50 to 300 ft (15 to 90 m). Cavern diameters have ranged between 100 and 1000 ft (30 to 300 m), but generally are 150 to 300 ft (45 to 90 m). Caverns have been constructed with floors as shallow as 300 ft (90 m), or as deep as 9000 ft (2700 m), but their general depth range is 1000 to 4000 ft (300 to 1200 m).

3. A gaseous diesel (air, nitrogen, carbon dioxide, natural gas) or liquid (crude oil, LPG) roof pad, or "blanket" is usually employed to assist in controlling the shape and dimensions of each cavern.

4. Well/cavern underground storage systems are usually subjected to higher design, construction, and operational criteria than their counterparts used solely for brine production. Storage systems pose greater risks to the subsurface and surface environment and to the safety of living beings. For both environmental and economic reasons, however, there is a rapidly growing trend toward strengthening the criteria governing brine wells, not only to protect the environment but also so that they can be more readily converted to storage/disposal facilities, if desired.

SINGLE-WELL OPERATIONS. Historically, top-injection (Tully method) techniques have resulted in low investment costs

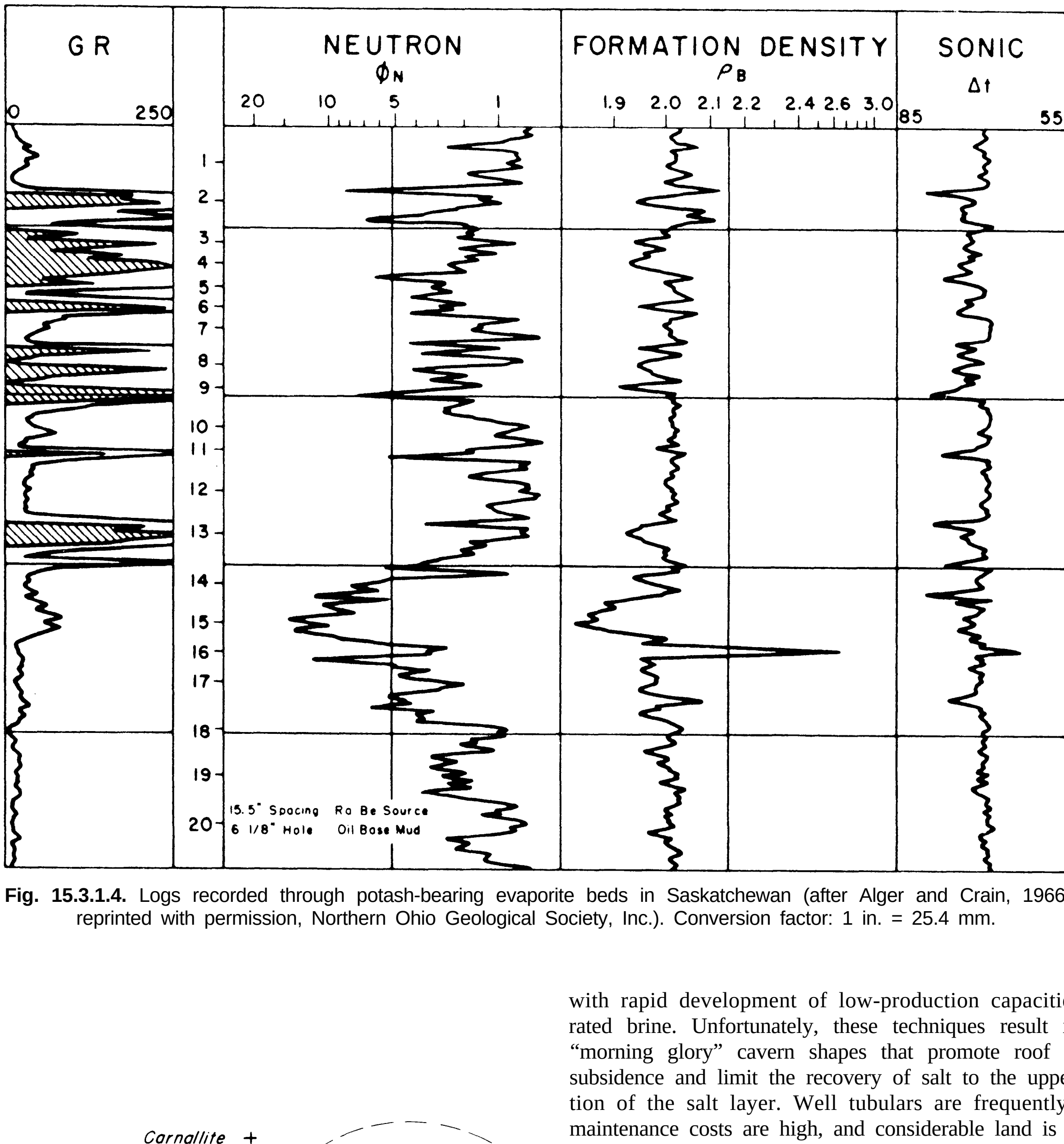


Fig. 15.3.1.4. Logs recorded through potash-bearing evaporite beds in Saskatchewan (after Alger and Crain, 1966, reprinted with permission, Northern Ohio Geological Society, Inc.). Conversion factor: 1 in. = 25.4 mm.

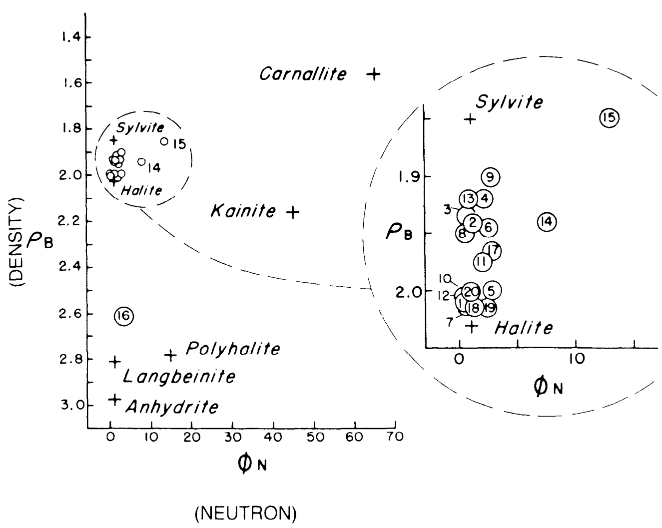


Fig. 15.3.1.5. Cross plot of density vs. neutron (derived from Fig. 15.3.1.4) identifies evaporite minerals (after Alger and Crain, 1966, reprinted with permission, Northern Ohio Geological Society, Inc.).

with rapid development of low-production capacities of saturated brine. Unfortunately, these techniques result in unstable "morning glory" cavern shapes that promote roof caving and subsidence and limit the recovery of salt to the uppermost portion of the salt layer. Well tubulars are frequently damaged, maintenance costs are high, and considerable land is required to support a long-term solution mining operations. This method, named for the earliest Allied Chemical operations at Tully, NY, is in general disfavor today.

The Allied Chemical Co. introduced the Trump method in the early 1930s whereby an undercut was achieved near the base of a selected salt layer. The original intent of this method was to develop a large-diameter undercut, a few feet (meters) in thickness, using a gaseous or liquid "blanket" to inhibit upward dissolution of the salt roof. More recent efforts have included coalescence of adjoining single caverns near the base of the salt, followed by two-well or multiple-well operation. This method frequently entails a 12 to 24-month cavern development period during which arrangements must be made for the disposal or usage of unsaturated brine. The method is most applicable to bedded salt deposits having low geologic dips.

The Detroit method, a bottom-tubing injection system, was developed to overcome many of the disadvantages of the Tully method. Cavern roof stability was improved, but salt brine (obtained from the top of the cavity) was usually below saturation, even at extremely low (50 gpm or 3 L/s) production rates.

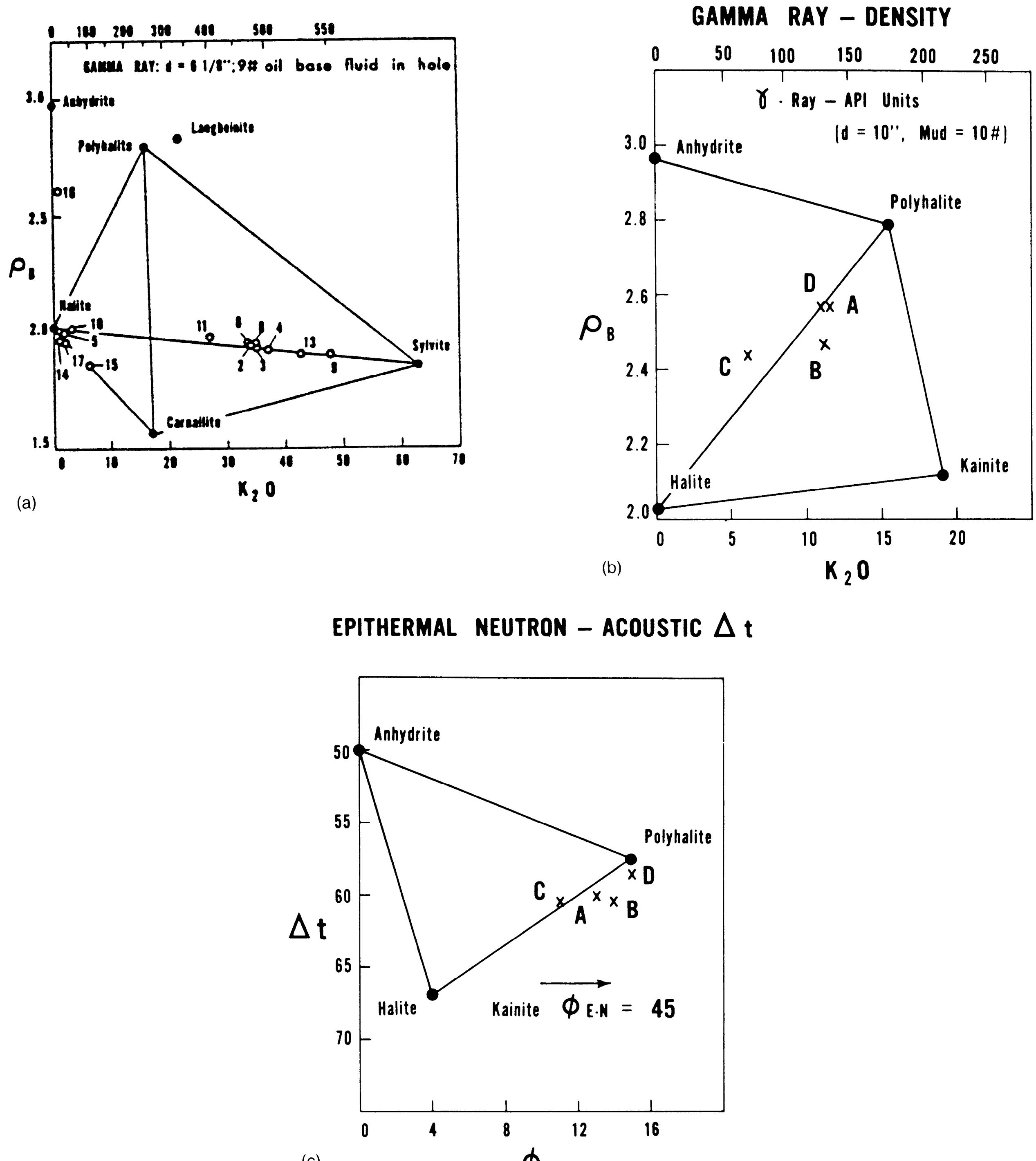


Fig. 15.3.1.6. Examples of various well-log cross plots derived from logs of one well through an evaporite section. (a) Plot of gamma ray vs. bulk density defines potash minerals; (b) plot of gamma ray vs. bulk density data from logs in (a); (c) plot of sonic neutron data from logs in (a). (Reprinted with permission, Northern Ohio Geological Society, Inc.).

Many single-well solution mining operations today are based upon a modified annular injection method wherein both the tubing string and the hanging casing string are positioned near the base of the lowermost salt layer to be dissolved. An undercut of 150 to 400 ft (45 to 122 m) diameter is achieved using an insoluble rock layer or a gaseous or liquid blanket to inhibit upward dissolution of the undercut. Saturated brine production rates in excess of 200 gpm (13 L/s) have been achieved from such wells after the protective roof pad has been removed. However, the effectiveness of the method is frequently dependent upon geologic factors such as formation dip, frequency and attitudes of discontinuities, and the in situ stress field. A schematic of this variable-point injection system is presented in Fig. 15.3.1.8.

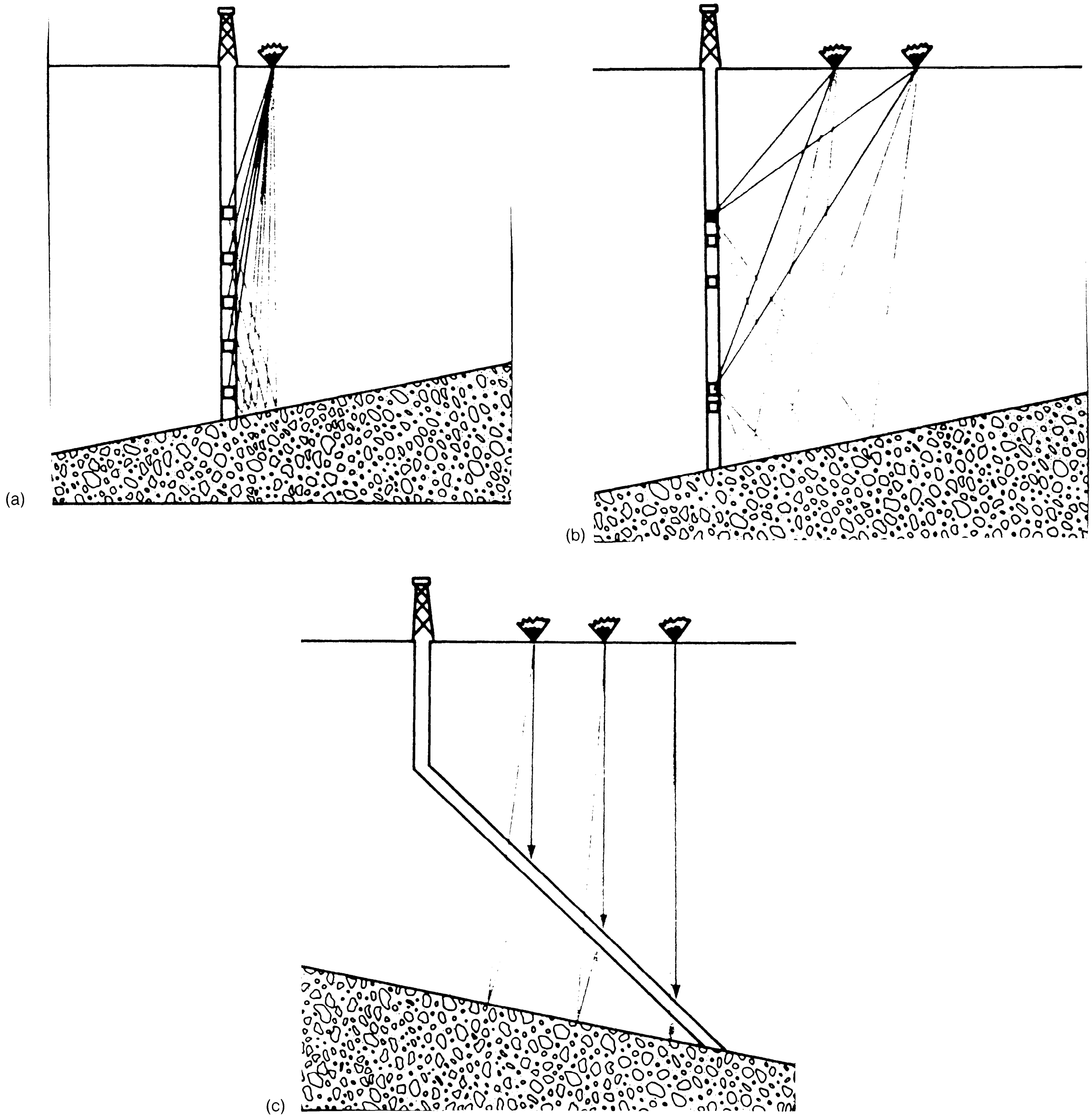


Fig. 15.3.1.7. Examples of vertical seismic profiles (*courtesy:* Schlumberger Well Services).

Deviated drilling techniques have been successfully used (Fig. 15.3.1.9) to avoid "rigging-up" within restricted or inaccessible land or water areas. Horizontal drilling is a technique used for thin-bedded evaporite strata such as nahcolite, trona, potash, and halite. The NATEC group (Houston, TX) expects to initiate a horizontal drilling program within their Piceance Basin (Colorado) nahcolite deposits in the early 1990s. In general, however, there are numerous technical and economic problems to resolve before horizontal drilling within evaporite strata can be regarded as other than experimental.

Many salt caverns have been developed through the use of multiple wells within a single cavern. This approach was taken during the construction of many of the SPR (Strategic Petroleum Reserve) caverns in the Gulf Coast of the United States in order to decrease the time required for full cavern construction and to provide for more rapid cavern fill/empty cycles.

At the Veedam brine field in the Netherlands, Billiton Refractories B.V. has constructed individual brine wells containing *separate* tubing strings, rather than the *concentric* tubing construction usually in practice. These wells are primarily for the

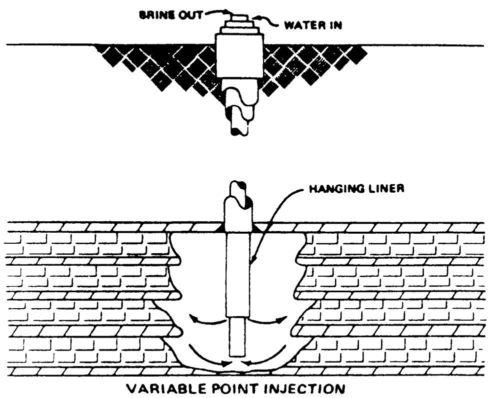


Fig. 15.3.1.8. Schematic of variable-point injection system (after Querio, *courtesy:* Salgema Mineracao, LTDA).

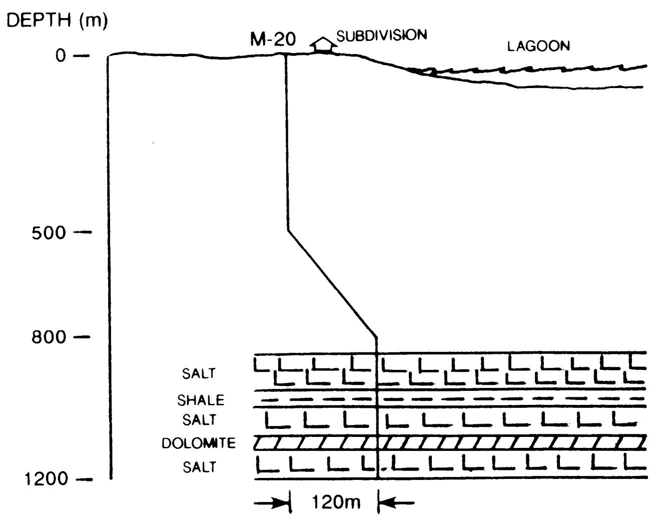


Fig. 15.3.1.9. Directional drilling: Salgema Salt Well No. M-20, Maceio, Brazil (*courtesy:* PB-KBB Inc.). Conversion factor: 1 ft = 0.3048 in.

production of highly plastic magnesium salts. An illustration of one of their well designs is presented in Fig. 15.1.3.10.

MULTIPLE WELLS. There have been few recent innovations in the technology of single-well interconnection to provide multiple-well operation.

The updip coalescence of single wells, particularly when assisted by some undercut technique near the base of a salt bed, is commonly used. Well interconnection by hydraulic fracturing is widely regarded as a high-risk technique, requiring considerable geotechnical acumen to be successful. However, rock mechanics studies in nonevaporite strata, promulgated by the petroleum industry, are beginning to be translated into new considerations and techniques for the "controlled" development of both "bedding-plane" and "vertical" fractures within evaporite beds.

Single-well interconnection by horizontal drilling to provide a rapid technique for multiple-well operation is still in its infancy. Vertical and horizontal drilling control within thin-bedded, undulating evaporite strata poses presently unresolved problems.

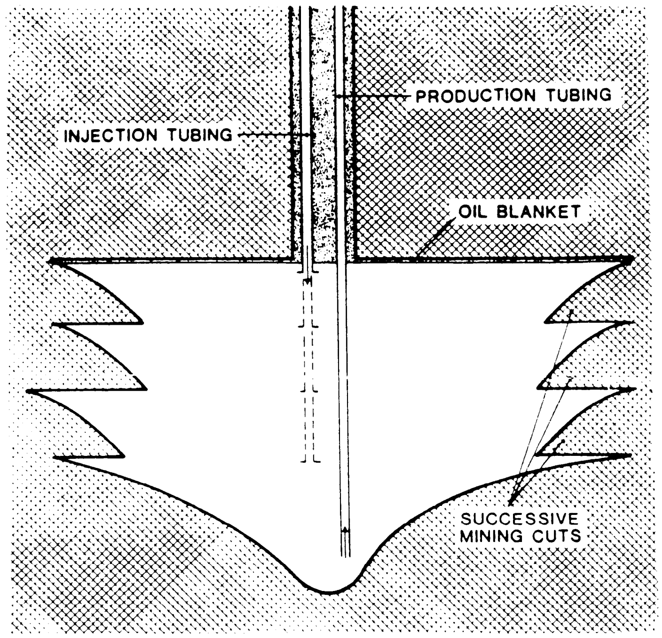


Fig. 15.3.1.10. Nozo strip mining concept (*courtesy:* Billiton Refractories B.V.).

Advances are also needed to resolve drilling fluid, cuttings transport, and cementing problems, as well as improvement in the use of well screens and articulated tubulars.

15.3.1.10 Underground Storage

Evaporite strata existing at depths below 1000 ft (300 m) frequently have very low porosities and permeabilities. These characteristics often permits their successful use for the storage of a variety of solids, liquids, and gases, as indicated in Table 15.3.1.3.

Where locations are favorable, most such storage has been developed by solution-mining techniques because of

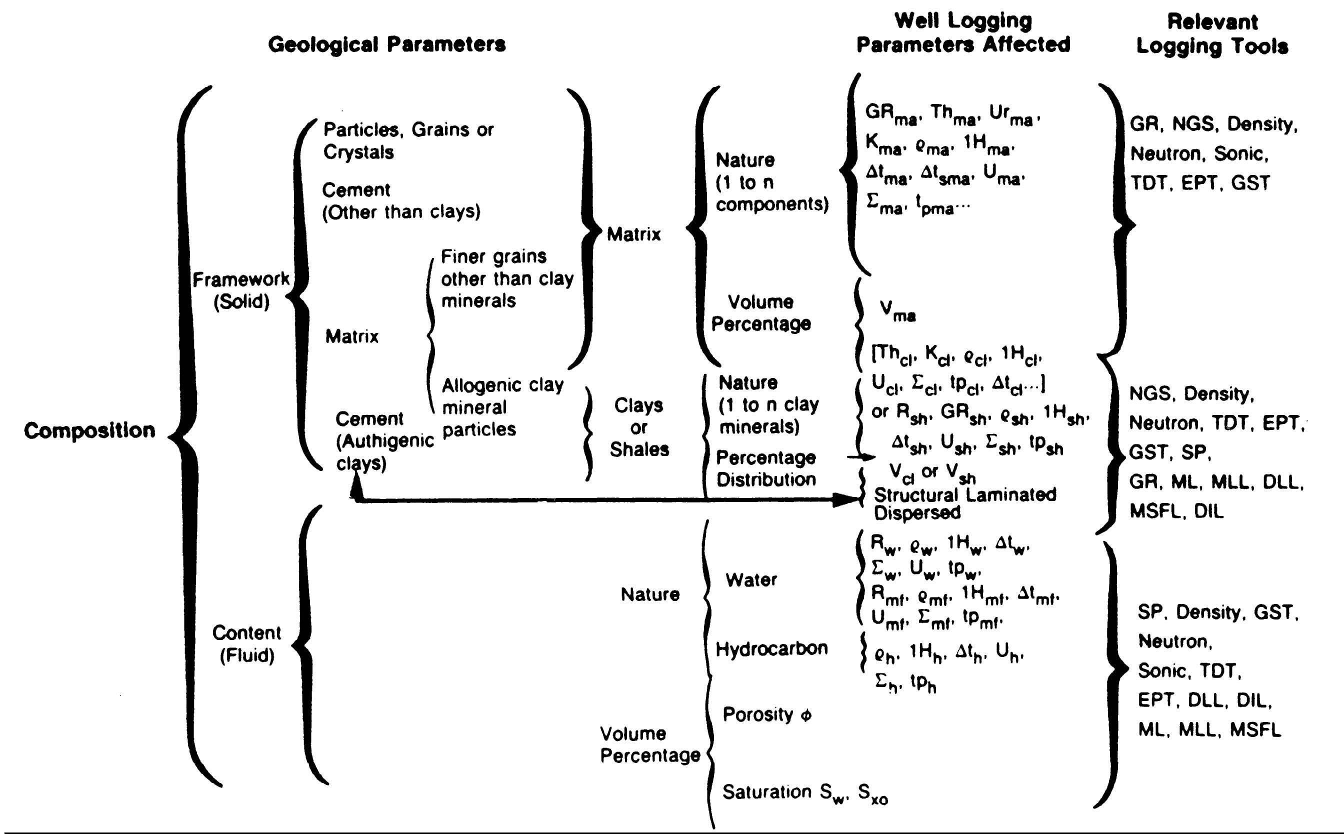
1. Simplicity of well/cavern design.
2. Flexibility in storage volume design.
3. Lower cost of construction.
4. Rapid installation.
5. Decreased operating and maintenance costs,
6. Increased safety to personnel, ground surface installations, and the environment.

COMMODITIES. National Petroleum Reserves: The largest underground storage system for crude oil was developed by the US Department of Energy (DOE) through its Strategic Petroleum Reserve (SPR). At the end of 1989, approximately 690 million bbl (109 Mm^3) of crude oil were being stored in one salt dome mine (Weeks Island, LA) and in salt dome cavern installations (Bayou Choctaw, LA; Big Hill, TX; Bryan Mound, TX; and West Hackberry, LA). Most of the SPR storage caverns were constructed within depths of 2000 to 4000 ft (600 to 1200 m), and have a storage capacity of about 10 million bbl (1.6 Mm^3) each of crude oil. A typical storage cavern is illustrated within Fig. 15.3.1.11.

National crude oil underground storage repositories within solution-mined caverns are also located in West Germany, East Germany, France, the USSR, Norway, Sweden, and Mexico.

LOOP/LOCAP Crude Oil Facilities: The LOOP (Louisiana Offshore Oil Port) is the only offshore deep-water port in the

Table 15.3.1.3. Relationship Between Risk Composition and Geophysical Well Logging Parameters



Courtesy: Schlumberger Well Services.

United States (descriptions of the LOOP/LOCAP systems have been summarized from in-house reports issued by PB-KBB, Inc.). It can handle tankers up to 700,000 DWT (635 kt). Crude oil is offloaded at the offshore marine terminal and is piped through an onshore booster station (Fourchon Booster Station) into salt dome storage caverns at the Clovelly terminal. The caverns serve as surge storage until the oil can be taken by pipeline. From Clovelly, crude flows through the LOCAP pipeline (jointly owned by several major oil companies) or to various refineries.

LOOP facilities include all equipment and piping necessary to receive crude and store it in the salt caverns. The LOCAP facilities include all equipment and piping necessary to transport crude from Clovelly to the LOCAP terminal and to the Capline terminal, both at St. James. A pipeline connection between the LOCAP facilities and the DOE facilities at St. James terminal provides capability for crude transfer between the two.

Fig. 15.3.1.12 depicts the location of the Louisiana Offshore Oil Port and connecting carriers. A preliminary design of a Clovelly terminal “surge” storage cavern is shown in Fig. 15.3.1.13.

Conceptual designs have been prepared by PB-KBB Inc. for the future enhancement of the SPR distribution system through flow reversal of existing LOCAP and LOOP oil terminalling transportation systems. Additional descriptions of these existing and proposed facilities are available through application to the appropriate DOE office.

LIQUEFIED PETROLEUM GAS (LPG)—There were more than 1200 solution-mined underground storage caverns in use within the United States in 1989. The liquid products stored include ethane, propane, E-P mix, normal butane, isobutane, natural gasoline, raw mix, ethylene, propylene, etc.

The Gas Processors Association, Tulsa, OK, conducts a biennial survey of such facilities. Cavern locations within the United States and Canada are shown in Fig. 15.3.1.14, and a tabulation of stored LPG substances is presented in Table 15.2.1.4. The rapid development of LPG storage within the United States from 1950 through 1987, ranging from 10 million to more than 433 million bbl (1.6 to 72 Mm^3), is illustrated in Fig. 15.3.1.15.

LIQUEFIED NATURAL GAS (LNG)—Much of the natural gas that is produced in association with crude oil has, in the past, been disposed of by flaring it in the field. However, the world’s ever-increasing energy demands have caused the prudent production, storage, transportation, and use of natural gas to become economically feasible. Since the density of natural gas increases almost 600 times by liquification to LNG (at –310°F or –190°C), overseas transportation within specialized tankers has become common. This gas is normally about 90% methane.

The benefits of underground storage in salt caverns are threefold:

1. The high density of the gas allows the use of considerably smaller storage caverns.

2. Underground storage is a safer process than aboveground terminals because, if LNG spills from its storage containers, it rapidly vaporizes (flashes) and becomes a highly flammable and explosive mixture in air.

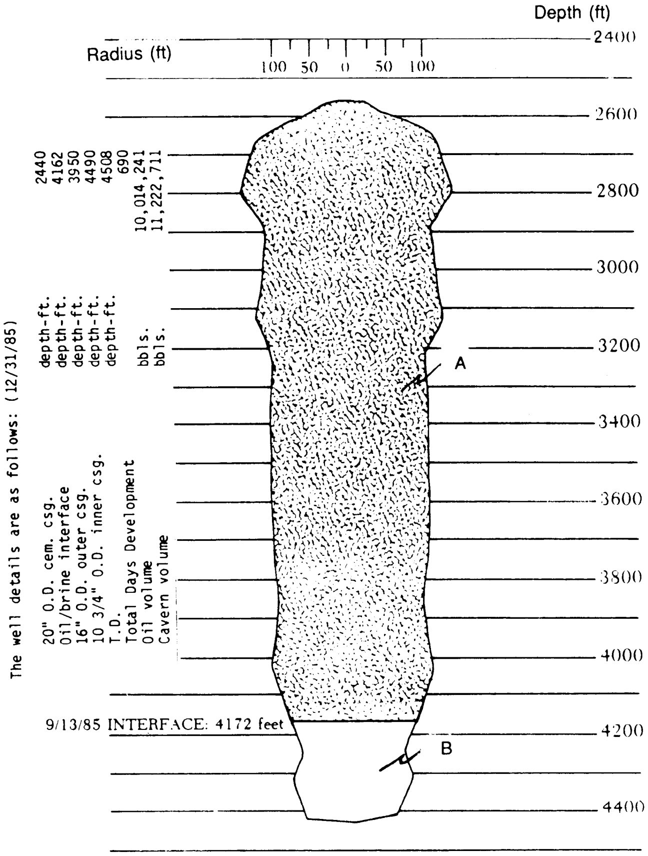


Fig. 15.3.11. Typical SPR crude oil storage cavern profile, West Hackberry Cavern No. 101, Louisiana. A=crude oil; B=rock sump (*courtesy:* PB-KBB Inc.). Conversion factor: 1 ft = 0.3048 m.

3. LNG stored in a salt cavern would cause heat flow from the surrounding salt mass into the cavern. This heat would increase the liquid gas temperature and pressure accordingly. Thus the total heat required for evaporation of gas prior to pipeline distribution is reduced by the energy from the natural source.

Library research and limited laboratory experiments were sponsored by KBB and Ruhrgas during the late 1970s. A field experiment was conducted in a small salt cavern under in situ conditions within a salt mine in Germany. This experiment is described in detail in various in-house reports at PB-KBB, Inc.

KBB has proposed the construction of a series of salt caverns to serve as interim storage for LNG arriving by tankers. These caverns would develop needed information for the analysis of thermodynamics (heat and mass transfer) and rock mechanics (cavern wall crack propagation) considerations.

The potential advent of superconduction technology during the next decade may completely change previous concepts concerning the economic feasibility of LNG storage in salt caverns.

NATURAL GAS (Methane)—Natural gas is normally stored within depleted or partially depleted gas fields or gas pools where the pertinent geologic parameters are usually well known and the infrastructure for the production, processing, and transportation of gas already exists.

Special geotechnical and economic conditions must exist in order that storage within salt caverns becomes a viable alternative.

The primary advantage of such storage involves the ability to withdraw gas very rapidly from a single-well/cavern system, independently from other well/cavern systems. The cost and time required for the development of a gas storage cavern system, of course, can be markedly decreased if existing LPG or brine storage caverns (and infrastructure) can be used with only minor modification.

The use of existing LPG or brine caverns, however, must be carefully investigated. Many of the readily available well/cavern systems cannot easily be adapted to storage of a highly mobile gas under considerably higher pressure. Moreover, well/cavern systems more than 10 years old may no longer meet the ever-increasing stringencies of environmental and safety requirements, or may not even be acceptable for repermitting for LPG storage.

The construction of new well/cavern gas storage systems is much more capital-intensive and requires associated water supply and brine disposal facilities. However, design, construction, and cavern development can be tailored to meet present day and/or anticipated future environmental and safety regulations.

Numerous candidates for conversion of existing LPG caverns for conversion to natural gas storage exist within bedded salt deposits (Kansas, Michigan, Ohio, etc.) and Gulf Coast salt domes (Texas, Louisiana). Local and regional natural gas utilities throughout North America are presently evaluating opportunities that will result from a resurgence of industrial and commercial activities.

COMPRESSED AIR ENERGY STORAGE (CAES)—Economical generation of electric power is difficult to achieve because power demands fluctuate widely both on a daily and seasonal basis. One of the "load-leveling" technologies presently being developed is compressed air energy storage within a salt cavern.

The basic elements of a CAES installation are shown in Fig. 15.3.1.16. Conceptually, air is compressed during periods of low electrical energy demand and temporarily stored within a salt cavern. This compressed air is later released, during a period of peak power demand, and allowed to energize a turbine generator.

The first large-scale CAES plant for electric utility load leveling was constructed at Huntorf, Germany, by the Nordwestdeutsche Kraftwerke AG (NWK). This plant stores off-peak energy from the NWK grid by compressing and storing air in caverns constructed within the Huntorf salt dome. The Huntorf dome was extruded from the Zechstein evaporite strata, is elliptical in shape, and has a layered and folded character. Salt was encountered at a depth of 1700 ft (500 m). Total as-constructed cavern volume was about 392,400 yd^3 (300,000 m^3). Minimum cavern air pressure is about 43 bars (4.3 MPa); maximum is 72 bars (7.2 MPa). Caverns were constructed about 700 ft (210 m) apart to minimize risks of salt cavern wall collapse. Minimum salt cavern roof thickness was established through rock mechanics studies at 2.6 times the diameter of the 170-ft (57-m) wide caverns. Cavern wall closure is expected to result in a 3% reduction in volume during the cavern's projected 30-year life.

The success of the Huntorf CAES project has generated considerable interest in the potential development of larger installations in other countries. Within the United States, preliminary studies have been conducted within northeastern Ohio (for the Cleveland Electric Illuminating Co.) and at the McIntosh salt dome in southwestern Alabama (for the Alabama Electric Cooperative, Inc.). The Alabama CAES installation has been constructed and was put into operation in mid-1991. Present construction includes a CAES plant having a nominal rating of 110 MW—supported by one CAES salt cavern having a volume of about $20 \times 10^6 ft^3$ (566×10^3 m^3).

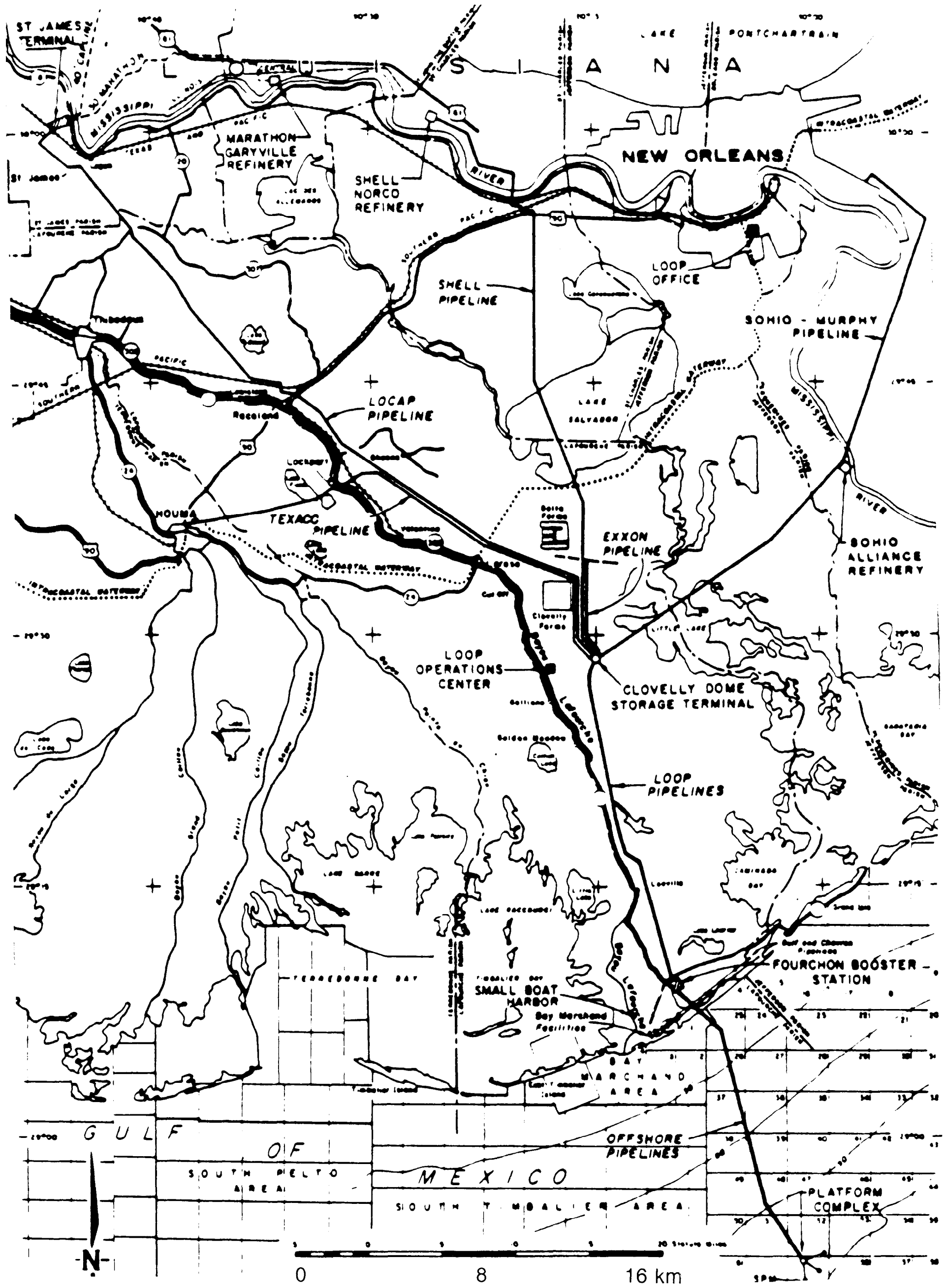


Fig. 15.3.1.12. Locations of the Louisiana Offshore Oil Port and connecting carriers (by permission: LOOP, Inc.). Conversion factor: 1 mi = 1.6093 km.

In designing the underground storage system, material properties of the McIntosh salt deposit, cavern geometry, and operational requirements were considered. Initial site exploration indicated a dissolution zone at the top of the salt, and scattered potash mineralization within the salt. Inasmuch as the solubility rate of KCl can be as much as 20 times that of NaCl, there appeared to be considerable risk that control of salt cavern shape and dimension might be difficult. A new site was selected and test drilled, with satisfactory results.

A preliminary design of the proposed CAES well and cavern construction is presented in Fig. 15.3.1.17. The Alabama CAES well and cavern was constructed at a 1991 cost of about US$/500 million.

MISCELLANEOUS GASES—During the past decade, the geotechnical and economic feasibilities of salt cavern storage of a variety of specialty gases have been evaluated. These gases have included hydrogen, helium, and chlorine.

Most of the time, the problems associated with these "specialty" gases, such as corrosion or embrittlement of tubulars, wellhead, instrumentation, and compressors are too severe to warrant more than feasibility studies and a few laboratory tests. However, the constant development of new materials of con-

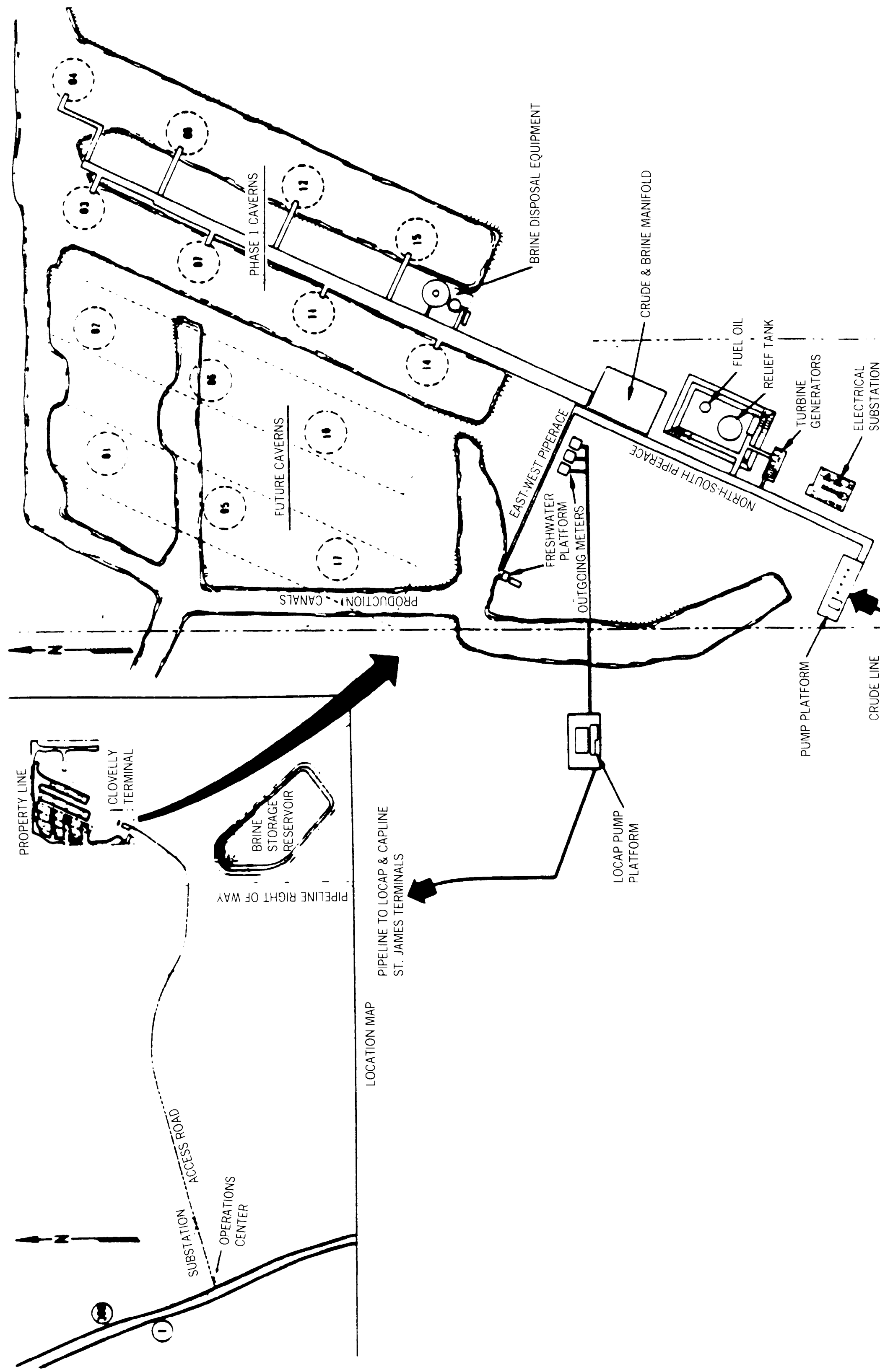


Fig. 15.3.1.13. Plan view of Clovelly terminal, Louisiana (by permission: Walk, Hydell & Associates, Inc.).

Fig. 15.3.1.14. Locations of LPG storage caverns in salt in the United States and Canada (1987 data from AGA). Conversion factor: 1 mi = 1.6093 km.

struction as well as changes in economic incentives generated by new safety and/or environmental requirements require a feasibility reevaluation every three to five years.

One example, the underground storage of hydrogen within aquifers or within conventionally mined or solution-mined facilities in the United States and Europe, has been proposed on numerous occasions. The primary driving force behind these proposals concerns safety. The hazardous nature of hydrogen arises chiefly due to its ability to combine with oxygen over a wide range of partial pressures, with very little energy requirement for activation.

Actual underground hydrogen storage experience has been very limited. Much of the relevant experience and operating data has been developed from an underground hydrogen-methane storage facility located at Reitbrook, West Germany. Here, more than 424 million ft^3 (12 Mm^3) of refinery gas, more than 60% hydrogen, was stored within an aquifer at a depth of 850 to 1300 ft (260 to 400 m).

Preliminary studies have indicated that solution-mined caverns will become the most attractive nonporous storage strata type, for all the reasons listed at the beginning of this underground storage segment.

15.3.1.11 Underground Disposal

Past and present regulations forbid the disposal or storage of radioactive wastes within solution-mined caverns. Industrial wastes from salt companies and chlor-alkali producers have been frequently disposed of in caverns with or without regulatory controls. The disposal of hazardous wastes, however, is a developing technology that is presently surrounded by political and geotechnical controversy. The topic of radioactive waste disposal is dealt with in Chapter 24.3.

INDUSTRIAL WASTES. Raw brine from a solution mine contains a small percentage of both water-soluble and insoluble substances that must be removed prior to introducing the brine into the chemical process plant system. Procedures used mainly involve clarification, settling, and/or filtration. Primary waste products contain calcium, magnesium, and iron, which can be treated and removed as solid precipitates (calcium carbonate, calcium sulfate, iron oxide, or sulfide, etc.) A portion of these sludges can be refined and marketed (e.g., calcium chloride, calcium carbonate); most have simply been contained within waste ponds, a form of landfill. Inasmuch as nearly 85% of these sludges consist of strong brine solutions, the potential for leakage

Table 15.3.1.4. North American Underground LPG Storage Capacity, Solution-Mined Caverns, 1987 and 1989 (Capacity in Thousands of Barrels)

	Caverns	Ethane	Propane	E-P Mix	Normal Butane	ISO Butane	Natural Gasoline	Raw Mix	Ethylene	Propylene	Other	Totals (1987)	Totals (1989)
United States													
Arizona	16	—	2,901		955	1,192	—	—	—	—	—	5,048	4,113
Kansas	586	1,114	40,712	3,446	11,345	4,176	3,812	3,411	—	81	330	68,427	67,114
Louisiana	76	6,591	23,213	1,547	13,830	5,693	2,289	5,726	10,261	3,882	16,877	89,909	90,646
Michigan	31	—	4,057		989	250	—	5,000	—	—	—	10,290	10,230
Mississippi	21	1,128	20,785	—	959	524	—	—	—	—	971	24,367	24,267
New Mexico	12	—	913	60	—	—	—	189	—	—	—	1,162	1,162
New York	15	—	916	—	120	115	—	—	—	—	955	2,106	2,512
Ohio	2		4,000	—	250	—	—	—	—	—	—	4,250	7,277
Oklahoma	73	—	815	281	333	17	61	499	—	—	—	2,006	2,327
Texas	294	13,502	59,633	38,984	34,200	18,301	7,791	19,526	24,688	15,994	33,354	265,973	264,076
Utah	1	—	—	—	—	—	—	—	—	—	—	200	115
Totals	1,127	22,335	158,145	44,318	62,975	30,268	13,959	34,351	34,949	19,957	52,487	473,738	473,839
Canada													
Alberta	34	2,960	7,585	—	2,887	683	—	2,245	—	—	1,400	17,760	
Maritime Provinces													
Ontario	50	990	2,192	230	2,819	60	—	4,160	—	87	1,857	12,395	
Saskatchewan	8	2,000	6,600	—	600	—	—	—	900	—	600	10,700	
Totals	92	5,950	16,377	230	6,306	743	—	6,405	900	87	3,857	40,855	31,231

Courtesy: Gas Processors Association. Conversion factor: 1 m^3 = 0.159 barrel.

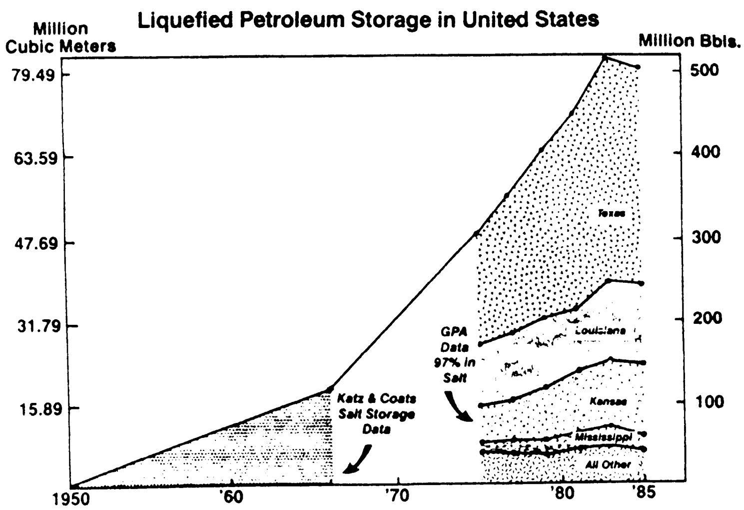


Fig. 15.3.1.15. Growth of LPG storage in the United States, 1950–1985 (*courtesy:* Gas Processors Association).

into the surface and subsurface drainage system has elicited considerable environmental concern.

Several companies have attempted to solve this problem by decreasing the liquid content of the sludge and pumping the solids into brine cavities that are being retired from active brine production. Many of these old well/cavern systems cannot meet modern environmental requirements, or are hydraulically connected to nearby production/storage caverns.

Under present federal standards, well/cavern systems must be permitted as Class II wells before disposal of industrial wastes is allowed.

HAZARDOUS WASTES. Many substances have been included within federal and state listings of hazardous and/or toxic wastes, subject to regulatory control.

During the 1970s, elemental mercury and many of its compounds were severely restricted because of their widespread usage and highly poisonous nature within the food chain. Mercury regulation has had considerable impact upon both the mining and the chemical processing (chlor-alkali) industries. Several companies using mercury cells for chlorine production had experimented with temporary disposal of mercury-bearing sludges within a solution-mined cavity, but these efforts had to be abandoned.

Landfill operations are rapidly being restricted as to location, operation, and usage. Subsurface disposal on a large scale in solution-mined cavities is now being investigated—amid strong controversy—in the state of Texas. As described in 15.3.1.12, federal, state, and local agencies all have a role in the permitting process.

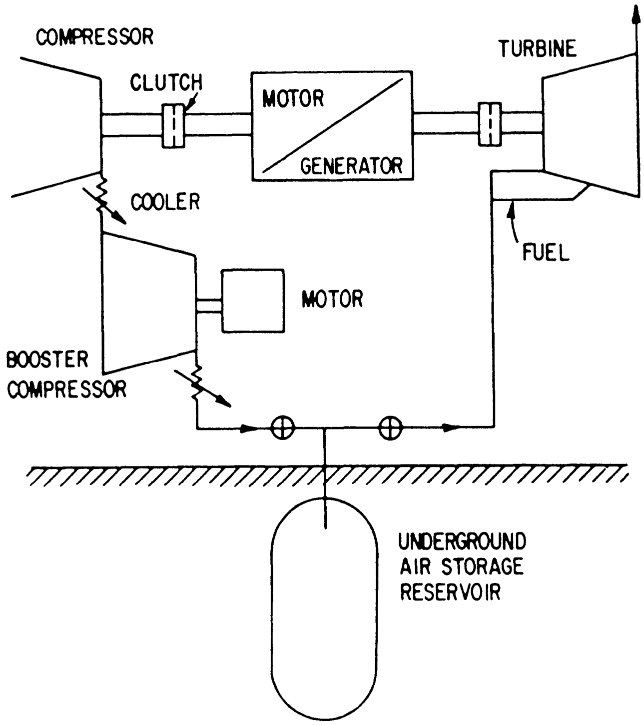


Fig. 15.3.1.16. Elements of a compressed air energy storage installation (*courtesy:* Electric Power Research Institute).

Five attempts to initiate hazardous waste disposal systems in evaporites in the United States are summarized in the following.

1. *Calcasieu Parish, LA.* The first proposed disposal installation was to be located in the Vinton salt dome, Calcasieu Parish, LA, where the top of a salt deposit was found at 700-ft (210-m) depth. The proposed installation was advanced by IMPACH, a Dutch-owned firm. This proposal was stopped by the Louisiana legislature, which voted for a two-year moratorium on disposal projects in salt domes. This moratorium has never been rescinded.

2. *Wharton County, TX.* United Resource Recovery Inc. prepared a study for the disposal of liquid toxic wastes in caverns to be constructed within the Boling salt dome in Wharton County, TX. The top of the salt deposit was at 975 ft (297 m), and proposed cavern diameters were about 100 ft (30 m). A permit application was submitted to the Texas Water Commission, which, after public hearings, rejected the permit application. This decision is being appealed.

3. *Anderson* County, *TX.* PB-KBB/ESE jointly prepared an extensive study of solid-waste disposal within the Keechi salt dome in Anderson County, TX. The project, named TEXTOR, found salt at a depth of 400 ft (122 m) in a corehole drilled in 1988. A multi-volume application for a permit was submitted to state authorities, but was withdrawn prior to required public hearings, again because of the loud outcry from local opposition forces.

4. *Liberty County, TX.* Hunter Environmental Inc., through it subsidiary Hunter Industrial Facilities Inc. (HIFI), has prepared a thorough study of solid-waste disposal within caverns to be constructed within the North Dayton dome in Liberty County, TX. Top of salt is at 800 ft (240 m) depth. HIFI proposes a system of 10 caverns, 120 ft (37 m) in diameter, constructed through the depth interval 1500 to 4000 ft (457 to 1219 m). Each cavern would have a fill capacity of about 4 million bbl (22.5 million ft^3, or 637 km^3).

The proposed surface plant would receive liquid toxic waste, which would be mixed with cementitious material and formed into solid waste pellets before injection into a cavern.

HIFI has submitted a multi-volume draft permit application to the Texas Water Commission and public hearings are expected soon. Again, there is considerable vocal opposition from regional and national environmental groups who dislike the concept of underground disposal of wastes.

5. *Detroit, MI.* A fifth waste disposal study and permitting attempt is underway at Detroit, MI, where a portion of the large, abandoned International Salt mine is available for storage/disposal activities.

There currently is controversy concerning "owners" and "users" rights within the mine and a formal application for a permit to dispose of hazardous wastes has not yet been submitted to Michigan authorities for consideration.

15.3.1.12 Government Regulations

All aspects of subsurface investigation, permitting, construction, operation, and abandonment of solution-mined well/cavern systems increasingly fall within the purview of government regulation (Chapter 3.4). Most of these regulations have evolved through environmental concerns about the long-term stability of underground openings, the potential for environmental damage to the enveloping strata, groundwater, ground surface, and atmosphere, and to the safety of personnel and other life forms at risk from an existing or proposed activity.

The primary federal regulating agencies are the EPA (Environmental Protection Agency) and OSHA (Occupational Safety and Health Administration). EPA now designates solution-mining and underground storage well/cavern systems as Class II wells and solid waste disposal well/cavern systems as Class I. Although headquartered in Washington, DC, EPA exercises control through regional offices and through publication of the *Code of Federal Regulations,* Title 40, entitled Protection of Environment and updated annually. Additional information and/or commentary is readily available through subscription to the *Environmental Reporter* (Bureau of National Affairs, Inc. [BNA], Washington) and the Underground Injection Practices Council, Inc. (UIPC) located in Oklahoma City, OK.

Regional, state, and local regulatory agencies occasionally promulgate modifications of federal law to meet specific geotechnical, environmental, economic, or political concerns. The potential contamination of groundwater supplies or of surface crops by brine leakage out of well/cavern systems has some historical basis in fact and can be readily conjured as a threat to local or regional populations.

The "permitting" process, the procedure necessary to obtain permits from *all* regulatory agencies for the drilling and testing of wells within a given area or region, may require several months to a year or more, largely depending upon the amount of controversy developed during the "public hearing" that follows submission of a permit application (Chapter 7.3). As of June 1990, no permit application for the storage of hazardous waste within a salt deposit has been approved.

Regional, state, and local regulatory agencies have frequently issued even more restrictive rules. Within the state of Texas, for example, an applicant should be familiar with the Texas Railroad Commission's *Forms and Procedures Manual,* Oil and Gas Statewide Rules, and the Texas Oil and Gas Conser-

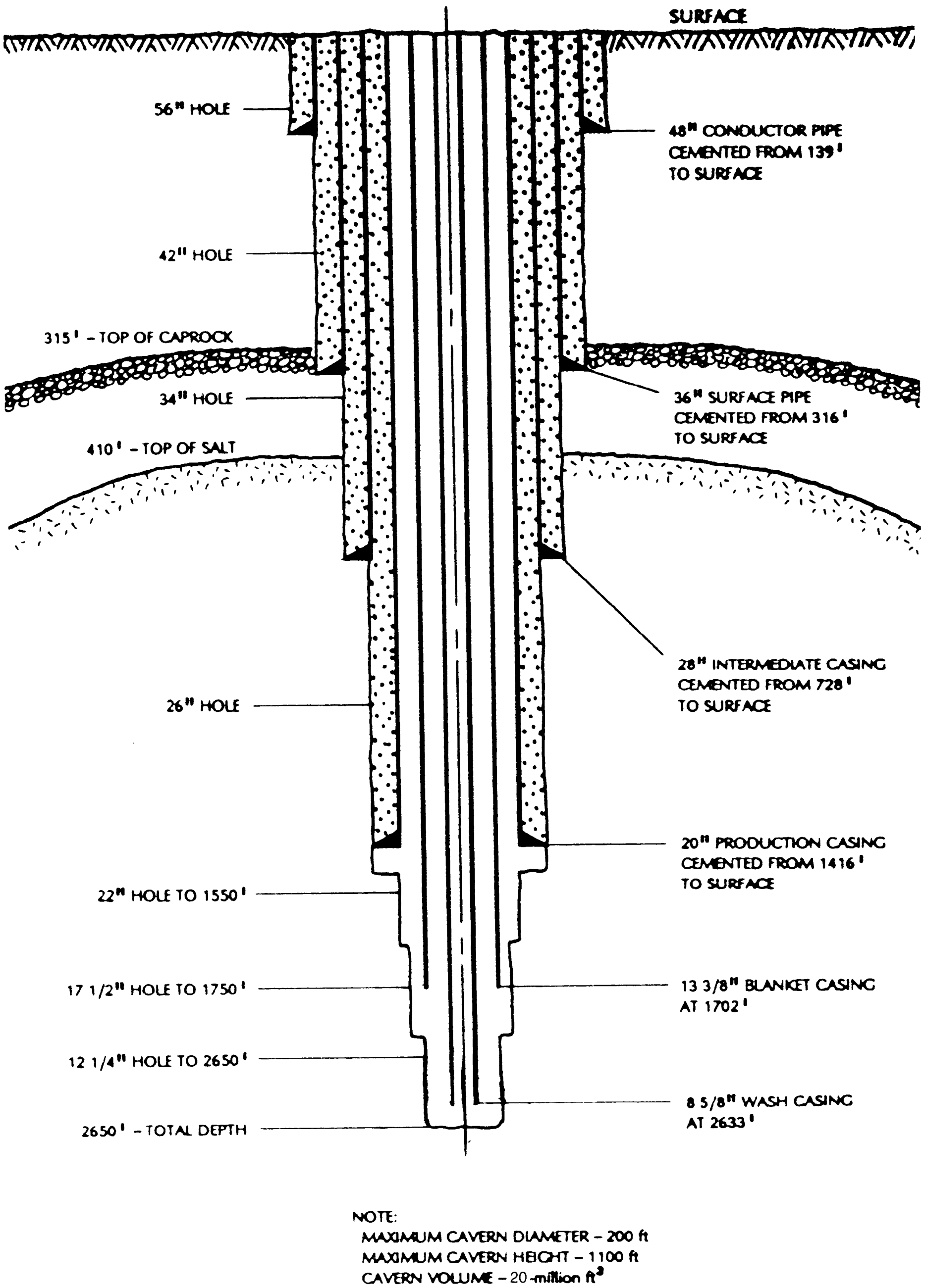


Fig. 15.3.1.17. Casing schematic of AEC CAES production well (not to scale) (*courtesy:* Electric Power Research Institute). Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 ft^3 = 0.02832 m^3.

vation Laws. The Texas Water Commission also exercises authority through the federal Safe Drinking Water Act.

The list of regulatory agencies that may influence a desire to construct an underground disposal well/cavern system is beyond the scope of this discussion.

15.3.1.13 Testing Solution-mined Well/Cavern Systems for Mechanical Integrity

Many well/cavern systems have developed avenues of fluid leakage into their subsurface and ground surface environments. This situation is particularly common with respect to wells drilled prior to 1980 when environmental regulations had neither the scope nor the impact that they have now.

During the past decade, the primary federal environmental agency, the US EPA, has promulgated increasingly more restrictive regulations concerning the design, construction, and operation of *all* wells, as briefly outlined in 15.3.1.12.

A major physical test, required by EPA before operational start-up of a well/cavern system, is commonly referred to as an MIT (mechanical integrity test). The EPA requires this test of the last cemented casing string at five-year intervals, at a minimum. Under special conditions, the MIT may be required by EPA, regional, state, or local agencies more frequently.

Several MIT procedures have been accepted by federal and state agencies, depending upon the region the test is to be employed. Nitrogen is a preferred testing medium although liquids or other gases may be used under special conditions.

An outline of a procedure commonly used in the Gulf Coast region is presented as follows:

1. *Cavern Preparation* .

a. Inspect wellhead for leaks or excessive corrosion. Correct as necessary.

b. Isolate cavern from other parts of the surface and subsurface system.

c. Provide access opening on each wellhead (a lubricator) to allow a logging tool unrestricted access to the center of the cemented casing or brine tubing.

d. Provide for pressure taps to connect continuous pressure recorders.

e. Provide source of saturated brine and pumping equipment to pressure the well.

f. Stop all stored product movement activity within the well/cavity system for at least 24 hr prior to test.

g. Prepare suitable access to the wellhead for logging and nitrogen trucks.

2. *Pretest Procedure* (minimum 24 hr before test).

a. Connect *calibrated* pressure gages to taps.

b. Pressurize well/cavern system to predetermined pressure, using saturated brine if possible.

c. Note and record pressures on the cemented casing annulus, casing/tubing annulus, and tubing string for at least a 24-hr period.

(Leakage discovered while using a liquid medium for pressurization will be greatly magnified under gaseous testing conditions.)

3. *MIT Procedure Using Nitrogen.*

a. Rig up wireline lubricator on the valve flange provided. Place interface/density logging tool into lubricator. Run tool into well casing (or tubing).

b. Make and record temperature survey of entire well to establish a baseline log.

c. Set interface/density tool at a predetermined depth. Inject nitrogen at predetermined rate (and volume) until gas/liquid interface reaches logging tool. Shut in for short test period; calculate measured nitrogen injection against casing (or casing/tubing) capacity at selected down-hole pressures. Compute amount and rate of leakage, if any.

d. Lower logging tool in depth stages, repeating nitrogen pressure test and subsequent calculations for each stage.

e. When the gaseous liquid interface reaches the casing shoe, shut in the well for at least a 24-hr period to test the entire cemented casing string for leaks.

4. *Cavern MIT Procedures.*

Principles and procedures involved in testing caverns are very similar to those previously described for testing the well. However, much larger volumes of the test fluid medium, higher wellhead and down-hole pressures, and greater testing-time intervals will be involved.

In general, most cavern leakage will occur behind the cemented casing shoe and/or within caved portions of the cavern roof. Specialized remedial techniques are frequently available for decreasing or eliminating such leakage.

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15.3.2 FRASCH SULFUR MINING

D'ARCY SHOCK

15.3.2.1 History

The process by which elemental sulfur is produced from a geological formation is singularly credited to Herman Frasch. His concept of the mechanism of melting sulfur and then airlifting the molten liquid to the surface required a persistent effort in technical development and an expenditure of time and capital for a span of over 10 years before realizing significant success. The process is rightly named the *Frasch process* in his honor.

The initial production of sulfur by the airlift Frasch process began in 1896 with the operation of three wells which produced 2100 long tons (2134 t) of sulfur (Hynes, 1942). Difficulties in keeping the wells in production and high operating costs delayed a rapid growth of the system. With the discovery of the Spindletop oil field in 1902 and subsequent availability of inexpensive fuel, as well as improved petroleum well technology, more salt dome sulfur deposits were located, and the Frasch sulfur mining system became well established. Frasch sulfur produced at the Gulf Coast salt domes constituted the major source of US sulfur production and dominated the world market until approximately 1970. At this point, sulfur recovered as a byproduct of oil operations became a significant factor (Morse, 1986).

Decreased oil exploration in and around the Gulf of Mexico reduced the discovery of sulfur-bearing salt domes. As a result, sulfur prices strengthened and provided the incentive for development of Frasch-process mines in West Texas where the sulfur occurs in structural traps not associated with salt domes (Ellison, 1971; Anon., 1970). At the same time, the incentive to remove the sulfur from petroleum products continued; so that in 1982, petroleum byproduct sulfur production exceeded Frasch production, and this gap has continued to grow. Frasch sulfur mining has been applied to Mexican deposits that are similar to those along the Texas and Louisiana Gulf Coast. The prospects for growth in the United States seem doubtful, unless significant offshore salt dome sulfur deposits are discovered during oil drilling in the Gulf of Mexico. From a worldwide standpoint, however, the Frasch process undoubtedly will be applicable to situations where fuel and water are available and the geological circumstances are favorable, for example, the Mid-East petroleum production areas, Poland, and Russia.

15.3.2.2 Basic Requirements for a Frasch Sulfur Mine

Three basic resources are needed to develop a profitable Frasch sulfur mine. They are (1) a large deposit of ore of good grade (5% sulfur is suggested as a lower limit); (2) an adequate, reliable, and inexpensive source of water, preferably containing little dissolved solids; and (3) a low-cost source of fuel. Other factors such as market price, royalty payments, tax liabilities, and transportation costs also influence the economic feasibility of the project.

Utilities required to mine sulfur have been quoted as (1) an average water requirement of 5000 gal/long ton of sulfur (19.2 m^3/t) mined, (2) a fuel requirement of 2700 to 3000 Mcf of natural gas/million gal of mine water (20,000 to 22,300 Mm^3 of gas/m^3 of water), and (3) power requirements 600 to 700 hp/million gal/day of mine water (118 to 138 W/m^3 of water) (Anon., 1967). Reliable and economic transportation systems

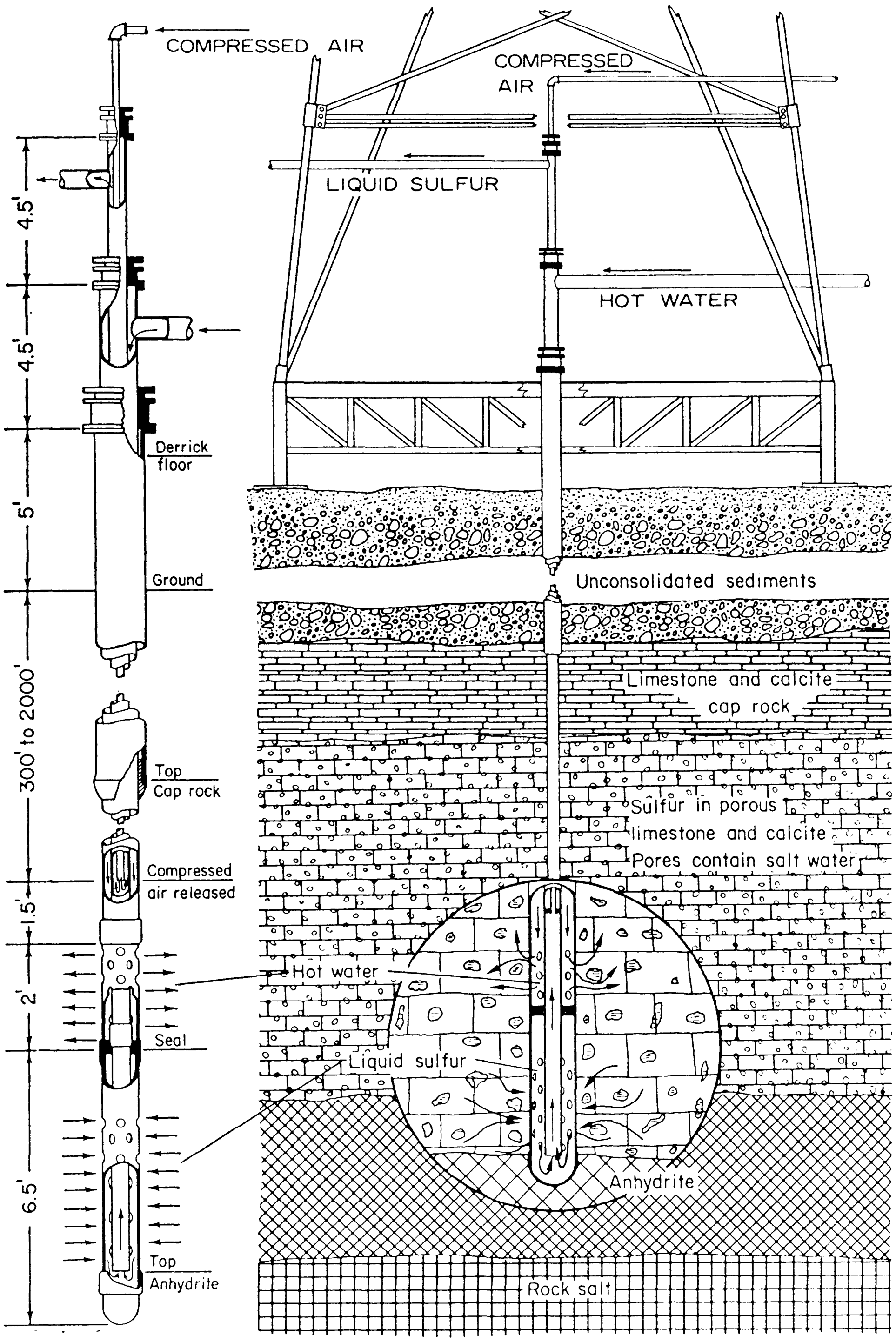

Fig. 15.3.2.1. Operation of a sulfur well (after Ellison, 1971). Conversion factor: 1 ft = 0.3048 m.

are also essential for distribution of the bulk sulfur. In this regard, transportation and storage of molten sulfur has been growing, particularly where low-cost water transport can supply shore-based distribution terminals. Sophisticated production personnel are important to operate and maintain the production and delivery systems.

15.3.2.3 Ore Deposit

The Frasch process was originally applied to removing sulfur from deposits located in association with salt domes. These domes have been formed by the intrusive flow of deep-bedded salt that penetrates the upper layers of sediments. Typically, the sedimentary layers are mud, shales, limestone, gypsum, and anhydrite. The salt plug normally is covered by an insoluble mineral cap. The sulfur is found below the cap in the vuggy porosity of the limestone. The sulfur location in the cap rock of the domes is highly variable, and not all salt domes contain sulfur-bearing rock.

Sulfate-reducing bacteria in conjunction with hydrocarbons and water are clearly capable of producing elemental sulfur. Thus sulfur can be formed from the gypsum and anhydrite and be precipitated in the porous limestone with the progression of water circulation.

The West Texas deposits of sulfur are associated with layers of gypsum, anhydrite, limestone, and dolomite. These deposits are not associated with salt plugs, and the porosity and permeability of sulfur-bearing beds follow the natural layered porosity of the strata and fracture system (Zimmerman and Thomas, 1969; Hentz et al., 1987).

15.3.2.4 Field Development

Efficient extraction of sulfur from the deposit depends on several interrelated factors. The hot water has to be introduced in such a way and at such a rate as to melt the sulfur so it will form a pool below the hot water. This pool of sulfur is then airlifted to the surface. The piping system consists of concentric pipes which extend into the minable formation (Fig. 15.3.2.1). The operating strings are a series of these concentric pipes. The outer casing is set into the caprock above the sulfur-bearing strata. The next string conducts the hot water and is set through the sulfur-bearing strata and is perforated at the bottom and also at some distance up the pipe. Another pipe is set concentrically within this pipe, and an internal seal is placed between the upper and lower perforations so that the flow of hot water is above the lower level. At a higher level, another pipe is run concentrically. The piping diagram is illustrated in Fig. 15.3.2.1, showing how hot water is introduced first, melting the sulfur. The sulfur, which is heavier, sinks to the bottom, runs into the lower pipe, and fills the second string. The molten sulfur is then airlifted with the compressed air and emerges as shown in the figure.

The molten sulfur is sent to storage, which can be handled in a number of ways. Since it may contain solid materials, or even small amounts of acid hydrocarbons, further processing may be required. Steam-heated leaf filters are commonly used for sulfur filtration. Precoating, usually with diatomaceous earth, is almost universal. The precoat should do the actual filtration to minimize blinding. The precoat will also neutralize some acid and reduce corrosion. However, small amounts of hydrated lime or ammonium bicarbonate may be added to the filter feed to neutralize acid. Such a system will produce sulfur that is free of ash and acid. If hydrocarbons must also be removed, activated clay may be added to bleach the dark sulfur. Other filter aids are also available to remove hydrocarbons. Filtration rates and precoat requirements will depend on the quality of the sulfur coming from the wells.

The sequence of the development of the well is shown in Fig. 15.3.2.2. The initial injection melts a pool of sulfur and seals the

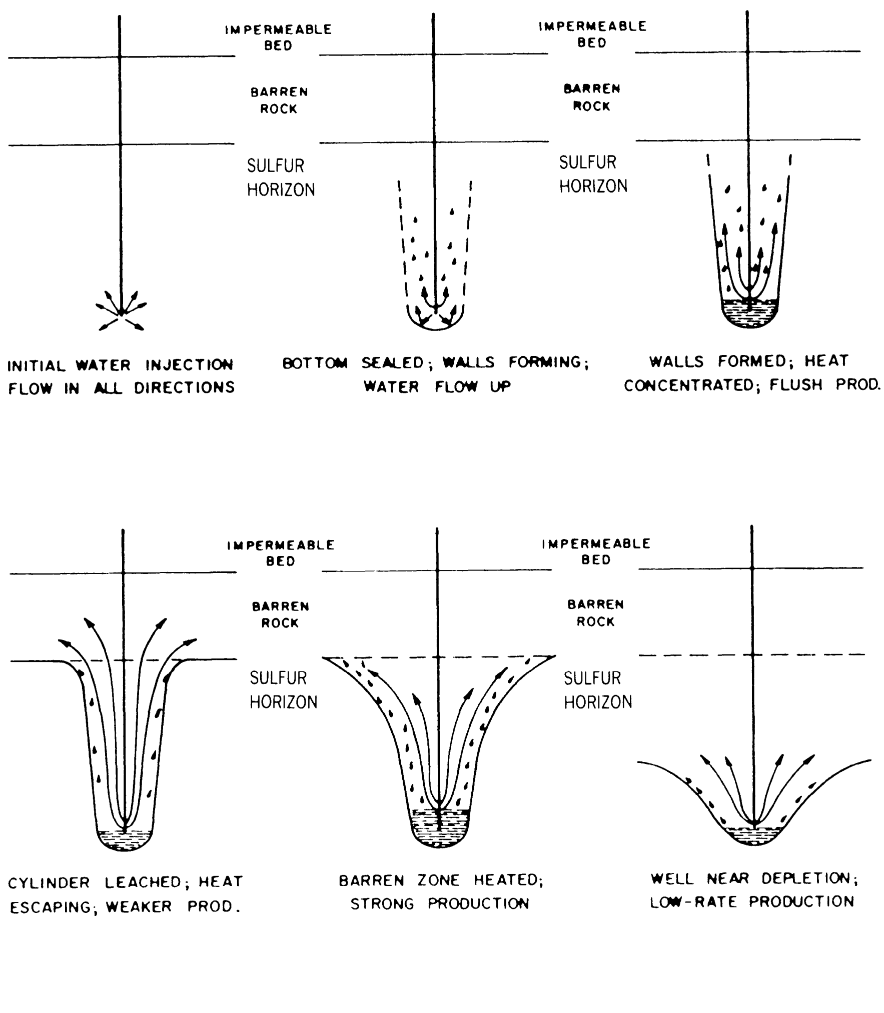

Fig. 15.3.2.2. Sequence of operations during the operation of a sulfur well (after Donner and Wornat, 1973).

bottom as shown in the figure. Hot water is forced further up the string, and a larger cylinder of rock is heated, melting more sulfur which is airlifted to the surface. This yields a concentrated flush production. This scenario is simple in concept; but in practice, where variability of permeability, sulfur content, heat flow problems, and the influence of other wells exist, the actual operation is very complex. It is necessary to reinject water produced in bleed wells to control flow, save heat, and dispose of waste water. The mining process is highly variable due to the nature of the deposits; and the uncontrollable nature of the reservoir system makes the process something of an art, that is gained by years of experience. Attempts have been made to apply reservoir engineering by substituting paradichlorbenzene as a meltable material for sulfur with some interesting insights (Rayne et al., 1956).

As mining proceeds, subsidence and collapse of overburden become a problem that can sometimes help, but may also shear tubing and cause many difficulties. Wells are often separated into groups to help control these problems and avoid total loss of field control. Corrosion due to molten sulfur contact is quite negligible. When the sulfur pool is drawn down to where the hot water and air come up together, the water flashes as steam, and a "blow" takes place. Hot water rising up the sulfur production string produces a corrosive environment; and with a blow, it becomes appreciable. The salinity of the water influences the rate markedly. If sea water is used and a blow occurs, the corrosion rate is disastrous (Hackerman and Shock, 1947; Shock and Hackerman, 1949). One of the practical solutions to controlling corrosion has been to use specially produced cement-lined pipes as well as limiting the salinity of the water. Treatment of the water to minimize corrosion and scale problems is essential.

Bleed wells are required to balance the water pumped into the formation at the production wells. Location of the bleed wells will depend on such site-specific factors as the depth, permeability, and hydrology of the sulfur-bearing structure. Due to environmental concerns, bleed water should be recycled to the maximum extent possible. When required, disposal of bleed water needs to be carefully planned due to the high concentration of dissolved hydrogen sulfide and other sulfur-bearing compounds. Aeration in ponds, chemical treatment, and use of deep disposal wells are all methods of handling these waters.

15.3.2.5 Facilities Needed at a Frasch Sulfur Mine

When a sulfur deposit has been located and defined by exploration drilling, a water supply located, and a fuel supply established, the design and construction of the needed facilities can proceed. Typical layouts showing both the well field and surface facilities are provided in the literature (Anon., 1970). Most mines will require some or all of the following:

1. Pumps, pipelines, and reservoir for the water supply.
2. Power plant for heating and treating the water, compressing the air, and generating the electricity and steam.
3. Equipment for drilling and servicing wells.
4. Pipeline system for carrying the mine water and air to the wells and the sulfur to collecting stations.
5. Control stations for the producing wells.
6. Sulfur collecting, loading, and storage facilities.
7. Purification equipment needed to meet customer specifications.
8. Shipping facilities for solid and/or liquid sulfur.
9. Bleed water disposal system.
10. Equipment needed for the maintenance of these facilities.
11. Employee transporting, dining, and housing facilities if the mine is remotely located.
12. Offices and buildings for material storage, laboratory, engineering, personnel, administration, and supervisory staff.

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15.3.3 IN SITU MINING OF HARD-ROCK ORES

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15.3.3.1 Overview of Methods

Today hard-rock in situ leaching techniques are being utilized to extract uranium and, on a small scale, oxide copper from undisturbed deposits. *In situ leach mining* involves the recovery of metal values from an ore body by circulating solutions through the ore in its native, essentially undisturbed geologic state and recovering those solutions for processing. In situ leach mining has been conducted from the surface (Fig. 15.3.3.1) as well as

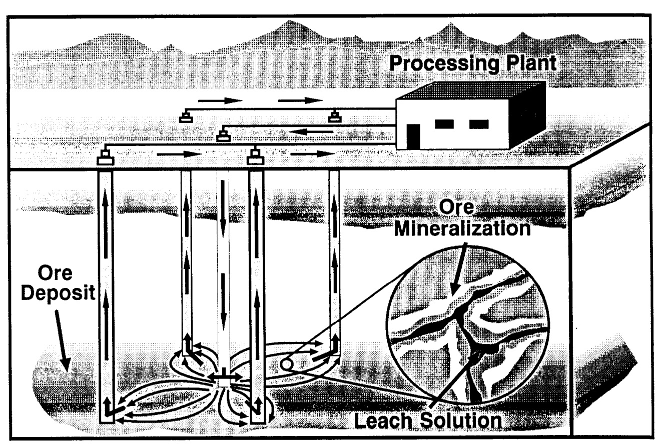

Fig. 15.3.3.1. In situ leach mining from the surface.

from underground workings (Fig. 15.3.3.2). By contrast, in conventional surface and underground mining, the ore is primarily recovered by large-scale, bulk handling techniques, through rock breakage and transport. In situ leach mining combines the operation of both surface processing facilities and subsurface remote mining in a way completely different from conventional technology.

A brief description of the in situ leach mining process is given here. A lixiviant appropriate to the target commodity or commodities (see Chapter 15.2) is used to dissolve the target metal and maintain production from the ore body, with solution processing occurring at a surface facility. A series of injection wells are used to inject or force the lixiviant into the pores and microfractures of the rock (ore). This solution travels through openings in the ore body and reacts with the minerals, mobilizing and transporting the metal species in solution. These openings may be the result of natural permeability or may be enhanced by using such oil field techniques as hydrofracturing (hydrofracing). A series of recovery wells are used to create a low-pressure hydrologic gradient where the metal-bearing solution can be collected and pumped to the surface. The metal-bearing solutions are processed for metal recovery and for regeneration of leach solution for subsequent reinjection through the flow paths and pores of the rock. Eventually, the metal values are depleted from the ore body, and a new series of wells are established to continue the mining operation.

In situ leach mining offers several advantages that may make metal mines more productive while reducing overall costs. Past in situ leach mining operations have shown that capital investment is reduced compared with conventional mining techniques. The advancement of this technology will ultimately mean that selected lower-grade deposits can be mined economically, thereby deriving maximum utilization from available resources. Because only fluid is being pumped, and ore is not being fragmented and transported to the surface, the energy requirements to produce the final product are significantly less. Other benefits of this technology include improvement of the health and safety environment of the miners, as they will not be directly exposed to the hazards of mining in and around the traditional rock handling and production areas.

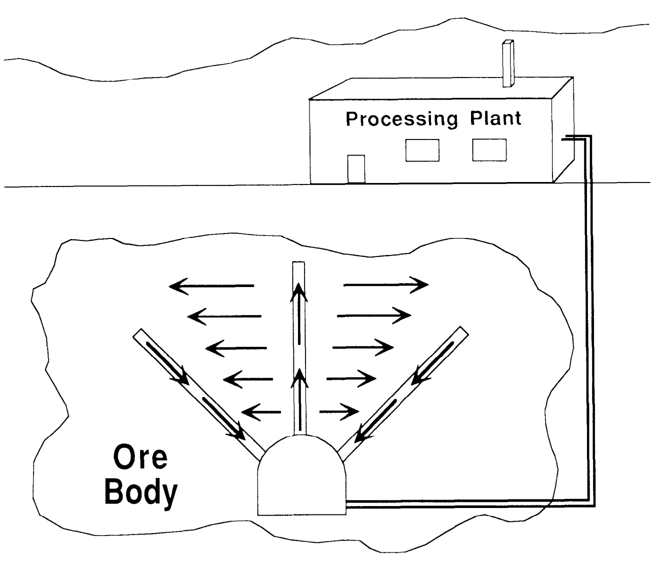

Fig. 15.3.3.2. In situ leach mining from underground.

The environmental impacts of this technology are much less than those of conventional mining from a waste-generation or air-quality standpoint. In situ leach mining removes the metal values from the host rock in place, and therefore will leave no waste piles or excavations to plague future generations. The primary environmental concern regarding this technology is the need to control and contain the leach solutions within the confines of the ore body. Restoration of the in situ leach mining zone may be required in order to prevent lixiviant excursions which could contaminate adjacent aquifers.

General advantages of hard rock in situ leach mining are summarized as follows:

1. No major surface disturbances, including open pits, waste dumps, or tailings.
2. No surface subsidence for most applications.
3. Minimal surface environmental impacts, including elimination of heavy mining equipment and processing plants such as mills and smelters.
4. Reduced labor requirements.
5. Reduced health and safety risks to workers.
6. Reduced energy requirements.
7. Quicker and more flexible startup and shut-down.
8. Lower capital investment.

However, in situ leach mining is not a universal panacea that is applicable to all types of deposits. Difficulties may include the following:

1. Difficulty in controlling or containing leach solutions due to faulting or lack of confining strata above and below the ore body.
2. Lack of suitable lixiviant for the contained metal value.
3. Inability to recover more than one or two values from polymetallic deposits.
4. Inability to accurately assess the level of recovery of values from the deposit.
5. Generally lower recovery of values than would be achieved in a heap leach on the same material.
6. Slower rate of recovery and lower overall level of recovery than that achieved in a surface milling and flotation or leaching operation.

15.3.3.2 Historical Perspective

URANIUM. Uranium is the only commodity that is successfully recovered in widespread, commercial, in situ leach mining operations. Commercial in situ uranium mining began in Wyoming in 1963 at the Shirley Basin site of Utah Construction and Mining Co., later Utah International (Anderson and Rithchie, 1968; Gardner and Rithchie, 1967). The company had started production in 1960 by underground mining, but poor ground conditions led to attempts at in situ leach mining. The company experimented with the system from 1961 to 1963. Commercial production with a sulfuric acid lixiviant continued from 1963 to 1969, when the company decided to extract the remaining uranium with open pit mining.

In Texas, experimentation with in situ uranium mining began in 1966 when the Dalco Co. (a subsidiary of Sabine Royalty) was organized to undertake uranium research and development (Knape, 1984). The first Texas permit for commercial leaching of uranium was issued in 1975 to Atlantic Richfield Company (ARCO) and its partners, US Steel and Dalco, for operation at their Clay West site in Live Oak County.

In situ uranium mining then expanded rapidly in the 1970s. In 1978, 12 in situ mines contributed 2,000,000 lb (900 t) of uranium out of total US production of 41,600,000 lb (19,000 t), or about 5% of the total (Shuck, 1979). US Bureau of Mines (USBM) personnel estimated the contribution, mostly from south Texas and Wyoming, grew to about 10% during the next two years. However, in the 1980s, the price of uranium decreased from over $40/lb ($88/kg) to less than $11/lb ($24/kg). Interest in opening new mines decreased along with the decrease in uranium prices. However, mines that had relatively low production costs and longer term contracts at higher prices continued to operate.

As of February 1989, there were three operating in situ uranium mines in Wyoming. One produced 923,000 lb (420 t) of U_3O_8 in 1988. The production of the other two was not reported. Only one of the other two was commercial. Informed sources in the industry unofficially estimate Wyoming in situ uranium production to be close to 2,000,000 lb/yr (900 t/yr). There was one mine in Nebraska. It was a pilot operation in 1988, so production was small. In Texas, there were four companies with six active commercial mines and at least one nearing the permitting stage. Although companies have not reported their production, informed sources estimated Texas in situ uranium mining produced somewhat more than 2,000,000 lb (900 t).

Total uranium production in the United States was 13,000,000 lb (5900 t) in 1987 (Anon., 1987), and was estimated to be about the same in 1988. Thus in situ uranium mining contributed about one-third of the total production. That fraction is expected to increase, as conventional mining declines. Byproduct uranium recovered from phosphate operations made Florida the largest producer of uranium in 1988, with an unofficial estimate of close to 4,000,000 lb (1800 t) produced. Conventional mining contributed the remaining one-third of production.

COPPER. There has never been a commercial in situ copper mining operation in undisturbed (unbroken) ore. However, a few in situ leach mining tests have been conducted and are described by Ahlness and Pojar (1983). These tests have been conducted in both oxide and sulfide ores at depths ranging from 100 to 2000 ft (30 to 600 m) below the surface. Lixiviant application and recovery have been conducted through wells and underground mine workings.

Perhaps the earliest in situ leach mining test took place at the Medler mine near Clifton, AZ, between 1906 and 1909. Mine water was pumped into underground mine workings on one level and allowed to seep through 60 ft (18 m) of rock to the next level where the pregnant solution was collected and processed. The copper mineralization was in sulfide form, and pregnant solution grade ranged from 0.2 to 0.6 g/L.

A major in situ mining research effort was conducted in the mid-1970s by Kennecott Minerals Co. at the Safford deposit near Safford, AZ. Testing was conducted in sulfide ore to a depth of about 2000 ft (600 m). A five-spot well pattern was constructed and leached, but the specific results are proprietary and have not been released. Commercialization of the site did not occur because of disappointing test results and the low price of copper at the time.

Another significant test was conducted at the Van Dyke deposit near Miami, AZ, in the late 1970s by Occidental Minerals Corp. A five-spot well pattern was constructed with corner-to-corner well spacing at 100 ft (30 m). Each of the wells was hydrofraced, and the leach interval ranged from 1000 to 1200 ft (300 to 600 m) below the surface. Dilute sulfuric acid lixiviant was injected in the center well, and pregnant solution was produced from the four corner wells. The test ran for several months and was considered successful. A commercial operation was not developed at the site because of difficulties in obtaining operating permits. The reason for the difficulty was that the ore zone partially underlay the town of Miami and was viewed as a possible threat to local water supplies.

There are currently two active in situ copper mining test sites (in undisturbed ore) in the United States, with one additional site in the development stage. The active sites are the Casa Grande mine near Casa Grande, AZ, and the Nacimiento mine near Cuba, NM. An in situ mining test facility is under construction at the Santa Cruz site near Casa Grande, AZ.

The Casa Grande mine is located about 30 miles (48 km) southwest of Casa Grande, AZ, and is operated by Cyprus Casa Grande Corp. The mine was originally developed as an underground block caving operation. At the conclusion of the caving operation, in situ leach mining of the remaining rubblized ore was conducted and is continuing to the present time. In addition to the block-cave leach, an in situ leach mining test is being conducted in an undisturbed zone of copper oxide ore. Injection and recovery wells are drilled in a fan pattern from the underground mine workings. Dilute sulfuric acid lixiviant is injected into the initially unsaturated ore zone. Pregnant leach solution is recovered from the recovery wells and is processed in a solvent extraction/electrowinning (SX/EW) plant on the surface. Details of this test are proprietary and have not been released.

The Nacimiento mine is located about 4 miles (6 km) southeast of Cuba, NM, and is operated by Leaching Technology Inc. The mine was originally developed as an open pit operation. The host rock is sandstone, and the primary copper mineral is chalcocite (sulfide mineral). A well field consisting of 9 contiguous five-spots (for a total of 16 injection wells and 9 recovery wells) has been constructed within the open pit. The ore zone is dipping and ranges from 150 to 400 ft (45 to 120 m) in depth to the bottom. The ore to be leached is saturated. The lixiviant used is a ferric sulfate solution. Operation of this in situ site has been sporadic. Decreasing communication between injection and recovery wells has apparently been a problem.

An in situ leach mining field test in a copper oxide deposit is being developed at the Santa Cruz site, which is located 7 miles (11 km) west of Casa Grande, AZ. This field test is a cooperative effort between the USBM and Santa Cruz Joint Venture formed between ASARCO Santa Cruz, Inc. and Freeport Copper Co. The goals of this test are to demonstrate the attainment of projected operating costs, copper production, equipment operability, and environmental compatibility. This will provide the mining industry with the data necessary to design and operate successful commercial in situ leach mining operations. The test may involve two contiguous five-spot patterns (Fig. 15.3.3.3) and is scheduled to run through 1996 (Ahlness, Millenacker, and Swan, 1989).

The Santa Cruz site has an estimated 97 million tons (87 Mt) of ore with an average grade of 0.7% oxide copper. The primary copper oxide minerals are atacamite and chrysocolla. The average depth to the bottom of the mineralization is 2200 ft (660 m), and the average ore thickness is 345 ft (105 m). The host rock is granite and quartz monzonite porphyry, with an in-place permeability of 5 to 20 millidarcies, a low value typical of such deposits.

An SX/EW plant will be built onsite. It will be designed to handle 10 to 50 gpm (0.6 to 3.2 L/s) of pregnant leach solution (PLS) with a maximum production capacity of 1000 tpy (900 t/a) of cathode copper. The dilute sulfuric acid lixiviant will be recycled. In other words, once copper is stripped from the pregnant leach solution, the solution will be reacidified and reinjected into the ore.

As part of its in situ copper mining research program, the USBM has developed a "Draft Generic In Situ Copper Mine Design Manual" (Davidson et al., 1988). This is a detailed source document for in situ leach mine design and economic modeling information and provides a systematic method for assessing the commercial feasibility of in situ leaching at any specific, undisturbed copper oxide deposit. The draft manual will be revised

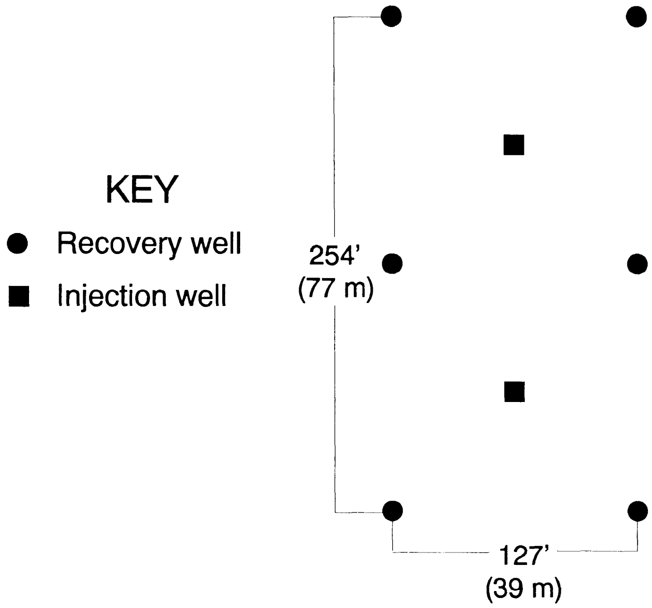

Fig. 15.3.3.3. Two contiguous five-spot well patterns.

by the USBM based upon data collected from the the Santa Cruz test.

15.3.3.3 In Situ Leach Mine Development

The following information is intended to help in determining the conditions under which in situ uranium and copper mining may be feasible and to provide suggestions for conducting initial tests. This information is not a substitute for the experienced, professional guidance required for planning an in situ leach mine. The discussion covers preliminary site evaluation, laboratory and field testing, economic analysis, well field development, product recovery, and environmental factors. Environmental issues are considered together as the last topic because initial permitting and final restoration are closely related.

This section will concentrate on in situ leach mine development for uranium because this technology is well developed and commercialized. In situ copper mining remains in the developmental stage. In general, the same basic steps as those discussed here for uranium will need to be conducted for any commodity to determine if a mineral deposit can be developed as a commercial in situ leach mining operation.

PRELIMINARY SITE EVALUATION. Many of the following suggestions for site evaluation were selected from an unpublished report prepared for the USBM (Buma, 1977). The general principles regarding site evaluation discussed in the report remain valid.

The following two questions require consideration:

1. Are the physical conditions of the deposit, including depth, hydrostatic pressure, and permeability, amenable to in situ leach mining?

2. Is the mineralogy of the deposit amenable chemically to a practical leaching system?

Physical Conditions—In general, uranium deposits amenable to in situ leach mining are associated with relatively shallow sandstone aquifers (less than 830 ft or 250 m subsurface) confined by low-permeability strata. Close proximity to faults could make control of leaching solutions difficult. From an environmental standpoint it is helpful if the deposit is in a well-confined aquifer and not in the same aquifer as municipal water supplies. Locating and plugging all old drillholes can be difficult. These drillholes must be plugged to prevent leach solution from flowing to overlying or underlying aquifers.

Upper and lower limits of acceptable permeability depend upon the depth of the aquifer. The flow depends upon the permeability and the pressure differential between the water in the wellbore and the water in the formation. The maximum allowable injection pressure is just less than the fracture pressure of the formation, which depends upon the depth as well as rock type.

The minimum permeability at most successful in situ uranium mines is 300 millidarcies. However, more important than absolute permeabilities are relative permeabilities of the mineralized and nonmineralized sandstone hosts. The relative permeabilities of different zones can be inferred from grain-size analysis. Zones having larger grain sizes generally have higher permeabilities. Sorting of the sands is also important. Better sorted sands have higher permeability than more poorly sorted sands of comparable size. The permeability of cores from different zones can also be measured. If higher grade correlates with higher permeability, then generally solutions can be controlled successfully. If the inverse is true, contacting the mineralization adequately with leaching solution may be difficult.

Reliable values of permeability or transmissivity are difficult to obtain. The best figures are obtained from draw-down tests performed in the field. These tests, however, yield an average result for the entire perforated interval. Spinner surveys can provide a flow profile for a well, but these are subject to error with low flow rates.

Mineralogy—Most uranium deposits suitable for in situ leach mining are of the sandstone roll-type. Roll-type denotes deposits in which uranium has precipitated from solution in ground water along an oxidation-reduction front (Larson, 1978; Langen and Kidwell, 1974), as shown in Fig. 15.3.3.4. Uranium was transported to its final location by the natural movement of oxygenated groundwater. Deposition occurred when ground water containing oxidized uranium as $[UO_2(CO_3)_3]^{-4}$ encountered a reducing environment where U^{+6} was reduced to U^{+4}. This caused precipitation from solution to form UO_2 as uraninite or pitchblende at the redox interface. The uranium mineral coffinite is also common, containing uranium in both the +4 and +6 valence states. Uraninite, pitchblende, and coffinite leach readily. If the reducing environment is disturbed, the uranium may be remobilized and form other uranium minerals, some of which may not be leachable. These include uranyl phosphate $[Ca(UO_2)_2P_2O_8] \cdot 8H_2O]$ and vanadates.

While the type of mineralization is certainly important, it is the distribution of the ore minerals within the host rock that is crucial. This will determine if the mineralization is sufficiently accessible to the lixiviant to permit economic in situ leach mining. The majority of ore mineralization needs to be located in or near the natural flow channels. Mineralization that is disseminated throughout the host rock matrix will not be accessible to the lixiviant within an economic time frame.

LABORATORY AND FIELD TESTING. Both laboratory and field testing must be conducted in order to determine the amenability of a deposit to in situ leach mining. Key questions that need to be answered are:

1. Is the ore mineralization amenable to selective dissolution within an economic time frame?

2. Can sufficient quantities of mineralization be contacted by the lixiviant to ensure the economic viability of the operation?

3. Can sufficient lixiviant flow be achieved in a well pattern with large enough well spacing to be economic?

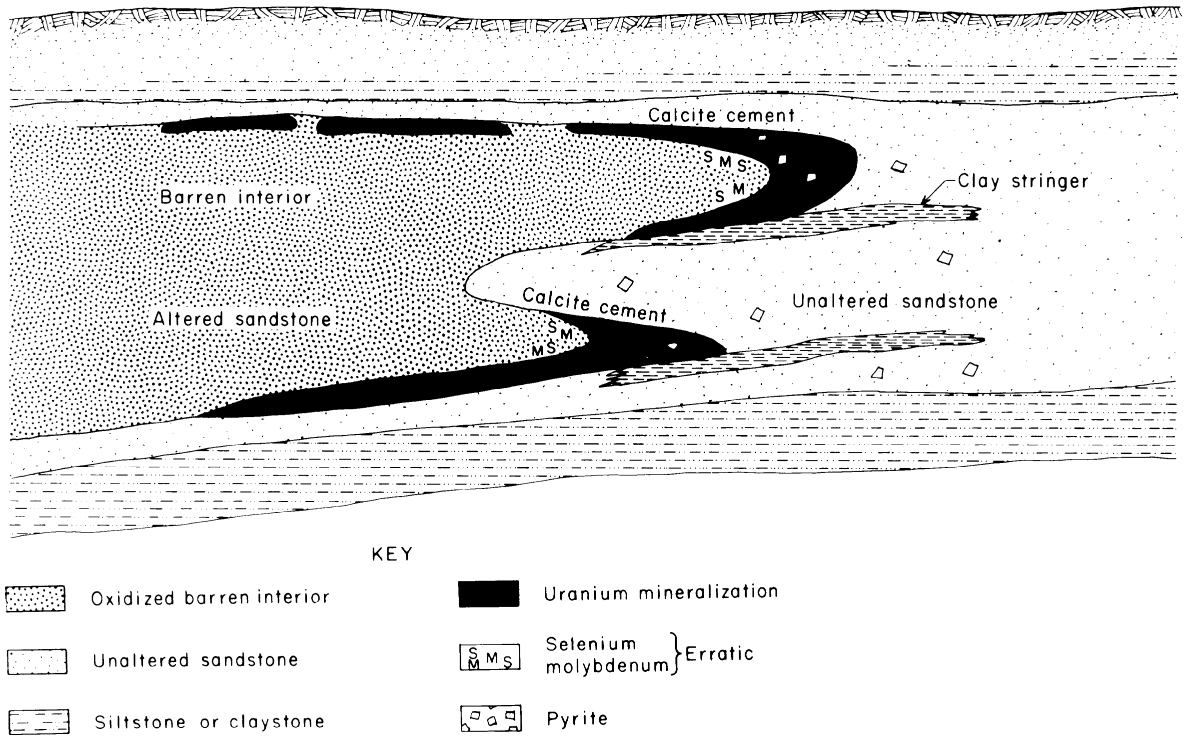

Fig. 15.3.3.4. Uranium roll-front deposit.

4. What is the preferred direction of fluid flow in the ore zone, and how will it affect sweep efficiency?

5. Can lixiviant be contained and recovered?

Laboratory testing can provide answers to the first two questions, and field testing can answer the others.

Solution Chemistry—Leach solution mobilizes uranium by reversing the chemically reducing conditions that originally led to deposition. It consists of ground water, an oxidant, and an anionic complexing agent that is usually carbonate. Using ground water from the formation minimizes swelling, precipitation, and other formation damage. The oxidant converts uranium from the +4 (reduced) to the +6 (oxidized) valence state, permitting its selective mobilization as an anionic complex. The ore-lixiviant systems for uranium are detailed in 15.2.2.3.

Many side reactions can occur between the leach solution and gangue minerals, such as calcite, clays, feldspars, zeolites, pyrites, and carbonaceous materials. These reactions can alter leach solution composition and/or mobilize contaminants, thereby impeding both uranium recovery and aquifer restoration. For example, oxidation of pyrite can produce sulfuric acid. Vanadium can also be mobilized and is recovered as a byproduct in some uranium operations.

Laboratory Testing of Leach Solutions—The two common types of laboratory tests are batch leach (sometimes called agitation leach) and column leach. Batch leach tests consist of agitating the ore and leach solution in a sealed container. Column leach tests consist of passing leach solution through a column packed with ore.

Much of the following information is from a USBM report (Tweeton and Peterson, 1981) describing methods for selecting leach solutions. Another reference, useful because of the similarities between milling and in situ leach mining, is Merrit's comprehensive book (1971) on chemical reactions in conventional uranium milling.

Sample Collection, Preservation, and Preparation—Obtaining meaningful results from batch or column tests requires care in obtaining, storing, and handling ore samples. When collecting the core sample, all drilling mud must be removed. The drilling mud can change the composition, Eh, and pH of the solution in contact with the core material. If the ore is inadvertently oxidized before being leached, then the measured oxidizer consumption will be too low. The degree of preoxidation can be estimated by conducting a leaching test under nitrogen. To avoid preoxidation, the ore can be sealed in airtight containers or frozen in dry ice soon after coring. If the ore is not preoxidized, the results will provide an upper limit to the oxidizer consumption because the oxidizing reactions will be more complete in the laboratory than in the field. Ore samples for permeability testing should not be allowed to dry. They should be preserved in sealed containers with water from the same hole or the moist samples may be frozen for shipping. Quick-freezing on site will retard chemical reactions that alter the sample, but may damage the clays.

Laboratory leaching solutions should be made from formation water, not from distilled water. A sample of formation water for use in the leaching tests must be collected and analyzed. Before collecting formation water, the well should be pumped until pH, conductivity, and Eh have stabilized and the water runs clear. One sample should be acidified immediately after collection and used for cation analysis. An unacidified sample should be kept for anion analysis. If precipitate forms before the sample can be analyzed, the sample should be shaken to mix the precipitate thoroughly and the precipitate analyzed along with the water.

To obtain the most reliable measurements of formation geochemical conditions, down-hole measurements of Eh, pH, conductivity, dissolved oxygen, and temperature should be made at the horizon from which the core sample was collected. Ideally, the water should be collected with a pressure sampler and the dissolved gas composition measured.

Laboratory studies should be conducted on homogenized ore samples. Meaningful comparisons of leach solutions require similar ore samples, but cores vary considerably in permeability and uranium content. Clay and calcite lenses in cores could have an overwhelming influence on core permeability while having very little effect on in situ horizontal permeability. The ore should be disaggregated gently to avoid breaking grains and thoroughly blended. Most sandstone uranium ore can be disaggregated easily. Some consultants advise removing clay lenses from the core prior to testing to reduce their effect on permeability. Their removal can also help avoid misleading results regarding the leachability of uranium adsorbed on clay or organic substrates.

Batch Leaching—Although batch leaching tests do not simulate down-hole conditions, they can indicate the relative rate and amount of uranium extraction and the relative consumption of oxidant as a function of leach solution composition. Variables include pH (e.g., 7, 8, and 9), carbonate or bicarbonate concentration (e.g., 1, 3, and 5 g/L), and oxidizer concentration (e.g., equivalent to 0.1, 0.3, and 0.5 g/L oxygen). The ore sample must be large enough to allow solution samples to be withdrawn without unduly disturbing the geochemistry and to provide sufficient dissolved uranium for analysis. Typical ore sample sizes are 100 to 500 g, with a liquid-solid volume ratio of three.

The measured parameters are usually pH, Eh, and concentrations of uranium, carbonate-bicarbonate, oxidizer, and site-specific elements such as vanadium. The Eh measurements do not necessarily represent in situ values, but changes in the Eh can indicate changes in the oxidation state of the uranium. Typical sampling times are 0, 1, 4, 8, 12, and 24 hours after the start of the test and once per day until equilibrium is reached, usually in 3 days or less. The uranium in the ore should be measured before and after leaching and a material balance made to check the validity of the measurements. Material balances must be calculated for proper control. Accounting for carbonate may be difficult because of losses in the production of CO,, but all cations should balance within reasonable limits.

Fresh solution should be added to the slurry after extracting the sample to maintain the liquid to solid ratio. Before extracting the sample, pH, Eh, and conductivity should be checked and recorded. The solution replacement for the sample should contain an adequate amount of fresh reagent to allow the leaching process to continue. Solution increments should not be so large as to overpower the system and mask other important effects.

Measuring the consumption of oxygen requires sealed experimental equipment and a method for withdrawing samples without losing oxygen. Some experimenters favor a chlorate oxidizer to avoid the need for pressurization. However, the consumption of chlorate and oxygen are not necessarily equal because their oxidizing powers are not identical.

Column Leaching—Column leaching tests simulate field conditions more closely than batch tests, but caution is still necessary when extrapolating from laboratory to field. The contact between ore and leach solution is more complete than in actual in situ leach mining, so the laboratory extraction of uranium is an upper limit for field results.

Permeability losses can be measured with column leach tests. Permeability is affected by ion-exchange reactions in clays, especially montmorillonites. Montmorillonite has exchangeable cations and a variable number of water molecules between the layers. Swelling can occur if the predominant cation in the leach solution has a larger ionic radius or lower valence state than those within the clay. A common example is the exchange of Na^+ for Ca^{++} in the montmorillonite. The Na^+ cation has a larger ionic radius than the Ca^{++} cation. Also two Na^+ cations are needed to exchange for one Ca^{++} cation to maintain electrical neutrality. This exchange increases the volume of the clay, causing swelling and decreasing permeability. Thus column tests are especially important in determining whether sodium carbonate leach solution will cause excessive clay swelling and loss of permeability.

Swelling can also occur if a solution having a lower ionic strength is introduced, as the clay admits more water molecules between layers. This problem can occur during laboratory leaching tests if the leach solution is made with distilled water instead of formation water.

A vertical column with leach solution entering at the bottom maximizes the rate at which air is swept out of the core. Horizontal tubes present the risk that ore will slump slightly, thereby creating a high permeability channel between the top of the ore and the inside of the column. Ideally, the column would be pressurized to simulate down-hole conditions and allow oxygen to be used. Using sodium chlorate in column leaching tests could reduce permeability.

In principle, the most reliable results will be obtained with the widest and longest columns. In practice, the amount of core is usually limited and the number of tests is large, so columns must be rather small. A column 2 in. (50 mm) in diameter and 3 ft (1 m) long appears to be a good compromise unless chromatographic effects are of special interest. Then a 1-in. (25-mm) diameter sectional tube with a total length of 10 to 17 ft (3 to 5 m) would provide more information. A series of vertical columns connected by flexible tubes is convenient for studying chromatographic effects. With small cells, the ore must press tightly against the sides of the cell to avoid large wall effects. For testing oxidizer consumption and response to oxygen under down-hole conditions, a small pressure cell can be used. Hassler pressurized cells 12 in. (300 mm) long gave good results at the University of Texas (Schechter, Lake, and Guilinger, 1984). Hassler cells can also be used for measuring the effect of various leach solutions on permeability, allowing careful control of experimental conditions.

Typical variables, measured parameters, and sampling times are similar to those for batch tests. Measuring permeability by measuring the pressure difference across the packed column at a constant flow rate is generally preferred over using a constant pressure. The velocity of the leach solution should be similar to that in the field, often about 10 ft/day (3 m/day). Excess flow rate or pulsing can lead to channeling, especially with horizontal columns.

Well Field Patterns—A pilot-scale field test is performed before starting commercial operation to evaluate leach solution behavior, to test well construction and completion techniques, and to demonstrate restoration procedures. These tests can be flow-through, where injection and production wells are separate, or push-pull (“huff-and-puff ”), where a well serves first for injection and then for production. Most commercial operations are flow-through. Push-pull may be used if hydrologic communication between wells is poor.

Experts disagree as to the value of push-pull tests. Some believe that these tests provide a reliable and relatively economical method of evaluating the leach solution under field conditions. For example, one push-pull well and two observation wells provided sufficient data for field evaluation of an experimental leaching process (Tweeton and Lake, 1986). The well pattern was an L, with the injection-production well at the corner. However, the results of flow-through tests can be quite different from those of push-pull tests. In unpublished flow-through tests at one site, increases in uranium concentration and Eh lagged behind the changes in the other components. Over three pore

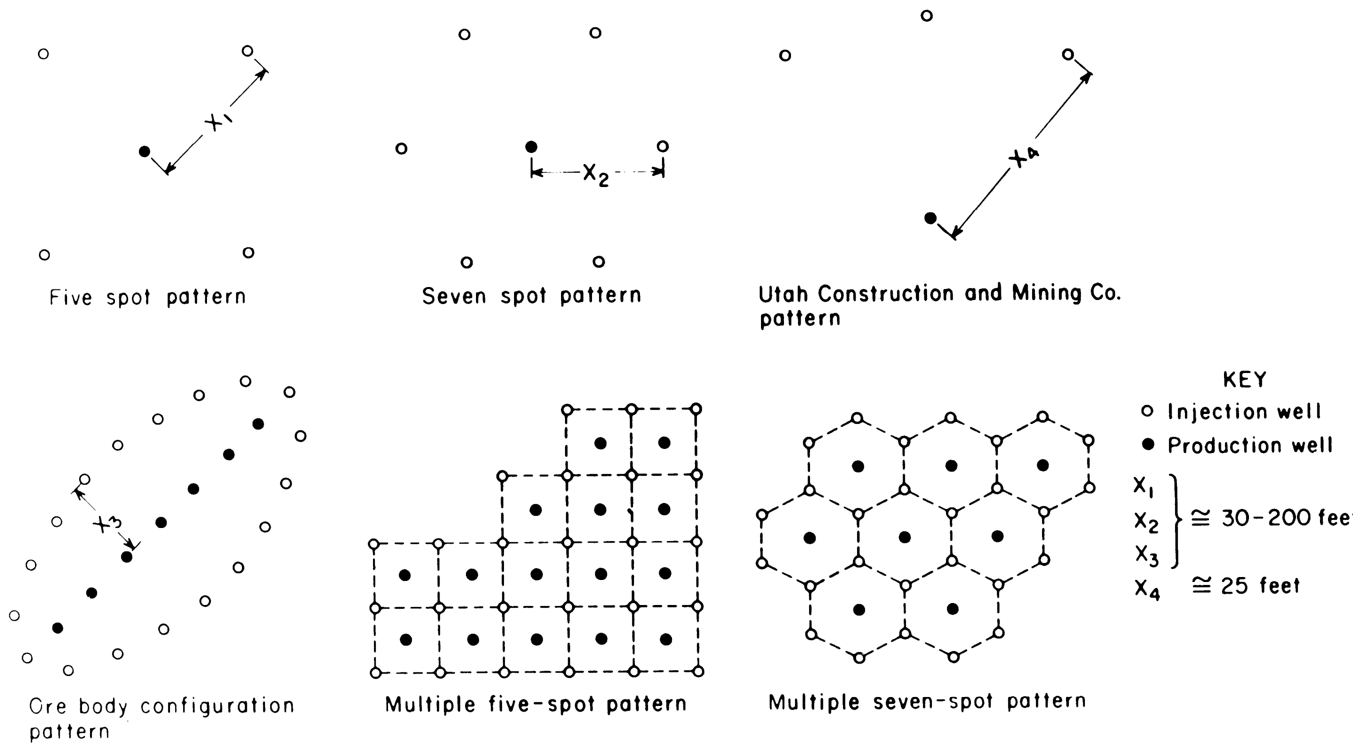

Fig. 15.3.3.5. Well field patterns. Conversion factor: 1 ft = 0.3048 m.

volumes of leach solution were injected before the uranium concentration and Eh rose, but that lag was not observed in a previous push-pull test at the same site. Therefore, the push-pull test resulted in an overly optimistic prediction about the ease of leaching. Conducting a flow-through pilot test before beginning a flow-through commercial operation is prudent if a push-pull test has been used for a preliminary evaluation.

Various well field patterns are shown in Fig. 15.3.3.5 (Larson, 1978). The most common configuration is a five-spot, either single pattern for tests or multiple pattern for commercial operation. A five-spot test provides directional permeability information, a reasonable sampling of the ore body, and closure for controlling solution flow. Capability for reversing the flow is sometimes helpful in restoring permeability. The usual flow pattern is to inject at the corners and produce at the center because there is generally more fluid resistance to injection than to production. This pattern also minimizes dilution. A disadvantage is that it is not possible to distinguish the arrival of solution via different paths. The distances between the injectors and producers can be adjusted to the permeability conditions. A 40- to 70-ft (12- to 22-m) span is typical. Experimentation with well spacing can be considered as part of development for production.

Advantages of a small field are that a test can be completed in less time, and the response time to operational changes is short. The smallest practical well spacing is about 20 ft (6 m) from injection to recovery well. This spacing is wide enough to allow drilling of post-test core holes between injection and recovery wells without danger of intersecting the wells.

Formation Clogging Causes and Cures—Clogging of the formation tends to be most severe near injection or production wells. Clogging of the formation near an injection well can occur due to poor well completion practices. Clogging will also occur if the lixiviant is incompatible with the formation. An example is clay swelling that occurs with a sodium carbonate-bicarbonate lixiviant used in a formation that is high in clay when the pre-leach groundwater is low in sodium. Usually, the incompatibility can be identified in laboratory column leaching tests. To prevent blockage of the formation by oxygen bubbles, the down-hole solubility of oxygen must not be exceeded.

Clogging of the formation near a production well can be caused by gas phase blockage or precipitation. The pressure near a production well is lower than in the rest of the formation, so some of the carbon dioxide undergoes exsolution. This can raise the pH and precipitate calcite. If the ground water is saturated with respect to calcium, the pH should be maintained at about the natural pH to avoid precipitation of calcite in flow channels. The solution pH should not be allowed to rise above 9.2, as this is the point at which silica is mobilized. Silica can form gels which block the formation and can poison the ion exchange resin in the extraction plant.

Reprecipitation of uranium is undesirable even if it does not cause serious clogging. It can occur if the strength of the leach solution (pH, Eh, or concentration) changes excessively from injection to production well. This problem can be minimized by increasing the strength of the leach solution gradually as leaching begins and by selecting a leach solution compatible with the formation. Reducing the well spacing is an expensive cure.

Hydrology—Darcy's Law and the equation for continuity of flow in an elementary volume of rock are the fundamental equations used to derive the partial differential equation that governs the unsteady-state flow of leach solution between injection and recovery wells. The analysis assumes a homogeneous isotropic aquifer setting. Darcy's Law states that the flow rate through a porous medium is proportional to the head loss and inversely proportional to the length of the flow path. The constant of proportionality in Darcy's Law (assuming a unit cross-sectional area of flow) is the *permeability*. *Storativity* is the volume of fluid that the rock mass releases from or takes into storage with a change in head (Walton, 1970).

Values for permeability and storativity in a saturated ore deposit may be obtained in situ from well tests or laboratory core leaching experiments. Field measurements of saturated permeability obtained from short-term well tests in fractured hard-rock deposits typically range from 0.1 to 10 millidarcys. For the most part these represent measures of effective fracture permeability rather than intergranular permeability. Intergranular permeability is commonly 1 to 4 orders of magnitude lower (Erskine, 1987). Typical values of storage coefficients for unfractured hard rock are 10^{-9} to 10^{-11}.

Insuring adequate flow rates between wells, and controlling the distribution of injected leach solution within the ore deposit are part of the in situ hydrologic design process. A number of important in situ design decisions must be made by site operators in order to meet these objectives.

In low permeability settings the design alternatives that are available to insure adequate well flow rates include imposing higher and lower head conditions on injection and recovery wells, choosing injection and recovery well locations closer together, and/or increasing the diameter of wells.

In saturated deposits, control of the flow and distribution of injected leach solution is accomplished by choosing wellhead conditions that ensure a net withdrawal of leach solution and ground water in the vicinity of the wells. Ground water injection wells, however, are sometimes used to create artificial hydraulic barriers at the boundaries of the site.

Leaching a low-permeability or poorly confined mineralized zone is difficult. An example is the mineralization tied up in the trailing edges on the oxidized side of a roll front. The unmineralized core of the oxidized side of a roll front is more permeable than the mineralized limbs. Leaching under such circumstances is possible, but the mode of operation must allow solutions to diffuse into the less permeable horizons and contact the mineralization rather than simply flowing through channels. Pressurizing the more permeable zones above and below the mineralized zone by injecting ground water can help to confine the leach solution to the mineralized zone (Schmidt and Ahlness, 1989).

Design decisions such as these must be made in the context of a complex hydrologic setting involving heterogeneities in permeability, flow in fractures, free saturated surfaces, partially saturated conditions, and mine void spaces. Application of an engineering hydrology model can greatly enhance understanding of this complex setting and make the decision process far easier (Schmidt, Salovich, and Jagusch, 1987).

Analytical or numerical hydrology models that are powerful enough to incorporate many of the complexities of an in situ field setting and at the same time permit easy trial-and-error manipulation of site conditions and design parameters are especially useful as engineering models. Most sites are too complex, and too little field data are generally available to permit precise prediction of field outcomes based on computer modeling. However, hydrologic modeling is extremely useful for exploring the effect of various parameters and testing their influence on desired outcomes (Strack, 1989).

At present, commercial copper in situ leach mining operations are being conducted in disturbed (broken) deposits that have been previously developed and exploited by prior conventional block caving or other underground mining methods. These settings offer a number of advantages over mining in undeveloped settings. Ore permeability is often greater as a result of mining-induced fracturing or rubblization of the deposit. Underground workings offer preferential access to a deposit and save on drilling costs. Underground mine void spaces are available as solution collection areas, and a developed deposit is generally better characterized geologically and mineralogically than an undeveloped deposit. However, prior mine development also contributes significantly to the hydrologic complexity of a leach site (Swineford, 1987).

While an ore body located in an undisturbed setting may be saturated and have a significant positive pore pressure, unsaturated conditions resulting from prior mine dewatering activity may exist initially in a partially developed deposit. Mining induced fracturing or rubblization of an ore deposit, and the presence of surface or underground mine void spaces results in a complex flow setting, making it difficult to isolate and assess the influence of individual hydrologic parameters.

The existence of unsaturated (negative pore pressure) conditions in a deposit prior to leaching can be a significant factor in controlling the distribution of injected leach solution, since practically speaking, the minimum head condition that can be imposed on a recovery well bore is zero. For leaching operations conducted under conditions of partial saturation, existing underground mine drifts rather than recovery wells may be the most effective collection mechanism (Schmidt, 1988).

Fracture heterogeneities can significantly affect the flow rates between individual wells, and the distribution of leach solution in the ore body. However, some of the most useful hydrologic information regarding fracture heterogeneities is acquired only as leaching operations are being conducted. In order to use this information fully, it is desirable to incorporate into a leach site design, as much flexibility as possible. The number of wells available for injection and recovery, their locations in the well pattern, and the head condition prescribed for each should be factors that can be independently manipulated in response to flow conditions that develop during leaching.

Injecting leach solution at pressures high enough to dilate existing rock fractures in the vicinity of the injection wells is a permeability enhancement technique that is currently being tested at a leach site. Preliminary results indicate that this technique can be an effective means of increasing solution injection and recovery rates. Questions remain, however, about the difficulty of maintaining fracture permeability at lower injection pressures, the influence of ambient pore pressure conditions on the pattern of dilated fractures, and most importantly the tendency of leach solution to flow preferentially between injection and recovery wells via a small number of highly conductive fracture flow paths (i.e., short circuits) as opposed to a more uniform distribution of flow in less-permeable, less-fractured rock.

A possible solution to this problem, which takes advantage of built-in flexibility in well specifications, involves converting injection/recovery well pairs that exhibit short-circuiting behavior to either paired injection or paired recovery wells. Such a well specification takes advantage of the highly conductive fracture flow path linking the two wells in order to distribute solution into lower permeability rock surrounding the fracture short circuit.

Data Collection—A five-spot field test will provide the most useful information for an economic evaluation, so care should be taken to collect an adequate number of samples during the test. As a minimum, four samples per day should be collected and analyzed for uranium, vanadium, molybdenum, selenium, arsenic, calcium, and CO_3. A spectrographic analysis of the pregnant solution should be obtained at the peak grade of uranium. The leaching conditions can be monitored by taking periodic in situ measurements of Eh, pH, conductivity, dissolved oxygen, and temperature.

To determine uranium recovery, a number of core holes should be placed within the test well field prior to initiating the test in order to obtain assay data. Because of the extreme variability of uranium deposits, the more pre-test and post-test core holes, the more reliable and consistent will be the results. As a minimum, post-test core holes should be placed midway between injection and recovery wells. Some post-test core holes should be placed outside the five-spot pattern to ascertain the extent of solution migration outside the well field. Statistical methods must be applied to data from field tests to get meaningful results.

ECONOMIC ANALYSIS. In situ uranium mining companies have lowered their costs of production since 1980. Informed sources in the industry estimate that direct production costs are as low as \$10 to \$12/lb (\$22 to \$26/kg) of uranium in some favorable deposits. These costs do not include royalties, taxes, or exploration. The cost reductions have been achieved through reducing the number of workers per unit of production, and by careful attention to technical procedures such as well field planning. At many sites, such cost reductions were not possible, and profitable production could continue only as long as higher-priced contracts with uranium buyers remained in effect. An example of the low labor costs possible with in situ leach mining was provided by Everest Minerals Corp. (Stover, 1989). During 1988, their Highland Uranium Project in Wyoming produced 2500 lb (1100 kg) of uranium/day while employing 55 people on site.

Economic evaluation models are available through private consulting companies in areas where in situ uranium mining is conducted. These models are mature, having been compared with actual costs. The models are proprietary, as are the costs for specific mines.

A nonproprietary cost model was released by the USBM (Toth and Annett, 1981; Toth and Larson, 1981, 1982) and can serve as a useful introduction to modeling of costs. The report presented the results of an assessment of uranium in situ mining costs through the application of process engineering and discounted cash flow analysis procedures. The model will generate mine life capital and operating costs as well as solve for economic production costs per pound (kilogram) of uranium. Conversely, the rate of return may be determined subject to a known selling price.

The databases of the cost model were designed to reflect variations in Texas and Wyoming site applications. Base case calculations gave the cost for producing uranium as over \$20/lb (\$44/kg) in both Texas and Wyoming. The biggest single item of expense was annual well field replacement costs, which were about \$5 to \$8/lb (\$11 to \$18/kg) of uranium for the base cases. Other direct costs were \$6 to \$8/lb (\$13 to \$18/kg). The results were most sensitive to pattern spacing and solution grade.

Perhaps the most important single factor in evaluating the economic viability of a potential in situ uranium mine is the grade-thickness product of the uranium reserve. The cutoff grade for in situ leach mining is lower than for conventional mining. Companies have used 0.03% as a cutoff, though the cutoff will vary with site conditions such as depth. Typical grade-thickness cutoffs range from 0.3 to 0.6%-ft (0.09 to 0.18%-m). In determining grade-thickness from logs, the nonleachable uranium associated with clays, shales, and coal must be eliminated. The cutoff depends not only on site-specific conditions but also on the price of uranium.

The advantages of the economic flexibility of in situ leach mining were discussed by Pool (1989). The flexibility occurs because fixed costs for in situ mining are a lower percentage of total costs than they are for underground or open pit mining, so production can more easily be changed to respond to price changes. Simulations with the USBM model indicated fixed costs are in the range of 35% of the total cost for a typical in situ leach mining operation. In contrast, fixed costs for underground mines are in the range of 50 to 65% of the total cost of full-scale production. For open pit mines, a 10:1 stripping ratio might easily put the fixed cost of an open pit at 90% of the total cost. Pool concluded that this higher degree of economic flexibility will stimulate the growth of in situ leach mining and provide additional insulation from fast-changing prices.

There are no economic in situ copper mining data available since there have been no commercial operations in undisturbed ore. However, a USBM publication contains a computerized cost model that can be used to estimate the economics of in situ leach mining for any specific oxide copper deposit (Davidson et al., 1988). The cost model develops detailed capital and operating costs and utilizes a discounted cash flow/rate of return (DCF/ROR) analysis to calculate either a pretax rate of return for a specified selling price of copper, or a pretax selling price of copper for a specified rate of return.

WELL FIELD DEVELOPMENT.—Many of the following recommendations are adapted from a USBM contract report (Shuck and Brooke, 1982) that describes in detail designing, drilling, grouting, and developing wells for in situ uranium mining. Where necessary, the recommendations are updated to reflect changes in practices.

Pump and Casing Selection—Pump specification and selection will depend on the characteristics of a particular site and the leach chemistry adopted. Because of the large number of pumps used, the total pumping cost should be used as the basis for their selection. The total pumping cost includes power costs, maintenance costs, and replacement costs. Excessive throttling or switching of oversized pumps should be avoided. The submersible pump must be hung on a pipe or tubing string of adequate strength to resist numerous starting-torque cycles. A separate safety line provides extra protection. The ability to withstand the corrosiveness of the leach solution is vital. Many operators have chosen stainless steel pumps, despite their higher price, to obtain higher reliability.

Casing must also withstand the corrosive effects of leach solution. PVC casing is used much more than fiberglass. Fiberglass is used only where it is needed because of unusually great depth. At normal depths (800 ft or 240 m), PVC is preferred because it costs less than fiberglass. If solvent welded PVC is used, proper procedures for joint make-up, casing emplacement, and grouting should be implemented and monitored to insure adequate casing integrity. Many operators use screws to hold the solvent welded joint while the solvent is drying. A yellow PVC casing which is marketed for mining use is available. It uses spline joints, so the problems associated with solvent welding are eliminated. However, it is more expensive than standard PVC.

Wherever practical, the use of PVC or fiberglass casing accessories is recommended over the use of steel accessories because of the reduced potential for casing damage during any subsequent work-over operations. Casing accessory specification will depend on the type of well completion.

Drilling—Because of cost and availability, hydraulic-rotary equipment with conventional circulation is recommended for in situ leach mining applications. The drill rig should have a rotating table which allows setting and grouting of the casing while still set up on the hole. The mud pump should be of sufficient size to maintain turbulent flow in the ascending fluid (i.e., a fluid velocity of 60 to 100 fpm or 0.3 to 0.5 m/s). Mud pits should be sized to permit maintenance of optimum drilling fluid properties for the particular site. This generally requires a pit volume at least 2 1/2 times the anticipated drillhole volume.

To obtain straight holes, it is important that sufficient weight and stiffness be incorporated into the drill string immediately above the bit. This is accomplished by means of drill collars to

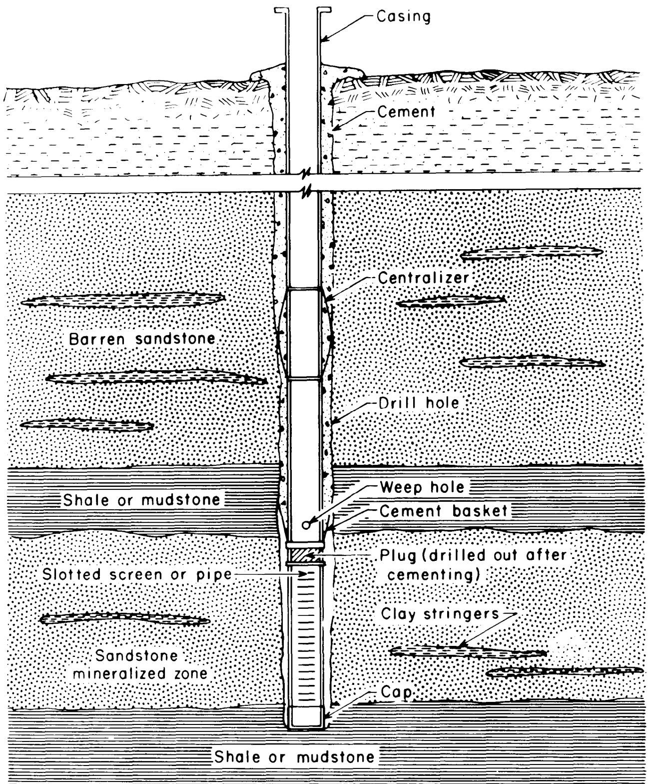

Fig. 15.3.3.6. Well completion using screen and cement basket.

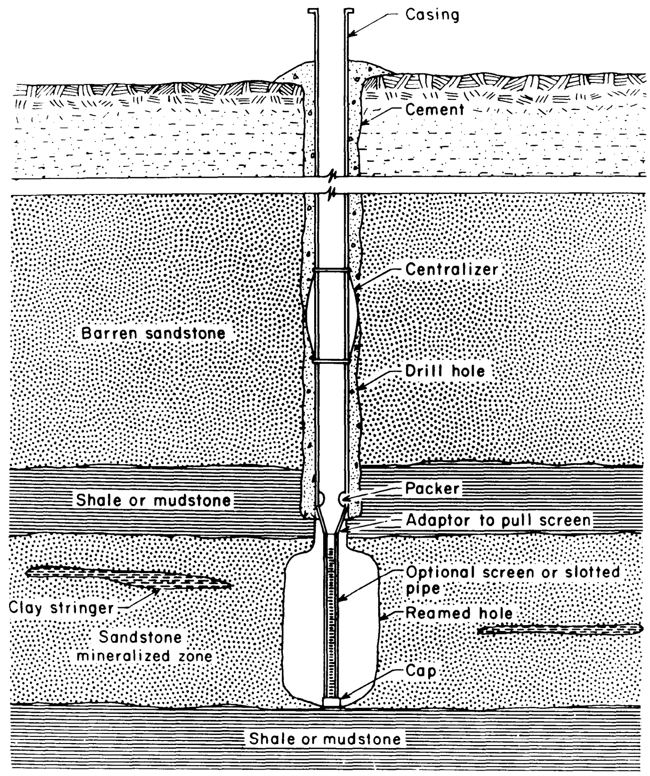

Fig. 15.3.3.7. Well completion using underreaming.

maintain the drill string in tension and obtain optimum drilling rates without pull down on the drill string. The type of bit(s) used to drill the hole will depend on the particular formation characteristics, but the bit should have a diameter at least 2 in. (50 mm) greater than the maximum external diameter of a casing joint. Drilling fluid selection should be matched to the well drilling and completing sequence to permit tight control of its characteristics when drilling the completion interval. To minimize the potential for damaging the formation while drilling, bottom hole pressures should be maintained well below the formation fracture pressure.

It is vital to avoid formation damage by drilling fluids. Wherever practical air-drilling of the completion interval is recommended to minimize damage to the exposed formation face. A chemically degradable polymer-based fluid is recommended where air drilling proves impractical. The four following principal fluid characteristics should be maintained while drilling the completion interval: recovered fluid density of 8.5 to 8.7 lb/gal (1.02 to 1.04 kg/L), Marsh funnel viscosity about 35 sec/qt (37 sec/L), filtration rate of 5 to 10 mL/half hour at 100 psi (690 kPa) pressure difference, and the minimum possible gel strength.

Well Completion—Two well completion methods are shown in Figs. 15.3.3.6 and 15.3.3.7. The method of Fig. 15.3.3.6 employs a screen or a slotted or perforated casing in the mineralized zone. Restricting the well completion interval to the mineralized zone is recommended but does not insure confinement of the leach solution to this zone. Most operators use screens attached to the casing rather than perforating or slotting casing. The screens provide better sand control without hampering well efficiency.

The method of Fig. 15.3.3.7 employs underreaming to enlarge the well bore in the mineralized zone to facilitate injection. A screen may be used, or, in competent ore, the underreamed portion of the hole may be left open.

The cement used to make the grout must be compatible with both the local groundwater and the leach solution composition throughout the course of the project. The grout should also be formulated to yield the required strength vs. time characteristics at the lowest practical slurry density. When the well design involves running casing through the target ore zone and either an under-reamed or perforated completion interval, the grout formulation should also have the lowest practical filtration loss. The need for other additives will depend on specific formation characteristics and ambient conditions. A minimum initial clearance of 1% of its calculated length should be allowed between the casing and bottom of hole to prevent contact during grout emplacement.

To assure the quality of grout seal required for in situ leach mining wells, through-casing emplacement with wiper plugs is recommended. Through-casing grouting involves injecting grout into the casing and allowing it to flow into the annulus between the casing and formation through holes (weep holes) near the bottom of the casing, as shown in Fig 15.3.3.7. A plug, generally plywood, below the weepholes prevents grout from falling into the inside of the well screen. This plug is drilled out along with the grout "heel" that remains above the plug. The cement basket prevents the grout from falling into the annulus outside the screen. The annulus is thus filled from near the bottom to the top.

A pregrout flush consisting of grout thinned to a density somewhat greater than the drilling fluid is recommended over water. It should be circulated at a flow rate sufficient to obtain turbulent flow in the annulus and continued for 5 to 10 minutes before grout emplacement is begun. The grout slurry volume used should be sufficient to obtain a return to the surface with the desired set characteristics. Once begun, grout emplacement should continue smoothly and continuously at a bottomhole pressure kept well below the formation fracture pressure. The volume of displacement fluid used should insure that a grout heel of at least 10 ft (3 m) is left within the casing when wiper plugs are not employed. To maintain grout displacement, it is essential that the grouting wellhead be leak-tight at the required shut-in pressure. Use of a weighted fluid which renders the casing neutrally buoyant is recommended where possible. As is true for casing accessories, PVC or fiberglass grouting accessories should be used down-hole wherever practical to minimize subsequent work-over damage.

Well Development and Stimulation—Current regulations require that injection-well integrity be verified prior to use. Pressure testing was once required in Wyoming, but resistivity logging is now accepted for verifying mechanical integrity in both Wyoming and Texas. If variable pumping arrangements are anticipated during operations, it may be preferable to test all wells rather than only injection wells.

Large leaks can be caused by well completion tools gouging holes in PVC casing. For example, leaks have been caused by underreamers when the tool has been pushed down or pulled up with the blade not fully retracted. Sand can interfere with retracting the blade. Damage can also be caused when any tool that fits tightly is forced past casing that is somewhat curved because of hole deviation.

Well development should begin as soon as practical after grouting is completed. The least vigorous method which proves effective should be used. The combination of chemical treatment to break down drilling fluid and grout residues followed by air lifting to remove the associated residues appears to be the most generally applicable and cost-effective development method. Chemical reagents used for this purpose should be limited to those with minimal impact on groundwater composition and related water processing for mineral recovery and groundwater restoration. During both chemical treatment and airlifting, caution should be exercised to prevent thermal damage to the casing as a result of either the heat of reaction or heat of compression. Only after such preliminary development should any of the more vigorous methods be used to complete well development.

Maintaining well performance requires the timely identification and correction of the causes for declining performance. When well stimulation is required, a combination of chemical treatment and pumping for reagent circulation and recovery is recommended. The type of pumping (cyclic followed by recovery vs. flow-through) employed in a given situation should be based on complete evaluation of the material to be removed, the reagents to be used, and the reaction products to be recovered. As in the case of well development, the chemical reagents should be carefully selected and sparingly used. Clogging of production well screens or the nearby formation can result from fines migration. Periodic vigorous well development such as airlifting may be required.

PRODUCT RECOVERY. Once the mineral values are in solution, they must be recovered. However, recovery is independent of the source of the pregnant leach solution (PLS) for in situ leaching, heap and dump leaching, or a hydrometallurgical plant. The recovery methods for uranium are covered in Chapter 15.2, and methods for copper are addressed in both Chapter 15.2 and Chapter 25.4.

ENVIRONMENTAL FACTORS.

Uranium—In situ uranium mining offers significant environmental advantages over conventional uranium mining. Much less radioactive material is left aboveground at the end of mining. In situ leach mine workers are exposed to less radon gas than underground miners are. Radium is less soluble than uranium, so most of the radium is left underground during in situ leach mining. In situ leach mining consumes less groundwater than dewatering for conventional mining. However, in situ leach mining, like conventional mining, has the potential to pollute aquifers. Therefore, regulatory agencies require companies to submit plans for monitoring of leach solution excursions and restoration of groundwater quality as part of a permit application.

The states in which in situ uranium mining is being performed include Texas, Wyoming, and Nebraska. There is some stope leaching in New Mexico, and interest in testing in situ leach mining, but there are no active in situ leach mines in undisturbed ore. Texas is an agreement state with the US Nuclear Regulatory Commission, so permit applications need to be filed only with the state. Permit applications should be submitted to the Texas Water Commission in Austin, TX. Also a Radioactive Materials Handling License is required from the Texas Department of Health. Wyoming, Nebraska, and New Mexico are non-agreement states, so permit applications must be filed with the NRC and state agencies. Applications to the NRC should be filed with the Nuclear Regulatory Commission Uranium Recovery Field Office in Denver, CO. The Wyoming Department of Environmental Quality, Land Quality Division, in Cheyenne, WY, receives the Wyoming permit applications (Anon., 1986a). In Nebraska, applications should be filed with the Department of Environmental Control in Lincoln. If the proposed mining zone is a fresh water aquifer, then an aquifer exemption from the US Environmental Protection Agency (EPA) will be required.

In Wyoming, Texas, and Nebraska, monitoring requirements are not given in detail, but rather are stated in terms of assuring that excursions (escape of leach solution from the well field) will be detected before leaving the operator's property or contaminating an aquifer. The pattern of monitor wells required will depend on local conditions such as velocity of groundwater flow and the proximity to aquifers from which water is being drawn for other purposes. In general, monitor wells will be required to surround the mine for sampling the mine aquifer and other monitor wells will be required for sampling overlying and underlying aquifers.

As shown in Fig. 15.3.3.8 (Stover, 1989) the overlying and underlying monitor wells sample aquifers that could become contaminated if leach solution were to migrate through the horizontal confining layers. Such migration could occur through natural discontinuities, through unplugged abandoned wells, or along poorly grouted injection or production wells.

In the past, restoration of groundwater quality after in situ leach mining was a long and costly process at sites that had been leached with ammonium carbonate. Restoring the ammonium levels to the required low levels was difficult because of the affinity of clays for ammonium ions and because the limits were set very low. Recent restorations have been more successful and less costly because sodium carbonate or carbon dioxide leach solutions have been used. The usual procedure now is to flush the field with water that is recycled and cleaned with reverse osmosis. Some companies inject a reductant, but most do not. The restoration is still a significant part of the operation, and its cost should not be neglected. However, it is less difficult than it was in the past. Typical requirements for restoration are to restore to drinking water standards or to the average concentration in the field before injection. In some cases, detection limits are the limiting factor.

The following information concerning restorations in Texas was supplied through the courtesy of the Texas Water Commission in Austin. A permittee is required to restore the aquifer after completion of mining in accordance with 31 Texas Administrative code 331.107 (Knape, 1989). Included in each production area authorization is a restoration table that provides resto-

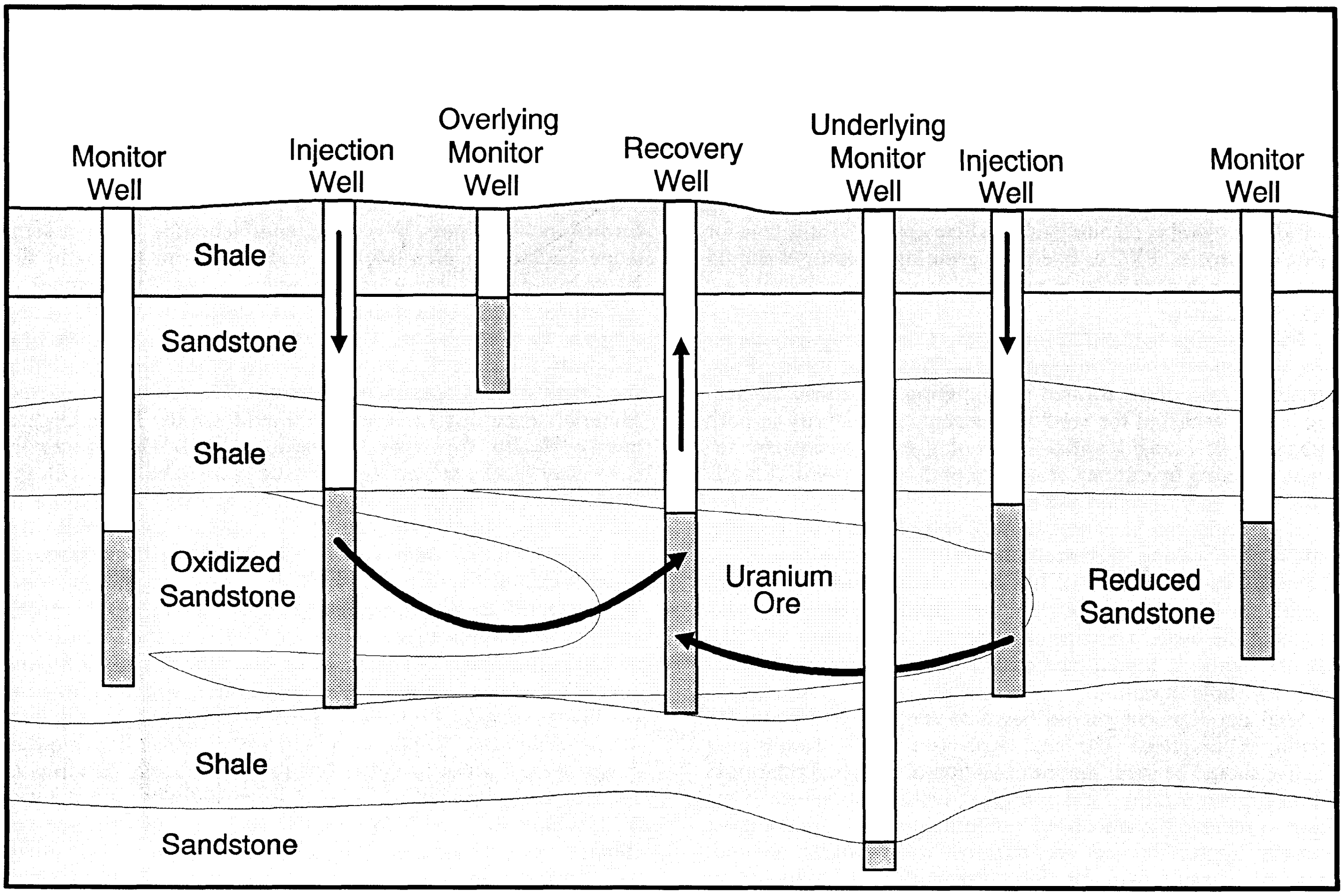

Fig. 15.3.3.8. Well field with overlying and underlying monitor wells.

ration values for 26 parameters. The 26 restoration parameters are calcium, magnesium, sodium, potassium, carbonate, bicarbonate, sulfate, chloride, fluoride, nitrate-N, silica, pH, total dissolved solids (TDS), conductivity, alkalinity, arsenic, cadmium, iron, lead, manganese, mercury, selenium, ammonia, uranium, molybdenum, and radium-226. The restoration values are based on analysis of water samples taken prior to mining. The radiometric parameters are set by the Texas Department of Health-Bureau of Radiation Control.

Required surface restoration basically consists of removing all pipelines, plugging and capping all well casings below grade, transporting contents of evaporation ponds to a licensed disposal site, filling in pits, and establishing appropriate revegetation. Financial security is also required for restoration and surface reclamation of the site. This financial assurance must be posted with the Texas Department of Health in accordance with a Memorandum of Understanding between the Texas Department of Water Resources and the Texas Department of Health.

Other environmental concerns, besides monitoring and achieving restoration, include the disposal of wastes generated during the mining and restoration. The following discussion is from Shuck (1979). Liquid wastes are generated from three principal sources: (1) well field over-production for the purpose of leach solution confinement, (2) process bleeds for the purpose of contaminant control, and (3) water treatment for the purpose of waste concentration and/or aquifer restoration. Liquid wastes are typically impounded for waste concentration and water reclamation via methods such as reverse osmosis. This approach minimizes consumptive water use during both production and aquifer restoration. Concentrated waste streams of approximately 15,000 ppm TDS are typically impounded and further concentrated by means of either natural or induced evaporation.

The principal sources of air emissions are the open surface of tanks and waste ponds and the product drying/packaging unit of the uranium recovery process. Generally, low TDS waste streams are utilized to scrub both particulate and gaseous emissions from the stack gases of the product drying/packaging unit. The spent scrub solution is impounded with other process wastes for further concentration via evaporation. Thus the principal source of untreated air emissions is the exposed surface of waste storage and concentration ponds.

Solid wastes result from the calcium control unit, waste concentration processes, filtered solids, laboratory wastes, and contaminated materials. These are typically impounded and maintained under a liquid seal to minimize atmospheric dispersion, pending permanent disposal.

Copper—Environmental issues specific to in situ copper mining are addressed through a permitting process administered by federal, state, and local governmental regulatory agencies. Each agency maintains jurisdiction over certain elements of the mining operation. Permitting involves preparation of comprehensive application documents by the mining company and subsequent document submittal to the appropriate agency. Authorization to proceed with the mining activity may be either granted or denied by the regulatory agency depending upon anticipated facility compliance with environmental protection requirements. Once permits are approved and the facility is constructed, facility operation requires strict adherence to environmental protection performance standards.

Environmental resources afforded special protection through permitting include groundwater, air, biological, and cultural resources. By the very nature of the in situ leach mining process, the greatest potential for environmental effect rests with the fluid injection-recovery system. Accordingly, the principal element of environmental permitting is protection of groundwater resources. Protection of air, cultural, and biologic resources is subordinate to this. Air emissions expected to result from an in situ leach mining operation include fugitive dust from access road traffic and volatile organics and sulfuric acid mist from operation of the SX/EW facility. Biological and cultural resources may be afforded special protection if there is public ownership of the property. Given its overriding significance, the emphasis of this discussion will be on groundwater permitting and protection.

Federal requirements that address the protection of groundwater from fluid injection activities are administered by the EPA under the Underground Injection Control (UIC) program. The UIC program was implemented under part C of the Safe Drinking Water Act, Public Law 93-523. Technical requirements of this program are identified under 40 CFR 144 and 146 (Code of Federal Regulations, Anon., 1986b). Requirements specified under part 144 address permitting and program specifications. Part 146 addresses technical criteria and standards for operation. Individual states may elect to develop their own UIC program and obtain primacy from EPA for program administration and enforcement. In developing its own UIC program, a state is not precluded from developing regulations more stringent than those established by EPA. States, by their own authority, may enforce additional requirements related to groundwater conservation and well construction.

EPA criteria under the UIC program specify that no fluid injection shall be authorized if it results in the movement of fluid containing a contaminant into an underground source of drinking water (USDW), if the presence of that contaminant may cause a violation of federal primary drinking water standards, or if it may adversely affect public health. If an aquifer within a prospective leach interval meets the definition of an USDW, it must be determined to be an “exempted aquifer” before lixiviant injection may commence. An exempted aquifer is one that would otherwise qualify as an USDW, but which has been determined to have no real potential as a drinking water source.

It is incumbent upon a mine operator to evaluate all environmental permitting and protection requirements as well as site conditions early in the mine planning process. This step is necessary to establish the feasibility, costs, and schedule for developing an environmentally acceptable mine design. The regulatory process remains in a state of flux, but permit applications must address ground water protection, a monitoring program, a groundwater restoration plan, and other site reclamation activities.

15.3.3.4 Future Trends

As the combination of more stringent environmental controls and lower-grade deposits continues to stretch conventional technology to its economic and environmental limits, in situ leach mining and other forms of advanced mining systems will emerge as viable options for mineral extraction. Depending upon the commodity and other hydrologic, geologic, and hydrometallurgical factors, in situ leach mining looks promising for small high-grade deposits, as well as large low-grade fracture-hosted deposits. If the development of in situ uranium mining technology is a precursor of what we might expect, then the future development of in situ leach mining technology looks very promising.

The next major demonstrated advance in this technology will be the in situ leach mining of copper oxides, due to the ease with which copper can go into solution using a dilute acid lixiviant. The primary candidates for this technology will be those deposits that are primarily fracture-hosted, which means that the copper mineral values occur predominately within the microfractures and joint system within the deposit. The state of Arizona has numerous oxide deposits that may be suitable for in situ mining technology. It is very likely that by 1993, in situ leach mining of copper oxides will be demonstrated and proved technically feasible. This will be a great advance for the mining industry in general because of the overall environmental advantages and significantly reduced surface disturbance.

Looking ahead, with the recent advances in biotechnology, down-hole directional drilling, and the ability to inject oxidants into the ore body, a fresh look will be given to in situ leach mining of copper sulfides. A focused, joint research effort by government, industry, and the universities will be required in order to solve the major technological barriers that are currently holding back such mining. Although several recent pilot tests have been tried on an industrial basis in the United States and Canada, a large-scale commercial operation has never been demonstrated. However, the extent of reserves in the United States almost assures that over the long term, in situ leach mining technology will be developed for copper sulfides.

In the longer term, some of the so-called critical and strategic minerals are candidates for in situ leach mining technology. These elements are so denoted because they are essential in the steel, aerospace, electronic, and defense orientated industries. The United States, for example, does not have high-grade deposits of manganese, cobalt, nickel, or platinum group metals and therefore must import most of its supply from foreign sources. In the United States, occurrences of small, deep and/or low-grade deposits of several critical and strategic minerals do exist. Included in these are the Co-Ni-bearing laterites of Oregon and California; the manganiferous ores of Artillery Peak in Arizona, the Cuyuna Range and Emily District in Minnesota, and the polymetallic deposits of the Blackbird District in Idaho and the Stillwater District in Montana. The problem, however, is that many of these deposits cannot be mined economically using conventional approaches. Possible leach systems for some of these are addressed in Chapter 15.2.

Ultimately, the advancement of in situ leach mining technology to the commercial scale will require the integration of numerous technical disciplines with advanced innovative mining techniques, and represents a significant challenge for the minerals industry. However, the challenge is worth the risk for several reasons. First and foremost, the successful development of this technology will keep United States producers competitive in the world minerals market. Secondly, from a domestic stewardship perspective, resource utilization will be maximized by making feasible the recovery of metals that would or could not be mined by conventional methods.

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Section 16 Surface Mining: Comparison of Methods

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Chapter 16.0
INTRODUCTION

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Each mining operation is unique in some aspect—ownership; ore body location, size, geometry, grade, mineralogy, hydrology, or geotechnical parameters; or constraints such as environmental regulations. However, all mining operations are alike in one aspect; the prime objective of any mining project is to maximize the return on the investment. This is the golden rule, or in the language of mining, the investor's law of conservation.

A mineral resource, by definition, does not become a minable reserve unless the mineral or minerals can be extracted economically (and legally). The three preceding sections have covered the various types of surface mining options. The choice of the most economic method is dictated primarily by the type of deposit to be mined, the mining rate, and the physical characteristics of the deposit.

A comparison of surface methods must take into consideration the prime objective of mine planning: (1) develop the most economic plan for the overall project that will maximize the return on the monies invested, and (2) maximize the recovery of the resource.

Surface mines can be subdivided into various classes and subclasses and which can be illustrated as follows:

Class	Subclass	Method
Mechanical		Open Pit Mining
		Quarrying
		Strip Mining
		Auger Mining
Aqueous	Placer	Dredging
		Hydraulic Mining
	Solution	Surface Techniques
		In Situ Leaching

The purpose of this section is to discuss the relative advantages and applications of one method vs. another, as well as some of the more important evaluation parameters that must be considered in any method comparison.

Chapter 16.1
FACTORS AND CONDITIONS

ERNEST BOHNET

Open pit mining and strip mining are the two most dominant types of surface mining methods in the world, accounting for approximately 90% of the surface mineral tonnage. Strip (open cast) mining is used for large, tabular, flat-lying ore bodies or mineral seams, such as coal, that are relatively close to the surface. Backfilling of these mines is usually economically feasible and desirable as part of the advancing mine operations.

Open pit mining is typically applied to disseminated ore bodies or steeply dipping veins or seams where the mining advance is toward increasing depths. Backfilling usually cannot take place until the pit is completed; even then, the prohibitive cost of filling these pits with all of the waste rock removed at the end of the mine life, would seriously jeopardize the project's economics. Very few large open pits in the world could support such a costly handicap.

Seam deposit mining, like coal, uses different planning methods and terminology as compared to open pit mining for base and precious metals (Section 13). However, all surface mining methods have one common element: mining proceeds until the economic limit is reached. The method of determining the economic limit is different for the two types of deposits. Coal, for example, in most instances, is readily distinguished from waste rock and is of a near uniform quality and value per ton mined in each seam. Therefore, mine design limits are determined by the break-even strip ratio, commonly defined as cubic yards (cubic meters) of waste to tons (tonnes) of coal that can be mined. Since one ton of coal is approximately equal to one cubic yard (1 tonne ~ 0.84 m^3), the ratio is similar to a volume ratio.

The pit limits for a disseminated deposit, such as base or precious metals, is determined by a break-even economic analysis of the value of the ore in each finite incremental pit expansion. This economic analysis is a two-step process where first the net operating value is determined for each ton of material in a block model. Second, the pit limits are determined by expanding the pit shape to a break-even point where the net operating margin equals the cost of waste stripping. Since each ton of ore has a different grade and net value, a break-even stripping ratio cannot be utilized. Complicating this matter further is the potential for dual ore processes, such as milling and leaching of the same ore type. Milling is a higher cost process method used to increase metal recoveries, whereas leaching is a slower process with correspondingly lesser recoveries and lower costs.

Therefore, the mining engineer must have sufficient knowledge of the overall project: from ore geometries to the process extraction methods. This includes an awareness of all cost and recovery parameters, as well as the factors that affect the value of the salable product. For example, where process capital and operating costs are high, ore selectivity plays an important role in mine design and selecting the most appropriate mining method. Mining cost increases can be more than offset by reduced process capital and operating costs per unit of salable product.

The total amount and type of material to be moved per time period, coupled with ore body geometries, are the prime criteria in selecting the most economic and applicable mining method.

Other factors that affect the type of mining method as well as type and size of equipment include:

1. Topography.
2. Remoteness of operation.
3. Support infrastructure.
4. Availability and cost of skilled labor.
5. Climate and altitude.
6. Environmental restrictions.
7. Groundwater quantities.
8. Slope stability and ground conditions.
9. Investment risk.
10. Blasting restrictions.
11. Life of operation.
12. Waste rock disposal.

Topography, as discussed in the preceding chapters, can play a major role in selecting the most appropriate mining method. Generally, the more rugged the topography, the more limited are the choices of mining method and level of overall production.

Remote operations require substantial infrastructure to maintain the standard of living needed to attract a well-qualified work force. Therefore, the mining and processing methods selected must result in high productivities per employee-shift. Better ore selectivity in the mine will reduce process plant requirements and could, therefore, reduce overall staffing requirements. If the ore grade can be raised sufficiently, the ore may be transported offsite for processing at a less remote location. Large equipment can be utilized for the bulk of the waste stripping and small equipment employed to selectively mine the ore.

Where labor costs are high, the same philosophy can be applied. Lower labor costs and the level of labor skills available can also affect the choice of the mining method. For example, front-end loaders and mechanical drive trucks will require less diverse maintenance skills as compared to the labor force needed to maintain electric/hydraulic shovels and electric haul trucks.

With adverse climatic conditions, mining operations may be seasonal. Seasonal and short-life operations may use contractors for all mining operations or for only the bulk of the waste stripping. The use of contract miners is discussed in detail in Section 6.0.

Environmental considerations play an increasingly important role in the type of mining operation permissible. Open pits in populous areas may require backfilling, while the same type of operation in a remote location may have minimal environmental constraints. For example, hydraulic mining, outlawed in California in 1884, was used to remove the unconsolidated overburden at the Bougainville Copper mine in 1969–1971. Environmental and aesthetic conditions may be so restrictive in some areas as to eliminate surface mining options altogether. In some cases, this forces the use of underground mining, which will result in reduced mineral resource recovery.

Groundwater conditions and slope stability can play an important role in mining method options, mine design, production scheduling, and equipment selection. A large dragline may not operate safely where ground conditions are wet and unstable and the risk of slope failure high. If large water inflows are anticipated, mining may have to be accelerated to avoid high pumping costs from a lengthy operation. Ore can be stockpiled at the processing plant and treated months to years after the mine is completed.

The risk of foreign investment varies from country to country; however, if risks are high, then capital expenditures are usually minimized. Front-end loaders, with shorter useful lives and lesser capital costs, are preferred over large shovels and draglines. The larger the project and the longer the development period, the greater the investment risk.

Blasting restrictions due to the operation's proximity to populous areas or in foreign countries with explosive use constraints also can affect mining method selection. Continuous mining machinery similar to pavement cutters or ripping can be utilized in some circumstances as an alternative to drilling and blasting.

Available waste disposal areas may be a significant distance from the surface mining operation. A viable alternative in this situation is crushing and conveying of the waste, and this in turn can affect the mining methods. Where the ore has a high value per ton, in excess of one hundred times the cost of mining, very high strip ratios can be economically justified. The mining method in these circumstances is designed to be primarily a waste removal operation. Ore mining is a small cost component of the overall operation and great emphasis must placed on efficient waste production and mining methods. Since most open pit and strip mines have waste to ore ratios higher than 1:1, the majority of surface mines are waste rock operations. It is, therefore, important to ensure that any effort to reduce ore mining costs does not increase waste mining costs. For example, in a coal stripping operation using shovels and trucks, often a slot is left on one side of the backfilled pit for coal haulage to minimize coal mining costs. This slot prevents efficient waste haulage around both sides of the pit, thereby increasing waste haulage distances and costs significantly. As strip ratios increase, the mining method and haul road system will require modification to allow more effective waste transportation and reduced overall mining costs.

Quarrying is a special type of open pit mining used to produce dimension-stone products. Rock joint fractures are infrequent, and, therefore, the bench faces are vertical. The lack of fractures often permits near vertical highwalls that can approach up to 1000 ft (300 m). Quarrying is a high-cost selective method that is only used to produce dimension stone such as marble, granite, limestone, sandstone, flagstone, and slate.

Auger mining is primarily used to remove coal from under a final highwall. The final highwall position is determined by economic stripping limits, property boundaries, or physical constraints such as a body of water.

In the case of physical constraints, auger mining may be the only alternative to increase the amount of coal recovered. However, if the highwall position was determined using a break-even stripping ratio analysis, the cost of stripping the final 100 to 300 ft (30 to 100 m) break-even increment should be compared to auger mining costs and profitability.

Aqueous surface mining methods, uniquely involving the use of water for extraction, can be used in special circumstances. The mining of placer-type deposits that contain concentrations of heavy metals in unconsolidated overburden are particularly suitable to dredging and hydraulic mining if:

1. An adequate water supply is available.
2. The mining operation can comply with the applicable environmental regulations.

Hydraulicking, now often termed hydraulic mining, also requires the presence of a natural gradient away from the deposit to facilitate hydraulic transport of the resultant ore or waste slurry to the process or disposal area.

Dredging can be used if a natural or man-made body of water overlies the placer deposit. Dredging machines can be classified as mechanical, bucket-line, bucket-wheel suction, and dipper; or hydraulic, suction, and cutterhead. The type of dredging equipment selected depends on the type of material to be excavated.

Solution mining using surface or in situ techniques is applicable to deposits of minerals that can be recovered usually by dissolution, but also by melting, leaching, or slurrying. The two methods, are similar and are differentiated primarily through locale and the type of minerals recovered. Surface leaching generally employs heap or dump leaching of mineral values; copper, gold, and uranium are common examples.

In situ mining uses water to dissolve, melt, or slurry the minerals. Barren solution is introduced down one set of wells and the loaded solution returns to the surface through concentric or another set of wells. This mining method is primarily used with sulfur, evaporite, or water-soluble minerals. In situ leaching uses chemical or bacteriological reagents, usually mixed with water to selectively dissolve the valuable minerals. Drillholes are typically used to inject and recover the solution. Extraction may also be enhanced by rubblization of the ore zone prior to solution application. Solution mining will increase in the future as more effective reagents are developed and application methods are improved.

The preceding discussion has focused primarily on factors and conditions that can affect surface mining options. New technologies will result in a greater range of alternatives in the future. Therefore, the mining engineer needs to continually review these developments as applicable to each old and new situation encountered. Basic assumptions must be revisited to ensure that past premises are still valid in light of new developments. Mining is dynamic in nature, and therefore, mining methods will continue to change to meet new economic and environmental conditions.

16.1.1 DEVELOPMENT AND PRODUCTION SEQUENCES IN OPEN PIT AND STRIP MINING

The mining method and the selection of the appropriate size and type of equipment are closely associated. The following primarily addresses loading and hauling fleets, as applied to both strip mines and open pit operations.

The size of the operation over time has a direct bearing on the type of equipment selected. Loading and hauling fleets are dependent fleets, and, therefore, the effectiveness and availability of each affects the fleet requirements of the other. For example, if a mining operation had only one large shovel, the truck production will only be effective when the shovel is operable and broken rock is available at the shovel face for loading. Therefore, to have an effective operation, more than one unit per equipment type is desirable. A mining operation should have one drill per shovel and at least one back-up shovel/loader to load trucks when a scheduled shovel breaks down.

Large mining operations require time to develop sufficient working area for effective use of the equipment. Therefore, the equipment selected for full production may not always be suitable for the development period. It must also be noted that the development or preproduction period may require different mining methods due to lesser material movements and different types of material to mine. During the development stage, access haul roads need to be constructed, and in rugged terrain, bench widths may be much narrower than those available during production. Smaller equipment may be needed during this stage of mine development.

Sufficient time must be allowed in the production schedule to drill, blast, and remove the material from the mine. Narrow benches do not allow continuous mining unless access for drilling and blasting is provided in the plan.

During preproduction and during the stripping phase of each mining sequence, higher benches may be used. For example, stripping may be accomplished on 40-ft (12-m) benches and ore production on 20-ft (6-m) benches to minimize ore dilution. The advantages of higher benches include the use of larger equipment, minimal access construction, and overall cost reductions.

Large open pit and strip mines are subdivided into smaller phases to improve project economics. The objectives of dividing a surface mine into phases or mining sequences are to

1. Reduce preproduction stripping requirements.
2. Shorten the time between stripping and ore production.
3. Develop the highest net value ore first and mine the lowest net value ore last.
4. Avoid high stripping peaks that result in excessive equipment purchases and manpower requirements.
5. Blend the ore or coal to produce a less variable product.

The development and production sequencing requirements must be considered in selecting the most appropriate mining equipment types and mining method.

Chapter 16.2
METHOD ADVANTAGES AND DISADVANTAGES

ERNEST BOHNET

The advantages and disadvantages of one type of surface mining vs. another is related to the types of equipment utilized and the associated costs and benefits derived from the use of that equipment. In the following discussion, emphasis is placed on the mechanical extraction methods, generally open pits and strip mining.

Strip mining has the greatest choice of equipment, particularly for the mining of relatively flat-lying coal seams in gentle topography.

Open pit mining conversely is usually limited to shovel/truck or front-end-loader/truck-type operations. Shovel/rail-type operations of the past have been almost phased out in favor of the greater flexibility of mining with truck haulage.

Sections 13 and 14 cover the various types of mining methods and equipment applications for strip mining using draglines, shovels, bucket wheel excavators, conveying systems, and combinations of these types of systems.

In these same sections open pit mining is usually conducted using cable shovels, hydraulic shovels, or front-end loaders. The loading equipment is usually matched with haul trucks that can be loaded in four to six passes.

The choice of loading unit is dependent on total production requirements vs. the minimum number of active work areas, as well as ore selectivity and blending requirements. Additional factors to be considered are the type of material to be loaded and the ground conditions. In hard digging or with sharp rock in wet climates, tracked shovels are preferable over rubber-tired loaders. However, if the mine production requirements are relatively small or the mine life is short, the lower capital cost of the front-end-loaders must be taken into consideration.

The rubber-tired loader is commonly used to load material that is easy-to-medium digging and for operations requiring less than 100,000 tons (90,000 t) of daily material movement. Loaders are highly mobile and, therefore, are ideally suited for situations requiring frequent movement from one mucking face to another for ore blending requirements. Loaders also can be used to load-haul-dump material into a crusher from blending stockpiles dumped near the crusher. Some operations have a mismatch of large shovels and a small crusher. A mismatch occurs when the crusher rating of tons per hour is much less than what the shovel can produce through the truck fleet. If this situation occurs, there are advantages to stockpiling the material at the crusher to avoid truck dumping delays and thereby maximize shovel and truck productivities. The extra cost of a stockpile-rehandling loader is more than offset by the increase in shovel and truck productivities and their respective reduced fleet requirements.

Loader cycle times are approximately 25 to 50% greater than a shovel's cycle time. Therefore, a loader has to be 25 to 50% larger (bucket capacity) than a shovel to attain the same productivity. The loader bucket fill factor is usually less than a shovel's in medium to hard digging. Hydraulic shovels have similar limitations and advantages as a cable shovel. The hydraulic shovel is typically not used in very hard digging. Hydraulic shovels are available in a range of sizes comparable to front-end loaders. However, since the larger sizes over 30 yd^3 (23 m^3) are not common, large cable shovels are used as the prime loader at mines where production requirements are in excess of 200,000 tons (180,000 t)/day.

Hydraulic shovels can effectively load from smaller bank heights than cable shovels and, therefore, if ore selectivity is very important, these shovels can improve ore selectivity and minimize dilution. A hydraulic shovel has additional advantages over a cable shovel in the sense that the bucket can be vertically positioned anywhere in the mucking face and selectively load from either the top half or bottom half of the face. This advantage is particularly useful in mining seam deposits where waste separation from the seam is necessary, or in some precious metal (gold) deposits that have highly variable grades and/or oxide-sulfide mineralization contacts.

Hydraulic shovels can also be equipped with a backhoe-type loading bucket. This permits the shovel to sit on top of the bench and load trucks that are positioned on the same bench or on the bench below. Backhoe loading is particularly useful for loading a dipping seam or wedge of material where loading with a cable shovel would be impossible. Backhoe loading is also very useful in wet conditions, permitting the trucks to travel on a drier bench above the water, with correspondingly increased tire life.

The useful life of loaders, hydraulic shovels, and cable shovels varies according to the application, maintenance standards, and the company's general replacement policy. However, life for a front-end-loader is typically in the three- to four-year range, hydraulic shovels in the eight- to twelve-year range, and cable shovels are usually replaced on 20-year intervals. The longer the useful life, the more obsolete the equipment becomes in comparison to new technological advances. In order to combat this technological aging factor, many equipment manufacturers are progressing toward complete modular part replacement designs. These designs allow the mine operator to discard older components in exchange for new, more reliable modules that may have more advanced features.

Other factors that influence the choice of loading equipment is the trade-off between lower and higher capital costs vs. higher and lower operating costs. Complicating this analysis is the difference in useful life. Since the prime objective in mining is to maximize return on the investment, time value of money must be taken into consideration in any trade-off analysis. Discount rates of 10 to 15% commonly are used to convert future costs into a net present value. The higher the discount rate, the greater the emphasis on reducing current costs. Expenditures after 10 years have little significance on the total net present value. This method of ranking alternative fleets can be applied to all choices: dragline vs. a shovel and truck fleet in a coal stripping operation, or cable shovels with 170-ton (150-t) trucks vs. front-end loaders with 85-ton (76-t) trucks. However, it is important to note that other tangible and intangible factors may be present that often are overlooked in this capital vs. operating costs analysis. An example of this would be greater ore selectivity, which means less ore dilution and less ore discarded as waste, thereby resulting in higher-grade ore to the processing plant. A small grade change can significantly affect overall project economics.

As an example of this, project economics are very sensitive to changes in commodity price. However, grade changes and recovery changes have the same effect as a commodity price change. Therefore, if ore selectivity can improve the ore grade by 10%, this is far more significant than a 10% increase in both mine capital and operating costs. This factor must be taken into consideration in any equipment trade-off economic analysis.

Other factors that influence the choice of equipment and mining method are less tangible. As previously discussed, these include mobility requirements, ore blending, hard or easy digging, maintenance skills of the available work force, size of the operation, working area, and advance rate required.

In respect to this last point, open pit mines are subdivided into phases. Each phase is somewhat independent of the following phases; however, the object is to remove waste rock in a timely manner that exposes ore before ore reserves are exhausted in the prior phase. There are circumstances where the advance rate or number of benches to be mined each year in a phase requires more than one loading unit. If the working width of a bench is too narrow to permit more than one unit, then larger equipment may be required.

An example of this situation is a mine that was using 15-yd^3 (11-m^3) hydraulic shovels as the prime loading fleet. The mine plan indicated that after the tenth year of operation, the fourth mining phase required the movement of 15 million tons (13.5 Mt) of material per year for five years to expose ore. Since one shovel could not move this material, and since two shovels could not effectively work on the bench, the alternatives were to purchase a large 40-yd^3 (30-m^3) cable shovel or start mining the phase earlier. A discounted cash flow analysis indicated the better choice was to purchase one 40-yd^3 (30-m^3) shovel for the bulk of the waste stripping and to use the smaller shovels for ore production.

The purpose of subdividing an open pit into phases is to improve the project economics. This also can mean that material movement requirements through time will change significantly. It does not mean that mining quantities will change dramatically each year. An effective mine production schedule must be developed to take into consideration equipment and personnel requirements. A smooth schedule will give the mine operations and maintenance departments sufficient time to build up their organizations without significant annual variances. If total material movements are increased during the life of the mine, these increases should be smoothed to form a new production level that remains constant at a higher level for a number of years.

During preproduction, or at some point in the life of the mine, this may not always be feasible, and an alternative to this is contract stripping. Contract stripping can be justified if the mine life is short or the stripping peak is of fairly short duration, for example, less than three to four years. The trade-off again is lower capital costs vs. higher operating costs. The intangible benefits include the avoidance of hiring and training a large work force for a relatively short time period, greater assurance that the stripping will be carried out on time and per the contract terms, and less management skills required. The intangible disadvantages are less control over the operation, and it must be recognized that a contractor's goal is to move a volume of rock, not necessarily to separate ore and waste in the mine owner's best interest. Adequate ore control procedures must be conducted as part of the mine owner's responsibility to minimize ore dilution and loss. It is also the mine owner's responsibility to ensure that the mine plan is accurate, otherwise extra costs will be charged by the contractor for scope deviations.

The life of a mining project plays an important role in selecting the most appropriate mining method, equipment fleets, and whether to use a contract miner. Particularly with precious metal mines, capital costs may have to be minimized due to a lack of available funding for whatever reason.

The lack of available capital also may force the mine operator to look for used equipment for initial mining. This equipment may be replaced within a few years with new equipment if mine life is sufficient and positive cash flow has been demonstrated to the lending agency's satisfaction. Any economic analysis of used vs. new equipment is extremely subjective. Equipment maintenance costs are composed of maintenance labor and repair parts. The amount of maintenance required is related to the condition of the equipment purchased and commonly related to the operating hours on the equipment. If the equipment has been overhauled adequately, maintenance costs will be similar to that of a piece of equipment that has been in operation one year. The maintenance requirements of rebuilt used equipment will increase at an aging rate that is approximately double that of a new piece of equipment.

In order to adequately explain this factor for economic analysis, consider the following example of haul trucks that normally have a useful life of a maximum of 40,000 hours. If the average maintenance cost during the life of a truck is 1.0, then the relationship of maintenance costs to life can be stated as follows:

Operating Hours	Maintenance Factor	Mechanical Ability
0– 5,000 hr.	0.30	85%
5,000–10,000 hrs.	0.70	80%
10,000–20,000 hrs.	1.00	75%
20,000–30,000 hrs.	1.20	70%
30,000–40,000 hrs.	1.35	65%

If a truck with 30,000 to 35,000 hours is overhauled, maintenance costs should drop to the equivalent of a new truck with 5000 operating hours. However, after operating only 15,000 hours, the overhauled used truck would have the same maintenance requirements as a truck with 35,000 hours.

It also must be noted that used equipment availabilities are much less than that of new equipment. New equipment usually will have manufacturers' warranties and close field support.

The effect of purchasing used equipment with lower availabilities is that more units are needed to obtain the same production level, and the maintenance work force has to be significantly larger initially. Nonetheless, if capital funding is limited, used equipment or a contract miner may be the only viable solution.

The comparison of one type of mining method vs. another should not neglect to compare the various types of equipment that are dependent on another type, such as loading and hauling equipment, loading productivity vs. crusher capacity, and drilling unit capacity vs. scheduled shovel production.

For any analysis, it is better to have the first unit in the chain of production with greater capacity than the downstream unit. Therefore, drill capacity should exceed shovel/loader capacity, shovel capacity should be slightly higher than crusher capacity, or the capacity of two shovels should be matched with the capabilities of one crusher.

A mismatch occurs when there are only a few units in production and the capacity of one unit dependent on the production of another is significantly different, For any comparison of different mining methods or equipment types, it is very important to conduct the economic analysis using the most accurate and correct assumptions.

Two of the major expense areas in any open pit or stripping operation are the functions of loading and hauling. Typically, 30 to 50% of the total mining cost is related to hauling all material types out of the pit or to the waste backfill area. A cost comparison of equipment sizes and alternatives must be based on an optimum selection of loading and haulage fleet sizes, otherwise the comparison may be unfairly skewed one way or the other.

Table 16.2.1. Fraction of Time That (x) Units are Available Out of a Fleet of (n) Units That Have a Probable Availability of 70%

Fleet Size (n)	Number of Units Available (x) 0	1	2	3	4	5	6
1	0.30	0.70					
2	0.09	0.42	0.49				
3	0.03	0.19	0.44	0.34			
4	0.01	0.08	0.26	0.41	0.24		
5	0.00	0.08	0.13	0.31	0.36	0.17	
6	0.00	0.01	0.06	0.19	0.32	0.30	0.12

As outlined in the previous chapters, the first step in any comparison is the determination of unit productivities per operating shift or per hour. Loader productivities can be estimated by using the anticipated load time per specified truck size and dividing this into the estimated effective minutes per shift. Productive minutes per eight-hour shift (with no downtime) will vary from 330 to 420 minutes, depending on the efficiency of the assignment systems used and the balance between loading and haulage units. Industry standards typically use 350 minutes as a base for comparison.

Truck productivities can be determined from haul profiles for each specific time period. For initial studies comparing one system to another, average productivities per time period can be used to determine fleet requirements.

One method that can be utilized to more accurately analyze the loading and haulage fleet requirements is the binomial method. The best way to explain this method is by using a simple case to demonstrate its use in determining dependent fleet production.

Example 16.2.1. Consider the following fleet consisting of

1. One loader, 80% mechanical availability and an estimated productivity of 9000 tons (8200 t)/operating shift.
2. Three haul trucks, 70% mechanical availability and an estimated productivity of 4000 tons (3600 t)/operating shift.

Determine the fleet capacity.

Solution. Assume that the fleet is scheduled 100% of the time and will only be inoperable if either the loader or all the trucks are down for repairs.

One could incorrectly assume that the average loader production would be 80% of 9000 tons (8200 t)/shift, or 7200 tons (6500 t)/shift. However, since the loader production is dependent on available haul trucks, the truck downtime distribution must be considered.

Table 16.2.1, developed using the binomial formula, shows the fraction of time that x units are available out of a fleet of n units with a given availability of p where this fraction equals:

$$\frac{n\,!}{x!(n - x)!} \times p^x \times (1 - p)^{(n-x)} \tag{16.2.1}$$

This table indicates that 34% of the time, all three trucks are available; 44% of the time, two are available; 19% of the time, only one is available; and 3% of the time, the whole truck fleet is down. When less than three trucks are operating, the loader production will drop to the amount that the trucks can haul, in this case 8000 tons (7300 t) 44% of the time, and 4000 tons (3600 t) 19% of the time.

In summary, the fleet capacity can be stated as follows: The loader operates 80% of the time, and during this time, 34% will be at 9000 tons (8200 t)/shift, 44% will be at 8000 tons (7300 t)/shift, and 19% will be at only 4000 tons (3600 t)/shift. Total fleet capacity would average:

0.80 × 0.34 × 9000	=	2448 tons/shift
0.80 × 0.44 × 8000	=	2816 tons/shift
0.80 × 0.19 × 4000	=	608 tons/shift
TOTAL	=	5872 tons/shift (5340 t)

From this simple example, one can see that production from the loader would be 18% short of the initial estimate of 7200 tons (6500 t)/shift that was determined without consideration of the haul fleet. The same analysis holds true for haul truck requirements determinations. For example, if only the number of haul trucks needed was examined, the following could result:

Annual target objective	1,800,000 tons (1,636,000 t)
Shifts scheduled	250 shifts
Tonnage requirements/shift	7200 tons (6550 t)
Average truck productivity	4000 tons (3600 t)/shift
Need 1.80 operating trucks/shift	
3 trucks @ 70% availability will average 2.1 shifts	

Therefore, it could be incorrectly assumed that three trucks would be sufficient. However, we know from the previous discussion that if the loading fleet contained only one loader, then 20% of the time the haul fleet would be idle waiting for the loader to be repaired. We also know that the loader could not keep up with three trucks, and production would be limited to 9000 tons (8200 t)/shift, not the 12,000 tons (10,900 t)/shift indicated by the haulage capacity. Therefore, haulage capacity would be

250 shifts × 0.80 × 0.34 × 9000 tons =	612,000 tons
250 shifts × 0.80 × 0.44 × 8000 tons =	704,000 tons
250 shifts × 0.80 × 0.19 × 4000 tons =	152,000 tons
	1,468,000 tpy (1,335,000 t/a)

The solution in this case would be to purchase another loader or work more shifts. With larger fleets, the shortfalls caused by calculating fleets independently are not so easy to quantify without utilizing a computer program that takes into consideration the availability interaction of the two fleets. Haulage truck productivities can be averaged for these calculations; however, in reality, haul distances will vary, and this can be used to counter some of the production shortfalls that would otherwise occur when short of trucks or loading units. For example, if the mine were short of trucks, then the shortest haul distances would be used; and conversely, if the mine were short of loading units, longer hauls would be used to counterbalance the shortfalls that would otherwise occur. This may not always be possible, but haulage distance flexibility should be one objective of a good mine plan.

The lack of a loading unit more severely impacts haulage capacity than the lack of one truck. For this reason, it is better to have slightly more loading standby capacity and use an undertrucking philosophy to minimize both capital and operating costs. Haulage costs per ton of material mined will typically be a several times the loading cost.

It is not always in the miner's best interest to have operators for each piece of loading or haulage equipment. In fact, as the fleet sizes increase or availabilities decrease, the extra operators are more of an operating cost liability.

The binomial distribution can be utilized to determine the most likely number of operating shifts that will be obtained for a given fleet size, number of operators or equipment scheduled,

Table 16.2.2. Equipment Operating Shifts Attainable/Shift With a Probable Availability of 70%

Fleet Size	Number of Operators					
	1	2	3	4	5	6
1	0.70					
2	0.91	1.40				
3	0.97	1.76	2.10			
4	0.99	1.91	2.56	2.80		
5	1.00	1.97	2.80	3.33	3.50	
6	1.00	1.99	2.92	3.66	4.08	4.20

and the equipment availability. Table 16.2.2 is derived from the following formula

Operating shifts/shift =

$$x \frac{\overset{n}{\Sigma}\, bj}{j = x} + x - 1\,(b_{(x-1)}) + x - 2\,(b_{(x-2)}) + \ldots 1\,(b_1) \tag{16.2.2}$$

where x is number of assigned operators, n is number of units in the fleet, and b is binomial probability of n things taken one at a time

For example, the anticipated operating shifts for five pieces of equipment with a 70% availability and four operators is 3.33. Adding one more operator to the payroll would only increase loader operating time by 0.17 shifts. From another viewpoint, if a truck fleet were purchased to cover five operating loading units, then the trucks purchased for the fifth loading unit would only operate 17% of the time. During the remaining 83%, the trucks and their drivers would be wasted capacity unless the remaining loading units could handle the extra trucks effectively.

In order for the remaining loading units to handle the extra trucks, the operation must be initially designed to be undertrucked when all the scheduled loading units are operating.

The following example is taken from an actual case in China for which the initial fleet specified for review was severely overtrucked.

Example 16.2.2. Initial production plans indicated an overburden stripping requirement of 160,000 tons (145,500 t)/shift. Production rates for the shovels and trucks were estimated to be 20,000 tons (18,200 t) and 2500 tons (2300 t)/operating shift, respectively. It was proposed to purchase and schedule 10 shovels to operate at 80% availability to give a loading capacity of 10 × 80% × 20,000 tons (18,200 t) = 160,000 tons (145,500 t). In order to cover each shovel, it was proposed to purchase 10 trucks and schedule eight to operate at 80% availability giving an average capacity of 10 × 80% × 2500 tons (2300 t) = 20,000 tons (18,200 t)/shift, thereby matching each shovel's productive capacity.

Solution. The production plan devised assumed that trucks and their drivers, due to physical constraints, were not transferable between the various loading units, and the haul distances were always the same length. Therefore, the total equipment requirements were estimated to be 10 shovels and 100 trucks, of which 10 shovels and 80 trucks would always be scheduled to operate.

By applying the binomial function to a group of one shovel and 10 trucks, the anticipated output would be approximately 15,000 tons (13,640 t)/shift, or 150,000 tons (136,400 t) for 10 shovels. This is 10,000 tons (9100 t) short of the objective and is caused by the fact that there will be times when less than eight trucks are available resulting in lost shovel production. There also will be times when more than eight trucks are available, but since the shovel can use only eight, no increase in production can be gained to counterbalance the shortfall.

Table 16.2.3. Optional Fleets vs. Cost (Ex. 16.2.2)

	Case				
	1	2	3	4	5
Shovels in Fleet	10	11	10	11	11
Shovels Scheduled	10	11	10	9	9
Truck Fleet	100	107	100	85	81
Trucks Scheduled	80	85	80	70	70
Truck Productivity, tons					
Minimum	2,500	2,500	2,500	2,500	2,100
Maximum	2,500	2,500	2,500	2,500	2,600
Fleet Production, tons	150,340	160,049	159,544	160,498	160,240
Shovel Operating Shifts	8.00	8.80	8.00	8.39	8.39
Production/Shovel Operating Shift, tons	18,792	18,187	19,943	19,125	19,094
Truck Operating Shifts	75.20	80.05	78.41	67.34	64.71
Effective Truck Shifts	60.10	63.98	63.82	64.20	64.13
Production/Truck Effective Shift	2,500	2,500	2,500	2,500	2,499
Operating Cost/Ton*	$0.4438	$0.4447	$0.4364	$0.4118	$0.4085
Fleet Purchase Cost ($ × 10^6)	$140	$151	$140	$129	$125

*Loading and haulage cost only.
Conversion factor: 1 ton = 0.9092 t.

If the trucks could be transferred from shovel to shovel to balance out truck shortfalls, then production will be increased by approximately 10,000 tons (9100 t) and the target attained. However, this is avoiding the real problem, over-trucking. The 160,000 tons (145,500 t)/shift target can be obtained with different shovel/truck fleet combinations with a lower capital expenditure and at a reduced operating cost. Table 16.2.3 illustrates various fleet sizes that would produce 160,000 tons (145,500 t)/shift. For comparison, operating and capital costs of each fleet are listed, and the basis of these estimates are provided in Table 16.2.3. Case 1 is the base as originally specified; Case 2 is the base case but with the equipment increases that would actually be needed to reach 160,000 tons (145,500 t)/shift with the same constraints as Case 1. Case 3 is the production obtained if trucks can be transferred from one shovel to another, Case 4 is an optimum fleet balanced to minimize costs with fixed hauls, and Case 5 is an ideal case where haul distances can be flexed to accommodate truck or loading equipment shortages and maintain peak production efficiencies. In this last case, production efficiency is increased by changing the mine plan to allow flexible haul distances. Instead of a fixed 2500 tons (2300 t)/truck shift, dump points can be varied to shorten or increase haul distances to give a truck productivity range of 2100 tons (1900 t) to 2600 tons (2400 t)/shift, but still average 2500 tons (2300 t)/effective truck shift.

This example shows the magnitude of the savings that can be achieved by selecting the optimal fleet size and removing some of the high-cost constraints.

Table 16.2.4 gives basic assumptions for all cases.

A computer program that models the dependent fleet interactions can generate and investigate a large number of loader/truck combinations, select those that meet the production goal, and finally determine which combination will give the lowest overall combined cost/ton of material mined.

This type of evaluation is also a very useful tool for sensitivity analyses to investigate how changes in parameters such as avail-

Table 16.2.4. Basic Assumption for All Cases (Ex. 16.2.2).

	Shovels	Trucks
Mechanical Availability	80%	80%
Maximum Productivity/Operating Shift	20,000 Tons	As Noted
Operating Cost/Effective Shift	\$1,500	\$665
Operating Cost for Non-Productive Shift (Waiting Time)	\$300	\$220
Operating Labor Cost/Shift	\$240	\$120
Capital Cost/Unit (\$ $\times$ 10^6)	\$4.0	\$1.0

Conversion factor: 1 ton = 0.9092 t.

abilities, depreciation, and equipment and labor operating costs change the optimum fleet size.

The methods described using the binomial distributions were developed to quickly and easily analyze various loading/haulage fleet combinations with respect to costs, capacities, and the optimal fleets for comparing ore mining method and equipment choices to another alternative.

There are obvious limitations to the accuracy of this method in its application to a true mining situation. This evaluation, for instance, cannot recognize the difference between the expected output from an operation that has a well-functioning dispatch system as compared to one that is run with batch assignments. This simple method also does not consider the total picture in the sense that there are multiple haul profiles and that the loading equipment may have varying productivities and/or targets according to the material being mined. Only a fully developed mine simulation program can analyze an operation accurately to reflect all of the numerous variables.

The desired accuracy level and the complexity of the operation to be studied dictates the depth of study and method that should be used. The numerical examples illustrate the importance of correctly selecting the best loading and truck fleet sizes.

This case study of equipment proposed in a large strip mine has been given as a good example of comparing non-optimal equipment fleets with other types of mining. If the comparison was made of a dragline vs. a shovel/truck combination and that comparison indicated \$10 million in capital cost savings, the mine owners would have made the wrong choice of mining method and equipment. Therefore, it is of significance to correctly analyze the proposed equipment fleet to ensure that any comparison is valid and the proposed savings of one system over another are real.

Chapter 16.3
IMPORTANCE AND SUMMARY

ERNEST BOHNET

The preceding chapters in this section have outlined various aspects that should be considered in choosing one mining method over another, or one type of equipment over alternative types. In all surface mining operations, where there is a choice of mining method and equipment, the optimal alternative is often difficult to chose because the differences may be within the degree of accuracy of the trade-off study.

The five most important criteria in selecting the best mining method and equipment are

1. Type of deposit, seam or disseminated.
2. Geometry of the deposit and terrain.
3. Total material movement per time period.
4. Projected project life.
5. Geographic location.

Other factors of lesser importance include

1. Climatic conditions and altitude.
2. Availability and cost of a skilled labor force.
3. Environmental restrictions.
4. Slope stability and groundwater.
5. Investment risk and available capital funding.
6. Blasting restrictions.
7. Method of waste rock disposal, external or backfilling.
8. Availability of support services and infrastructure.
9. Availability of electric power vs. the local cost of diesel fuel.

Table 16.3.1 presents a comparison of ore body conditions favorable to the different surface mining methods.

A summary of the advantages and disadvantages of each surface mining method is presented in Table 16.3.2.

In conclusion, many factors need to be taken into consideration in selecting the best method coupled with the most economic fleet of equipment. When there is more than one option, it is best to select the alternative that gives the mining operation the greatest degree of flexibility. Mining is dynamic, and the operation that can adjust the easiest to changing conditions is the operation that will survive future crises.

Successful analysis of the options available ensures that the proposed project will operate at its competitive peak. The ideas presented in this section will hopefully help the mine operator meet the prime objective, that is, the investor's law of conservation.

REFERENCE

Hartman, H.L., 1987, "Surface Mining: Method Comparison and Summary," Chap. 8, *Introductory Mining Engineering,* Wiley-Interscience, New York.

Table 16.3.1. Comparison of Deposit Conditions Favorable to Surface Methods

	Mechanical Extraction				Aqueous Extraction			
Characteristic	Open Pit	Quarrying	Strip Mining	Augering	Hydraulic Mining	Dredging	In situ Mining	Surface Leaching
1. Ore Strength	Any	Any (sound structure)	Any	Any	Unconsolidated, few boulders	Unconsolidated, some boulders	Consolidated	Rubblized or cavable, permeable
2. Rock Strength	Any	Any	Any	Any	Unconsolidated	Unconsolidated	Competent, impervious	Competent, impervious
3. Deposit Shape	Any (preferably tabular)	Thick-bedded or massive	Tabular, bedded	Tabular, bedded	Tabular	Tabular	Any	Massive or thick tabular
4. Deposit Dip	Any (preferably low dip)	Any, if thick	Any (preferably low dip)	Low dip	Low dip	Low dip	Any (preferably low dip)	Steep
5. Deposit Size	Large, thick	Large, thick	Large, moderate thickness	Limited extent, thin	Limited extent, thin	Moderate extent and thickness	Moderate to large	Any (preferably large)
6. Ore Grade	Low	High (assay not critical)	Low	Low	Very low	Very low	Intermediate	Very low
7. Ore Uniformity	Uniform (or sort or blend)	Uniform	Fairly uniform	Uniform	Fairly uniform	Fairly uniform	Variable	Variable
8. Depth	Shallow to moderate	Shallow to moderate	Shallow	Shallow	Very shallow	Very shallow	Moderate to deep	Shallow to moderate

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Table 16.3.2. Comparison of Advantages and Disadvantages of Surface Methods

	Mechanical Extraction				Aqueous Extraction			
Characteristic	Open Pit	Quarrying	Strip Mining	Augering	Hydraulic Mining	Dredging	In situ Mining	Surface Leaching/ Hydraulic Mining
1. Mining Cost (relative)	10%	100% (highest)	10%	5%	5%	<5%	5%	5%
2. Production Rate	Large-scale	Small-scale	Large-scale	Moderate	Moderate	Large-scale	Moderate	Moderate
3. Productivity	High	Very low	High	Very high	High	Highest	Very high	Very high
4. Capital Investment	Large	Small	Large	Small	Small	Large	Large	Moderate
5. Development Rate	Rapid	Moderate	Rapid	Rapid	Rapid	Moderate	Moderate	Moderate
6. Depth Capacity	Limited	Limited	Limited	Limited	Limited	Limited	Unlimited	Unlimited
7. Selectivity	Low	High	Low	Low	Moderate	Low	Low	Low
8. Recovery	High	High	High	Moderate	Moderate	High	Low	Very low
9. Dilution	Moderate	Low	Low	Low	High	High	High	Very high
10. Flexibility	Moderate	Low	Moderate	Very low	Moderate	Low	Low	Low
11. Stability of Openings	High	Highest	High	High	Moderate	Moderate	High	Moderate
12. Environmental Risk	High	Moderate	Very high	Low	Severe	Severe	Moderate	Moderate
13. Waste Disposal	Extensive	Moderate	Minor	None	Moderate	Extensive	Minor	Minor
14. Health and Safety	Good	Good	Good	Good	Fair	Good	Good	Good
15. Other	Low breakage cost; rainfall and weather problems; large-scale best.	Waste-intensive; labor-intensive; high breakage cost.	No waste haulage; low breakage cost; large-scale best.	Restrictive; used for remnant coal.	Unconsolidated deposit; water required; no breakage cost; beneficiates	Unconsolidated deposit; water required; no breakage cost	Unconsolidated deposit; water required; no breakage cost; beneficiates	Unconsolidated deposit; water required; no breakage cost

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Part V
Underground Mining

Section 17 Underground Mine Development

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Chapter 17.0
INTRODUCTION

G.T. LINEBERRY

Underground mine development begins with the positive investment decision to mine and ends with the inception of full-scale exploitation. It incorporates all activities, personnel, and equipment required for creation of underground access to a mineral deposit.

Because of the depletion of near-surface mineral resources, the reduced mobility of large surface mining equipment, increasingly stringent environmental constraints, and the introduction of promising innovations to rock penetration and rock excavating equipment (e.g., continuous miners, rock borers), underground mining is projected to account for an increased share of worldwide mineral production in the foreseeable future. Without proper engineering during the mine development phase, failure of the underground mining project is axiomatic.

A frustration attendant to coverage of the complex area of underground mine development lies not only with the need for treatment of the many disparate specialties within the area but even with the lack of systematization of these specialties in what is recognized as a vital precursor to successful mining. Section 17 offers insights into (1) the basic principles of the planning and design process, including baseline assessment, reserve determination, premine planning, and subsystem design (Chapter 17.1); (2) equipment selection and sizing, with emphasis on excavating and bulk materials handling (Chapter 17.2); (3) mine plant layout, divided into three major subcategories of surface, shaft, and underground plant (Chapter 17.3); (4) construction of development openings, including location and selection of shafts, methods of shaft sinking, and principles of slope construction (Chapter 17.4); and (5) hoisting systems, detailing the design and technical considerations to be examined when selecting components of a shaft hoisting system (hoist, conveyance, rope, shaft, and headframe) (Chapter 17.5).

The broad coverage of this section is intentional. It targets the engineer without a career emphasis in underground mine development, presumptive that involvement in one or two major mine development projects in a career is considered a probable maximum. The underground mine development specialist, however, should find utility in the comprehensive treatment of such an array of multifarious topics as well, some of which undoubtedly lie outside one's own area of technical expertise. For example, although details of shaft location and selection would be familiar to shaft contractors, principles of the analysis of underground excavating equipment might not be, despite the fact that both are mine development concerns. Throughout the section, the inquisitive reader, interested in adding depth to knowledge in any subtopic, is directed to related references.

Chapter 17.1
MINE PLANNING AND DESIGN

GEORGE W. LUXBACHER AND RICHARD J. KLINE

17.1.1 INTRODUCTION

An underground mine planning and design project is unique within the sphere of design engineering in that the primary infrastructure (mine workings) advance into conditions that vary over both space and time. Although generally a mine design moves through the three basic phases—conceptual, preliminary, and final design—the initial design remains the key. Any interpretive error in geology or design error in layout or facilities siting may result in technological, organizational, or operational difficulties, leading to excessive mineral loss or uneconomic operation that could continue through the life of the mine (Pazdziora, 1988). *Underground mine planning and design* has as its goal an integrated mine systems design, whereby a mineral is extracted and prepared at a desired market specification and at a minimum unit cost within acceptable social, legal, and regulatory constraints.

A large number of individual engineering disciplines contribute to the mine planning and design process, such that it is a multidisciplinary activity. Given the complexity of the mining system, *planning* assures the correct selection and coordinated operation of all subsystems, while *design* applies to the traditionally held engineering design of subsystems. A mining operation should be more correctly viewed as a system, because of the diversity of technological processes, facilities, and personal skills required, the large capital invested, the mutual relations that exist between subsystems, etc. The planning process necessitates a systems engineering approach suitable for complex design problems (see Section 6 and Chapter 8.3 of this *Handbook*).

Mine planning can be compared with organizational strategic planning; although the scale may differ, corporate strategy management (Thompson and Strickland, 1989; Steiner, 1979) is very similar to plan development and implementation. A planning process that considers both external (e.g., inflation, technological, industry, and market changes, environmental and political effects, competitor practices) as well as internal forces at work during the mine life will have an improved chance of success and a competitive advantage in the market place. Planning in this context could be considered an entrepreneurial activity while design is more administrative in context.

Although many of the basic principles of mine planning and design have not changed over many years, advancements made in individual systems such as rock mechanics or equipment design have either made or have the potential for making a significant impact on mining. Design guidelines developed as empirical rules based on experience or derived from regulatory requirements are no longer adequate for mine design. A proactive posture regarding mine planning and design is mandated for successful mining in today's complex and competitive minerals market.

Two of the major driving forces in this trend are the application of technical computing and the introduction of a systems engineering approach (see Chapters 8.3 and 8.4). The widespread acceptance of technical computing in the context of computerized mine design has expanded the scope of many older design techniques, yet has permitted the introduction of new techniques. This trend has been driven by the increasing sophistication of readily available software and hardware, augmented by the decreasing cost of hardware (personal computers, workstations, micro and mini systems), the range of software developed (spreadsheets through comprehensive mine planning systems utilizing interactive graphics), and the increasing applications of databases.

The planning process will, in general, move through four steps, irrespective of the design phase: (1) baseline assessment, (2) reserve determination, (3) premine planning, and (4) subsystem design. For a comparison with surface mine planning, see Chapters 13.1 and 13.2.

17.1.2 BASELINE ASSESSMENT

In any mining project, a baseline assessment of all available data precedes any planning efforts. The *baseline assessment* is a comprehensive initial review of all available information on the potential reserve or mine from geographic, geologic, environmental, technical, and economic standpoints. For example, the geographic location of a resource will have a great influence on economics and may dictate the mining method due to equipment and power availability, labor availability and skills level, supplies transport, etc. As the size of the resource is defined, it may be large enough to overcome the negative aspects of its location, but other problems may preclude its development. The major factors to consider in a baseline assessment can be classified as geologic, environmental, geographic, and economic.

17.1.2.1 Geologic Factors

A mine plan must build on the fundamentals of a sound mineral inventory base. It must be remembered, however, that a geologic model is only an interpretation of actual conditions based on data that are, of economic necessity, limited, and subject to the skill of the geologic professional making the interpretation. Exploration efforts are rarely fully achieved because of core recovery, alteration by drilling fluids, water inflow modified by drilling fluid migration, etc., and are often directed toward a specific purpose or goal other than that to which the data are finally applied (Pazdziora, 1988). New tests or additional drilling may be made as a project moves through the design phases that could dictate changes in interpretation, and consequently change the geologic model, the interpretation, or both. Techniques such as geophysical logging, core photography, and petrographic identification, that replace qualitative data of a general and subjective nature with quantitative data where parameters are measured in a systematic and verifiable manner, are often invaluable (Merritt, 1986). Geologic considerations are dealt with more fully in Sections 4 and 5.

Two approaches can be used to handle the uncertainty in the geologic model in mine planning efforts based on a quantification or qualification of risk: 1) accept the geology and develop the plan accordingly, or 2) acknowledge the uncertainty in the geologic model and direct the planning effort to assure sufficient subsystem flexibility to accommodate any potential impact.

17.1.2.2 Environmental Factors

Planning must account for both environmental protection, beginning as early as the initial exploration, and for reclamation.

It is critical that planning alleviate or mitigate potential impacts of mining for two key reasons: (1) the cost of environmental protection is minimized by incorporating it into the initial design, rather than performing remedial measures to compensate for design deficiencies, and (2) negative publicity or poor public relations may have severe economic consequences. If baseline data collection in the field is required, adequate time must be allowed for a comprehensive effort; an average of one to three years of baseline data may be required, although in an environmentally sensitive area, this may increase to as much as five years (assuming an environmental impact statement is required) (Anon., 1987).

Minimum necessary baseline environmental data required for the planning process include (1) overburden analysis, (2) soil surveys identifying topsoil and subsoil, (3) hydrologic studies, (4) a determination of characteristics of surface and ground water, (5) vegetation and existing land use surveys, (6) air quality analyses, (7) wildlife surveys and (8) archeological surveys.

For further discussion of environmental protective measures, see Chapters 7.3, 12.1, 12.2, and 12.3.

17.1.2.3 Geographic and Economic Factors

In the baseline assessment, there are a number of additional factors that must be considered; these can generally be grouped as either geographic or economic. The obvious, as previously used as an example, are geographic: the location of a reserve with respect to existing transportation infrastructure (for movement of equipment to the mine as well as shipment of the product); the type, size, and skill level of work force available to draw upon; the private and public facilities available to support additions to the local work force; climate; power availability; etc. These factors may strongly influence the choice of a mining method (e.g., labor intensive methods such as cut-and-fill or shrinkage stoping if equipment is not readily available, highly mechanized methods if the labor availability or skill level is insufficient or if labor costs are high). Other factors that strongly influence economics must be considered: the political and tax environment, the stability of the existing government, and the availability and replacement of equipment and supplies (Barnes, 1980; Anon., 1987).

17.1.3 RESERVE DETERMINATION

From a mine planning and design perspective, the characteristics of a reserve are as crucial as the reserve magnitude or grade; the depth, inclination, geometry (delineating shape and extent), type and properties of host and deposit rocks, quality, etc. may play a key design role. For example, concerns such as wall rock characteristics or structural features of a deposit strongly influence the choice of mining method and the mine operating costs (Barnes, 1980; Anon., 1987).

17.1.3.1 Criteria

A mineral deposit/resource can be classified as an ore deposit or reserve only when it can be successfully mined at a profit. Although the planning and design process attempts to identify the method of exploitation that allows this to occur, all or part of a deposit may not meet the reserve criteria. The impact of the geographic and geologic environments have been previously discussed. Criteria are also a function of market (demand) and mining technology. Categorization changes occur as the level of inventory and knowledge of a particular deposit increase. Reclassification will also occur corresponding to geographic and economic factors, technological advances in either mining, preparation, transportation, or utilization (impacting market or demand), and depletion. Classification methodology is often mineral- or country-specific. For example, several books have been written on the assessment and classification of coal reserves (Todd, 1982; Wood et al., 1983). From the planning and design perspective, reserve determination can best be expressed as a qualitative assessment of overall potential, expanded to include engineering and economic aspects. For details of mineral resource estimation, see Chapter 5.6.

17.1.3.2 Data Presentation

Reserve/resource data typically are presented in a matrix form, with one side of the matrix indicating the degree of certainty and accuracy of the existence of the ore or mineral, based on the distance from a point of measurement, and the other side indicating the comparative economic recovery viability. Although this format is adequate for data summary, it is of necessity a compromise presentation. Distance categories chosen to represent the degree of certainty may be derived from published sources or government agencies (e.g., Wood et al., 1983); however, categorizations specifically derived for a given deposit using techniques such as geostatistics or depositional modeling may yield more valuable information.

The widespread use of computers often leads to a comprehensive reserve summary of tabulation that is too voluminous for practical utilization. Effective graphics can often maximize communications.

17.1.3.3 Mathematical Methods

Reserve estimation involves taking point data (samples from drilling or prospecting) and extrapolating those data to blocks or grids for calculation purposes. Intermediate steps, such as mapping (determination of the sample point coordinates), compositing of sample data into a common unit or horizon, or determining categorization boundaries (reserve classification, leasehold boundaries, etc.), are often required. The point data are then extended using a wide range of mathematical techniques, of which the most common are (1) polygonal, (2) inverse distance weighing, and (3) kriging. Block size in terms of length and width may be defined on the basis of geologic structure, deposit variability, and data spacing (determining grid size in computer modeling) or may be dictated by mining methodology or quality forecasting requirements. Mapping of the block data and comparison with in-mine sampling is critical to confirm the extension methodology.

17.1.4 PREMINE PLANNING

A mine plan relates a detailed view of a complex process in action at only one specific point in time. The plan constantly evolves as physical changes occur to the existing infrastructure and as new conditions are encountered. Engineering science and mining technology are constantly advancing, while the mine, once constructed, is essentially locked in its initial physical framework. Although fixed assets such as equipment change with time, the basic design remains the same. This fact causes an ever-widening gap between existing and new mines.

When possible, a plan is chosen that will minimize the construction or development time required to get an operation into production. The shorter schedule will allow a more rapid cash flow and consequently better return on the capital investment. Although it may be preferable to drive to the property or deposit extremes and mine back to the plant for ground control or logistics reasons, this may not be economical. Plans are made to access the ore/resource soon after development is started, and to sequence development and production until the deposit is completely mined.

The mining method selection may be a tradeoff between capital investment, time from development to full production, and production costs (Chapter 8.1). As an example, in metal/nonmetal mining, a mining method such as block caving may require extensive capital and development time, yet yield lower production costs. Similarly, in coal mining, a selection of longwall mining may require extensive, high-cost development before lower operating costs, as compared with continuous mining, are realized. Conversely, in either coal or metal/nonmetal mining, a room and pillar mining method might be chosen to yield immediate production, although at a higher mining cost than other methods.

Most plans start with a feasibility study to provide an engineering assessment of the potential viability and minability of the project (Chapter 6.2). This study generally is an overview of the project, making reasonable assumptions about the physical and other key operating factors, to see if the project justifies further effort. Variations in the critical assumptions should also be evaluated to determine the sensitivity of the results to the corresponding input data.

The only way to obtain accurate cost forecasts (Chapter 23.3) for the complete project is to develop a life-of-mine plan for the reserve block, including costs associated with postmining reclamation and final land use. Postmining reclamation costs can be of such a magnitude that the project feasibly may be questionable. Potential damage to the land, water system, or community can preclude mining and can relegate a promising reserve to a resource classification.

Despite the fact that an almost limitless list of factors goes into a mine plan, the following is recommended as a checklist of concerns. Lineberry and Adler (1989) offer another organization of the complex array of concerns for underground coal mine design, as an atypical mine design model.

17.1.4.1 Regulatory and Legal Factors

From the start of the planning process, adequate consideration must be given to regulatory affairs (Chapter 7.3). The cost of compliance may be significantly reduced when taken into account in the design or planning process, in a proactive manner, rather than being addressed on an ad hoc basis as problems develop or enforcement actions occur.

For activities ranging from exploration drilling, taking grab samples, sampling streams for background water data, surveying property and well locations, etc., through mine construction, operation, and eventual closure, various regulatory agencies require the proper permits and approvals. Since the permits and approvals required may be at the level of a federal, state, county, local, regional, or multi-agency, depending on the location, the potential project impact, the regulatory and social climate, and other factors, a comprehensive examination for approvals must be made. These laws and regulations are further subject to continual revision or reinterpretation, necessitating an ongoing review throughout the planning process.

A review of the leases is required to determine mineral and surface rights for any mining venture. In some instances, lease requirements or compliance provisions may place a substantial burden on an operations plan.

As the plan develops, initial designs of the mine infrastructure or compliance plans for key subsystems will need to be submitted to the various agencies for approval. These items, needed before mining can be initiated, may include at least the following: (1) mine layout with projections, (2) strata/roof control plan, (3) ventilation plan, (4) fan stoppage plan, (5) dust control plan, (6) medical/emergency evacuation plan, (7) fire control/mine evacuation plan, and (8) escapeway map/plan. Most of these regulatory requirements must be considered in the plan from the onset. Special care must be taken to insure that the latest regulations, policy releases, court rulings, and proposed rule-making have been reviewed and are being incorporated.

17.1.4.2 Geologic/Geotechnical Factors

From the time an area is determined to have potential value for mining, data collection starts with a determination of the need for additional exploration (Chapters 5.2 and 5.5). Few properties have been so adequately explored that no more information is required. There is never enough information on a deposit to fulfill all design requirements. Ore-body variability can be mathematically defined, and depending on the detail needed, the number of data points and required spacing can be estimated for a required degree of confidence. From the start of the planning process, however, exploration permits are invariably needed to collect additional information to formulate the plan.

An understanding of the regional geology and depositional features forms the basis for every mine plan, although this is often acknowledged only for those mines operating in difficult geologic conditions. Structure and quality variations are mapped and an overall layout scheme meeting the extraction criteria is developed.

As was previously mentioned, economics usually favor the extraction of best-grade material or lowest mining cost areas at the onset of mining to maximize the return on investment and shorten the payback period. While immediate extraction in this manner enhances the economics, it usually creates a compromise in design.

Many geologic and geotechnical factors must be considered for a deposit, depending on the mining techniques under consideration. Among these factors include the identification of ore horizons within a potential minable zone, the quality of individual benches, horizons or plies, and the material type forming the mining horizon roof and floor. The location of oil and gas wells or exploration drillholes that have not been sealed must be determined. Areas of potentially adverse geologic conditions, such as faults, wants, rolls, low cover, or water inflow must be located or defined. Seam or horizon conditions are important also. Of practical interest is knowledge regarding seam/ore hardness and presence of partings/impurities. All of these factors are utilized in the planning process.

17.1.4.3 Environmental Factors

From the start of data gathering and permitting, environmental considerations are important, although benefits from a strictly economic sense may be intangible (Chapter 7.3). From exploration, where core holes must be sealed and the site reclaimed, through plan development, the impacts on the environment must be considered. These impacts include aesthetics, noise, air quality (dust and pollutants), vibration, water discharge and runoff, subsidence, and process wastes; sources include the underground and surface mine infrastructure, mineral processing plant, access or haul roads, remote facilities, etc. If mining will cause quality deterioration of either surface water or groundwater, remedial and treatment measures must be developed to meet discharge standards. The mine plan must include all the technical measures necessary to handle all the environmental problems from initial data gathering to the mine closure and reclamation of the disturbed surface area.

Reclamation plans include many of the following concerns: drainage control, preservation of top soil, segregation of waste material, erosion and sediment control, solid waste disposal, control of fugitive dust, regrading, and restoration of waste and mine areas. The plan must also consider the effects of mine subsidence, vibration (induced by mining, processing, transport, or subsidence), and impact on surface water and groundwater. These environmental items often dictate the economics of a planned mining operation and determine its viability.

17.1.4.4 Technical Factors

From an engineering perspective, the technical area is the most extensive in the overall plan (see other chapters in Section 17). Plan detail is defined using the data from regulatory, geologic, and environmental considerations. All information is used to develop each part of the plan, utilizing an iterative process to account for the many interrelationships that exist. The layout of the mine is determined by the size and shape of the ore deposit. The depositional features and conditions are used to calculate the mine reserves. After the mine area is delineated, access development to the deposit is considered. Access can be by vertical shafts, inclined slopes and drifts, or horizontal entries and adits. The relative size and production levels of the mine area will dictate the number and size of the access openings made. The larger and more extensive the area, the more complicated the plan will be. The optimum mine size or production level can be determined from a combination of mining area, workable reserves, and daily production. A small mine may not generate sufficient income to justify the cost, while a large mine may be penalized by high fixed costs incurred in the later years, primarily due to the cost of maintaining the infrastructure.

The size and technical parameters will form the conceptual basis for the plan; from this, the detailed plan takes shape. Each item is defined: key assumptions, physical factors, equipment, facilities, infrastructure, and transportation are detailed and modified as each item falls into place, dependent on the base data and "best-fit" technical determinations.

SURFACE FACILITIES. The productive capacity of the proposed operation and the reserve size determine the size and site placement of the facilities; extensive, large-capacity facilities need a correspondingly larger surface area than similar type smaller facilities (Chapter 7.2). Consideration must be made for access, extraction, removal, and storage of the ore, the physical needs of the workforce, and the operational needs of the facility (Anon., 1987). Transportation of the product may be by rail, truck, water, or a combination of modes. The ore may have a high reject or waste percentage and is consequently too costly to transport for processing; partial processing at the mine site with final processing at another location may be preferential. The processing facilities could be designed as a closed circuit system, recycling water internally, or could use a slurry system, settling out fine refuse. If slurry disposal is required, the facility design must insure adequate life. Land acquisition for disposal areas must be addressed. Water (potable and plant makeup) and power requirements must be considered. Dust, noise, safety, and layout efficiency are other design considerations. Future access as the mine is extended further beyond the initial development area will require life-of-mine planning to identify the location and function of the ancillary facilities in later years.

PHYSICAL FACTORS. Where the dimensions of the deposit determine working height (e.g., coal, trona), physical factors relate primarily to deposit height. Isopach mapping is used in these cases to develop the best mine layout, often giving consideration to the number of benches to be mined or the portion of the seam best matched to market requirements. The reserve can be constituted by a single vein deposit or by multiple veins, splitting and joining at different places throughout the mine area. Several levels may be operating simultaneously to produce the required product quality. The sequence of extraction can be important for quality and for maximizing the reserve recovery. The proper sequencing of mining in a multilevel deposit or multiple seam situation can greatly increase the recovery and can avoid mineral losses from mining sterilization. Quality parameters for each mining area must be considered, so that areas not meeting market criteria can be blended or avoided. Poor mining conditions must be factored into the analysis to account for changes in productivity rates and mine costs.

EQUIPMENT. The dimensions of the deposit and the hardness of the mineral are the primary determinants of the types of equipment needed (Section 9). Additional factors include seam or working height, mining dilution limits, production rates, and property extent. Ventilation, size constraints, regulations, and floor pressures may impact the choice of diesel- or electric-powered equipment. A large flat-lying deposit may allow the use of longwall mining equipment. The floor condition plays a big part in the equipment type.

Desired product size may also influence equipment selection. For example, the washability analysis for a particular coal seam at various size fractions may suggest the choice between continuous mining (fine-particle generation) and conventional mining systems (coarser product) for room and pillar mining, using preparation (washing) as a criteria. Consideration of other factors, such as equipment productivity, however may negate this particular criteria.

Equipment overhaul schedules should be developed to assure sustained productivity rates. Equipment replacement schedules should consider the incorporation of new technology as it becomes available.

SUPPORT SYSTEMS/INFRASTRUCTURE. Subsystems critical to, but not a part of, production at the face form the support systems, installed in the mine entries/drifts/levels as mining proceeds. This network forms the mine infrastructure.

All parts of the system must be evaluated for capacity and availability, and must be matched. Most systems are developed in series; if one part of the system fails, the whole system must be halted until the failed part is corrected. Although series systems are generally used out of necessity (due to the costs of redundant components or parallel systems), low availability cannot be accepted because of its potential negative profitability impact. The support systems are designed to be as maintenance free as practical, while minimizing capital and labor intensity requirements (although these two items are, in turn, classical trade-offs). They must have designed into them (albeit at a potential increased capital cost) the flexibility needed to serve the expected production levels. Given the number of support systems and the potential for delays outby the face area, the support systems may impact productivity more than the face systems.

Transportation—Transportation encompasses provisions for the movement of materials, personnel, and equipment into and out of a mine. Workers must reach their designated work area in a prompt and expeditious manner. Supplies must get to the proper area for use in the system before the need becomes critical. The equipment itself must be transported through the mine to get to the working area. And, of course, the ore or mined material must get from the face to the processing facility. A smooth flow of the transport systems must be ensured in a successful mine design.

The greatest design work is expended on the transport of ore. Usually other transport systems are dictated from that system. The haulage system must be compatibly sized, from the face extraction cycle to the facilities outside. Productivity rates must be matched, so that a highly productive face is not hindered by the capacity and availability of outby transportation. Many mines have failed or not achieved their profit potential because proper consideration had not been given to system capacity compatibility.

Regardless of how well capacities are matched in any materials handling system, a “bottleneck” will likely occur. The key is to keep the most expensive links in the system working at or near 100% capacity to maximize the return on net assets. The need to decouple the haulage system must be considered. Surge systems are weighed against large-capacity haulage units. Surge storage units (e.g., bins, bunkers) can serve both as a decoupler and as a ratio feeder for improved materials handling.

Future technology must be considered also. Is there new technology being evaluated now which will be available soon? It may be prudent to design haulage systems to meet capacity requirements from an improved production basis or from planning basis where large capacity longwalls will be installed later. The equipment from the face, through the intermediate haulage system, to the outside must be given due consideration.

The workers may get paid for portal to portal work, for 8- or 10-hour shifts, possibly changing at the face. Allowances must be made for overtime work. The various classification grades and possible contracts with labor groups must be considered. The fastest, most efficient system must be used, so that the non-face time will be minimized. Not only does this give more production time, but it also maximizes the utilization of costly labor. Clearance must be provided, both to the sides and overhead, for safe movement of personnel and equipment.

Manpower—Staffing of the system is a function of the required production level, selection of equipment, and technical objective selected to accomplish the objectives. Typically the manpower level is inversely related to the relative level of capital spending (in turn, usually a function of reserve size). Adequate personnel must be provided to allow the system to function properly, at the design utilization levels; personnel includes the supervisory workforce as well.

Consideration must be made for support staff levels (e.g., administration, engineering, financial). Centralization or a mine-based staff each may be more effective depending on the particular circumstances. The physical location of the mine must be considered. Are the environmental, engineering, personnel, and other support staffs a part of this staffing level? These considerations all become part of the staffing effort.

Mine Power—The mine productive capacity and mineral processing requirements govern the electrical power needs of the operation (Chapter 12.4). The regional distribution systems should be evaluated to determine if the power is available or whether a new distribution system must be designed and built. These systems must be of adequate size for the life of the mine. They must also be easily maintained and must provide adequate reliability. The distribution of power is designed from the power supplier to the location in the mine where it is being used. Adequate capacities and safeguards are considered so that the total system will be available when needed. In some areas, both on the surface and underground, backup systems may be needed, Also, in many mine areas, backup systems are being designed where different forms of power are being utilized, including solar radio transmitters, methane power generation, and hydroelectric power.

Mine communications systems are a part of this also (Chapter 12.6). Several types may be utilized for wired or wireless. Backup systems are often used. Many systems are now a part of the telephone system so that communications are complete to remote offices all over the world, from the mine face. Couple this with the mine monitoring systems and computers, and the design variables are almost unlimited. The proper communication of voice and data is vitally important. Timely and accurate documentation of the mining systems status can lead to very efficient operation and utilization of the mine resources.

Water—The supply of water is very important to the operation of the mine and preparation facility and is a subsystem in itself with its own distribution system from source to face (Chapter 12.1). Both process water and potable water must be supplied. There may not even be a source nearby, requiring that one be designed for in the premining phase. Public sources may be available, but they must be adequate for the amounts needed. Wells might have to be drilled, or impoundments made, to get the needed supply. Various mine systems need the water for cooling, dust suppression, fire fighting, processing, and personnel needs.

Services—The mine plan support systems also include distribution of dust suppression material, hydraulic oil, and other supplies (Chapter 12.7). Most mine equipment requires good, clean hydraulic oil for proper operation and continuous service. This system must be well thought out so that it is properly implemented. The use of diesels will require even more oil distribution systems, properly installed and maintained. The dust suppression materials, usually rock dust, in bag or bulk form, must be distributed properly. Bulk packaging of supplies for work areas will minimize handling, leading to less labor intensive supply systems.

Ventilation—After most of the other factors are laid out, the ventilation is designed to provide the mine’s life support system (Section 11). First consideration is for providing clean respirable air to the workers. The dilution of contaminants is next. In other cases, air can be used for cooling also. The mine life must be considered when doing the design. Changes in the mine extent or size will determine what areas need to be serviced. Contaminants (e.g., dust, radiation, and methane) must be diluted to safe levels and/or carried out of the mine. Areas may need to be sealed or isolated from the system to make the system efficient.

Mine layout is dramatically impacted by the ventilation system. Proper airflow requires proper sizing, location, and numbers of airways keeping in perspective ground stability requirements. Design considerations often require large-sized and high-horsepower fans for the proper utilization of the support systems. Combinations of holes, shafts, and adits are often utilized. Minimum and maximum velocities and quantities are often specified by the regulations and mine conditions. Petitions for variance are often needed because the regulations do not cover all circumstances found in the mines.

Development sections are sometimes harder to ventilate than production sections because they are usually driven into virgin territory, where conditions are not as well known. In many cases, the development unit is extended a long distance from the primary aircourses and adits, necessitating high pressures to provide the proper air quantity and quality. Production units can also be troublesome where high production causes high liberation of contaminants. In these cases, high quantities are required for proper ventilation.

17.1.4.5 Mine Closing and Reclamation

After the deposit has been completely mined, the mine area must be cleaned up and returned to approximately its original condition. Permits require bonds to be set for protection against not completing this reclamation. Sinking funds are usually made a part of the mine costs to cover these reclamation costs. Openings are filled and sealed, waste piles are covered and revegetated, and structures are removed. For details of mine closures and sealing, see Chapter 8.7.

17.1.5 CONCLUSION

The mine planning and design process is detailed and complicated, requiring comprehensive knowledge of geology, laws and regulations, equipment specifications, mechanical, electrical, and civil engineering technology, and general mining engineering. Throughout the process of planning, all the factors discussed above are used in a methodical way to arrive at a baseline assessment, to develop an initial plan, and then to evaluate various scenarios.

A plan is developed in a window of time that is stopped for the moment, and finalized, based on the then available data. When the plan is reviewed tomorrow, it is quite likely that new data will be found, or that different emphasis will be put on existing data, such that changes will be required to the plan.

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Chapter 17.2
EQUIPMENT SELECTION AND SIZING

G.T. LINEBERRY AND A.P. PAOLINI

Equipment selection and sizing is a major mining engineering activity. It follows the positive investment decision to mine (Section 6) and begins with selection of a mining method (Sections 8, 21, and 23). Although equipment selection begins during the formulation of a "milestone diagram," in which the minable reserve is parceled out into zones distinguished by method of mining, suitability and production analyses for mining equipment are not conducted until after layout and sequencing are accomplished and before scheduling is undertaken (Lineberry and Adler, 1989). It is therefore apparent that equipment selection and sizing is a vital activity for a comprehensive, balanced underground mine design.

At the heart of mining is the excavation and handling of bulk materials. In fact, the primary objective of mining is to win ground, convert it into a fragmented solid, and then move this material to a suitable location for processing (Adler, 1984). In accomplishing this, the excavating and bulk handling system sets the pace for all other systems (ground control, life support including ventilation, and support services including power distribution and maintenance). It further determines suitable equipment type, equipment size and horsepower, and productive capabilities, and therefore to a large extent influences layout, sequencing, and scheduling. For these reasons, the selection and sizing of equipment, whose purpose is for the accomplishment of excavating and bulk materials handling, is emphasized here. Details for the selection and sizing of materials handling devices are subordinated, since these are treated in Section 9, Production Operations. Offered here are guidelines for the selection and evaluation of underground equipment, both mobile and fixed, as well as a standardized procedure for their analysis.

This chapter combines qualitative descriptions of many common underground excavators and handlers with an attempt to classify each as an aid to their proper selection and evaluation. Included in the classification system for completeness, however, are many common pieces of surface mining and construction equipment. Limits and ranges are offered, recognizing that these are valid only as frames of reference in a market area that is undergoing rampant innovation as operators and manufacturers seek higher productivity and enhanced safety. These goals are sought through technological improvements (e.g., coal interface detection, on- and off-board navigation, robotics, autonomous vehicle technologies) and improved management strategies (e.g., artificial intelligence, decision support systems, management information systems).

Major topics to be included here are (1) the development of the viewpoint of the primacy of excavating and bulk handling in mining, (2) a description of a recommended equipment classification system, (3) a review of the characteristics of the most common underground excavating and handling equipment, with major recent and impending innovations noted, and (4) an overview of a procedure for determining the performance of excavating and bulk handling equipment. Recommendations are also given for improving the selection and evaluation procedure.

Because there is little agreement as to what specific points should be considered, and even less on what procedure to follow, much of the material presented here combines established and time-honored principles with a new structure and framework. Specific goals are (1) the development of excavating and bulk materials handling as an area that is distinctively mining's, (2) introduction of a classification system that provides an organizational structure upon which to build, and (3) standardization of a procedure for analyzing excavating and bulk materials handling equipment.

Table 17.2.1. Dimensions and Symbols

Dimension	Notation	Query	Symbol (Suggested)	Description
Matter (material)	M	What?		Substance or agent in question
Space	L	Where?		Locate, delineate, measure, represent, classify
Time	T	When?		Specific time frames (past, present, future), or increments (Δt)
People	—	Who?		Skill, ability, experience, safety training, motivation
Money	—	How much?	$	Monetary value, cost, profitability
Purpose (major)	—	Why?	?	Reason for doing job or element of it (safety, economy)

Source: Lineberry and Adler, 1988.

17.2.1 BASIS FOR EQUIPMENT DIFFERENTIATION

17.2.1.1. Organizing Principles

Essential to proper selection of mining equipment is an appreciation for and an understanding of the unique conditions under which excavating and handling equipment is employed. In an integrated and intensive effort to define these mining conditions (termed encumbered space), the introduction of a standard set of engineering dimensions is appropriate. The complete range of engineering dimensions, which efficiently encompasses a mass of diverse properties, is shown in Table 17.2.1 and provides an essential checklist (Lineberry and Adler, 1988). These six key engineering dimensions not only aid in the characterization of the highly adverse and distinctive working conditions associated with underground mining but also assist in formulating a consis-

Table 17.2.2. Total Systems Context

Systems	Description
Ground Control (esp. underground)	Support, caving, closure, bursts, subsidence
Excavating and Handling	Rock penetration, fragmentation, bulk solid movement, storage, transfer
Life Support (esp. underground)	Ventilation, drainage, illumination, personnel safety and health
Normal Support (logistics & environment)	Power, supplies, repair, maintenance, surveying, construction, management, personnel, environment

Source: Adler and Lineberry, 1988.

tent methodology for equipment selection and sizing in encumbered space.

17.2.1.2. Mining's Uniqueness

To develop a consistent and replicable procedure for selecting and sizing underground mining equipment, the basic and unique attributes of mining engineering must be determined, described, and then appreciated. When systematically compared to other engineering fields, mining engineering is distinctive in (1) having a total systems concern, (2) dealing with encumbered space, and (3) requiring *full-spectrum practice.* Each of these, respectively, corresponds to the requirement of specifying tools and activities, conditions, and management. By considering an extreme case situation, underground mining variables which are often too subtle or ubiquitous to notice can be more readily identified.

Thoughout this development, the *mine face* will be emphasized because it is the focal point of all activities. Here, tools are marshalled, concentrated, and applied under the worst conditions. The remainder of the mine extending back to the surface, including worked-out areas, is termed the *mine plant.* The mine plant more closely resembles a normal, structured, manufacturing plant and is easier to operate and maintain. It is often the shared professional jurisdiction of mining, mechanical, civil, and electrical engineers. Only the mining engineer is willing and able to deal with the unstructured mine face with its difficulties, inherent hazards, and potential for disaster. The underground mine face has been termed "hostile" (Zegeer, 1986) and characterized as being "dirt-infested" (Sitek, 1980; Lineberry and Adler, 1988).

TOTAL SYSTEMS CONTEXT. The total systems concern ensures that vital excavating and bulk handling activities will occur and be assisted by the necessary synchronized supporting sytems: ground control, life support, and normal support, as shown in Table 17.2.2. In the aggregate, all of these systems are so complex that mining can be compared to a busy city, with concern for water, light, power, communications, transportation, supplies, sewerage, and construction. Because mining also requires rapid advance and retreat, an unprecedented demand for mobility exists, particularly at the face.

Excavating and Handling System—In accomplishing mining's mission, the excavating and bulk handling system sets the pace for all other systems. A fundamental choice must be made, especially at the face, between using mobile equipment or fixed machinery (Adler and Lineberry, 1986). Face mobility is exemplified by the trackless equipment characteristic of room and pillar mining, coupled with a possible switch to fixed machinery, such as a belt, further back in the plant. By way of contrast, longwall mining, with continuous haulage, is considerably more rigid. When using mobile equipment, the mining engineer is indispensable; with fixed machinery, the mechanical engineer assumes a more important role. Even with fixed machinery, mobile equipment is still necessary to prepare, trim, support, grade, and clean up.

It is important to note that in addition to milling and digging, excavating also employs the mechanisms of blasting, boring, ripping, scooping, caving, and impacting. All are carried out in the geologic mass, which, in combination with the tool, forms the *job-end.* Engineering dimensions of matter, space, and time are emphasized.

The bulk solids produced must be handled by the appropriate job-end tool (e.g., box, blade, belt). When the bulk solid lies in a muckpile or stockpile, loading is first necessary. The bulk solid material, which is also best described in terms of matter, space, and time, is also termed "loose," "fragmented," or "non-cohesive." It can be further processed by crushing, screening, sizing, and blending.

Ground Control System—Ground control is vital in all underground mining, especially of coal. It provides control of roof falls, closure, bursts, and subsidence and significantly assists excavating and handling. Because it depends upon poorly defined and highly variable geologic conditions, the ground control analysis is often questionable. Underground mining methods have been traditionally classified on the basis of ground control as open (self-supported), supported, or caving. When recovering pillars, a rapid in-out approach is favored, facilitated by highly mobile equipment.

Life Support Systems—Life support systems are essential in underground mining. They consist of (1) ventilation, (2) drainage, (3) illumination, and (4) personnel safety and health subsystems. In coal mining, ventilation demands major attention because it radically impinges on the face operation and influences the overall mine layout. As with ground control, the engineering of these systems depends on poorly defined geologic data (e.g., rock permeability) and uncertain engineering data (e.g., friction factor), requiring the practiced judgment of mining engineers.

Normal Support Systems—Normal support systems are required for surface as well as for underground mining. They provide (1) power, fuel, and supplies; (2) maintenance and repair; (3) management, communications, and personnel transport; (4) surveying, construction, moving, and cleanup; and (5) environmental protection. Hazards associated with some of these systems include exposure to high-voltage electricity and handling of explosives, volatiles, solvents, and cutting gases.

ENCUMBERED SPACE. To mine safely and efficiently, working conditions must be systematically defined. Their uniqueness distinguishes mining from other engineering professions. The worst case in underground mining is termed *encumbered space.* Working space is often inherently hazardous, transitory, and expensive. It is also tight, distorted, congested, isolated and inaccessible, of poor quality, and deteriorating. These adverse conditions endanger personnel, damage mobile equipment, and affect all activities, even on the surface. An integrated and intensive effort has been made to define mining conditions (Lineberry and Adler, 1988). Only a brief summary can be presented here.

General—Encumbered conditions are those which "impede motion or activity, as with a burden." They are singular to mining and convert normal difficulties into hazards, and hazards into potential disasters. Their distinctiveness and significance is sufficiently great to justify an engineering career emphasis.

Table 17.2.3. Causes of Encumbered Space

Causes	Description
Primary (Inherent)	Geology; geography; underground, ground control
Operational (Induced)	Total systems activities, overlapping (esp. excavating and handling), people
Worse-case Situation	Superposition of above at mine face (esp. disaster potential)

Source: Adler and Lineberry, 1988.

Encumbrances primarily impact on the excavating and bulk handling system, but all systems are involved.

The mining situation contrasts sharply with that in heavy construction, even with tunneling. In mining, the value of the excavated material is often just marginally higher than the cost of mining. In construction, the value resides in the resulting access structure and is several times greater than the cost of excavating. In mining, the life of the opening is brief and intense compared to that in construction. Consequently, initial costs in mining must be kept unusually low, especially those for ground control. Full mineral recovery is sought, and dilution and deadwork must be minimized. All of these factors intensify hazards and mandate that mobile equipment be flexible, versatile, and highly maneuverable.

The full impact of encumbered space does not really exist in a manufacturing plant nor in construction with its alternative sites and routes; it exists only in mining. At the mine face, conditions are sufficiently unprecedented so that traditional engineering no longer suffices. As conditions worsen, the operation slows down. This allows even further worsening, a downward spiral to failure. The underground structure can be dramatically likened to a large collapsing edifice, still occupied and undergoing demolition from within. Potential causes of accidents and even of disasters abound. Only the mining engineer can hope to compete with these severe constraints in time and space.

Inherent Causes—The causes of encumbered space, as shown in Table 17.2.3, are the result of natural conditions and of an underground workplace. A geologic mass is often practically impenetrable and highly variable, making its properties extremely difficult to predict. Natural conditions constitute the primary set of independent variables and are virtually unalterable. A material description of any geologic mass must include the rock type, geologic structures, contained fluids, and rock temperature and pressure. Fluids, gases, and water are often hazardous, and rock temperature discomforting. Subregions composed of uniform material properties must be located, measured, and described. However, because of impenetrability, both material and spatial descriptions are frequently inadequate.

Underground mining can pose severe ground control problems. In coal, the normal quasi-stable roof span is only about 15 ft (5 m) and, therefore, very restrictive. Pillars are often large, possibly 120 ft (35 m) for coal, and cause major detours in equipment routes. Deterioration often results in a progressive decrease of structural integrity, reinforcing the rapid in-out requirement, especially during secondary recovery of pillars.

Severe ventilation and drainage problems can result from contained fluids. Pillars serving as seals can deteriorate with time, compromising their effectiveness. The human environment is usually poor. Vision is limited due to darkness, fog, and obstructions. It can be damp, hot or cold, noisy, gassy, slippery, grimy, and dirty.

In closely following the deposit, especially when employing blasting, poor quality access space results. Cross sections are often misshapen, tight, and rough, while the centerlines are lengthy, circuitous, and abrupt. Underground mine surveying reflects these distinctive problems. Mine faces are dead-end and are often remote and inaccessible. Haulway surfaces typically are littered, dirt-infested, slick, rough, weak, or heaving. Since the bulk solid is raw, it often contains oversized fragments, resulting in spillage and interference. It can also be leaky, sticky, abrasive, and dusty. Water exacerbates many of these conditions and accelerates deterioration. The disposal of any waste or unwanted water is a major problem.

A more detailed approach to organizing the inherent components of encumbered space (Lineberry and Adler, 1988) describes the geologic mass, bulk solid, ground structure, running surface, man and machine environment, and centerline profile in terms of the five basic engineering dimensions. These components are then matched to major equipment components, including (1) job-end (to excavate and handle), (2) running gear (to provide mobility and/or reactions), (3) chassis and cab (the base for all components, the operator, and attachments), and (4) power and drive train (energy source and controls).

Operationally Induced Conditions—In order to cope with the many intrinsic adversities of the inherent state previously noted, the total systems are stuffed into already tight space to yield induced conditions. Each individual system to some extent bears a deliberate one-to-one relationship with an inherent component. Ground control heavily influences the ground structure; life support relates to the human environment; and, clearly, excavating and bulk handling incorporates the geologic mass, the bulk solid, and the running and reaction surface. The normal support system and the centerline profile remain ubiquitous, but in all cases, interactions occur. The addition of any remedial system is partially self-defeating, since in itself, it occupies valuable space, causes wear to running surfaces, deteriorates, and, in turn, must be serviced.

Four induced components of encumbered space are recognized as problematic: (1) interference, caused by adding moving equipment (e.g., bolters); (2) congestion, due to introducing stationary facilities, installations, and stores; (3) traffic, resulting from the keen competition for space by multiple haul units (e.g., ramcars); and, worst of all, (4) high labor intensiveness, exposing personnel to serious jeopardy in tight, short-lived, but very active areas. Special safety categories such as “struck by” and “caught between” emphasize the potential effects of these induced components of encumbered space.

Congestion in ground control can be caused by props, cribs, sets, and trusses. Bolts and grouting, being internal or superficial, are excluded. Short spans and large pillars have already been considered as inherent conditions. Roof bolters can prolong interference, especially when roof conditions are poor. Excavating and handling equipment for other parallel systems and for serial elements add to interference, while loading and transfer points promote congestion.

Ventilation near the face causes extreme congestion, with curtains, stoppings, ducts, and fans. Drainage requires pipes, ditches, sumps, and pumps. Water and firelines must be included, along with suitable escapeways. Rock dusting can offer interference also. For normal support, congestion is created by distribution systems for power (cables), water and compressed air (pipes), as well as for supplies and dumps. Explosives, volatiles, and cutting gases require special handling and storage (magazines).

Numerous personnel in constrained geometries and in nonideal conditions further complicate underground mining operations. Low spatial volume per worker ratios—with the lowest in the Appalachian Coal Basin of the United States—contribute to an increased hazard potential. Because activities are concen-

Table 17.2.4. Full-spectrum Practice

Phases	Description
Plan	Conceive, study, investigate
Design	Analyze, organize, synthesize
Implement	Install, construct, staff
Operate	Run, function, supply, maintain
Control	Manage, monitor, vary, alter

Source: Adler and Lineberry, 1988.

Table 17.2.5. Alphabetical Inventory of Equipment Types[a]

Airplane	*Grader*
Auger	*Hoist* (vertical, incline)
Backhoe	Hydraulic (pipeline)
Barge	Hydraulicking
Bin	Impactor (hydraulic)
Blasthole loader	Loading machine (ramp)
Borer (raise, shaft, tunnel, reamer)	Longwall (LW) and Shortwall (SW)
Breaker	Mobile crusher, feeder, conveyor
Bucket chain excavator (BCE)	Muckers (shaft, tunnel)
Bucket wheel excavator (BWE)	Orepass
Cable scraper (slusher and tagline)	Pipeline (hydraulic, pneumatic)
Chute	*Railroad* (RR)
Clamshell	Ramcar
Compactor	Reclaimer
Continuous miner (CM) (road header, boom)	Ripper (dozer, LW-plow)
Conveyor (belt, chain, etc.)	*Scoop* (FEL type)
Crane	*Shovel*
Crushers (fixed and mobile)	*Shuttle car* (S/C)
Cutting machine (undercut, top, side, universal, rock saw)	Side bucket wheel excavator (side-BWE)
Demolition ball (impactor)	Stacker
Dozer	*Tractor scraper*
Dragline	Tramway
Dredge (bucket chain)	Trencher
Drill	*Truck*
Dump (tipple, ramp, grizzly)	
Feeder (with or without breaker)	
Front-end loader (FEL or LHD, load-haul-dump; many variants)	
Grade-all	

[a] Abbreviations given, with major types underlined.
Source: After Adler and Lineberry, 1986.

trated at the point of deepest penetration, it is not surprising that most underground mining accidents occur within 25 ft (8 m) of the face.

FULL-SPECTRUM PRACTICE. Engineering spans a spectrum from conception, through planning, to operating, as shown in Table 17.2.4. Different engineering disciplines stress different phases: civil engineering concentrates on plan and design. Electrical, mechanical, and chemical engineering shift towards design and control, with the latter done virtually automatically. Mining and industrial engineering, both of which are labor intensive, must shift even further to operate and control. For mining, this shift is intensified by geologic variability, geometric constraints, and severe market fluctuations. The mining engineer is one of the few practitioners who often "flies" his own design. Full-spectrum practice also requires applying administrative techniques, which already receive priority in many smaller mines. The transition from engineering to management remains awkward, although their close ties are well recognized.

In this discussion of equipment selection and sizing, engineering is emphasized over administration, with the analyses being conducted in the early phases of full spectrum practice. Operation and control will not be addressed, except as it applies to the safe and efficient use of underground excavating and handling equipment. Most coverage will focus on the use of performance analyses to properly select and size common pieces of underground equipment.

17.2.2 EQUIPMENT CLASSIFICATION SYSTEM

All types of mining and reclamation equipment can be systematically classified in order to reduce a vast quantity of information to manageable proportions. This classification serves to control the development of a market area undergoing rapid, even rampant, changes and assists in the safe and economical use of equipment. It also fosters an integrated view of what are often considered fragmented, disparate specialties.

An efficient and standardized classification system is the starting point for most analyses. However, an examination of the current classification systems for underground equipment indicates serious limitations. While the proposed classification is primarily intended for mining engineers, other engineers will also benefit. Many factors are considered in the classification. A cursory knowledge of the equipment is essential for its use.

17.2.2.1. Background

ALPHABETICAL INVENTORY. To fully appreciate the range and complexity of equipment types, an alphabetically-arranged inventory is presented in Table 17.2.5. Over 50 types are included, with their commonly accepted abbreviations noted. The term *equipment* often refers to mobile units (e.g., dozers, shuttle cars), while *machinery* refers to fixed assemblages (e.g., conveyors, borers), a distinction that is often blurred. Although this section is devoted to selection and sizing of common underground equipment, presentation of a complete inventory is preferred because performance analyses of all mining and reclamation equipment follow common basic principles.

NECESSARY REQUIREMENTS. In order to properly analyze excavating and bulk handling problems, the objectives, conditions, activities, and tools must be clearly stated (Adler, 1984). The objectives are either to excavate, to handle, or to do both. *Conditions* are the natural setting or environment in which mining objectives are being pursued. The *activities* are how the tool is used, maneuvered, or manipulated. The tool itself must be designated as to *type* (e.g., the continuous miner is a *type* of integrated excavator; a tunnel boring machine (TBM) is a *type* of rapid excavator).

CURRENT CLASSIFICATIONS. Existing systems are highly simplistic and usually consider only one condition: surface vs. underground, hard vs. soft rock, hoisting vs. transporting, gravity vs. mechanical flow, mining vs. construction use, and so forth (Apple, 1963; Carson, 1961; Nunnally, 1982; Anon., 1958; Anon., 1987). An industrial engineering classification for materials handling equipment distinguishes three types of equipment: (1) fixed-path types (e.g., conveyors), (2) limited-area types (e.g., cranes), and (3) wide-area types (e.g., trucks) (Peurifoy, 1979). This system, together with another developed for loader selection (Anon., 1958), are precedents for the proposed classification.

PROPOSED CLASSIFICATION. The most apparent and striking visual distinction between equipment types is their *degree of activity:* mobile equipment (e.g., scoops) vs. fixed-base equipment (e.g., TBMs). The differences are clearly self evident. An intermediate distinction covers equipment that remains fixed and in

Table 17.2.6. Major Equipment Groups Related to Types[a]

Group	Types
Mobile loaders and excavators	Dozer (and ripper), tractor scraper, FEL
Rotating excavators	Power & hydraulic shovels, backhoe, dragline, grade-all, clamshell (crane)
Integrated excavators	CM, BWEs, (and side-BWE), BCE trencher, auger dredge, cutting machine
Rapid excavators	Borers (raise, shaft, tunnel), reamer auger
Ground preparation	Drill/blast (hole loader), impactor (hydraulic or ball), ripper, cutter, hydraulicking
Loaders and muckers	Tunnel (ramp), shaft
Mobile conveyors	Modular (extensible), shiftable (in-pit), bridge (continuous handling)
Conveyors	Belt, bucket, flight, spiral, high-angle conveyors; hydraulic and pneumatic pipelines

[a] See Table 17.2.5 for meaning of abbreviations.
Source: Adler and Lineberry, 1986.

place while excavating but is mobile when tramming between sites (e.g., shovels). A second dimension is added to the classification by the *objective of activity:* whether it is for excavating (e.g., continuous miners) or for bulk handling (e.g., trucks). Again an intermediate distinction includes equipment doing both (e.g., LHDs loading from muckpiles or from stockpiles) or equipment used only for short-distance handling (e.g., shuttle cars). This matrix of degree of activity vs. objective of activity enables all equipment to be located systematically.

17.2.2.2. Classification and Value

An initial consolidation of over 50 equipment types is shown in Table 17.2.6, combining many types into well-accepted groups. Integrated and rapid excavators are distinct because they simultaneously excavate, load, transfer, and dump. On the basis of these groups and the remaining individual types, a comprehensive equipment classification is offered in Table 17.2.7, with abbreviations used whenever possible. Additional refinements concerning the relative location between types could be made within each block. In fact, an attempt has been made to supplement the classification system with greater detail, based on conspicuous visual distinctions within each equipment type (Adler, 1989a). By adopting classes, groups, types, and kinds, at successive degrees of refinement, a taxonomy has been developed based on clear visual differentiation, or on a morphology. Although promising, this refined classification has not been critiqued and standardized. Furthermore, uncertainties remain in the proposed taxonomy, especially below the level of type, and graphical deficiencies are still present in the morphology.

Production analyses of equipment with long cycle times (> 2 minutes) are notable since they use a *tabular duty cycle* approach. Short cycle times (< 2 minutes) are usually found in tables (with appropriate correction factors) or are time-studied on the job. Hoists have simple, uniform, precise, and repeated short cycles and employ a *graphical duty cycle* approach. Slushers and some tramways can be handled either way. In general, analysis of equipment with minimum operational variability and short cycles (e.g., hoists and rail haulage) favors a graphical technique.

17.2.2.3. Exceptions and Future Trends

Clear classification of equipment is difficult. The front-end loader acts sometimes as a tractor shovel (excavating), at other times as a load-haul-dump (LHD) (both excavating and handling). Some apparently fixed assemblages, such as the slusher or hoist, have considerable internal relative motion of the job-end and of the overall center of gravity. It is this motion that is deemed cyclical, locating these types in the intermediate rather than fixed classes. Judgments as to relative mobility are made in locating the rotating excavators alongside integrated ones. The bucket wheel excavator and boom-type continuous miner can be considered either rotating or integrated. Also fixed-base continuous excavators must periodically be advanced (borers) or the job-end direction reversed (longwall). Therefore, these excavators have broad secondary cycles.

It is often difficult to distinguish between equipment types that are intended exclusively for surface or for underground use, because variants have appeared of both. The continuous miner has been used infrequently on the surface, where there is a recent variant, the surface drum miner (e.g., Easiminer). The longwall shearer has been proposed for the secondary mining of coal

Table 17.2.7. Classifying Excavating and Bulk Handling Equipment[a]

	Activity		
Objective	Mobile (Cyclic)	Intermediate (Cyclic-Continuous)	Fixed-base (Continuous)
Excavating	*Mobile Loaders* (Grader, Compactor)	*Rotating Excavators* *Integrated Excavators* *Ground Preparation* FEL as Tractor Shovel	*Rapid Excavators* LW (SW)
Load (muck or stockpile)	FEL as LHD or Scoop	*Loaders and Muckers* Cable Scraper	Stacker, Reclaimer
Intermediate Haul (short)	Ramcar S/C	*Mobile Conveyors and Crushers*	Crusher, Bin Chute, Ore-pass, Dump (Feeder)
Bulk Handling	Truck RR Barge Air	Hoist Tramway	*Conveyors* Pipelines

[a] See Table 17.2.6 for makeup of equipment groups and Table 17.2.5 for meaning of abbreviations.
Source: Adler and Lineberry, 1986.

under highwalls (Moomau, Zachar, and Leonard, 1974). Tagline cable scrapers, a surface type, are obsolete, having been replaced by tractor scrapers, but slushers are still widely used underground due to their minimum cost and headroom requirement. Underground conditions are unique in having (1) tighter clearances (i.e., low profiles and restricted opening width) and (2) unusually tough excavating (e.g., hard rock). For surface work, not only are top clearances almost unlimited, making large chassis and long booms possible, but excavating is often easier, occurring in a soil or in a loosely consolidated, blocky, or bedded geologic mass (either naturally occurring or induced by light blasting or ripping). On this basis, equipment types exclusively for surface use consist of only draglines, bucket wheel excavators, and tractor scrapers (secondarily, also dredges, trenchers, stackers, and reclaimers). Equipment intended exclusively for underground use is self-evident: tunnel, raise, and shaft borers.

In excavating, there have been evolutionary trends from fixed-base to mobile and consequently to highly flexible equipment. Continuous processes are also being stressed, while miniaturization has frequently closed the gap between equipment types (e.g., between backhoe and front-end loader). Another trend that affects the classification is hybridization, resulting in many variants, which provides increased versatility but with major compromises. At worst, an equipment type will appear in the classification immediately next to other possible locations. The need for alertness in applying this classification does not diminish its usefulness, and exceptions frequently establish a general rule.

The major advantages of the proposed classification lie in its dense informational content, far exceeding previous ones, and in the many practical insights it gives on the functioning of equipment. Because "form ever follows function" (Sullivan, 1896), numerous clear relationships emerge, yielding considerably more information than all previous classifications. A wide variety of factors, conditions, and trends, ranging from output through diggability to costs, has been offered (Adler and Lineberry, 1986) for each degree of activity (mobile, intermediate, and fixed-base).

17.2.3 EQUIPMENT DESCRIPTION

17.2.3.1. Mine Plant vs. Mine Face Equipment

Before a standard procedure for analyzing and selecting underground mining equipment can be offered, a distinction should be drawn between what is termed mine plant equipment and mine face equipment. From this distinction, it will become clear why mine plant equipment selection and evaluation is more routine than that for mine face equipment. As the understanding of mine face analysis increases, these analyses will become more deterministic. Currently, the best computerized tools for underground production analysis are simulators (e.g., CONSIM, LONGSIM, UGMHS, FLEXMOD), which rely upon the user to vary equipment models and their capabilities when evaluating alternatives. No standard methodology exists for evaluating many pieces of mine face equipment and for selecting from alternative types and models. This is in sharp contrast to the well-developed and standardized procedures for analyzing and selecting mine plant machinery (e.g., conveyor belt, hoists, rail haulage, aerial tramways).

The classification system for mining equipment offered in 17.2.2 forms the basis for this discussion. This system organized equipment on the basis of its objective (excavating vs. handling) and its activity (mobile vs. fixed). A simpler classification system distinguishes underground mining equipment as being either a part of the mine face or of the mine plant. Any equipment used at the physical mine face for removal of material or for near-face transport of mined material can be termed *face equipment.* Equipment used to handle all products of the mine face to its final destination can be classified *as plant equipment.* In addition, plant equipment must provide all the needs of the activities, equipment, and personnel associated with the mine face from a source outside the face. Continuous miners, cutting machines, and loading machines are clear examples of face equipment. Belt conveyors, hoists, and rail haulage equipment are examples of mine plant equipment. Shuttle cars and other mobile bulk handlers can be classified as either mine face or mine plant equipment, although it is accepted that most activity of these devices is concentrated during loading near the face and, therefore, can perhaps best be classified as face equipment.

Generally, face equipment is transitory, that is, it is often switched about from place to place (trammed) and is mobile and impermanent. Another characteristic of face equipment is its high operating cost yet relatively low capital cost. Shuttle cars, scoops, and ramcars are examples of mobile haulage equipment, characterized by good maneuverability, flexibility, relatively low speeds 50 to 100 fps (15 to 30 m/s), and low capacities and production rates. Mobile equipment used in the face area is usually either rubber-tired or crawler-mounted. Rubber-tired vehicles, such as shuttle cars and scoops, are typically used when high speeds are desired and when road conditions are fair to good. Crawler-mounted equipment, such as continuous miners, is used when the terrain is rough or when the bearing capacity of the floor is problematic.

Plant equipment is typically static, fixed, and permanent to semi-permanent. Characterized by low operating cost but high capital cost, it can remain in place for a few months to 20 years or more, often for the life of the mine. Hoist, belt, and rail installations are examples of fixed haulage devices and are characterized by poor maneuverability, inflexibility, relatively high speeds, and high capacity and production rates. Examples of fixed plant equipment include conveyor belts, track haulage systems, and pipelines.

17.2.3.2 Common Components of All Mine Face Equipment and Mine Plant Machinery

MINE FACE EQUIPMENT. Because space is at a premium in underground mining, and because a rapid in-out approach is favored in selecting equipment, mobile equipment is emphasized here.

Mobile Face Excavating Units—Major components of all excavators are (1) a cutting mechanism (drum, borer, shearer); (2) loader or elevator (gathering arms and pan, auger, bucket); (3) conveyor for material transport through the mechanism (chain, flight, push blade); (4) running gear (wheels, crawlers, skids); (5) power supply (usually electric cable); and (6) cab (unless remotely operated). Continuous miners and boring machines are covered here. Selection and evaluation of other rock penetration devices (e.g., drills, shearers, plows) are covered in other sections.

Mobile Face Haulage Units—Major components of mobile face haulage equipment are common to all devices, with differences only in configuration, mechanism, and dimension. These components include (1) box (load-carrying component); (2) running gear (typically, rubber tires); (3) power supply (electric cable, battery, diesel); (4) conveyor (for material discharge); and (5) cab. Differences in construction, application, and limitations are evident in the routine review of the descriptions of shuttle cars, ramcars, and scoops, the three most common underground haulage devices (Lineberry, 1985; Stefanko and Bise, 1983). Front-end loaders and diesel trucks, although common in underground hard-rock applications, are not discussed here but are covered in Chapters 9.4 and 13.3.

MINE PLANT MACHINERY. Major components of mine plant machinery appear to be largely dependent on whether the equipment is mobile or fixed-base (stationary) and whether it is cyclic or continuous. In general, mobile, cyclic plant equipment (e.g., front-end loaders, trucks, rail haulage devices) compares favorably to mobile face haulage devices in regard to both major components and analysis. Mine plant machinery in the intermediate category (see Table 17.2.7) is both cyclic and continuous, yet is more closely akin to cyclic equipment in regard both to major components and to performance analysis. Common to rail haulage and hoisting, both used strictly for bulk handling, is that motive power is provided by an external source (locomotive and hoisting engine, respectively) and that more control is placed on the path of the load-carrying component (rails and shaft guides) than for mobile excavators and intermediate handlers. Fixed-base, continuous excavators, such as the rapid type (borers and reamer augers) and the longwall type, are comparable to mobile excavators in description and analysis, except that these devices have sacrificed mobility and flexibility for continuity of face operation. Fixed-base, continuous base handlers (e.g., conveyors, slurry pipelines) have two common major components: a load carrier (belt, pipeline) and power supplier (drive pulley, booster pump). Performance analysis for these two devices is also quite similar.

17.2.3.3 Common Underground Equipment: Excavators

CONTINUOUS MINING MACHINES. The decade of the 1980s has seen the continuous miner evolve as the most effective excavating tool for underground mining of coal and other friable materials. Because of regulations on underground blasting in coal mines and for reasons of safety and labor intensiveness, mineral extraction with conventional equipment (e.g., drills and cutting machines) is no longer cost effective in many instances.

Longwall mining is the most productive mining method, considered on a tons per employee basis. Longwall mining, however, requires a sizable investment in capital. Furthermore, longwalling must be complemented by continuous miners for entry development. Continuous miners are flexible and economical, being especially suitable for small two to three unit mines. Room and pillar mining, with application in undulating seams, is highly productive in low to high seams.

The three basic types of continuous miners are (1) rotating drum machines and auger miners, (2) boring machines, and (3) roadheaders. Rotating drum machines have evolved as the most productive and universal. Boring and roadheader types are limited to specialized applications. In this section, selection principles for drum-type continuous miners will be emphasized. Analyses of other underground excavators and handlers follow a similar line or reasoning.

Rotating Drum Machines—All manufacturers have concentrated on designs with increased horsepower and weight in order to maximize productivity to meet the economic needs of the underground mining industry. Simultaneously, machine design and development have addressed reliability to maximize availability and subsequent utilization, as well as ease of maintenance to minimize repair time and cost. Figs. 17.2.1 through 17.2.3 show three common drum-type continuous miners in use today.

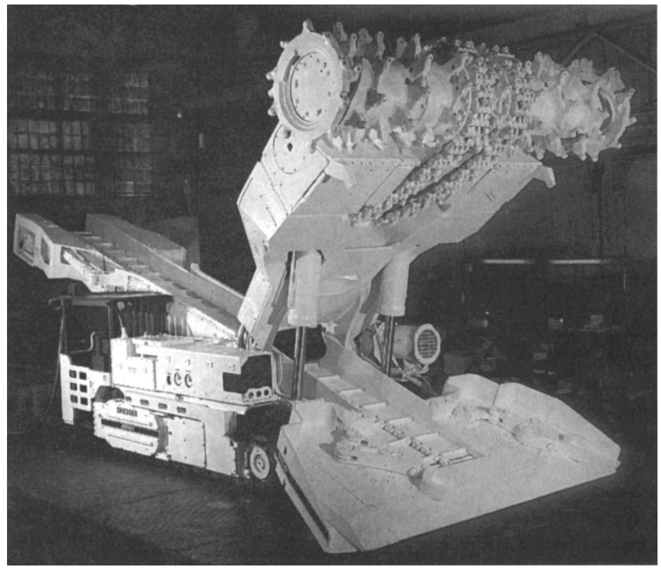

Fig. 17.2.1. Dresser 1060HP, 920 horsepower, rotating drum-type continuous mining machine. (*Courtesy:* Dresser Industries, Inc.)

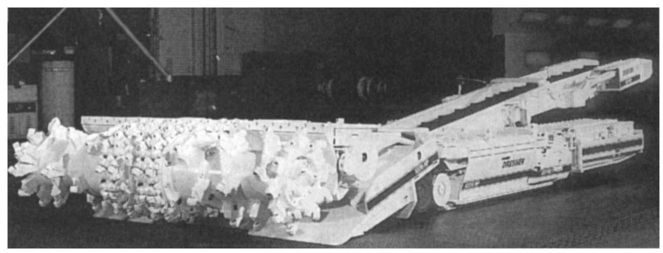

Fig. 17.2.2. Dresser 1026HP, 690 horsepower, low-seam, rotating drum continuous miner (*Courtesy:* Dresser Industries, Inc.)

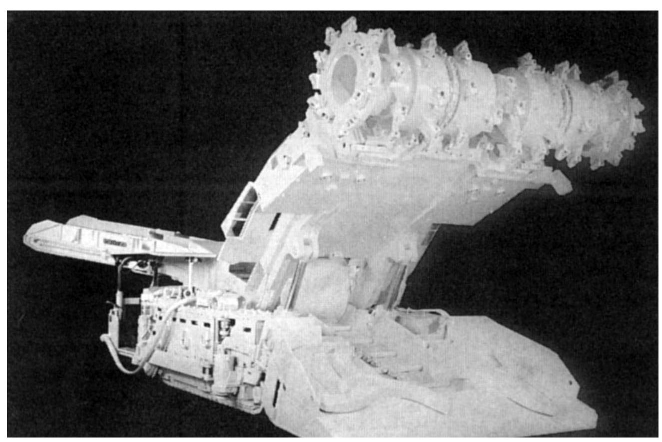

Fig. 17.2.3. Joy Model 12CM12, drum-type continuous miner. (*Courtesy:* Joy Technologies, Inc.)

GEOMETRY OF DESIGN. An aspect of the selection process that must be considered by users is the static design geometry of the miner. A simple static analysis of the physical characteristics of the machine is primary to an understanding of its effect on machine performance. A static diagram (Fig. 17.2.4) can be easily used to determine the sump and shear stability of a given machine, as well as its ability to utilize the full cutting power of the cutter head in applying bit force to break out the material

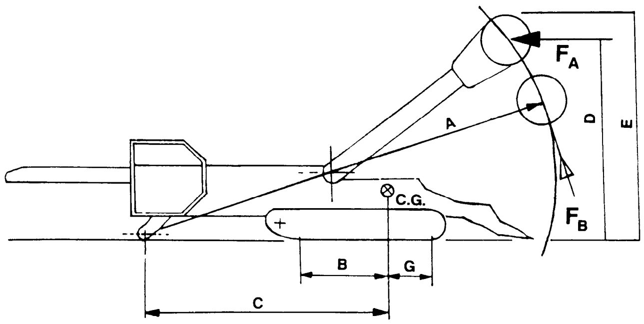

Fig. 17.2.4. Static diagram for typical rotating drum-type continuous miner.

from the solid face. The static elements of the machine must produce a force component F_A that is equal to or greater than the similar component produced by the sump force of the miner, with the sump force being determined by the machine weight and the coefficient of friction of crawler chains on the mine bottom. (The design of the crawler drive system must, of course, provide sufficient tractive effort to exceed this force.)

Concurrently, the static component F_B must be reasonably close to a similar component produced by the cutter head torque. Design expertise is required in this calculation in applying modification factors, such as the efficiency of the cutter head design and the bit force required to break out the material being mined.

SEAM HEIGHT PARAMETER. The flexibility of a continuous miner has a major constraint: the physical height of the machine. Manufacturers, while not having any published standards, generally classify continuous miners into four basic categories, on the basis of coal seam height:

Classification	Seam Height	
Very Low	26 to 30 in.	(0.66–0.76m)
Low	Over 30 to 42 in.	(0.76–1.07m)
Medium	Over 42 to 72 in.	(1.07–1.83m)
High	Over 72 in.	(+ 1.83m)

The only basic criterion for selection by seam height is operating clearance. Two basic factors influence machine selection for a particular seam height: the minimum practical mining height of the miner and its chassis height. In low and very low seams, without undulations, a minimum of 6-in. (150-mm) operating clearance is considered adequate. While more clearance is desirable, machine designs are not available to achieve this. Thus, in medium seams 12 to 15 in. (300 to 380 mm) is the desired operating clearance and 18 to 24 in. (460 to 610 mm) in high seams. This selection by operating clearance is clearly dictated by the economic need to utilize the most productive machine in each application. Tables 17.2.8 through 17.2.10 illustrate the physical specifications of commercially available continuous miners.

PRODUCTIVITY/AVAILABILITY. The economics of operation in parts cost per ton of material produced is an essential factor in the selection of any continuous miner. However, a low operating cost without good productivity will result in low-margin returns on investment. Thus availability and productivity must be carefully considered in the evaluation. The following example illustrates the importance of this comparison.

Example 17.2.1. Machine "A" mines at a rate of productivity of 10 tpm (9 t/m):

a. 1 minute cutting
b. 0.5 minute loading
c. 1 minute wait time
TOTAL: 2.5 minutes

Therefore, productivity is: 10 ÷ 2.5 = 4 tpm

Shift production is 180 minutes × 4 = 720 tons at 100% availability

Annualized productivity is 2 shifts × 5 days × 50 weeks × 720 tons = 360,000 tons.

Today, 93 to 96% availability is the accepted norm. Thus, at 96% availability in this example, the miner would produce 345,600 tons/year.

However, if replaced with miner B with 20% more productivity, utilized 3% less, or 93% availability, the results would be an annual production rate of 401,760 tons, an increase of 56,160 tpy, or additional annualized gross income of approximately $1,291,700 (based on a coal price of $23/ton). Lower availability would reflect higher parts cost per ton:

Parts cost, machine A: 345,600 × 0.30/ton = $103,680

Parts cost, machine B: 401,760 × 0.35/ton = $140,616

Thus the annualized gross income increase from Machine B is reduced to $1,254,700. If a profit margin of $4.00/ton is used, machine B produces $187,700 more profit dollars per year. The principle of miner selection by considering productivity and availability is graphically shown in Fig. 17.2.5.

MINER UTILIZATION. On-board machine operators have the effect of limiting the depth of a continuous miner's advance to 20 ft (6 m) or less, depending on roof conditions. Operation by remote (radio) control extends the potential depth of cut to 40 ft 12 m). Place-changing time is thereby reduced by approximately 50%, allowing increased utilization of the miner in cutting and therefore increasing productivity.

Two methods of on-board ventilation are available to dilute methane and reduce respirable dust concentrations in the working place with the deep-cut application.

On-board Dust Collectors—An electrically driven fan connected to a machine-mounted ducting system picks up the dust at the cutter head. It is then exhausted and collected on a filter screen. Downstream, a de-mister, mounted in the ductwork, separates the water from the air, allowing clean dry air to be exhausted at the rear of the miner. The criterion for efficient operation of dust collectors is that they must be used with blowing face ventilation. A typical dust collector arrangement is shown in Fig. 17.2.6. Fig. 17.2.7 shows its actual position on a continuous miner.

Fan Sprays—A second system is the Foster-Miller type of fan spray. This system increases the flow of incoming air, in combination with an increased number of water sprays that effectively control dust and increase the airflow sweeping across the face, thus permitting the use of deep cuts.

LONGWALL DEVELOPMENT. Contrary to room and pillar production openings, longwall development headings are normally driven as narrow as possible, consistent with accommodating face end equipment, ventilation requirements, etc. This introduces several disadvantages for continuous mining, such as inefficient two-pass operation, difficulty of place changing in more restricted space, and limitations on haulage equipment. The haulage system is also affected by the tendency to locate crosscuts on longer centers. This extends haulage times.

As a result, a different approach to these productivity contraints is occurring. Modern higher-powered continuous miners have no difficulty in cutting the full width of the heading (15 to 16 ft, or 4.6 to 4.9 m) in one pass. In addition to the obvious advantage of increased efficiency, single-pass mining ensures a consistent width of entry and makes it easier to maintain straighter roadways.

On the other hand, place changing with a full-width auger, even with pull-ins, can be difficult and, therefore, interest in roof bolting off the miner has increased. Early development in 1976

Table 17.2.8. Specifications for Very-low-seam Continuous Miners

	Dresser 102hp	Dresser 1026	Simmons-Rand 500	Joy 14CM10/10AA	Joy 14CM12/10AA	Joy 15CM2/10A	Fairchild Intl. Mark 21	Fairchild Intl. F-410
Basic Machine Height	24 in.	26¾ in.	25½ in.	25½ in.	25½ in.	23 in.	23 in.	27 in.
Cutter Support Height	24½ in.	30¾ in.	29¾ in.	28¾ in.	30¾ in.	26 in.	—	—
Machine Body Width	9 ft 6 in.	9 ft 5 in.	9 ft	9 ft 3 in.	8 ft 9½ in.	8 ft 4 in.	9 ft 5 in.	8 ft 0 in.
Ground Clearance	5¾ in.	6 in.	6 in.	6 in.	5 in.	7 in.	—	6 in.
Overall Length	23 ft 7 in.	34 ft 6 in.	35 ft	35 ft 7 in.	35 ft 7 in.	35 ft 0 in.	24 ft 0 in.	42 ft 3 in.
Bumper to Face	19 ft 5 in.	24 ft 10 in.	24 ft 7½ in.	25 ft 11 in.	25 ft 11 in.	24 ft 9 in.	22 ft 4 in.	32 ft 5½ in.
Cutting Diameter	26 in. 30 in.	31½ in.	29¾ in.	30 in.	30 in.	28 in.	26 in./36 in.	30 in./42 in.
Cutting Width	10 ft 1 in.	11 ft 0 in.	10 ft 10 in.	10 ft 10 in.	10 ft 0 in.	10 ft 0 in.	11 ft	16 ft
Maximum Mining Height	60 in.	101 in.	85 in.	90 in.	90 in.	60 in.	48 in.	72 in.
Minimum Mining Height	26 in.	31½ in.	29¾ in.	31 in.	32 in.	28 in.	26 in.	32 in.
Minimum Tram Height	26 in.	31½ in.	29¾ in.	31 in.	31 in.	28 in.	26 in.	28 in.
Machine Weight	44,000 lb	85,000 lb	85,000 lb	87,000 lb	79,000 lb	68,000 lb	32,000 lb	72,000 lb
Cutting Head Horsepower	250 hp (2@125)	250 hp (2@125)	270 hp (2@135)	330 hp (2@165)	250 hp (2@125)	230 hp (2@115)	230 hp	310 hp
Total Horsepower	345 hp	440 hp	545 hp	510 hp	430 hp	400 hp	230 hp	410 hp
Conveyor Width	30 in.	30 in.	30 in.	30 in.	24 in.	24 in.	18 in.	24 in.

Conversion factors: 1 in. = 25.4 mm, 1 lb = 0.4536 kg, 1 hp = 0.7457 kW.

Table 17.2.9. Specifications for Low/Medium-seam Continuous Miners

	EIMCO 2810	Joy 14CM10/10A	Joy 14CM15/10D	Joy 14CM14/10B	Joy 14CM9/10B	Dresser 1026hh	Simmons Rand SR-700
Basic Machine Height	28 in.	25½ in.	28½ in.	28½ in.	28½ in.	26¾ in.	28 in.
Machine Body Width	9 ft 7 in.	9 ft 3 in.	9 ft 11 in.	9 ft 11 in.	9 ft 3 in.	9 ft 7 in.	9 ft 0 in.
Ground Clearance	6 in.	6 in.	9 in.	9 in.	9 in.	6 in.	6 in.
Overall Length	35 ft 9 in.	35 ft 9 in.	35 ft 9 in.	35 ft 9 in.	35 ft 9 in.	34 ft 7 in.	36 ft 3 in.
Bumper to Face	26 ft 7 in.	25 ft 11 in.	26 ft 11 in.	26 ft 1 in.	26 ft 1 in.	26 ft 0 in.	26 ft 9 in.
Cutting Diameter	36 in.	32¾ in.	44 in.	36 in.	36 in.	42 in.	36 in.
Cutting Width	11 ft	10 ft 10 in.	10 ft 10 in.	10 ft 10 in.	10 ft 10 in.	11 ft 0 in.	10 ft 10 in.
Maximum Mining Height	120 in.	120 in.	126 in.	122 in.	122 in.	120 in.	120 in.
Minimum Mining Height	36 in.	32¾ in.	45 in.	37 in.	37 in.	42 in.	36 in.
Minimum Tram Height	36 in.	32¾ in.	45 in.	36 in.	36 in.	36¾ in.	36 in.
Machine Weight	100,000 lb	90,500 lb	105,000 lb	101,500 lb	92,500 lb	108,000 lb	107,000 lb
Cutting Head Horsepower	400 hp (2@200)	330 hp (2@165)	420 hp (2@210)	420 hp (2@210)	330 hp (2@165)	500 hp (2@250)	400 hp (2@200)
Total Horsepower	650 hp	510 hp	670 hp	670 hp	510 hp	790 hp	685 hp
Conveyor Width	36 in.	30 in.	38 in.	38 in.	30 in.	36 in.	36 in.

Conversion factors: 1 in. = 25.4 mm, 1 lb = 0.4536 kg, 1 hp = 0.7457 kW.

Table 17.2.10. Specifications for High-seam Continuous Miners

	Dresser 1038HC	Joy 12CM12-10B	Joy 12HM9-10B	Dresser 1060hp	Joy 12HM10
Basic Machine Height	40 in.	39 in.	56 in.	58 in.	56 in.
Machine Body Width	10 ft 3 in.	9 ft 7 in.	9 ft 10 in.	9 ft 10 in.	10 ft 4 in.
Ground Clearance	12 in.	9 in.	13 in.	12 in.	13 in.
Overall Length	35 ft 10 in.	35 ft 11 in.	34 ft 0 in.	36 ft 10 in.	42 ft 3 in.
Bumper to Face	25 ft 4 in.	25 ft 7 in.	24 ft 10 in.	27 ft 2 in.	30 ft 7 in.
Cutting Diameter	36¾ in.	44 in.	36 in.	47 in.	58 in.
Cutting Width	11 ft 0 in.	10 ft 10 in.	10 ft 10 in.	11 ft 0 in./18 ft 0 in.	15 ft 0 in.
Max. Mining Height	158 in.	145 in.	180 in.	186 in.	168 in.
Min. Mining Height	50 in.	50 in.	72 in.	72 in.	64 in.
Min. Tram Height	48 in.	48 in.	60 in.	58 in.	60½ in.
Machine Weight	127,000 lb	126,500 lb	129,500 lb	200,000 lb	200,000 lb
Cutting Head Horsepower	500 hp (2@250)	420 hp (2@210)	400 hp (2@200)	600 hp (2@300)	630 hp (2@315)
Total Horsepower	790 hp	680 hp	710 hp	920 hp	920 hp
Conveyor Width	36 in.	38 in.	30 in.	36 in.	30 in.

Conversion factors: 1 in. = 25.4 mm, 1 lb = 0.4536 kg, 1 hp = 0.7457 kW.

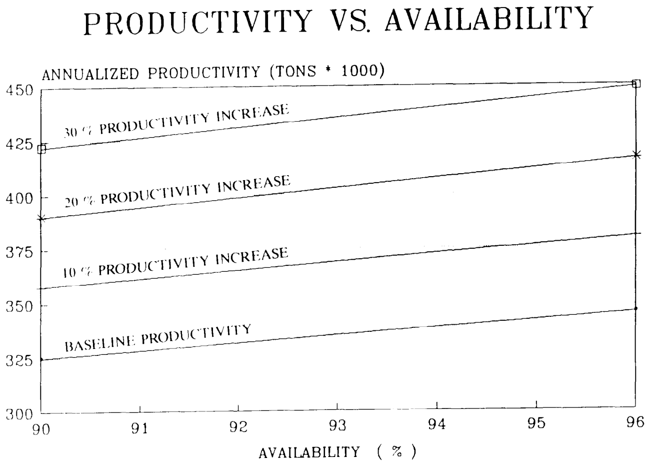

Fig. 17.2.5. Continuous miner selection, productivity vs. availability. Conversion factor: 1 ton = 0.9072 t.

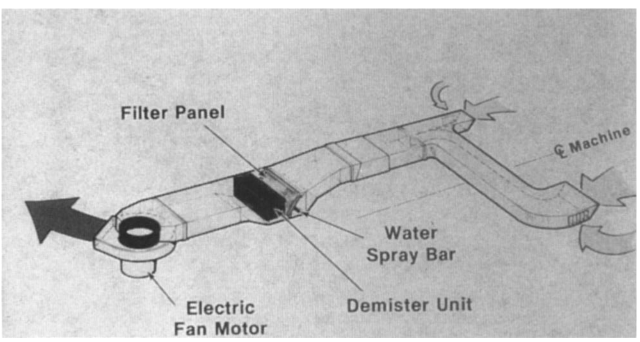

Fig. 17.2.6. Dust collector arrangement for continuous mining machine.

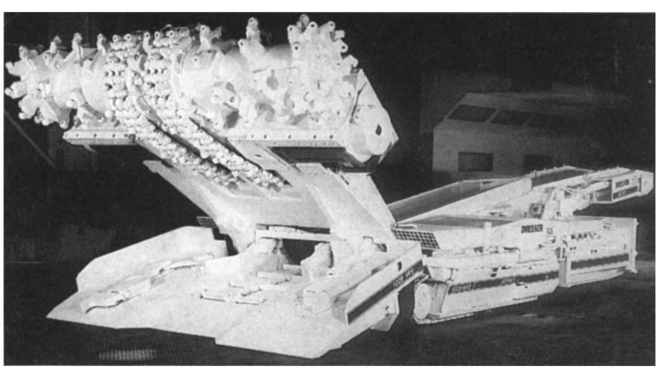

Fig. 17.2.7. Dust collector installed on continuous miner. (*Courtesy:* Dresser Industries, Inc.)

placed the bolters immediately in front of the operator's pit. Today the bolts can be placed within 11 ft (3.4 m) of the face.

By bolting off the machine, place changing can be virtually eliminated. However, production must stop while bolting takes place. This is obviously not achieving the aim of making "contin-

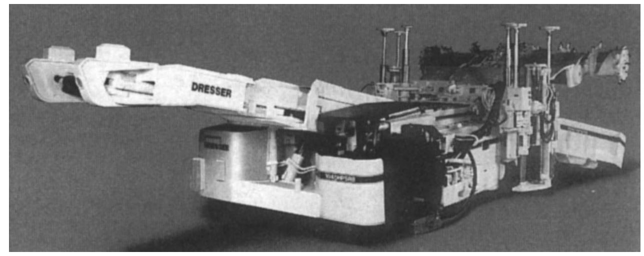

Fig. 17.2.8. Shuttle bolter. (*Courtesy:* Dresser Industries, Inc.)

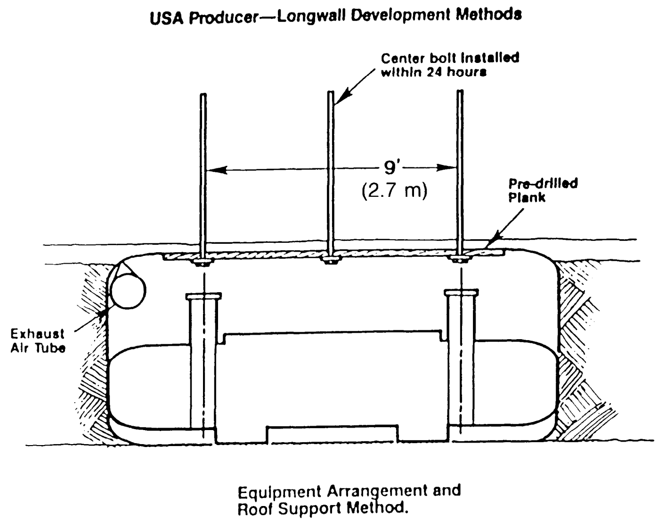

Fig. 17.2.9. Bolting plan for shuttle bolter.

uous" miners more continuous, so a new development became necessary. This is the shuttle bolter (Fig. 17.2.8).

Basically, the mining and bolting functions are separated, although the latter is powered from the miner. The bolters are mounted on a bridge structure that straddles the miner. This frees the machine to mine continuously while bolting takes place.

The shuttle bolter eliminates the dead time generated when the fixed bolting machine stops mining to complete the bolting cycle. This should allow approximately a 33% increase in productivity.

Where possible, the distance between crosscuts in longwall entries should be maximized. This, of course, introduces haulage constraints, and thus there has been a keen interest in continuous haulage systems that can operate in narrow entries. This equipment eliminates the change-over time in the entry. However, these systems do require excessive time to establish and to negotiate a place change. This system, integrating the mining and roof bolting operations and utilizing continuous haulage, shows potential as a highly productive system for longwall development. It greatly reduces lost time by moving the machine out of the place to allow the roof to be bolted. The bolters set two bolts, approximately 9 ft (2.7 m) apart, and utilize roof planks to support the roof. While not universally applicable due to variations in roof conditions, they have proved very effective in increasing the rate of entry development (Fig. 17.2.9). Further increases in the rate of development can be achieved with the use of continuous haulage to support the system. Fig. 17.2.10 illustrates a typical development plan.

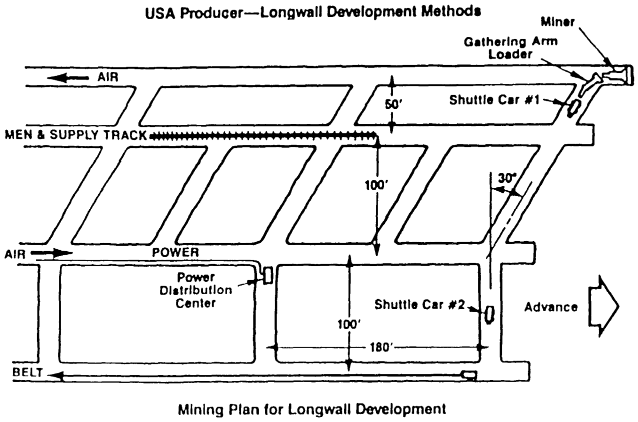

Fig. 17.2.10. Typical development plan for rapid longwall development, with use of continuous face haulage. Conversion factor: 1 ft = 0.3048 m.

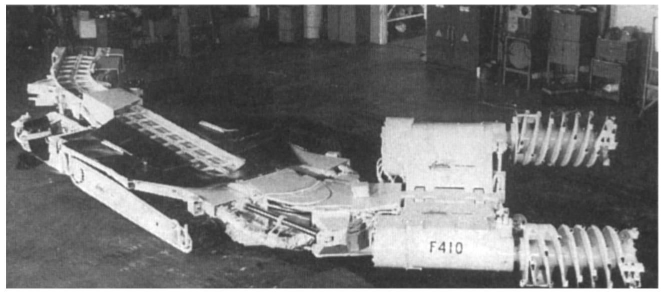

Fig. 17.2.11. Auger miner for thin-seam coal extraction. (*Courtesy:* Fairchild International.)

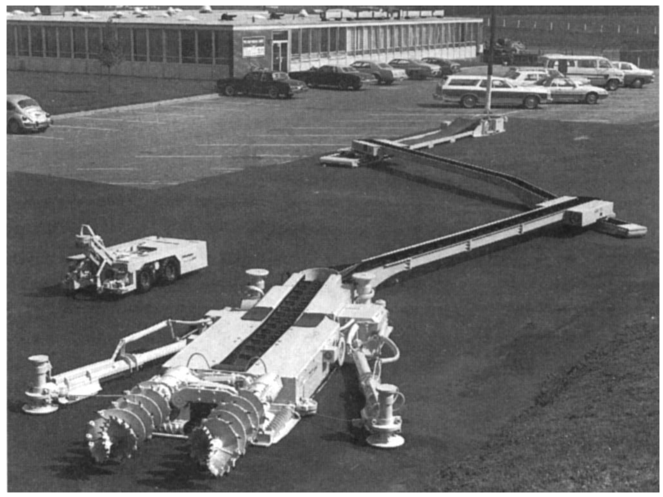

Fig. 17.2.12. Auger miner with hydraulic jack extensions. (*Courtesy:* Fairchild International.)

Continuous miners have now been developed into extremely productive machines. Place change time and haulage support systems are the limitations to further increases in productivity.

AUGER MINERS. One of the major mechanical design problems in the past has been packaging auger-type continuous mining machines into low-seam models. The result has been that, for a particular model, the high-pedestal mounting for thicker seams has resulted in a very good machine, but the low-seam models have been far less satisfactory. Originally, this model was developed by removing the cutter bar from a shortwall cutter, a type of cutter now obsolete, and by adding two cutter augers and a conveyor. The twin-cutting augers have an effective cutting depth of 5 ft (1.5 m) and are available with diameters of 20, 24, and 28 in. (508, 610, and 711 mm), to provide maximum mining heights of 30, 34, and 41 in. (762, 864, and 1041 mm) respectively, and can be lowered and raised at will.

In operation, the auger miner is positioned at the side of the heading with both augers at floor level. The rib-side auger is raised, and the machine is sumped into the face. The rib-side auger is then lowered and the other auger raised, and the machine is hauled across the face by ropes in the same manner as the shortwall cutter. A gathering conveyor, consisting of a single-strand chain with cast-steel flights, gathers the loose coal from between the augers and delivers it to the rear of the machine, where it is discharged into a bridge conveyor. A modern auger miner is shown in Fig. 17.2.11.

One of the undesirable features of an auger miner is that the jacks around which the ropes pull the machine across the face must be moved by the helper to maintain a cutting capability. This has resulted in a high incidence of accidents. To reduce the exposure of the helper to such accidents, a recent modification has been made to the machine. Two heavy-duty pivot jacks are frequently are mounted at its rear corners. The miner is moved with these jacks in conjunction with the anchor jacks and the ropes normally used. The difference is that the two anchor jacks are installed on each rib before the machine begins cutting. They remain stationary, the necessary maneuvering of the machine into and across the face being assisted by the pivot jacks. Among the advantages claimed for this mode of operation is that time-consuming sumping is eliminated; face workers to set jacks are not required, cutting down on exposure to hazards; and permanent roof supports can be set closer to the face, leading to greater safety and a need for fewer temporary supports and less labor. Finally, it is claimed that the semicircular coal face created by the auger miner offers better roof support and ventilation, although this is subject to question. The most recent modification uses hydraulic jack extensions for the pull ropes (Fig. 17.2.12).

TUNNEL BORING MACHINES. Full-face tunnel borers (Fig. 17.2.13) for a range of rock hardnesses are made by relatively few companies; the experience and expertise required to design and build such machines is considerable. Almost all those made are designed to match ground characteristics at specific projects, although machines have been reconditioned and sometimes modified after completing a task and used again. Boring machines are also available for shaft and raise driving.

The success of a TBM is highly dependent on the extent and accuracy of the ground investigations prior to design. The advances made in recent years in geotechnical sciences have contributed greatly to the increasing applicability of tunnel borers in mining and civil engineering works (Chapter 22.1). Considerable delays and increased costs can occur if there are, for example, unexpected fault zones, especially when filled with clay or water; wide variations in rock strength, toughness, abrasive-

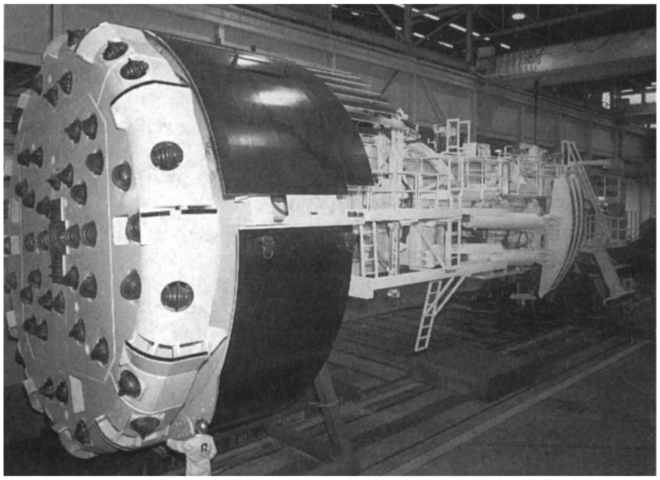

Fig. 17.2.13. A 27.9-ft (8.5-m) diameter hard-rock tunnel boring machine. (*Courtesy:* Robbins Co.)

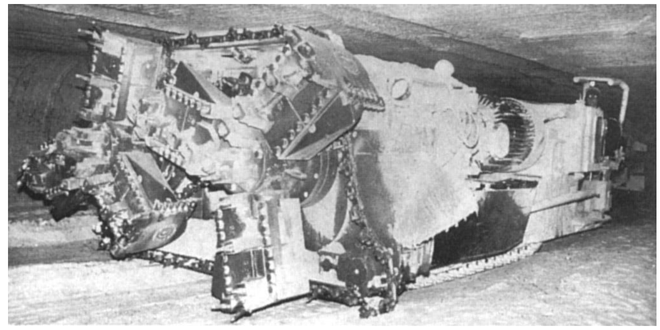

Fig. 17.2.14. Two-rotor boring miner. (*Courtesy:* EIMCO Coal Machinery Inc.)

ness, hardness, jointing, or lamination; or rapidly squeezing ground that may close onto the machine. Jointed and fractured formations may allow considerable improvement in boring rates compared with unfractured rock, but samples selected for testing purposes are often unfractured. Evaluation of this effect is, therefore, difficult and the many formulas proposed often provide widely differing results and thus are not presented here.

In brittle rocks, fractures may propagate rapidly from a high stress point if there are no inhibiting cracks or transverse platey minerals; in softer rocks that fail elastically after deformation, the pressure from a cutter may be released before a chip can form. Thus, for brittle rocks, high speeds would be preferred, but for softer rocks, low speeds and high torques are used. For minimum cutter costs, the cutter diameter, the cutter-ring cross section, and the cutter edge thickness should all be substantial. The hardness of the cutter edge is a compromise between abrasion resistance and toughness. Rotation speed must allow a balance between penetration rate and shock loading, especially in blocky ground. Increased cutter spacing results in reduced cutter costs and increased penetration, but too great a spacing will not allow a chip to be liberated by the cutter's shearing action, and grooving will occur.

Greater understanding of these complex variables over the last decade has led to considerable improvements in penetration rates and cutter costs, although the rate of improvement has shown some decrease. During boring operations, the front housing supporting the rotating head mechanism of the TBM must be held rigidly against the tunnel walls. After each complete stroke, the housing must be released to move it forward. The methods used to clamp the machine take several differing forms.

Mechanically positioned supports are fitted to stabilize the cutting head. These are adjustable to take account of gage cutter wear. On all these machines, steering is aided, normally with a laser target.

The choice between a roadheader or a TBM for development works is influenced by a number of factors. Favoring roadheaders are tasks involving roadways with changing cross sections, directions, or gradients, and where the lengths to be driven are relatively short (< 6600 ft or 2000 m). When access to the face is required for exploratory drilling, grouting, or drilling and when blasting through very hard strata, roadheaders present no problem. Rock compressive strength should not exceed 22,000 psi (152 MPa), although manufacturers stress that other important factors are tensile strength, abrasivity, cementation, lamination, jointing, and water content (which can increase the abrasive effect in siliceous rocks). The wear rate on the cutters may be a crucial cost factor, and pick manufacturers can usually conduct laboratory tests and advise accordingly. Tunnel boring machines are best employed for drivages in excess of about 10,000 ft (3000 m), when the required diameter is between 5 ft to 40 ft (1.5 to 12 m), and when the rock compressive strength is below about 40,000 psi (276 MPa). The best ground for TBMs is sedimentary, with bedding planes and jointing, and with good fracturing capability. Water inflow should be low and ground treatment unnecessary. Grades should be below ± 10° and curves less than radius, although there are special designs capable of tackling steeper gradients and tighter curves. (45° inclines and 100-ft [300-m] curves have been achieved.)

Tunnel boring machines will normally achieve increased advance rates than roadheaders in comparable conditions. Achievements reported from a variety of sources indicate speeds of advance for roadheaders between 50 to 275 ft (15 to 84 m) per week and with TBMs between 165 to 650 ft (50 to 200 m) per week. Thus, for about eight times the capital cost, depending on sizes and ancillaries, a doubling of advance rate is achievable. Because time costs money, as in many main mine development projects, this increased cost will often be well justified. The results achieved with either type of machine, however, will be influenced very strongly by the adequacy of the waste handling system, whether conveyor or rail car, by the efficiency of the supporting method, and by the competency of the project management team. These are always essential ingredients for project success.

BORING MINERS. As an alternative to ripper machines in coal and soft rock, boring-type machines were introduced almost at the same time. These machines have rotating arms equipped with bits that "bore" out the mineral product, hence the name.

The boring miner, as shown in Fig. 17.2.14, has a cutting chain on the bottom that creates a flat working bottom and an upper trimming chain, which, together with the boring arms, produce a roof configuration that is an arched or ovaloidal opening. The arms, which revolve relatively slowly, produce a much coarser product than does the rotating drum miner, minimizing gas and dust problems. Since cutting is accomplished by advancing the machine on its cats, it is very difficult to roof bolt and cut simultaneously with it, although drills have been mounted on telescopic devices and slide carriages on the machine with varying degrees of success. The greatest disadvantage of the machine is its great bulk; it occupies almost the entire entry. Although the extreme tips of the arms usually can be extended

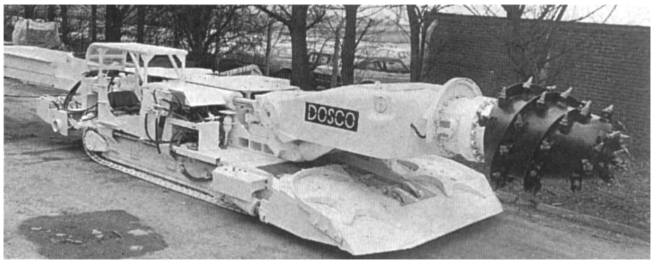

Fig. 17.2.15. Typical roadheader for use in coal or soft rock. (*Courtesy:* Dosco Corp.)

or contracted hydraulically over certain limits to provide some machine clearance during tramming, this generally is insufficient to allow very fast movements. Where the top and bottom have a tendency to converge, the machine can easily become wedged and, therefore, cannot be employed. Also the machine's bulk makes it difficult to keep the face ventilated. Because of these severe disadvantages, borers are rarely employed in mines today, except for occasional development work.

ROADHEADERS. Primarily intended for hard-rock mining and arched-roof profiles, roadheaders (Fig. 17.2.15) can mine materials that are harder to cut than continuous miners, although at a reduced productivity rate.

Roadheaders, originally built for coal and for relatively soft rocks, incorporate the cutter drive motor in the boom structure, which can be slewed by a rack and pinion. Machines that are built now, for high outputs in strata with compressive strengths up to 20,000 psi (138 MPa), have very strong, box-type booms that are hydraulically raised and swung. The power and the weight of the largest machines have increased dramatically over the past few years. The production obtainable has created a growing interest in the use of these machines for stoping operations (e.g., in borate and copper ores in the United States).

The slewing force on a transverse head is at right angles to the circumferential force, and the reaction is countered directly by the mass of the machine. When sumping, the direction of cut is the same as the circumferential force, and a large forward thrust is necessary. Several repeated cutting procedures may be required in all but the softest ground, pushing forward on crawler tracks to obtain sump depth that at most is about two-thirds of the head diameter. To obtain a clean final profile, irregularities on the roof and sides may need to be removed after moving back on the tracks. Sometimes the required profile is best obtained by the use of a telescopic and/or articulated boom. Movement of the head when excavating is, as far as possible, vertical.

With in-line heads, circumferential forces and slewing forces act in the same plane across the face. The reacting force will result in a sideways thrust, which with light machines, may need counteracting by bracing jacks. This is not a serious problem with heavyweight models, particularly those with wide gathering aprons and tracks, unless the floor is wet and slippery. Sumping requires less forward thrust compared with transverse heads, and any area of the face can be selected for the first cut, enabling weaker beds or lenses to be selected. The sump depth can equal the length of the head, although lesser depths may be selected when the ground is strong.

The sumping thrust available varies according to the machine weight, the gradient, and the floor conditions. Sumping thrust may be expressed as some fraction of the tractive effort available at the tracks, perhaps between 25% to 75%. Very few machines have telescopic booms, but these can exert an alternative means of applying sumping thrust, up to 20 tons (18 t), with the machine stationary, without respect to floor conditions, and, if necessary, spragged to the sides of the floor.

The design of in-line cutterheads shows considerable variation. Some are relatively short with closely spaced picks; others have long, conical, spiral vane heads, which are designed to ensure that a high number of picks are in contact with the rock and that vibration is minimized. The spiral vanes transfer the debris towards the gathering apron.

Pick speeds may vary between 200 and 680 fpm (1.0 and 3.5 m/s), the higher values tending to be employed for transverse cutters and for softer rocks. In harder ground, excessive pick speed usually results in high pick wear rates, increased dust generation, and, important in gassy mines, more risk of sparking. The most effective parameters are high forces and low speeds, provided the machine maintains stability. Some manufacturers offer alternative speeds, either by variable-speed motors or by gear changing.

Picks are generally point-attack conical types, especially in harder rocks, although chisel- or pentaprismic-shaped picks can be used. Angles and spacings for the picks are assessed by manufacturers to suit particular rock conditions, but modifications often have to be made after trials to achieve optimum speeds and pick life, especially on new projects. Contact pressure and torque increase with rock hardness.

Selection for any particular task is influenced by the power and weight, as determined by the type of ground and the output required. The robustness (ruggedness) of the machines and the engineering skills employed are also important factors, as are the availability of service and spares. Some general items stressed are the need for crawler tracks built for low ground pressure and high tramming speeds (similar to those on heavy-duty bulldozers) and hydraulics with a reasonably low pressure, preferably with gear-type pumps, especially if nonflammable fluid must be used. Easy changes of pick types and speeds are often important in variable ground or new projects. Debris handling might be by gathering arms onto a chain conveyor or by rakes or flight chains collecting from each side of a wide apron. The latter types are intended for advancing along the centerline of the tunnel, enabling the apron to extend the full road width and to hold debris during cutting operations. This makes possible accurate directional alignment and facilitates the provision of ancillaries such as a bridge conveyor, mechanized support, and dust extraction systems. However, widths must be above about 13 ft (4 m) and cannot vary during the job without machine modification.

17.2.3.4 Common Underground Equipment: Haulage Systems

Because face haulage systems for usual mining situations are relatively straightforward, both in equipment description and in terms of equipment selection and analysis (refer to Chapters 8.3 and 9.3), discussion here is limited to issues related to thin-seam mining of coal, which will assuredly receive increased attention in the future.

While the major changes have been made in underground excavation systems, face haulage has not been ignored, especially for underground coal mines. It is recognized that the capacity of the face-haulage vehicle is the major factor. Increased capacity reduces the number of change-outs and thus reduces lost production time. Commensurately, the higher-capacity vehicle takes longer to fill, allowing more time for the second or third vehicle to return from the dump point and be available for change-out.

As a result, there has been increased interest in battery- and diesel-powered vehicles. In certain conditions, they have been shown to have significant advantages over cable-powered shuttle cars:

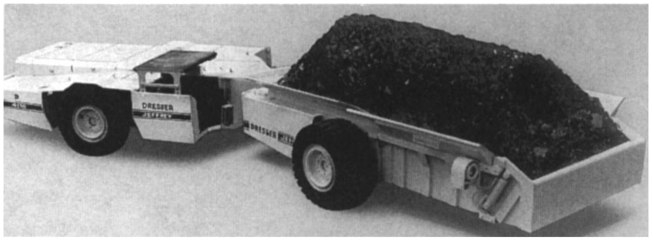

Fig. 17.2.16. Diesel-powered mobile handler. (*Courtesy:* Dresser Industries, Inc.)

1. Articulated design allows higher capacity within specific roadway dimensions.

2. Elimination of trailing cable allows use of three vehicles to reduce delays in change-out and provides more flexibility.

3. Better visibility is achieved for higher tram speeds. A diesel-powered ramcar is shown in Fig. 17.2.16.

Articulated vehicles do have the potential disadvantage of a three-point turn at the dump point, but this is offset by the much higher rate of material discharge. In addition, battery recharging or diesel maintenance are more onerous than cable-reel shuttle car maintenance. However, these requirements can be met off-shift and need not limit productivity. In addition, the shuttle car does have downtime from cable damage.

Example 17.2.2. In order to compare haulage vehicles, assume operation in a 6-ft (1.8-m) seam with 18-ft (5.5-m) wide entries in panels with 100- by 40-ft (30- by 12-m) centers. With 83-lb/ft^3 (1330-kg/m^3) material, an advance of 4.48 tons/ft (14.7 t/m) is obtained. It is also assumed that the continuous miner loads at 13 tpm (12 t/min), and the average load in a shuttle car and articulated vehicle is 6.5 tons (5.9 t) and 8.5 tons (7.7 t), respectively.

With respect to vehicle delays, it is assumed that all vehicles average 0.33 minutes to change-out behind the miner. Where shuttle cars are used, experience shows that, on average, there are "other delays" of 1 minute/car/load due to slowness in the trip to and from the dump point. Time studies show "other delays" of 0.6 minutes for two articulated vehicles and zero in the case of three vehicles, which shows the advantage of multiple haulage unit usage. A summation of time elements in a typical face haulage cycle is as follows:

Type of Vehicle	Two Shuttle Cars (min)	Two Articulated Vehicles (min)	Three Articulated Vehicles (min)
Loading Time Per Car	0.5	0.65	0.65
Change-Over in Entry	0.4	0.4	0.4
Other Vehicle Delays	1.0	0.6	0.0
Cycle Time	1.9	1.65	1.05
Vehicle Capacity (tons)	6.5	8.5	8.5
(t)	5.9	7.7	7.7
Mining Rate (tpm)	3.42	5.15	8.10
(t/min)	3.10	4.67	7.35

The above analysis is intended to be a relative one. Obviously, local conditions will modify performance, but the potential improvement in production with three vehicles is too large to ignore.

CONTINUOUS CONVEYOR HAULAGE. Continuous conveyor haulage has been widely used behind continuous miners for many years, although, until recently, its use has been largely confined to thinner seams. Below 36 in. (900 mm), the shuttle car becomes very inefficient because ground clearance requirements, particularly in rolling seams, limit capacity.

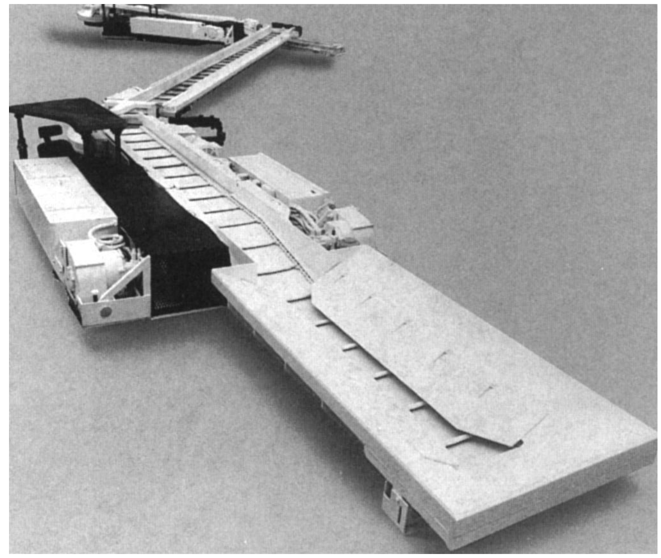

Fig. 17.2.17. Disconnected haulage system with surge hopper.

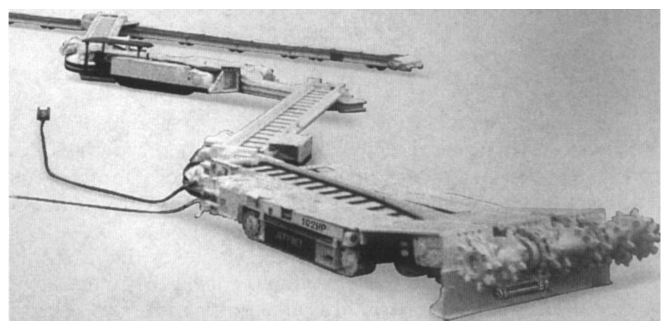

Fig. 17.2.18. Low-seam continuous haulage system. (*Courtesy:* Dresser Industries, Inc.)

Additionally, mining in these low seams is largely uneconomic unless entries of 18 ft (5.5 m) or more are maintained. This is the point at which cascading-type chain conveyor systems become practical, with component lengths of up to 30 ft (9.1 m). These systems comprise caterpillar-tracked mobile bridge carrier (MBC) units connected by simple bridge conveyors, which provide a degree of flexibility by riding along the receiving end of the MBC.

In the lowest seams, a swan-neck bridge conveyor is attached to the end of the continuous miner conveyor because there is not normally sufficient clearance for the miner conveyor to feed directly into the MBC. As seam height increases, a disconnected system is adopted. The miner conveyor feeds into an MBC with a limited capacity receiving hopper. This accommodates the surges that start to become more significant as the tonnage mined in the shear-down part of the cycle increases. The disconnected system provides increased length and flexibility (Fig. 17.2.17).

Low-seam systems, such as the one shown in Fig. 17.2.18, have capacities of 6 to 8 tpm (5.4 to 7.2 t/min), while seam installations can handle up to 13 tpm (11.7 t/min), with increased conveyor widths and sideboards (Fig. 17.2.19).

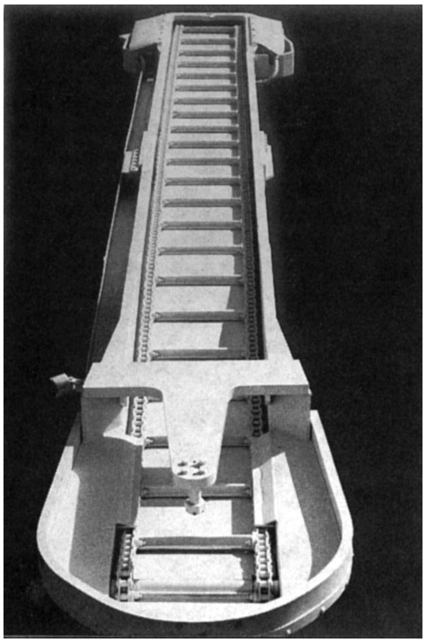

Fig. 17.2.19. High-capacity bridge carrier (*Courtesy:* Long Airdox Company)

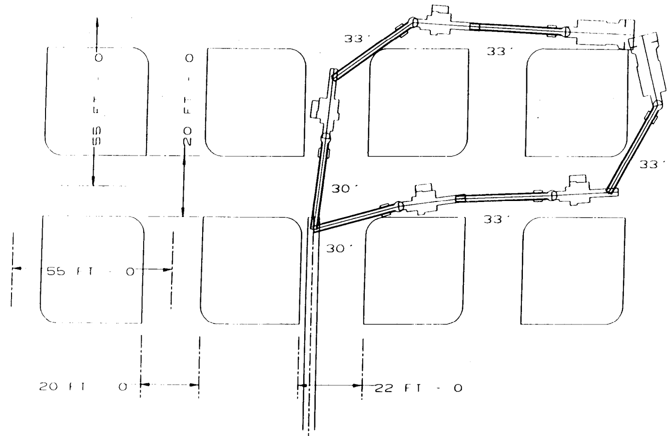

Fig. 17.2.20. Continuous haulage development plan. Conversion factor: 1 ft = 0.3048 m.

In higher seams, wider conveyors become a problem, as roadways tend to decrease in width. In addition, the mining rates on shear-down again overcome conveyor capacity, resulting in spillage or a need for operator restraint that is seldom maintained.

Despite doubts about reliability and efficiency that can only be obviated by industrial use, continuous conveyor haulage obviously offers a productivity improvement over rubber-tired haulage vehicles with their intermittent operation. However, there are significant trade-offs that must be evaluated:

1. Continuous haulage systems contain many more components than haulage vehicles and can be more prone to breakdown.
2. Conveyor breakdowns completely stop production, while rubber-tired haulage can still produce 60% of normal production with one of two cars available.
3. Entries have to be established and conveyor installed equivalent to the length of the continuous haulage system before operation. This is to accommodate retraction of the system during place changing.
4. Place changing of the continuous miner requires movement of the haulage system, which adds to the time taken and provides opportunities for further delays.

A typical continuous haulage development plan is shown in Fig. 17.2.20. Tables 17.2.11 and 17.2.12 give specifications for mobile bridge carriers and bridge conveyors, respectively. Table 17.2.13 illustrates a typical selection procedure for a recent continuous conveyor haulage system installation.

17.2.3.5 Common Underground Equipment: Face Preparation

To avoid repetition, face preparation by cutting machine is the only pre-extraction activity treated here. Face drilling for relief (e.g., presplitting) is covered in Chapter 9.2. Also omitted are the mechanics of rock penetration and treatment of drilling machines (Chapter 9.1), since drilling is more appropriately termed a production operation than true underground mine development.

CUTTING MACHINE. Drilling and blasting—the drilling of holes at the face in a certain pattern, the placement of explosives in them, and the subsequent blasting to fragment rock, coal, and ore—is conventionally practiced as an alternative to continuous mining. However, this method of fragmentation, called shooting from the solid, although practiced in nearly all hard-rock mining, tunneling, and similar construction operations, is frequently restricted by law in coal mining. It is both potentially dangerous and very inefficient. It requires the drilling of a large number of holes and the use of a sizable amount of explosives. Also there is a chance that the explosive may blow out of the hole during blasting, even if it is properly stemmed, igniting the coal dust and gas usually present and creating a serious explosion and fire hazard. Finally, the large amount of explosives required may result in roof damage from blasting vibration and shock.

Therefore, all coal seams using conventional mining processes should first be cut, a term unique to the mining of coal and other sedimentary mineral deposits such as salt and potash and the quarrying of dimension stone. This is accomplished by a coal cutter (Fig. 17.2.21), which is basically a giant chain saw. The cutter bar is generally 9 to 11 ft (2.7 to 3.4 m) long and equipped with bits to cut a 6-in. (150-mm) kerf. Most cutting machines are rubber-tired for mobility and are "universal," in that the cutter bar can be positioned to cut vertically or horizontally at an infinite number of planes. For low coal with a seam thickness no greater than about 5 ft (1.5 m), a horizontal slot is cut at the bottom. This provides a free face toward which the explosives in the single top row of holes can break the coal. A thicker seam requires two rows of holes for a bottom cut, but will be better fragmented if a shear (vertical) cut is also employed. While placing a horizontal cut at the roof line (top cutting) reduces vibrations in the roof from blasting and perhaps improves roof control, it is not often done because the ragged bottom that results hinders loading.

17.2.4 BASIC ANALYSIS FOR EXCAVATING AND BULK HANDLING

A fairly complex analysis has been offered (Adler, 1984) for selecting or evaluating a piece of excavating and bulk handling

Table 17.2.11. Range of Specifications for Mobile Bridge Carriers

	Jeffrey 506C5	Jeffrey 5010	Jeffrey 5010HPL	Long-Airdox 27C-L	Long-Airdox 27C	Long-Airdox 30C	Long-Airdox 36C
Height (Overall MBC)	24½ in.	26½ in.	26½ in.	24½ in.	25½ in.	26½ in.	40 in.
Height (Overall System)	26½ in.	29⅛ in.	29⅛ in.	25¾ in.	25½ in.	29½ in.	43 in.
Width (Overall)	7 ft 9 in.	8 ft 9 in.	9 ft 0 in.	8 ft 3 in.	8 ft 3 in.	8 ft 6 in.	9 ft 7 in.
Length (Overall)	20 ft 11½ in.	26 ft 10 in.	26 ft 10 in.	27 ft 0 in.	27 ft 0 in.	27 ft 0 in.	27 ft 3 in.
Weight (lb)	11,000	17,250	19,500	20,000	20,000	22,000	28,000
Cont. Rated Capacity (tpm)	6	10	12	10	11	15	27
Width of Conveyor	21 in.	24 in.	27 in.	27 in.	27 in.	30 in.	36 in.
Number of Motors	2	2	2	2	2	2	3
Conveyor Motor Horsepower	7½	20	20	20	20	20	40
Hydraulic Horsepower	20	40	40	40	40	40	75
Tram Speed (fpm)	40	65	65	65	65	65	65

Conversion factors: 1 in. = 25.4 mm, 1 lb = 0.4536 kg, 1 hp = 0.7457 kW, 1 ton = 0.9072 t.

Table 17.2.12. Range of Specifications for Bridge Conveyors

	Jeffrey 94L	Jeffrey 5010	Jeffrey 5010HPL	Long-Airdox 27C-L	Long-Airdox 27C	Long-Airdox 30C	Long-Airdox 36C
Height (Overall)	25½ in.	25½ in.	25½ in.	24½ in.	25½ in.	28½ in.	40 in.
Width of Conveyor	21 in.	24 in.	27 in.	27 in.	27 in.	30 in.	36 in.
Depth of Conveyor	6¾ in.	7 in.	7 in.	5 in.	7 in.	7 in.	10 in.
Length Between C.L. of Pivots	30 ft	30 ft–39ft	30 ft	30 ft–39ft	30 ft–39ft	30 ft–39ft	30 ft–40ft
Length (Overall)	32 ft 6 in.	32 ft 6 in./41 ft 6 in.	32 ft 6 in.	32 ft 6 in. 41 ft 6 in.	32 ft 6 in. 41 ft 6 in.	32 ft 6 in. 41 ft 6 in.	33 ft 6 in. 41 ft 6 in.
Swing Each Side of C.L. Conveyor	90°	90°	90°	90°	90°	90°	90°
Cont. Rated Capacity (tpm)	8	10	12	10	11	15	27
Motor Horsepower	15	20	20	2@10 hp	2@10 hp	2@15 hp	2@20 hp
Weight (lb)	4,600	6,000	6,600	9,000	9,000	9,500	10,300

Conversion factors: 1 in. = 25.4 mm, 1 lb = 0.4536 kg, 1 hp = 0.7457 kW, 1 ton = 0.9072 t.

equipment and for predicting its performance. At present, there is little agreement as to what specific points should be considered, and even less on what procedure to follow. Both of these are covered here in a consistent and direct form. The analysis applies to all types and sizes of equipment, from continuous miners to front-end loaders to shuttle cars. Because so many equipment types can be identified, and because of the complexity of providing detailed examples of all variants of the many analyses that constitute selection and evaluation of underground mining equipment, this basic procedure is preferred. If followed, this standard procedure serves well for the analysis of all mining equipment, both now and in the future.

The importance of equipment characteristics, job activities, and work conditions are stressed, and separate analytical stems are established in a flow chart for these factors. They are combined, using suitability tests, in the following steps: (1) problem statement, (2) natural conditions, (3) suitability guidelines, (4) appropriate equipment types, (5) operational activities, (6) encumbered space, (7) geometric design, (8) equipment sizing, (9) failure and limits, (10) performance and consumption, (11) public policy checks, and (12) costs and profitability. Public policy and economics, while considered throughout, receive emphasis toward the end. The procedure is iterative, containing distinct feedback loops. Of value is the lack of the need for hundreds of rules, formulas, variants, and exceptions for analysis in what is recognized as both a complicated yet essential area of mining engineering.

The purpose of this discussion is to describe a procedure for determining the *performance* of excavating and bulk handling equipment that is comprehensive, concise, and systematic. The procedure itself can be adopted as a standard and at least will provide a checklist of concerns. In doing so, the safe and efficient use of equipment is promoted.

The importance of excavating and bulk handling equipment in mining has already been established. This activity accounts for at least 50% of total direct costs, and perhaps for as many as 30% of all accidents. Of the several systems employed in mining, excavating and handling is the most important and sets the pace for all others. Recently, there has been an increased introduction of innovative types of equipment from manufacturers, some having a large and immediate impact. The number of types of excavating and bulk handling equipment is already prodigious, and their sizes vary vastly. This proposed analysis will be applicable to all types and sizes, and the approach developed should be of value to all professionals concerned with equipment application. The analysis relies heavily on the engineering sciences, and the procedure, while focusing on the excavating and bulk handling system, does consider the surrounding total systems context (see 17.2.1.).

Table 17.2.13. Selection Procedure for Conveyor System, Model 5010-HP-L System

Seam Height:	52 in.
Sump Speed:	8 fpm
Sump Depth:	26 in. = 2.16 ft
Miner Head Width:	10 ft 6 in.
Miner Head Diameter:	33 in.
Shear Rate:	12 fpm
Sump Volume:	62.56 ft^3
Shear Height:	52 in. − 33 in. = 19 in. = 1.58 ft.
Shear Volume:	36.02 ft^3
Coal (Solid):	80 lb/ft^3
Coal (Broken):	50 lb/ft^3
Run of Mine (Solid):	90 lb/ft^3
Run of Mine (Broken):	60 lb/ft^3
Percent Reject:	32%

Cycle Times (for 52-in. coal seam)

Sump	= 2.16 ft @ 8 fpm = 0.27 min
Shear	= 1.58 ft @ 12 fpm = 0.13 min
Rev. Cut Cusp	= 1 ft @ 30 fpm = 0.03 min
Tram to Face, Raise	= 3 ft @ 30 fpm = 0.10 min
TOTAL CYCLE TIME	= 0.53 min

Material Removed Per Cycle

$$\text{Sump} = \frac{62.56\ \text{ft}^3 \times 90\ \text{lb/ft}^3}{2{,}000\ \text{lb/ton}} = 2.81\ \text{tons}$$

$$\text{Shear} = \frac{36.02\ \text{ft}^3 \times 90\ \text{lb/ft}^3}{2{,}000\ \text{lb/ton}} = 1.62\ \text{tons}$$

Rate of Coal Removal Per Cycle

Sump = 2.81 tons/0.27 min = 10.4 tpm
Shear = 1.62 tons/0.13 min = 12.4 tpm

Capacity of "Wide-5010" at 439 ft^3/min

$$= \frac{439\ \text{ft}^3/\text{min} \times 60\ \text{lb/ft}^3}{2{,}000\ \text{lb/ton}}$$
$$= 13.1\ \text{tpm}$$

$$\text{Continuous Duty Cycle} = \frac{2.81\ \text{tons} + 1.62\ \text{tons}}{0.53\ \text{min}} = 8.58\ \text{tpm}$$

Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 lb = 0.4536 kg, 1 ton = 0.9072 t.

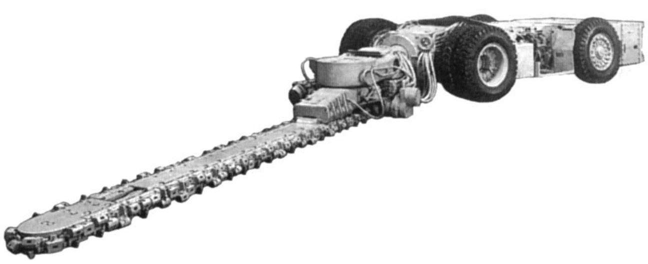

Fig. 17.2.21. Coal cutting machine for use in underground conventional mining. (*Courtesy:* Joy Technologies, Inc.)

All disciplines require a standardized, well-established approach to solve their particular problems. Structural analysis, for example, requires specification of the loads, material properties, and structural geometries to determine stresses. Although arguably more complex, a comparable approach is proposed for excavating and bulk handling.

17.2.4.1 Requisites for Basic Analysis

MAIN STEMS. To determine the performance of a given piece of equipment, three necessary and sufficient requirements must be described: (1) *work conditions,* (2) *activities,* and (3) *equipment characteristics. Work conditions* are the broad encompassing factors associated with a work site, including, among others, diggability, available space, the haulway surface, and human factors. *Activities* are the movements or motions prescribed by the operation and cover such aspects as loading, hauling, and dumping. *Equipment characteristics* describe the tool or instrument required to do the job. Mining equipment is routinely described in manufacturers' specifications by capacities, weights, dimensions, and power. Merging all of these aspects together through *suitability tests* enables the performance rate—the heart of the analysis—to be determined. This yields the time it takes to accomplish the specific job. For excavating and bulk handling, the job is usually production, the rate of mining or moving material, termed *loose mean output* (LMO), in units of tons (tonnes) per hour.

PROBLEM STATEMENT. Problem formulation must be as definite and concise as possible. A statement that lacks clarity leads to vague, muddled, and even inconclusive analysis. Two basic types of problems can be distinguished: (1) an evaluation when the equipment has already been given and (2) a selection of the equipment when the conditions are provided. Combinations are also possible, and in all cases, performance assessment is the most critical analysis. When the performance rate is specified, along with the conditions, an indirect approach is necessary, with a trial selection of equipment in order. Successive approximations using additional trial selections are executed until adequate agreement is reached between the desired and calculated performance rates. If the equipment is already available, and if the site conditions can be altered, performance rates at each of their altered states are found until agreement is reached.

Usually, there are a number of alternative choices, exemplified by the choice of several small mobile haulers or one large stationary conveyor. Public policy concerns must then be added to the straight engineering approach, and when the results are adjusted for these, a feasible solution is achieved. The final selection will depend upon company capabilities and expectations. Economically, this is formalized in profitability and cost-benefit analyses.

In any engineering study, it is always useful to avoid serious omissions by checking the *basic engineering dimensions.* These are matter, space, time, people, and money—which correspond loosely to "what?," "where?," "when?," "who?," and "how?" (Anon., 1966). The "why?" is the objective of estimating the performance rate, and the "how?" is the proposed analysis itself (Table 17.2.1).

17.2.4.2 Analyses

The analysis yields the performance rate for a single piece of equipment of a specified type and usually of the largest feasible size. Alternatives would therefore include several units of decreased size as well as other suitable types. The final choice will depend mainly on economics, but also on availability, familiarity, and other intangibles.

TRADITIONAL ANALYSIS. The procedure presently used is exemplified in the duty cycle calculations performed for mobile equipment where the input data are highly selective and limited (Bishop, 1968). Conditions mainly cover the bulk solid density, distances, grades, and rolling resistance factors. Equipment characteristics are only capacity, vehicle weight, and horsepower. Activities are usually lumped into fixed times for loading, dump-

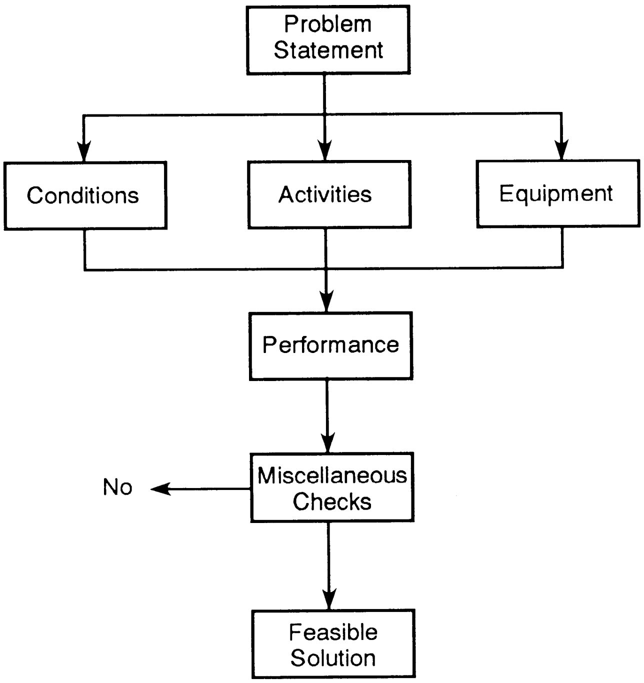

Fig. 17.2.22. Present flow chart for traditional analysis (Adler, 1984).

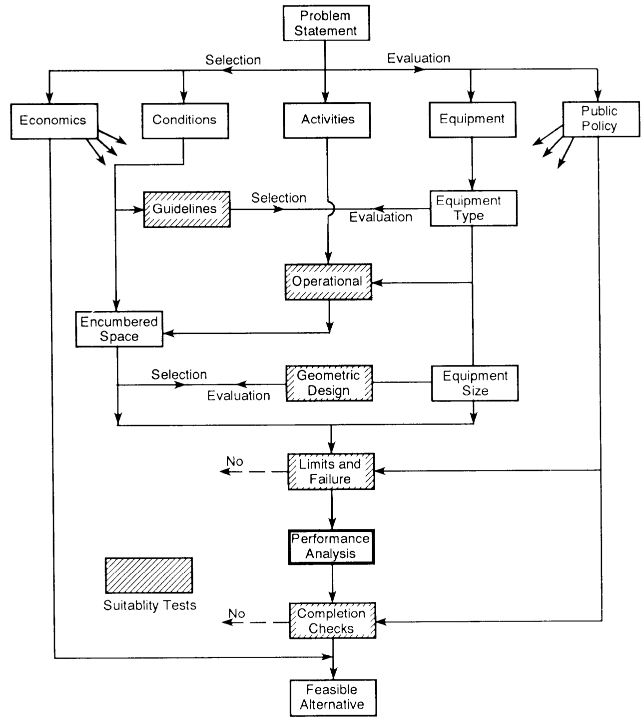

Fig. 17.2.23. Recommended flow chart for basic analysis (Adler, 1984).

ing, manuevering, and waiting. On this basis, the production rate is calculated and various constraints are subsequently invoked on speed limits, traction, and clearances. If any of these are exceeded, the input is varied and the solution reworked. Fig. 17.2.22 illustrates this procedure. It is important to add that this procedure is often valid for a first-order estimate of performance but is insufficient when selecting and operating underground mining equipment, particularly when the value of a single piece of equipment is in the hundreds of thousands of dollars.

The primary deficiency of the traditional, implicit method is its failure to discriminate between the various types of constraints. By lumping these constraints together and considering them only at the end, an inordinate number of iterations may become necessary. This number can be drastically reduced by examining and sorting out the constraints as early as possible, with the use of suitability criteria to yield a systematic flow from the general to the particular. The dependency between the working conditions and suitable equipment types is well established in various *"guidelines."* The spatial dependency between conditions and equipment *size,* which is of most concern in first-order selection and evaluation of mobile excavators and handlers (see 17.2.3.3), can be deliberately attacked in "geometric design." A part of any equipment's activities includes normal operating patterns and motions, which are dependent on the specific equipment *type.* By judiciously employing these suitability criteria at an early stage, the number of iterations is minimized. In addition, specific failure modes and regulations must be carefully evaluated, and, finally, the stage will have been set for a human factors analysis.

PROPOSED ANALYSIS. By establishing the three main stems—conditions, activities, and equipment—the necessary dependent links can be made early in the analysis and the stems systematically grafted and pruned. Fig. 17.2.23 shows the proposed analytical flow chart. It also indicates whether a selection or an evaluation procedure is being performed. For selection, the flow proceeds through conditions to guidelines to determine suitable equipment *types.* From equipment types, the analysis goes unidirectionally to operational activities, which are then added to other activities. The activity stem is then merged with the conditions stem in encumbered space. After this, the analysis proceeds through geometric design to establish the equipment *size.*

Knowing the equipment's type and size, its specifications can then be obtained. Merging encumbered space with the equipment stem is followed by examining limits and failure. The next step is the performance analysis, the focus of the procedure. Performance rate estimation (e.g., LMO), move and setup time, advance rate, clearing rate, bolting time is mainly considered here, but consumption rates are included also. When possible, all adjustments should be made prior to this step.

Additional checks and rechecks are then made, particularly for public policy (especially health and safety regulations), followed by the "completion" phase and human factors analyses. Throughout the procedure, public policy and economic concerns are regularly, but implicitly, dealt with. The final result, including necessary iterations, represents one of several feasible solutions.

17.2.4.3 Data Processing

CONDITIONS. The conditions under which underground equipment is operated can be divided into *natural conditions* and *site conditions* and can be differentiated by material descriptions. The natural and site conditions refer to the virgin state initially encountered in excavating, and then to the site developed for bulk solid handling by excavating, clearing, grading, and ripping. The natural conditions are immutable and therefore constitute independent variables. Of all the major inputs, these are the most complex, and their organization by engineering dimensions (matter, space, and time) is recommended (see 17.2.1.1). Spatially, subregions of uniform properties are established and described by location, size, and shape. Geography, predominately a surface mining concern, covers such material aspects as topography, climate (temperature, atmospheric pressure, precipitation), drainage, cultural features, and legal boundaries. A geologic description (ignoring resource values) concentrates on engineering capabilities, which entail costs. Material properties, predominately an underground mining concern, cover the rock type, geologic structure, fluids, and only peripherally rock pressure and temperature.

For excavating purposes, *properties* include rock density, strength, seismicity, jointing, abrasiveness, and water. These are recast into the extremely valuable job properties category, covering the ability to blast, drill, bore, dig, and rip (Adler and Krishnan, 1983). The bulk solids produced by excavating have unique properties, as will be noted later.

Handling involves moving any bulk solid, usually over a haulway. The haulway can be described in terms of rolling resistance, traction, bearing capacity, and roughness. Water again is significant and can dramatically influence the selection. Spatially, the haulway has various transverse cross sections strategically located along a longitudinal centerline. The centerline profile itself can be described by grades, curves, and distances, which may limit equipment selection. Uniform subregions or sections along the centerline can also be delineated. The properties of bulk solids, which can influence the equipment design, include density, fragment size distribution, fragment shape, friction angles, water content, stickiness, dustiness, leakiness, and friability. Chemical properties affect pollution, cementing, and oxidation and should be considered, although these do not influence the design or selection of handling equipment that contain the bulk solids for relatively short periods.

ACTIVITIES. Information on equipment activities is essential and comes initially from manufacturers. It covers normal vehicle motions and the total systems. (The latter has been extensively treated in 17.2.1.1, the former in 17.2.4.4.) These systems interact, and their impacts on the excavating and bulk handling system are of immediate concern. The ground-control system, for example, constrains with small spans, large support structures, and mobile bolting machines. Life support impedes through its demands for ventilating, water handling, and escape facilities. Normal support introduces impediments of power distribution cables, supply equipment, maintenance, and cleanup, as well as the personnel required to provide these services. Within the excavating and handling system itself, interferences also occur. All systems must be accounted for spatially, and contingencies must be anticipated. An excessive number of impediments represent a hazard.

EQUIPMENT. The numerous types of equipment available have already been inventoried, classified, and broadly described in 17.2.2. The major distinctions are between excavating and handling equipment and also between mobile (cyclic) and fixed (continuous) types. Intermediate equipment types exist, such as load-haul-dump units and rotating excavators. Manufacturers' specifications range from being highly detailed to severely condensed and rarely provide cost data. Each component of the equipment must be systematically addressed and understood. These include the job-end, running gear, power train, chassis, and operator's compartment. For fixed equipment, structural supports, accessories, and monitors require attention. Vital statistics, which relate horsepower to capacities (or to LMO), to dimensions, or to weights are very useful (Adler, 1986; Basu and Lineberry, 1988). It is important to note that it is common practice to customize excavating and handling equipment to meet operators' specifications, particularly for equipment with a value over about US$100,000 and especially in times of intense competitiveness in the minerals industry.

17.2.4.4 Suitability Analyses

GUIDELINES. Guidelines match conditions to equipment types. This information comes from texts, handbooks, periodicals, company files, and personal experience. While sometimes denigrated as "handbook engineering," its role is extremely important for quick and rough solutions, yielding orders of magnitudes and trends. Guidelines do have serious deficiencies, being often vague, incomplete, disorganized, unsubstantiated, and even contradictory. The basis for preference is often implicitly economic and only for normal circumstances. For excavating, tables are available comparing equipment to rock classes and mining methods to broad classes. For handling, there are tables relating equipment type to running surfaces, grades, distances, velocities, and acceleration. Mixed tables exist, covering flexibility, production rates, engineering intensity, site preparation, and costs. While assembling appropriate guidelines, the database can also be strengthened. This base includes values for limits and failure analysis (safety factors, strengths, and speed limits), performance analysis (fill and derating factors, fixed times, and availability), consumption rates, and estimates for equipment life. While it is deemed inappropriate to provide an exhaustive reference list here for such an area requiring broad treatment, the reader is directed to vendors' pocket guidebooks and performance handbooks, texts, and handbooks in the field of civil engineering construction and excavation, underground mining methods handbooks, and earlier editions of this *Handbook.*

EQUIPMENT TYPE. In following the selection procedure through the guidelines or in performing the evaluation procedure, with the equipment itself as the given information, the equipment type has now been established in both cases. A concentrated effort now is necessary to describe functional aspects and especially possible variants. Additional background can be acquired for such items as (1) history of the equipment type's use, (2) applications, (3) comparisons, and especially (4) potential failure modes. Size determination is deferred to geometric design.

OPERATIONAL ACTIVITIES. For each type of equipment, both surface and underground, there are characteristic motions associated with their normal operation. These operational motions, the study of which has been referred to as *operational kinematics,* is another essential ingredient of a successful suitability analysis.

Normal vehicular motions for mobile equipment, largely in the horizontal plane, can be categorized as (1) the I or shuttle pattern (shuttle car), (2) the O or loop pattern (ramcar), (3) the Z or parallel shuttle pattern (bulldozer), and (4) the Y or backup and turn pattern (scoop). Combinations also exist. The job-end motion can also be described as the boom and bucket digs, loads, or dumps. Horizontally, the boom often can swing through a specified arc, and vertically it describes slices or cuts. Vertical cuts can be classed as (1) at-grade (continuous miner), (2) below-grade (dragline), and (3) above-grade (shovel). Combinations again exist. Whether full- or partial-face extraction occurs must also be noted.

For fixed equipment, mostly mine plant machinery, the operational equivalent involves setting up, taking apart, and moving, as the working face advances or retreats (e.g., conveyor belts). This also applies to many support facilities and accessories. Since determination of the operational activities closely describes the equipment operator's job, the operator should be consulted in order to define intangibles such as contingencies, general problem areas, and to provide tips for safety and efficiency. This material, often subjective in nature, can be supported by appropriate field measurements and then converted into engineering terms.

ENCUMBERED SPACE. The cumulative effects that result from merging the conditions and activities is termed encumbered space (see 17.2.1.2). For underground coal mining, the worst-case situation, there is the potential for disaster from a safety viewpoint and the need for a clearer understanding of derating factors for production estimation.

GEOMETRIC DESIGN. The term geometric design derives from highway engineering. Using templates, sections, and layouts on graph paper or, alternatively, computer-aided design packages, equipment can be spatially matched to the encumbered space in which it must operate. If this is not done, equipment can get wedged, suffer glancing blows, and possibly collide. In the selection procedure, this establishes the equipment size; in evaluation, it determines the workplace dimensions.

For velocities of less than about 2 mph (3 km/h), static relationships apply and two concerns exist: to provide *clearances* and *reaches.* Clearances require that special attention be given to turning radii, dumping heights, and boom interference. When shifting or moving equipment between workplaces, clearances again must be considered. Reaches or ranges provide maximum digging and dumping heights (or depths), radii, and swing angles.

When velocities exceed 2 mph (3 km/h), dynamic effects are added to the static ones. Velocity-clearance curves must be considered, along with lines-of-sight for emergency stopping (Lineberry and Adler, 1984).

This analysis will yield a maximum acceptable size of equipment, with a minimum reach. In the selection procedure, the next step is to use the vital statistics (Adler, 1986) to approximately determine the horsepower, capacity, and weight and to pick a particular model. A complete set of individualized manufacturer's specifications then becomes useful. In evaluation, conditions and activities can now be modified to fit the equipment.

LIMITS AND FAILURE. Limits and failure will be treated here in simple terms, ignoring their complex and often progressive nature. Ideally, modes of failure are enumerated, their severity described, and their probability estimated. Although an attempt has been made to accomplish this (Adler, 1989b), limits and failure remain perplexing difficulties within the area of equipment selection and evaluation. Whenever possible, limits and failure should be considered prior to the performance analysis to reduce iterations. Failure falls in three broad categories, consistent with the earlier treatment of the basic engineering dimensions: (1) geometric failure, (2) material failure, and (3) time-related failure. Geometric failure has already been considered in terms of geometric design (see earlier in this section).

Material failure is of varying severity. Modes include (1) rupture and separation (belts, cables), (2) penetration (bearing surface, tires), (3) buckling (booms), and (4) cuts and bruises (belts, tires). In addition, skidding and overturning can be dangerous, and spillage should be avoided. The effects of water are obvious and particularly troublesome in mining.

Time-related failure is more subtle. It includes decreased life, capacity, availability, and performance, as well as increased wear (abrasion), maintenance, and consumption. Most weak spots tend to concentrate wear and then accelerate deterioration, especially for haulways, tires, and belts.

17.2.4.5 Performance Analysis

The performance analysis represents the end-product, providing the rate of doing a specific job. All analytical stems are involved, having first been sorted out and screened for suitability and failure.

TYPES OF PERFORMANCE. Several types of performance exist, the foremost being production rate or *LMO*, in units of tons (tonnes) per hour. Next in importance is the advance rate, in units of feet (or meters) per shift, which also corresponds to the penetration rate for boring. The advance rate is convertible to production by:

$$LMO = AR \times \rho \times SF \times A \times F \qquad (17.2.1)$$

where *LMO* is loose mean output in tons per hour, *AR* is advance rate in feet (or meters) per shift, *r* is in-place rock density in lb/yd^3 (kg/m^3), *SF* is swell factor in %, *A* is cross-sectional area in ft^2 (m^2), and *F* is a conversion factor.

Clearing, grading, and ripping rates, in acres per hour (square meters per hour), apply to site preparation. Other performance rates include those for bolting, timbering, moving, construction, and clean up. Equipment-specific measures of performance include mechanical availability, utilization, and overall efficiency.

PRODUCTION ANALYSIS. For cyclic equipment, the production rate is given by

$$LMO = (CAP/TIME) \times GF \qquad (17.2.2)$$

where *CAP* is capacity in yd^3 or tons (m^3 or kg), *TIME* is cycle time in minutes or seconds, and *GF* is a general factor to account for nonstandard conditions and conversion factors.

For short cycles of less than about 2 minutes (e.g., shovels, hoists), the cycle time and even the production rate can often be obtained directly from existing tables. Load-growth calculations can be very useful (Adler and Naumann, 1970). For long cycles, greater than about 10 min (e.g., front-end loaders, trucks), the cycle time becomes

$$TIME = FXD + VAR \qquad (17.2.3)$$

where *FXD* is fixed time in minutes and *VAR* is variable time in minutes.

Fixed times include delays, loading, dumping, and maneuvering, and are often obtained empirically or measured in the field. Variable times (travel times) are determined from duty cycle calculations, which include velocities, accelerations, and distances. For continuous operations (e.g., belt conveyors), the production rate is determined by:

$$LMO = A \times \rho \times VEL \times GF \qquad (17.2.4)$$

where *VEL* is velocity in fpm (m/s). In this case, *GF* is the product of load efficiency and fill factor.

CONSUMPTION RATES. Consumption rates are often estimated from the production rates, as with electric power in hoisting and battery selection for locomotives and scoops. For fuels, lubricants, explosives, and replacements, consumption rates are usually obtained from empirical data or manufacturers' specifications and their empirically derived information and guidelines.

17.2.4.6 Completion Analyses

PUBLIC POLICY. Checks on public policy should be done early. These are legally mandated regulations, codes, and contracts, covering safety and health, environmental protection, and labor, among others. Public policy checks are intrinsically controlled, monitored, and regulated, with much of the activity relating to equipment selection and evaluation going unrecognized.

HUMAN FACTORS. Human factors analyses serve to limit the mental and physical stress on an operator to make the job safer and more efficient. This is done by converting the operator's experience into simple engineering terms. Human factors is a significant specialty in industrial engineering and psychology, and, since modest beginnings in the 1960s, human factors (ergonomics) is now attracting wide attention in underground mining (Sanders and Peay, 1988).

The primary application of human factors in equipment selection and design is through the workload intensity concept (Adler and Lotfi, 1983). The mental stressors on an equipment operator due to conditions and the physical tasks due to activities are summed to obtain a plot of total stimuli vs. time. These data come largely from the operational activities and the performance analysis (see 17.2.4.4 and 17.2.4.5), where velocity was determined. When peaks in the workload curve violate a specified limit, corrective action is required. This can include altering conditions by (1) improving the human environment (illumination, noise, temperature, humidity), (2) improving the haulway (clean up bottom, broaden curves, widen cross sections), and (3) reducing congestion and interference (increase percentage of usable space, reduce traffic). Alternatively, activities can be staggered, simplified, and even slowed down to alleviate task peaks. The equipment itself can be redesigned to decrease both stressors and tasks, with enclosed cabs, automation and robotization, more precise and responsive controls (including computer controls), and by providing greater stability and agility. Training to improve operator performance has made strides by the use of physical simulators, such as for continuous miners, shuttle cars, front-end loaders, draglines, and shovels.

Other important areas of human factors, following more traditional organizational trends, are (1) systems design, (2) human capabilities and limitations, (3) human error and accidents, (4) design of displays, controls, equipment, and tools, (5) work physiology, (6) effect of environmental factors on safety and productivity, (7) training and retraining, and (8) motivation and organizational development (Sanders and Peay, 1988).

FINAL CHECKS. All limits, failure modes, and suitability criteria, especially those previously deferred, must now be examined or reexamined. Further iterations may still be necessary. When everything is satisfactory, a single feasible solution has been produced, one among several alternatives.

17.2.4.7 Economic Analysis

The profitability of each feasible alternative must be addressed before a final decision is made. All costs are estimated in dollars per unit time, and then converted into dollars per ton by using the production rate. Knowing the market value yields the unit profit, and multiplying this by gross tonnage gives the gross profit. Profitability measures for a specified time period can then be employed, such as (1) payback period, (2) discounted cash flow, and (3) rate of return. When evaluating the site conditions for possible upgrading, a cost-benefit analysis is appropriate. In any final decision, familiarity with the equipment, along with other intangibles (e.g., training requirements, level of confidence, after-sale service, and technical support), all play an important role.

17.2.4.8 Summary and Conclusions

A basic procedure has been developed to determine the performance rate of excavating and bulk handling equipment. By systematically involving the conditions, activities, and equipment characteristics, along with periodic suitability tests, the performance rate can be determined most efficiently. The analysis applies to all types and sizes of equipment—underground or surface, mobile or fixed, large or small—and can be used either to select or to evaluate. A flow chart guides the procedure through specific phases until one of several feasible alternatives has been found. The procedure is then repeated for other possibilities, with the final choice being largely determined by economic factors.

Numerical examples demonstrating equipment selection procedures appear elsewhere in this *Handbook* (Chapters 9.4 and 13.3).

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Chapter 17.3
MINE PLANT LAYOUT

SCOTT G. BRITTON

Mine plant layout is a general term for describing the process of configuring a complex and often expensive portion of an underground mine. It encompasses the placement of all development facilities such as buildings and structures, machinery, pipelines, power lines, equipment, cables, ponds, roads, rails, and other auxiliary works needed to support any underground mine activities.

This chapter provides the reader with (1) an historical look at mine plant layout, (2) a general understanding of surface, shaft, and underground plant structures and systems, and (3) guidelines for basic mine plant layout.

Mine plant layout is defined as the design for integrating all structures, systems, or activities required to support the mining and processing of ores, minerals, or fuels for economic gain. Throughout man's history of mining endeavors, mine plant layout has required considerable attention, and in today's complex mining world, detailed planning for the mine plant layout is even more important and necessary.

As an example of this, consider the decision for choosing site locations for the original openings and plant buildings of a new mining complex. These decisions are influenced by a number of major factors, namely, (1) depth of cover, (2) location with respect to the reserve perimeter or ore body, (3) surface topography, (4) proximity to contract services (e.g., power, water, etc.), (5) location of railroads and market destinations, (6) geological structure, (7) proximity to population centers, (8) regulatory and environmental constraints, and (9) ease of access by personnel. Additional minor factors are numerous and relate specifically to each mining site, company objectives and strategy, and economics of the project.

Mine plant layout is broadly divided into three major subcategories: surface, shaft, and underground plant. As Fig. 17.3.1 depicts, each category describes the general location of the layout but does little to identify what each category encompasses.

The *surface plant* commences at the entrance to the property and mine opening site. This is generally seen in the form of roads, fencing, drainage and runoff ditches, lighting/power lines, and other items needed to provide the site with materials and services. The surface plant extends throughout the area to a point at the shaft collar or slope/drift portal and encompasses all the buildings, yards, controls, surface control structures, and equipment needed to service the underground mine. The surface plant also includes the hoisting headframe(s), hoist house, motors, cable, skips, hoists, and all other items included with the aboveground structure of the shaft.

The *shaft plant* subcategory begins at the shaft collar and consists of the airways, pumps, piping, water collection structures, communication and power lines, transportation systems, and other interconnecting components between the surface and underground mine operations. The shaft plant continues to a point where the underground development workings originate. At this point—and for anything inby this point—these systems are part of the *underground mine plant.* These systems would include, but not be limited to, ventilation (including heating and cooling), drainage, transportation, supply and materials handling, mine power, and communications.

The reader should distinguish between designing the mine plant for large and small mines. There is a cutoff that roughly parallels the expectation of management on three basic parameters of the mine complex: (1) the duration of the underground facility, (2) the profit expected from the mine, and (3) the needs of the mine for auxiliary services. As a rule of thumb, large or long-lived mines (i.e., with a life greater than 10 years) almost certainly need detailed and thorough mine plant engineering and a large layout area. Smaller mines with low production volumes and with an operational life of three to five years may have a portable surface plant, little or no shaft plant (especially with drift or adit type coal and metal mines), and a very basic underground plant layout.

Such a difference in size will ultimately influence the detail required for each plant component design. It will not, however, reduce the need for considering the component during the layout effort. For example, a large mine using a shaft entry system will in all probability have some type of surge bin storage system located underground at a point near the shaft skip pocket or hoisting shaft. This bin is used to lower the potential disruption and to maintain a high continuity of mining operations should a problem develop in the shaft hoisting system itself. For large coal mining operations, horizontal bunkering systems are becoming popular as an answer to this storage/continuity problem. In contrast to this is a small mine that is accessed via a series of adits or drifts. Although some thought is given to storage, it is probably limited to the stockpile area located on the surface and connected directly with a beltline or rail system.

17.3.1 HISTORICAL ACCOUNTS OF MINE PLANT LAYOUT

As an overall introduction of the subject, the first serious treatment of plant functions within a mining context for reference works was *De Re Metallica,* published in the Middle Ages by Georgius Agricola (1556). In this landmark work, Agricola wrote on the machines and techniques used to support the mining of ores and their preparation. Pumping, ventilation, and processing functions were the main interests at that time, and Agricola devoted much space faithfully describing these techniques and operations. The text provided word and picture descriptions, something rare in books at that time. The pictures were carved from drawings into woodcut blocks and used in a new invention of the time, the printing press. Figs. 17.3.2 and 17.3.3 are copies of woodcut diagrams included in the book and demonstrate the extensive treatment of such discussions. This text retained its currency and enjoyed wide usage until the 19th century.

Since the publishing of *De Re Metallica,* there have been fragmentary accounts of mine plant design in other texts. Every modern mining engineering text since that time (e.g., Peele, 1941; Lewis and Clark, 1964; Cummins and Given, 1973; and Hartman, 1987) has at least one discussion concerning the design and construction of various mine plant structures such as shafts, drainage and pumping stations, processing plants, haulage systems, and their layout. Specific to underground coal mining, several discussions appear in Crickmer and Zegeer (1981), Brit-

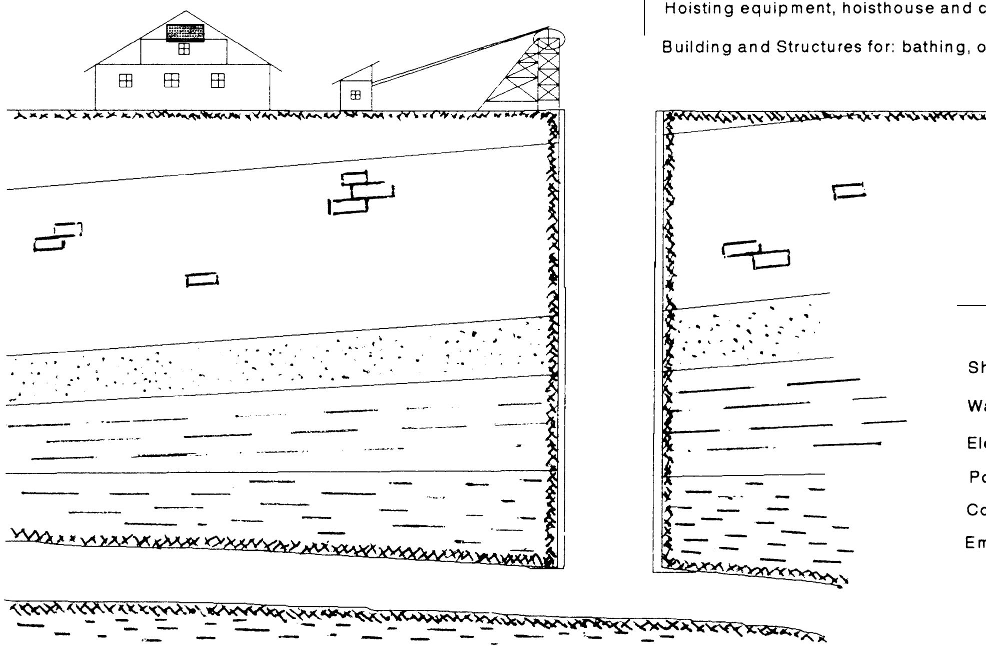
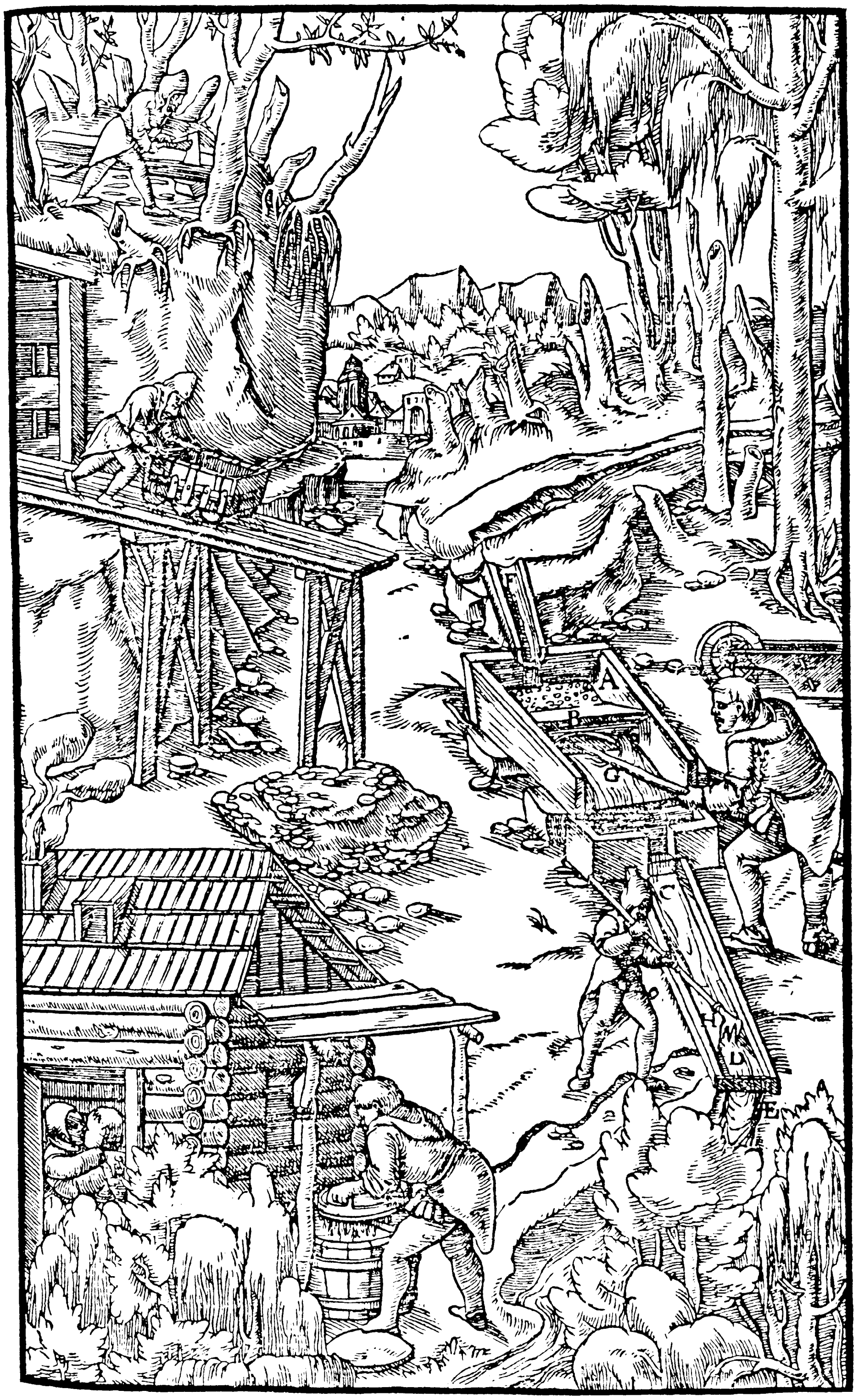


Fig. 17.3.1. Mine plant layout is broken into three broad areas.

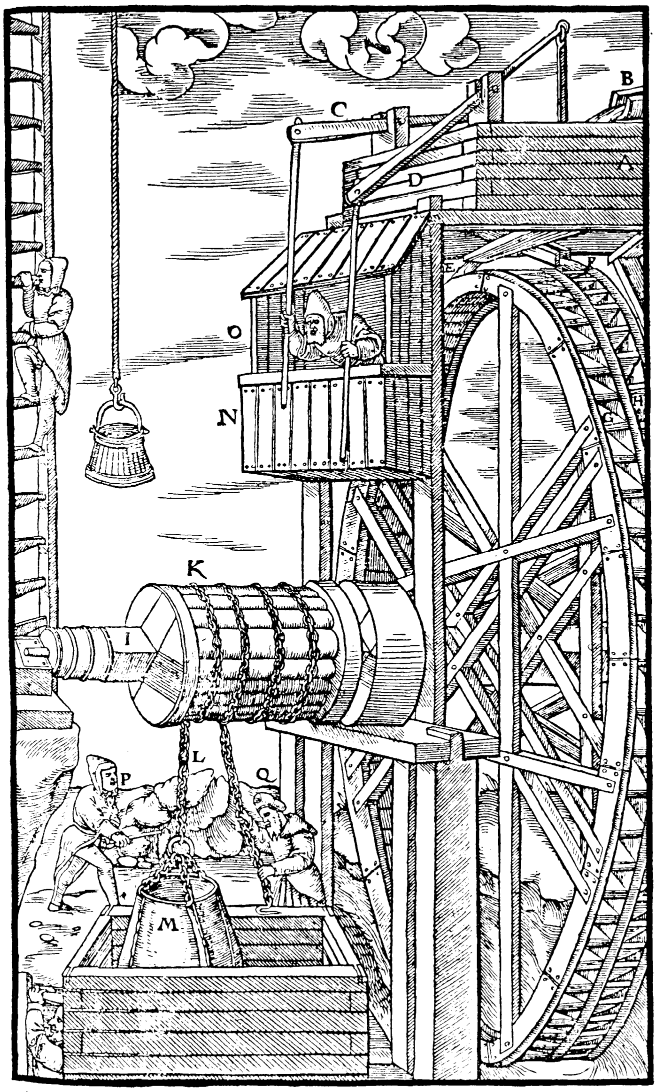


A—Box. B—Perforated plate. C—Trough. D—Cross-boards. E—Pool. F—Launder. G—Shovel. H—Rake.

Fig. 17.3.2. Wood cut drawing (*De Re Metallica,* Dover edition).

ton (1983) (Figs. 17.3.4 and 17.3.5); and Stefanko and Bise (1983). However, the chief drawback to these accounts has been and still remains the problem of fully discussing the variety of solutions needed to solve mine plant problems when confronted by unique sets of design circumstances. It is almost an impossible task.

However, one source stands out in its approach to covering the variety of solutions found within the mine plant. It is a definitive reference called *Mine Plant,* authored by Tillson (1938) and published by SME-AIME. In this unique text, Tillson did away with the usual word descriptions for plant functions and provided complete engineering drawings of specific mine-related structures used at that time. The text covers nearly all major plant functions and equipment design, including ore processing, shaft design and construction, headframes, pumps, fans, conveying, transportation, and so forth. The author attempted to



A—Reservoir. B—Race. C, D—Levers. E, F—Troughs under the water gates. G, H—Double rows of buckets. I—Axle. K—Larger drum. L—Drawing-chain. M—Bag. N—Hanging cage. O—Man who directs the machine. P, Q—Men emptying bags.

Fig. 17.3.3. Wood cut drawing (*De Re Metallica,* Dover edition).

address layout planning by showing specific solutions to actual problems. Unfortunately, the material is dated and is primarily useful as a historical look at recent bygone mining practices rather than as a guide to modern mine plants. Figs. 17.3.6 and 17.3.7 are representative of the level of detail and complexity found throughout the text. *Mine Plant* currently suffers from not having any updated material. The text does not consider surface mining support or layout. Finally, it is out of print and difficult to find.

The preceding discussion is intended to leave the reader with the impression that it has been difficult trying to cover a complex subject within the changing mining field when no specific set of design parameters exist. However, behind any design layout there exists a general analysis approach required to successfully design a plant layout. It is this approach which will be emphasized in the following discussion.

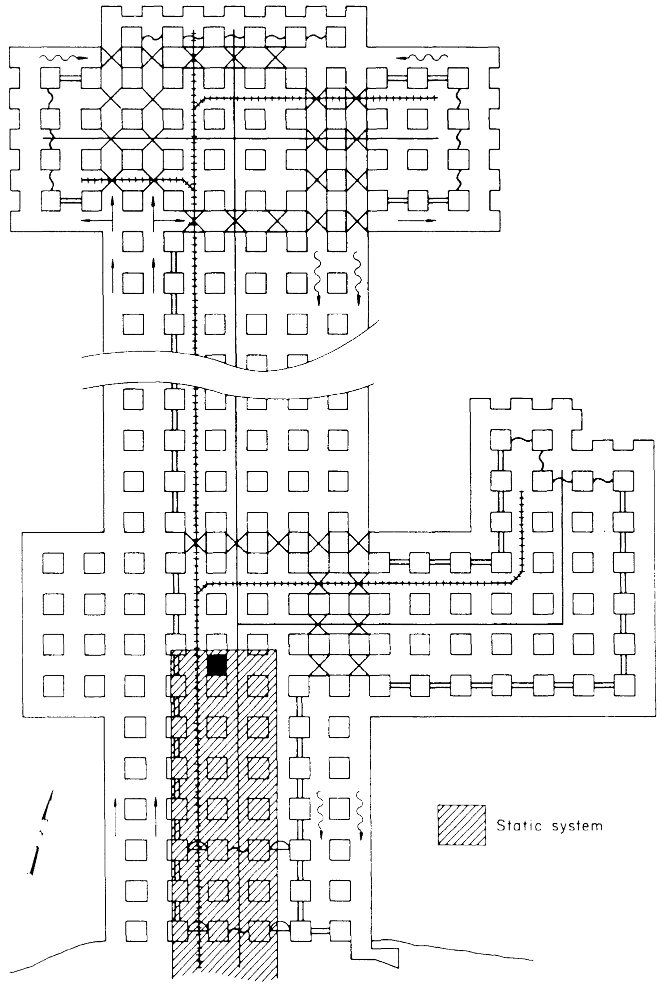


Fig. 17.3.4. Underground mine plant layout.

17.3.2 GUIDELINES FOR BASIC MINE PLANT LAYOUT

The mine plant layout is the backbone of efficient mine and plant operation. Whether in a remote mountain valley in Alaska, or on the outskirts of a major metropolitan area, mine plant design and layout does not vary in terms of the guidelines needed to successfully approach its design. These guidelines are considered both strict (because they are required for any design) and yet flexible (because they must meet each individual or site specific problem presented).

Guidelines for plant layout can be thought of as tiers of a pyramid. These guidelines have changed over time and will continue to change as new technological developments and government regulations impact their design. There are primary guidelines that govern the development of the mine plant design, secondary guidelines to aid in the interplay of subsystem designs, and tertiary guidelines to govern the overall fine tuning of the plant design to conform with regulatory approvals and unforeseen circumstances.

Primary guidelines are those guidelines that allow the engineer to (1) conform to effective, tried-and-true design practices, and (2) take advantage of current industry standards for the type of design being laid out. They are simple, straightforward, and based primarily on common sense and experience. There are five guidelines:

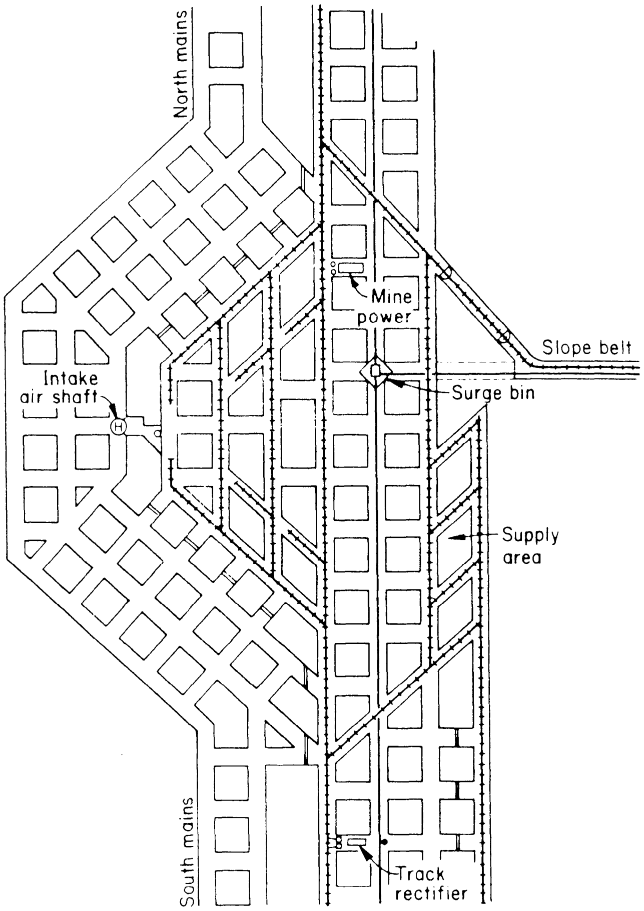


Fig. 17.3.5. Underground shaft/slope bottom layout.

1. Primary mine and preparation facilities should be designed to last the the life of the mine unless other circumstances (economics, safety, regulations, etc.) dictate a change.

2. Size of the stockpile areas, mine supply yard capacity, bathhouse space, and throughput of the preparation facilities should reflect the expected maximum design output of the mining operations. Limitations to this guideline include space, climate, and topography considerations, such as steep hillsides in West Virginia, deep cold temperatures of the arctic, or locations near urban areas.

3. Primary design components such as power, water, and access routes should reflect the most current available technology. For example, in underground coal mining, mine power voltages of 20 years ago reflected the general thinking that 440 V was adequate for primary production machinery. Today, a new trend towards 950 V is being heralded as the best available. Or, for example, heap leaching recovery of gold ores from waste tailings was unheard of 20 years ago, but today it is the preferred technology for most secondary recovery operations.

4. The shaft plant design should reflect sufficient flexibility in the placement of piping, cables, machinery, and wires to allow individual repair or replacement without significantly impacting any other component.

5. Shaft plant systems should be designed for the life of the mine unless circumstances dictate otherwise.

Secondary guidelines are concerned with the interdependence of the various mine plant systems. This can be best illustrated by examples, such as process water and mine dewatering, or underground machinery repair shops with surface repair facilities, or even power for the mine fan as it relates to other underground and surface uses. Secondary guidelines are used during the second stages of mine plant layout after the initial design considerations are determined, hence the term secondary. There area three secondary design guidelines:

1. Competing uses for primary resources such as power and water should be designed to complement one another and be able to service all needs during peak demand periods. For example, when a thermal dryer fan motor is turned on, the portal building lights should not dim or production machinery underground slow down. However, the rated voltages of the motors should be similar and circuits providing the power need to be isolated and properly protected.

2. The layout of any system should consider available built-in safety considerations so that it can function without problems (given reasonable maintenance), and that rupture, breakage, or failure of one system cannot directly affect another system (i.e., the "domino" effect). For example, power boreholes have become popular for several reasons, but mainly because they keep the cable out of the hoisting shaft and separated in view of the danger and potential damage if the electrical cable were to blow up or the skip cage were to derail and cut the cable.

3. The layout should, to the greatest extent possible, minimize any waste or inefficiency in repetitive operations between systems. For example, the mine material supply yard should be located next to—or as close as possible to—the point of entry underground (either a slope, drift, or shaft). It should not be located across the valley or behind the portal building. Likewise, the hauling of ore or waste should be so directed to provide the shortest route that is free of nuisance traffic and environmentally sound.

The final set, *tertiary guidelines,* is directed toward the layout of systems when competing regulatory agencies, outside organizations, and other unforeseen circumstances compel a change in the design. These instances generally occur when competing uses for natural resources are regulated by governmental actions based on law. They are set apart from the other guidelines because they do not impact the design process until the design itself is under review by external authorities such as state and federal environmental or safety agencies, labor unions, and the general public. These guidelines can be summarized as follows:

1. To the greatest extent possible, input from such agencies or outside organizations should be gathered early in the design process. Although, for example, an environmental permit may not be submitted until the end of the design process, it is prudent to obtain as much common understanding while changes can be made relatively easily. Public comment sessions are a necessary ingredient to good public relations and acceptance of the desired design layout.

2. In public debates or public meetings over technical design issues, "facts speak louder than words". For the mining engineer, this means being prepared with correct and complete data, and that facts should be stressed against emotional arguments. This is especially true for design issues surrounding environmental and safety issues. Both issues are extremely sensitive for the mining industry as a whole, and it generally falls to the mining engineer to correctly plan to mitigate any adverse impacts due to mining or milling activities.

In being effective with the use of "facts", the engineer should be able to reduce the numbers to easily understood graphs, charts, and tables. Audiences without technical training must be able to make decisions in favor of the mining project, its design, and its impact on sensitive issues by using this data. The use of impartial or outside expertise in communications may be required.

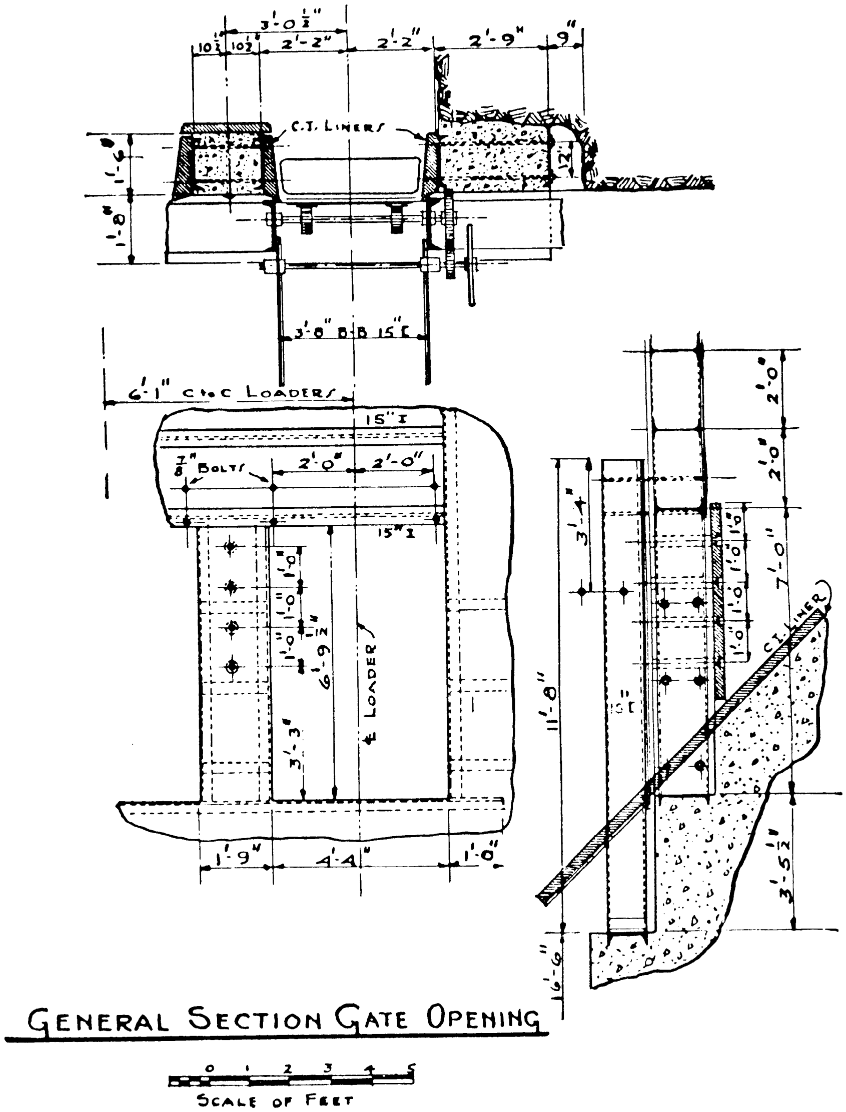


Fig. 17.3.6. Typical drawing from *Mine Plant* (used by permission of SME-AIME). Detail of gate opening for skip loader, Homestake Mining Co., Lead, SD.

3. Acceptable alternative solutions should be developed for those items within the design layout that may represent points of contention between various groups. These alternatives may serve to avoid costly delays or redesign problems later in the project design or construction phases. Refuse piles, air shafts, roads, plant effluent discharge, and other noticeable aspects related to mining operations should have their designs with acceptable alternatives or changes already considered by the mining engineer.

4. Keep design parameters conforming to known standards to aid in maintaining effective project design. For example, in the US, electrical installations should reflect current National Electrical Code Standards as well as local utility construction standards, if necessary. Mine materials should conform to US Bureau of Mines published standards or ASTM standards when cited by other government agencies such as the Mine Safety and Health Administration. Attention to health standards from the National Institute for Occupational Safety and Health or Occupational Safety and Health Administration increasingly are demanded during the design phase by third-party groups and corporate risk managers.

If these standards are not observed during this time, and there is a challenge over the particular design element, the engineer will be called to change the element to conform to the standard unless an overwhelmingly compelling reason for deviating from the standard is present. Changes to incorporate standards can be expensive and complicated after the design stage, so it is important to consider them at the appropriate time.

This concludes the basic guidelines needed to design adequate mine plant systems. Because of the infinite variety of design elements and the changing nature of technology, these guidelines are not hard and fast in terms of specific rules. They do, however, provide the basic fabric over which the mine plant design can be developed.

17.3.3 PLANT LAYOUT METHODOLOGY

Plant layout methodology differs from the above discussion in several fundamental ways. First, the methodology puts into practice the guidelines outlined prior to this point. Secondly, the methodology focuses on the procedures needed to bring mine plant concepts to fruition.

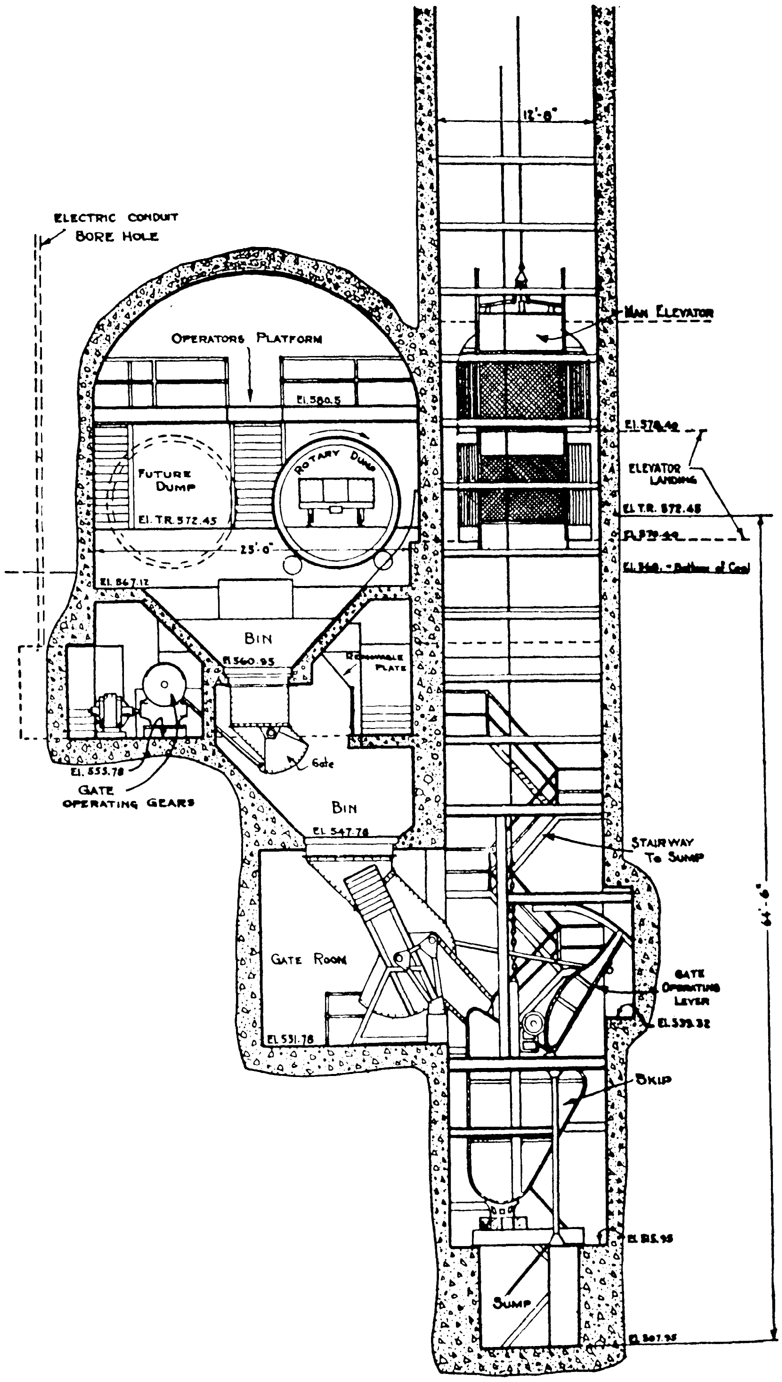


Fig. 17.3.7. Typical drawing from *Mine Plant* (used by permission of SME-AIME). Shaft loading pocket to hoist 100 fifteen-ton skips per hour, Buckeye Coal Co., Nemacolin, PA.

Basic mine plant layout is centered on devising a plan to balance the required operational goals of the company or business with the cost, time, and technical constraints of the project at hand. This plan is composed of numerous design and analysis steps that can be used again and again as needed. These steps are reviewed during the initial stages of mine development or project analysis when a planning team or engineering team is estimating the basic costs needed for extraction and ore processing.

The first type of mine plant layout always occurs in the initial efforts after one or more deposit access sites are identified. In this stage, a "thumbnail" sketch of the expected operations is laid out for all the potential opening sites. These expected operations are the product of initial mineral production and processing considerations, such as whether the ore is to be concentrated or milled onsite, or the mined material should be washed (either onsite or off), or whether there will be an onsite power plant or power is available from a local generating station, and so on. These considerations surface during the exploration program and are raised by the technical and marketing teams prior to the actual detailed engineering efforts. The engineer will note any special cost considerations for any one of the opening sites, if necessary. The sketch will encompass all known plant operations of major importance for all three plant areas.

From the thumbnail sketch, the engineer and others can begin the painstaking work of design—and with it—estimating costs of construction. The design consists of developing specifications for each subsystem, given the guidelines provided earlier. The costs can be obtained in the form of request for proposals to construction and design firms or developed from in-house resources. It may be necessary to develop detailed plan and construction drawings to fully support basic plant layout. At this time, the engineer will begin to piece together each supporting element based on the guidelines mentioned earlier. Firms experienced in the design and construction of particular elements (slopes and shafts, processing plants, grade work, etc.) are called to provide detailed plans along with quoted prices for completing such elements. Other firms capable of supplying equipment are contacted to provide quotes on specific items required by the design layouts.

During this stage of plant layout and design, the primary role of the engineer or planner is one of coordinator and compiler. It is his/her job to ensure that all elements required of the design are covered with detailed specifications and that these specifications coincide with other components within the overall design. It is not uncommon to spend a considerable amount of time modifying submitted designs, changing specifications to obtain a more efficient and cost-effective layout.

Once the basic parameters are satisfied, filling in the details between component systems becomes one of redesigning in relation to the overall plant layout and the differences between each component system. For example, power required for the site must be sufficient to run all operations, from the mine to the plant to the area lights. The power system should interface with each substation, breaker box, transformer, and other switching gear. Or, for example, water supplies must be adequate to support bathing and processing needs, sewage, and treatment needs, and be able to handle peak demands. Water must be filtered and treated before release into the environment. Each system has to be looked at before the mine plant can be pieced together.

The rule of thumb in determining the hierarchy of design after the initial plans are made can be simply stated as beginning from "the ground up," that is, the first system to focus in is the one most basic to operations. This progression of design is followed throughout until each system has been covered. For example, roads, ponds, and land contouring are the basic components of the surface plant. From this base, power lines, disconnects, and substations are added. The next system would be the fresh/process water system, followed by the supply/material yards, buildings, slopes, shafts, boreholes, hoists, fans, etc. Although each component system can be designed by individual teams, allowances must be made for limiting parameters of prior systems.

Within the auspices of the plant layout, the design engineer team begins to place the physical and structural components of the design to the actual plant site when all the components of the system are blocked out and defined. Especially important is the integration of the shaft plant and surface plant areas (the shaft collar, hoisting system, etc.) as well as the interface between the shaft and underground plant areas (sumps, power boreholes, skip stations, etc.) that must consider geologic stability. Where recognized areas of rock or strata weakness, or other unsuitable surface or subsurface areas are identified, the plans must be altered to work around or mitigate these circumstances.

In considering the layout of each necessary mine plant item, care must be given to human factors, cost, efficiency, and risk in the layout. For example, one does not lay out a design which results in putting a hazardous chemical storage depot next to an active travelway. For further clarification, thought and planning in the layout should consider (1) normal traffic patterns for employees, mobile equipment, customers, and vendors; (2) placement of aerial and underground pipelines and cables; (3) patterns of material flow to minimize health and environmental hazards (dust, water contamination, trash, etc); (4) orientation of facilities to minimize construction costs (excavation, etc.); and (5) placement of systems to take advantage of—or not be affected by—prevailing weather conditions.

Underground plant layout requires additional consideration for the confined space dimensions, the hazardous geologic conditions, and the dependency on artificial environmental systems to maintain a safe working environment (see the discussion on unencumbered space in 17.2.1.2). These additional considerations are not present in the surface plant design and require considerable practical experience on the part of the engineering design team.

Where materials and ore transportation systems enter the mining environment there is a tendency to underdesign the long-term nature of the transition area. For example, skip pockets, surge bins, sumps, or other systems that must endure the heavy operation imposed by life-of-mine duty are sometimes designed without consideration for changeouts, upgrading, or preventive maintenance activities.

Mine plant layouts, whether surface, shaft, or underground, are generally placed on drawings called schematics, plats, or layouts. All of these types of drawings orient the items placed on it to an exact scaling and position. In all cases, common sense and experience can help to reduce any inadvertent shortcomings.

The methodology of mine plant layout can be summed up as a straightforward application of good technical judgment when piecing together a "jigsaw" puzzle of interconnecting systems.

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Chapter 17.4
CONSTRUCTION OF DEVELOPMENT OPENINGS

KOT F. UNRUG

17.4.1 LOCATION AND DESIGN OF VERTICAL SHAFTS

17.4.1.1 General Considerations

Shafts are the most important capital openings of deep mines, providing all services for underground operations including fresh air, transportation of ore and supplies, personnel traffic, power (electricity and compressed air), communications, water supply, and drainage.

Depending on the depth of the mine, shaft sinking may consume as much as 60% of the mine development time. Because of this, a proper choice of a method to minimize sinking time and assure uninterrupted operation is of great importance. In determining the shaft diameter and hoisting depths (the latter especially for three-dimensional or multiseam deposits), future mining needs have to be evaluated beyond the first stage of the project. Generally speaking, it is better to overdesign the shaft in the first stage of the project life span than later to face a bottleneck, preventing an otherwise feasible increase of production or requiring the sinking of a second shaft. If shaft extension is foreseen, part of the area in the shaft cross section can be used for this operation without affecting operation of the shaft, serving production from any upper levels.

Technical data required for the design of the shaft consists of a project outline and determination of the approximate shaft location. The outline of the shaft project should contain a section describing its purpose and justifying selection of the shaft site. Also included in the outline should be a short description of the characteristics of the shaft and its function; the hoisting gear; the shaft capacity, diameter, and depth; the type of shaft lining; the main pipelines and cables; number and functions of shaft insets; the depth of the shaft sump; and the quantity of airflow through the shaft, with corresponding drawings and cost specifications.

Excavation design requires the gathering of pertinent data and solving particular elements of the design, the most important of which are

1. Description of the geologic column in the form of a table identifying the rock strata, their geotechnical parameters, and groundwater levels together with water heads, calculated inflow, and degree of chemical contamination, if any.
2. Determination of the shaft diameter, with justification.
3. Choice of shaft sinking technology and its justification.
4. Description of the shaft lining and a list of lining sections with their thicknesses.
5. The shaft foundations (footings), their locations and dimensions.
6. The shaft collar, its depths and foundation, thickness and construction material, kind and number of openings with their function, and dimensions and elevations.
7. The shaft sump, its depth, structural characteristics, pumping arrangement, and cleaning system.
8. Surveying data for particular shaft elements,
9. Calculations comprising ground and water pressure acting on lining, resulting lining thicknesses, shaft insets with their dimensions, and airflow capacity.
10. Timetable of construction, with such elements as preparatory works, shaft sinking, lining erection, shaft equipment installation, and liquidation of construction arrangements.
11. Cost specifications.
12. Drawings of the general mine layout, with shaft location, plan of shaft site (construction stage), and shaft cross section with an outline of equipment and compartments.

GEOLOGIC COLUMN AND GEOMECHANICS PROPERTIES OF ROCKS. Reliable geologic information based upon core drilling near the shaft site is required for shaft design. In a virgin area, more detailed investigation is required than on a site where shafts previously were sunk nearby.

The package of hydrogeologic tests is usually based on the determination of the lithological character of rocks along the geologic column; the number of water levels, their depths and thickness; the hydrostatic pressure in particular water levels; coefficients of filtration of particular water levels; indicated water inflow to shaft from particular water levels; and chemical characterization of the various water levels. The geomechanics package contains a description and granulometric analysis of unconsolidated strata, if applicable, angles of repose or internal friction, specific gravity, porosity and fracture description, rock quality designation (RQD), inclination of strata, and uniaxial compressive and tensile (Brazilian) strengths. Geophysical logging usually includes neutron density, self-potential, and resistivity.

An exploration hole is drilled 35 to 100 ft (10 to 30 m) from the shaft axis. Such a distance ensures that test results from the drillhole correspond to conditions in the shaft, without the hole itself having a detrimental effect on the sinking operation (e.g., interconnecting water-bearing levels, resulting in an erroneously high hydrostatic pressure). Since the hole is a major source of information for shaft sinking, it must be placed with utmost care and under the supervision of a geologist. Core and water samples should be preserved in their natural state for testing.

DETERMINATION OF WATER INFLOW INTO SHAFT. A shaft face intersecting a water-bearing stratum will collect water similar to the way wells collect water. Water inflow must be controlled by pumping; consequently, a depression cone is created around the shaft. When certain parameters are defined, water inflow into the shaft during sinking can be predicted. They include (in SI units) the coefficient of filtration K in m/day; head of the water table H in m; head of the depressed water table h in m; radius at the depression cone R in m; radius of shaft r in m; and depression $s = H - h$ in m. The inflow often has a decisive influence on the choice of sinking method (Fig. 17.4.1). There are several inflow relationships.

1. An inflow q in m^3/day into the shaft through the walls (free water level) is calculated from

$$q = 1.36\, K \frac{H^2 - h^2}{\log (R/r)} \tag{17.4.1}$$

2. An inflow into the shaft through the walls for water level with a head is

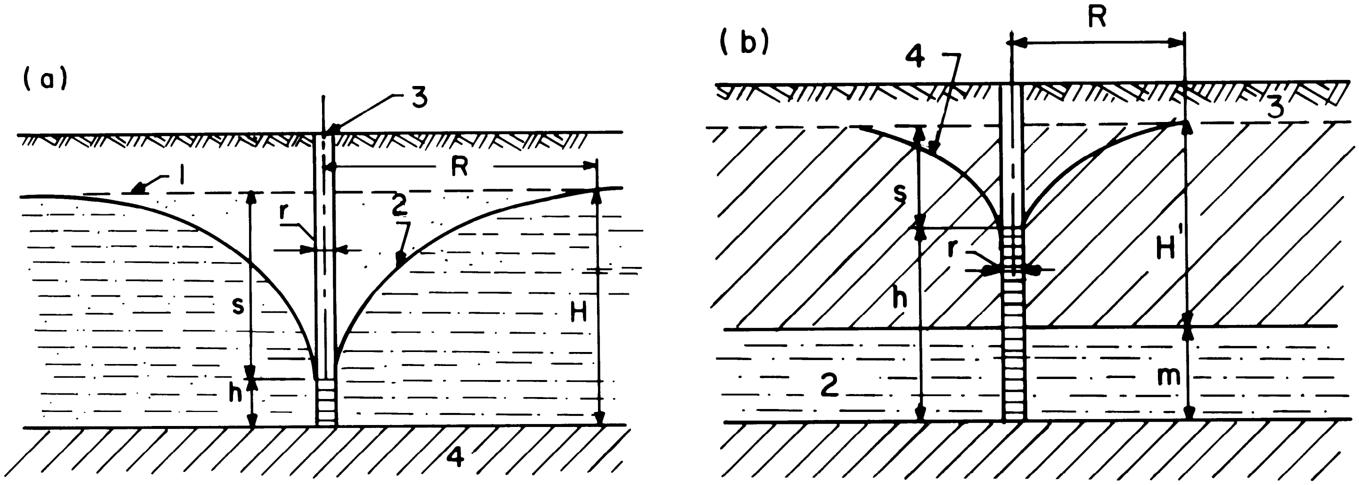


Fig. 17.4.1. Schematic of drawdown groundwater. (After Walewski, 1965.)

a) water-bearing strata with free ground level
1. initial groundwater level
2. drawdown water level
3. depression well
4. nonpermeable strata

b) water-bearing strata with head
1. nonpermeable upper strata
2. water-bearing strata
3. initial head of groundwater
4. drawdown water level

$$q = 2.73\ K\ m\ \frac{H' - h'}{\log\ (R/r)} \qquad (17.4.2)$$

where m is thickness of the nonpermeable strata.

3. Inflow through a flat bottom is

$$q = 4K\ r\ s \qquad (17.4.3).$$

4. Inflow into the shaft through the shaft walls in fissured rocks and free water level (simplified formulas) is

$$q = 2\pi\ K\ H\ \sqrt{r\ s} \qquad (17.4.4)$$

5. Inflow for this same condition but with a head is

$$q = 2\pi\ K\ m\ \sqrt{r\ s} \qquad (17.45)$$

The above formulas correspond to stabilized (equilibrium) inflow. Initial inflow will be greater.

SHAFT SITE SELECTION. The number of shafts in a mine depends directly on the daily production rate and the dimensions of the mining area. To obtain a minimum cost per ton of production, it is essential that an optimum balance between capital expenditure and operating costs be found.

Two Shafts—For a simple two-shaft mine, the location of a production/personnel/material/intake shaft is considered first, followed by the location of the ventilation shaft. Shafts can be located (see Fig. 17.4.2) in the central part of the property or on the footwall side of a deposit, with its major axis in the strike direction.

A central location of shafts in the mining area is most advantageous because transport costs (production, supplies, personnel) are minimized, and ventilation airflow routes to the production faces are also minimized. However, a central location causes losses of minable reserves within safety pillars and under the surface facilities at the mine site. In tabular deposits, with one-seam mining at moderate depths, it is the most effective system.

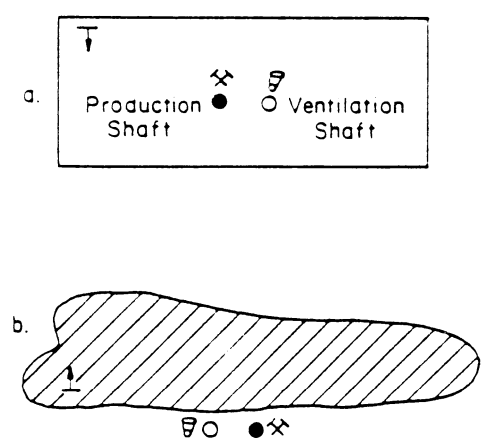


Fig. 17.4.2. Shaft locations for a two-shaft mine (a) centrally located and (b) on the footwall side of a steeply dipping deposit with its major axis in the strike direction. (After Jawien and Suchan, 1980.)

A side location (Fig. 17.4.2b and 17.4.3) increases the costs of development and transport by about 50% when compared with a central location (Fig. 17.4.2a), with simultaneous lengthening of ventilation airways. However, no part of the reserves is affected by the extensive shaft pillars that become essential for three-dimensional deposits at greater depths.

Three Shafts—Three shafts are required when the mining area is spread to the extent that a two-shaft arrangement is insufficient to provide the required ventilation. In such a layout, the intake shaft should have a larger cross-sectional area (e.g., 23 to 27 ft, or 7 to 8 m, in diameter), and ventilation shafts

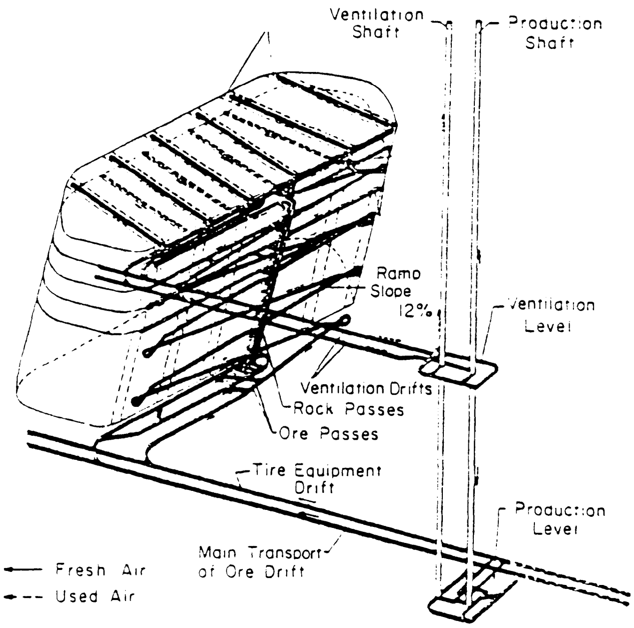


Fig. 17.4.3. Additional development resulting from side-shaft location for a magnetite ore deposit.

should be smaller (18 to 20 ft, or 5.5 to 6.0 m, in diameter). A centrally located intake shaft may also serve as a production and personnel/material shaft. Peripheral ventilation shafts should be placed along the strike direction when the mining property is two to three times longer in this direction than along the dip (Fig. 17.4.4a). In the case of a separated three-dimensional deposit elongated in the strike direction, the main production shaft is sunk in the footwall in the middle of the deposit. Ventilation shafts are located outside, on both ends of the deposit (Fig. 17.4.4b). When the mining area is elongated in a direction along the dip, the production shaft and one ventilation shaft are located in the middle, while the third shaft (second ventilation shaft) should be in the part of the deposit that is closest to the surface (Fig. 17.4.4c).

Four Shafts—A four-shaft system is necessary when the mine has a larger area and planned production is roughly double the size of a two-shaft mine. All four shafts should be approximately the same diameter and will have the following functions: (1) one production/intake shaft, (2) one personnel/material intake shaft, and (3) two ventilation shafts. The first two shafts should be centrally located, while the ventilation shafts will be close to the limits of the property, along the strike direction.

As was demonstrated for a three-shaft layout (Fig. 17.4.3b), a similar principle is applied to locate four shafts in the case of three-dimensional deposits. Another variant is possible when ventilation shafts are in the footwall side close to the highest part of the deposit. When the mining area is elongated in the direction of the dip, shafts may be located in the configuration with three shafts in the central area (production, personnel/material, and ventilation) and the fourth (a ventilation shaft) in the upper part of the deposit.

17.4.1.2 Selection of Shaft Diameter

The lateral cross section of the shaft is found graphically according to

1. Lateral dimensions of the hoisting conveyances and other installations (such as a ladderway), maintaining adequate distances between each and the lining.

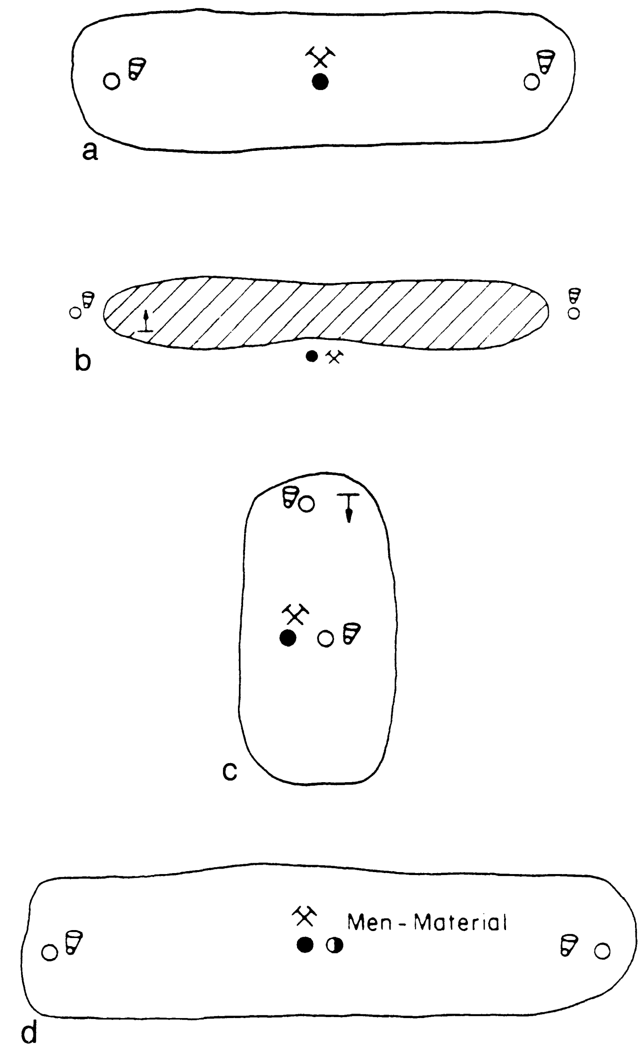


Fig. 17.4.4. Shaft locations for three- and four-shaft mines. Three-shaft layouts: (a) elongated deposit with major axis parallel to strike; (b) steeply dipping deposit with major axis parallel to strike; (c) elongated deposit with major axis normal to strike. Four-shaft layouts: (d) elongated deposit with major axis parallel to strike (after Jawien and Suchan, 1980).

2. Designed amount of airflow to fulfill ventilation requirements.

The required useful load Q in tons (tonnes) of the hoisting conveyance, generally a skip, for the planned daily production W in tons (tonnes)/day through the designed shaft is calculated for new production as follows:

$$Q = \frac{k\ W\ t}{3600\ T} \tag{17.4.6}$$

where k is irregularity factor = 1.15 for two or more vessels and = 1.25 for only one vessel, t is overall time of one cycle in seconds = $t_1 + t_2$ (t_1 is hoisting time and t_2 is brake time), and T is working hours of the hoist per day.

In SI units, the useful volume of the skip P in cubic meters is then

$$P = \frac{Q}{\gamma} \tag{17.4.7}$$

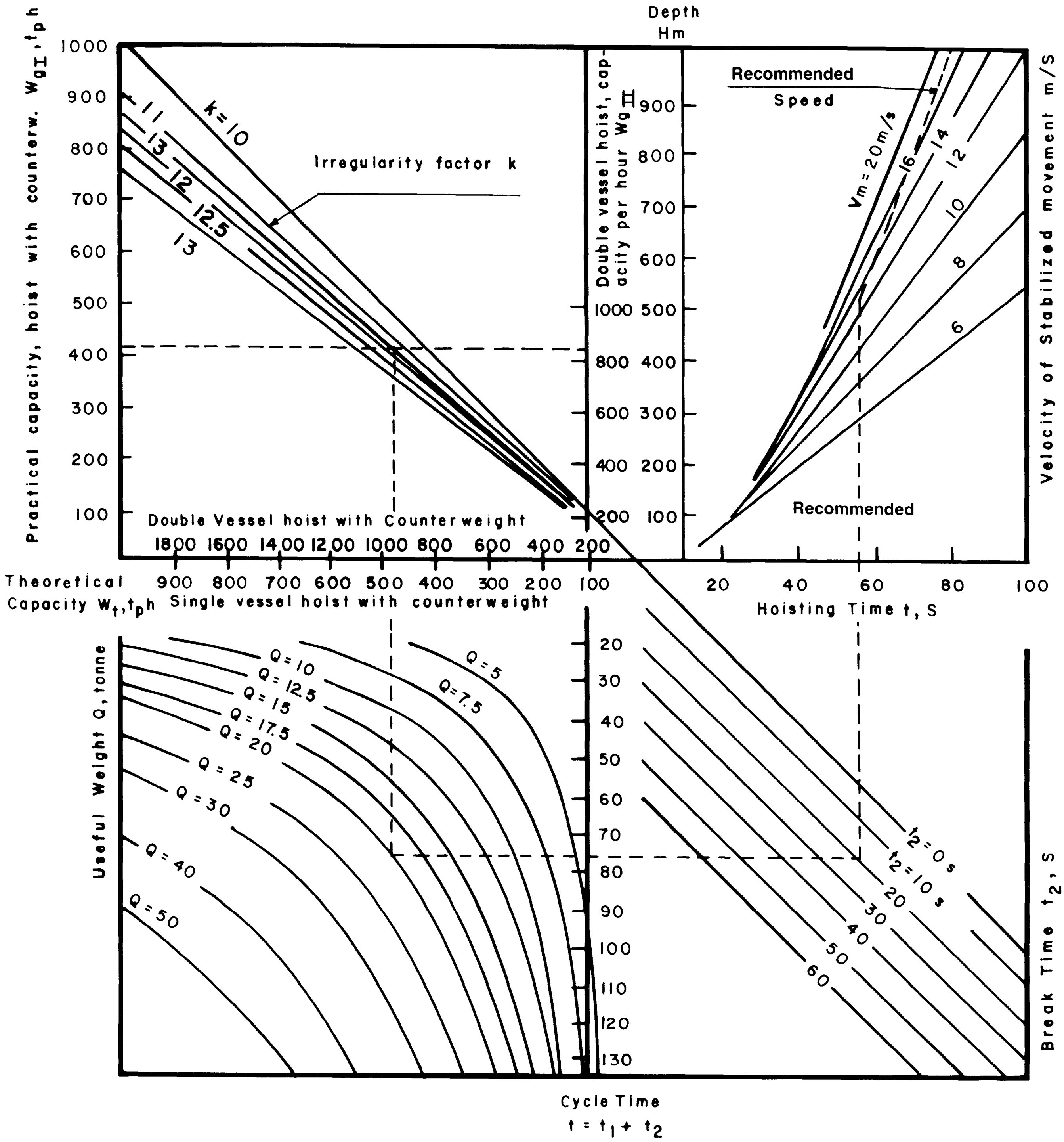


Fig. 17.4.5. Diagram for determination of single- and double-conveyance hoist capacity. Conversion factors: 1 ton = 0.9072 t, 1 fpm = 0.005 m/s.

where γ is bulk density of the hoisted material in t/m^3. For coal, γ = 0.8 to 0.85 and for stone, γ = 1.4 to 1.5.

Based on calculations and taking into consideration design principles and constraints, a graph for fast determination of the main hoisting parameters has been constructed (Fig. 17.4.5).

17.4.1.3 Shaft Linings

The lining of the shaft serves two purposes: support of shaft equipment and support of the walls of the excavation. Depending on ground conditions, wall support may require an additional

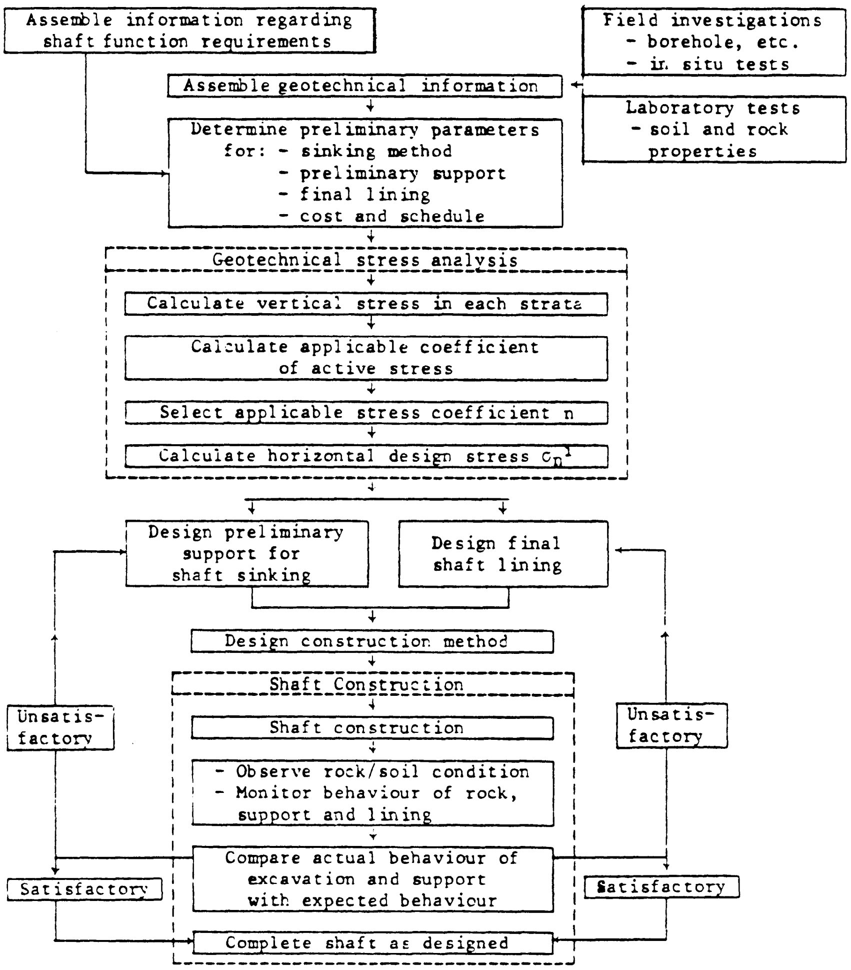


Fig. 17.4.6. Flow diagram for shaft ground support and lining design. (After Roesner et al., 1984.)

thickness of lining over the minimum 8 in. (200 mm) needed for fastening shaft equipment. In short life-span shafts sunk in competent rock, a rectangular shape with timber support is still commonly used. In modern large projects, however, round (or elliptical) shafts with concrete lining are the design used almost exclusively. Such a shape reduces airflow resistance, makes the sinking process easier, and permits taking full advantage of the structural features of concrete. Where possible, concrete lining with its numerous advantages is used over formerly popular materials such as brick and concrete blocks. Its placement is mechanized, resulting in high sinking rates as well as lower cost. Concrete strength can be adjusted according to need (2800 to 7100 psi, or 20 to 50 MPa), and watertightness of the lining can be achieved within aquifers with moderate head. In difficult ground conditions associated with high water pressure, cast iron tubbings with concrete mantle and welded steel lining are applied, usually in conjunction with one of the specialized methods of shaft sinking (see 17.4.3.2.). If mining within the shaft pillar is planned (deep multilevel coal mining), then a special elastic shaft lining is designed, consisting of two layers separated most often by a bitumen envelope. Such construction prevents deformation of the inner structural lining caused by shifting of the subsiding rock strata.

CALCULATION OF GROUND PRESSURE ACTING ON THE LINING OF CIRCULAR SHAFTS. The design procedure for shaft ground support and permanent lining is given in Fig. 17.4.6. When designing a new shaft, experience gained from data obtained during sinking of a nearby shaft should be used, provided that geologic conditions remain similar. Otherwise, only the results of geotechnical investigations obtained from the pilot hole are usually available. For complete treatment of geomechanics, see Section 10.

For temporary support, the safety factor need not be higher than one. For permanent linings, a higher safety factor is required, taking into account work conditions of the lining and its life span.

PRIMARY STRESSES. The vertical stress σ_v is proportional to the weight of overburden and is calculated from the relation,

$$\sigma_v = \Sigma \gamma h \tag{17.4.8}$$

where γ is density of the rock stratum, and h is thickness of the rock stratum.

Horizontal stresses may vary substantially from one location to another within the range of a minimum value, derived through elastic theory (Poisson's ratio), to a maximum value resulting from active or residual tectonic stress. From a practical standpoint, primary stresses may influence to some degree the level of the secondary stresses acting on shaft support. The pressure is caused by the overstressed zone of rock in a post-failure state developed around the shaft, resulting from the primary stress concentration. In most of the mining fields where primary stresses were measured, the horizontal to vertical stress factor is two to three, at a depth of less than 1500 ft (500 m).

STRESS DISTRIBUTION AROUND THE SHAFT. The stress distribution around the shaft differs in practice from the idealized solution derived from the elastic theory. Factors that influence stress are existing fracture pattern in rock, its anisotropy, shaft diameter, method of sinking (drill-blast or boring), time of exposure without support, type of temporary support used, and delay in installation of permanent lining. Additionally, the variety of strata penetrated by the shaft and their individual strength characteristics further complicate the stress analysis. For engineering purposes, however, a simplified approach is adequate where extreme pressure conditions, critical for shaft lining design, are assumed.

CALCULATION OF HORIZONTAL PRESSURE. Horizontal pressure appears when the strength of rock is reached by stress building up around the excavation and the limit of elastic deformation of the rock is exceeded. Then the fractured zone around the shaft spreads outward until the new equilibrium is achieved. Outside this zone, rocks are in an elastic state of deformation and hence self-supporting. The fractured (clastic) zone around the excavation tends to deform towards the center of the shaft and has to be controlled by an adequate support. In weak unconsolidated strata of a young geologic formation (usually at the upper part of shaft), ground pressure can be substantial, especially when combined with the head of the aquifers. In consolidated rock material, however, remaining in the elastic state of stress and deformation, no pressure is expected above the critical depth at which the stress reaches the strength of rock at the perimeter of the excavation. Some strata intersected by the shaft may be strong, thick, and therefore rigid. Such strata carry the load of the above weaker rock layers; thus there is no transfer of stress of the clastic zone from above the strong stratum to the weak rocks below this stratum. Such strata are also called cantilever strata. Their presence in the geologic column should be identified, and if they are below the critical depth, their influence on pressure distribution is significant. To act as a cantilever, the rock stratum has to satisfy the requirement (in SI units),

$$E\,I >= 206\,\sigma_t\,h^3 \qquad (17.4.9)$$

where E is elastic modulus in MPa, I is moment of inertia of section area of the rock stratum in m^4, σ_t is tensile strength of the rock in MPa, and h is thickness of the rock stratum in m. Above the first cantilever (if such exists), ground pressure is calculated in cohesiveless soils with $\phi = 5°$ and $c < 10$ kPa:

$$\sigma_h = \sigma_v\,Ka \qquad (17.4.10)$$

where σ_v is vertical stress in MPa, Ka is coefficient of active stress (ratio of horizontal to vertical stress), and $\sigma_h/\sigma_v = (1 - \sin\phi)/(1 + \sin\phi) = \tan^2(45° - \phi/2)$. In cohesive soils ($\phi <$ 25° and $c > 10$ kPa), the horizontal stress can be calculated from the equation,

$$\sigma_h = \sigma_v\,Ka - 2c\sqrt{Ka} \qquad (17.4.11)$$

where c is cohesion.

WATERTIGHT SHAFT LINING. The technical limit of water inflow for the conventional method of sinking is approximately 132 gpm (0.5 m^3/min). However, even a smaller inflow creates considerable problems and requires special arrangements to control it. Water seeping into the finished shaft causes corrosion of steel installations, requiring costly and troublesome replacement. Therefore, it is essential to minimize water inflow into the shaft unless it is controlled through a specially arranged drainage system, preventing a buildup of hydrostatic pressure behind the lining.

Over the years, different systems for control of water inflow have been tried. The most effective have proven to be those that take advantage of the energy of water itself to seal the lining. Quite often, water around the shaft is chemically aggressive to the concrete, so the lining can deteriorate when not insulated. Presently, the available system that is most effective, cheapest, and easiest to install is based on polyethylene (PE) or other insulation membrane sheets welded in the shaft. Double PE, 0.02-in.- (0.5-mm-) thick sheets have proven easy to handle and very successful for water insulation of concrete lining. A PE diaphragm is placed between the shaft wall and the concrete lining. In this case, the concrete does not need to be watertight, but it satisfies design strength requirements. Conversely, the PE insulation does not have any mechanical support function, but it provides 100% watertightness. Besides the desirable effect of a dry shaft resulting from this, deterioration of the lining associated with water seepage through concrete is also prevented.

Roesner et al., 1984, recommend the following method of determining the coefficient of the active stress for the calculation of horizontal stresses acting on the shaft excavation in cohesionless soils, cohesive soils, and rocks.

The single factor used for calculation of the coefficient of active stress Ka is either calculated from the readily obtainable friction angle ϕ_o (for soils) or the strength factor f (for rocks) (Table 17.4.1.).

The relationship between apparent friction angle and strength factor is (in SI units) ϕ_o = arctan f (for rocks) and f = tan ϕ (for soils), where ϕ_o is apparent friction angle (rocks) in degrees, ϕ is friction angle (in soils) in degrees, f is strength factor $= \sigma_c/10$, and σ_c is uniaxial compressive strength in MPa.

Very often the rock mass property is described either by the rock mass rating RMR or by the quality index Q (see Chapter 10.5). If either RMR or Q is known, the coefficient of active stress Ka can be read directly from the graph, Fig. 17.4.7.

Once the coefficient of the active stress Ka is established by either method, then the apparent coefficient of active stress Ka' is calculated by multiplying Ka by the appropriate stress coefficient n (Table 17.4.2.).

The stress coefficient n takes into account the influence of original tectonic forces n_o, the influence of the shaft diameter n_1, the influence of the shaft station n_2, the influence of the dip of the strata n_3, and finally the influence of the hydrostatic pressure on the lining (seepage coefficient) n_5.

By using the adjusted coefficient of active stress Ka' in the formulas for the horizontal stress, the horizontal design stress may be calculated from relations $\sigma'_h = \sigma_v Ka'$ for cohesionless soils, $\sigma'_h = \sigma_v Ka' - 2c\sqrt{Ka'}$ for cohesive soils, and $\sigma_h' = \sigma_v Ka'$ for rocks.

The detailed procedure is shown in Table 17.4.3.

From the same graph (Fig. 17.4.7), the indicated safe unsupported height between the permanent lining and the shaft bottom can be established and the ground support method can be selected (Table 17.4.4.).

The range of the maximum unsupported height varies from 45 to 75 ft (15 to 25 m) and is governed not only by the rock quality but also by mining regulations.

If preliminary ground support is installed, the relationship between Ka' and the unsupported height is no longer valid.

Depending upon the type and rigidity of the preliminary support, the stability conditions of the shaft excavation wall may improve; therefore, the unsupported height of the shaft can be increased accordingly.

CALCULATION OF THE LINING THICKNESS. The compressive strength and elastic moduli of concrete are given in Table 17.4.5. The coefficient of lateral deformation $\upsilon = 1/6$, and the

Table 17.4.1. Determination of Apparent Coefficient of Active Pressure

Strength grade	Descriptive of rock/soil	Compressive strength σ_c (MPa)	Strength factor f	Apparent friction angle $\phi_o = \arctan f$	Apparent coefficient of active pressure $Ka = \frac{1 - \sin \phi_o}{1 + \sin \phi_o}$
Extremely high	Extremely hard, solid, dense: quartzite, basalt, granite and other exceptionally high-strength rocks.	200–500	20–50	87–89	0.0006–0.0001
Very high	Granite, quartzporphyry, gabbro, granodiorite, diabase, gneiss	150–200	15–20	86–87	0.001–0.0006
High	Medium-grained granite, quartz.	120–150	12–15	85–86	0.002–0.001
	Small-grained sandstone, strong limestone, solid conglomerates with limestone matrix.	110–120	11–12	84–85	0.002
	Strong dolomites and limestones.	100–110	10–11	84	0.002
	Weathered granite, basalt, diabase.	70–100	7–10	81–84	0.005–0.002
Moderately strong	Sandstone, limestone.	60	6	80	0.007
	Weathered granite, limestone, sandstone, sandstone-shale.	50	5	78	0.01
Medium strong	Shale, sandstone, limestone,	40	4	76	0.015
	loose conglomerates, shales, slates.	30	3	71	0.025
Moderately loose	Loose shale, limestone, gypsum, frozen ground, blocky sandstone, cemented gravel, rocky ground.	—	1.7–2.7	60–70	0.07–0.03
	Gravelly ground, blocky and fissured shales, hard clay, coal, salt, marl.	—	1.2–1.7	50–60	0.1–0.07
Loose	Dense clay, clayey soil, soft coal, lignite.	—	1.0	45	0.1
	Loose loam, loess, gravel.	—	0.8	40	0.2
Soils	Soil with vegetation, peat, soft loam, wet sand.	—	0.6	30	0.3
	Soft, fine gravel, backfill.	—	0.5	25	0.5
Plastic soils	Silty soil and other soils in liquid condition.	—	0.08–0.5	5–25	0.8–0.5

Source: Modified after Roesner et al., 1984.

coefficients of work conditions for the lining are given in Table 17.4.2.

The thickness of the shaft lining d_i in m may be found from the relation (in SI units)

$$d_i = a\left(\sqrt{\frac{R_c}{R_c - np\sqrt{3}}} - 1\right) \qquad (17.4.12)$$

where a is shaft radius in light of lining in m, R_c is allowable stress of the concrete in MPa (see Table 17.4.5.) n is coefficient of lining work conditions (see Table 17.4.2.), and p is calculated outside pressure acting on the lining in MPa.

Based on Eq. 17.4.12, a diagram for determination of concrete lining thickness has been constructed (Fig. 17.4.8).

17.4.1.4 Shaft Collars

The shaft collar is the upper part of the shaft extending to the first footing and, of necessity, must be anchored in competent rock. The dimensions of a collar, such as its depth, cross section, and thickness, depend on the shaft function, character of overburden rocks, hydrologic conditions and resulting water and ground pressures, sinking method, and additional loading conditions when applicable (in a vertical direction from the headframe and horizontally as a reaction from the foundations of nearby structures).

The shaft collar of the production/intake shaft must be planned with an outlet from the ladder compartment and a connection for a heated air intake in the winter or cooled air in the summer (when required by climatic and mine conditions). Additional outlets as needed accommodate water pipes, compressed air lines, and electrical and communication cables.

The collar of a ventilation shaft always has a connection to the ventilation drift conducting exhaust or intake mine air to the main fan if located on the surface. Additional outlets may be provided to meet special needs (e.g., outlets for water pipes, air lines, cables). In hoisting shafts, the collar must be designed to provide support for the headframe. When overburden is composed of strata with low-strength parameters, placement of the headframe to provide firm support from the collar is the most desirable solution. The base of the headframe can rest on the collar wall, provided allowance is calculated for the additional load (Fig. 17.4.9a), and the collar is constructed as a foundation. It can also be placed on steel beams fastened in the collar wall (Fig. 17.4.9b).

When a large hoist is installed at the top of the headframe, additional foundations outside the collar wall are needed. For smaller hoists, a round tower is designed, similar to an extension of the shaft and integrated with the reinforced concrete collar (Fig. 17.4.9c) at a certain depth below the ground. Since the required wall thickness of the shaft tower is 7 to 24 in. (0.2 to 0.6 m), the diameter of the collar must be adequately increased.

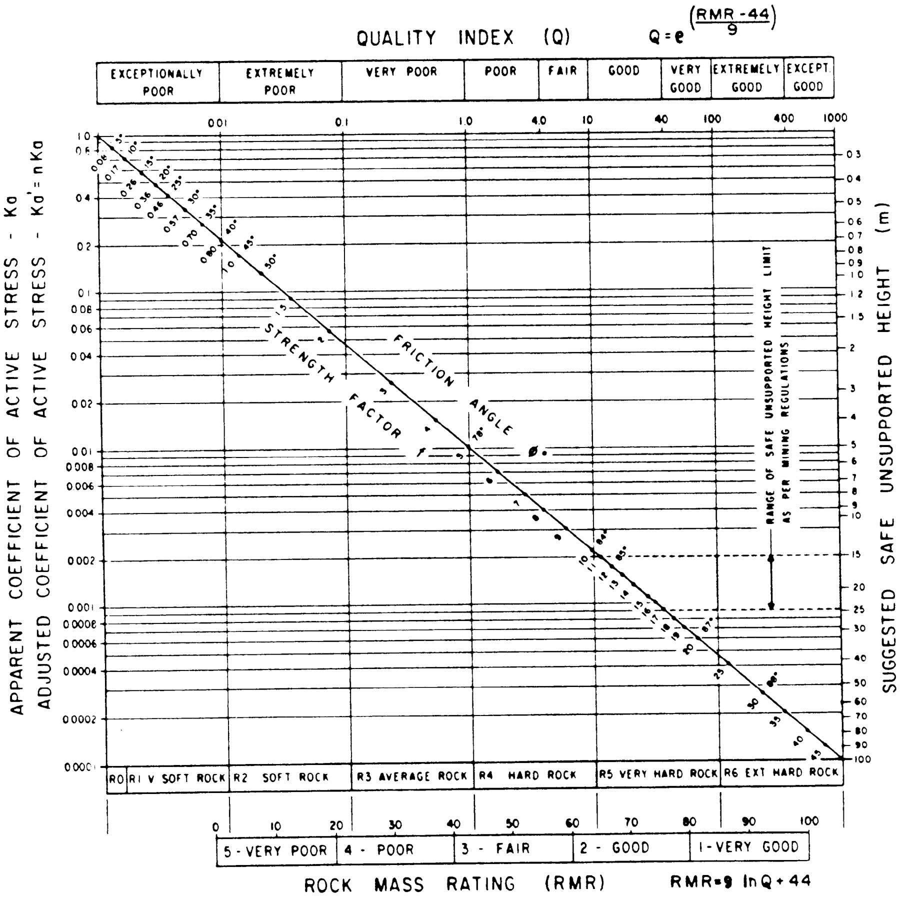


Fig. 17.4.7. Empirical strength factor criterion for ground support during shaft sinking. (After Roesner et al., 1984.) Conversion factors: 1 ft = 0.3048 m, 1 psi = 6.895 kPa.

The tendency now is to use multirope friction hoists installed on top of the shafts (Fig. 17.4.9d), so this design solution is often used in new projects.

The lining of the collar is made of construction material, similar to that for the rest of the shaft, but it is thicker because of additional loads and the weakening structural influence of multiple outlets and connections. Additionally, the collar is exposed to changes in temperatures and to dynamic loading in both vertical and horizontal directions, caused by movement of conveyances in the shaft.

Collar linings can be constructed of concrete blocks, concrete, precast reinforced concrete elements, or high-strength bricks. Reinforced concrete is chosen for large and eccentric loads existing, for example, as in steep strata. Collars usually have a step-down shape (Fig. 17.4.10), with a footing foundation at the bottom. The first step must be deeper than frost penetration.

Choice of a collar shape depends on ground and load conditions. The first lining step is usually 3 to 5 ft (1 to 1.5 m) and sometimes up to 7 ft (2 m) thick; the next step is usually 2 to 3 ft (0.6 to 1 m) thick or approximately two times the thickness of the actual shaft lining. Wall thickness of the third step, if such is planned, should be somewhere between the ground step and the regular shaft lining. The foot of the collar is placed in firm stratum, 7 to 10 ft (2 to 3 m) below the overburden. Its shape is often double-conical to better transfer the load to the surrounding rock.

The influence of foundations adjacent to the shaft supporting such structures as headframe coal silos, etc., on the shaft collar lining can be evaluated using a graphical system recommended by Cimbarievich (1951) (Fig. 17.4.11). When the distance is less than the minimum distance l_o, the additional load must be found from a stress distribution diagram, which may require that the collar dimensions be increased.

Calculation of Collar Lining Thickness. In practice, the collar thickness is chosen according to structural needs, following the recommendations described above. Actual stresses are then calculated and the safety factor is evaluated.

The shaft in Fig. 17.4.12a shows the load distribution over the shaft head lining, with horizontal and vertical stresses unevenly distributed due to eccentric loading of the headframe. The shaft head is assumed to be an independent, hollow cylindrical, thick-wall foundation, with a footing. A conservative assumption is made that there is no friction between the lining and the surrounding ground. The four headframe legs carry the vertical forces P_1, P_2, P_3, and P_4. The resultant force and its coordinates are then calculated. Several design solutions of shaft collars are shown in Figs. 17.4.13 through 17.4.15.

17.4.1.5 Shaft Insets and Sump Design

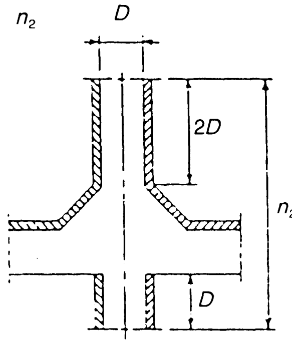
Insets of Hoisting and Ventilation Shafts. Insets of ventilation shafts without hoisting gear must be designed to minimize airflow resistance. The size of an inset for a hoisting shaft depends on the width and number of skips and cages being hoisted on this level, number of decks in the cages, and maximum length of supplies to be delivered. Additionally, an inset sectional area is checked for ventilation requirements. (The recommended

Table 17.4.2. Determination of Stress Coefficient *n*

Symbol	Type	Magnitude	Remarks
n_o	Influence of tectonic forces	In situ measurements	n_o
n_1	Influence of the shaft diameter D	$n_1 = \frac{1}{2}\sqrt[3]{D + 1}$	
n_2	Influence of the shaft station	$n_2 = 1.5$	n_2
n_3	Influence of the strata dip x	$n_3 = 1.0$ if $x < 30°$ $n_3 = 1.25$ if $x > 30°$	n_3
n_4	Influence of the weak strata of thickness h	$n_4 = 1.0$ if $h > 1.3$ m $n_4 = 0.7$ if $0.8 < h < 1.3$ $n_4 = 0.3$ if $h < 0.8$	n_4
n_5	Influence of the water filtration through the lining (seepage)	$n_5 = 1.0$ for watertight lining $n_5 = 0.1$ for formation grouting $n_5 = 0.1$ to 0.2 for water drainage $n_5 = 0$ for preliminary lining	n_5

Remarks

n_2



n_o—apply coefficient if $\sigma_h/\sigma_v > 1$

n_3—applies for cohesive soil and rock without water
n_4—applies for calculation of $H_{critical}$ in rocks
n_5—applies for calculation of water pressure

Source: Modified after Roesner et al., 1984.

air velocity is 800 fpm or 4 m/s for production shafts, and 1575 fpm or 8 m/s for ventilation exhaust shafts.) Within the inset, other openings must be provided as abutments (basements) for pushers and swinging platforms, drifts for simultaneous entrance and departure by personnel from multideck cages (if required), niches for control equipment, bypass around the shaft, etc.

Determination of the height of an inset (Fig. 17.4.16) is governed by the maximum length of material to be transported (e.g., rails, pipes).

17.4.1.6 Loading Chambers

Skip shafts and production levels are connected by a system of openings, the functions of which are interrelated and serve to accommodate the flow of material. Their shape and dimensions are dependent on skip size and loading system, as well as the types of mechanical and electrical service to the system.

Insets and sumps for shafts equipped for hoisting (Figs. 17.4.17 and 17.4.18) are more complicated than those of ventilation shafts. The sump in this case must house reverse sheaves for the bottom rope, end supports for stiff guides or weights for rope guides, and in the case of a skip hoist, must be equipped to catch and handle materials that fall.

The sump must also be of sufficient depth to assume adequate leeway for the hoisting conveyance below its lowest allowable position during operation. At the bottom, the sump must also provide space to store water flowing into the shaft and pump it into the central dewatering system. Fig. 17.4.18 shows an arrangement for a skip hoist loading level. Depending on the skip loading system and horizontal transportation arrangement, loading facilities may be required that incorporate the following:

1. Rail horizontal transport. (a) Tippler chamber or unloading ramp chamber (for Granby cars), batchers chamber, and skip chamber (so-called stiff system). (b) Tippler chamber or unloading ramp chamber, retaining bunker, loading devices chamber, batchers chamber, and skip chamber (so-called elastic system).

2. Belt horizontal transport. This system consists of unloading chamber, retaining bunker (vertical or horizontal), loading chamber, batchers chamber, skip chamber, retaining bunker, loading chamber, drift with belt scales, and skip chamber (also elastic system).

17.4.2 TECHNOLOGY OF SHAFT SINKING

17.4.2.1 Classification of Sinking Methods

Engineering properties of rocks and hydrogeologic conditions are the basis of the classification of sinking methods, with the main division on the conventional and special methods.

Table 17.4.3. Determination of Horizontal Design Stress

Soil/Rock Type	Friction angle ϕ	Cohesion intercept c(kPa)	Strength factor f	Coefficient of active stress $Ka = \sigma_h/\sigma_v$	Adjusted coefficient of active stress Ka'	Vertical stress σ_v (kPa)	Horizontal stress σ_h (kPa)	Horizontal design stress $\sigma_h{}^1$
Cohesionless	$\phi \geq 5°$	$c < 10°$	—	$\dfrac{1 - \sin\phi}{1 + \sin\phi}$	5) $Ka \cdot n$	2) $\sigma_v = \Sigma\gamma h$	$\sigma_h = \sigma_v \cdot Ka$	$\sigma_h{}^1 = \sigma_v \cdot Ka'$
Cohesive soil	$\phi < 25°$	$c \geq 10°$	—	$\dfrac{1 - \sin\phi}{1 + \sin\phi}$	5) $Ka \cdot n$	2) $\sigma_v = \Sigma\gamma h$	3) $\sigma_h = \sigma_v \cdot Ka$ $-2c\sqrt{Ka}$	3) $\sigma_h{}^1 = \sigma_v \cdot Ka^1$ $-2c\sqrt{Ka^1}$
Rock	$\phi > 25°$	$c \geq 10°$	1) $f = \sigma c/10$	$\dfrac{1 - \sin\sigma_o}{1 + \sin\sigma_o}$ $\phi_o = \arctan f$	5) $Ka \cdot n$	2) $\sigma_v = \Sigma\gamma h$	4) $\sigma_h = \sigma_v \cdot Ka$	4) $\sigma_h{}^1 = \sigma_v \cdot Ka'$

1) See Table 17.4.2, Stress coefficients n.
2) Below the water table, net hydrostatic pressure σ_w must be added to the submerged density of soil to obtain total σ_v
$\sigma_w = \gamma_w \cdot$ net water head $\cdot\ n_5$
3) In cohesive soils, the critical depth h at which pressure begins to act is calculated from equation
$h = 2c/\gamma\ Ka$
4) In elastic rocks, $\sigma_v = 0$ and $\sigma_h = 0$ to the depth $h < h_{critical}$

$$h_{critical} = n_4 \cdot \sigma_c/(2_{\gamma av}\ n_o n_1 n_2 n_3)$$
$$\gamma_{av} = (\gamma_1 h_1 + \gamma_2 h_2 + \cdots \gamma_i h_i/(h_1 + h_2 + \cdots h_i)$$

5) See Table 17.4.4, Determination of stress coefficient n
Conversion factor: 1 psi = 6.895 kPa.
Source: Modified after Roesner et al., 1984.

Table 17.4.4. Suggested Ground Support for Shaft Sinking

		Preliminary Support									
Quality Index Q	Strength Factor f	Shield caisson	Tubbings: cast iron, concrete	Concrete: cast-in-place, precast	Liner plates: bolted, welded	Ribs and Lagging, Forepoling	Shotcrete	Rock bolts grouted	Rock bolts: mechanical	Rock bolts, wire mesh straps	Ground stabilization (grouting, freezing)
Exceptionally poor 0–0.1	$f < 0.8$	X	X	X	X	X					X
Extremely poor 0.01–0.1	$f > 0.8$	X	X	X	X	X					X
Very poor 0.1–1.0	$f > 2$				X	X	X	X	X	X	
Poor 1–4	$f > 5$						X	X	X	X	
Fair 4–10	$f > 8$								X	X	
Good 10–40	$f > 10$								X	X	
Very good 40–100	$f > 18$								X		
Extremely good 100–400	$f > 25$										
Exceptionally good 400–1000	$f > 35$										

Source: Modified after Roesner et al., 1984.

Table 17.4.5. Ingredient Proportions and Requirements for Various Classes of Concrete

Saturated Surface Dry Aggregate, lb/yd.³ (lb/bag)				Approximate Percent Fine To Total Aggregate		Maximum Free Water Gal. Per Bag of Cement (lb/bag)	28-day Compressive Strength-psi (1) (MPa)	Slump-In.(4)	Minimum Cement Factor Bag/yd.³ (lb/yd.³)
Sand	Gravel	Sand	Stone	Gravel	Stone				
1146 (191)	2052 (342)	1272 (212)	1926 (321)	36	40	6.00 (50.0)	3,500 (26.4)	2–4	6.00 (564)
1071 (153)	1918 (274)	1190 (170)	1799 (257)	36	40	6.00 (50.0)	3,500 (26.4)	6–7	7.00 (658)
1155 (175)	2066 (313)	1280 (194)	1934 (293)	36	40	5.00 (41.7)	4,000 (28.1)	2–3	6.6 (620)
1310 (273)	1962 (297)	1440 (300)	1648 (385)	40	44	7.50 (62.6)	2,500 (17.5)	3–5	4.80 (451)
1054 (155)	1972 (290)	1176 (173)	1650 (272)	35	39	6.00 (50.0)	4,000(3) (28.1)	1–5	6.8 (639)
998 (125)	1864 (239)	1108 (142)	1747 (229)	35	39	5.75 (48.0)	5,000 (35.1)	1–5	7.8 (733)

Conversion factor: 1 lb = 0.4536 kg.

NOTE: R_c, allowable stress, is obtained by application of safety factor-3

NOTE: The above dry weights per cubic yard of concrete are based upon a specific gravity of 2.64 for sand and 2.66 for stone or gravel. The Engineer will adjust aggregate proportions to compensate for volume due to entrained air as necessary to maintain the specified cement factor. The air content, by volume, shall be 5.5% plus or minus 1.5%.

(1) All class AA concrete shall contain a water-reducing admixture meeting requirements when the ambient air temperature at the time of concrete placement is 70°F or less. When the ambient air temperature at the time of placement of class AA concrete is 71°F or more, the concrete shall contain a water-reducing and retarding admixture. At the direction of the engineer, water-reducing and retarding admixtures may be required or allowed for ambient air temperatures of less than 71°F. The type of admixture used shall not be changed for concrete placed during any individual contiguous pour, unless permitted.

(2) For prestressed members, 5000 psi expected at or prior to 28 days.

(3) Slumps less than the minimum listed herein will be permitted by the engineer provided satisfactory results are obtained.

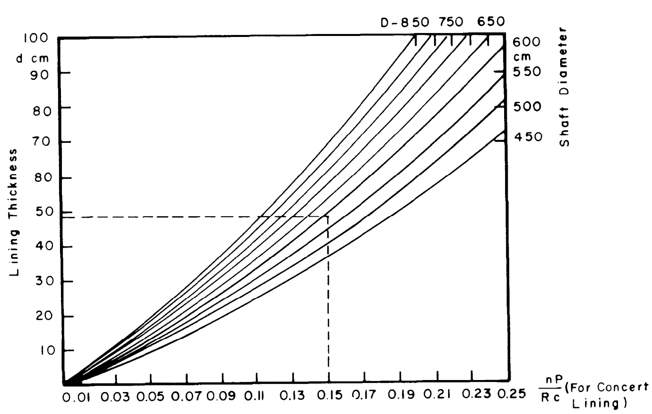


Fig. 17.4.8. Diagram for determination of lining thickness (concrete). Conversion factor: 1 in. = 2.54 cm.

Conventional methods consist of repeating cycles of face advancing and erecting of lining without any previous ground stabilization. Conventional methods are applicable in good- and fair-quality rocks with limited water inflow to the shaft not exceeding 130 gpm (0.5 m^3/min).

Special methods of shaft sinking are used in

1. Strong rocks but with an extensive fracture pattern, resulting in a substantial water inflow of more than 130 gpm (0.5 m^3/min).
2. Coherent, weak, plastic, and flowing ground.
3. Loose, water-bearing grounds (e.g., flowing sands, silts).

17.4.2.2 Conventional Drill and Blast Method

Taking into account the order of the main components of the sinking operation with the conventional method, the following classification can be employed:

1. Sinking with non-simultaneous face advancing and erecting of permanent lining, the so-called *series system.*
2. Sinking with simultaneous face advancing and permanent lining erection in certain distances behind the face, the so-called *parallel system.*
3. Advancing the face with simultaneous lining erection in the same shaft section, the *simultaneous system.*

Shaft beams, guides, etc., are usually installed following the advance of the permanent lining, although sometimes also after completion of the shaft lining (when a shaft boring system is employed).

In the series system, a shaft is sunk in sections, with length depending on the rock quality (between 100 and 170 ft, or 30 and 50 m). In a particular section, the face is advanced first with the temporary lining, and then when sinking is suspended, the permanent lining is constructed. This system does not assure a fast sinking rate, nor does it require a high capital expenditure, so it is used for sinking of shafts with smaller diameters and small and moderate depths.

The parallel system is characterized by a simultaneous advancing face and erection of permanent lining, but in various shaft sections. A temporary lining is required too. This system is applied for deep and large-diameter shafts, where the high capital (equipment) investment cost can be paid over the life of the project.

The simultaneous system can be further subdivided into (1) series-simultaneous and (2) parallel-simultaneous. In the *series-*

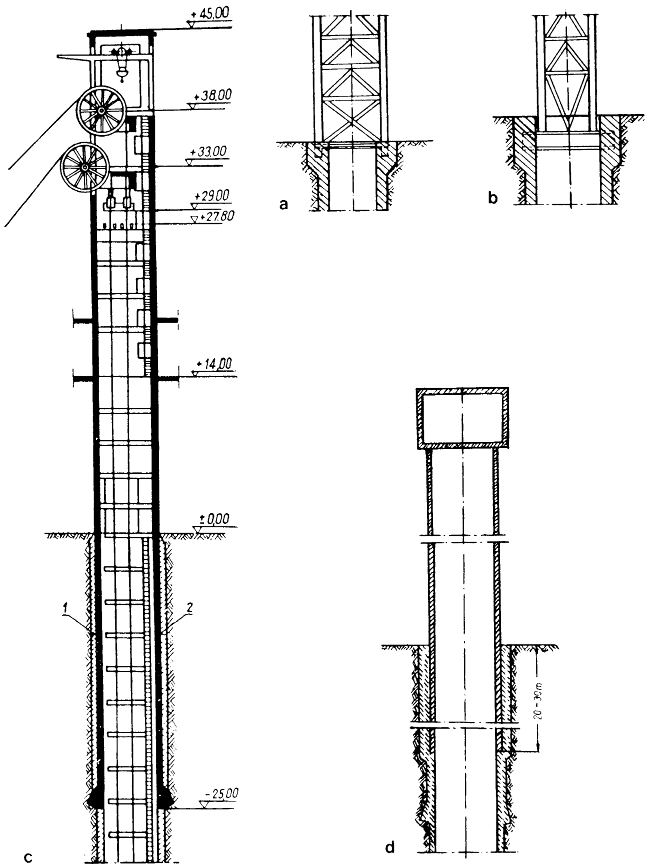


Fig. 17.4.9. Sections through headframe and headgear foundations: (a) foundation on shaft head; (b) foundation on beams located in shaft collar wall; (c) circular headgears founded on shaft head, comprising concrete shaft head and reinforced concrete headgear. Dimensions in meters. (After Unrug, 1985.) Conversion factor: 1 ft = 0.3048 m.

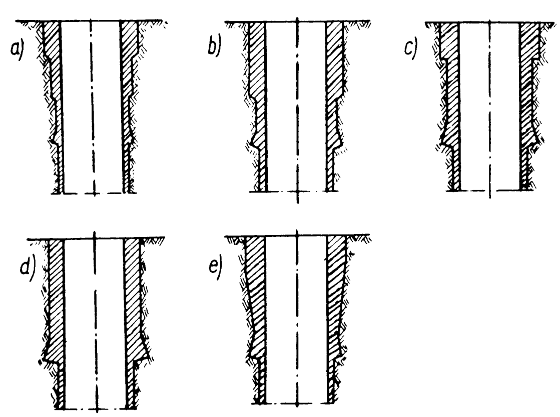


Fig. 17.4.10. Different shapes of shaft collars. (After Unrug, 1985.)

simultaneous system, face advance and lining are completed in one cycle, being done one after another. The lining is erected either upwards (e.g., brick lining in short segments), or downwards (e.g., underhanging tubbing lining or timber lining), eliminating temporary lining. The *parallel-simultaneous system* consists of simultaneous face advancing and permanent lining with segments 12 to 25 ft (4 to 8 m) long in competent rocks without temporary support, or with temporary support in weak, unstable rocks.

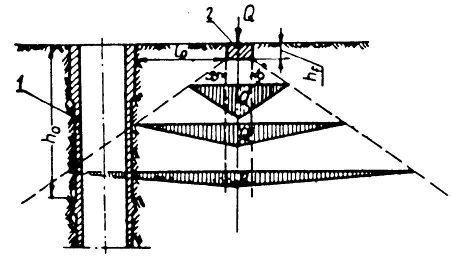


Fig. 17.4.11. Simple geometric method for determining additional load acting on the shaft collar lining from nearby foundations. A zone of influence is defined by lines drawn at 35° from the vertical at the base of the foundation. The effect of additional foundation loading may be considered negligible when the horizontal distance from the shaft to the foundation l_o is greater than $(h_o - h_f)$ tan 55°, where h_o is depth of the shaft collar, and h_f is depth of the foundation. (After Unrug, 1985.)

Another group of sinking methods is based on utilization of a central borehole for removal of muck (shaft sunk to the existing mine level).

Shafts can be sunk to the final depth or only to depths required in the first stage of mine development. Extension of the shaft to the next level should be synchronized with reserve depletion at the upper level, ensuring uninterrupted production. Shafts can be extended by sinking or raising.

PREPARATORY WORK. Shaft sinking is preceded by preparatory work as follows:

1. Determination of geologic and hydrogeologic conditions usually by drilling a hole or by examining an existing exploratory hole close by.
2. Preparation of technical documentation pertaining to surveying and permitting.
3. Construction work involving site preparation and construction of shaft head.

Site Preparation—The shaft site should be secure from flooding (above 100-year water level). Site preparation consists of

1. Construction of roads and material storage areas (open and covered).
2. Grading of the terrain (when required, this could also be done during sinking, using muck material as a fill) for further construction of mine facilities.
3. Water supply for drinking, industrial uses, and fire extinguishing. When a water line is not available, an adequate supply has to be built in compliance with existing regulations. When wells are drilled, they should be located outside the drainage cone of the shaft. Water pumped from the shaft should be utilized as well. The required daily supply is 20,000 to 26,000 gal (80 to 120 m^3) of plant water and 10,000 to 16,000 gal (40 to 60 m^3) of tap water. The fire tank should be 20,000 to 26,000 gal (80 to 100 m^3) capacity. An adequate ditch for storm water should be planned.
4. Electrical power supply can be provided in two stages: for preparatory work, from small (to 260 kVA) transformers, and during sinking when more power is needed, from larger units (e.g., 1200 kVA at 500 V). The transformer station should

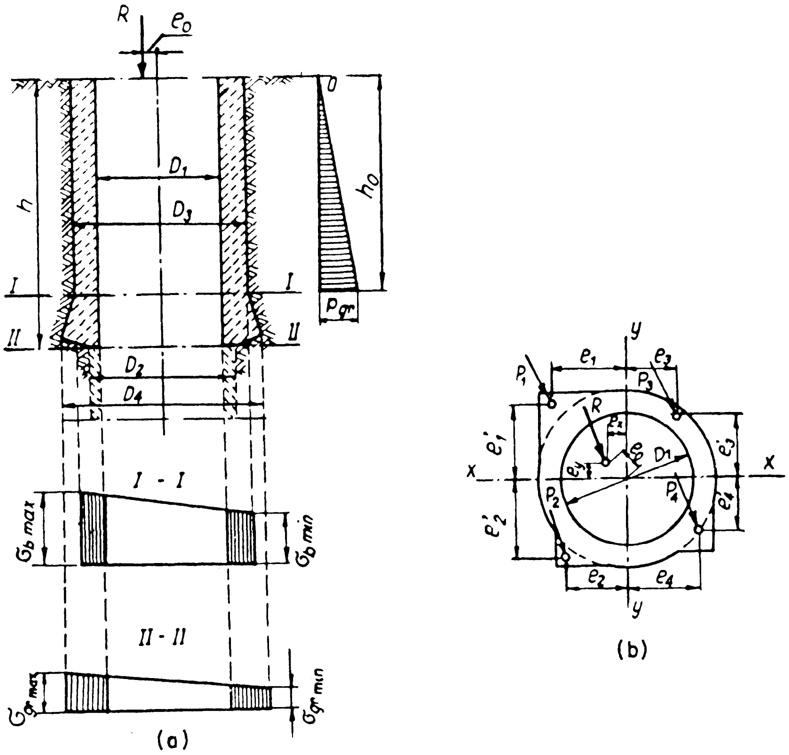


Fig. 17.4.12. Load distribution on shaft collar lining [*p*—forces, *e*—coordinates of forces, *R*—resultant force, *D*—shaft diameters]. (After Walewski, 1965.)

have two independent sources of power, and when that is impossible, a diesel generator is required for emergency use. Requirements include:

a. Provision of covered storage for construction materials and ramp for unloading heavy machinery.

b. Hook up to telephone lines.

c. Construction of temporary buildings for shaft sinking and foundations for head gear.

d. Installation of sinking equipment (hoists, etc.).

e. Surveying of elevation of shaft collar, with markers of main shaft axes and north direction (Fig. 17.4.19).

Temporary Buildings—Temporary buildings should be located in a way that will not interfere with permanent buildings to be constructed. For shaft sinking, the following is needed:

1. General use building, housing an office, bathhouse, first aid room, heating and cooling units, warehouse for small equipment. (When permanent mine facilities are available, such temporary facilities are not required.)

2. Lamp-house, complying with existing Mine Safety and Health Administration (MSHA) standards.

3. Mechanical and blacksmith shop for minor repair work and sharpening bits.

4. Sheds housing compressors (with foundation under particular machines) with outside tank for compressed air and cooling water tank.

5. Transformer station, housing transformers, high- and low-voltage switches, and an electrical shop. Sometimes high-voltage transformers are located outside; then the low-voltage switch-board is placed in the compressor building to avoid loss of cables.

6. Shed beside the headframe, approximately 32 to 64 ft (10 by 20 m), where concrete, mucking, and other equipment is placed.

7. Material shed, basically for cement storage and other construction materials (approximately 20 by 40 by 40 ft (6 by 12 by 12 m).

8. Winch building for the shaft stage.

9. Fan housing, approximately 13 by 13 ft (4 by 4 m).

10. Flammable supplies warehouse (fire resistant structure).

11. Hoist building, depending on machine size, approximately 16 by 33 ft to 40 by 40 ft (5 by 10 to 12 by 12 m).

12. Storage for 200 to 400 lb (100 to 200 kg) of explosives and 500 to 1000 detonators, when current supply is secured, and where not, a larger storage area is needed.

13. Living accommodations for sinking crew, depending on needs.

SHAFT COLLAR CONSTRUCTION. Shaft collar sinking and construction depend on geomechanical and hydrologic conditions. In the case of loose water-bearing rocks, one of the special sinking methods has to be used. In good ground conditions, an ordinary method is applicable to sink the collar to its whole depth, 26 to 40 ft (8 to 12 m).

In favorable ground conditions after the collar is constructed, further sinking of the shaft down to 160 ft (50 m) can proceed with the application of provisional equipment. This allows for installation of specialized sinking equipment and other required work, for example, grouting.

During collar construction, rock is excavated by means of jackhammers, mechanical excavators, or explosives. Muck is removed by suitable construction machines, and where beyond their reach, by bucket and mobile crane.

DRILLING AND BLASTING IN SHAFTS. Properly done, drilling and blasting work is supposed to ensure:

1. Correct size and shape of planned excavation.

2. Even surface of the face that makes easier mucking and drilling of the next round.

3. Uniform size of broken rock for effective mucking operation.

4. Safe and economical operation.

In shafts, a selection of high explosives is generally used so that in case of excessive methane liberation, adequate blasting

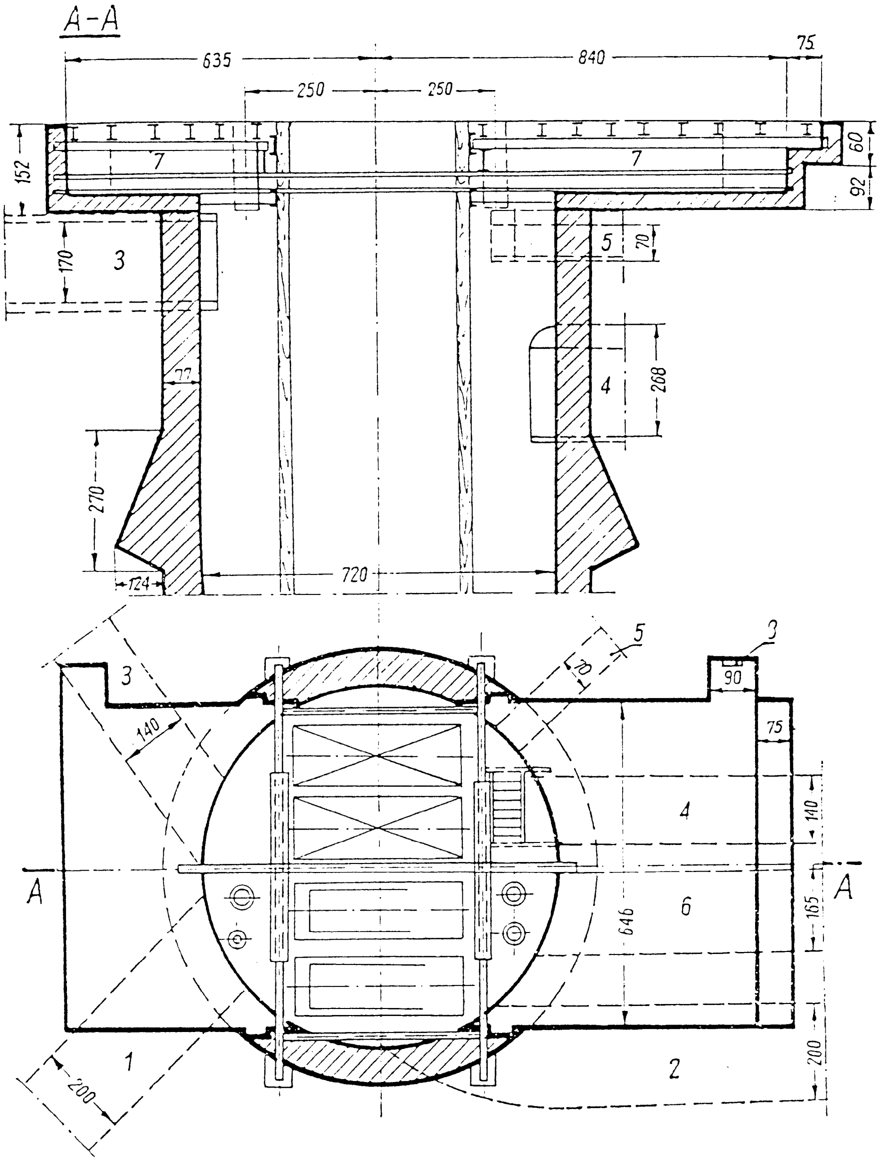


Fig. 17.4.13. Production shaft head 24 ft (7.2 m) in diameter.
1. heated air channel I
2. heated air channel II
3. channel for water pipes
4. outlet from ladder compartment
5. cable channel
6. channel for compressed air lines
7. basements for pushers

Dimensions in cm. Conversion factor: 1 in. = 2.54 cm.

material is available to ensure safety. Electric caps with millsecond delays are used.

The theory and practice of explosives and blasting is covered in Chapter 9.2.1.

Consumption of Explosives—The *unit consumption* (or *powder factor*) q is the amount of explosives in lb (kg) required to excavate 1 yd^3 (1 m^3) of solid.

It has been established that:

1. For practical purposes, it can be assumed that q is independent of the area of the shaft cross section S, provided that $S > 24$ yd^2 (20 m^2).

2. The unit consumption of explosives depends on the hole length; with a constant hole utilization factor (see Eq. 17.4.13), the minimum q is achieved in holes with length 8 to 10 ft (2.5 to 3.0 m) in shales and 7 to 8 ft (2.0 to 2.5 m) in sandstones (with other parameters constant) (Fig. 17.4.20).

3. With increase of hole diameter, q decreases.

4. Unit consumption decreases with the increase of hole utilization factor.

5. With q approaching its optimum value, the intensity of rock breaking increases.

6. q depends on the physical properties of the rock and is 20 to 30% less in shales than in hard sandstones.

Hole Utilization Factor—During shooting, rock material does not separate from the solid at the bottom of the hole. The *utilization factor* h is defined as follows:

$$\eta = \frac{l_o}{l} \tag{17.4.13}$$

where l_o is effective hole length utilized, and l is total hole length. The difference l-l_o represents hole length that was not utilized. An optimum factor h, which characterizes a proper selection of drill and blast parameters, assures a good sinking rate.

In the coal basin of Donieck, USSR, in carboniferous rocks where many shafts were sunk, research gave the following results (Walewski, 1968):

1. The ratio h increases (with other conditions unchanged) to a maximum, corresponding to an optimum unit consumption q, and then decreases.

2. The ratio grows with an increase of hole diameter (cartridge diameter).

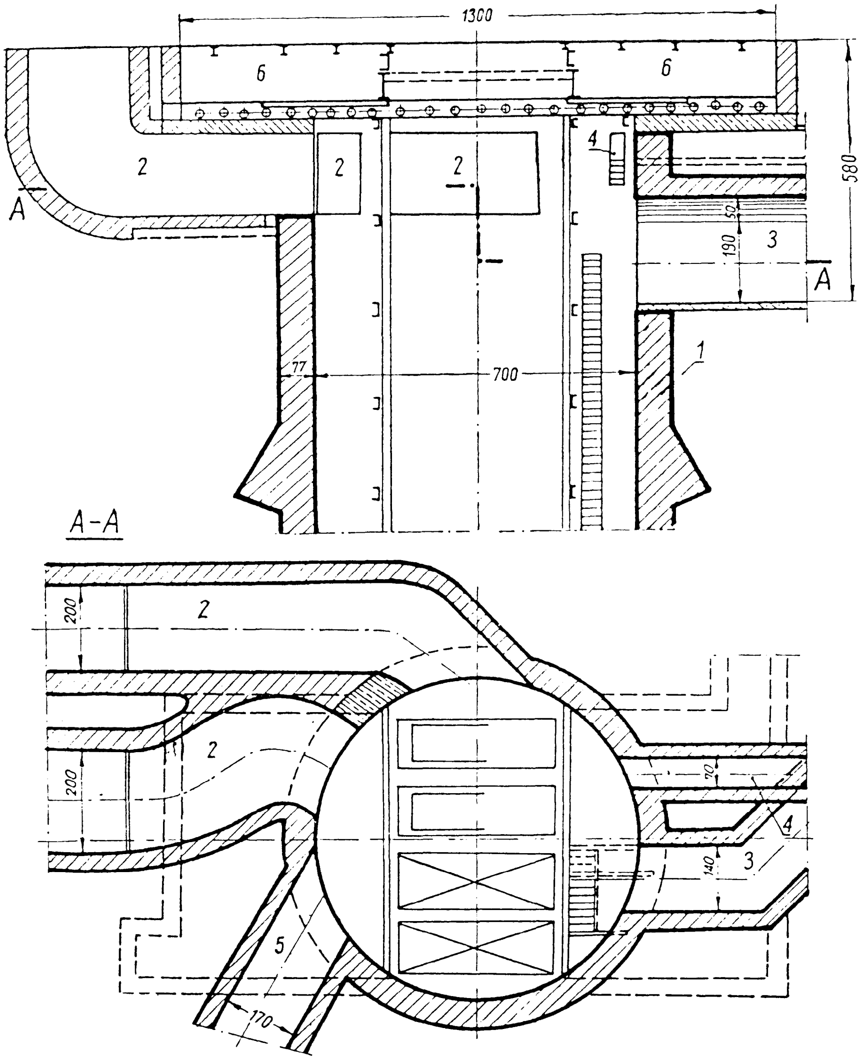


Fig. 17.4.14. Design of production shaft head 23 ft (7 m) in diameter.
1. lining (bricks)
2. heated air channels
3. outlet from ladder compartment
4. cable channel
5. channel for water pipes and compressed air
6. basement for fire closings

Dimensions in cm. Conversion factor: 1 in. = 2.54 cm.

3. A maximum value is achieved with an optimum length of holes. When the holes are too long, the shooting effect diminishes; in the upper part of the hole, crushing of the rock is not sufficient, while in the deeper part, the material is crushed excessively.

4. The factor increases with the growth of the hole loading factor until it reaches its optimum value (one has to remember that the hole loading factor decreases with a hole cartridge diameter increase).

Hole loading factor a is the ratio of the length of the hole filled with explosives to the entire length. In practice this relationship is as follows:

Hole diameter, in. (mm)	1.26 to 1.42 (32 to 36)	1.77 (45)
Factor *a*	0.6 to 0.7	0.45 to 0.5

Research on number of holes has shown that:

1. The number of holes does not depend on depth in the range of 10 to 12 ft (3 to 3.5 m).

2. The number of holes per unit area of shaft face does not depend on face area S when $S < 24$ yd^2 (20 m^2).

3. The hole number decreases 1.8 to 2.0 times when the hole diameter is 1.8 in. (45 mm) and 2.5 to 3.0 times when the diameter is 2.2 in. (55 mm). The use of shaft jumbos able to drill larger holes is therefore recommended (Fig. 17.4.21).

An empirical formula developed for hard rock gives the number of holes N as

$$N = 0.234\,A + 22 \qquad (17.4.14)$$

$$N = 2.55\,A_1 + 22 \qquad (17.4.14a)$$

where A is area of shaft face in ft^2 or A_1 is area of shaft face in m^2.

Hole Length—Hole length is different from hole depth; it is the shortest distance between the free face and the end of the hole. (Hole inclination creates this difference.) The length of holes has to be justified in three respects:

1. Length as a factor being determined by geologic and mining conditions (e.g., rock type, water inflow, diameter).

2. Length being determined according to the applied drilling and shooting technology (e.g., drilling equipment, explosive material).

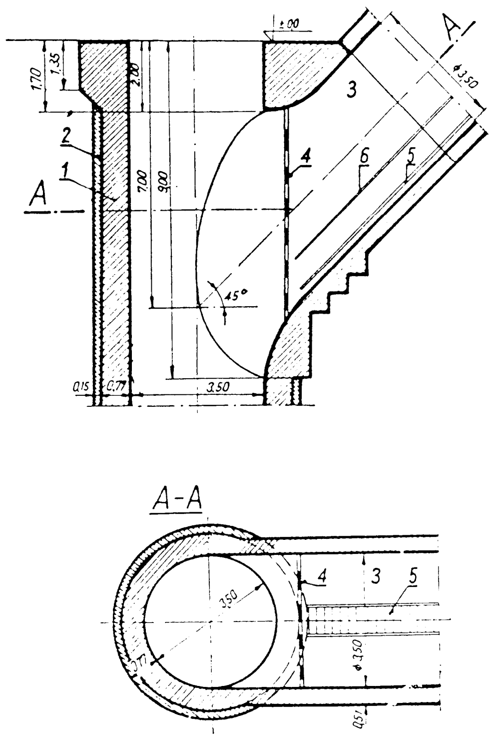


Fig. 17.4.15. Design of ventilation shaft head 12 ft (3.5 m) diameter.

1. hanging brick lining
2. insulation concrete outer ring
3. ventilation channel
4. safety grate
5. ladder
6. railing

Dimensions in cm. Conversion factor: 1 in. = 2.54 cm.

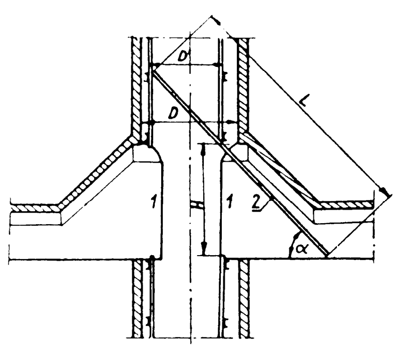


Fig. 17.4.16. Inset height calculation. *L* is maximum length of transported material, α is angle from level, *D* is shaft diameter. For α = 45°, *D′* = 0.7 *D,* which is the free space between shaft beams. *H* = 0.7 (*L* – *D*).

3. Length of holes determined according to the organization of shaft sinking, which consists of a sequence as follows: drilling + mucking/hoisting + lining. Shorter holes are chosen when one cycle per shift is planned rather than one per day. The most desirable will be the length that assures that a unit advance with minimum time and cost is achieved.

An optimum hole length should be found from the minimum cost principle. The main cost contributors are drilling of holes, explosives, and mucking, depending on the depth of round.

Fig. 17.4.22 shows the timing of particular operations vs. hole length.

An important consideration when choosing hole length is any deviation from the planned shaft section; for example, for holes 6.5 to 8 ft (2 to 2.5 m) long, this deviation is 2 to 3.5%, and for holes 13 to 16 ft (4 to 5 m) long, the deviation reaches 5 to 10%. Most records in shaft sinking have been achieved with hole lengths of approximately 10 ft (3 m). For carboniferous rocks such as shales, an optimum length is about 10 ft (3 m); for sandstones, about 8 ft (2.5 m). The sinking cycle time, dependent on hole length, must be considered when planning the length of holes.

It should be emphasized that the above calculation, essential during the designing stage, has to be verified by actual sinking conditions and can be changed as required.

Drilling Round—The drilling pattern has a strong influence on sinking performance. It should ensure:

1. Accuracy in the shape of the excavation.
2. Clean separation of rock from the bottom of the excavation (shaft face).
3. Even and sufficient comminution of rock.
4. Avoiding cutoffs of neighboring active holes.
5. Avoiding damage to nearby shaft installations.

The drilling pattern depends on such factors as the shape of the shaft section, rock strength, cleavage, dip of strata, water inflow, and hole loading structure.

In circular shafts with small and moderate strata dip, holes are placed on concentric circles, drawn from the center of the shaft. The number of circles is three to five, depending on the shaft diameter (Fig. 17.4.23).

The central hole is vertical and shorter, about two-thirds the length of the other holes. This hole initiates conical excavation by cut holes and is especially recommended in strong rocks.

Holes of the first (central) circle, so-called wedge holes inclined towards the shaft center, are shot first to open an additional free surface and to make easier and more efficient the work of the remaining external holes. They are drilled with an opposite (outward) inclination of 65 to 75°. Four to ten holes are drilled for the first circle. The ends of four holes are intended to create a square with sides of 7 to 12 in. (180 to 300 mm). For a larger number of holes, a circle 1.3 to 2 ft (400 to 600 mm) in diameter is used. In general, the harder the rock, the more first-cut holes are required.

Holes of the next two, three, and sometimes four circles crush and lift the excavated material.

The diameter of circles when the cartridge is 1.25 in. (32 mm) is dependent on the shaft diameter *D:*

1. With three circles, use 0.37, 0.66, and 0.93 *D.*
2. With four circles, use 0.35, 0.54, 0.7, and 0.93 *D.*
3. With five circles, use 0.27, 0.43, 0.6, 0.73, and 0.93 *D.*

The ratio of the hole number in particular circles is

1. For three circles: 1—2—3
2. For four circles: 1—2—3—4
3. For five circles: 1—2—3—4—5

When cartridges 1.78 in. (45 mm) in diameter are used, holes are placed usually in three or four circles with these diameters:

1. For three circles, use 0.3, 0.6, and 0.95 *D*
2. For four circles, use 0.25, 0.48, 0.72, and 0.96 *D*

The ratio of hole number in particular circles is then for three circles, 1—3—6, and for four circles, 1—2—3—5.

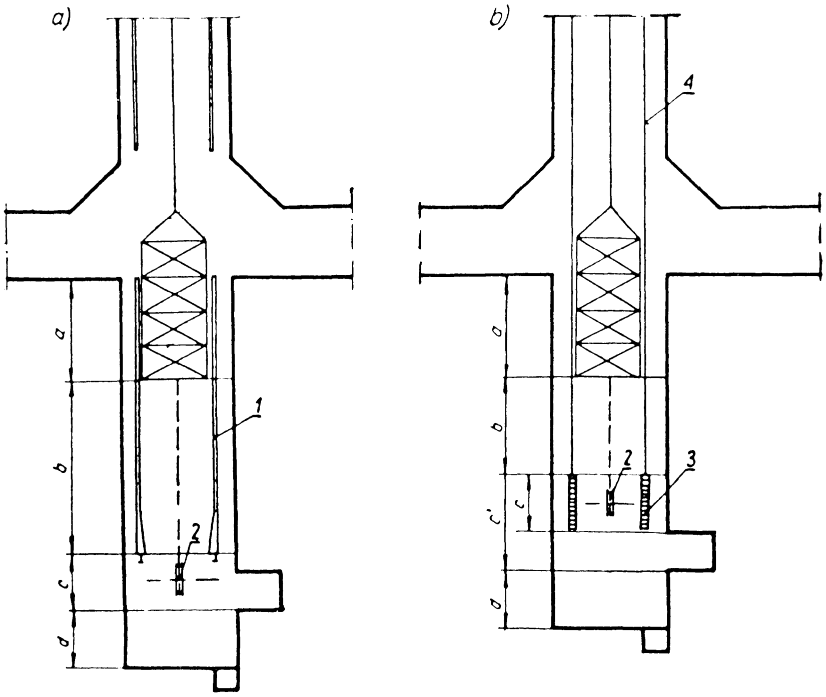


Fig. 17.4.17. Determination of shaft bottom depth (a) in personnel shaft using stiff guides (1), and (b) for material hoisting using rope guides (4). Space must be allowed for return sheave (2) and rope guide weights (3). Then depth (in m) = *a+b+c+d,* where *a* = (*n* – 1) *w, b* = (1–1.5) *w, c* = 2–10 m, and *d* = 3m, *n* is the number of cage floors, *w* is the distance between floors, *v* is the maximum hoisting speed in m/s, *b* is the freeway (max. 35 ft or 10 m), and *c* is the sheave clearance in m. (After Walewski, 1965.) Conversion factor: 1 ft = 0.3048 m.

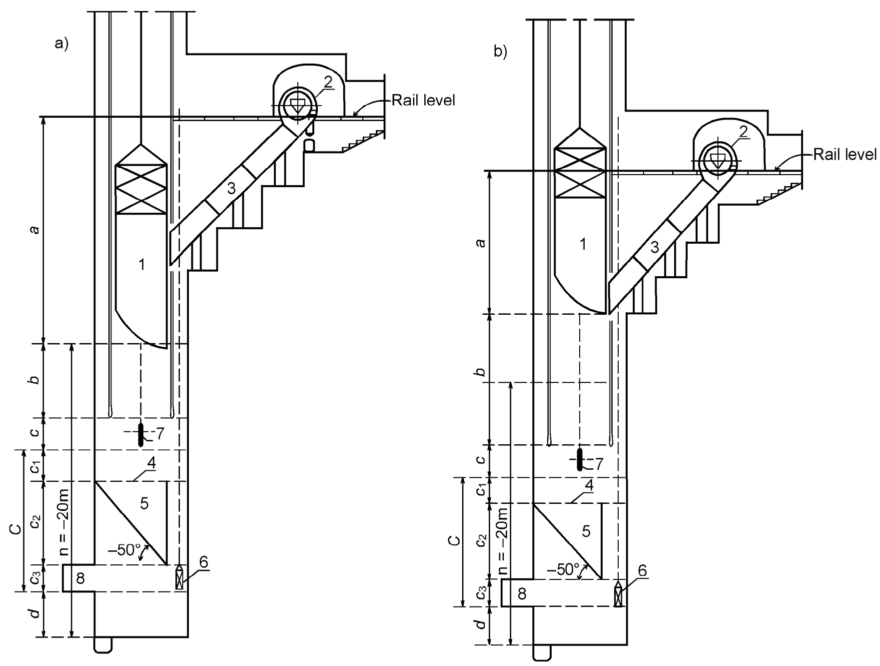


Fig. 17.4.18. Determination of shaft bottom depth in a skip shaft (a) for material hoisting and (b) for personnel hoisting. In this case, depth = *a+b+c+d,* as in Fig. 17.4.17, but a includes additional clearance for the skip (1), tippler (2), and batchers (3); and *c* includes clearance for the protecting grill (4), spillage bunker (5), auxiliary hoisting bucket (6), and return sheave (7); *d* is the sump pump chamber (8). (After Walewski, 1965.)

The inclination of holes varies from 65 to 85°, and the further they are from the center of the shaft, the more vertical they are.

The distance between adjacent holes on a circle should not exceed 2.6 to 3 ft (0.8 to 0.9 m) for 1.26-in. (32-mm) and 1.4-in. (36-mm) cartridges; for a 1.77-in. (45 mm) cartridge, it should be in the range of 3.28 to 4.92 ft (1 to 1.2 m). The last circles of so-called rib holes are supposed to leave the shaft wall smooth. They consist of holes drilled toward the shaft wall, 65 to 85° (less inclination for weak rocks and more for stronger ones). In very weak rocks, they can be slightly inclined toward the center of the shaft as are all the others.

Rib holes are placed in a circle with a diameter of 2 ft (0.6 m) less than the shaft diameter in rocks with a coherence factor of f = 2 to 8 and 1 ft (0.3 m) for rocks with f = 9 to 20. The distance from the shaft wall has to be such that in rocks with $f < 8$, holes should not project from the design shaft section, and in rocks with $f > 8$, they should protrude about 4 to 8 in. (100 to 200 mm) outside the shaft contour. The distance between rib

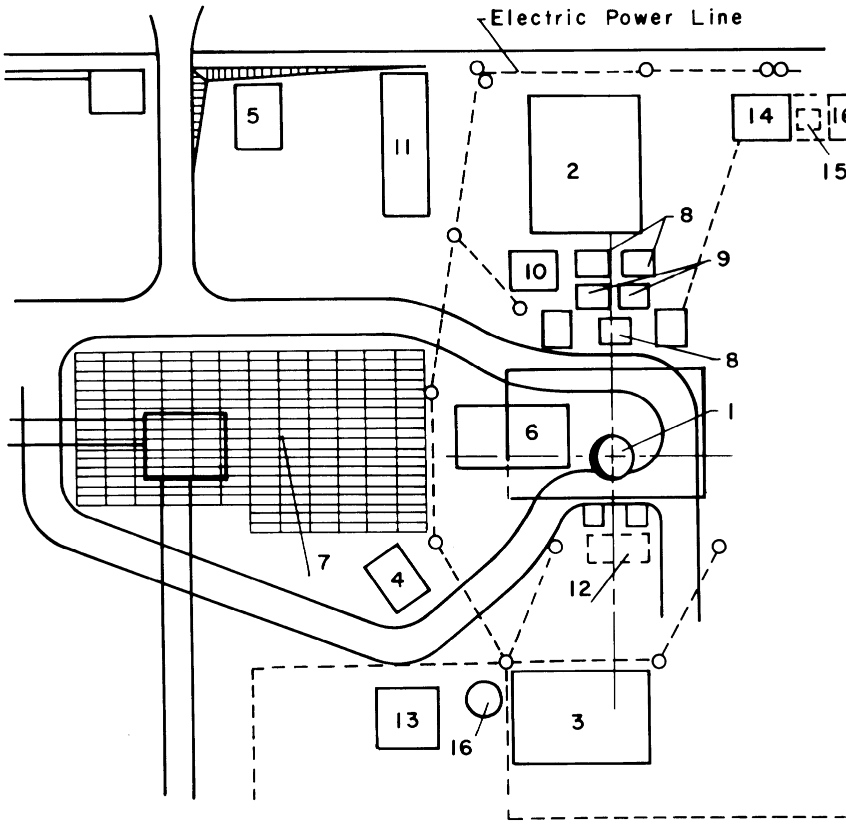


Fig. 17.4.19. Plan of the shaft-sinking site.
1. headframe
2. hoisting machine house
3. compressor house
4. fan
5. blacksmith shop
6. concrete plant
7. open storage for construction materials
8. reel hoists for concrete form
9. reel hoist for hanging stage
10. control house
11. machine shop
12. place for emergency reel hoist
13. water tank for compressor and fire hazard
14. temporary transformers 6.0/0.5 kV
15. compressed air tank, 353 ft^3 (10 m^3) volume
16. diesel generator

Conversion factor: 1 ft^3 = 0.0283 m^3.

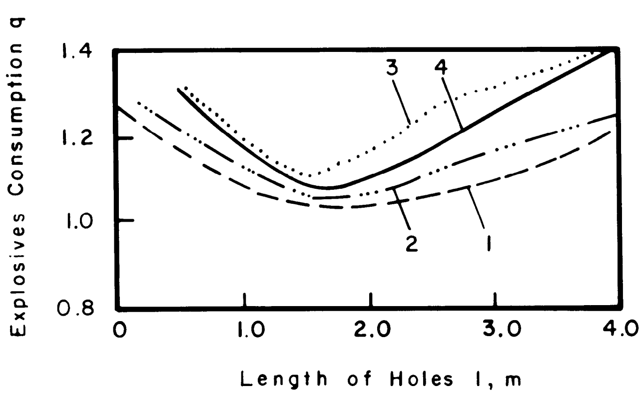


Fig. 17.4.20. Dependence of unit consumption of explosive q = kg/m^3 on length of blastholes (for shales):

1. $\eta = 0.55$
2. $\eta = 0.65$
3. $\eta = 0.75$
4. $\eta = 0.85$

Conversion factor: 1 ft = 0.3048 m.

holes in moderate-strength rock is 3 to 4 ft (0.9 to 1.2 m) and in strong rocks 2.23 to 3 ft (0.7 to 0.9 m). Sometimes it is necessary to place them as close as 7 to 12 in. (0.2 to 0.3 m).

When strata have a dip of more than 45°, the wedge holes are located along the strike, decreasing utilization of holes along the stratification (Fig. 17.4.24).

Overbreakage will be larger on the side of the hanging strata. The distance of the rib holes from the shaft contour on the dip side is 0.75 to 1.2 ft (0.25 to 0.4 m) in moderate-strength strata

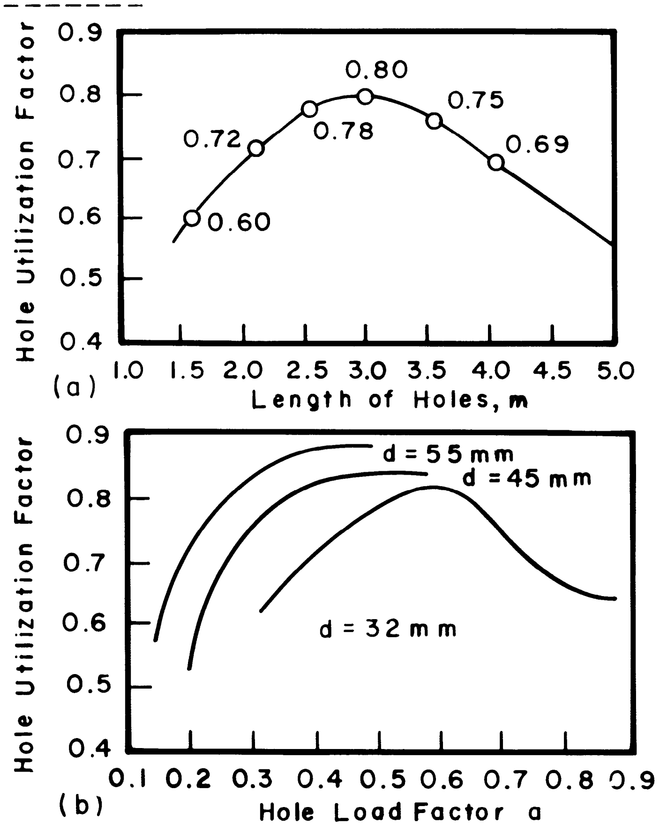


Fig. 17.4.21. Dependence of the hole utilization factor on:
(a) length of holes (shales, q = 1.1 kg/m^3), cartridge 32 mm.
(b) hole load factor with different cartridge diameters.
Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m,
1 lb/ft^3 = 16.02 kg/m^3.

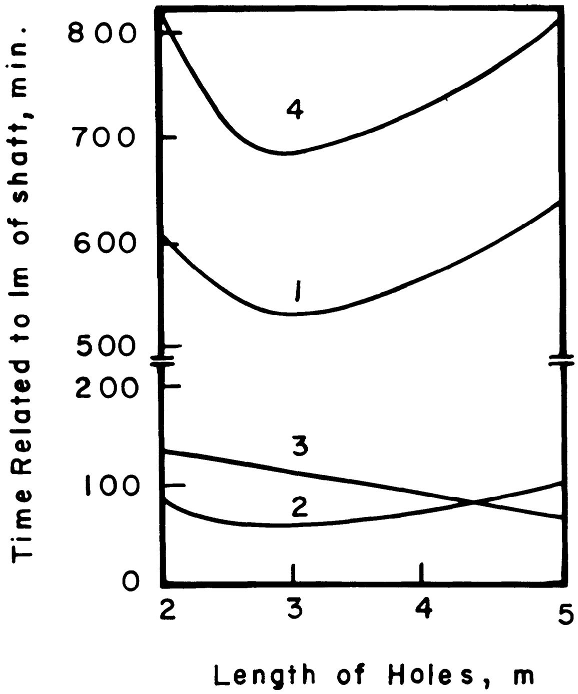


Fig. 17.4.22. Time dependence for a unit length of shaft sunk and length of holes:

1. mucking
2. drilling of holes
3. loading of holes
4. total time 1 to 3 min.

Conversion factor: 1 ft = 0.3048 m.

and 0.15 to 0.25 in strong rocks. In inclined strata, the rock being blasted tends to form a pile on one side of the shaft. It creates difficulties in stationing buckets and in mucking operations.

In rectangular shafts, blasting holes are distributed similarly to horizontal drifts. The most typical and effective systems are the wedge and pyramidal ones (Fig. 17.4.25). A step pattern is used for conditions of larger water inflow (20,000 gal/hr or 80 m^3/h). The deeper part works as a sump for a pump.

Another development in drilling patterns, originating in the Scandinavian countries, is based on a central larger diameter hole (or holes) drilled parallel to the axis of the shaft (tunnel). This is possible only with application of the drilling jumbo, and in shafts, requires special organization of the drilling. Such a relief (burn cut) hole provides an additional free surface and increases the efficiency of blasting, reducing the consumption of explosives. This method was tested in hard rocks of the Canadian Shield for extension of an existing shaft 550 to 1350 ft (180 to 443 m) deep, where the powder factor was 5 to 5.4 lb/yd^3 (3 to 3.2 kg/m^3).

An example of the drilling pattern for a shaft 30 ft (9 m) in diameter in high-strength siltstone (Fig. 17.4.26) shows the distribution of holes and a schematic of the wiring of electric caps. Table 17.4.6 lists other data.

Shooting features include back initiation of shot holes, electric millisecond caps with consecutive delay steps in particular circles (with this same delay in each circle), a three-ring connecting system of electric caps (to diminish the probability of misfire), lighter shooting, and separate shot cables (exclusively for shooting).

A wedge cut was chosen for this round (Fig. 17.4.26). The largest charges were in lever holes (II, III, IV circles) and the smallest in wedge and rib holes (I and VI circles). Blastholes are charged with dynamite (381 lb or 173 kg/round). Millisecond fuses are initiated in six series, one after the other. Four workers load holes; one of them is the firemaster. The cartridge with the cap goes into the hole first. Each shotfirer makes the connections for his circle. Then the firemaster checks connections and makes the main one, to the shot cables. Afterwards, the shot firers go to the surface, and the resistivity of the wiring is measured. Shooting is done with a high-capacity shot lighter. The blast is supposed to crush the rock and lift it and, in this process, increase its volume by the swell factor.

By visual inspection, the shooting effect can be assessed. A centrally located pile of muck implies a correct shot. Piles on the side of the shaft indicate misfires there, or nonuniform hole charges. Since the bottom of the shaft is covered with muck, it is difficult to find misfires. A basic safety precaution in the shaft is to watch for misfires. When localized, an additional shot hole is drilled parallel to shoot out the remnant stump of rock and blasting material in the unfired hole.

Drilling of Shot Holes—Blastholes can be drilled with sinker percussive drills, hand-held or mounted on a special shaft jumbo. Sinkers are self-rotating, compressed-air or hydraulically driven with air or water flushing.

There are two organizational systems: (1) *series,* in which the drilling starts from the shaft center toward its perimeter after completion of mucking, and (2) *parallel,* where mucking and drilling are done at the same time. Drilling starts from the perimeter towards the shaft center. A shaft jumbo can be used only in the series system. A series system is used in smaller shafts when a Cryderman or similar type of mucker is used. This system assures precision in blasthole drilling.

The parallel system is sometimes chosen in large cross-sectional shafts where cactus grab cabin muckers start to clean the sides of the shaft moving towards its center. Drilling is done in the cleaned area. Owing to parallel drilling and mucking, some time saving can be achieved; however, it is at the expense of drilling accuracy. Blastholes can be drilled individually or by groups of drillers. In the first case, approximately 25% of the time is lost in changing bits. In the second, one shaft area is divided according to the number of drillers; each group drills in one sector holes of a certain depth only. Bits are changed only when they get dull (usually after two to four holes). Then the drilling rod is changed (lengthened), and work is done in the next sector. Wooden sticks secure holes against muck.

Drilling equipment is delivered in a special bucket or basket to the shaft bottom. An elastic hose with quick coupling is connected to the main steel compressed air tubing. At the end of the elastic hose, there is a distributor with multiple outlets and valves for particular drills. Drilling is a labor-intensive operation, which takes about 16% of the cycle time.

Shaft drill jumbos have been developed to mechanize blast hole drilling (Fig. 17.4.27). These permit the drilling of larger-diameter holes more precisely, faster, and deeper.

Loading of Blastholes and Shooting—Before the drilling of blastholes is finished, the shotmaster installs caps in the cartridges in a secure, fire-resistant shelter a minimum of 60 ft (20 m) from the shaft. These cartridges are carefully lowered in a special soft-lined container. The shot firer is responsible for appropriate loading of holes. The shot firer checks the hole and then loads the first (charged with a cap) cartridge and puts the first stemming in top of it. Further loading is done by the rest of the crew.

Wedge holes have zero-delay caps. The next circles have millisecond caps. Detonating wires should be well insulated. To

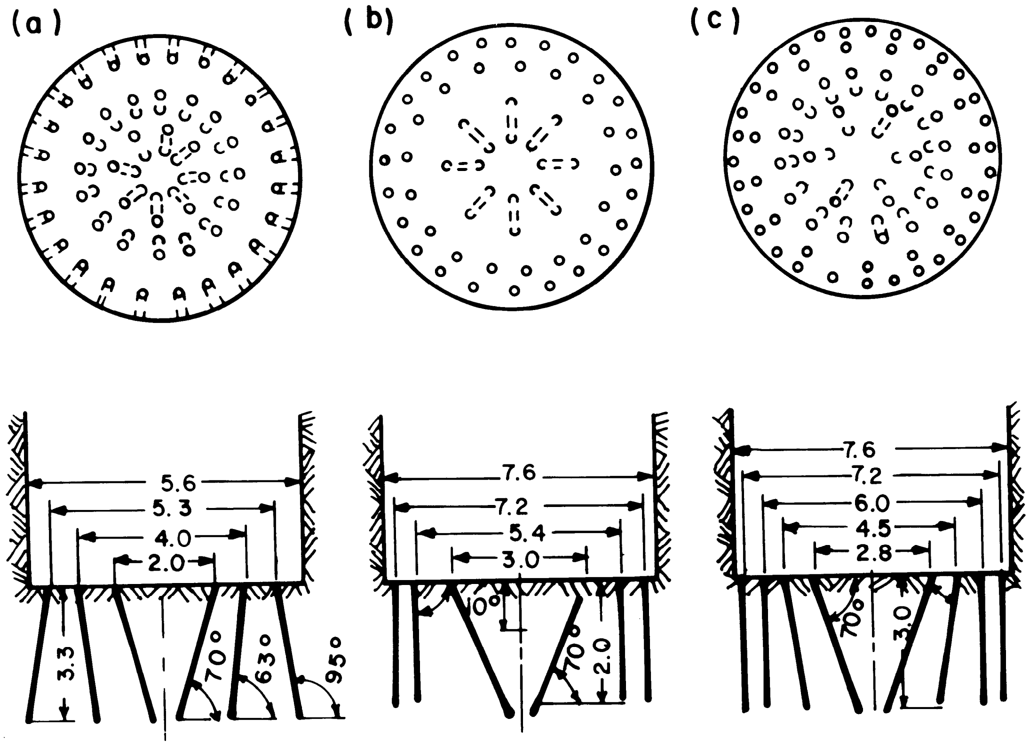


Fig. 17.4.23. Examples of drillhole pattern in circular shaft.
a) in siltstone
b) in shale
c) in sandstone
Dimensions in m. Conversion factor: 1 ft = 0.3048 m.

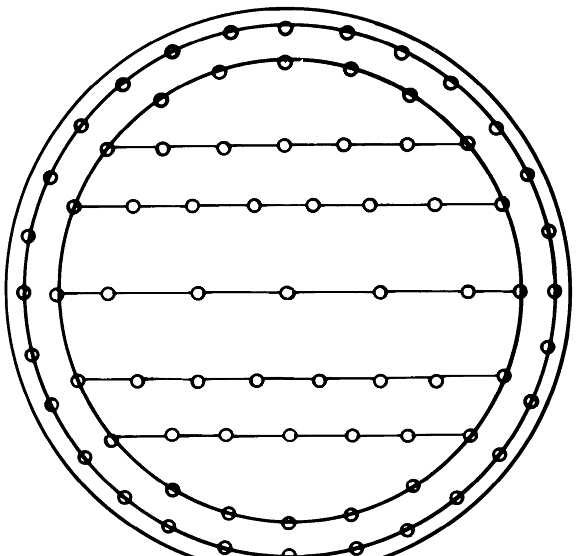


Fig. 17.4.24. Example of drillhole distribution in shaft sunk in inclined strata.

prevent the hazard of stray current, the ends of wires are short circuited until they are connected to the shot cable.

A clay-sand mixture is used as stemming. In wet shafts, flooding to a 10-in. (250-mm) depth substitutes for stemming. Loading time for one hole is about 6 to 8 minutes and for the whole round, about 60 to 90 minutes. Caps of detonating cartridges can be connected in series, in parallel, or in a mixed manner. Connection in series is the simplest method but also the least reliable. Failure of one cap means a misfire of all the holes.

A mixed connection is done in such a way that a group of holes is connected in series, and particular groups are connected parallel to the main shot cable. An example of a mixed connection is shown in Fig. 17.4.28.

When connections are completed, face lights are pulled up on the stage. The shot master is the last to leave the face. Firing is done by a shot lighter from the surface.

SHAFT VENTILATION. Shaft ventilation is carried out by fan and vent tubing, as described in Chapters 11.6 and 11.7.

MUCKING IN SHAFTS. Mucking in shafts is the most time- and effort-consuming operation, accounting for as much as 50 to 60% of cycle time. Table 17.4.7. gives different factors influencing mucking efficiency. Fig. 17.4.29 shows a schematic of sinking phases with an Eimco 630 loader. Fig. 17.4.30 illustrates the section through a shaft sunk with a cactus grab loading unit.

Mucking efficiency is dependent also on the degree of rock fragmentation and the height of the bucket. For mechanical mucking, medium-sized fragmentation, approximately 5 in. (125 mm), is most efficient.

Generally, in a muck pile after a shot, 70 to 80% of the material is fine fragmentation, and the rest is composed of larger fragments.

Muckers—Several types of muckers are in use depending on shaft shape, size, depth, and location, as listed below:

1. Scrapers, used in rectangular shafts and very large-diameter shafts (underground bunkers).

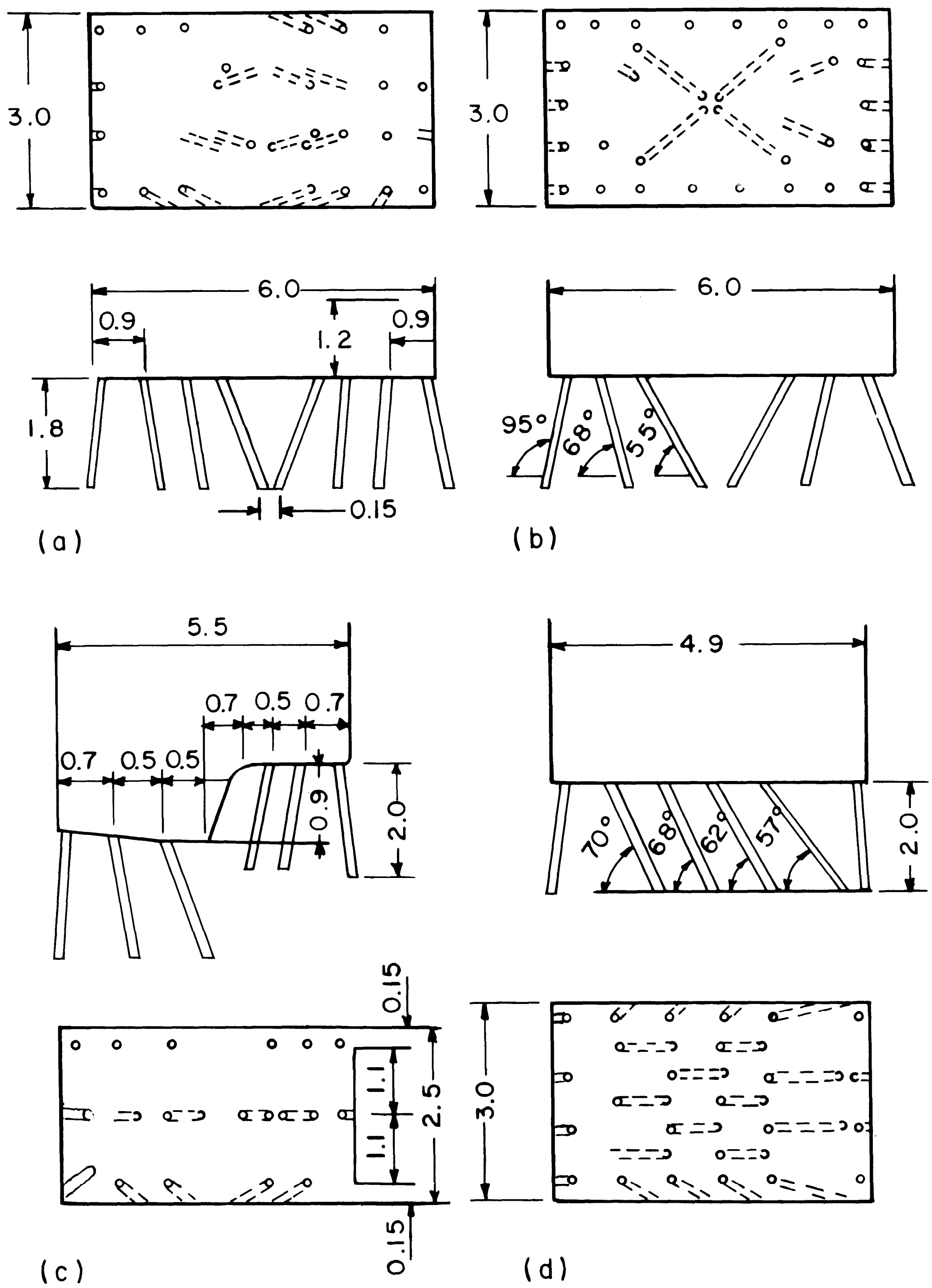


Fig. 17.4.25. Examples of hole distribution in rectangular shafts.

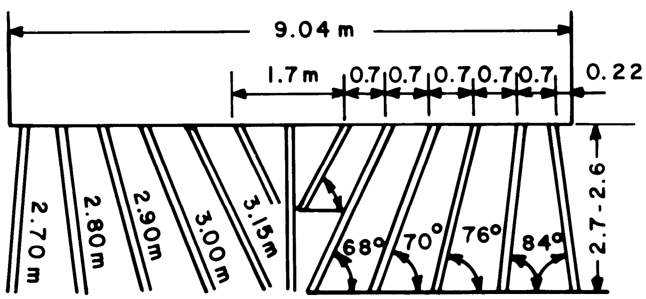
a) wedge cut
b) pyramidal cut
c) step cut
d) wedge-side cut

Dimensions in m. Conversion factor: 1 ft = 0.3048 m.

Table 17.4.6. Data for Example in 17.4.2.2.

Circle	Number of holes in circle	Number of cartridge in hole	Size of charge in hole, kg	Size of charge in circle, kg	Fuse no.	Circle no.	Shot firer
I	4	9	1.125	4.500	0		
II	8	18	2.250	18.000	1	I	A
III	16	16	2.000	32.000	2		
IV	20	15	1.875	37.500	3	II	B
V	24	14	1.750	42.000	4	III	C
VI	24	13	1.625	39.000	5	IV	D

Conversion factor: 1 lb. = 0.4536 kg.



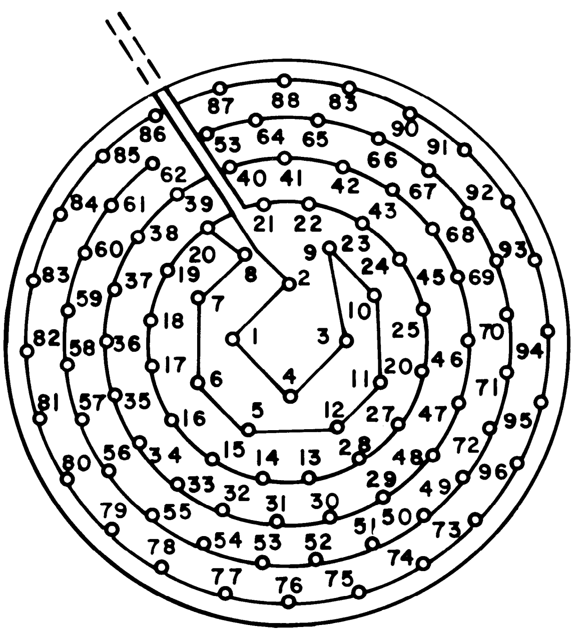


Fig. 17.4.26. Drilling pattern for shaft 27.5 ft (9.04 m) in diameter. Conversion factor: 1 ft = 0.3048 m.

2. Overshot loaders on caterpillars (e.g., Eimco 630).
3. Cryderman muckers.
4. Cactus grab (light).
5. Cactus grab (heavy, on central pivot).

A variety of systems is used. Some are driven by an operator (e.g., Eimco), some are hand-guided at the face when hanging from a platform, and some are attached to the shaft wall (Cryderman) and operated from a control cabin. Heavy cactus grab units have a rotating arm on a central pivot below the stage. The largest muckers are 1.3 yd^3 (1 m^3) capacity. Generally, large-capacity muckers are used for deep shafts with large diameters, where a high level of equipment investment is justified. Muckers can be electric, pneumatic, or hydraulically driven.

Determination of Mucker Capacity—The theoretical mucking efficiency P_t in yd^3 (m^3)/hr is

$$P_t = 3600\, k \frac{V_b}{t} \tag{17.4.15}$$

where k is the fill factor (0.8 to 0.9 for sandstone and 1.1 to 1.2 for shales), V_b is bucket volume in yd^3 (m^3), and t is mucking time in sec.

The mean actual efficiency is found by calculating the actual mucking time and all time losses for unproductive activity such as mucker lowering time to work position, time of temporary support erecting when mucking has to be suspended, and time for final cleaning.

DEWATERING OF SHAFTS. Most shafts intersect one or more water courses, so shaft dewatering is an essential part of sinking.

The characteristic features of shaft dewatering consist of changing water inflow, changing pump head, water containing rock and mud particles, and periodic changes in positioning dewatering devices.

Dewatering can be considered either as limiting water inflow into the shaft or pumping water out of the shaft.

Water inflow into the shaft can be diminished by initial drainage of the rock mass, sealing of the water courses, or by tapping the water.

For dewatering the face, the following devices are used:

1. Rock kibbles and special water kibbles.
2. Face pumps.
3. Combination of face and stationary pumps.
4. Combination of hanging and stationary pumps.
5. Submersible pumps.
6. Airlifts.

Dewatering with Kibbles—Removal of water with the muck being hoisted is the simplest dewatering system. A small portable pump is usually driven by compressed air with a head of 55 ft (18 m) water and output of 40 to 132 gal/hr (0.15 to 0.5 m^3/h) (it could also be any other type of electric pump approved for shaft face conditions).

Usually a kibble (bucket or skip) exclusively filled with water is periodically hoisted up, depending on the need.

A limit of the kibble dewatering system derived from the condition not to flood the face during shooting and ventilating is given by the formula (in SI units):

$$q > 0.8\, D^2 (kl + 0.1)/t \tag{17.4.16}$$

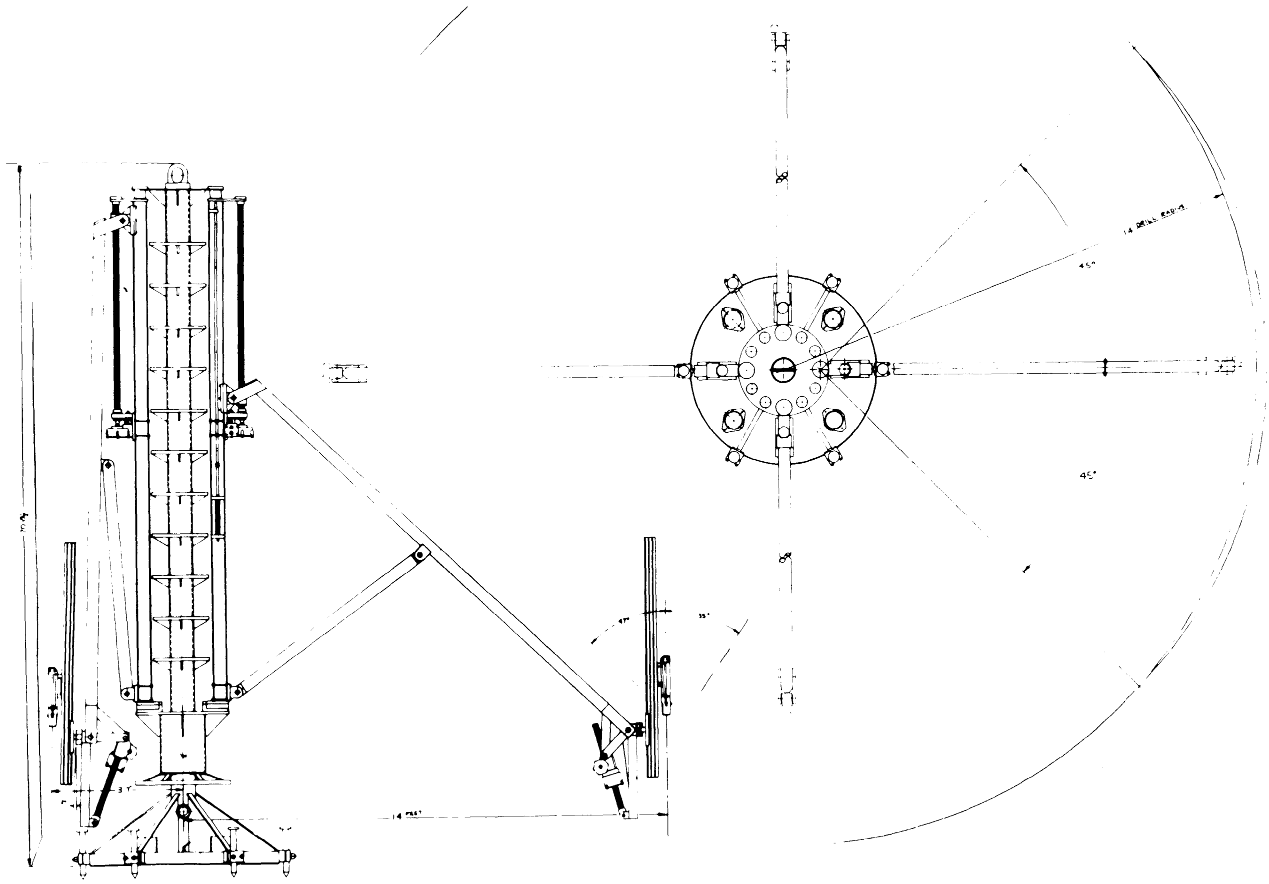


Fig. 17.4.27. ASJB-3B Shaft Jumbo (courtesy: Acme Machinery Co.). Conversion factor: 1 ft = 0.3045 m.

where q is water inflow in m^3/h, t is pumping time in coordination with blasting work in h, k is the coefficient characterizing the size of voids in the muck (0.5 to 0.6 for hard rocks and 0.3 to 0.4 for weak rocks), D is kibble diameter in m, l is depth of blasthole in m, and 0.1 m is the allowable water table level in the shaft.

Table 17.4.9. shows the amount of water inflow that can be controlled by dewatering with kibbles, depending on kibble capacity and the number of kibbles hoisted per hour. For example, from European practice, it is known that the limit for this system of dewatering for shafts down to 490 ft (150 m) deep is 800 gal/hr (3 m^3/h).

Face Pumps—Face pumps work on various principles. Compressed-air-driven membrane pumps are particularly effective in handling dirty water as it gathers in the shaft face and therefore are often used.

Hanging Pumps—Hanging pumps are used to control larger inflows, and when required, heads of 600 to 1200 ft (200 to 400 m). An advantage of these pumps is that they can be easily lifted up during shooting.

Dewatering with Airlifts—The airlift pump is powered by compressed air. The two main systems for these pumps are (1) parallel, where the air pipe runs beside the water pipe, and (2) concentric, in which arrangement the air pipe is inside the water pipe.

Airlifts require substantial submersion, so they are applicable in dewatering flooded shafts. They can handle dirty water, even quicksands.

Intermediate Pumping—When sinking a deep shaft, intermediate pumping has to be applied. The number of stages depends on the depth. On levels chosen as stages, overflow chambers are built. Pumping in stages can be arranged in the following systems:

1. The face pump delivers water to the overflow chamber, and from this it is pumped up by stationary pumps to the surface or to another chamber.
2. The hanging pump delivers water to the overflow chamber from where the stationary pump sends it upward.
3. The face pump pumps water to the overflow chamber where an airlift delivers it up to the surface.

SHAFT LINING CONSTRUCTION. The following factors determine the type of lining: hydrogeologic conditions, shaft function, planned lifetime, shape of shaft section and its depth, availability of construction materials, and construction cost.

When choosing a shaft lining, geotechnical properties of the rock and hydrologic conditions are decisive in most cases. However, the chemical activity (corrosiveness) of the water can also be an important consideration. Since modern new shafts often have automatically operating hoisting gear (moisture sensitive), they should be dry. Main shafts are usually planned for the entire mine life span, so they should be constructed to minimize repairs and maintenance time.

TEMPORARY LINING. Temporary support is placed to protect the crew and equipment against falling rocks from the exposed shaft wall before a permanent lining is placed. Temporary support is used when the face is advancing, leaving a longer lift

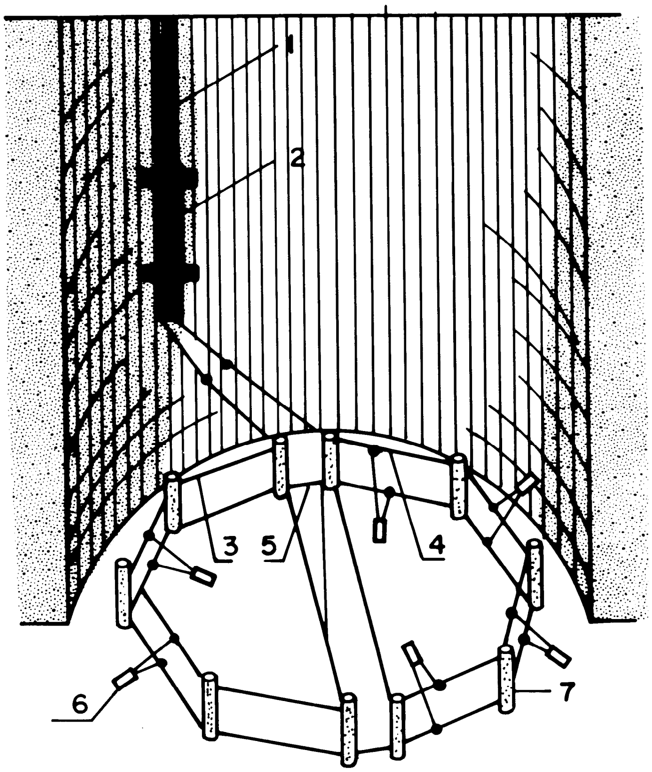


Fig. 17.4.28. Three-ring connection of electric caps.
1. shot cable
2. protecting pipe
3–4. active antenna
5. passive antenna
6. electric caps (in holes)
7. wooden stick supporting antennas.

Table 17.4.7. Factors Influencing Mucking Efficiency

Depth, m	to 150	150–300		300–500		> 500
Factor	1.0	0.9		0.85		0.8
Rock strength *f*	12–15	10–12	6–10	3–6	2–3	0.5–2
Mechnical Mucking Factor	0.85	0.89	0.92	1.0	1.1	1.4
Shaft section area, m^2	up to 4.0	4–8	8–12	12–16	16–20	20
Factor	0.65	0.75	0.85	0.9	0.9	1.0
Water inflow, L/min	up to 100	100–200		200–300		300
Factor	1.0	0.9		0.8		0.75

Conversion factors: 1 ft = 0.3048 m, 1 ft^2 = 0.0929 m^2, 1 gpm = 3.786 L/min.

up to 20 to 120 ft (6 to 40 m) of unlined shaft. Work at the face is then suspended, and the temporary support is dismantled, allowing positioning of a concrete form.

When removal of the temporary lining creates safety problems, it can be left and concrete poured over it.

Numerous systems of temporary lining are in use. One of the most popular types is shown in Fig. 17.4.31. Steel rings in profile are assembled and suspended from round, 1-in. (25mm) steel hangers. Each ring consists of six to eight separate segments bolted together with the possibility of a diameter change to fit the excavation. Boards, iron grates, or sheets are placed behind the rings. The first ring is supported by pins (into the shaft wall), placed every 4 to 5 ft (1.2 to 1.5 m) around the shaft perimeter and every five to six rings vertically. Wooden wedges are driven to make the rings tight.

Permanent Lining—According to the shaft design and environmental conditions, several different permanent lining systems can be applied: (1) timber, (2) brick or concrete blocks, (3) concrete monolithic, (4) reinforced concrete, (5) tubbing (cast iron and precast elements), (6) shotcrete, various systems (e.g., with mesh), or (7) anchor bolts. Quite often, the lining consists of a combination of the above listed types.

Timber Lining—In contemporary shafts, timber lining is very seldom used, and when it is, only in auxiliary shafts with a short life span (interlevel blind shafts).

Timber lining can be "full" with sets lying one on top of the other in more difficult ground conditions, or in better conditions, on posts with the distance of particular sets 2 to 5 ft (0.5 to 1.5 m). In both arrangements, every 25 to 45 ft (8 to 15 m), bearing beams are set, placed in nests 2 to 2.5 ft (0.5 to 0.7 m) deep. They are wedged to get the entire structure tightened into the surrounding rocks (Figs. 17.4.32 and 17.4.33).

Brick Lining—Brick lining was popular before the mechanization of shaft sinking. There are some characteristic features of this lining such as simplicity and ease of construction, ability to carry load immediately, ease of repairs, and resistance to corrosive waters (Fig. 17.4.34) that can be advantageous in certain conditions.

Among the disadvantages of brick lining are the following:
1. Time- and labor-consuming erection.
2. Low strength.
3. High cost (when labor is expensive).
4. Substantial permeability.

Concrete Block Lining—Concrete block lining is a type of improved brick lining possessing a decreased number of seams and the higher strength of a concrete shaft wall. This lining is also less labor-intensive when compared to brick lining.

Monolithic Concrete Lining—This is the most popular shaft lining. It has many advantages when compared to other types, such as:

1. The possibility of complete mechanization of the construction process with slip or switch forms and by transporting concrete through the slickline.
2. Good bond between the lining and shaft (rock) wall such that shaft footings (foundations) are not needed.
3. Decreased labor intensity (three to six times) and costs (30 to 40%) compared to bricks.
4. High strength, resulting in less required excavation.

Disadvantages are
1. Less resistance to corrosive water.
2. Sensitivity to rock mass movement.
3. Inability to take load immediately after placement.
4. Difficulty of repairs.

In ordinary conditions, a moderate-strength (2400- to 3500-psi, or 17 to 25 MPa) concrete is sufficient for lining purposes. It is made of natural aggregate and Portland or slag cement (3500 to 5000 psi, or 25 to 35 MPa). The concrete mix is prepared in the vicinity of the shaft (in a mechanized concrete plant) and sent down the shaft by one or two pipelines (sticklines). The minimum lining thickness is 8 in. (200 mm) and the maximum 32 in. (800 mm). There are several technologies of construction. When sinking in series, short segments of shaft, 13.1 to 16.4 ft (4 to 5 m) in length, are lined with the application of a collapsible steel form, which is relocated after the concrete sets. Lining made in that way is not a monolith but consists of segments with seams (Fig. 17.4.35).

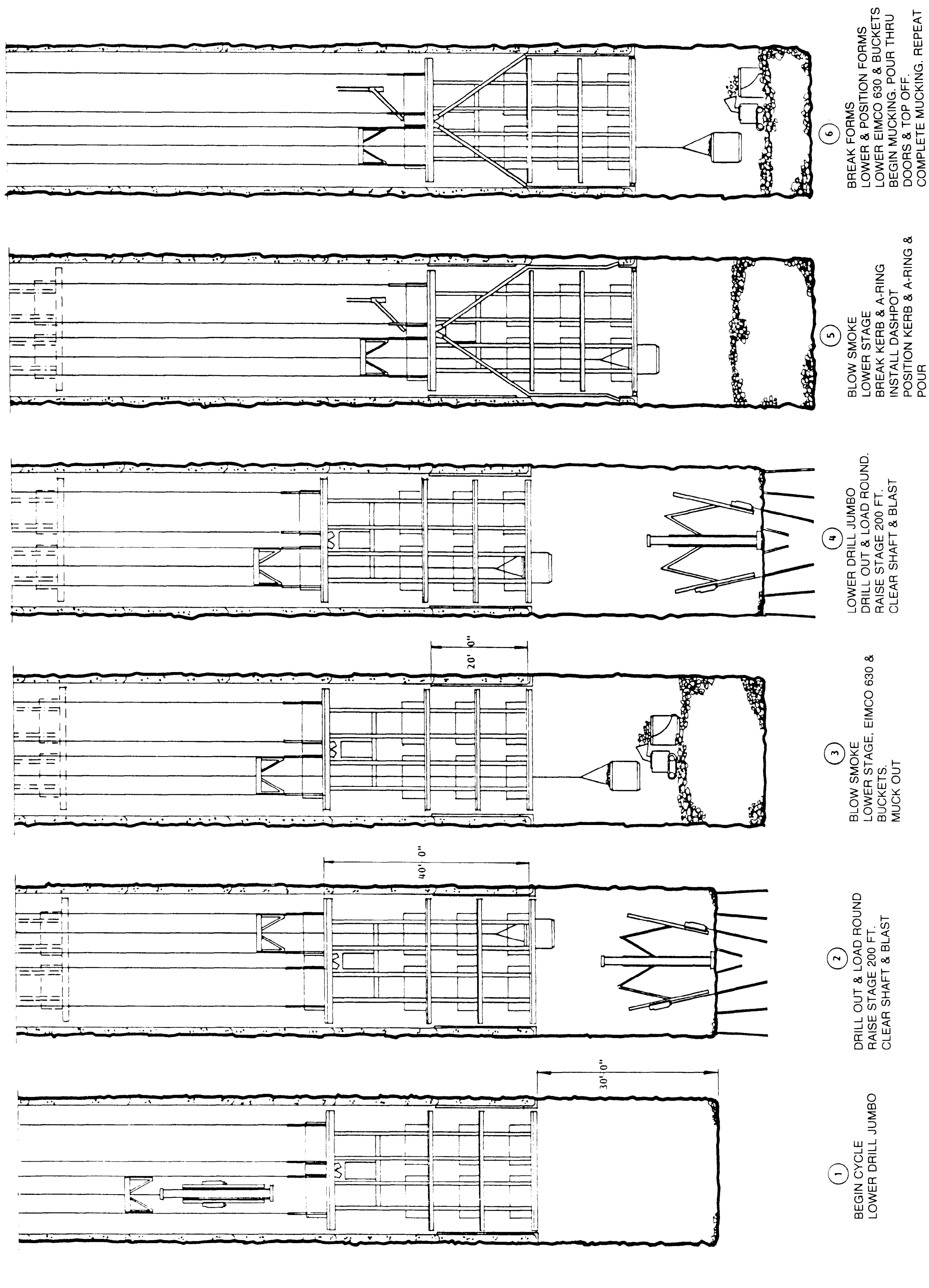


Fig. 17.4.29. Schematic of sinking operations. (After Hynd, 1982.) Conversion factor: 1 ft = 0.3048 m.

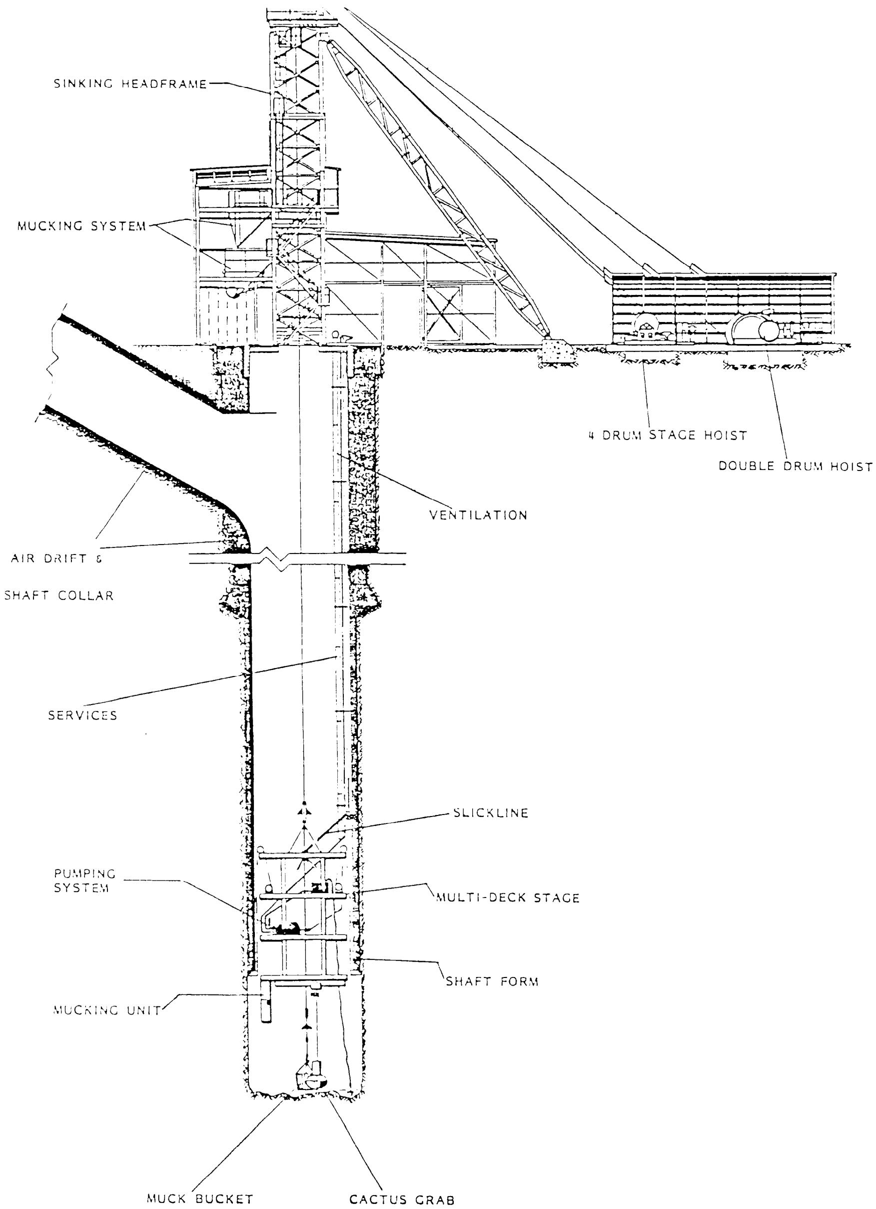


Fig. 17.4.30. Typical shaft sinking arrangement. (After Hynd, 1982.)

Table 17.4.8. Allowable Water Inflow, When Dewatering Is Done with Kibbles (Buckets)

Number of kibbles hoisting n/h	Water inflow (m^3/h) when controlled with kibbles of capacity m^3			
	1.0	1.5	2.0	2.5
5	2.25	3.4	4.5	6.75
10	4.5	6.8	9.0	13.5
15	6.75	10.2	13.5	20.3

Conversion factor: 1 ft^3 = 0.0283 m^3.

When the shaft is sunk in long lifts, a sliding concrete form can be applied. As a result, longer sections of monolithic lining are obtained.

Shotcrete Lining—Shotcrete lining can be made in a few variants, such as regular shotcrete, shotcrete with rock bolts, reinforced concrete, and reinforced concrete with bolts.

Shotcrete lining is applicable in dry shafts in rocks with good strength. Especially in blind shafts of smaller diameter where the use of concrete forms is limited, shotcrete is very useful. Shotcrete has some very attractive technical features such as very good binding with rock, tightness, and high strength due to low water/cement (w/c) ratio.

Shotcrete, however, is more often used in combination with rock bolts and mesh. The last two are generally installed as a temporary support, prior to or after the first shotcreting. Finally, shotcrete covers the mesh pinned to the shaft wall with bolts and creates a strong but thin shell of lining.

Rock Bolt Lining—Rock bolt lining is used primarily in salt mine shafts. Salt has creep characteristics and by flowing toward the opening, will exert an excessive pressure on a rigid shaft lining.

Two main systems are in use in such conditions. When salt rock is in good shape, no lining is required for ground control. In the presence of laminations and an overall tendency to weathering that creates worse conditions, rock bolts and mesh made

Table 17.4.9. Applicability of Special Methods of Shaft Sinking

	Method	Rock Mass	Thickness of Water-bearing Strata (m)	Water Inflow	Range of Depth (m)	Approximate Rate of Advance m/month	Comments
1.	Pile wall a) wooden b) sheet pile wall (steel profiles or pipes)	Quicksands, soils (saturated with water without stones and rock fragments)	2–6 6–10	limited	6–15 down to 30	2–5 6–10	Required presence of impermeable strata of clay, into which the piles could be driven to lock water in above water-bearing strata
2.	Concrete pile wall drilled or driven piles	Quicksands, soils, loose rocks (sands, gravels), clays saturated with water (also with stones)	10–20	any	25–30	8–10	Concrete pile wall can be used as a foundation for a shaft headgear.
3.	Caisson method a) Without pumping water in flow rocks b) With steady pumping in weak and loose water bearing rocks	Uniform flowrocks or loose rocks (sands gravel), clays, without intrusion of hard rock or boulders	10–15	not limited up to 0.3 m^3/min	down to 30, in special cases more when vibrators or compressed air applied	10–20	Flat-lying strata required or with little dip. Presence of the clay strata below water-bearing zone 2–3m thick is necessary.
4.	Caisson with compressed air	Loose or noncoherent water bearing strata	8–30	moderate	down to 40	20–30	Maximum air pressure in the face chamber 3–3.5 atmospheres
5.	Depression of groundwater level	Mid and coarse solids and gravels with coefficient of filtration 10–100 m/day. Fractured competent rocks.	20–30	great	around 100–150	15–20	Not supposed to be applied in clay rocks, quicksands, and alternating packages of permeable and not permeable rock.
6.	Chemical grouting	Sands with clay content not exceeding 10–15%. Coefficient of filtration 2–80 m/day. Also competent rocks with small tissue	a few	moderate	unlimited		Usually used as an additional interventional method
7.	Grouting (cement)	Competent rocks, with tissues and cracks filled with water (sandstones, shales, mudstones, limestones), and coarse sands and gravels	not limited	mid and large, up to about 10 m^3/min	any	10–15 in good conditions 20–30	Not recommended in flow rocks or quicksands, with water flow velocity more than 80 m/day, or in presence of caverns and large voids
8.	Grouting with clay	Water-bearing rocks with large cracks and voids	not limited	not limited	not limited	10–15	Can be used in presence of aggressive waters. Usually is used combined with grouting (cement).
9.	Bitumination	Competent rocks with large cracks and voids, also gravels	20–30	mid and large	down to about 40		Used as an auxiliary method for local sealing of the rocks with high flow velocity and aggressive waters.
10.	Freezing of the rock mass	All types of water-bearing rocks	10 to 1000	not limited	down to 1000	differs depending on depth, sinking frozen rocks up to 50 m overall less due to length of freezing method	Most reliable method, however, costly and time consuming because of expensive and time-consuming preparation works. (drilling of the freezing holes freezing themselves).
11.	Shaft drilling	Weak, water-bearing strata	several hundred	not limited	to 600		Thickness of competent hard rocks not supposed to exceed 10% of overall thickness of water-bearing strata.

Conversion factors: 1 ft = 0.3048 m, 1 ft^3 = 0.0283 m^3.

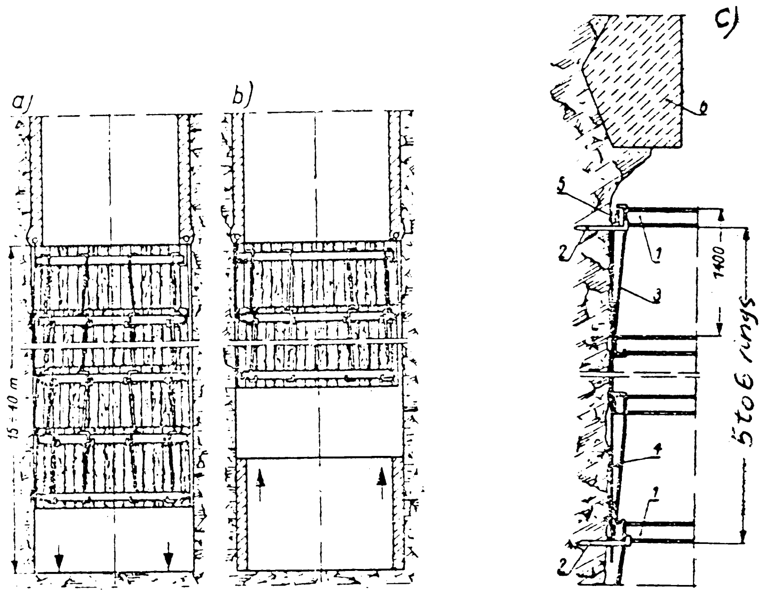


Fig. 17.4.31. Schematic of shaft sinking with long lifts and temporary support.
a) phase of sinking
b) phase of lining (permanent)
c) temporary lining
1. ring
2. rock pin supporting the ring
3. hook to suspend next ring
4. iron sheet liners
5. wedges to fasten lining
6. base ring or permanent lining
Conversion factor: 1 ft = 0.3048 m.

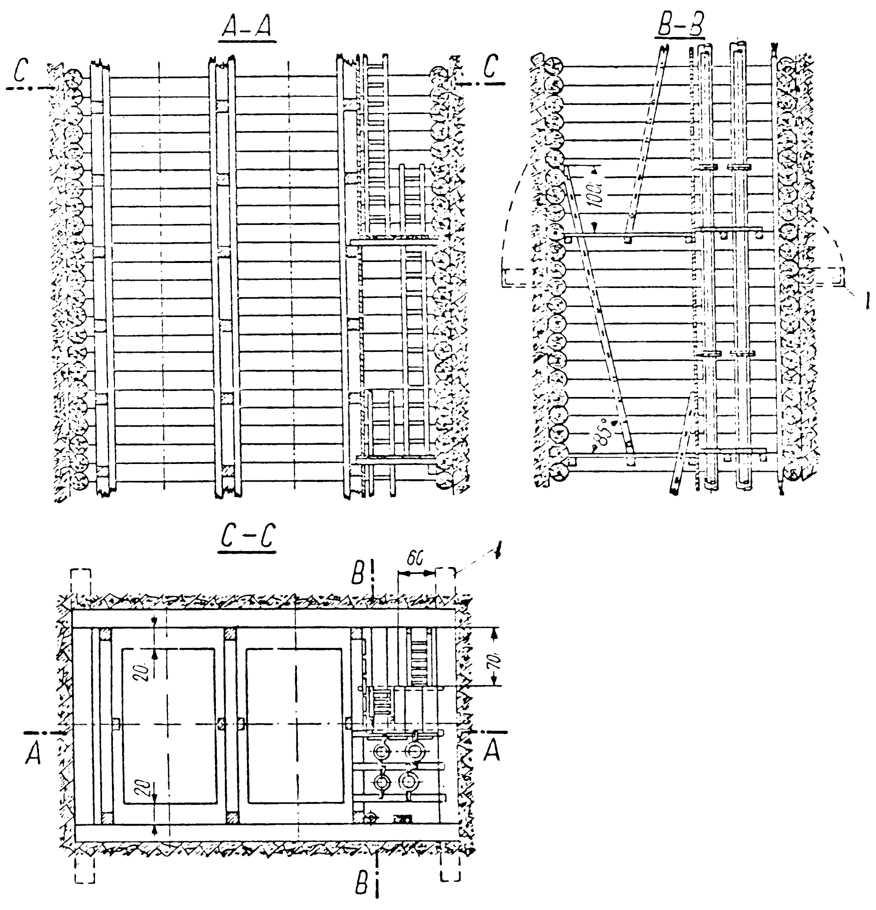


Fig. 17.4.32. Timber lining—full.
1. bearing set
Round 16–22 cm dia. or square 14–20 cm timber used for sets. Timber should be treated to avoid fast decaying. Dimensions in cm. Conversion factor:
1 in. = 2.54 cm.

from synthetic materials (e.g., plastic, which does not corrode) are used. This same support system in salt is applied in insets, chambers, and other capital openings.

17.4.2.3 Specialized Methods of Shaft Sinking

CLASSIFICATION. Special methods of shaft sinking are used when the following conditions are encountered:

1. Incompetent, water-bearing rocks (e.g., quicksand).
2. Weak, unstable, soil-type rocks.
3. Competent fractured rocks, with high water inflows above 130 gpm (0.5 m^3/min).

There are numerous special methods varying among themselves in applicability because of environmental conditions and range of depths. The characteristics of these are given in Table 17.4.9.

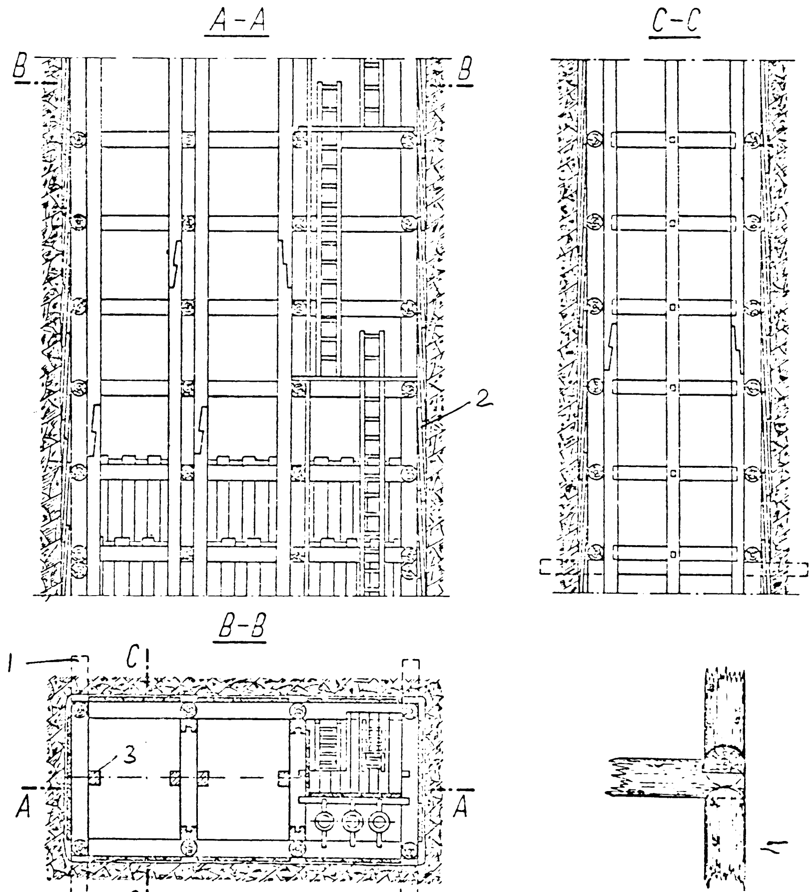


Fig. 17.4.33. Timber lining on posts.
1. bearing set
2. lining with 1-in. (25-mm) boards
3. guides.

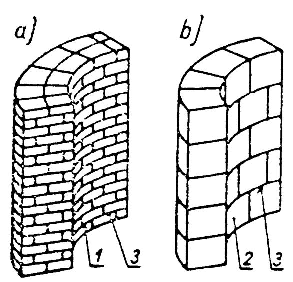


Fig. 17.4.34. Schematic of the cut-out of:
a) brick and b) concrete block shaft lining.
1. brick
2. wedge concrete shaft block
3. seams filled with mortar.

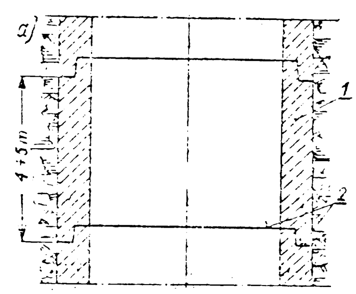


Fig. 17.4.35. Section through the shaft with concrete lining, made with application jump form.
1. concrete
2. seam.
Conversion factor: 1 ft = 0.3048 m.

WOODEN PILE WALL. The initial diameter of the shaft collar D_g (Figs. 17.4.36 and 17.4.37) should be established according to the following:

$$D_g = D + 2(d + 2b + c + f) \qquad (17.4.17)$$

where d is permanent lining thickness, b is width of guide rings, c is thickness of driven piles, and f is thickness of guide beams.

SHEET PILE WALL (STEEL). Construction of a steel sheet pile wall is shown in Fig. 17.4.38.

CONCRETE (REINFORCED CONCRETE PILE WALL). Two currently used methods of placing a concrete pile wall are piles driven with an outer steel tube with a diameter of 20 in. (520 mm), and drilled piles, with or without slurry, leaving diameters 20 to 48 in. (520 to 1200 mm). The outer steel pipe is forced into the ground by a pile driver, whereas drilled piles are made by means of a drilling rig. Where the ground does not assure stabil-

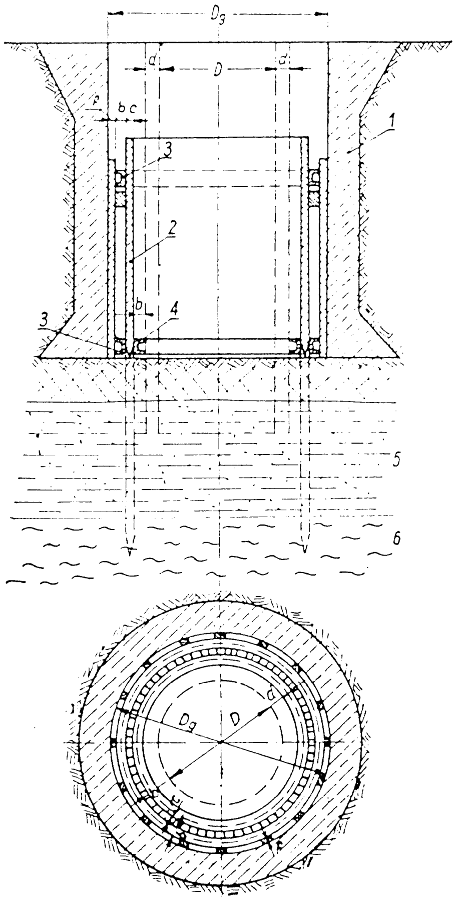


Fig. 17.4.36. Sketch of vertical wooden pile wall for circular shaft.
1. shaft collar
2. piles
3. outer guidance rings
4. inner guidance rings
5. water bearing strata
6. underclay strata.

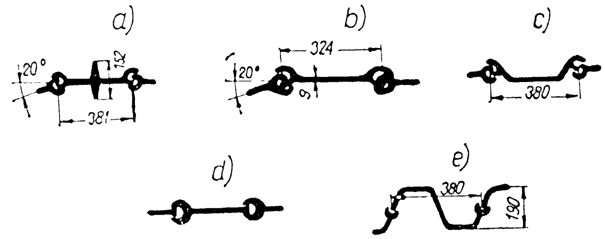


Fig. 17.4.38. Sheet piles.
a to e) different type of joints of sheet pile walls. Lengths up to 15 m. Joints allow consecutive sheet pile to deviate 20°. Dimensions in cm. Conversion factors: 1 in. = 2.54 cm, 1 ft = 0.3048 m.

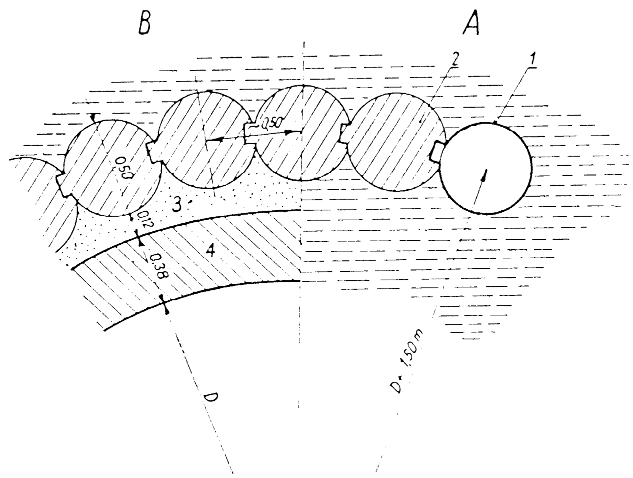


Fig. 17.4.39. Lateral section.
a) driving construction
b) with shaft lining erected with protective pile wall.
Conversion factor: 1 ft = 0.3048 m.

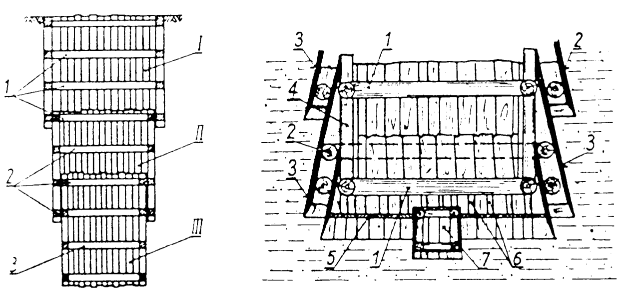


Fig. 17.4.37. Sketch of wooden pile wall for rectangular shaft.
a) vertical method, with diminishing lateral section, I,II,II,IV subsequent segments, b) oblique method, with steady lateral section
1. main rings
2. auxiliary rings
3. driven piles
4. post fastening rings
5. floor (boards)
6. expanding posts
7. well for intake basket of the pump.

ity of the side wall, a slurry is used to prevent failure of the hole. The concrete used for piles has a low water/cement ratio (approximately 0.4). To assure tightness of the concrete, it is placed in batches and compacted with a rammer.

Piles driven beside each other (Fig. 17.4.39), cutting into the side of the fresh concrete of the neighbor pile, create a tight wall, preventing water and loose rock from penetrating the excavation (Fig. 17.4.40). For longer pipes, steel reinforcement is used. The limit of depth for this method is around 120 ft (40 m).

CAISSON METHOD. To permit advance of the caisson with weight Q and resisted by friction F, the following conditions have to be fulfilled (in SI units).

1. With water being pumped out: $Q > F$
2. Without pumping water: $Q > F + G$

$$Q = \frac{\pi(D_2^2 - D_1^2)}{4}\gamma H + B \tag{17.4.18}$$

a. Weight Q^1 of caisson partly submerged to depth H is $Q^1 = Q - G$, where G is calculated,

$$G = \frac{\pi(D_2^2 - D_1^2)}{4} H_w\,\delta_w + V_B\,\delta_w \tag{17.4.19}$$

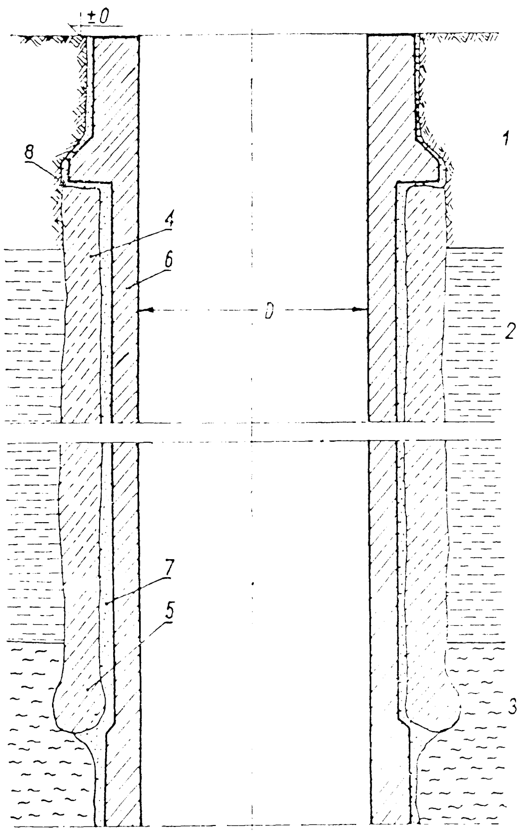


Fig. 17.4.40. Vertical section through the shaft.
1. dry ground
2. water-bearing ground
3. nonpermeable clay strata
4. pile wall
5. base of the pile
6. shaft lining
7. inner layer of concrete
8. upper end of the pile wall.

where D_1 is inner diameter in m, D_2 is outer diameter in m, H is height of caisson in m, H_w is height of submerged part of caisson in m, B is weight of cutting toe in t, V is volume of cutting toe in m^3, δ_w is specific gravity of water in t/m^3, and γ is specific gravity of lining material in t/m^3.

b. Friction force, *F:*

$$F = \sum_{i=1}^{i=n} F_i = \pi D_2 \sum_{i=1}^{i=n} h_i \, f_i \qquad (17.4.20)$$

where f_i is friction force per unit in a particular strata in t/m^2, and h_i is thickness of particular strata thickness in m (Fig. 17.4.41).

For shallow caissons, the recommended value of friction force per unit area is in the range of 2.25 to 2.5 t/m^2 (when the depth of caisson does not exceed 75 ft or 25 m).

If either condition, $Q > F$ or $Q > F + G$, is not fulfilled, then the caisson will not sink and special methods are required, such as (1) some means to diminish friction between the lining and ground, or (2) additional load Q provided by ballast or jacking.

It is possible to diminish the friction by injecting water at 30 to 45 psi (210 to 320 kPa), or more than hydrostatic pressure, through the holes in the cutting toe.

Caisson Method With Compressed Air. The range of application of this method is limited to about 90 ft (30 m) because the maximum air pressure should not exceed 50 psi (350 kPa). Decompression to avoid the bends associated with working in a pressurized atmosphere is required for face crews. Compressed air in the caisson forces water back into the ground; therefore, face operations are in dry conditions. The pressure should be so adjusted that water will stay about 4 to 25 in. (100 to 400 mm) away. In this method, a lock chamber or chambers are required for transfer of the crew and for materials handling.

The thickness of roof at the working chamber is calculated according to the maximum pressure with a safety factor of 1.5. It is always a very solid reinforced concrete structure.

The lock dock has usually more than one chamber to accommodate hoisting facilities, personnel traffic, and materials handling.

To assure proper sinking, the vertical force downwards must be larger than the combined effect of friction and the vertical force acting upwards caused by positive air pressure in the caisson.

In the Moscow (USSR) coal basin, the caisson method with compressed air was often used, with rates of advance in sinking of (mean) 60 ft (20 m)/month and (maximum) 150 ft (45 m)/month.

Sinking Method With the Depression of Water Table. Design consideration for application of this method must be preceded by:

1. Choice of the most suitable way for depression of the water table in given conditions.
2. Determination of the number, spacing, diameter, and depth of the wells.
3. Calculation of the radius of the depression cone, amount of water to be pumped out, and the height of the remaining wet portion of the water-bearing strata.
4. Equipment, hardware, and pumping facilities.
5. Choice of the auxiliary method to go through the wet remainder of the water-bearing strata.

To lower the water table in the area of shaft sinking, the following system can be applied:

1. Pumping water from three to six depression wells drilled around the planned shaft perimeter, placed on a circle about 12 ft (4 m) larger than the diameter of the shaft excavation.
2. Draining water through drillholes to existing openings in the mine below.
3. Draining water through drillholes to water-absorptive strata (e.g., dry sands under water-bearing strata).

The first of these systems is the one most often used.

Wells. Wells are drilled with 14 to 20 in. (300 to 500 mm) diameters, without drilling mud. They are spaced uniformly around the future excavation. Within the range of the water-bearing strata, they are equipped with filters perforated by round or elongated holes 0.2 to 0.35 in. (5 to 10 mm) in diameter, spaced in a checker design, with an outside screen having eyes 0.015 to 0.035 in. (0.4 to 0.9 mm) in diameter, sized against entry of sand particles. Generally, the construction of the filter must be suitable for the character of strata to be drained.

Submersible drainage pumps have a wide capacity range, varying up to 92,500 gal/hr (350 m^3/h).

Calculations for well efficiency are based on principles of flow through porous media. The objective of the design is to determine a pumping regime to achieve a dry-out zone surrounding the shaft. This is created by the cumulative effect of the depression cones around particular drainage wells (Fig. 17.4.41). The movement of water through a porous media is given by the Darcy formula,

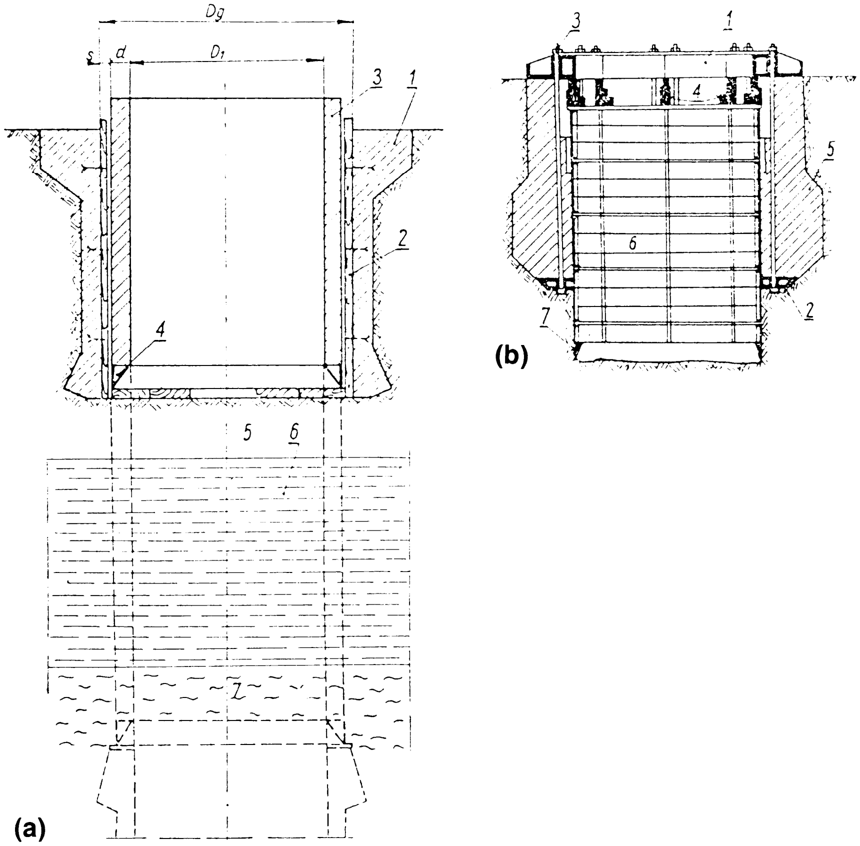


Fig. 17.4.41. (a) Shaft collar for the caisson method.
1. shaft collar
2. guide beams
3. brick lining
4. cutting toe (boot)
5. dry strata
6. water-bearing strata
7. nonpermeable (clay) strata

b) Shaft collar for jacking at the cast iron tubbing lining.
1. reaction ring for hydraulic jacks
2. auxiliary reaction ring
3. pull rods connecting rings with the wall of the collar
4. hydraulic jacks
5. shaft collar wall
6. water-bearing strata
7. nonpermeable strata.

$$V = K\frac{dh}{dl} = Ki + K\frac{h}{l} \qquad (17.4.21)$$

where V is mean velocity of flow per unit time through a unit area or mean filtration velocity in m/s; $h = h_2 - h_1$, or the difference of water levels in the ground (depression) created by pumping water or head loss during flow; l is distance in which head loss appears in m, K is the coefficient of filtration (permeability factor) in m/s; and $i = \frac{h}{l} = \frac{h_2 - h_1}{l}$, or head loss over a unit distance (hydraulic gradient), which is a dimensionless quantity.

The coefficient of filtration K may be expressed in the following units: ft/day (m/s, cm/s, m/day). It has a characteristic value for a particular rock, depending upon the rock structure, grain size, etc.

The best drainage results are obtained in coarse and mid-size sands with K = 30 to 300 ft/day (10 to 100 m/day). Within the feasible range of drainage methods, it will successfully compete with other methods in terms of costs. It could be combined with the caisson method (with compressed air) to allow lowering the air pressure in the working chamber.

One of the negative features, especially in fine sand, is the carrying away of silt particles entrained in the water being pumped. This can cause substantial subsidence effects. An application of adequate filters can help to control a siltation problem.

SHAFT SINKING WITH APPLICATION OF GROUTING METHOD. Based on geologic data, the following decisions have to be made prior to the start of the project:

1. Choice of the grouting system, including whether grouting will be done from the surface or from the bottom of shaft; determination of the grouting depth; and determination of the length of the grouting sections.
2. Determination of the number of grouting holes and their respective distances, diameters, depth and angle of dip, and applicable drilling method.
3. Construction and calculation of the grouting plug.
4. Grout type to be used, its density and admixtures.
5. Injection system and required equipment.
6. Determination of working pressure to be applied in particular sections, and approximate calculations of grout consumption.
7. Number of control holes and their locations.
8. Set of drawings corresponding to the above list of problems.

Necessary hydrogeologic information should contain:

1. Number and characteristic of water horizons.
2. Chemical analysis of water flows and determination of their degree of corrosiveness.
3. Flow velocity.
4. Coefficient of filtration.

GROUTING SYSTEMS. Properly performed grouting seals the rock mass in the vicinity of the shaft, making sinking possible by a conventional method. Grouting is performed from the surface when the water-bearing strata are relatively thick and close to the surface, or from the shaft face when the strata to be grouted are located at such a depth that it is not feasible to drill long holes from the surface. Basically, grouting is most successful in water-bearing, competent, but fractured rocks. Grout fills the fractures and cracks, creating a sealed zone around the shaft to be sunk and protecting it against water inflow.

Grouting for sealing is often applied to improve tightness of the lining in existing shafts and to eliminate the voids behind the

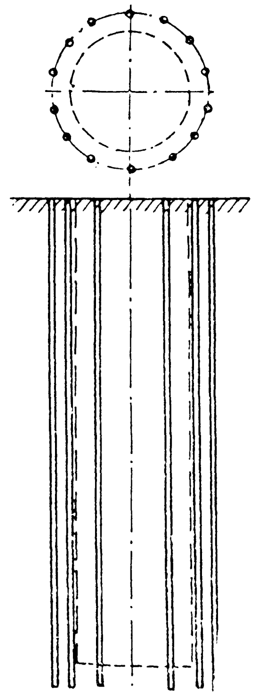


Fig. 17.4.42. Schematic of distribution of grouting holes drilled from the surface.

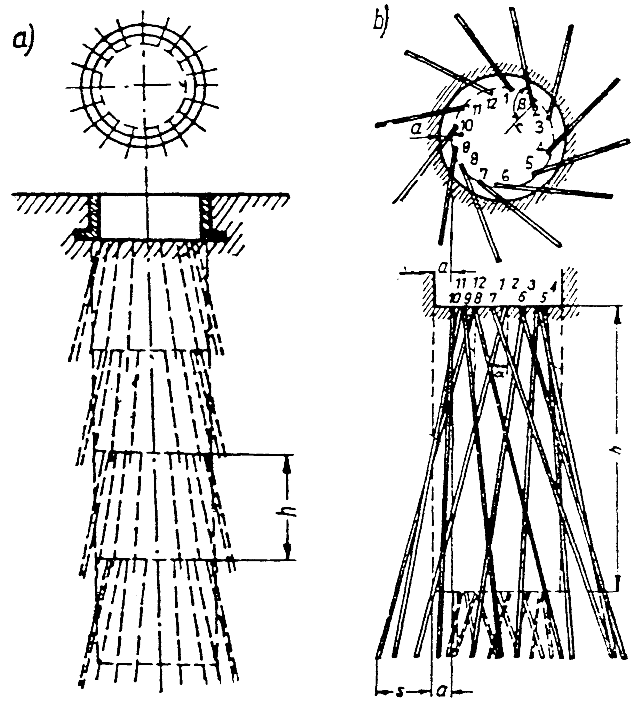


Fig. 17.4.43. Schematic of grout hole distribution drilled from the face.
a) general schema
b) sketch of one section with diagonal holes.

lining, as well as to seal rocks in the immediate vicinity of the lining.

GROUTING FROM THE SURFACE. Grouting holes with an initial diameter of 3 to 5 in. (75 to 150 mm) are drilled vertically around the future shaft on a circle with a radius 10 to 12 ft (3 to 4 m) larger than the excavation diameter.

The diameter of the holes should be as small as possible to minimize drilling cost and to maintain a sufficient velocity of grout flow to avoid sedimentation of cement particles. The distance between the holes usually is in the range of 6 to 12 ft (2 to 4 m) (Fig. 17.4.42). Basically, holes are without casings, excluding parts in loose rocks (usually near the surface). The entry to the hole has a cemented guide pipe 10 to 15 ft (3 to 5 m) long.

Depending on the thickness of the zone to be grouted, one or more grouting sections are chosen. The practical length of the section is 75 to 150 ft (25 to 50 m). In rocks with uniform fractures, distribution, and porosity, longer sections are used.

About 24 hours after the upper section is grouted, the hole is redrilled, in relatively soft grout; it is extended by the length of the next section and then grouted. After the drilling holes are flushed with water until clear, water comes out from the hole. Holes should be straight, and with their greater depth, any deviation should be recorded.

GROUTING FROM THE SHAFT FACE. This system (Fig. 17.4.43) is recommended when water-bearing strata occur below 300 ft (100 m) and when their thickness does not exceed 210 ft (70 m). The length of sections vary between 36 to 75 ft (12 to 25 m).

Holes are drilled along the length of grouting section, usually by means of heavy drill hammers or small rigs when space allows. Holes are directed at an angle in the vertical plane to get the hole bottoms 3 to 6 ft (1 to 2 m) away from the future excavation.

To increase the probability of encountering fractures by the hole, they are also tilted in a tangential direction (Fig. 17.4.43b). The distances between the hole entries are 2.5 to 4.5 ft (0.8 to 1.5 m). Spacing between the circles of holes is 4.5 to 6 to 9 ft (1.5 to 2 to 3 m).

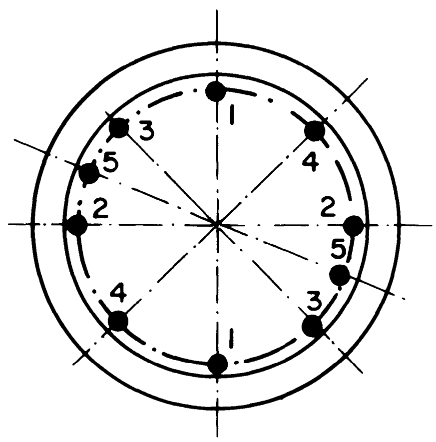


Fig. 17.4.44. Sequence of drilling and grouting opposite holes in pairs.

The number of holes drilled depends on the rock character, shaft diameter, and hydrogeologic conditions. Practically, it varies from 10 to 20 to 30. The initial drillhole size is 2 to 3 in. (50 to 75 mm); the end size is 1 1/4 to 2 in. (30 to 50 mm). The sequence of drilling and grouting is as follows:

1. The first one-half of the total number of holes is drilled and grouted; during hardening of the cement, the second half is drilled and grouted.

2. All holes are drilled and grouted, one after another, starting with the hole with the largest water inflow.

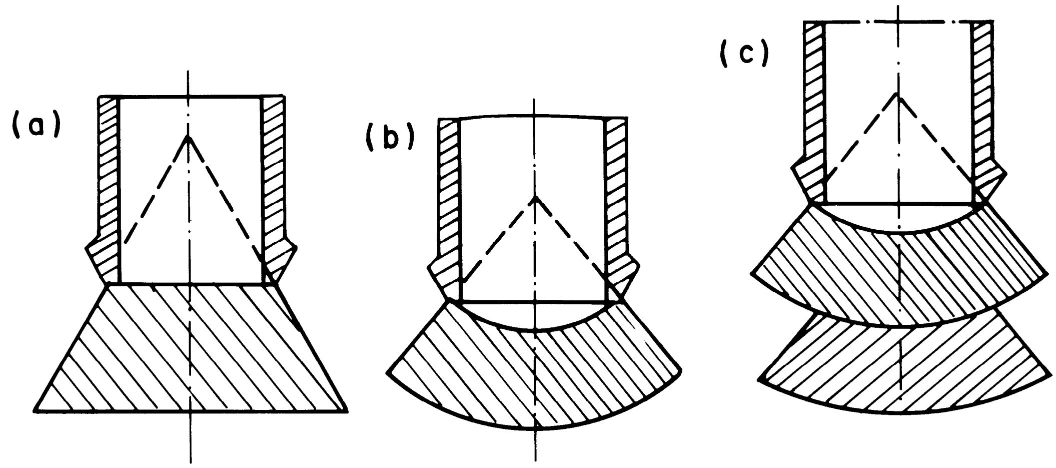


Fig. 17.4.45. Shaft grouting plugs.
a) flat
b) spherical, one stage
c) spherical, double stage.

3. A pair of holes lying on opposite sides are drilled and grouted; then the pair lying in the plane perpendicular to the first one is processed, etc., as in Fig. 17.4.44.

This system is the one most often used, being recognized as the best, because of the initial maximum distance between the holes, diminishing the probability of grout penetration from one hole to another.

The entrances of the grout holes are equipped with entrance pipes with diameters of 2 to 3 in. (50 to 75 mm) and lengths of 4.5 to 6 ft (1.5 to 2.0 m).

GROUTING PLUG. Grouting plugs are put in the shaft bottom to prevent water and grout inflow into the shaft. Entrance pipes to the holes are secured firmly by grouting them in the plug. The plug can be either natural or artificial.

Natural Plug—The natural plug is composed of a protective layer of grouted rock 9 to 15 ft (3 to 5 m) thick, depending on the strength of rock and expected pressure (water and grout).

Artificial Plug—An artificial plug is used more often and is constructed of concrete or reinforced concrete. It is constructed on the shaft bottom, 6 to 12 ft (2 to 4 m) from the water-bearing strata. When grouting is completed, the plug is demolished with explosives, and sinking proceeds through the cemented section, but again is suspended 6 to 12 ft (2 to 4 m) above the end of the grouted zone. The shaft lining is constructed and the procedure with the plug is repeated until the shaft passes the whole water-bearing zone.

During sinking of the shaft through the grouted zone, test drillholes are required to monitor hazardous water inflow. Artificial plugs can be flat, spherical, or multistage (Fig. 17.4.45). Appropriate design calculations are required.

CONSTRUCTION METHODS FOR ARTIFICIAL PLUGS. The construction method depends on the volume of water inflow to the shaft. When an inflow is small enough that it can be controlled by pumps, a wooden chest with a volume of about 35 ft^3 (1 m^3) is set on the shaft bottom (Fig. 17.4.46), with holes about 1.2 in. (30 mm) in diameter allowing water to enter. In the chest, a vertical drainage pipe is installed with a diameter larger than the suction pipe of the pump, and about 3 ft (1 m) longer than the thickness of the plug. On the shaft bottom, a layer of gravel 2 to 3 ft (0.6 to 1.0 m) thick is placed for water filtration. Through the drainage pipe, the suction pipe of the pump is lowered so that during the plug construction water inflow is controlled.

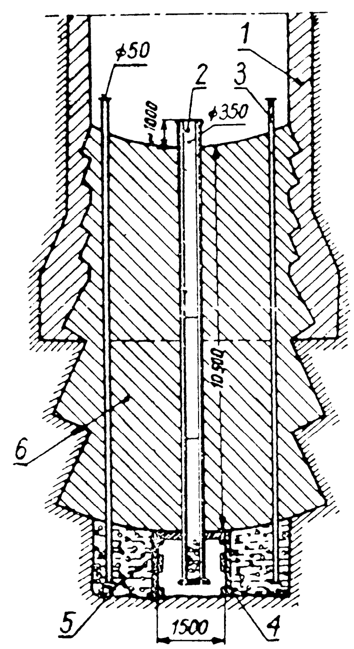


Fig. 17.4.46. Construction method with grouting plug.
1. shaft lining
2. drainage pipe
3. grouting pipes
4. wooden chest with holes
5. gravel or crushed stones
6. double-stage concrete plug.
Dimensions in mm. Conversion factor: 1 in. = 25.4 mm.

When the plug has hardened, a gravel layer is grouted through pipes previously installed within the plug. Then the grouting holes are drilled through the entrance pipes already installed in the upper part of the plug (during placing of concrete).

When the water inflow is too great to control with pumps, an underwater plug is placed. Concrete is pumped on the shaft

Table 17.4.10. Selection of Grout Density Depending on Water Absorbability of Grouting Hole

Absorbability L/min	Weight Proportions			
	Cement	Sand	Water	Proportion of Cement and Sand to Water
0.0–0.001	1	0	12	1:12
0.001–0.01	1	0	10	1:10
0.01–0.1	1	0	8	1:8
0.1–0.5	1	0	6	1:6
0.5–1.0	1	0	4	1:4
1.0–3.0	1	0.5	3	1:2
3.0–5.0	1	1.0	2	1:1
5.0–10.0	1	2.0	1.5	1:0.5
over 10.0	1	3.0	2.0	1:0.5

Conversion factor: 1 gal = 3.786 L.

bottom through tubing to form layers 3 to 3.5 ft (0.3 to 0.4 m) thick. (It is important to keep the outlet of the rising tubing below the concrete surface.)

SELECTION OF THE CEMENT AND GROUT PARAMETERS. Selection of cement depends on:

1. Character of water (salty or fresh).
2. Mineralization and chemical corrosiveness.
3. Value of the filtration factor.

A proper choice is made on the basis of results obtained from chemical analysis and pumping tests.

DENSITY OF GROUT. The choice of the grout density depends on water absorbability measured in the hole. It may be determined by pumping clear water for a 20-min period to the grouting hole with a head exceeding 45 to 75 psi (300 to 500 kPa) hydrostatic pressure.

Water absorbability in L/min can be determined by (in SI units),

$$q = \frac{Q}{Hh} \tag{17.4.22}$$

where Q is water volume being pumped in L/min, H is water head in m, and h is vertical height in m of the hole section where water absorbability is measured. Selection of grout density vs. water absorbability is shown in Table 17.4.10.

In the presence of salty water, this same salt concentration in water is used to prepare grout. The grout density depends on the character of the discontinuities in the rock mass and the applied grout pressure. This density changes over wide limits ranging from 3 to 50%. When larger cavities are encountered, then a thick grout (with sand fillers) is first pumped in, followed by a thinner mixture consisting of water-cement only. Similarly, for larger fissures, the recommendation is first to inject dense grout (25 to 30%), while for rock with small cracks, thin grout (3 to 5%) is preferable. The density of grout must be changed to match the absorbability of the hole.

In rocks with very small fissures where grout cannot penetrate well even under high head, special chemicals are used to decrease friction. To increase absorbability, small explosive charges sometimes are used in the holes.

To decrease absorbability and lessen the cost of the grouting operation, clay-cement grout is used with the cement-clay proportions of 1:2, 1:3, and 1:4, and solutions of cement-clay-sand with corresponding proportions of 1:1:1.5 or 1:1:2, respectively. By using clay-cement grout, cement is spared without sacrificing tightness of the rock mass.

It is advisable before planning a grouting operation to carry out a study and laboratory examination of rock samples to determine an optimum grouting regimen.

The hardening time of cement should be selected to allow grout preparation and injection to the desired radius of penetration. From an engineering standpoint, only the volume of injected grout is justified that seals the rock to meet minimum design requirements. This has a special meaning in cases of high rock permeability, when a substantial loss of grout may easily occur, thus raising the cost of shaft sinking. Admixtures speeding the time of setting up the cement are required to limit the range of grout penetration into large fractures, especially in the presence of water flow.

EQUIPMENT USED IN GROUTING OPERATION. The main components of the grouting setup are:

1. Cementation pumps (two units) for grout injection.
2. Mechanical mixers where grout is prepared.
3. Grout tanks.
4. Flexible pressure hoses.
5. Injectors.
6. Other hardware (pipes, valves, manometers, tools, etc.).

Cementation pumps are able to change head and output over wide ranges. Piston or plunger pumps with a capacity of 18 to 40 in./min (0.2 to 1 m/min) are commonly used at working pressures up to 2200 psi or (15 MPa). These may have electric or compressed-air drive.

When the grouting operation is carried out at substantial depths, the hydrostatic pressure of grout being prepared on the surface can be sufficient. Otherwise, cementation pumps are essential.

GROUT PRESSURE. The applied grout head should be established separately for each section of cementation. When grouting is done from the surface, the maximum pressure p in Pa is given by the formula,

$$p = H\gamma\, k \tag{17.4.23}$$

where H is depth counted from the surface to the top of the strata in m, γ is density of rocks = 1.5 to 2.0 t/m^3, and k is coefficient of cohesion of the strata above the grouting zone = 2 to 3.

When grouting from the shaft bottom, the required pressure has to be found by taking into account the strength of the shaft lining. In practice, a grout pressure is set 150 to 450 psi (1 to 3 MPa) higher than the hydrostatic head. Control calculations to ensure that the lining is not threatened should be done. Also recommended is the observation of the shaft lining during injection. Observation is done in control pipes with plugs installed in the lining, 9 to 15 ft (3 to 5 m) above the shaft bottom. Prior to injection, a tightness test of the installation is done with the pressure 1.5 times the maximum scheduled. Following that, the grout hole is flushed with water to clean out silt and clay until the return water is clean. Then the grouting operation begins. When grouting is terminated, the cutoff valve is closed, and the grouting installation, when not immediately used for another hole, should be flushed with water to prevent the grout from solidifying in the pipes.

After 24 hours, a grout hole can be drilled again. In the case of water outflow, the grouting procedure is repeated. Often, one injection may not be sufficient and one or more additional treatments is required.

CONTROL OF THE PROCESS. Effectiveness of the process is assessed by measuring the water inflow to the shaft after grouting and by examining the rock core obtained from the control diamond drillholes.

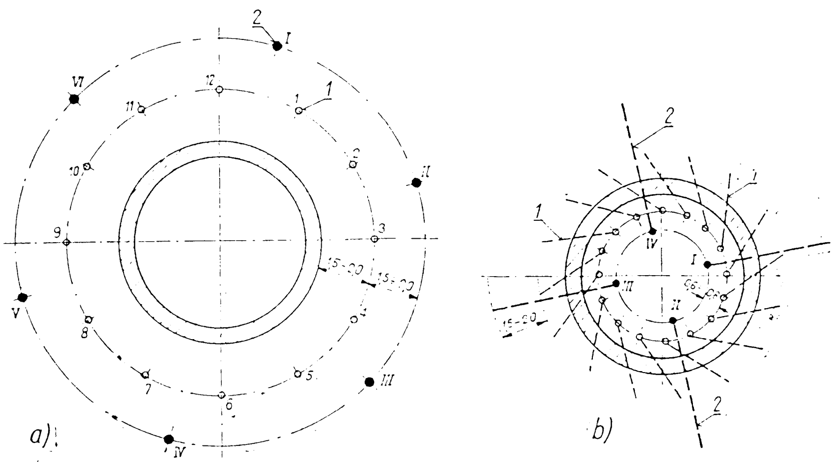


Fig. 17.4.47. Location of control holes.
a) for grouting operation from the surface
b) for grouting from the shaft bottom
1. grouting holes
2. control holes
The depth of the control holes is the same as the depth of the grouting holes. Dimensions in m. Conversion factor: 1 ft = 0.3048 m.

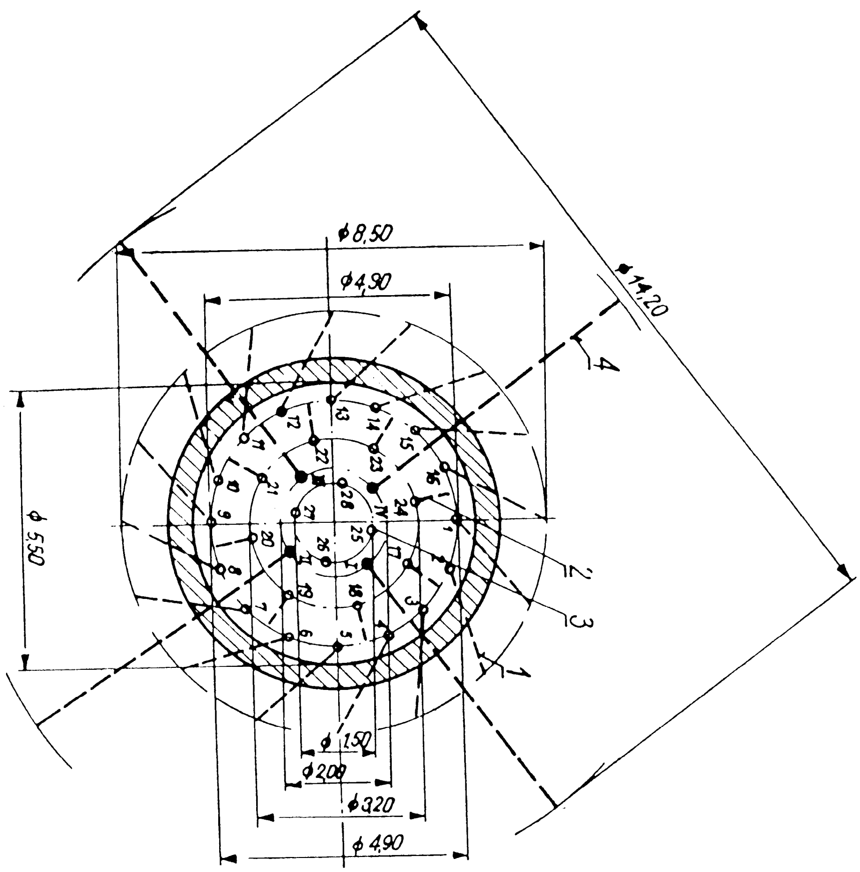


Fig. 17.4.48. Schematic of distribution of grouting and control holes in 18-ft (5.5-m) diameter shaft. Dimensions in m. Conversion factor: 1 ft = 0.3048 m.

The grouting operation is successful when control holes prove that water absorbability within the grouted rocks is less than 0.04 gpm/ft (0.05 L/min/m).

Control holes, where the grouting operation is carried out from the surface, have diameters of 3 to 4 in. (80 to 100 mm) and are located on opposite sides, between grouting holes, in a circle with a diameter of 9 to 12 ft (3 to 4 m) larger than the circle of grouting holes (Fig. 17.4.47a).

When grouting from the bottom of the shaft, control holes are drilled in a circle within the grout holes circle at a distance of 2 to 2.6 ft (0.6 to 0.8 m) (Fig. 17.4.47b).

Shown in Fig. 17.4.48 is an example of the hole plan for grouting carried out from the shaft bottom with a 36-ft (11-m) height for grouting sections.

GROUT CONSUMPTION. Grout consumption per 1 ft (0.3 m) of shaft varies widely and depends on the fissility of the rocks and the grouting system. The approximate mean figures for grout consumption for designing purposes that can be used are as follows: for sandstone, from 1.1 to 5.5 tons/ft (3.3 to 16.5 t/m); for dolomite, from 0.7 to 1.5 tons/ft (2.2 to 4.4 t/m); for chalk, about 1.3 tons/ft (4.0 t/m); and for diabase from 0.2 to 0.3 tons/ft (0.6 to 1.0 t/m).

EXCAVATION OF CEMENTED ZONE. The excavation method in the cemented zone is similar to the conventional method of shaft sinking. However, the depth of the blasting holes should be limited to 3 to 5 ft (1 to 1.5 m). In addition, the size of the charge in a particular hole and the overall charge being blasted should be reduced.

Explosives consumption can be estimated as follows: (1) in sandstone, about 2.5 lb/yd^3 (1.3 kg/m^3); (2) in shale about 2.2 lb/yd^3 (1.1 kg/m^3), and in coal about 1.2 lb/yd^3 (0.6 kg/m^3).

Excavation proceeds only to a certain depth in the grouted zone. When approaching the limit of the zone, an outgoing hole has to be drilled (through a valve) to test for the existence of water hazards.

SHAFT LINING. In spite of the long-lasting effect of properly done grouting, there is always a possibility of some water penetration in the shaft. Therefore, the permanent lining for the grouted zone should be watertight.

SHAFT SINKING PERFORMANCE AND COSTS. For a grouting operation, an average of 35 to 50% of the overall time is required for sinking within the water-bearing strata being grouted. The mean monthly advance is in a range of 25 to 50 ft (8 to 15 m)/ month. An approximate cost of sinking in water-bearing strata is about 30 to 50% more than the cost of sinking in normal conditions.

COMPLEMENTARY GROUTING. The complementary grouting of a shaft lining in the void between the lining and rocks, as well as in the rock itself in the vicinity of the shaft, is always done after shaft sinking in difficult hydrogeologic conditions, and especially when the freezing method has been used. In this method, a consolidation of the rocks disappears with thawing of the water and with a substantial change in stresses due to volumetric changes that often cause new cracks in the lining.

Complementary grouting is beneficial also after sinking with the normal method when tightening of lining, filling voids, or strengthening of rocks is required for better service conditions at the shaft. It is also used to improve conditions in existing shafts where, due to multiple causes (corrosion, vibration from shaft vessels, subsidence, etc.), the lining tightness has deteriorated.

A longer section of the shaft to be grouted is divided into smaller sections, with heights of 50 to 75 ft (15 to 25 m). These sections are grouted downwards, but in each of them, grouting goes from bottom to top. In every section on the circles 3 to 6 ft (1 to 2 m) apart and in a vertical direction, grouting holes are drilled with a 1.2- to 2-in. (30- to 50-mm) diameter and with depths depending on need from 2 to 6 ft (0.7 to 2.0 m), locating the holes in a checkerboard pattern. Holes in the circle are 3 to 6 ft (1 to 2 m) apart.

Grout pressure should exceed the hydrostatic water head but should not exceed the critical pressure for the lining. Assuming short-term loading, the safety factor for the lining can be lowered to 2 or 1.5.

If the first grouting operation is unsuccessful, it is repeated. Tubbing segments have special holes with plugs for grouting purposes. Grouting in shafts is usually conducted from a hanging platform.

The grouting operation is considered satisfactory when the water inflow in the shaft does not exceed 6.5 gpm (25 L/ min).

FREEZING METHOD. A suitable design procedure requires solution of the following problems:

1. Calculation of the wall thickness of the frozen rock cylinder.

2. Calculation of the diameter of the circle where freezing holes are located, their number, depth and spacing, and the number and distribution of surveying and control holes.

3. Selection of the drilling system for freezing holes, specification of equipment, choice of freezing pipes and their diameter, and assessment of time required for drilling.

4. Design of a temporary shaft head with a freezing basement.

5. Planning of the freezing procedure in active and passive periods, with specifications of freezing temperature, heat balance, required efficiency of freezing installation, and freezing time.

6. Determination of the system and time of thawing and liquidation of freezing holes.

7. Initial planning of freezing technology, shaft sinking and lining, and complementary grouting.

Calculation of Thickness of Frozen Rock Wall—An artificially created cylinder of frozen rock around the shaft creates a kind of lining, shielding the inner space against water and unstable rock. Thickness of the frozen rock cylinder e_c in m, for shallow and moderate depths is calculated for external pressure p in MPa, from the Lamé formula (in SI units):

$$e_c = r\left(\sqrt{\frac{k_c}{k_c - 2p}} - 1\right) \qquad (17.4.24)$$

where r is shaft radius of the excavation in m, and k_c is allowable stress of the frozen rock in MPa. The allowable stress of the rock is 50% of its compressive strength in the mean range of freezing temperatures.

In well-defined and favorable conditions, the allowable stress can be increased to 75% of the rock compressive strength. Because of the creep characteristics of ice, the compressive strength of frozen rocks is a time-dependent value.

For calculation of the thickness of the cylinder of frozen rock, the mean temperature of the rocks should be taken. The temperature within the wall of frozen rock changes, from a minimum value just beside the freezing hole to a maximum (around 32°F or 0°C) on the outer wall limit.

The outside pressure p is calculated according to guidelines given in 17.4.1.3.

The cylinder of frozen rock, when properly created, permits sinking a shaft of moderate depth even in long sections using temporary lining. This is desirable because there are fewer seams between lining sections. The concrete lining length made in one pass changes from several feet (meters) to about 60 to 75 ft (20 to 25 m), depending on the strength of the frozen rock mass, overall depth, and effective temperature (Fig. 17.4.49).

The thickness of the cylinder of frozen rock at greater depths within the plastic zone is calculated from the Domke formula:

$$\frac{e_c}{r} = 0.29\,\frac{P}{k_c} + 2.3\,\frac{P}{k_c} \qquad (17.4.25)$$

The other condition that is to be fulfilled is $p < P < k_c$, where $p = H\gamma_m$ in MPa is the vertical pressure (dead load) of the cylinder of frozen rock at depth H, with mean density γ_m, and with allowable compressive stress k_c. When the above conditions are met, it means that e_c has been properly chosen.

For greater freezing depths, another design concept of frozen rock thickness is recommended. It is usually impossible to arbitrarily change the freezing temperatures because of multiple limitations. At this pressure level, frozen rock creeps according to the Maxwell or Bingham rheological model. *R'c is* adequately evaluated due to the exposure time of the unsupported shaft wall, that is, the interval between excavation and installation of lining.

Lieberman (1962) has proposed the following formula for determination of frozen wall thickness e_c in these conditions (in SI units) (Fig. 17.4.50):

$$e_{c_{max}} = r\left(\frac{\gamma H}{e^{R'c}} - 1\right)s \qquad (17.4.26)$$

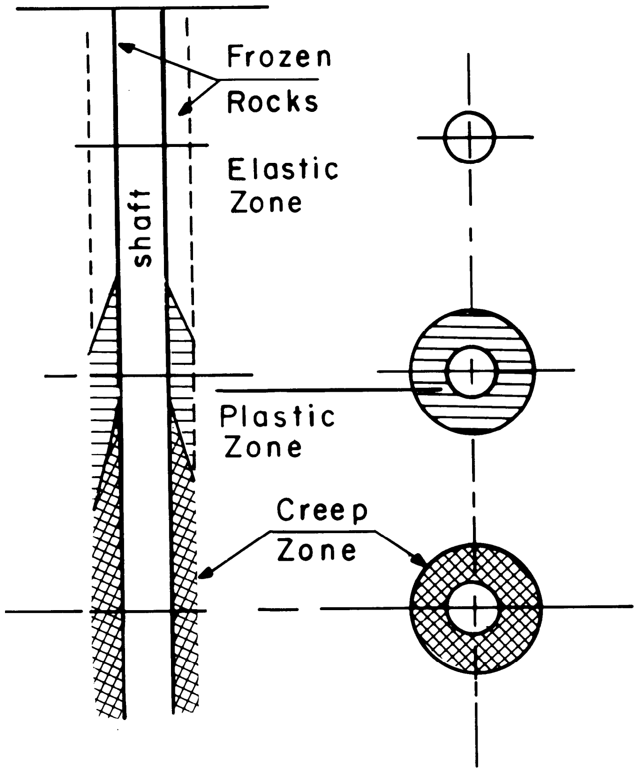


Fig. 17.4.49. Idealized schematic of state of stresses in cylinder of frozen rocks with growing depth.

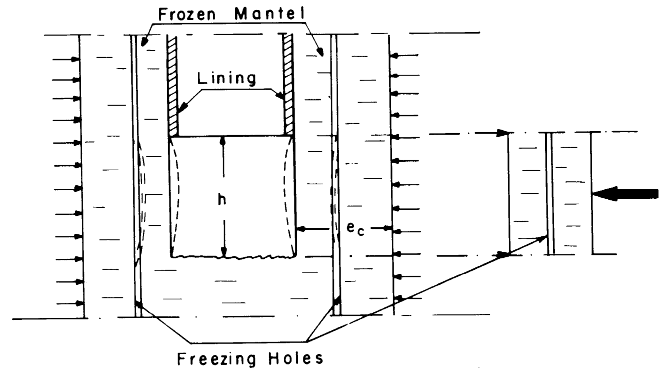


Fig. 17.4.50. Sketch of creep (bending) on frozen cylinder wall at great depth.

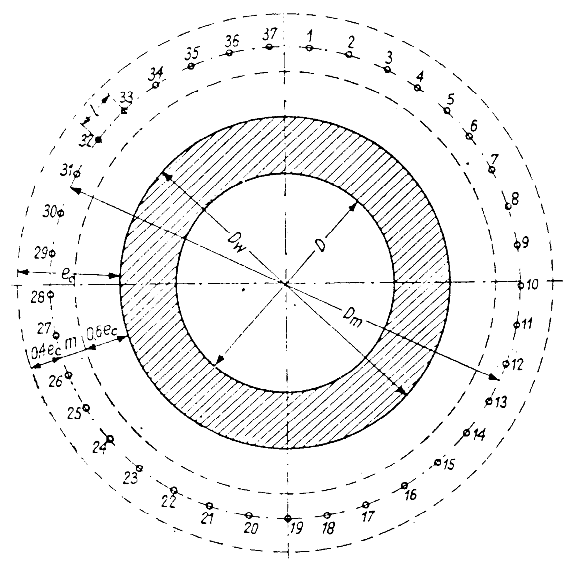


Fig. 17.4.51. Arrangement of freezing holes around the shaft.

Table 17.4.11. Creep Characteristics of Frozen Soil Materials

Type of Material	Compressive Strength R_c, MPa at Varying Temperature 14°F (−10°C)	5°F (−15°C)	−13°F (−25°C)
Clean ice	1.2	1.8	2.1
Clay, silt with organic matter	—	3.0–6.0	—
Clay, silty fine sand	5.5	7.2	3.5
Fine and medium-grained sand saturated with water	8.7	13.3	15.3
Quartz sand	12.0	15.0	20.0

Conversion factor: 1 MPa = 142.2 psi.

where h is height of excavated section in m, s is safety factor = 1.1 to 1.2, e is the natural logarithm = 2.718. Find h in m from the expression,

$$h = e_c \frac{R'_c}{\gamma H} \tag{17.4.27}$$

The author recommends the following procedure in applying the preceding formulas: Because of technological and economic limits, a practical thickness of frozen cylinder is assumed to be 15 to 17 ft (5 to 5.5 m). Then the allowable height of excavation h is calculated. When the assumed thickness $e_c > e_{max}$ from the above, then there is no restriction on the height of excavation. In these formulas, H is measured to the shaft bottom.

FREEZING HOLES, CONTROL AND TESTING HOLES: DIAMETER OF THE CIRCLE OF FREEZING HOLES. Centers of holes are placed on a circle with diameter D_m in ft (m) (Fig. 17.4.51), calculated as follows:

$$D_m = D_i + 2(0.6\, e_c + m) \tag{17.4.28}$$

where D_i is diameter of the excavated shaft in place with thickest lining in m, e_c is thickness of frozen cylinder in m, and m is a correction taking into account the vertical deviation of the freezing holes in m. The factor 0.6 represents nonsymmetrical (in a horizontal, radial direction) growth of the frost cylinder, 0.6 in the direction toward the center of shaft, and 0.4 toward the outside boundary where heat flow losses are greater. In practice, the diameter D_m is larger, at least 6 ft (2 m) more than the shaft diameter in excavations with freezing depths to 250 ft (80 m). For a depth of 450 ft (150 m), D_m is at least 8 ft (2.5 m) greater, and below 450 ft (150 m), no less than 10 ft (3 m).

Some well-trained crews are able to drill freezing holes at a depth of 1500 ft (500 m) within a cylinder with a 3-ft (l-m) radius.

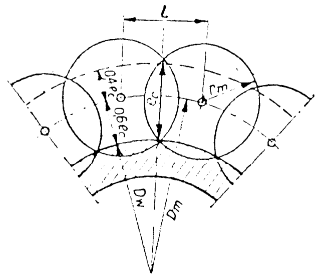


Fig. 17.4.52. Cylinders of frozen ground around the freezing holes.

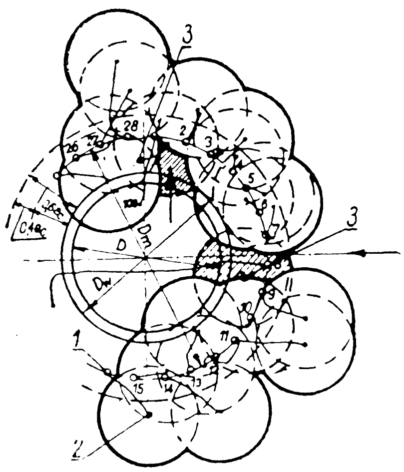


Fig. 17.4.53. Map of frozen rock cylinder with freezing hole locations and their deviations on the control horizon.
1. inlets of freezing holes
2. bottoms of freezing holes on control level
3. additional holes.

DISTANCE BETWEEN FREEZING HOLES. The spacing of freezing holes is taken from practical experience and varies between 35 and 44 in. (0.9 and 1.1 m); however, for deep freezing with turbulent flow of brine, spacing can be increased to 50 in. (1.25 m) (Fig. 17.4.52).

Basically, hole spacing should be arranged such that the chord of overlapping circles will be equal to the calculated thickness of the cylinder of frozen rock e_c.

The diameter of the frozen column around the freezing holes at a temperature of –1° F (–15°C) is assumed: in sandy rocks, 9 ft (3 m); in clay and silt rocks, 7.5 ft (2.5 m); and in rocks with water flow and large coefficient of filtration, 6 ft (2 m). The number of freezing holes is calculated from $n = p\,D_m/l$ (Fig. 17.4.52).

DEPTH OF FREEZING HOLES. When the freezing depth does not exceed 180 ft (55 m), the bottom of the hole should reach 6 to 15 ft (2 to 5 m) below the water-bearing strata. In inclined strata or when the total depth is greater, holes are drilled 15 to 30 ft (5 to 10 m) below it.

ADDITIONAL FREEZING HOLES. Verticality of freezing holes is measured every 150 ft (50 m), and results are plotted on a plan of the whole cylinder of frozen rock (Fig. 17.4.53).

If deviation of holes causes the cylinder of frozen rock not to close (assuming radius r_m of frost around the hole), drilling of an additional hole or holes is required. Emphasis should be put on accuracy of drilling to keep to a minimum the number of required additional holes.

CONTROL HOLES. To provide information about the freezing process, additional control holes are drilled along a radial line drawn in the middle between two freezing holes. Control holes are drilled 1.8, 3, and 10 ft (0.5, 0.9, and 3.0 m) from the freezing-hole circle toward the outside and 1.3 and 4.3 ft (0.4 and 1.3 m) toward the shaft center. These holes are filled with brine and are used for temperature measurements on different levels. They are cased in steel pipes with welded bottoms.

A water-level observation hole is drilled inside the frozen cylinder, usually in the center of the shaft. It has a casing without a bottom, and at levels corresponding with water tables, it has perforations with filter screens. Observations of changes in the water table are taken in that hole. It also plays an important role in stress relief, allowing water entrapped within the frozen cylinder to flow up into the hole. Otherwise, volumetric changes during the freezing process would increase the pressure.

DRILLING OF FREEZING HOLES. The most important problem during the drilling of freezing holes is to assure verticality of holes. Contemporary, modern rotary drilling rigs, when operated by an experienced crew, can maintain verticality within acceptable tolerances. Freezing holes are drilled with mud fluid of specific gravity 1.15 to 1.25. To shorten the time required for drilling of many (sometimes 40) holes, two or three rigs are used at a time.

FREEZING HOLES EQUIPMENT. Every hole is equipped with two columns of tubing.

1. Outside: the freezing pipe itself has a diameter of 4 to 6 in. (102 to 152 mm) and is constructed of steel with inside connectors so it is smooth on the outside and has a welded bottom.

2. Inside: The inner tubing has a smaller diameter of 1.25 to 2 in. (32 to 51 mm), is made of steel or polyethylene (PE) pipe, and has inside diameter and wall thickness of 3 in. and 2.25 in. (75 by 43 mm), respectively. Polyethylene pipes are used because they have several advantages over steel. They are cheaper and easier to install (by unwinding from a reel), have lower thermal conductivity, and the specific gravity of the PE is close to the specific gravity of the brine, resulting in less suspended weight (Fig. 17.4.54). Freezing brine flows at high velocity down the inner pipe. It rises more slowly in the annulus, absorbing heat from the surrounding rocks.

There are two distinct flow regimes: laminar and turbulent. For great depths and more difficult freezing conditions associated with the presence of clay rocks and organic matter (e.g., lignite), turbulent flow is recommended. It assures a more uniform heat flow from the rock to the freezing hole, making it less dependent on depth.

Laminar flow, which is adequate for moderate depths, exhibits a distinct difference in velocity of brine flowing down (2 to 4.5 fps, or 0.6 to 1.5 m/s) and upward (0.25 to 0.67 fps, or 0.08 to 0.2 m/s). Brine circulation initially cools the rocks and later freezes them, creating rigid columns 6 to 9 ft (2 to 3 m) in diameter. There are large differences in the thermal conductivity of rocks, resulting in differences in freezing time, varying from two to eight months. For greater freezing depths, larger-diameter freezing pipes are required. The ratio of diameters between the outer and inner pipes has to be chosen for particular freezing depths and estimated heat transfer rates. This keeps the cost of energy (freezing compressors, pumps) to a minimum and also ensures an installation capable of freezing the rock in the scheduled time interval.

Complete tightness of joints among individual segments of freezing pipes is necessary. Any brine leak will thaw the rock,

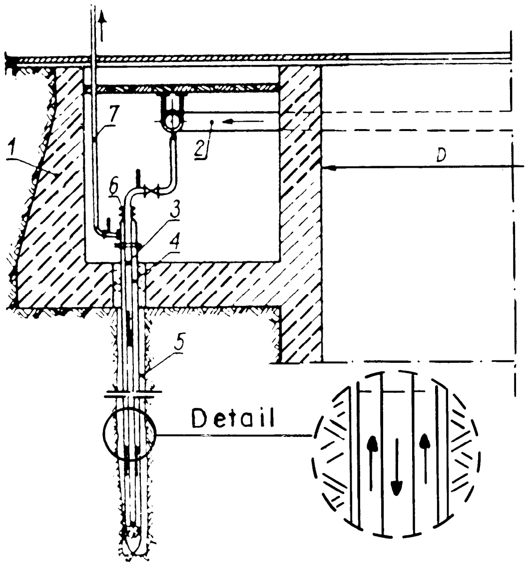


Fig. 17.4.54. Freezing brine circulation in freezing hole.
1. freezing cellar lining
2. distribution ring
3. entrance tubing
4. freezing pipe
5. freezing hole
6. tee
7. outflow pipe to inflow brine tank.

and in extreme cases, can cause catastrophic inflow of outside water and flow of rock to the shaft. It is also impossible to freeze rocks contaminated with brine without substantial lowering of the temperature, which in turn is limited by the capacity of the particular freezing station. Therefore, during installation of freezing pipes, every column 150 ft (50 m) or longer is hydrostatitally tested.

When the whole column is lowered to its final position, casing pipes are removed. The space between the wall of the hole and the freezing pipe is then filled with dense drilling mud.

SECTIONAL FREEZING. Sometimes water-bearing strata underlie strata of competent rocks. In this case, freezing is required only at particular depths; however, double inner-pipe installation is necessary. A brine circuit is designed to enable control of a return flow from particular holes (visually), and the flow-alarm gage in the tank gives a warning signal when the brine level goes down, indicating a break of the freezing pipe and escape of brine into the rock mass. Such a hole has to be immediately shut off (valve on inflow line).

FREEZING CELLAR. The collars of freezing holes with pipes connecting them with the distributing ring, shut-off valves, and thermometers are installed all together in a so-called freezing cellar.

A freezing cellar is designated either as (1) *temporary,* to speed up construction time (it will be demolished when the permanent shaft headframe is installed); or (2) *permanent,* considered a part of the permanent shaft headframe, requiring a longer construction time. An example of freezing cellar design is given in Fig. 17.4.55.

FREEZING OF ROCK MASS. Freezing media used as refrigerants are

1. Ammonia (NH,): assures a freezing temperature of –4°F (–20°C) with one stage and –40°F (–40°C) with two stages of compression, and is used in most cases.

2. Carbon dioxide (CO_2): provides a freezing temperature of –58°F (–50°C).

A schematic of an ammonia freezing station is shown in Fig. 17.4.56. It is a one-compression-stage system, and the freezing temperature is around –4°F (–20°C). The ammonia system is closed. Compression raises pressure to 150 psi (1 MPa), with a simultaneous increase of temperature to 167°F (75°C). Through the oil separator, the compressed ammonia goes to the sprinkle condenser, where it is cooled down with water (about 54°F, or 12°C) to a temperature of around 68°F (20°C). At this temperature, gaseous ammonia changes to a liquid, and, passing an expansion valve, the pressure (150 psi or 1 MPa) is reduced to 7 to 20 psi (50 to 140 kPa), expanding and boiling. Because of this rapid evaporation, the temperature drops to –4 to –13° (–20 to –25°C). Vapors of ammonia travel through the coil pipe within the evaporator filled with brine, which cools the brine. Then the ammonia returns to the intake of the compressor and the cycle starts again.

BRINE CIRCUIT. A water solution of calcium chloride ($CaCl_2$) is used almost exclusively for temperatures not lower than about –49 to –58°F (–45 to –50°C). For lower temperatures, lithium chloride solutions can be used (e.g., the Yarbo shaft, Saskatchewan, Canada). Most often used is an aqueous solution of $CaCl_2$, with a specific gravity of 1.25 to 1.26, and a freezing point of –29 to –36°F (–34 to –38°C). The brine circuit begins with the outflow brine being heated up (about 5°F or 3°C) in a rock mass and returning (through a collector ring or an outflow tank on the shaft gear and main pipeline) back to the heat exchanger, where it is cooled down. Here the cycle starts again.

FREEZING PERIODS. Two main periods of freezing can be distinguished, the active period and the passive or maintaining period.

During the active period, a frozen mantel of design thickness is created around the future shaft. To achieve the designed capacity of the freezing station, two to three aggregates are installed together to provide the required heat efficiency. During an active period, all units work at full capacity. The passive period starts after the frozen mantel is created and lasts through the sinking, lining, and assembling of the shaft installations. During the passive period, usually only one unit works while the others stand by as a reserve.

HEAT BALANCE CALCULATIONS. The volume of frozen rock V has to be found. It is calculated as a sum of the volumes of the cylinders in particular strata of different physical properties.

The volume of each stratum is the sum of the volume of rock material (solid) and the water volume remaining in rock pores.

The water content of the rock depends on the rock porosity and in competent rock also on the fracture and pore volumes. The required data are obtained from rock tests.

The amount of heat Q_o in Btu (kcal), which is to be extracted to cool down the rock mass from the initial temperature t_i to the mean freezing temperature t_m, is

$$Q_o = Q_1 + Q_2 + Q_3 + Q_4 \tag{17.4.29}$$

where Q_1 is heat in kcal to be extracted from the rock mass to lower the initial temperature of the solids to the mean freezing temperature, Q_2 is heat in kcal to be extracted to cool down the water in pores from initial temperature to temperature 32°F (0°C), Q_3 is heat in kcal to be extracted to change water at temperature 32°F (0°C) into ice at constant temperature, and Q_4 is heat in kcal to be extracted to cool down ice at initial temperature 32°F (0°C) to the final freezing temperature. The mean initial rock and water temperature is approximately 50 to 54°F (10 to 12°C).

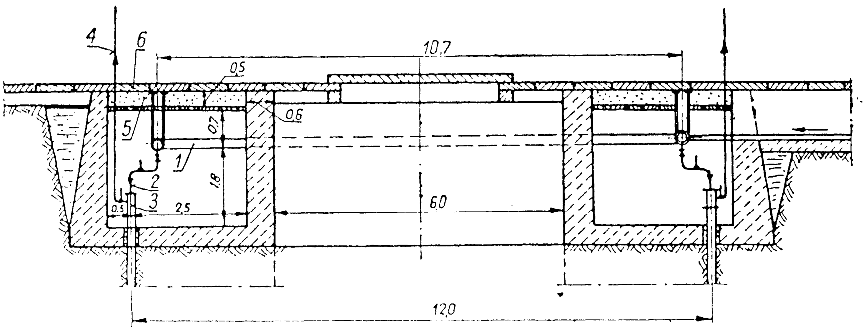


Fig. 17.4.55. Freezing cellar,
1. distributing ring
2. inflow pipes (supply cold brine)
3. freezing pipes
4. pipes with returning warmed up brine
5. beam floor with thermal insulation
6. surface beam floor.

Dimensions in m. Conversion factor: 1 ft = 0.3048 m.

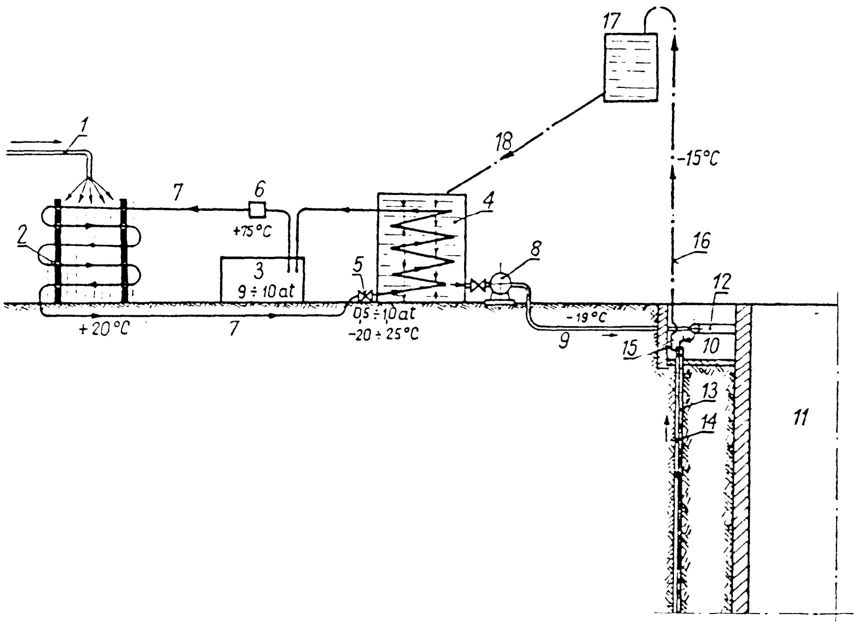


Fig. 17.4.56. Schematic of freezing station for shaft sinking with ammonia as freezing agent.
1. water line
2. sprinkle condensor
3. compressor
4. evaporator
5. reduction valve
6. oil separator
7. ammonia line
8. brine pump
9. brine pipeline
10. freezing cellar
11. shaft
12. distribution ring
13. inflow pipe
14. freezing pipe
15. thermometers
16. outflow pipe of warmed up brine
17. outflow tank on headgear
18. outflow pipeline of warmed up brine.

The above heat components are calculated by applying calorimetric formulas that take into account the dimensions of particular rock cylinders with their corresponding content of solids and pore water, the specific gravity of those components, heat capacity of rock and water, latent heat of change of water into ice at 32°F (0°C), heat and density of ice, and the temperature difference achieved by freezing. The required net capacity of the freezing station can be calculated, as well as the heat loss in the rock mass and in the installation. The size of the freezing station is based on heat exchange during the active freezing period.

HEAT LOSSES. There are two kinds of losses in the freezing process: (1) loss in the rock mass as a result of the steady flow of heat from outside the frozen mantel, and (2) losses in the freezing installation, because of imperfect insulation. The gross capacity of the freezing station is equal to its net capacity minus heat losses in the rock mass, plus heat losses in the equipment.

TIME OF ACTIVE FREEZING. The time of active freezing of a rock mass depends on the rate of frost penetration radiating outward from the hole to reach a radius r_m, assuring a sufficient thickness e_c of frozen mantel, measured midway between the two holes.

The frost penetration velocity for sands is about 1 in./day (25 mm/day) and for clays about 0.2 to 0.25 in. (10 to 15 mm/day) (Pokrowskij, 1962).

The freezing velocity in practical terms for a given set of rock mass parameters and hydrologic conditions depends mainly on the net capacity of the freezing station.

From actual records, the time of active freezing is about 2 to 4 months for a freezing depth as great as 750 ft (250 m), and about 9 to 10 months when the freezing depth is around 1500 ft (500 m).

A nomograph (Fig. 17.4.57) allows the determination of the active freezing time for a frozen mantel with a required thickness, depending on brine temperature, spacing of freezing holes, initial temperature, and coefficient of thermal conductivity. The nomograph shows an example of water-saturated sand with a thermal conductivity of 0.13 in./hr (3 mm/h), with an initial temperature of 46°F (8°C) with a hole spacing of 4 ft (1.2 m) and brine temperature of –13° (–25°C) for a required thickness of frozen mantel 9 ft (3 m).

PASSIVE PERIOD OF FREEZING. During this period, the already frozen mantel of the rock mass should be maintained at the designed freezing temperature during the entire period of sinking and lining of the shaft. The required capacity of the freezing station during this period is smaller, just sufficient to balance heat losses. In practice, during the passive period, one-half to one-third of the initial capacity is sufficient.

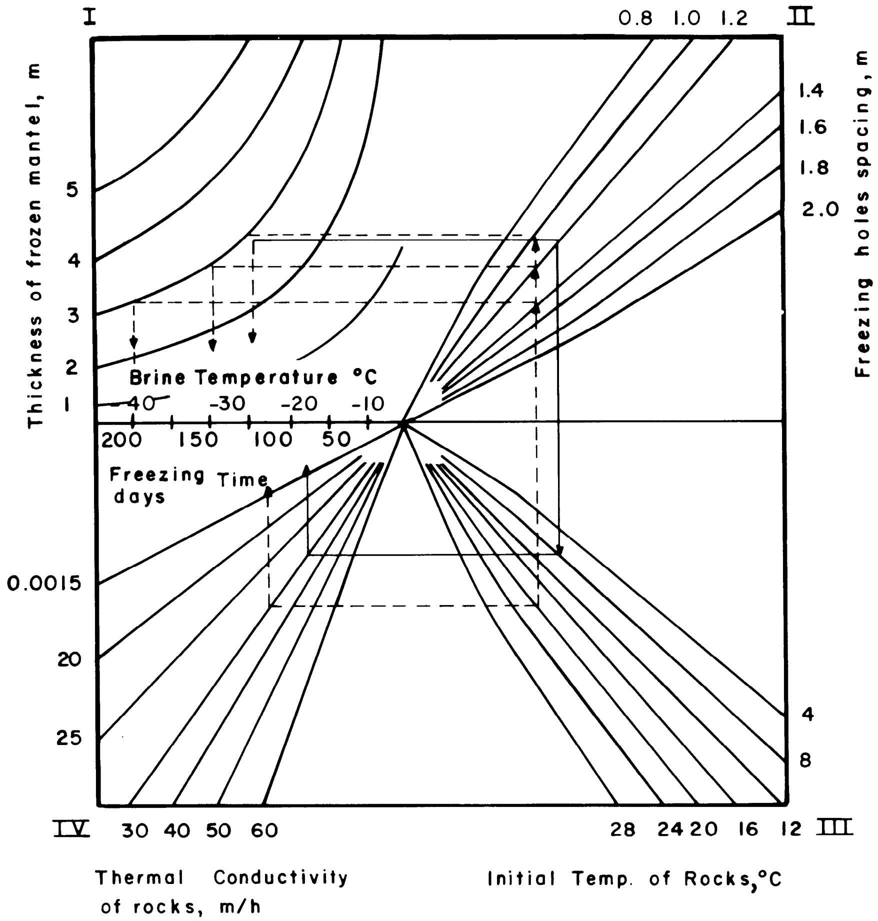


Fig. 17.4.57. Nomograph for determination of time required to create a frozen mantel with given thickness, depending on brine temperature, hole spacing, natural rock temperature, and thermal conductivity of rocks. Conversion factors: 1 ft = 0.3048 m, °F = 1.8°C + 32°.

Computer numerical modeling methods are applicable for heat transfer calculations and are often used in design of shafts sunk with application of the freezing method.

EXCAVATION OF FROZEN GROUND. Methods of excavation are similar to those under normal conditions; however, when explosives are used, special care is required because of the proximity of the freezing holes and the potential danger of their destruction. When the core of a shaft is not frozen, a cactus grab or other mechanical mucker can dig the shaft bottom. When the rock is partly frozen or too hard to be excavated, hand-held drill hammers with chisel picks are used; however, these working conditions are very unhealthy. In hard frozen rock, the drill and blast method is used with limitations as follows: the minimum distance between the freezing holes and the blastholes should be 5 ft (1.5 m), and the depth of the blastholes should not exceed 3 to 6 ft (1 to 1.7 m) in shafts of large diameter having a density of 0.5 to 1.5 holes/10 ft^2 (/1 m^2).

Explosives consumption is estimated as follows: (1) in alluvial overburden rocks, 0.5 to 1 lb/yd^3 (0.25 to 0.5 kg/m^3); and (2) in sandstones, sandy shales, and limestones, 2 to 3 lb/ yd^3 (1 to 1.4 kg/m^3).

The newest excavation system, which avoids exposing workers to unhealthy conditions at the face and permits good performance without disturbing freezing holes, was first developed in Poland (Lubin copper basin) in the early 1970s. A specially designed shaft miner was built from coal shearer parts. This system proved itself in several shafts. A system developed later utilized a roadheader cutting system in a similar arrangement.

17.4.2.4 Extension of Shafts by Raising

The use of conventional raising methods to drive vertical openings has decreased; however, under some conditions, especially for smaller mines, it remains the most attractive solution.

Development of raise boring machines and raise climbers has limited the application of conventional raising to short raises. This method of shaft extension is possible only when the lower level has been developed from another shaft or slope. Raising is recommended when there is no possibility of sinking due to a lack of free space in the existing shaft and a lack of room on the production level for production purposes.

Hydrogeologic and geomechanic conditions, however, limit the applicability of the raising method. It can be utilized only in competent rocks of high and medium strength, without fractures, and without a large water inflow.

The advantages of raising include:

1. Raising is independent of mining production so its influence on mine performance is minimal.
2. Loading of muck into buckets is avoided.
3. Blasting effectiveness is greater.
4. No water pumping is required.

The disadvantages of raising include:

1. The height of raising is limited to about 300 to 360 ft (100 to 120 m).
2. Ladder climbing is an inconvenience for personnel and the transport of materials to the face.

3. There is a potential hazard of pieces of rock falling from the face.

4. There is a potential threat of muck jamming in the chute compartment.

5. Surveying is difficult (deviation is possible).

6. There is time and cost involved in preparing the opening for raising purposes.

A raise must have three compartments—chute, hoist, and ladder—and must have a brattice installed from the bottom of the ladder compartment up to the safety platform placed below the working platform. The chute compartment is of solid construction and is lined tightly with thick 2-in. (50-mm) planks (lagging) to withstand the high-side pressure of the muck. For long raises, a rebound niche is made every 100 to 170 ft (30 to 50 m) in a sidewall of the raise.

Working platforms in raises are made with timbers strong enough to withstand the load of piled-on muck and are supported by steel girders or wooden beams with strength adequate to support the weight of muck, equipment, and workers. For carrying steel beams, a safety factor of six is recommended.

Safety platforms are of a lighter construction, lined with 2-in. (50-mm) lagging laid on beams of adequate strength.

There are two entrances with hatches, one in the ladder and one in the hoisting compartment. The distance between the working and safety platforms cannot be less than 5 ft (1.5 m) or more than 10 ft (3 m). The chute compartment should be filled with rock when it has rebound platforms in niches, and only an excess of muck is removed by opening the bottom damper. In hard rock, a raise face can be advanced up to 15 ft (5 m) without lining, and in weaker rocks, proportionally less. When a lining consists of timber sets, every third set has to be inserted in sockets in the shaft wall. When full timber lining is used, curb sets support at least each platform of the ladder compartment. The chute compartment is guarded by a railing to prevent accidental falls of personnel and larger pieces of rock. For ventilation, a hole 8 to 12 in. (200 to 300 mm) in diameter is drilled in the center of the future shaft. It is cased within the limit of the sump of the existing shaft and projects above the water level in a sump.

Three raising methods are used, depending on the size of the shaft and the geomechanical rock properties. They are as follows:

1. Raising shafts with small diameters up to 12 ft (4 m) is done full diameter with a temporary lining. A permanent lining is erected later. Good rock conditions are required (Fig. 17.4.58).

2. In competent rock, with medium strength, regular-sized and small-diameter shafts are raised full diameter, but in short cuts with immediate construction of a permanent lining.

3. Shafts of diameter greater than 12 ft (4 m) in competent rock of high and medium strength are first raised upward at reduced diameter; then the small shaft is widened downward to the required size. This is the most common method (Fig. 17.4.59).

MECHANICAL RAISE CLIMBERS. Mechanical raise climbers have been developed primarily for application in hard-rock mining. The most popular of this kind is the Swedish Alimak raise climber. The machine runs on a guide rail that incorporates a pin rack. The segmented guide rail is fastened to the side of the excavation with rock bolts. Figure 17.4.60 shows the different stages of system operation. The Alimak method has the following advantages over conventional systems:

1. Permits the driving of very long raises, vertical or inclined, straight or curved.

2. When traveling to the head of the raise, personnel are well protected in a cage under the platform.

3. The miners work from a platform that is easily adjusted for convenient height and angle.

4. Risks involving gases are reduced, as the raise is ventilated after blasting with an air-water mixture supplied in the raise climber.

5. All equipment and material required can be taken to the head of the raise in the raise climber.

6. Timbering is avoided, and if needed, rock bolts, with or without a screen, are used for support.

7. The raise climber can be used for areas up to 85 ft^2 (8 m^2).

The disadvantage is the cost of the raise-climber system, which cannot be justified for short raises; therefore, the most frequent application is in mines where driving raises in hard rock is part of the mining method. A new development is the change of pneumatic drive to electric for long raises. Performance data from recent applications can be summarized as follows.

Three ore passes were excavated with dimensions of 7 by 7 ft (2.1 by 2.1 m), with a combined length of 1097 ft (334 m) and a maximum length each of 427 ft (340 m). One was a stope fill/ventilation/service raise with this same cross-sectional area and length of 315 ft (96 m), and one a main ventilation/service raise 7 by 14 ft (2.13 by 4.26 m) and length of 314 ft (96 m). The raise was completed in 11 months for an average advance rate of 160 ft/month (49 m/month). The peak advance was 312 ft/month (95 m/month). Work was performed by a two-worker crew on a 2 shift/day, 6 days/week schedule. Costs in 1989 dollars, including contract labor, operating supplies, and services for the raise excavation, including setup and tearing down of equipment, was \$197/ft (\$646/m) of advance. Muck load out and haulage was an additional \$17/ft (\$56/m) of advance. The excavation rate increased slightly for an advance beyond 150 ft (50 m). These costs do not include the excavation required for Alimak equipment setup and do not include ground support in the raise. Neither do they include mine engineering and surveying, mine overhead, etc.

17.4.2.5 Raise Boring

Raise boring is one of the newest technologies in shafts and raises, maturing through improvements made during the three decades since its initial application. Steady increases in technical parameters of raise boring, such as diameter, length, and advance per unit time, followed research on system principles and improvements in performance of the critical components. Large-diameter holes or shafts can be drilled vertically or inclined. The costs of raise boring have been substantially reduced due to improvement of the cutters, the average life of which has been extended fivefold in the last fifteen years.

Raises are being utilized as a part of multilevel mining systems for orepasses and ventilation, as coal mine ventilation shafts, and also as full-sized shafts, primarily in coal mining.

Excavating rock by a continuous boring system has several advantages over the conventional drill-and-blast methods, especially in vertical excavations such as shafts and raises, where it eliminates the hazards associated with the presence of workers at the face. In industrialized countries, reducing the skilled labor required in conventional shaft sinking and raising is an important factor in cost reduction. The continuous action of raise boring usually assures faster completion of the project than conventional blast and drill methods. Boring does not disturb the surrounding rock, and therefore, most often there is no need for support or, alternatively, only minimum support is required for the finished hole. A perfectly round shape makes the hole inherently stable, and its smooth walls reduce resistance for ventilation as well as for passing ore. Provided conditions are right, the cost advantage over other systems is clearly apparent for long raises and for shafts.

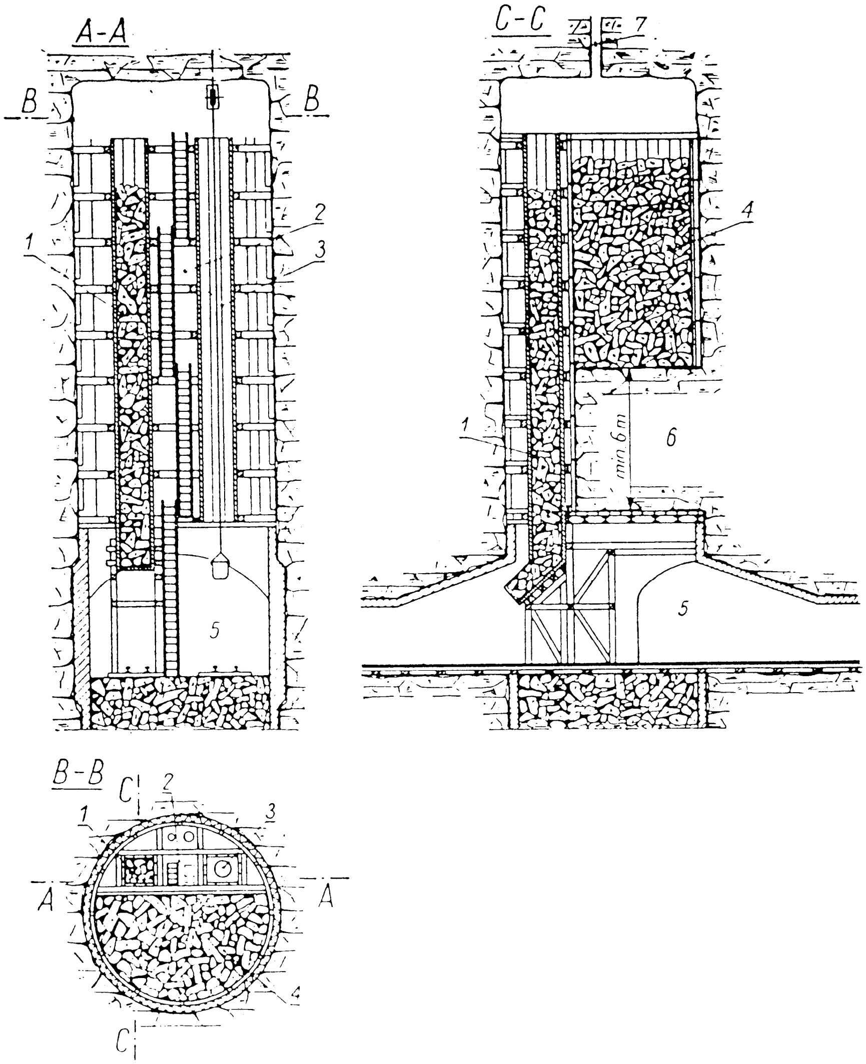


Fig. 17.4.58. Raising method with full diameter of the shaft, with temporary and a later-erected permanent lining.

1. chute compartment
2. ladder compartment
3. bucket compartment
4. storage of muck
5. lower shaft station
6. protective shelf
7. ventilation hole (to sump of existing-upper shaft).

Conversion factor: 1 ft = 0.3048 m.

The disadvantages of a continuous boring system include a general requirement that rock conditions be similar within the length of the raise bore. Another disadvantage is that drilling in very hard rock is slow, and costs rapidly increase. Also access to upper and lower levels is necessary, or in other words, raise boring requires a pre-existing level. A raise-bored rock mass has to be dry. The initial cost of the rig is also high.

EQUIPMENT AND SYSTEM DESCRIPTION. Raise boring equipment is produced in the United States, in other industrialized western countries, and in the USSR. Machines that are designed primarily for soft rock are applied mainly in European coal fields. US manufacturers produce machines for hard- and soft-rock conditions as well.

There is a large variety of machines available that are designed for certain sets of rock conditions and nominal sizes and lengths of the finished hole. Typical characteristics of the machine specify the pilot-hole diameter, drill pipe diameter, reaming torque, type of drive (ac, dc, or hydraulic), its rpm and power (hp or kW), and type of gear reducer with corresponding rpms. Additional specifications are the pilot-hole thrust and feed rates, reaming pull, size of drill pipe, base plate and derrick dimensions with characteristics (especially important for underground application), and finally the type of transporter/erector, if applicable, and the water and air consumption.

Raise boring machines (RBMs) operate on the principle of first drilling a small pilot hole and then reaming the hole, in one or more stages, to the desired size. The modes of operation are shown in Fig. 17.4.61. The most frequently used is a conventional scheme with the pilot hole down and reaming upward. However, in some mining systems, box-hole drilling upwards in one stage is chosen.

The RBM consists of several parts (Fig. 17.4.62). The rig itself is composed of a rigid plate and a structure on which are assembled drilling, hydraulic, and electrical equipment. The rig has its own crawler driven by compressed air, or it is mounted on rail wheels for haulage. The rig can be disassembled and hauled in underground service tire vehicles. A pilot hole is drilled through the stem with stabilizers and a conventional drilling bit (Fig. 17.4.63). Drill rods have outer/inner diameters in a range of 8 to 12-7/8 in. (20.3 to 31.4 mm) to 4-3/4 to 5-7/16 in. (12.06 to 13.1 mm). The length of the sections for underground application is usually 5 ft (1.5 m), dictated by clearance restrictions. Connections are made with tapered thread joints that have to be kept absolutely clean.

The purpose of the stabilizer is to assure directional accuracy of the pilot hole. Several types are in common use. They are of a larger diameter than drill rods, with strips of tungsten carbide welded vertically or spirally on the outside, bringing the outer

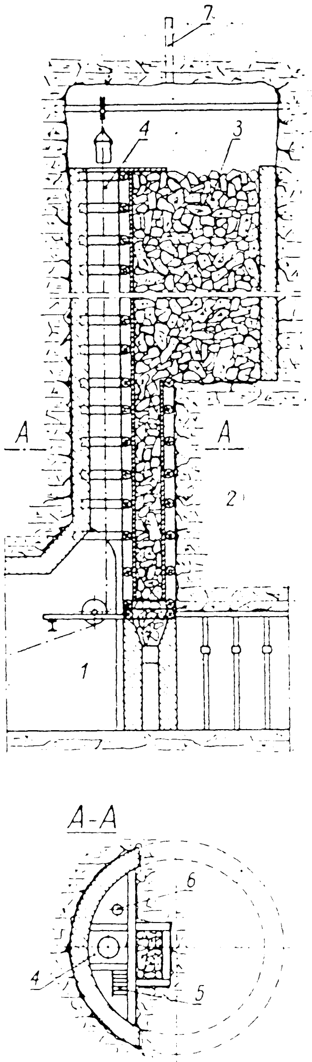


Fig. 17.4.59. Raising method with full diameter, short cuts, and immediate permanent lining.
1. shaft station on a new level
2. protective rock shelf
3. storage of muck
4. bucket compartment
5. ladder compartment
6. tubing compartment
7. drillhole for ventilation (to the sump of existing upper shaft).

dimension close to the hole diameter. These stabilizers are used for reaming up. Other types, for pilot holes and reaming down, have more clearance, allowing cuttings to pass.

Once the pilot hole penetrates to the mine opening, the pilot bit and stabilizer are removed. After necessary servicing, the stabilizer is mounted on the drill stem and the reamer is put in the place of the small pilot-hole bit. Starting the hole (collaring) is a critical operation, and damage to the cutters or bit body can occur due to the uneven surface. The tension on the stem has to be adjusted to allow starting the hole without a highly variable, dynamic load acting on the drilling components. The stem is the longest at this stage of drilling and, therefore, prone to maximum elongation and twist. The design of the reamer depends on the final size of the hole and the number of reaming stages. Structural analysis is required and is usually performed by computer, taking into account necessary balancing of the tool, achieved by optimum positioning of the cutters (Fig. 17.4.64). In the past, the primary body of the reamer was often supplemented with stages mounted below, resulting in a "Christmas-tree"-shaped tool (Fig. 17.4.65). Because of large torque requirements, this system is presently used only for shaft-sized holes, together with large non-rotating stabilizers. In another configuration, extensions are attached to the primary body. By doing this, an increase of the final diameter is achieved by adding more cutters in the same planes as the cutters on the primary body itself.

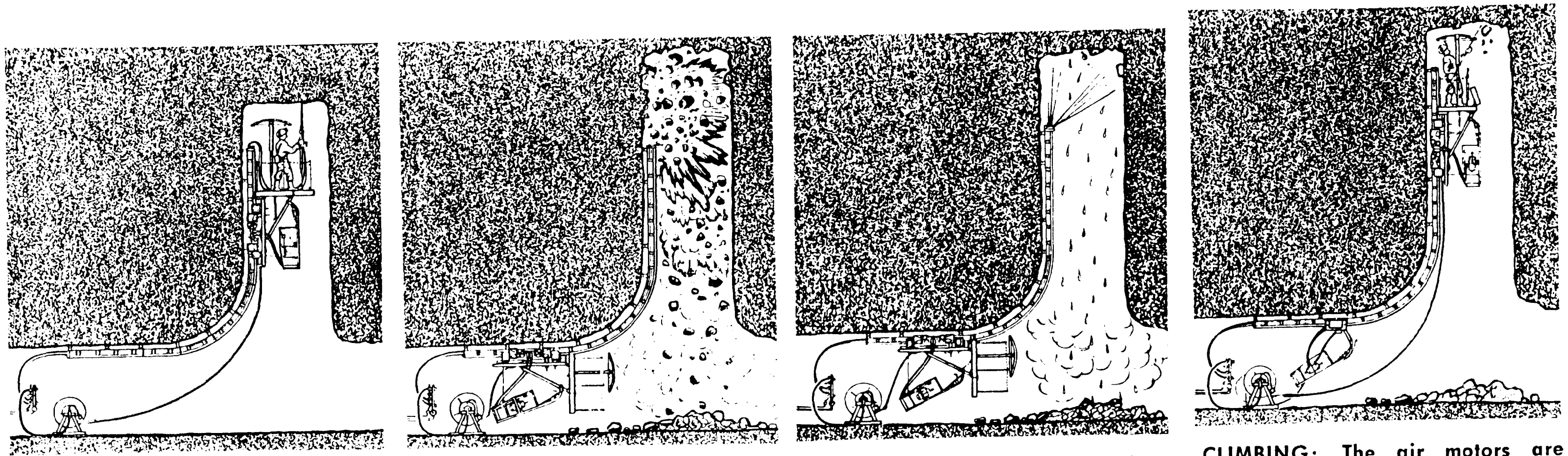
Geared reamers are being used for reduction of required torque when compared to conventional reamers. The bit turns at a slower rate than the drill pipe due to the gearing mechanism. Shaft reamers are used for very large-diameter holes. This is a separate machine that is installed after the standard-sized raise has been completed, and is used for enlarging the raise to the full diameter required. The shaft reamer stabilizes itself by side jacks pressing against the hole and develops both drilling load and torque reacting against the anchored portion of the machine. The cuttings fall freely down the previously completed raise. Shaft lining can be installed when drilling. A service hoist to the working level is, however, necessary for men and materials.

The accepted measure of drilling performance is the rate of penetration, which depends primarily on bit design. The cutting action has to match the rock properties to optimize drilling rate (Fig. 17.4.66). During the reaming operation, in addition to the cutters, the design of the reaming head also influences the raise boring performance. If no other source of information is available, an analysis of the wear of presently used cutters can help in the design of an improved cutting structure. Independent variables when drilling are rock properties, which are usually characterized by compressive, tensile (indirect), and shear strength measurements. The drilling tool should break rock primarily in tension and shear, because the strength of rock in these modes of loading is minimal. Strength alone does not determine drilling performance; additional factors intervening are rock ductility and abrasivity. For example, strong but brittle rock can have better drilling response than weak, clayish rock. Abrasivity is dependent on the mineral content of rock and is responsible for the wear of drilling tool elements in contact with the rock face and cuttings. The properties of the rock mass, such as joints, faults, and structural features, also influence drill response and, therefore, must be evaluated. The lowest drilling cost will occur at the point where both cutter life and rate of penetration are optimized.

One of the most important technological aspects of boring is the removal of cuttings. Research has shown that a layer of cuttings on the uncut rock reduces penetration up to 40%. In other words, the energy of drilling is spent on regrinding the cuttings rather than penetration of the intact rock. In raise boring, this situation is encountered during drilling of the pilot hole. Standard drilling technology with compressed air, air and foam, or drilling fluid is used for cuttings removal. During the reaming operation, cuttings simply fall down the hole.

Drilling load should be at the level required by the cutters to penetrate the rock and create fractures. Knowing the geometry of the cutting tool, it is possible to calculate the load exerted by particular cutters. Once the threshold strength of rock is reached, a further increase in drilling load does result in greater depths of penetration, but not in direct proportion to the increase of load. The general expression for this relationship is

$$D = W^d \tag{17.4.30}$$



DRILLING: Scaling and extension of guide rail is safely and easily carried out from the platform under the steel protective roof. Tubes in the guide rail are used for the supply of air and water to the drills.

BLASTING: During blasting the Raise Climber is well protected in the tunnel.

CLEANING: After blasting the face is cleaned with an air and water mixture sprayed through jets in a plate protecting the top of the guide rail.

CLIMBING: The air motors are controlled from the cage, and they are supplied with air via a hose connected to a reeling drum in the tunnel. ALICAB or ALITROLLEY Reserve and Service Lifts provide fast communication with the Raise Climber and is useful for the transport of men and material, or in the case of accidents.

Fig. 17.4.60. Stages of operation of the Alimak mechanical raise system (*courtesy*: Alimak Co.).

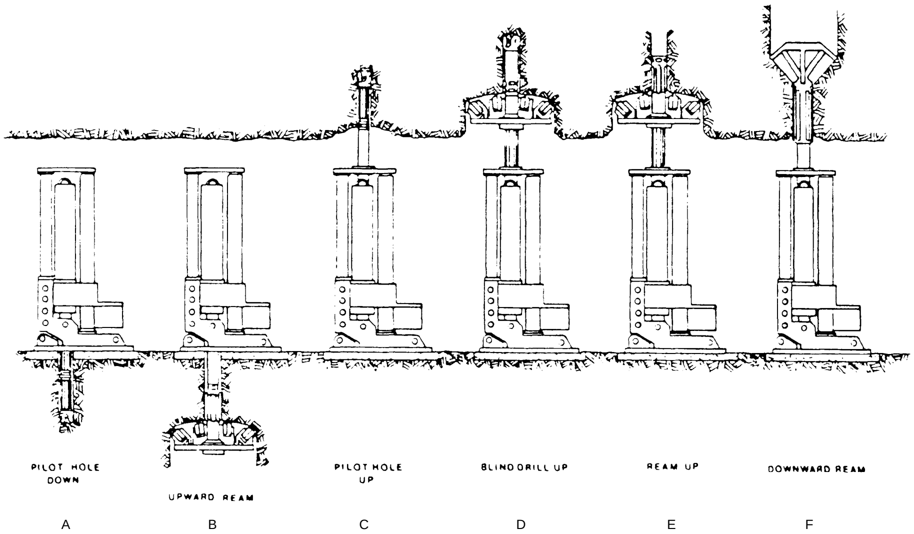


Fig. 17.4.61. Modes of operation in raise boring.

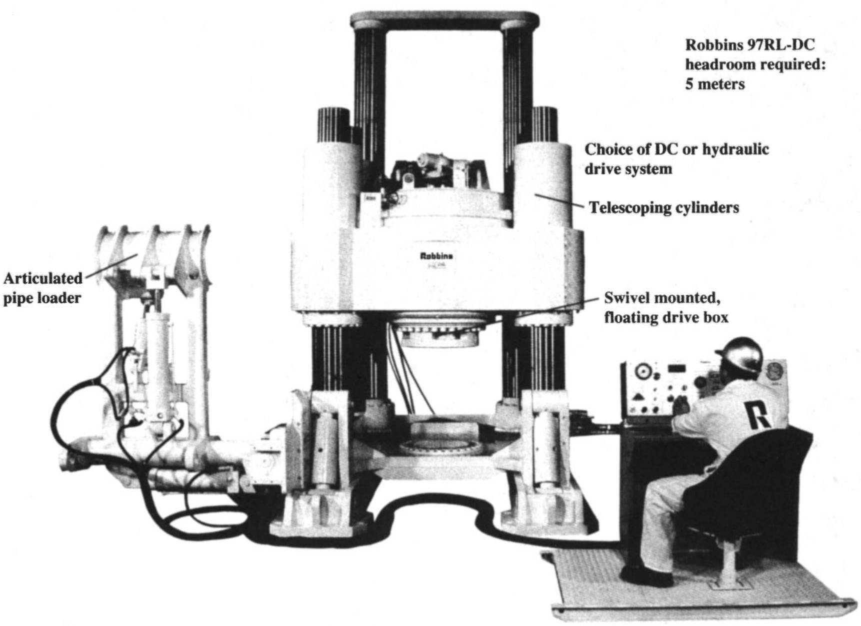


Fig. 17.4.62. Robbins 83RM-DC raise drill (*courtesy:* Robbins Co.).

where D is depth of penetration, W is the applied load, and d is a load exponent, which varies from 1.5 in soft rock to 1.0 in harder rocks.

During the reaming operation, the above equation is used for determination of load. The operator measures the rate of penetration over several minutes. He then adjusts the load and through several trials, finds the optimum load, above which the rate of penetration does not noticeably increase.

Rotary speed also affects the penetration rate. The higher the rotary speed, the faster the rate of penetration. However, restrictions other than the cutter-rock contact exist, namely, the wear of bearings and the tool itself. Vibration effects of uneven work by cutters result in wear of the drill pipe by rubbing against the wall of the pilot hole.

Table 17.4.12 lists the main parameters from several raise-boring projects.

METHOD SELECTION. The decision whether to drill raises from the top or bottom usually determines the access point. If only one end of the raise has adequate room, the machine must be put there. The extent to which raise boring will interfere with production must be evaluated. Location of the borer where less disturbance will be caused is preferred. Cost is also of major concern. Generally, drilling down and reaming up is cheaper than other configurations. Rock quality and its predicted response to drilling can influence the choice of direction as well.

LINING OF RAISES. In strong rocks without fractures, usually no support is needed. If support is required, shotcrete, rock bolts, with or without wire mesh, or steel lining can be applied. Shotcrete can be placed remotely or by workers from a cage. Rock bolts must be installed manually. Steel lining, required in unstable formations, is installed remotely, directly behind the reamer.

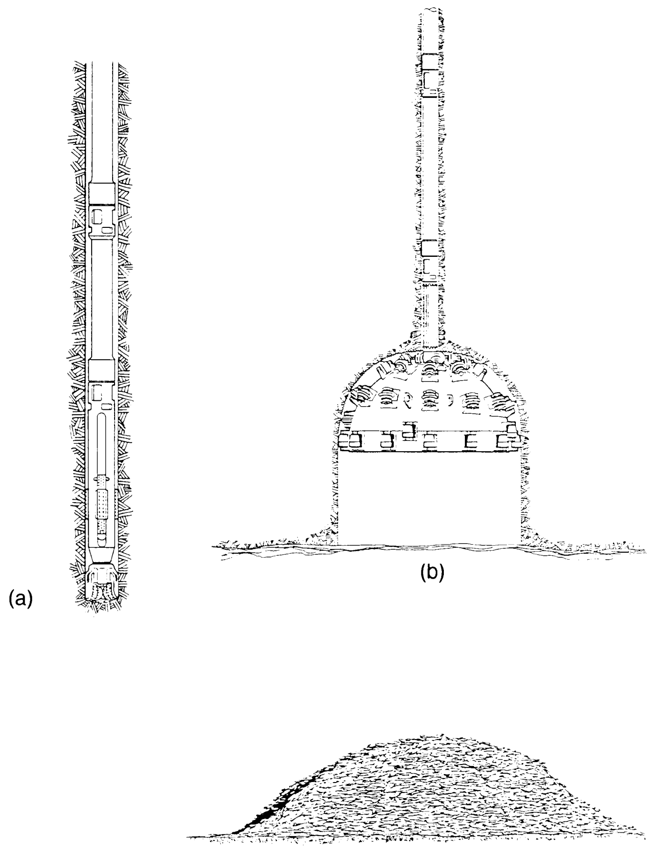


Fig. 17.4.63. Pilot hole (a) drilling and (b) reaming (*courtesy:* Robbins Co.).

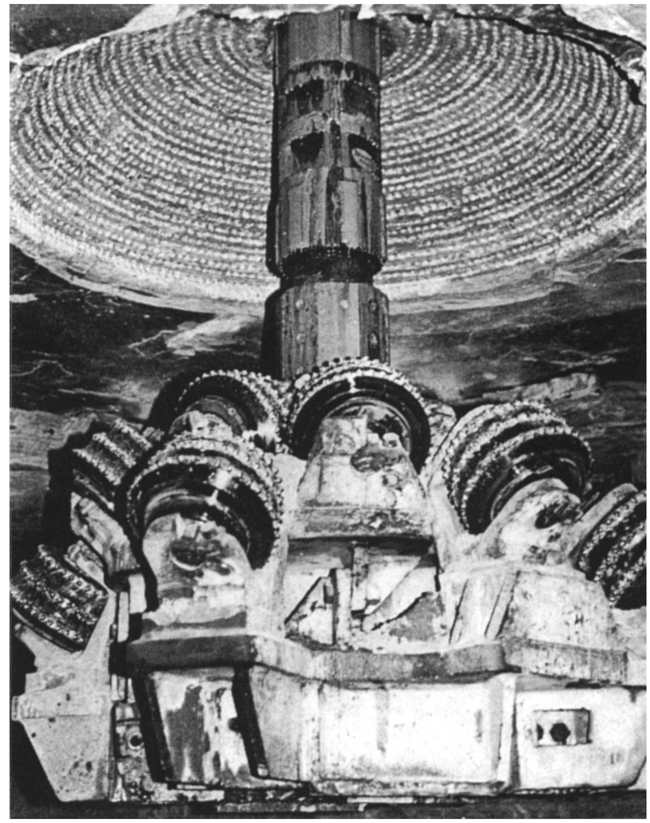


Fig. 17.4.65. Top section of 16-ft (4.8-m) diameter reamer breaking through the surface after 1490 ft (453 m) of reaming.

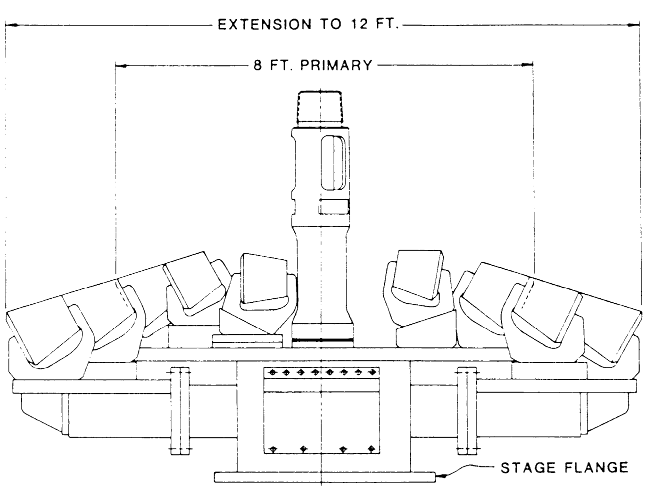


Fig. 17.4.64. Raise bit with extensions. Conversion factor: 1 ft = 0.3048 m.

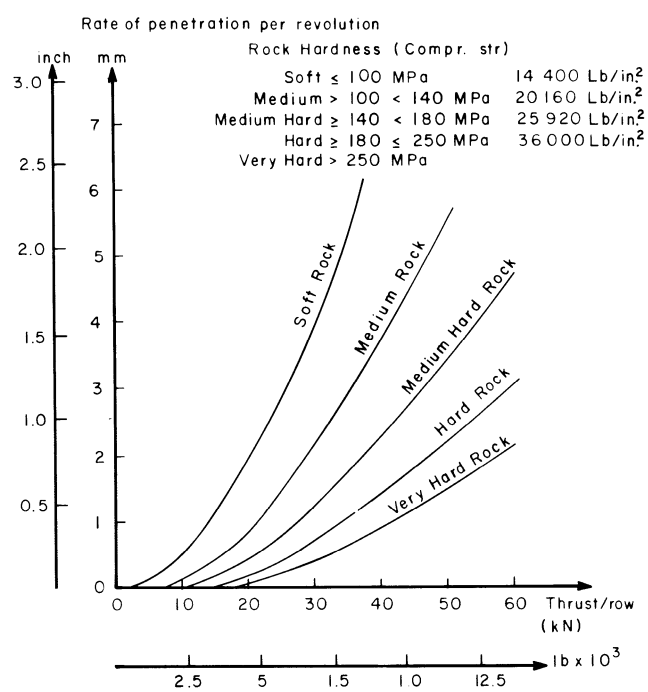


Fig. 17.4.66. Relationship between the rate of penetration thrust per row and rock strength.

For further discussion of rock boring, see Chapters 9.1 and 22.1.

17.4.2.6 Shaft Drilling and Boring

In recent years, a substantial effort has been made to improve methods and equipment used for drilling and for mechanical

Table 17.4.12. Examples of Performance in Raise Boring

Diameter ft	Max/Min Length/ft	Mobilization Max/Min Cost $	Setup Max/Min Cost $	Pilot Max/Min $/ft	Ream Max/Min $/ft	Lining $/ft
6	1150/129	89,000/13,500	26,000/10,000	128/36	196/75	200
7	1150/90	70,000/23,000	34,380/8,500	70/41	180/90	200
8	1280/160	37,400/20,450	19,125/12,700	76/30	175/105	—
9	1240/200	80,820/15,025	35,000/6,900	101/46	225/103	—
10	2000/110	58,286/4,436	29,100/4,500	94/40	267/157	—
11	280/180	20,075/14,750	19,950/15,025	71/44	305/216	—
12	180/120	28,000/20,500	26,350/10,000	93/43	530/175	486
14	255/150	47,250/18,000	91,700/13,000	71/70	765/500	500

Conversion factor: 1 ft = 0.3048 m.

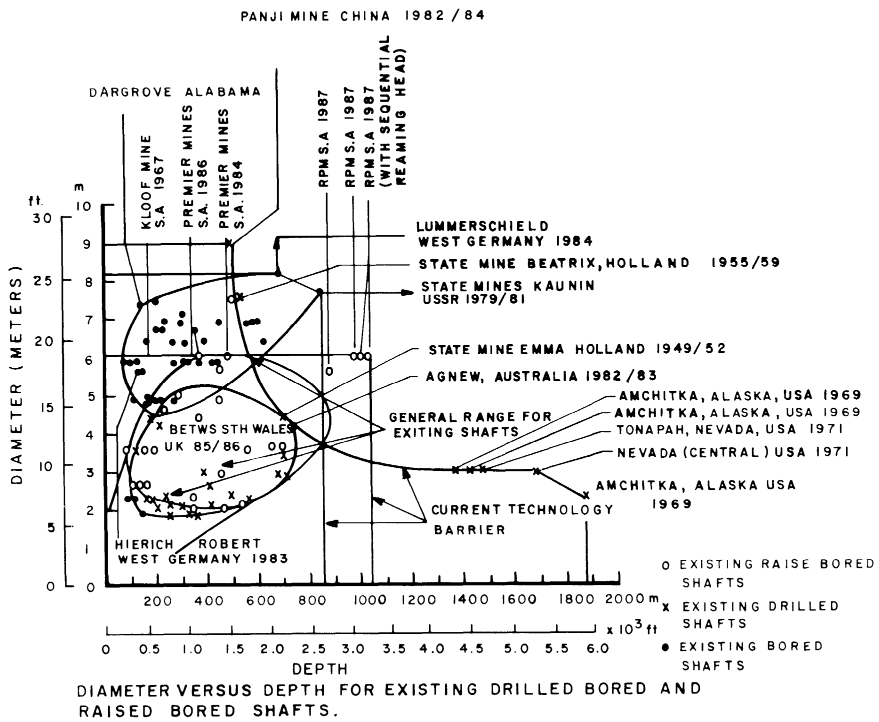


Fig. 17.4.67. Diameter vs. depth for existing drilled, bored, and raised-bored shafts. Conversion factor: 1 ft = 0.3048 m.

boring of shafts and large-diameter holes. In addition to classical mining, hydro and civil projects, a new application for the foundations of off-shore drilling platforms has emerged. Since conventional shaft sinking is the most hazardous work of all mining operations—and also, shaft conditions are among the most uncomfortable—methods eliminating human presence at the face are of great interest. Presently, two methods, shaft drilling and shaft boring, where applicable, can compete successfully with other conventional methods of shaft sinking. The cyclic nature of the mineral industry causes periods of activity in mine construction followed by periods of stagnation. Consequently, the rate of development of new technology and progress in existing technology is affected. Mechanical methods of shaft construction provide a good example of this process, as is the rate of their improvement.

The present state of technology permits shaft drilling to a depth of 3000 ft (1000 m) and up to 30 ft (10 m) in diameter (Figs. 17.4.67 and 17.4.68). Shaft drilling generally competes with conventional methods in weak and moderate-strength rock and can cope with the most difficult hydrogeologic conditions, which otherwise require application of expensive special methods. The most powerful rigs can even drill formations with compressive strength up to 42,000 psi (300 MPa).

Designing the drilling operation requires consideration of the following steps:

1. Choice of the drilling method according to the hydrogeologic conditions and geotechnical parameters of rock along the shaft profile. Next the drilling rig has to be selected based on the planned shaft diameter and depth, adequate cutters for the type of rock, stages of reaming, if any, and properties of drilling fluid.

2. Design of shaft head.

3. Selection of a type and technology of lining placement (buoyancy or casing type), determination of footing type and location, and fastening and sealing of lining in the shaft.

4. Design of the system of sealing water within aquifers, if present.

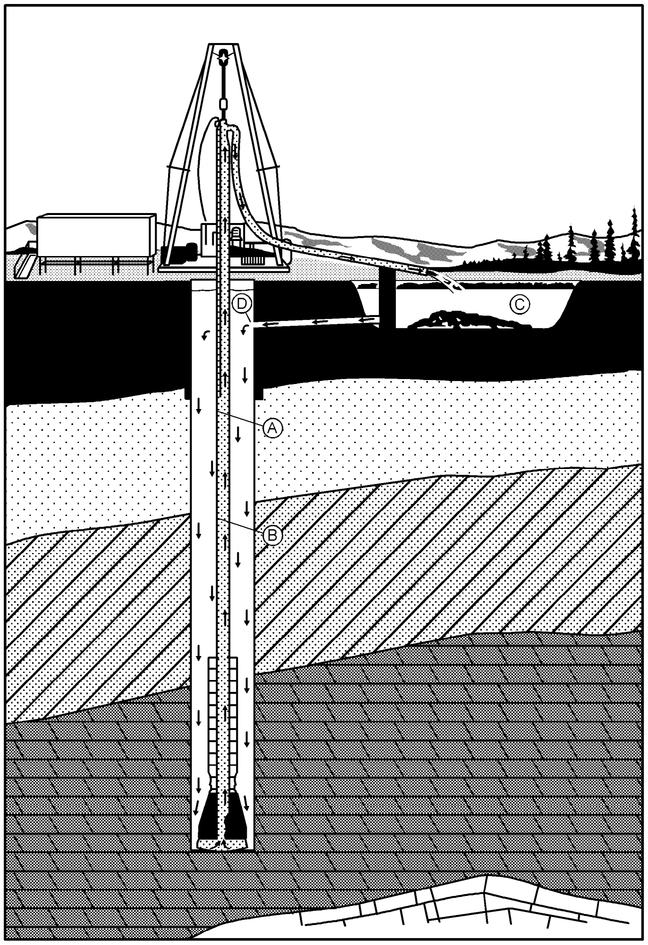


Fig. 17.4.68. Blind hole drilling system.

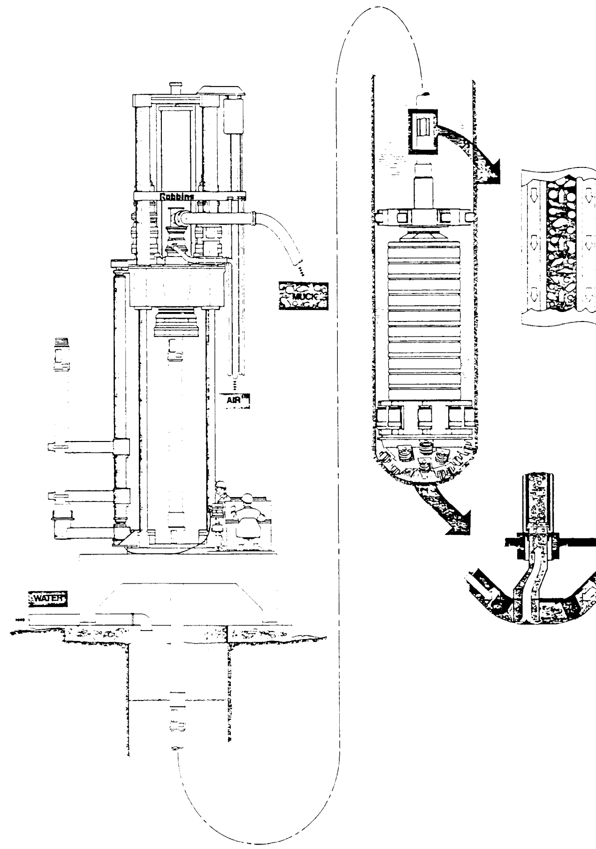


Fig. 17.4.69. Blind hole drilling rig (*courtesy;* Robbins Co.).

5. Organization of the drilling site, location of ponds for drilling fluid and a site for the fluid-making plant, and storage of lining components and materials.

DRILLING FLUID. The drilling fluid is a mixture of water and natural clay partly dispersed and partly colloidal, which by its hydrostatic head, keeps the wall of the hole in an equilibrium state and also transports chips of drilled rock and cools the bits. Ordinary drilling fluid should have a specific gravity of 1.15 to 1.25. Drilling fluid is prepared in mixers from which it flows to a tank and is pumped by slurry pumps to the shaft. During drilling, the shaft has to be filled completely with drilling fluid. To impose movement of the fluid, a compressed air airlift is used (Fig. 17.4.69). In small holes where the annular space between the drill string and hole wall is small, normal (right) circulation is applied. Air is injected in this space, and the drill fluid flows to the bottom through the drilling pipe, rising around it to lift cuttings to the surface. For large holes with a large sectional area of annular space between the drilling string and wall, the reverse system is used. Air-assisted, so-called reverse (or left) circulation systems, can be applied as single-, double-, and triple-wall pipe. In the simplest case, for example, a 3-in. (75-mm) pipe (if 13 3/8-in., or 33.4-mm, drill pipe is used) is run down the drill string, approximately 330 ft (100 m), sometimes to the end of the string. Compressed air is injected through the 3-in. (75-mm) pipe. When it rises in the annulus between the inner and drill pipes, it creates a mixture of air-drilling fluid of decreased density. The positive pressure of the drilling fluid (with unchanged density) in the shaft induces circulation with a resulting high flow velocity in the drill pipe, enabling transport of cuttings to the surface. The more viscous the fluid, the less the fluid velocity need be. Usually, the maximum particle load any system can carry is 5% of its volume.

A new development is termed pumped reverse circulation. Instead of air, a submersible pump is installed as a part of the drilling string. This allows an increase in the volume of circulating fluid (in drill pipe of 12.8-in. or 300-mm bore, 4000 to 5300 gpm, or 15 to 20 m^3/min) and consequently, better evacuation of chips from the face. To increase the size of cuttings transported, in addition to enlarging the flow volume, flow velocity can also be increased by restricting the cross-sectional area with skirts and shrouds at the very bottom of the hole. Here rock chips are lying exposed to unnecessary regrinding by the bit. Changing the angle of the sucking hole and adding chemicals to the drill fluid can also serve that purpose.

DRILLING EQUIPMENT. Originally, the drilling rigs used for large-hole drilling were converted heavy oil rigs. These were adequate when dealing with holes in a range of 10 to 12 ft (3 to 4 m), but for shafts of 15- to 25-ft (5- to 8-m) diameter, a rig had to be designed on a different set of principles. The rig must be able to produce large torque, and the required hoisting capacity must be very large. For example, CSD-300 is a rig manufactured by Hughes Micon, with six hydraulic motors producing 500,000 lb-ft (70 kNm) of torque. The power swivel is raised and lowered on hydraulic cylinders with a 2-million-lb (880-t) capacity (Fig. 17.4.70).

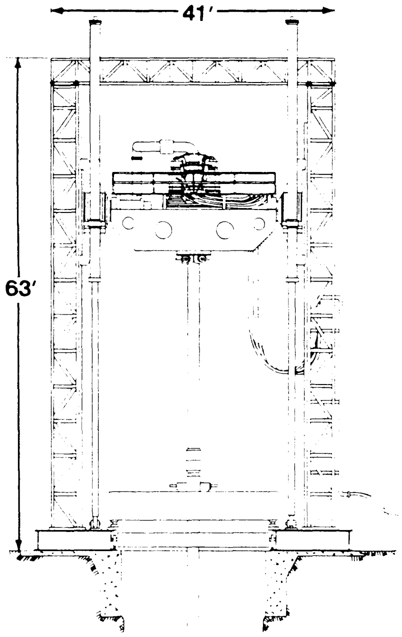


Fig. 17.4.70. CSD 300 hydraulic rig (*courtesy:* Hughes-Micon).

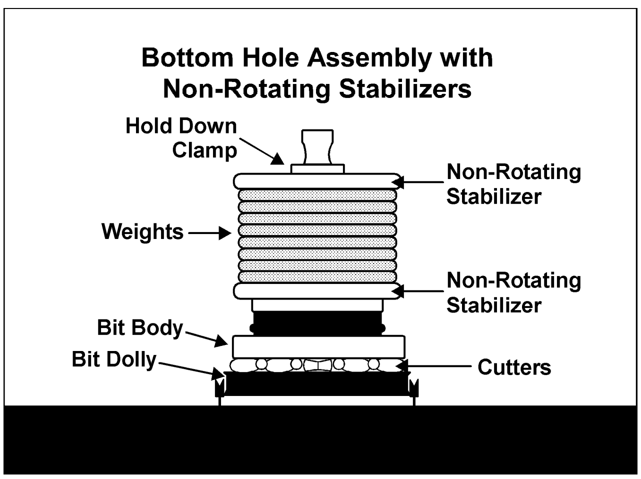


Fig. 17.4.71. 14-ft (4.3-m) diameter bottom-hole assembly.

Verticality of a large blind hole is a necessary condition that can be achieved by guiding the bit by a pilot hole or by the plumb bob effect of the very heavy end of the drilling string with large bit, weights, and stabilizers. For very-large-diameter holes, it may be necessary to use two or more passes with different sizes of bits.

Weights above the drilling structure serve several purposes other than assuring verticality of the hole. They produce the load required for bits to penetrate the rock and prevent reversal of stresses on the drill string by the flywheel effect, smoothing the dynamic reaction on the string from jerks and bumps caused by bits rolling over the rock with unevenly distributed strength. Two basic types of stabilizers are used, rotating and nonrotating. The first rotates together with the string and is often equipped with rollers (Fig. 17.4.71). Those rollers sometimes have cutting structures and work as reamers. Nonrotating stabilizers have an outer shell or pads that remain stationary while the inner core rotates. They are appropriate for softer rocks because they do not damage the shaft wall formed by weak material. Stabilizers assure vertical alignment of the hole by restricting lateral movement of the drilling bit. By using multiple stabilization, it is possible to drill shafts with an alignment of 1 in 1000.

Breaking of the rock at the base of the hole is achieved by cutters attached to the bit body. Great effort has been made over the last few decades to increase the reliability and life of cutters while permitting increased loading to be applied.

Milled steel-tooth cutters are most suitable for soft rock, producing large cuttings. This gives the potential for rapid rates of penetration, but the performance of the cutters diminishes rapidly as the teeth become dull.

Tungsten carbide cutters are more appropriate for harder rock or in particularly abrasive formations; they require substantially larger bit loading than other bit types and produce smaller chips. Their rate of penetration is slow, although performance remains steady throughout the life of the cutter. Disk or kerf cutters are not normally used for blind hole drilling because they tend to produce large chips that cannot be transported across the bottom of the hole and up to the surface. In soft and plastic strata, they can produce concentric furrows in the base of the bore with no excavation progress.

The cutters themselves are mounted in saddles that are attached to a fabricated steel body of the bit. Various basic shapes have been used, including flat-bottomed, conical, hemipherical, or a combination of shapes. There is not enough evidence to determine which shape is the most advantageous, because varying conditions in particular projects make such comparisons difficult. Some believe that flat-bottom bits drill straighter holes than do conical bits; however, the latter give a better rate of penetration in medium to soft formations.

LINING FOR DRILLED SHAFT. Types of permanent lining for drilled shafts depend on the character of rock formations intersected by the shaft. In strong rocks, minimum lining requirements exist; therefore, shotcrete usually is sufficient. In fractured zones, a combination of rock bolts with mesh and shotcrete can be applied. For shafts with hoisting installations, usually a minimum thickness of 8 in. (200 mm) of concrete lining is required, needed for the fastening of consoles for shaft beams. These types of lining must be installed from a hanging stage using conventional technology.

Permanent lining for water-bearing, weak strata can be made of reinforced concrete or steel, as a single (with stiffening rings) or double cylinder with concrete fill between the outer and inner segments. The work is performed at the surface and the lining is lowered to the shaft filled by drilling fluid.

In the floating method, a steel bottom is made (welded) beside the shafts, lined with 16 to 32 in. (400 to 800 mm) of concrete, and set afloat in the shaft. On this bottom, a first segment of lining 8 to 11.5 ft (2.5 to 3.5 m) high is welded. Extending the lining column with consecutive segments causes the weight of the lining to increase and, consequently, it sinks. Drilling fluid is displaced by lining with the equivalent volume. When the lining has positive buoyancy, by pouring water into it, the required balance can be achieved. The rate of lining placement is about 30 ft/day (10 m/day). If required by the depth of the shaft and the formation pressure, a double steel-cylinder lining with concrete filling the space between them can be made, just before sinking it into the shaft, using steel segments with the required curvature and automatic welders. The concrete inside the lining is usually made with application of the Colcrete method. Injectors are placed vertically with uniform spacing. The space is then filled with gravel, sand, and aggregate (mixed in the desired proportion), and cement of high quality is pumped into the stone skeleton through injectors, filling voids from the bottom up and pushing the air out. Vibrators are used for upgrading, compacting, and strengthening the concrete (5700 to 7100 psi, or 40 to 50 MPa, after 28 days). If for some reason, any difficulties appear during the sinking operation, the lining can be turned around, or filled with drilling fluid instead of water to make it heavier.

Another system of lining installation applies reel drum or hydraulic hoists to the lower segments of the lining 50 to 75 ft (15 to 25 m) high, without a bottom. This system is more appropriate for lighter single-cylinder lining, sufficient for conditions with lower formation pressure. After lining the entire length of the shaft, the annular space between the lining and the shaft wall is filled with cement-clay grout (proportion 2:1 and specific gravity 1.65 to 1.75) and is pumped upwards through the hoses from the bottom. The grout seals the contacts of the particular lining sections. When the grouting operation is completed and the grout has hardened to sufficient strength, the drilling fluid is pumped out from the filled shaft. If required, additional sealing can be done from the stage by grouting.

SHAFT BORING. Shaft boring is the newest and probably the least-mature technology of shaft sinking, even though this concept has been tried since the 1960s and earlier. Equipment in this category, called shaft boring machines (SBMs), could be considered as vertical tunnel boring machines (TBMs), with operating personnel traveling on board, and muck transportation and service facilities provided to the surface. The major problem encountered in this type of configuration is picking up the cuttings from the sump face for removal up the shaft. Wirth machines were quite successful with a pilot hole for free-muck removal to a lower (accessible) level. However, complete blind-hole operations have had only limited success, although the newest Robbins-Redpath machine seems to have a good potential for success. This machine has been developed utilizing experience gained with previous systems.

The SBM consists of a cutter wheel with a horizontal axis mounted on a carriage with vertical travel. The carriage is mounted on a slewing structure that rotates about the vertical axis of the shaft. The entire assembly is positioned in the shaft by an upper and lower gripper system that engages the sides of the shaft. Walking cylinders between the gripper assemblies allow the machine to advance up or down the shaft in increments. A clam-type mucking unit is mounted opposite the cutting wheel on the slew structure and loads directly from the shaft bottom into a hopper. A conventional mine hoisting system allows muck buckets to be lowered into the SBM to transfer muck from the hopper to the surface. The cutter wheel itself is fitted with disk cutters and is powered by two 261 kW motors. In the design of cuttings removal, all the advantages of disc cutters can be realized, similar to the TBM. All components of the machine fit through a 6- by 6-ft (2- by 2-m) opening, with the longest component approximately 17 ft (5.5 m) in length. The heaviest is 14 tons (13 t). This makes for easy transport and assembly and disassembly of the SBM, which can be employed even in remote locations (Fig. 17.4.72).

Carboniferous rock is probably the most suitable environment for an SBM. Formations which have a large amount of water inflow must be sealed by grout before the SBM begins boring.

17.4.3 SLOPES AND INCLINED SHAFTS

17.4.3.1 General Rules for the Design of Slopes

For shallow deposits, especially coal, *slopes* (also called *inclines* or *declines)* are more often chosen than shafts as the main entrance to a mine primarily because of their lower cost and shorter construction time. With belt transport of the coal or ore, a 15.5° slope angle is the maximum. Since a minimum of two entrances to a mine is required, slopes can serve as an intake airway while a ventilation shaft is raise bored from the mine level. As with a shaft, a slope provides all services for the mine such as main transport of production, personnel, materials and equipment, pipes, and cables (for power and communications); therefore, the cross section of the slope is designed to accommodate the required space. In coal mines, the main belt often is placed in an overhead compartment isolated from the rest of the cross-sectional area. Equipment, supplies, and miners are hoisted by means of the appropriate type of rail cars. Often the personnel walkway is also in a separate compartment to prevent exposure to the stream of intake air.

When comparing a slope with a shaft, the advantages of slopes are as follows. The beltway transportation system is uninterrupted all the way to the surface, either to the preparation plant or to the loading facility for surface transport. Slope construction is quite rapid and can be performed with mine personnel. In good quality rock, minimum support is required; often shotcrete with rock bolts is sufficient, providing additional insulation against the weathering of rocks. The main disadvantage of a slope is its increasing length with deposit depth, which makes shafts less expensive for depths exceeding about 1150 ft (350 m). However, for high-output mines, operating costs of a belt vs. shaft hoisting may still make the slope a more attractive solution, even for deeper mines.

A similar type of opening often encountered in hard-rock metal mines is a *ramp*. Ramps may be spiral or constructed as straight ramps with sharp turns. Their primary function is to connect adjacent mine levels and to allow for movement of rubber-tired haulage or tire equipment. Consequently, ramp floors have to be carefully designed and constructed to prevent tire wear on the vehicles. Good surveying of spiral ramps during construction is of great importance.

17.4.3.2 Ground Support

Slope portals at the surface are usually, as with tunnels, made of concrete. Depending on the configuration of the terrain, portals may be located at the highwall of the hill, as is practiced in Appalachian coal mines, or on even ground. In every case, they must be placed at a higher elevation than the surrounding area to avoid flooding. Further, slope entrances, similar to shafts, should be above the 100 years' flood level. Slopes intersect all strata of the overburden above the mine level, and ground support must be provided during construction and permanent service. Depending on the geotechnical parameters of rocks, support systems most often used are rock bolts, rock bolts with mesh, shotcrete, and steel arches. Selection of the support is also dependent on the technology of slope construction, size of its cross-sectional area, and utilization of slope. Principles of support selection, based on ground conditions, are given in Chapter 10.5.

17.4.3.3 Technology of Slope Construction

Technology of slope construction is similar to that of drift and tunnels; however, it is more difficult because the inclination of excavation makes the removal of muck more difficult.

A slope can be driven from the surface or upper level down, or inclined upwards if the lower level is already accessible. In rocks of compressive strength not exceeding 15,000 psi (110 MPa), roadheaders are recommended to achieve rapid progress and low cost per unit length of excavation. In rocks stronger

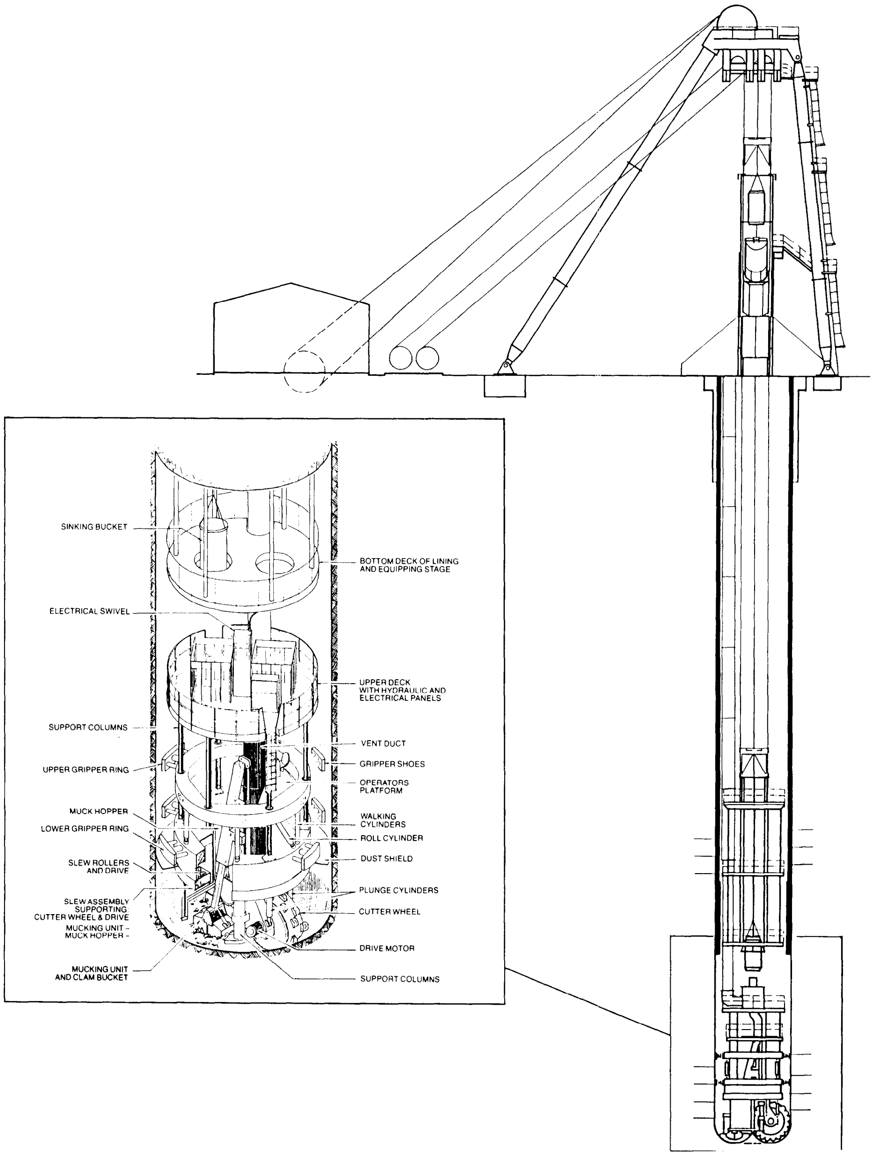


Fig. 17.4.72. Robbins shaft boring machine (*courtesy:* Robbins Co.).

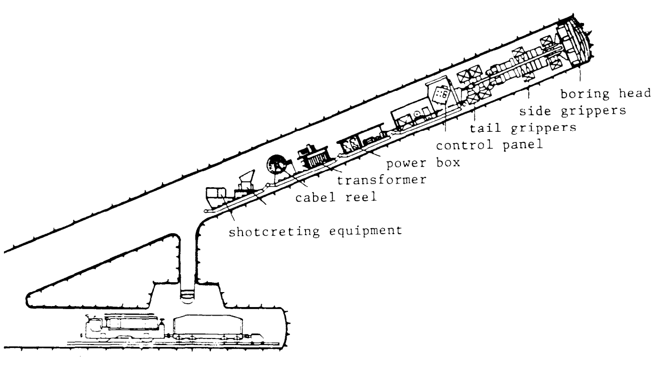


Fig. 17.4.73. Driving incline with tunneling machine (*courtesy:* Thyssen Schachtbaus Co.).

than indicated above, conventional drill-and-blast methods are usually used. In slopes where a circular cross-sectional area is required, and the total length justifies the high cost of equipment, a TBM can be applied (Fig. 17.4.73).

TECHNOLOGY WITH ROADHEADER CONFIGURATION. A boom-type roadheader cuts the face in a desirable shape, usually with an arched roof (Fig. 17.4.74). Laser beam guidance systems are used for maintaining direction along the excavation axis. Dust is controlled by exhausting tube ventilation and with water sprays installed on the cutterhead. The roof is supported by steel arches erected by a miner's boom with yoke. Steel arches can be left as the elements of permanent support or reused at the face area if another type of permanent lining is chosen (e.g., shotcrete). Muck is removed from the face by means of gathering arms on the apron of a roadheader. A conveyor at the back of the machine feeds another conveyor suspended on a monorail. A conveyor loading rail car, or cars, is periodically hoisted upward. During the car trip, the face loading system is shut down, and the apron of the machine then serves as a surge pile.

CONVENTIONAL CONFIGURATION IN HARD ROCK. The following machines are employed at the face: drill jumbo (two-boom), side loader (e.g., Eimco 633), face conveyor with impact roller crusher and loading stage for the next element of transport system (e.g., rail car), lifting cage, and shotcrete machine. Because this configuration is used in hard rocks that most often

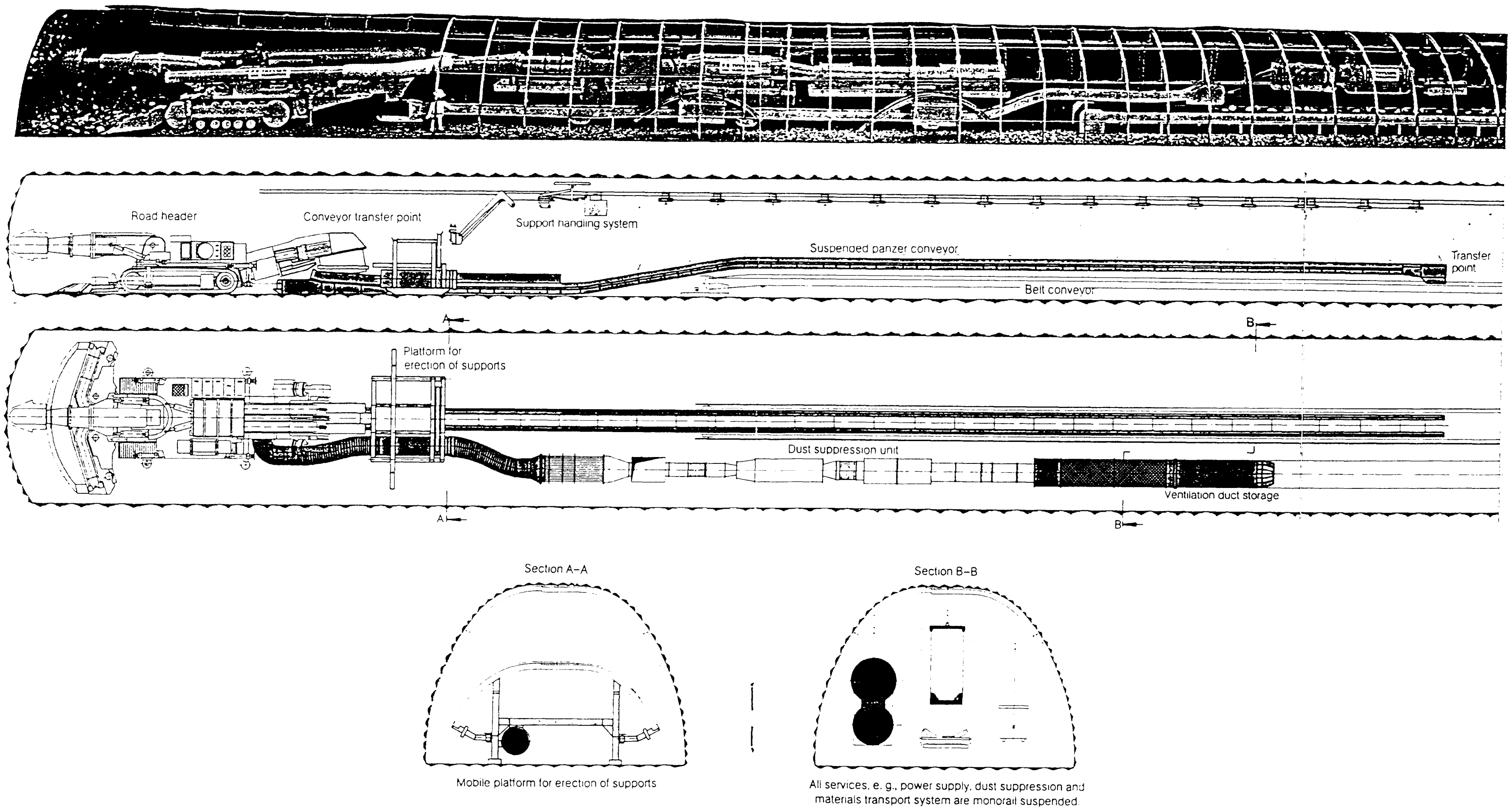


Fig. 17.4.74. Slope development with application of the roadheader (*courtesy:* Thyssen Schachtbaus Co.).

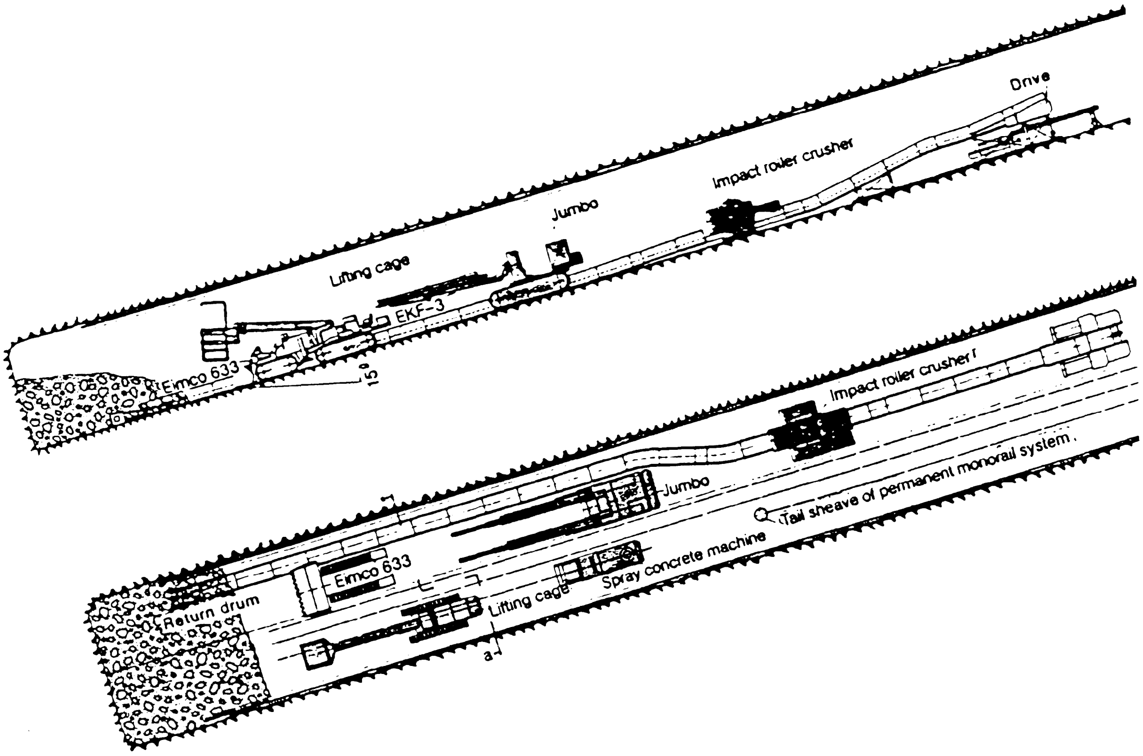


Fig. 17.4.75. Slope development with conventional drill-and-blast method. Development of a lower level by an incline, face equipment with drill jumbo and side-tipping loader (electro-hydraulic), steel support and sprayed concrete (*courtesy:* Thyssen Schachtbaus Co.).

are self-supporting, shotcrete is sufficient for temporary as well as for permanent support (Fig. 17.4.75). If water gathers at the face in excessive quantities (a small amount is beneficial for dust suppression), a face pump must be used. With a long slope, an additional unit is required to pump water to the surface (upper level).

Excavations with an angle of more than 45° are called inclined shafts. They are most often found as part of the materials handling system (orepasses) in hard-rock mining. At the present time, technology of construction is usually based on raise boring or drill-and-blast with application of a climbing platform as described in Segments 17.4.2.4 and 17.4.2.5.

17.4.4 SHAFT STATIONS

Types of shaft stations are diversified to the same degree as underground mines. Generally, the longer the life span of the mine and the larger its output, the more complex the shaft station becomes. The design considerations depend on the number of shafts within the station, type of deposit, mode of materials handling in the mine and in the shaft, water inflow, ventilation requirements, mining equipment, etc. Some typical solutions will be briefly characterized. Shaft stations in hard-rock mines with large production have several chambers that may have substantial dimensions. Those related to materials handling are skip loading pockets, retention bunkers (horizontal or vertical), pump chamber and main power station, explosives storage chamber, locomotive room, mechanical and electrical shops, miner station, dump chamber with bunker and sometimes primary crusher, and acclimatization chambers. If unit trains are the main mode of transportation, appropriate circuits are designed for empty and full trains. In cases of large water inflow in a deep mine, the pump station and sublevel water reservoirs will have large dimensions.

High-output coal mines operating in inclined seams, and consequently having primary development in rock (so-called rock structure), will have a design similar to that of hard-rock shaft stations. This is typical, for example, of European coal mines. US mines in the eastern coal fields operate in generally horizontal seams, where development with room and pillar mining equipment takes place exclusively in coal (so-called coal structure). As a result, shaft stations are developed within the limitations of the applied technology and often—due to the nearly exclusively used belt transportation—are quite simple. The geometry of such a shaft station resembles a set of main entries with larger pillars. Rail transport is limited to the handling of supplies and transportation of personnel. Service areas are arranged in particular entries.

17.4.4.1 Shaft Station Components

INSETS OF THE CAGE AND VENTILATION SHAFTS. The size of the inset of a cage shaft depends on the width and number of cages being hoisted on this level, number of decks in cages, and length of supplies to be delivered. Additionally, the inset cross-sectional area is checked for ventilation requirements. (The maximum recommended air velocity is 700 fpm (4 m/s) for production shafts and 1500 fpm (8 m/s) for ventilation exhaust shafts). Within the inset, other openings must be made, such as basements for car pushers and swinging platforms, basements for simultaneous entrance and exit of workers from multideck cages (if required), a niche for control equipment, passage to personnel station, by-pass around the shaft, etc. (Fig. 17.4.76).

Insets used only for ventilation purposes are less complex, due to their simple function.

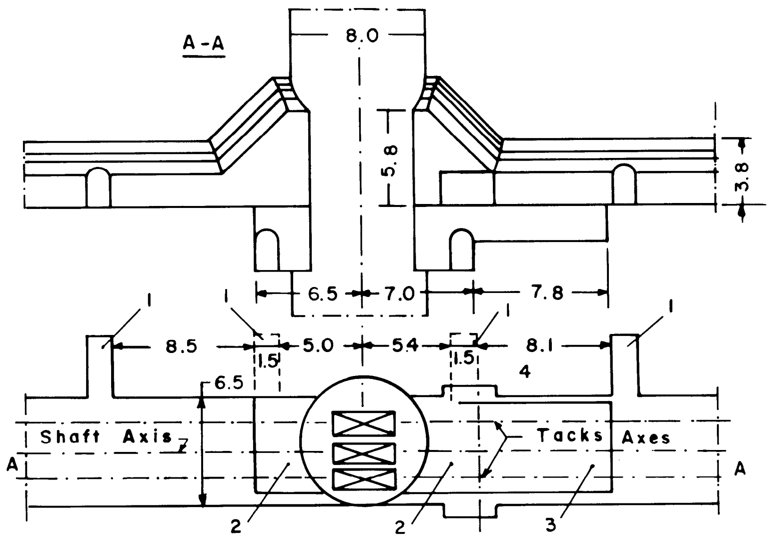


Fig. 17.4.76. Inset of cage shaft with three levels to step in and out for crew.
1. access drift to waiting room
2. basement for two level traffic and swinging platforms
3. basements for pushers and barrages (blocking, cars)
4. niche for control equipment.

Dimensions in m. Conversion factor: 1 ft = 0.3048 m.

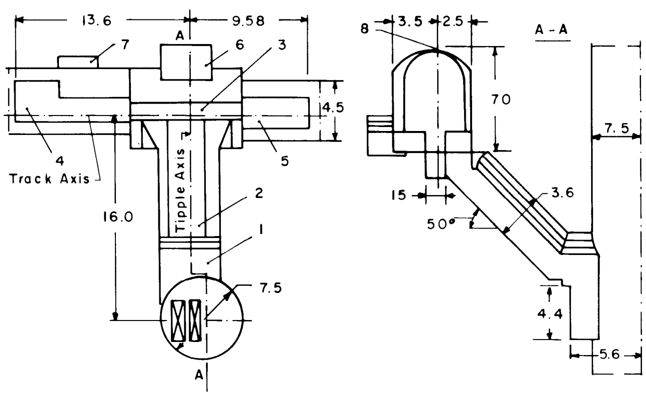


Fig. 17.4.77. Connection of production skip shaft with the opening of loading system for rail transport system.
1. skip chamber
2. batcher chamber
3. tippler chamber
4. basement of shifting mechanism
5. basement of braking system
6. drive niche
7. electrical equipment niche
8. ventilation (only in methane mines).

Dimensions in m. Conversion factor: 1 ft = 0.3048 m.

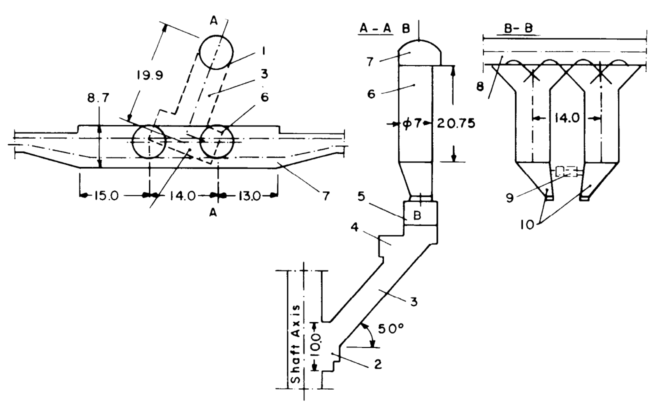


Fig. 17.4.78. Connection of production skip shaft with the openings of the loading devices for horizontal rail transport.
1. skip shaft
2. skip chamber
3. batchers chamber
4. switches chamber
5. loading chamber
6. retaining bunker
7. distribution chamber
8. distribution ramp
9. drift for clearing away jams
10. chutes.

Dimensions in m. Conversion factor: 1 ft = 0.3048 m.

INSETS OF PRODUCTION SHAFTS WITH LOADING POCKET FOR SKIPS. Skip shafts and production levels are connected by a system of openings of interrelated function that accommodate the flow of material. Their shape and dimensions are dependent on the skip size and loading system as well as on the type of mechanical and electrical equipment in the system.

Depending on the skip loading system and horizontal transportation arrangement, there could be the following sets of openings for loading facilities:

1. For rail horizontal transport:

a. Dump (tippler) chamber or unloading ramp chamber (for Granby cars), batchers chamber, skip chamber (so-called stiff system) (Fig. 17.4.77).

b. Tippler chamber or unloading ramp chamber, retaining bunker, loading devices chamber, batchers chamber, skip chamber (so-called elastic system) (Fig. 17.4.78).

2. For belt horizontal transport: unloading chamber, retaining bunker (vertical or horizontal), loading chamber, batchers chamber, skip chamber, retaining bunker, loading chamber, drift with belt scales, skip chamber (also elastic system) (Figs. 17.4.79 and 17.4.80).

SHAFT STATION CHAMBERS. These chambers are an important link in the extraction process, transport, and energy supply. Because of their function, they are located in the vicinity of the main or auxiliary shafts.

The first group includes the main dewatering pump chamber, explosive storage, machine shops, fire extinguishing chamber, and crew station. Chambers serving the transportation system are the locomotive, car repair and cleaning, rectifiers, and dispatcher room. Examples of particular chambers are shown in

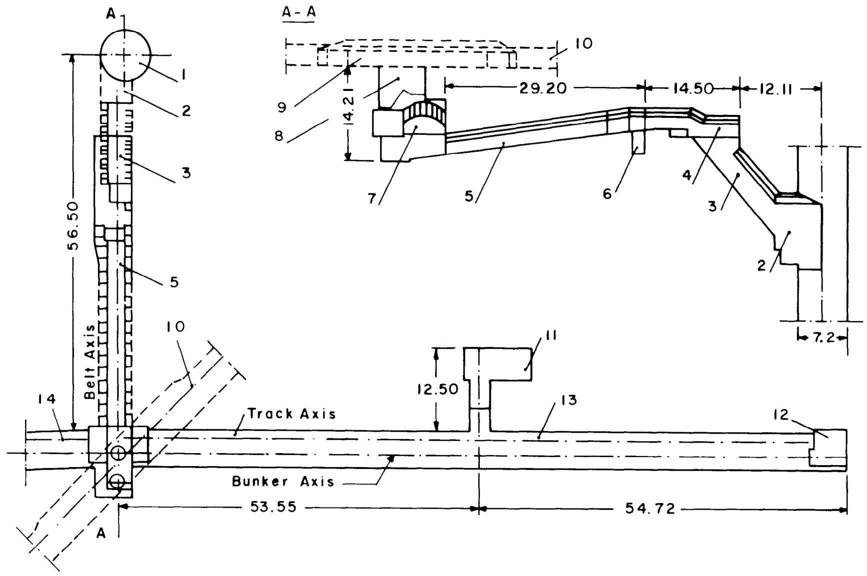


Fig. 17.4.79. Connection of production skip shaft with openings of the loading devices for horizontal belt transport system, elastic system with horizontal bunker-mechanical type.

1. skip shaft
2. skip chamber
3. batchers chamber
4. rectifiers chamber
5. belt drift
6. basement for belt tensioning weights
7. loading chamber
8. bunker-chute
9. unloading chamber
10. main belt drift
11. chamber of hydraulic pumps for mechanical bunker
12. chamber of turning station
13. horizontal bunker I (mechanical type) 300 t (330 tons) capacity
14. bunker.

Dimensions in m. Conversion factor:
1 ft = 0.3048 m.

Figs. 17.4.81 through 17.4.83. Two examples of the general layout of shaft stations are presented in Figs. 17.4.84 and 17.4.86. Power distribution chambers include the main power distribution chamber, main switch chamber, and other energy-related openings.

The technology for construction of shaft station chambers is similar to that for tunnels of corresponding size. Ground conditions (mixed rock face) have great influence on sequencing of the excavation in layers. The conventional drill-and-blast method is usually used; however, continuous miners and roadheaders may also be used in weaker rock formations.

17.4.5 HORIZONTAL DEVELOPMENT OPENINGS

Primary and secondary horizontal openings play a major role in the development of a mine. This is well documented in statistics showing that out of the total length of development openings driven in 1980, 82% were horizontal openings, 16% were raises, and the rest were shafts, slopes, etc. (Martens, 1982).

Horizontal openings for primary development are applicable for mineral deposits in mountainous terrain. The deposit can be an outcropping (coal seam) or one that can be reached most conveniently by a horizontal tunnel (adit or drift) nearby. The cross-sectional area of the opening, as in the case of shafts and slopes (declines), must accommodate all required services for the mine (17.4.1.1). To meet large production and ventilation requirements, a cross section of large size is required. An alternative is the use of multiple openings, customary in coal mines. The shape of the cross section of the opening may vary from rectangular (in outcropping seam deposits such as coal) to any convenient shape (horseshoe, round) as dictated by ground conditions, construction technology, and required services of the capital opening.

Secondary development openings can be driven within the deposit, as is the practice in flat-lying coal seams (e.g., drifts, entries, crosscuts), or outside the deposit, as is generally practiced in three-dimensional ore bodies in metal mines (e.g., adits, tunnels, drifts). In the case of coal mines, there is no difference between secondary development and the production openings. The only difference is in the number of parallel entries, which can vary from seven to eleven mains and usually five to seven submains. The number of entries depends mainly on ventilation and haulage needs.

TECHNOLOGY OF CONSTRUCTION. Two major systems are in general use: 1) the drill-and-blast method, and 2) mechanical systems with the application of roadheaders, continuous miners, and tunnel boring machines. The drill-and-blast method is the most flexible system, relatively insensitive to changing rock conditions. The cross section of an opening may be changed in one step, or gradually, as may be needed in intersections and junctions of different size openings. Explosives remain the most energy-efficient means of excavating rock. Most of the energy available is utilized for rock breakage, and heat losses are minimum. The power factor per unit mass of rock obtained is 0.75 to 1.5 kWh (1 to 2 hp-hr)/ton of rock. By comparison with the most efficient for mechanical methods—a TBM with disk cutters—the corresponding factor is 2.25 to 3.75 kWh (3 to 5 hp-hr)/ton of moderately hard rock. Another advantage of the drill-and-blast method is that the capital cost is lower, by a factor of 3.5, as compared to TBMs.

Disadvantages of the drill-and-blast method are associated with the necessity for sequential drilling, blasting, mucking, and supporting of the excavation. The cyclic nature of that system is its limitation, especially with a single heading in which equipment and personnel not required for a particular operation at the face must remain idle. Another negative aspect is the hazard of handling explosives. A crew possessing higher skills is needed as compared with the TBM method, and the working environment is more physically demanding and hazardous. Finally, that part of the explosive energy that goes beyond the planned contour of the excavation often weakens the rock and produces overbreaks. This requires potentially hazardous scaling and increases the cost of support.

Depending on the face area, and whether it is uniform or mixed face (different rock strata in the face), full-face advance or in stages (with benches) is chosen. Drilling of blastholes for a larger face area is done with multiboom drilling jumbos, mounted on crawlers or on a rail carriage. For smaller cross sections, small jumbo or leg-mounted drills may be the optimum solution. Equipment selection is based on the minimum cost principle for the planned rate of excavation advance. Drilling patterns for different shapes of excavations are shown in Figs.

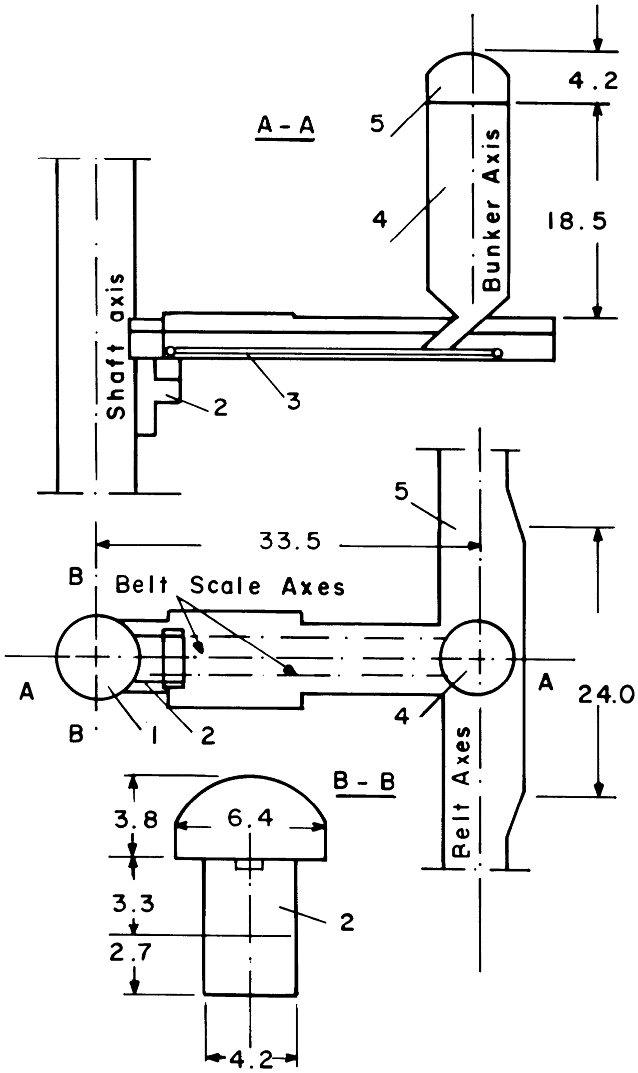


Fig. 17.4.80. Connection of production skip shaft with the opening of loading devices for horizontal belt transport system (elastic system with vertical retaining bunker).

1. skip shaft
2. skip chamber
3. belt scale
4. retaining bunker
5. unloading chamber.

Dimensions in m. Conversion factor: 1 ft = 0.3048 m.

17.4.87 through 17.4.89. V cut patterns are usually used in drifts of small face area, where only small drilling equipment is applicable. In larger drifts and tunnels, burn cuts are preferred. This requires larger equipment capable of drilling large-diameter parallel holes, which are left unloaded, providing additional free surface and thus increasing blasting effectiveness. There are a variety of drilling patterns, which are individually designed for the size of the excavation, the type of rock at the face, and the drifting technology applied. The planned contour of excavation should be followed as closely as possible, avoiding overbreaks. Therefore, a cumulative effect of blastholes should ideally reach this same surface, resulting in the designed size of excavation. There are also smoothwall blasting techniques that are based on adequate sizes of contour holes (smaller than other holes) with smaller charges. However, they require more precise drilling, affecting the rate of advance; therefore, this method is not as popular. Consumption of explosives per unit volume of solid rock varies substantially, depending on tunnel or adit size and rock strength. A similar relationship exists for the required number of holes per round. In other words, the stronger the rock, the more holes and more blasting material is needed per round. With other factors constant, for a larger face area, fewer holes are needed per unit area.

Mechanical systems of face advance are based on roadheaders or full-face borers continuously cutting the face. Technological improvements resulting in system availability and durability of cutters are gradually increasing possible applications of those systems for hard rock (Figs. 17.4.90 and 17.4.91). New-generation TBMs can cut even the strongest rock and successfully compete with drill-and-blast methods in terms of rate of advance. As in softer rock such as shales and weak sandstone, a mechanical system based on roadheaders has a clear advantage over conventional drill-and-blast methods. In a rock with moderate to high strength, careful evaluation is required of the advantages and disadvantages of both systems in order to make the proper choice. The capital cost of roadheaders and especially TBMs is high and can only be recovered over a considerable length of tunneling. Because of the very high cost of TBMs, they are seldom seen in US mining applications, whereas they are commonly used in civil construction. In coal mines of western Germany and England, main transportation tunnels have been successfully driven with TBMs (Table 17.4.13). They produce a smooth surface on the excavation, which is inherently more stable than those produced by blasting.

While TBMs have a full-face cutting head and thus produce circular openings, roadheaders mine the face with the cutterhead mounted on a hydraulically controlled boom. Therefore, any shape of cross section of the opening can be obtained. Within the new generation of roadheaders, there is a tendency towards heavier machines equipped with stiff box-type cutter booms supported by forward mounted cylinders. They operate with high torque and low RPM and are more powerful than the previous generation of machines. The roadheader principle of operation is based on the application of drag bits, which break rock in the shear and tensile modes, thus taking advantage of rock characteristics of least fragmentation resistance. The heaviest roadheaders equipped with special bits (picks) can cut rock up to 30,000 psi (200 MPa) (Trehelles et al., 1983). Roadheaders are very popular in Europe where mine development openings are usually made in rock. For instance, in Germany the average annual advance rate of roadheaders is more than twice that of conventional drill and blast headings at 10 to 15% lower cost per meter (Boldt, 1988). Two main types of cutterheads are in use, the in-line (milling) heads rotating on axis parallel to the boom and transverse (ripper) cutterheads. The latter work better in hard rock, while machines with in-line heads achieve a higher cutting rate in soft rock.

Bit consumption and cost depend on rock properties; cutter bit costs range for soft and nonabrasive materials up to $4/yd^3 ($5.20/m^3) (Kogelman, 1989). Roadheader production rate vs. rock strength is given in Fig. 17.4.91.

In the United States, the continuous miner is the most popular member of the rapid excavation family, because flat-lying seams in coal mines allow in-seam mine development. If possible to apply, this is the most economical development system, with the continuous miner applied in both development and production faces. In a weak-rock formation such as coal, shale, and salt, the continuous miner with drum cutting head is probably the most refined and most productive machine, widely applied in coal mines in North America, Australia, and South Africa.

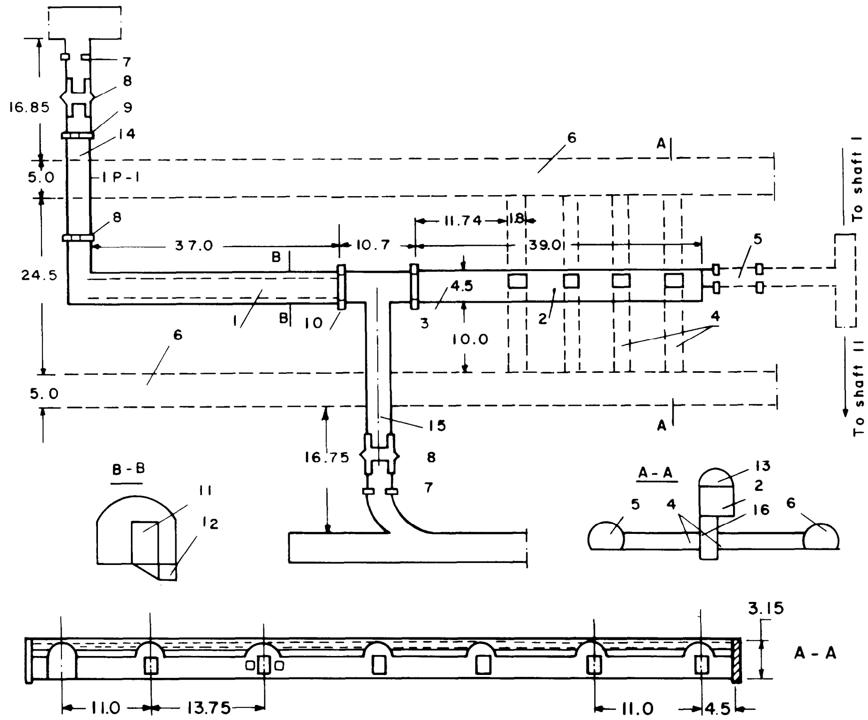


Fig. 17.4.81. Main dewatering and power distribution for 6-kV chambers.

1. power distribution chamber
2. pump station chamber
3. mechanical shop
4. drift to sumps
5. incline (pipe) drift to shafts
6. water retaining reservoir (drifts)
7. safety lock
8. water drain (with doors)
9. ventilation stopping
10. steel doors
11. switch and control panel
12. cable channel
13. girder crane
14. walk in drift
15. delivery drift
16. water sump.

Dimensions in m. Conversion factor: 1 ft = 0.3048 m.

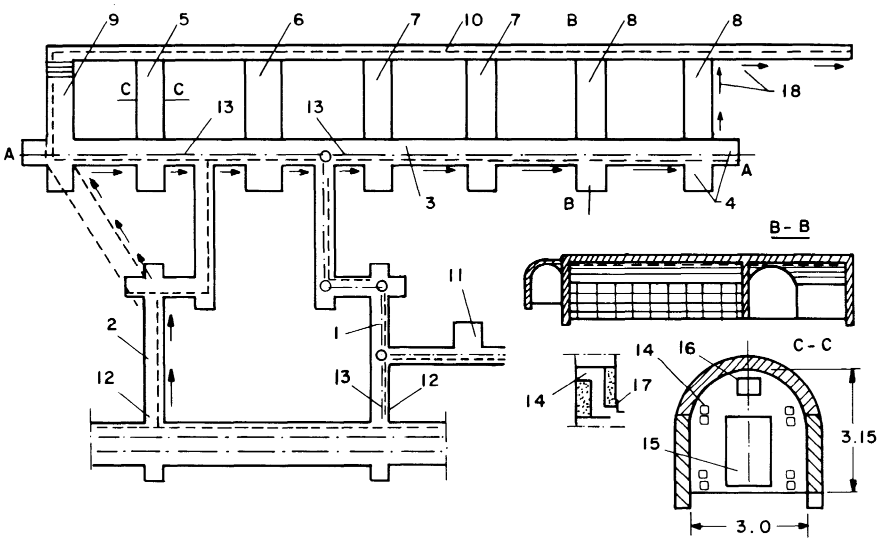


Fig. 17.4.82. Storage of explosives.

1. entrance drift
2. outlet drift
3. storage drift
4. buffer niche
5. caps chamber
6. handling-over chamber
7,8,9. chambers for different types of explosives
10. ventilation drift
11. niche for empty boxes
12. safety door
13. grate door
14. outlet ventilation hole
15. steel doors
16. light
17. screen 10 × 10 mm (0.4 × 0.4 in.)
18. airflow (ventilation).

Dimensions in m. Conversion factor: 1 ft = 0.3048 m.

MUCKING. A variety of possible configurations is available depending on the drifting system. Generally, side-casting or overshot loaders with track or tire undercarriage are used. In large tunnels, front-end loaders, arm loaders with flying conveyors, or backhoe shovels can be applied. Depending on equipment configuration, they discharge either to rail cars or use some sort of intermediate system, such as a hanging conveyor or shuttle car side casting to the rail cars. Beyond the face, muck transport in the tunnel can be either by rail cars or conveyor belt. The most practical and economical method is to apply the system that will be in service after the development stage is completed, during the exploitation stage of the mine (see Chapters 9.3 and 9.4). Some examples of equipment configuration are given in Figs. 17.4.92 and 17.4.93.

Support of the excavation has to ensure safety of the crew and the equipment. It is selected on the basis of rock conditions, technology of face advance, future function of the drift/tunnel, and existing regulations. In self-supporting rocks, only minimum support is needed, such as rock bolts, and even this is sometimes mainly to satisfy safety regulations. On the other hand, in unstable rocks, temporary support has to be installed at a minimum distance to the face to shorten delay time, and a permanent support of adequate strength must be erected following the face advance. Table 17.4.13 lists the support systems used for development drifts in reference to the engineering MBR classification. Clearly, the required type of support affects the equipment needed during the face advance. With the drill-and-blast method, the support is based on rock bolts, and if required, shotcrete will require a minimum number of machines in the face. If the support of steel arches is needed, the roadheader is usually employed where its boom yoke is used for erection of a roof arch. When rock pressure conditions make the installation of additional support necessary, it is erected at some distance to the face where it does not interfere with face activities. For instance, the initial

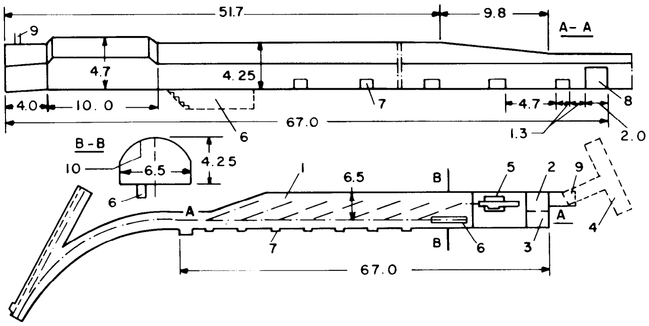


Fig. 17.4.83. Locomotive depot for trolley locomotives.
1. parking area
2. lubricating chamber
3. spare parts warehouse
4. ventilation drift
5. canal for jack lift
6. inspection canal
7. switch niche
8. sand niche
9. ventilation hole
10. trolley.

Dimensions in m. Conversion factor: 1 ft = 0.3048 m.

support (rock bolts or arches) can be shotcreted over or a structural concrete lining (with application of form) can even be poured for more difficult conditions.

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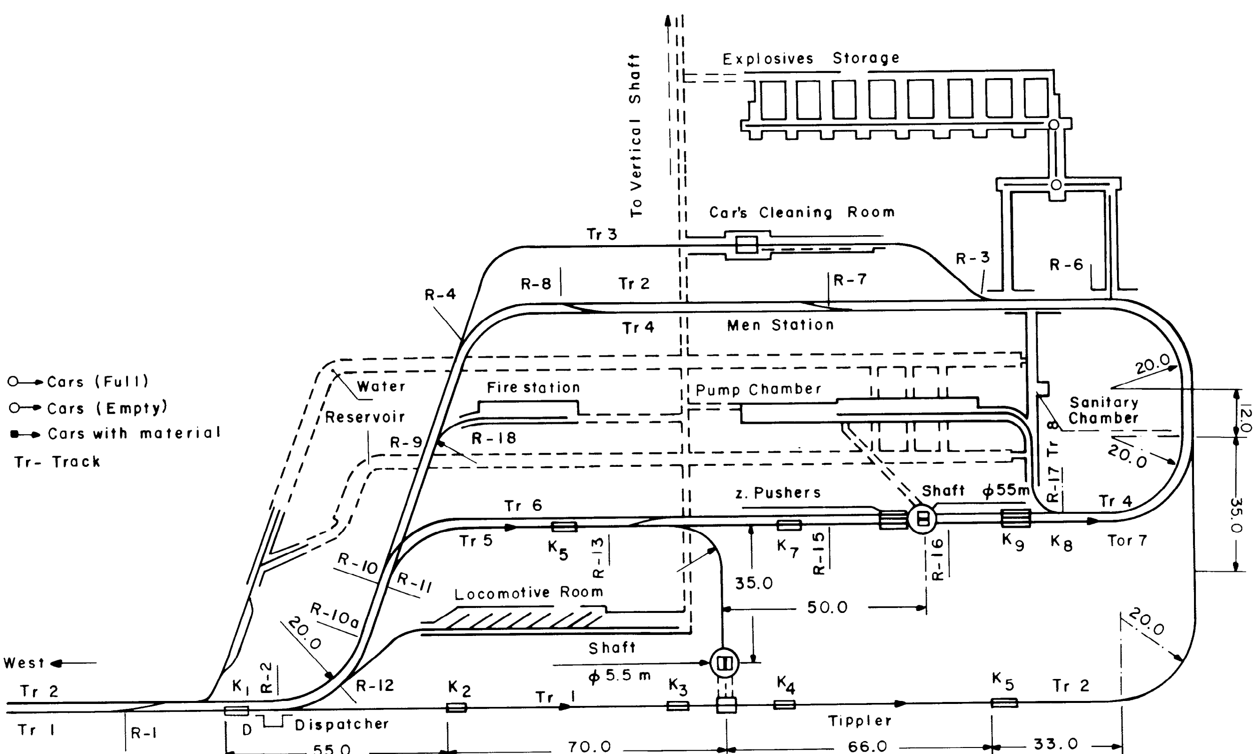

Fig. 17.4.84. Parallel side shaft station with 3000- to 6000-t/d (3300- to 6700-tpd) capacity. K_1 to KG—chain drives. R-1–R-18—rail switches. Conversion factor: 1 ft = 0.3048 m.

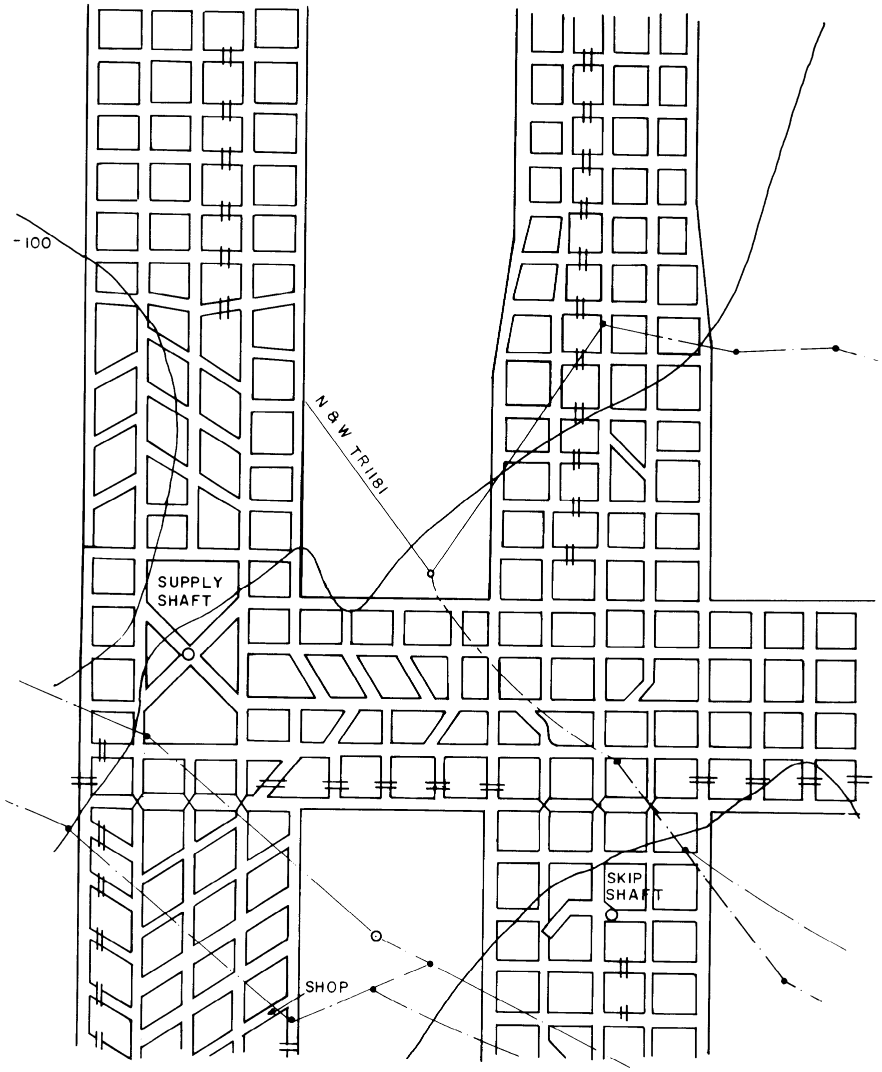

Fig. 17.4.85. Example of the shaft station in a coal mine operating in a horizontal seam.

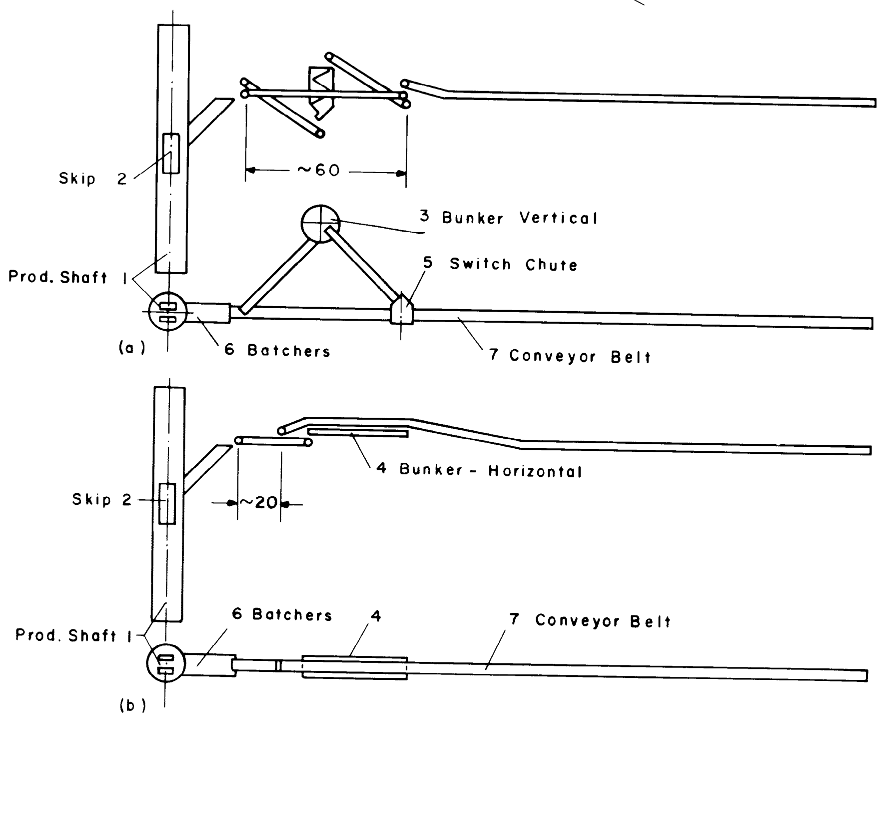

Fig. 17.4.86. Shaft station with conveyor belt transport system and skips.

a) version with permanent vertical retention bunker
b) version with horizontal retention bunker.

Dimensions in m. Conversion factor: 1 ft = 0.3048 m.

Table 17.4.13. Examples of Mining Applications of Tunnel Boring Machines (TBM)

Model No./ Type	Boring Period	Project Name and Location	Rock Description: Type	Rock Description: Strength 1000 psi (MPa)	Tunnel Length ft (m)	Machine Diameter ft-in. (m)	Cutterhead hp (kW)	Thrust lb (kg)	Torque lb-ft (kgm)	Weight tons/t	Cutting Discs Face and Gage Center
202-201 Hard Rock Rotary	1979	Westfalen Mine RR German Federal Republic	Shale, Sandstone	5–20 (35–140)	41,670 (12,700)	20-0 (6.1)	720 (537)	1,400,000 (635.040)	950,000 (131.385)	220 (200)	39-14″ 4–12″
193–214 Hard Rock Rotary	1981–88	Selby Mine Road Selby, U.K.	Sandstone, Mud-stone, Coal, Siltstone	7–21 (500–1476) (50–148)	44,620 (13,600)	19-0 (5.8)	900 (67.1)	1,274,000 (578.0400)	943,000 (130.400)	263 (240)	36-14″ 6–13″

Source: After Robbins Co., Kent, WA.

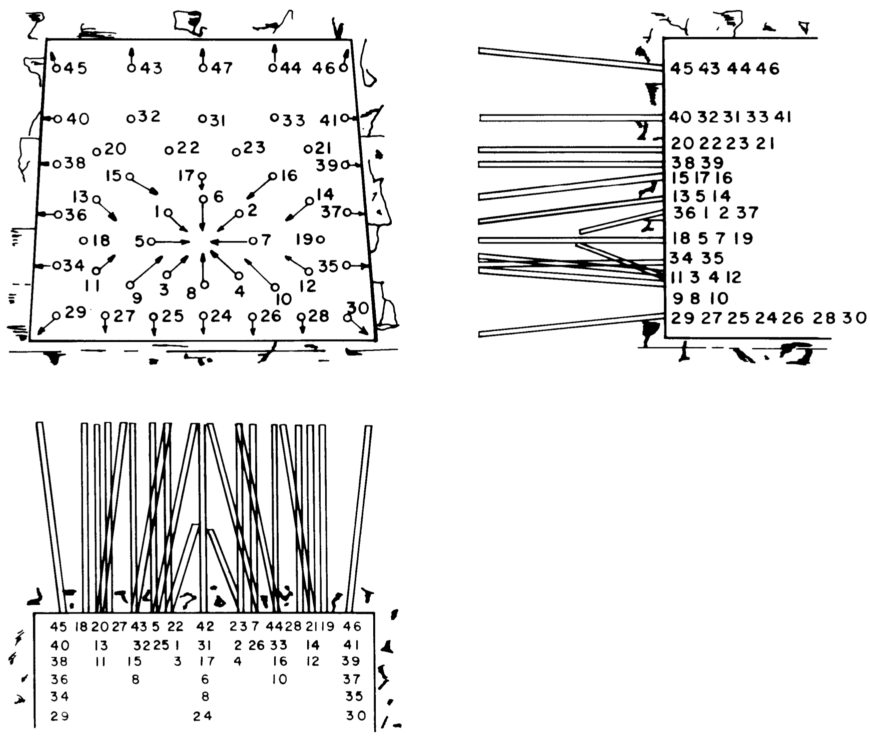

Fig. 17.4.87. Drilling pattern, cone cut, with drillhole numbers.

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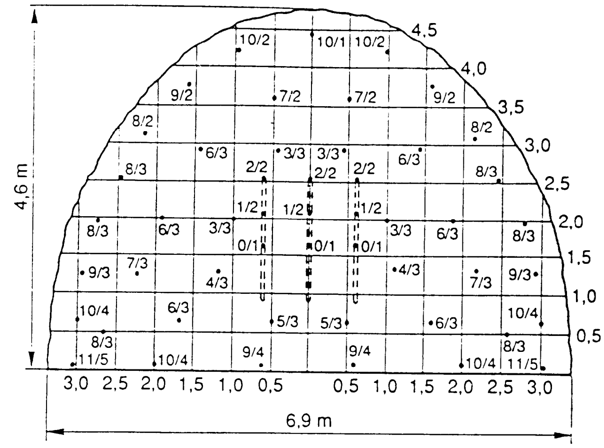

Fig. 17.4.88. Shot-firing pattern in conventional blasting system. Conversion factor: 1 ft = 0.3048 m.

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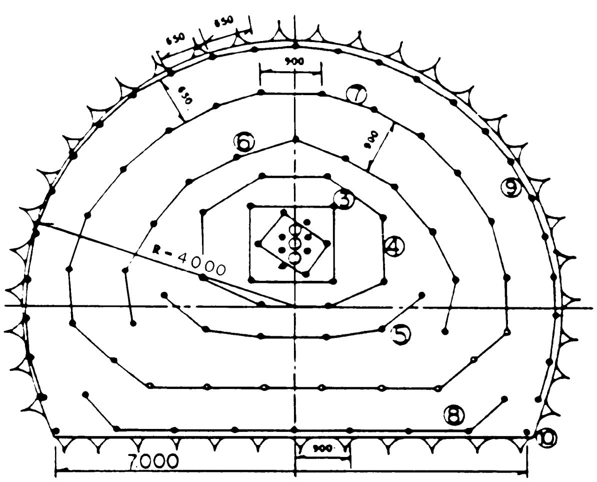

Detonator		Number of holes	Quantity of Charge	
			Per hole (kg)	Total (kg)
MS	1	1	3.00	3.00
	3	1	3.00	3.00
	5	1	3.00	3.00
	7	1	3.00	3.00
	9	2	3.00	6.00
	12	4	3.00	12.00
DS	3	4	3.55	14.20
	4	5	3.55	17.75
	5	6	3.55	21.30
	6	10	3.55	35.50
	7	20	3.55	71.00
	8	9	3.55	31.95
	9	25	1.60	40.00
	10	2	3.55	7.10
Sum Total		91		268.80 kg

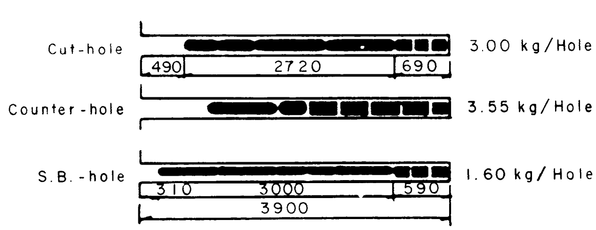

Fig. 17.4.89. Application of burn holes.
a) blasting patterns for top heading
b) number of blastings for headrace tunnel by drilled length.
Conversion factors: 1 ft = 0.3048 m, 1 lb = 0.4536 kg.

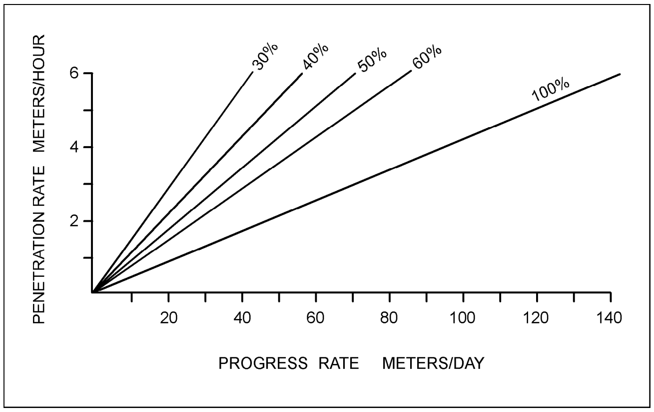

Fig. 17.4.90. Tunneling machine utilization. Conversion factor: 1 ft = 0.3048 m.

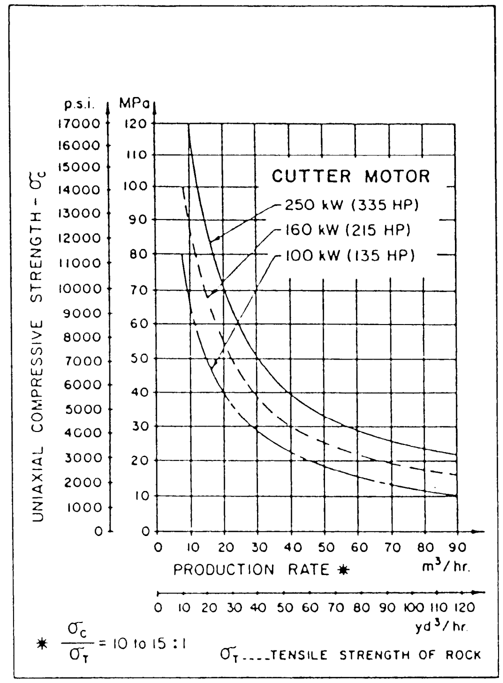

Fig. 17.4.91. Roadheader production vs. strength of rock.

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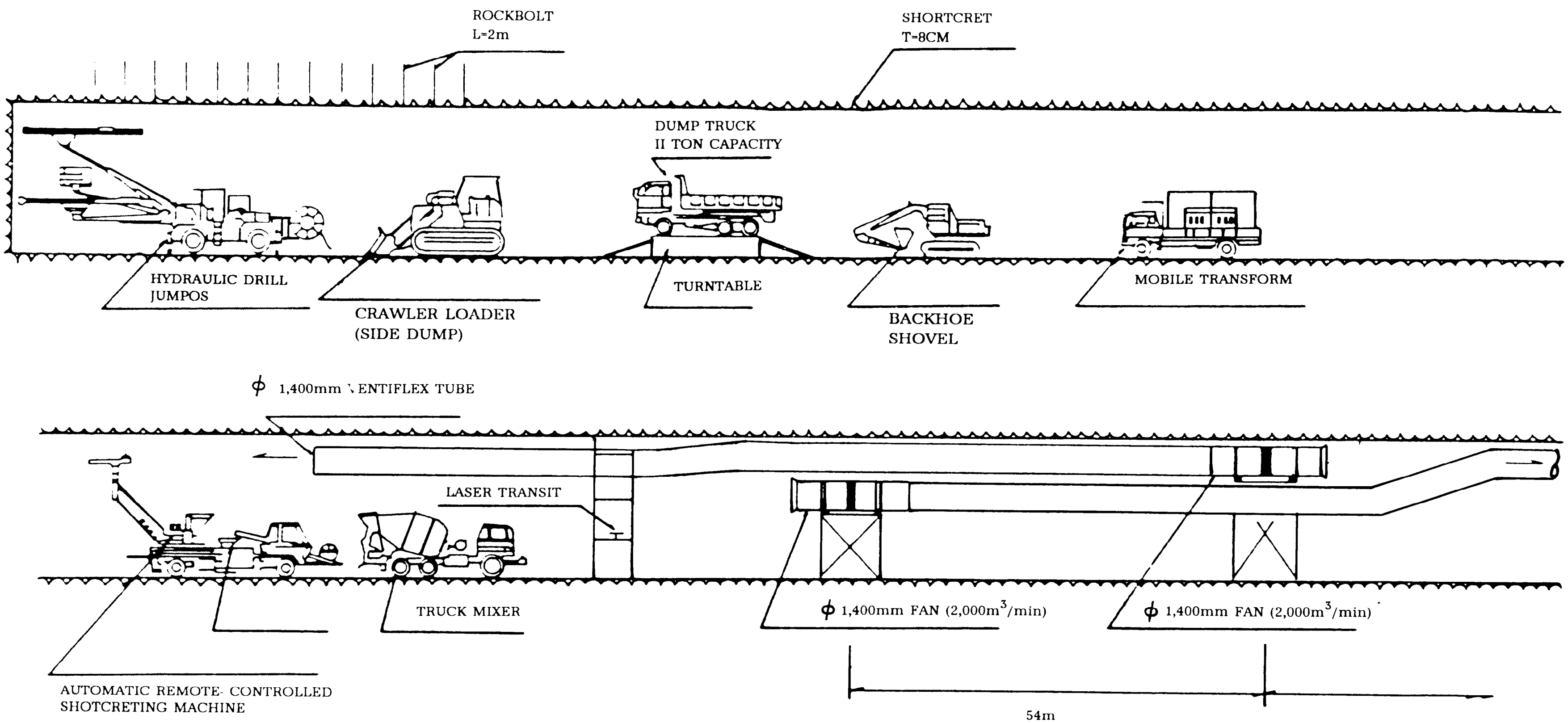


Fig. 17.4.92. Excavation procedure for top heading. Conversion factor: 1 ft = 0.3048 m.

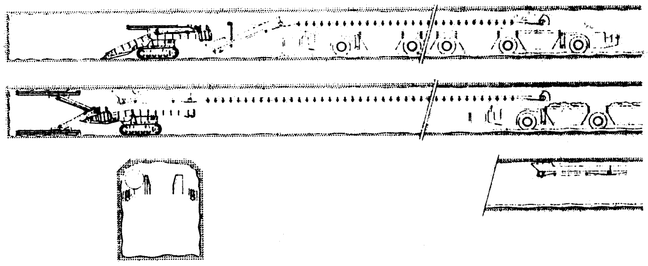


Fig. 17.4.93. Schematic of face configuration in drill and blast method.

Chapter 17.5
HOISTING SYSTEMS

FRED A. EDWARDS

17.5.1 INTRODUCTION

After a detailed technical feasibility study has indicated the commercial viability of exploiting an ore body using underground methods, an adequate planning and design process for mine access should begin. Two general types of access should be considered: (1) vertical to near-vertical shafts using hoists and cable-suspended conveyances, or (2) horizontal or inclined openings using rail, trucks, conveyors, or cable-operated conveyances.

Because a shaft often provides the most direct access over the longest period of time, there is an advantage to designing a shaft for maximum duty, consistent with economy. The current trend in shaft design is to provide multipurpose shafts. These shafts contain facilities for handling ore, waste, materials, personnel, services, manways, and ventilation.

During the process of identifying the purpose of the shaft, it should be realized that once a shaft is excavated and equipped, it cannot be enlarged easily in the future. Therefore, the shaft's initial and ultimate requirements must be defined during the design phase.

This chapter presents the information and criteria necessary to design or select a mine shaft hoisting system. Specific topics such as hoist selection, hoist rope details, headframe design, shaft furnishings and equipment, as well as shaft lining design, are included in the discussion. Ramps and tunnels are dealt with in a similar fashion in Chapters 17.4 and 24.1.

A *system* is defined as a group of units so combined as to form a whole and to operate in unison. Thus, when designing a shaft hoisting system, one must consider not only each part of the system separately but the interrelationships among the parts of the system in its entirety. For the purpose of this discussion, the shaft hoisting system has been divided into five main components: (1) hoist, (2) conveyance, (3) rope, (4) shaft, and (5) headframe (Fig. 17.5.1).

Following a description of each of the five main hoisting components, a segment describing a systematic procedure that can be used to design a shaft hoisting system is presented.

17.5.1.1 Design Parameters for Evaluating Alternative Modes of Access

The first step in evaluating an access system for a mine is to determine the principal purpose of the opening. The opening could be used for production, service, ventilation, exploration, development, or a combination of these. Next, duty requirements for the opening must be established. Questions include: how much ore and waste? how many personnel? how much material and supplies? how much air?

With this information, the basic design parameters, including the size of the opening, configuration, ground support, and inclination can be developed. This allows preliminary designs and comparative capital and operating cost estimates to be made between alternative systems.

Detailed cost analysis for the mine access cannot be carried out without additional design consideration involving the following items: (1) geology of the ground to be excavated, (2) strike, dip, plunge and depth of the ore body, and (3) hydrogeology of the strata to be penetrated. With the above information, the engineer is in a position to prepare conceptual designs and order-of-magnitude cost estimates.

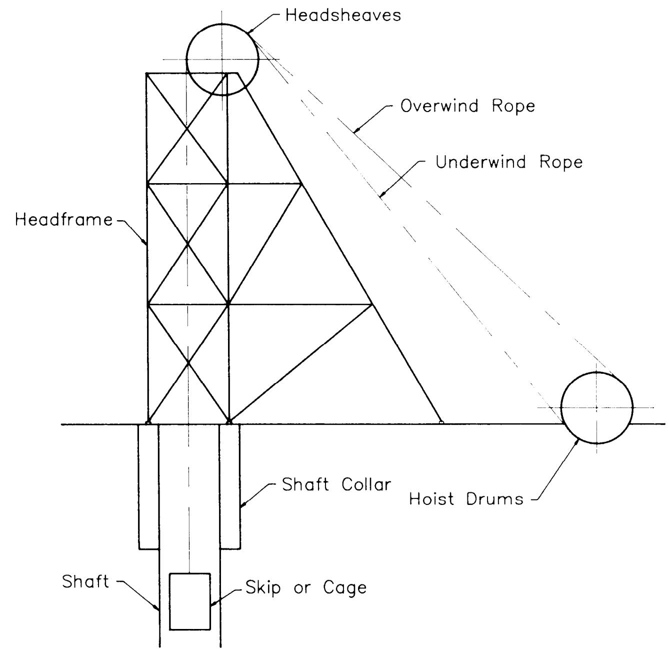


Fig. 17.5.1. Shaft hoisting system.

Outside influences on the cost of any new project must be taken into account. These are (1) the geographic location of the project, affecting cost of materials; (2) climatic conditions at the site, especially heavy rainfall, snow, extreme cold and high winds; and (3) availability of skilled manpower to carry out the work, noting labor relations in the area. While these items do not generally affect the design considerations, they affect the development costs for the access system.

After the above terms have been defined, trial calculations and alternate designs can be considered. During this process, the interrelationships among the major components of the system can be examined. Each trial design must be evaluated to ensure that it meets the design objectives. These objectives are concerned with capital costs, operating costs, dependability, efficiency, flexibility, suitability to the mining plan, and time to construct. This procedure of defining, examining alternate designs, and conducting evaluations is continued until an optimum solution is reached.

17.5.2 SHAFT HOISTING SYSTEMS

In this chapter, the major components associated with vertical to near-vertical shafts using hoists and cable-suspended conveyances are described briefly. The term shaft hoisting system is used to describe collectively the openings and the equipment being considered.

In addition to the five major hoisting components, Edwards (1988) has identified an additional 277 subcomponents. The number of subcomponents and their interrelationship with the main components are indicative of the complexity involved with the design of shaft hoisting systems.

A brief description of each of the above main components is now presented. In the following segments, information on the design and technical considerations to be examined when selecting a particular component is presented in more detail.

17.5.2.1 Hoists

There are two basic types of hoists in common use today. These are the *drum hoist* in which the hoist rope is stored on the drum, and the *friction hoist* in which the rope passes over the wheel during the hoisting cycle. Within each category there are several variations.

Drum hoists are usually located at some distance from the shaft and require a headframe and sheaves to center the hoisting ropes in the shaft compartment. Friction hoists may also be located directly over the shaft and, depending upon the wheel diameter, may require deflection sheaves to center the rope in the shaft compartment.

17.5.2.2 Conveyances

Conveyances used in mining operations are classified according to their use. Those for handling personnel and material are generally termed cages. Conveyances for handling broken ore or coal and waste are termed skips. Combination skip-cages are used in some areas. A counterweight may also be considered a conveyance.

17.5.2.3 Rope

When considering the complete shaft hoisting system, there are three common uses for steel ropes, with a particular construction applicable for each use. A list of ropes by use and most common construction follows:

Rope Use	Rope Construction
1. Hoist rope	Round strand
	Flattened strand
	Locked coil
2. Balance rope	Non-rotating
3. Guide and rubbing rope	Half-locked coil

17.5.2.4 Shaft

Brucker (1975) has provided two definitions that can be used to describe shafts for mining purposes: (1) a vertical, deep, restricted cross-section excavation, and (2) a vertical or inclined primary opening in rock that gives access to and serves various levels of a mine.

Primary openings can be further defined as those that are considered to be permanent and require a high degree of safety. Although the above definitions can be used to describe any type of shaft, they are somewhat general and do not provide sufficient information to allow the design and construction of a shaft as a useful structure for mining purposes.

There are several classifications that can be used to differentiate shafts by type. For the purpose of this discussion, four commonly used classifications are presented. Shafts can be classified (1) by purpose, (2) by configuration, (3) by ground support, and (4) by excavation method.

SHAFT PURPOSE. One of the first items to be examined when designing a shaft is to identify its intended purpose. When defined by purpose, shafts usually fall into the following categories:

1. Production: ore and waste handling.
2. Service: personnel and materials handling.
3. Ventilation: upcast or downcast airflow.
4. Exploration: for defining mineral deposits.
5. Escape: for emergency.
6. Combinations of the above.

SHAFT CONFIGURATION. Shafts may be classified according to their size and configuration. The most common shaft configurations for any size shaft are circular, rectangular, and elliptical. Size of a shaft can be small (32 to 160 ft^2, or 3 to 15 m^2), medium (160 to 2150 ft^2, or 15 to 200 m^2), or large (> 2150 ft^2, or 200 m^2), depending upon its service.

SHAFT GROUND SUPPORT REQUIREMENTS. Shafts may be classified according to the type of ground support to be used to maintain the stability of the shaft structure, for example, a timber shaft or concrete-lined shaft. Ground support applied to the shaft structure can be classified as temporary support or permanent support. Ground support requirements are thoroughly discussed in Chapter 17.4.

SHAFT EXCAVATION METHODS. Shafts may be classified by the excavation method used during construction. There are two types of excavation methods: conventional and bored. *Conventional shafts* are excavated with standard drilling, blasting, and mucking methods, in combination with various methods of ground support. They may be of any configuration or inclination. *Bored shafts* are excavated using a mechanical shaft boring machine (SBM). Several boring methods are available. Most methods remove cuttings from the collar of the hole. Some methods drop cuttings from the machine into the mine. All bored shafts are circular in configuration, with various methods of ground support. They may be of any inclination. Shaft excavation techniques are discussed in Chapter 17.4 and Section 9.

17.5.2.5 Headframe

Headframes may be constructed of wood, steel, or concrete. They are generally divided into two types:

A. With backlegs
   1. A-frame
   2. Four-post
   3. Six-post
   4. Other
B. Tower
   1. Ground-mounted hoist
   2. Tower-mounted hoist

17.5.3 HOISTS

Generally, the mine hoist selected and installed at a mine remains in position for the life of the operation. It is therefore imperative that the proper hoist be selected. In order to select the proper hoist, it is important to understand the basic design parameters, the various types of hoists available, and the relationship between the mine hoist and the other components of the hoisting system. Each major type of hoist is now considered separately (Fig. 17.5.2).

17.5.3.1 Types of Hoist

SINGLE DRUM. The single drum hoist may be used for balanced or unbalanced operation. When used for *unbalanced*

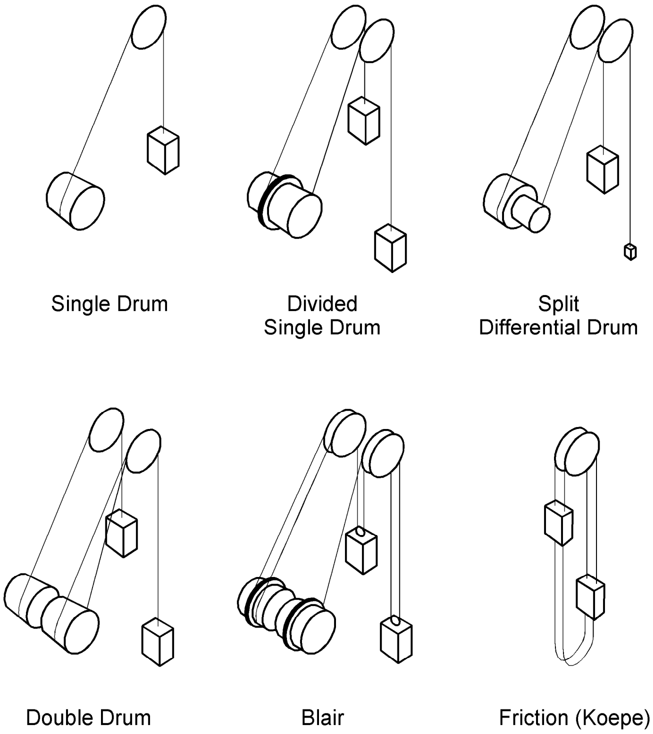


Fig. 17.5.2. Arrangements of various hoists.

hoisting, the cost of the electric drive becomes quite high for long hoisting distances and high tonnages. This is because the motor must have sufficient torque to handle the weight of the rope, conveyance, and payload.

In a *balanced hoisting system,* one rope winds off the drum as the other winds on. When used with a skip or cage in balance with a counterweight, a single drum hoist can service one or more levels since the location of the counterweight is not important. When used with two skips in balance, the single drum hoist is best used for single-level hoisting. Any rope adjustments to locate the conveyance must be done manually. For shallow shafts with one layer of rope, no dividing of the drum is required. For deeper shafts, the drum must be divided.

DIVIDED SINGLE DRUM. This type of hoist is used for deeper shafts with balanced hoisting when several layers of rope must be stored on the drum. Peak horsepower is less than with unbalanced hoisting because the skip weights are balanced. Since the payload and weight of the rope is not balanced, the maximum unbalanced load occurs when the loaded conveyance is at the bottom of the shaft.

SPLIT DIFFERENTIAL DIAMETER DRUM. The third type of single drum hoist available is the split differential diameter drum. This type of hoist is used with a conveyance and counterweight in balance. If the counterweight is wound on the smaller diameter drum, it moves less than the main conveyance, and rope adjustment problems are reduced.

DOUBLE DRUM, ONE DRUM CLUTCHED. Although more expensive than a single drum hoist, the double drum hoist with one drum clutched has certain advantages. With this type of hoist, it is possible to make quick adjustments to the ropes due to initial stretch. As a service hoist with cage and counterweight, this type of hoist can serve several levels efficiently. As a production hoist with two skips, the ropes can be adjusted to maintain balanced hoisting at any level in a multilevel operation.

DOUBLE DRUM, BOTH DRUMS CLUTCHED. The double drum hoist with both drums clutched has the added feature of allowing hoisting to continue in one compartment should something happen to the other compartment. This is an excellent feature if there is only one hoist available. This type of hoist is also favored during shaft sinking operations.

MULTIPLE DRUM, BLAIR TYPE. With this type of hoist, each conveyance is suspended from two hoist ropes that are each coiled on a drum. The advantage of this is that smaller diameter ropes and drums may be used. This type of hoist was developed in South Africa. There are no Blair-type hoists in use in North America. However, for very deep mines, they should be considered.

FRICTION HOIST, SINGLE ROPE, AND MULTIROPE. The Koepe or friction hoist was developed by Frederick Koepe in 1877. It consists of a wheel with a groove lined with friction material to resist slippage. The hoist rope is not attached or stored on the wheel. In early installations, the hoist was mounted on the ground, and a single rope was wound around the drum and over the headsheaves to the conveyances, in a balanced arrangement. In addition, a tailrope of the same weight per unit length as the headrope was suspended in the shaft below each conveyance. Thus the only out-of-balance load was the payload.

As hoisting loads became larger, the number of headropes and headsheaves increased to the point where it became more practical to install the hoist in the headframe directly over the shaft. In North America, many friction hoists are mounted in this way. In order to bring the rope centers in line with the compartment centers, deflection sheaves must also be installed in the headframe below the hoist.

17.5.3.2 Comparison of Friction and Drum Hoists

A friction hoist system differs from a drum hoisting system in performance as well as components. Therefore, when attempting to decide which type of hoist to use, it is necessary to compare the two complete systems rather than the two hoists alone. In addition to comparing the total capital costs of the hoist, headframe, ropes, conveyances, and shaft, it is necessary to consider operating costs, maintenance costs, reliability, power supply system, local custom, and individual preference.

Brucker (1975), Schulz (1973) and Tudhope (1973), among others, have discussed drum hoist and friction hoist applications. The following general statements help distinguish between these two hoisting systems: (1) double-drum hoists are the preferred hoist for shaft sinking; (2) double-drum hoists are the best choice for hoisting in two compartments from several levels; (3) drum-type hoists are best suited for high payloads from shallow depths; (4) the limitation on a drum hoist employing a single rope is the ultimate strength of the rope, because large ropes are difficult to manufacture and handle; (5) the depth capacity of drum hoists can be extended by using two ropes per conveyance (Blair-type hoist), and with this arrangement, Blair hoists can be used for depths exceeding those of either single-rope drum hoists or friction hoists); (6) friction hoists with multiple ropes can carry a higher payload and have a higher output in tons per hour than drum hoists within a range of depths from 1500 to 5000 ft (460 to 1520 m); (7) friction hoist mechanical operation is very simple, has a low rotational inertia, and is less costly than a drum hoist; (8) friction hoists have a lower peak power demand than drum hoists with the same output; and (9) the friction hoist can operate on a relatively light power supply.

17.5.3.3 Hoist Component Considerations

When selecting (or comparing) hoists, the components to be considered during the evaluation process include (1) drum, (2) bearings, (3) gearing, (4) brakes, (5) drive motor, and (6) control.

DRUM. For drum hoists, the drum must be designed to store the required length of rope, meet the statutory requirements concerning fleet angles and rope ratios, and must be sufficiently strong to withstand bending and crushing forces. The design of drums is well documented (Atkinson, 1973) but beyond the scope of this *Handbook.*

Drums may be plain faced or grooved. With plain-faced drums, a section of the drum is unavailable for carrying live turns of rope. However, with grooved drums, the entire face can be utilized.

There are three types of drum grooving available: (1) helical, (2) parallel, and (3) Le Bus (or antisynchronous). *Helical grooving* is a continuous spiral providing smooth winding with a single layer of rope. It can also be used with multilayer winding. *Parallel grooving* is made up of individual grooves evenly spaced over the drum width. This grooving is used for single layer winding only. *Le Bus grooving* is a combination of parallel and helical. It is deeper than helical and parallel grooves, thus providing better support. It is most suited for multilayer winding as it reduces rope whip at the crossover points.

For friction hoists, the drum must be sized to meet statutory requirements for rope-to-drum ratios and must be wide enough to carry the required number of ropes.

The tread of a friction hoist drum is lined with a friction material to resist slippage. In the past, this material was wood or leather. At present, polyurethane, PVC, or combination blocks are used.

BEARINGS. When there is a relative motion between two members of a machine, one of which supports the other, the supporting member is called a bearing. Bearings are classified into two general types: sliding bearings and roller bearings. *Sliding bearings* are those in which the surfaces are in sliding contact, and the supported member runs on a cylindrical, conical, or flat surface. *Roller bearings* have surfaces that are in rolling contact, and the supported member runs on hardened steel balls or rollers.

In some hoists, sliding bearings lined with babbitt are used for the hoist drum shaft, pinions, and headsheaves. During operation, the bearing surfaces are separated by an unbroken film of oil. Although sliding bearings are, in principle, the perfect type of bearing, they are being replaced by roller bearings. The advantages of roller bearings are their low coefficient of friction, economy of space, simplicity of lubrication, practical elimination of wear due to roller point contact, maintenance of accurate alignment, and their consistency in design and manufacture. For a hoist, roller bearings having a rated life of 300,000 hours are generally selected.

GEARING. The hoist may be driven by either ac or dc electric motors. Depending upon the hoisting speed, these motors may be connected to the shaft directly or through a gear drive. Low-speed motors may be connected directly; high-speed motors require a gear reducer drive.

With high-speed motors (300 to 900 rpm), power is transmitted from the motor to the hoist through coupling-connected single or multiple-stop, helical gear reduction drives. Beerkircher (1975) recognizes the following as consideration for the design and selection of a suitable gear drive: (1) size, type, speed, location, and number of motors; (2) size, type, speed, torque requirements, and location of hoist; (3) type of operating cycle; and (4) physical restrictions.

BRAKES. The braking system is required to decelerate, stop, and hold the hoist drum. This may be accomplished using electrical and mechanical braking systems, which must operate both under normal and emergency conditions.

Electric braking may be accomplished through regenerative braking, counter-torque braking, or dynamic braking. In regenerative breaking, the motor, when connected to a hoist operating on an overhauling load, performs as an induction generator, developing braking torque and returning energy to the system. With counter-torque and dynamic breaking, there is no energy returned to the system; instead, it is consumed and dissipated in the secondary resistance as heat.

When electric braking is used, and during normal operating conditions, the *mechanical brake* serves two purposes. Firstly, it helps to slow down the drum after the speed has been greatly reduced electrically, and secondly, it holds the drum at rest. Under these conditions, the mechanical brake does little work, and its design and operating characteristics can be fairly simple and straightforward. Under emergency conditions, however, the hoist must be stopped with the mechanical brake as quickly as possible, without damaging the hoist, brake, or conveyances.

In North America, there are three main types of brakes used: (1) the jaw, (2) the parallel motion, and (3) disk brakes. Disk brakes are used mainly on friction hoists, whereas jaw and parallel motion brakes may be used on both drum and friction hoists.

With the *"caliper-type" brakes,* two shoes are pressed against the periphery of the brake drum through the pull of a single or double draw bar acting through a series of linkages. The source of force can be a brake weight or a nest of springs. When not in use, the weight or springs are held open by a hydraulic oil cylinder. When the brakes are applied, oil flows from the cylinder to allow the force of the weight or springs to apply the brakes. The control of this flow is extremely important, as it determines the speed and amount of force to be applied.

The use of *disk brakes,* particularly with friction hoists, has increased. Two types of disk brakes are used. In one type, the brake is operated via a lever system and during emergency stops via a falling weight. This is similar to conventional post brake systems. With the other type of system, hydraulically operated, spring applied units are used. These units are mounted around the brake disk. The required number of units is determined by the total braking force required by the particular hoist. The braking control unit is similar to that used in drum hoists. The advantages of disk brakes are their relatively small dimensions, light weight, and ease of replacement.

CLUTCHES. Drum hoists used for unbalanced hoisting, or when operating from several levels, should have at least one of the drums clutch-connected to the drum shaft. With such a hoist, the clutched drum(s) is supported on the shaft through a sleeve bushing and the clutch is used to transfer the motor torque to the drum. In order to prevent the drum from rotating when it is unclutched, it is necessary that the clutch operating mechanism and the drum brake be interlocked.

The clutch is operated through a two-, three-, or four-arm *spider* that is attached to the turning drum shaft. These arms are perpendicular to the shaft and are shifted parallel to the axis of the drum shaft. A clutch ring with matching teeth on its entire periphery is bolted to the inside of the drum, and as the spider is moved, these two sets of teeth are free to engage at any point on the circumference of the drum. Driving torque is then transmitted from the clutch to the drum through the bolts used to hold the clutch ring to the drum flange.

DRIVE MOTOR. Staley (1936) describes four methods of providing power to drive the hoist: (1) electric power, (2) steam, (3) compressed air, and (4) internal combustion engine. Electric power is by far the most common method and is the only one treated here.

Table 17.5.1. Capital and Operating Cost Comparison Between M-G Set and Static Drive

	Type of Drive	
	M-G Set	Static Drive
Initial cost of system	100%	85% to 110%
Installed cost	100%	70% to 90%
Operating cost, kWh/ton	100%	90% to 92%
Power factor	0.8 leading	0.7 lagging
kW demand effect	100%	92%
Estimated maintenance time	100%	70%
Estimated downtime for major repair	100%	30%
Operating characteristics	No appreciable difference	
Technical competence required	100%	100%

Source: Muller, 1978.

The choice of electric motor drives lies between alternating-current motors and direct-current motors, with direct current being supplied from either a motor-generator (MG) set or from a silicon-controlled rectifier (SCR). The correct choice for a particular application is complex and requires advice from persons expert in this field. The principal factors to be considered in selecting a motor are the mechanical configuration, electric power system, and economics.

The ac motor has the advantage of having the lowest initial cost and can use the power normally supplied. However, it has a higher starting torque and is difficult to automate. The ac motor generally requires a gear train to drive the hoist.

The dc motor provides accurate and sensitive control, is easily automated, and has a lower starting torque. The dc motor may be directly connected to the hoist, thus eliminating the gear drive and the space requirements of such a drive.

The choice of power-conversion equipment is affected by the quality of the power system available, the ratio of the hoisting load (electrical) to the total plant load, and personal preference. Table 17.5.1 shows an operating and cost comparison of motor-generator sets and static drives for a mine hoist.

HOIST CONTROL. Control systems are required to monitor the speed and location of each conveyance moving in the shaft. Whenever a conveyance exceeds a preselected, safe, speed-distance profile, the control systems initiates an action to prevent the moving conveyance from striking a permanent obstruction regardless of its speed or direction of travel. In addition, control systems may be used to shut down equipment due to high operating temperatures, excess brake wear, rope slip, slack rope, loss of power, etc. Discussion with hoist manufacturers reveals that the control systems for friction hoists are more numerous and complex than for drum hoists.

Because a full discussion of all of the systems (and their variations) is beyond the scope of this presentation, it may be more useful to discuss the factors to be considered when reviewing various systems presented by suppliers. The three most important factors to be considered, according to Eastcott (1977), are (1) reliability, (2) simplicity, and (3) ease of adjustment.

17.5.3.4 Hoist Selection

The size of a hoist is expressed by the drum dimensions, horsepower rating of the motor, and rope pull. The following segments show how these factors are calculated.

Both drum hoists and friction hoists consist of two separate machines, each of which must be designed to produce the desired result. These types of machines are as follows: (1) mechanical (drum, drumshaft, gears, brakes), and (2) electrical (drive motor, controls). The *mechanical portion* of the hoist is designed to support the hoisting rope and its loads. The *electrical portion* of the hoist is designed to provide sufficient torque to turn the drum and raise or lower the rope, conveyance and payload.

In order to design and build a hoist, certain basic duty information is required, namely, (1) hoisting distance, (2) production rate (tons per hour), (3) maximum loads, and (4) types of guides (wood, steel, rope). It can be shown that for a given depth and production rate, there is an optimum load that results in the lowest cost for the hoist.

Due to competition among suppliers and the tendency for owners to purchase on the basis of lowest cost, it is possible that the selected hoist will not have any excess capacity for the drum and motor. This may mean that any change in the future, however slight, could not be accommodated. It is thus important that the purchaser provide the supplier with the following additional information: (1) ultimate use of shaft (men, materials, ventilation, exploration, proposed phasing of expansion); (2) surface plant layout (type of headframe, bin location, etc.); (3) underground layout (levels, station, loading pocket); and (4) type of power distribution system available.

Although the complete detailed design of a hoist is not possible here, the following segments provide sufficient information to allow the reader to determine the best type of hoist for a particular set of conditions, together with some indication as to the power and current consumptions involved.

In order to determine the dimensions, capacity, and size of the mechanical components, the designer must determine certain basic criteria. These include:

1. Hoisting speed, including acceleration, deceleration, and maximum speed.
2. Production rate, in tons per hour.
3. Maximum load to be hoisted.
4. Hoisting distance.
5. Weight of payload and conveyances.
6. Diameter of hoisting rope.

After this information is determined, the designer can then determine the capacity of the electric motor required to raise and lower the loads in the required time.

The order in which the data should be generated is as follows: (1) duty cycle times, (2) hoisting rate, (3) payload and conveyance weight, (4) rope size, (5) drum dimensions, and (6) RMS horsepower requirement.

These items are discussed in more detail in the following.

DUTY CYCLE. The *duty cycle* describes the total time it takes to move a conveyance from the bottom of its wind to the top. It is often depicted graphically as a time-speed diagram. In order to be complete, the duty cycle must include periods of time for initial creep, acceleration, full speed, deceleration, dumping, loading, and rest. The physical laws of uniformly accelerated motion are used to determine the time required and the distance traveled once values for acceleration rate, final velocity, and shaft depth are known.

The relationships between maximum speed, length of wind, and running time are summarized as follows (Fig. 17.5.3).

Accelerating time, seconds:

$$t_1 = \frac{V}{a} \tag{17.5.1}$$

Accelerating distance, ft or m:

$$\frac{Vt_1}{2} = \frac{V^2}{2a} \tag{17.5.2}$$

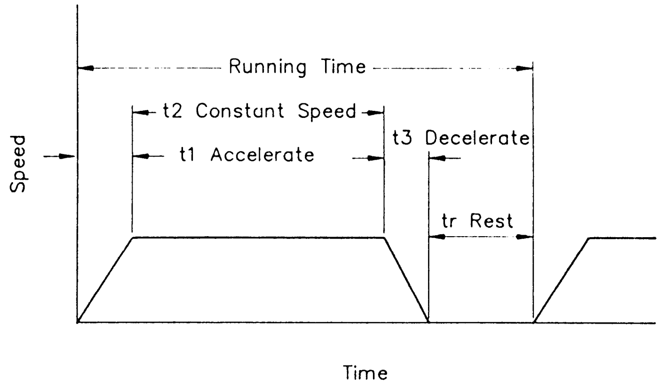


Fig. 17.5.3. Time-speed diagram.

Decelerating time, seconds:

$$t_3 = \frac{V}{r} \tag{17.5.3}$$

Decelerating distance, ft or m:

$$\frac{Vt_3}{2} = \frac{V^2}{2r} \tag{17.5.4}$$

Full-speed time, seconds:

$$t_2 = \frac{L}{V} - \frac{V}{2} \times \left(\frac{1}{a} + \frac{1}{r}\right) \tag{17.5.5}$$

Full-speed distance, ft or m:

$$L = \frac{V^2}{2} \times \left(\frac{1}{a} + \frac{1}{r}\right) \tag{17.5.6}$$

where a is acceleration in ft/s^2 or m/s^2, r is retardation in ft/sec^2 or m/s^2, t_1 is acceleration time in sec, t_2 is full speed time in set, t_3 is retardation time in sec, V is maximum speed in ft/sec or m/s, and L is length of wind in ft or m. Now if we call t_r the time at rest (loading and dumping),

Cycle time in seconds = $t_1 + t_2 + t_3 + t_r$

$$= \frac{V}{a} + \frac{L}{V} - \frac{V}{2} \times \left(\frac{1}{a} + \frac{1}{r}\right) + \frac{V}{r} + t_r \tag{17.5.7}$$

For initial estimating, the following assumptions may be used:

V, fps = $0.742L^{1/2}$ for drum hoists (0.410 $L^{1/2}$ m/s)
V, fps, = $0.790L^{1/2}$ for friction hoists (0.436 $L^{1/2}$ m/s)
a, fps^2, = $r = 2.2$ (0.76 m/s^2)
t_r = 20 sec for good skip loading and dumping conditions

Substitution of the above values in Eq. 17.5.7 gives the total cycle time for the hoisting operation:

Drum Hoist (English units)
Cycle time = 0.337 L + 21.388
Drum Hoist (SI units)
Cycle time = 0.612 L + 22.439
Friction Hoist (English units)
Cycle time = 0.359 L + 21.266
Friction Hoist (SI units)
Cycle time = 0.651 L + 22.294

The maximum velocities used above should not exceed the following values for different types of guides:

Wood guides 35 ft/sec (10 m/s)
Steel guides 50 ft/sec (15.25 m/s)
Rope guides 70 ft/sec (20 m/s)

In addition, the designer should check any statutory restrictions on velocity and acceleration in the jurisdiction (country or state) in which the hoisting system will operate.

PRODUCTION RATE. The required production rate (or hoisting rate as it is sometimes termed) for a skip hoist is expressed in terms of the average tons per hour hoisted. If the mill operates on a different schedule to the mine, the production rate must be adjusted to reflect the difference between hoisted tons per day and milled tons per day. In addition, the moisture content of the ore must be taken into consideration and included in the quantity of material to be hoisted.

At this stage the concept of hoist utilization factor should be introduced. In certain well-organized shafts, the utilization factor can be as high as 0.92 but a more normal figure is 0.70.

SKIP SIZE. After the cycle time and production rate are known, the payload requirements for the skip are derived simply as:

$$\text{Payload p} = \frac{\text{production rate}}{\text{no. of trips/hr}} = \frac{\text{production rate (tons/hr)}}{\text{cycle time (s)} \times 3600\text{ (s/hr)}}$$

The approximate weight of the skip can be derived from one of the following relationships:

$$= 0.5\text{ payload} + 1500\text{ in lb} \tag{17.5.8}$$

$$(0.5\text{ payload} + 680\text{, in kg}) \tag{17.5.8a}$$

or roughly, skip weight = $\frac{5}{8}$ payload.

Conveyance manufacturers should be consulted for more accurate figures.

ROPE SIZE. After the weights of the payload and the conveyance are known, the correct *size of rope* can be selected by applying the required (usually specified by statute) safety factor. The type of rope must also be considered and is covered in more detail in 17.5.5.

For initial estimating purposes, the size of the rope required may be selected by using the following relationships (de la Vergne, 1978).

1. For a depth of 4500 ft (1370 m) or less:

$$W = \frac{P(1 + s) \times 2000}{Ls - 4500} \tag{17.5.9}$$

$$\frac{P(1 + s) \times 1000}{Ls - 1370} \tag{17.5.9a}$$

where W is rope weight in lb/ft or kg/m, P is payload in tons or tonnes, s is mass of skip + mass of payload, Lu is maximum rope length that can be suspended in ft or m, and Ls is $Lu/5$ in ft or m.

2. For depths greater than 4500 ft (1370 m):

$$W = \frac{P(1 + s) \times 2000}{Ls - 1} \tag{17.5.10}$$

$$W = P(1 + s) \times \frac{1000}{Ls - 1} \tag{17.5.10a}$$

DRUM SIZE. Having determined the rope diameter, it is possible to establish a *hoist drum diameter.* Minimum allowable drum diameters are usually covered by legislation. In the United States, drum and sheave diameters must be

1. Not less than 60 times the hoist rope diameter for slope or inclined shaft applications.
2. Not less than 80 times the rope diameter if the hoist ropes are 1 in. (25 mm) in diameter or greater, or not less than 60 times the rope diameter if the hoist ropes are less than 1 in. (25 mm) in diameter for vertical shaft applications.
3. Not less than 100 times the diameter for locked coil ropes.

For drum hoisting, the drum width and center to center distance of the drums (for double drum hoists) must be calculated. The *drum width* may be determined from the following equation:

$$\text{capacity} = 0.252\ WN\,[D/d + 0.85\,(N - 1)] \tag{17.5.11}$$

where capacity is length of rope to be stored in ft, W is drum width in in., N is number of layers, D is drum diameter in in., and d is rope diameter in in. In SI units,

$$\text{capacity} = 3.024\ WN\,[D/d + 0.85\,(N - 1)] \tag{17.5.11a}$$

where capacity is length of rope in meters, W is drum width in cm, N is number of layers, D is drum diameter in mm, and d is rope diameter in mm.

The *center-to-center distance of the drums* can be calculated using the requirements for proper fleet angles (covered later) and knowing the center-to-center distance of the hoisting compartments being serviced.

In the case of tower-mounted friction hoists, the center-to-center distance of the compartments must be equal to or less than the drum diameter. If it is less, a deflector sheave must be installed below the hoist wheel to deflect the ropes on one side of the hoist into their correct position. In addition, the tail rope also has a natural bending radius that must be considered.

POWER REQUIREMENTS. After the duty cycle and rope size have been determined, it is possible to calculate the *hoist motor power* requirements. A quick, approximate method follows.

$$\text{HP required} = \frac{\text{foot-pounds work done per minute}}{33{,}000 \times \text{efficiency}}$$

$$= \frac{\text{payload in pounds} \times \text{speed in feet per minute}}{33{,}000 \times \text{efficiency}}$$

$$= \frac{\text{fpm} \times \text{tons} \times 2000}{33{,}000 \times \text{efficiency}} = \frac{\text{fpm} \times \text{tons} \times 2000}{\text{HP constant}} \tag{17.5.12}$$

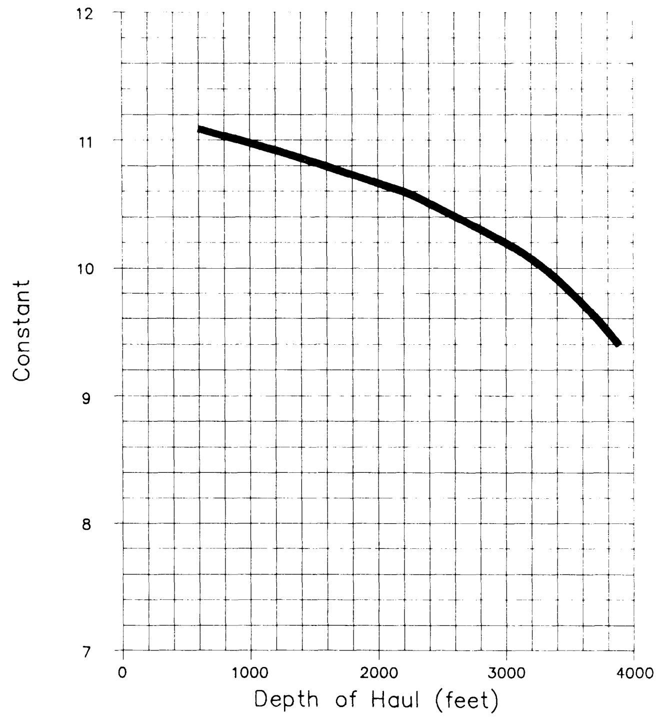


Fig. 17.5.4. Horsepower constant (Anon., 1948). Conversion factor: 1 ft = 0.3048 m.

$$\text{Watts required} = \frac{\text{watts}}{\text{efficiency}}$$

$$= \frac{\text{payload in kilograms} \times \text{speed in meters per second} \times 0.1020}{\text{efficiency}} \tag{17.5.12a}$$

Since the efficiency of the hoist varies according to depth, a graph has been developed from calculated data showing the variation in the constant for different depths (Fig. 17.5.4).

A more exact method of calculating the hoist motor horsepower is presented for both drum hoists and friction hoists as follows (Harmon, 1973).

Drum Hoist Horsepower Calculations— The equations needed to calculate horsepower at all points A through F, as shown on the horsepower vs. time cycle graph in Fig. 17.5.5, are as follows (in English units only):

1. Horsepower HP, required to accelerate the hoist system (motor inertia not included):

$$\text{HP}_1 = \frac{TSL \times V^2}{32.2 \times T_a \times 550} \tag{17.5.13}$$

where TSL is total suspended load and mass of hoist rotating parts to be accelerated $TSL = EEW$ (refer to Fig. 17.5.6) + SL + $2SW$ + $2R$, SL is skip load, SW is skip weight, R is rope weight, V is rope speed in fps, and T_a is total acceleration time (i.e., t_a to creep + t_a from creep to full speed).

2. The horsepower HP, regenerated into the electrical power system by suddenly decelerating is shown as a negative horsepower as it is created by energy pumped back into the power system (all terms the same as in Eq. 17.5.13):

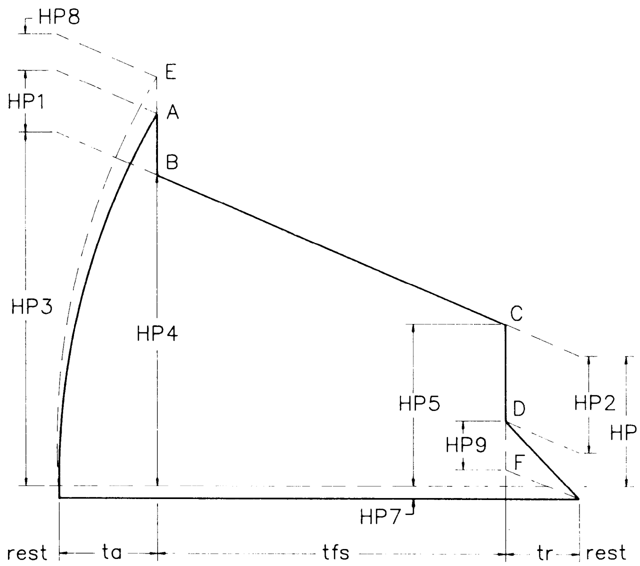


Fig. 17.5.5. Horsepower vs. time cycle, drum hoist (Harmon, 1973, p. 15-54).

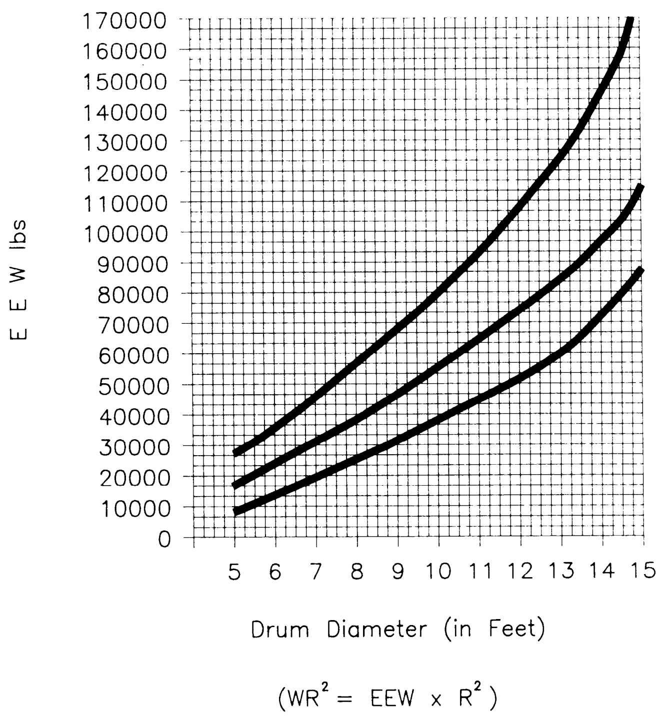


Fig. 17.5.6. Equivalent effective weight chart (EEW) (Harmon, 1973, Fig. 15-39). Conversion factors: 1 ft = 0.3048 in., 1 lb = 0.4536 kg.

$$HP_2 = -\frac{TSL \times V^2}{32.2 \times T_R \times 550} \tag{17.5.14}$$

where T_R is total retard time, i.e., from full speed to creep speed to stop.

3. Running horsepower at the shaft bottom without acceleration, that is, the horsepower to hoist a full skip load at the shaft bottom when traveling at full speed at that particular point in the shaft:

$$HP_3 = \frac{(SL + R) \times V}{550} \tag{17.5.15}$$

4. Running horsepower at the end of acceleration to full speed:

$$HP_4 = \frac{SLB \times V}{550} \tag{17.5.16}$$

where SLB is suspended load at shaft bottom =

$$(SL + R) - (V \times T_a \times \text{rope weight in lb/ft})$$

5. Running horsepower at the end of full-speed-run period and start of deceleration:

$$HP_5 = \frac{SLT \times V}{550} \tag{17.5.17}$$

where SLT is suspended load at shaft top =

$$(SL - R) + (V \times T_R \times \text{rope weight in lb/ft}).$$

6. Running horsepower at the end of deceleration:

$$HP_6 = \frac{(SL - R) \times V}{550} \tag{17.5.18}$$

7. The horsepower correction factor for motor and gearing efficiency is less than 100%:

$$HP_7 = \frac{SL \times V}{550} \times 0.176 \tag{17.5.19}$$

a. Peak accelerating horsepower of the hoist system:

$$A = HP_1 + HP_7 + \frac{HP_4 + 2HP_3}{3} \tag{17.5.19a}$$

b. Full-speed horsepower at the end of acceleration period:

$$B = HP_4 + HP_7 \tag{17.5.19b}$$

c. Full-speed horsepower at the start of deceleration period:

$$C = HP_5 + HP_7 \tag{17.5.19c}$$

d. Deceleration horsepower:

$$D = HP_2 + HP_7 + \frac{HP_5 + 2HP_6}{3} \tag{17.5.19d}$$

(1) Horsepower required to accelerate motor rotor (or armature):

$$HP_8 = \frac{0.6A \times 1.2}{T_a} \tag{17.5.20}$$

(2) Horsepower required to decelerate motor rotor (or armature):

$$HP_9 = \frac{-0.6A \times 1.2}{T_r} \qquad (17.5.21)$$

e. Total horsepower to accelerate the hoist system and motor rotor (or armature):

$$E = A + HP_8 \qquad (17.5.19e)$$

f. Total horsepower to decelerate hoist system and motor rotor (or armature):

$$F = D + HP_9 \qquad (17.5.19f)$$

8. Finally, equations to calculate the RMS horsepower requirement are as follows:
a. dc motors:

$$\text{RMS hp} = \sqrt{\frac{E^2 \times T_a + \frac{B^2 + C^2 + BC}{3} \times TFS + F^2 \times T_r}{0.75\ T_a + T_{FS} + 0.75\ T_r + 0.5\ \text{rest}}} \qquad (17.5.20a)$$

b. ac motors:

$$\text{RMS hp} = \sqrt{\frac{E^2 \times T_a + \frac{B^2 + C^2 + BC}{3} + TFS + F^2 \times Tn}{0.5\ T_a + T_{FS} + 0.5T_r + 0.25\ \text{rest}}} \qquad (17.5.20b)$$

9. As a check on calculations, the following equation can be solved:

$$\left(\frac{A}{2} \times T_a\right) + \left(\frac{B+C}{2} \times T_{FS}\right) + \qquad (17.5.21)$$

$$\left(\frac{D}{2} \times T_r\right) \cong \frac{SL \times \text{depth}}{0.85 \times 550}$$

The relation should agree to within 1 or 2% with the RMS horsepower calculation. This check is simply an approximation of the area under the horsepower vs. time-cycle diagram (Fig. 17.5.5) and represents work done, which is load times depth divided by 550 ft-lb/set-hp with an efficiency factor of 0.85 incorporated.

For unbalanced hoists, the only change is in the total suspended load *TSL*. In this case,

$$TSL = EEW + SL + 1SW + 1R. \qquad (17.5.22)$$

Friction Hoist Horsepower Calculations—Horsepower calculations for friction hoists are similar to those for drum hoists but a little simpler (several are identical). This is because the running horsepower after initial acceleration does not change as a result of a constant rope weight (head rope plus tail rope) during travel up the shaft.

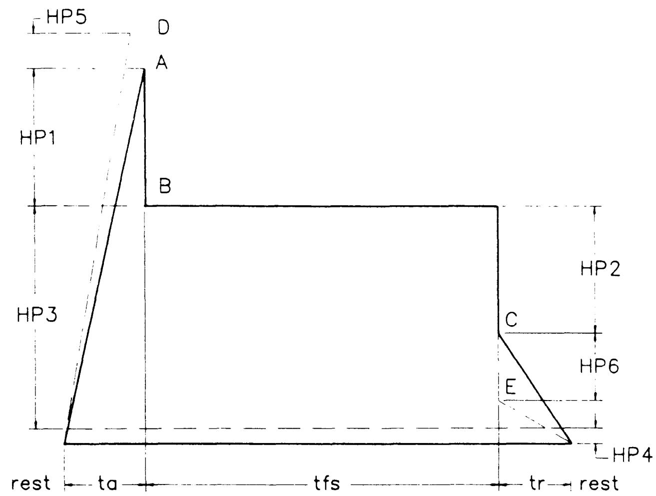


Fig. 17.5.7. Horsepower vs. time cycle, Koepe hoist (Harmon, 1973, p. 15-54).

The equations used to calculate horsepower requirements at all points (in English units only), A through E, as shown on the horsepower vs. time cycle graph in Fig. 17.5.7, are as follows:
1. Acceleration horsepower:

$$HP_1 = \frac{TSL \times V^2}{32.2 \times T_a \times 550} \qquad (17.5.23)$$

where $TSL = EEW + SL + 2SW + R$, and R = (depth × rope weight) per ft × 2 × no. of ropes.
2. Regeneration horsepower:

$$HP_2 = -\frac{TSL \times V^2}{32.2 \times T_r \times 550} \qquad (17.5.24)$$

3. Running horsepower at shaft bottom:

$$HP_3 = \frac{SL \times V}{550} \qquad (17.5.25)$$

4. Running horsepower at end of acceleration:

$$HP_4 = \frac{SL \times V \times 0.111}{550} \qquad (17.5.26)$$

The efficiency of the motor and drive system is greater than in the drum hoist:

$$0.111 = \frac{100 - 3}{E} \text{ where } E = 0.90$$

5. Summary calculations:
a. Acceleration peak of Fig. 17.5.7

$$A = HP_3 + HP_4 + HP_1 \qquad (17.5.27a)$$

b. Full-speed running horsepower:

$$B = HP_3 + HP_4 \qquad (17.5.27b)$$

c. Total retardation horsepower:

$$C = HP_3 + HP_4 + HP_2 \quad (17.5.27c)$$

(1) Running horsepower at end of full-speed run:

$$HP_5 = \frac{0.75A \times 1.2}{T_a} \quad (17.5.28)$$

(2) Horsepower required to retard motor rotor:

$$HP_6 = -\frac{0.75A \times 1.2}{T_r} \quad (17.5.29)$$

d. Total horsepower to accelerate hoist and motor rotor:

$$D = A + HP_5 \quad (17.5.27d)$$

e. Total horsepower to retard hoist and motor rotor:

$$E = C + HP_6 \quad (17.5.27e)$$

6. RMS horsepower
a. dc motors:

$$\text{RMS hp} = \sqrt{\frac{D^2 \times T_a + B^2 \times T_{FS} + E^2 \times T_r}{0.75\ T_a + T_{FS} + 0.75\ T_r + 0.5 \text{ rest}}} \quad (17.5.30a)$$

b. ac motors:

$$\text{RMS hp} = \sqrt{\frac{D^2 \times T_a + B^2 \times T_{FS} + E^2 \times T_r}{0.5\ T_a + T_{FS} + 0.5\ T_r + 0.25 \text{ rest}}} \quad (17.5.30b)$$

The check for Eqs. 17.5.23 through 17.5.30b is the following:

$$\left(\frac{A}{2} \times T_a\right) + \left(B \times T_{FS}\right) + \left(\frac{C}{2} \times T_r\right) = \frac{SL \times \text{depth}}{0.90 \times 550} \quad (17.5.31)$$

Example 17.5.1. Drum Hoist Calculation (Harmon, 1973) (in English units): Determine RMS hp of a hoist system using an ac motor and hoisting two 10-ton skips in balance in a 1000-ft shaft. Rope speed is 20 ft/sec. Skip-weight/skip-load ratio = 0.75. Acceleration rate is 2.0 ft/sec^2; deceleration, 2.0 ft/sec^2; rest period is 10 sec. The hoist uses two l-in. dia. round-strand ropes at 3.78 lb/ft. The hoist is a 15-ft dia. single-drum unit. The hoist duty cycle, in seconds, is

1. Accelerate to creep	1.0
2. Creep at 2.0 ft/sec^2	2.0
3. Accelerate to full speed	9.0
4. Travel at full speed	39.4
5. Decelerate to creep speed	9.0
6. Creep at 2 ft/sec^2	4.0
7. Decelerate to stop	1.0
8. Rest (load and dump)	10.0

Solution. Then T_a = 10 sec (1 + 3), T_r = 10 sec (5 + 7), T_{FS} = 39.4 sec (4)

$$HP_1 = \frac{TSL \times V^2}{32.2 \times T_a \times 550}$$

TSL: 1. EEW from Fig. 17.5.6 for 15-ft single drum is 110,000 lb
2. SL = 20,000 lb
3. 2 SW = 2 × 0.75 × 20,000 = 30,000 lb
4. 2R = 2 × 1000 ft × 3.78 lb/ft = 7560 lb

TSL = 167,560 lb

$$HP_1 = \frac{167{,}560 \times (20)^2}{32.2 \times 10 \times 550} = 376 \text{ hp}$$

$$HP_2 = \frac{-TSL \times V^2}{32.2 \times T_r \times 550} = 376 \text{ hp since } T_r = T_a HP_1 = -HP_2$$

$$HP_3 = \frac{(SL + R) \times V}{550} = \frac{(20{,}000 + 3{,}780) \times 20}{550} = 865 \text{ hp}$$

$$HP_4 = \frac{(SLB) \times (V)}{550}$$

$$\begin{aligned} SLB &= (SL + R) - (V \times T_a \times R \text{ per ft}) \\ &= (20{,}000 + 3{,}780) - (20 \times 10 \times 3.78) \\ &= 23{,}780 - 756 = 23{,}024 \text{ lb} \end{aligned}$$

$$HP_4 = \frac{23{,}024 \times 20}{550} = 839 \text{ hp}$$

$$HP_5 = \frac{SLT \times V}{550}$$

$$SLT = (SL - R) + (V \times T_r \times R \text{ per ft})$$

Since $T_a = T_r$, then ($V \times T_r \times R$ per ft) = ($V \times T_a \times R$ per ft), or SLT = (20,000 − 3780) + 756 = 16,976 lb

$$HP_5 = \frac{16{,}976 \times 20}{550} = 617 \text{ hp}$$

$$HP_6 = \frac{(SL - R) \times V}{550}$$

From HP_5 above, ($SL - R$) = (20,000 − 3780), or $HP_6 = \dfrac{16{,}220 \times 20}{550} = 591$ hp

$$HP_7 = \frac{SL \times V}{550} \times 0.176 = \frac{20{,}000 \times 20}{550} \times 0.176 = 128 \text{ hp}$$

$$A = HP_1 + HP_7 + \frac{HP_4 + 2HP_3}{3} = 376 + 128 + \frac{839 + 2(865)}{3} = 1360 \text{ hp}$$

$$B = HP_4 + HP_7 = 839 + 128 = 967 \text{ hp}$$

$$C = HP_5 + HP_7 = 617 + 128 = 745 \text{ hp}$$

$$D = HP_2 + HP_7 + \frac{HP_5 + 2HP_6}{3} = -376 + 128 + \frac{617 + 2(591)}{3} = 352 \text{ hp}$$

$$HP_8 = \frac{0.6A \times 1.2}{T_a} = \frac{0.6 \times 1260 \times 1.2}{10} = 91 \text{ hp}$$

$$HP_9 = -\frac{0.6A \times 1.2}{T_r}$$

Since $T_r = T_a = 10$ sec, $HP_9 = -91$ hp

$$E = A + HP_8 = 1{,}260 + 91 = 1{,}351 \text{ hp}$$
$$F = D + HP_9 = 352 + (-91) = 261 \text{ hp}$$

Check:

$$\frac{A}{2} \times T_a + \frac{B + C}{2} \times T_{FS} + \frac{D}{2} \times T_r \cong \frac{SL \times \text{depth}}{0.85 \times 550} \qquad (17.5.21)$$

Then

$$\frac{(1260 \times 10)}{2} + \frac{(967 + 745)}{2}(39.4) + \frac{(352 \times 10)}{2} \cong \frac{20{,}000 \times 1000}{0.85 \times 550}$$

or $6300 + 33{,}726 + 1760 = 41{,}786$ }
$\frac{20{,}000 \times 1000}{0.85 \times 550} = 42{,}780$ } less than 2% difference

Therefore, the check indicates that the solution will be accurate to within 2%.

The RMS hp for the ac motor

$$= \sqrt{\frac{(1351)^2 \times 10 + \frac{((967)^2 + (745)^2 + (967)(745))}{3} + (261)^2 \times 10}{0.5(10) + (39.4) + 0.5(10) + 0.25(10)}}$$

or approximately 1000 hp

Hence, for the skip load, skip weight, rope, and rope speed used in this example, a 1000-hp ac motor would be sufficient for this duty cycle.

Example 17.5.2. Friction Hoist Calculation (Harmon, 1973) (in English units): Determine the RMS hp for an ac drive running a friction hoist operating 5-ton skips in balance in a 1000-ft shaft with the following given information:

Given: 1. Hoist uses four l-in. dia. F.S. ropes at 1.80 lb/ft
2. Rope speed = 20 fps
3. Wheel diameter = 10 ft
4. Skip-weight/skip-load ratio = 1.2
5. Duty cycle T_a = 10 sec
T_{FS} = 39.75 sec
T_r = 8 sec
Rest = 10 sec

Solution.

1. HP_1:

$\frac{TSL}{EEW}$ (from Fig. 17.5.7)	= 37,000 lb
+ *SL*	= 10,000 lb
+ 2*SW* (2 × 1.2 × 10,000)	= 24,000 lb
+ *R* (1000 ft × 1.80 lb per ft × 2 × 4)	
TSL	= 14,400 lb
	85,400 lb

$$HP_1 = \frac{85{,}400 \times (20)^2}{32.2 \times 10 \times 550} = 193.5 \text{ hp}$$

2. $HP_2 = \frac{85{,}400 \times (20)^2}{32.2 \times 8 \times 550} = -241$ hp

3. $HP_3 = \frac{(10{,}000) \times (20)}{550} = 363$ hp

4. $HP_4 = \frac{(10{,}000) \times (20) \times 0.111}{550} = 40.3$ hp

5. $A = HP_3 + HP_4 + HP_1 = 363 + 40.3 + 193.5 = 596.8$ hp

6. $B = HP_3 + HP_4 = 363 + 40.3 = 403.3$ hp

7. $C = HP_3 + HP_4 + HP_2 = 363 + 40.3 + (-241) = 162.3$ hp

8. $HP_5 = \frac{0.75A \times 1.2}{T^a} = \frac{0.75(596.8) \times 1.2}{10} = 54$ hp

9. $HP_6 = \frac{-0.75A \times 1.2}{T_r} = \frac{-0.75(596.8) \times 1.2}{8} = -67.2$ hp

10. $D = A + HP_5 = 596.8 + 54 = 650.8$ hp

11. $E = C + HP_6 = 162.3 + (-67.2) = 95.1$ hp

Check:

$$\frac{A}{2} \times T_a + B \times T_{FS} + \frac{C}{2} \times T_r \cong \frac{SL \times \text{depth}}{0.90 \times 550}$$

$$= \left(\frac{596.8}{2} \times 10\right) + (403.3 \times 39.75) + \left(\frac{162.3}{2} \times 8\right) = \frac{10{,}000 + 1000}{0.90 \times 550}$$

$$= (2984) + (16{,}000) + (644) = 19{,}633$$

$$= \frac{10{,}000 \times 1000}{0.90 \times 550} = 20{,}200 \text{ hp}$$

$$= \frac{20{,}200}{19{,}663} = 1.0178,$$

or a difference of 1.78%, which is within the allowance of 1 to 2%; therefore, the solution is accurate.

$$\text{RMS hp (ac)} = \sqrt{\frac{D^2 \times T_a + B^2 \times T_{FS} + E^2 \times T_r}{0.5\ T_a + F_{FS} + 0.5\ T_r + 0.25 \text{ Rest}}}$$

$$= \sqrt{\frac{(650.8)^2 \times 10 + (403.3)^2 \times (39.75) + (95.1)^2 \times 8}{0.5\,(10) + 39.75 + 0.5\,(8) + 0.25\,(10)}}$$

$$= 458.9 \text{ hp}$$

17.5.3.5 Number of Hoists Required

Since most shafts are designed to be multipurpose (i.e., for ore, waste, personnel, materials), one of the first items to be considered is the number of hoists required to meet all the hoisting requirements. The best approach is to list all requirements, determining the daily hoisting time required for each. If the total is more than 24 hours, two hoists must be employed; or if close to 24 hours, compromises made. The items to be included in the list are as follows: (1) number of ore hoisting trips per day, (2) number of waste hoisting trips per day, (3) number of material trips per day, (4) number of man trips per day, (5) time required for hoist, shaft, rope, and conveyance inspections as required by law and good maintenance practices, (6) time required for internal movement of personnel, and materials, and (7) idle time, per day, per week.

After the number of trips has been determined, it is then necessary to calculate the length of time or duty cycle for each trip (see 17.5.3.4).

ORE HOISTING TIME. In order to determine the number of hours per day required to hoist ore, the following should be known: (1) planned tonnage to be hoisted daily, and (2) production rate for the skip hoist in tons per hour. This may be expressed as follows:

time required for ore hoisting (hours/day)

$$= \frac{\text{daily ore production (tons/day)}}{\text{hoist production rate (tons/hour)}} \qquad (17.5.32)$$

WASTE HOISTING TIME. Waste hoisting time is calculated in a similar manner as for ore hoisting. When calculating daily waste production, consideration should be given to using waste underground as backfill as this will reduce the daily waste hoisting requirements:

time required for waste hoisting (hours/day)

$$= \frac{\text{daily waste production (tons/day)}}{\text{hoist production rate (tons/hour)}} \qquad (17.5.33)$$

MATERIALS AND SUPPLIES HOISTING TIME. In order to determine the number of hours per day required to hoist materials and supplies, the following information should be known: (1) number of material and supplies trips per day, and (2) hoist duty cycle, defined as the total time it takes for the hoist to move a conveyance from one elevation to the next:

time required for materials and supplies (hours/day)

$$= \frac{N \times T}{3600} \qquad (17.5.34)$$

where N is number of material trips, and T is cycle time per trip in sec.

PERSONNEL HOISTING TIME. The projected daily employee requirement that must be lowered and raised through the hoisting facilities is either known or calculable. Using the daily manpower data, the hoisting time required for transporting personnel to and from underground is calculated after determining the cycle time, using Eq. 17.5.7.

In estimating the cycle time, consideration is given to the following: (1) cage capacity, (2) number of working levels underground to which personnel are to be transported, (3) hoisting speed to the first level, then between each level to conform with the mining regulations, (4) loading and unloading time (per level), and (5) idle time (per cycle).

MAINTENANCE AND INSPECTION TIME. Although statutory maintenance, inspections, and test requirements, as regulated by federal or local authorities, vary for each country and jurisdiction, the following procedures are carried out on most hoisting facilities:

1. Shaft and conveyance (including shaft signal) inspection—once per week.
2. Hoist (electrical and mechanical components) inspection—once per week.
3. Rope and rope attachments inspection—once per week.
4. Rope maintenance (including attachments)—once per month.
5. Electromagnetic test to ropes—once every three months.
6. Conveyance drop test—once every three months.

In order to allow maximum time for ore hoisting, it is normal practice at most major mines to carry out maintenance and repairs of a major nature on the weekend when the requirements for lowering personnel and material is at a minimum, unless such repairs are deemed necessary for the safe running of the facilities. However, statutory maintenance and inspections consume some of the time which would have been available otherwise for hoisting. For a typical mine, the average time lost for this activity can be approximated, as shown in Table 17.5.2.

Table 17.5.2. Hoisting Time Used For Maintenance and Inspection of Shaft and Equipment

Items	Time (h)	Remarks
Shaft, conveyance & rope inspection & signal bell test	1.5	Weekly
Sheave wheels inspection	0.5	Weekly
Hoist inspection	1.5	Weekly
TOTAL WEEKLY:	3.5	
Rope maintenance (lubrication)	4.0	Monthly
TOTAL MONTHLY:	4.0	
Electromagnetic test to ropes and attachments	2.0	Quarterly
Drop test	2.0	Quarterly
TOTAL QUARTERLY:	4.0	

Note: If the above is interpreted as unavailability of the hoists for production hoisting, then the average time lost per day at the typical hoisting facility can be taken as one hour.

INTERNAL HOISTING TIME. In some mining operations it is necessary to transfer personnel, materials, and equipment between levels using the shaft hoisting equipment. The work required to carry out these tasks is termed internal hoisting, and the time required to complete this type of hoisting can best be obtained through personal experience or by discussion with operators of similar mines.

IDLE TIME: HOIST UTILIZATION. It is a fact that it is not possible to plan to operate 100% of the time and that some idle time must be provided in any hoisting schedule. The amount of time to be allowed is generally a matter of experience on the part of the designer. However, the total operating time scheduled during planning stages should not exceed 70% of the total operating time available, that is, 16.8 hr/day (Harvey 1973a).

The hoist utilization factor H_u is defined as:

$$H_u = \frac{\text{No. of trips actually made in a month}}{\text{No. of trips possible on the basis of cycle time}} \qquad (17.5.35)$$

In certain exceptionally well-organized shafts, utilization factors as high as 0.92 have been achieved, but a more reasonable figure of 0.70 should be adopted. With multipurpose hoists, the utilization factor will be much lower.

SUMMARY. After the daily hoisting time for each of the above time subcomponents has been calculated and a hoist utilization factor estimated, a total daily hoisting time T can be determined. If this total is greater than 24 hours, two (or more) hoists are required:

$$T = \frac{T_o + T_w + T_s + T_p + T_m + T_i}{H_u} \qquad (17.5.36)$$

where T_o is ore hoisting time, T_w is waste hoisting time, T_s is materials hoisting time, T_p is personnel hoisting time, T_m is maintenance hoisting time, T_i is internal hoisting time, and H_u is hoist utilization factor.

17.5.4 SHAFT CONVEYANCES AND ACCESSORIES

The most common shaft conveyances considered are skips, cages, and counterweights. Accessories consist of loading pockets, spill pockets, and arrestor gear.

Skips are used to hoist broken ore and waste from the mine. Cages are used to raise and lower personnel, material, and equipment. Counterweights may be used in balance with either skips or cages. Loading pockets can improve the efficiency of hoisting systems by ensuring that the optimum amount of material is fed to the skip during each cycle. Arrestor gear is provided in the headframe and shaft bottom for friction hoist systems to protect personnel, the hoist, and the conveyances in the event overtravel occurs at the end of the hoisting cycle.

17.5.4.1 Description of Skip Hoisting

Skipping, as described by Souter (1973), consists of filling, hoisting, dumping, and returning to be filled again. The ore and waste may be crushed or uncrushed, filling may be volumetric or by weight, and the operation may be manual or automatically controlled. During filling, the skip is subjected to high dynamic loads, and depending upon depth, a certain amount of rope stretch occurs. Although chairing of the skip is possible, this practice can also result in higher impact loading, difficulties with slack rope, and the danger of leaving the chairs in the shaft, which could cause collision.

In order to permit hoisting at high speeds, skips are guided as they travel in the shaft. Difficulties with guides due to misalignment or loosening generally result in lower hoisting speeds and, correspondingly, reduced production rates.

Skips may be dumped at any point in the shaft, although the most common location is in the headframe. Dumping is usually accomplished by means of scrolls located at each side of the skip compartment at the dump. A roller on the side of the skip body (for swing-out body skips) or on the door (for fixed body skips) engages the scroll and causes the skip to be dumped. In the Sala-type skip, an air cylinder actuates the dump door.

17.5.4.2 Types of Skips

There are three main types of skips in use today: (1) the overturning, (2) the swing-out body, and (3) the fixed body.

OVERTURNING. The *overturning* or *Kimberley skip* consists of a bail frame, a shaft to support the skip, and the skip body itself. The skip body rests on the shaft. The skip is dumped through the action of a bull wheel, mounted on each side (at the top) of the skip body, engaging a scroll and causing the skip

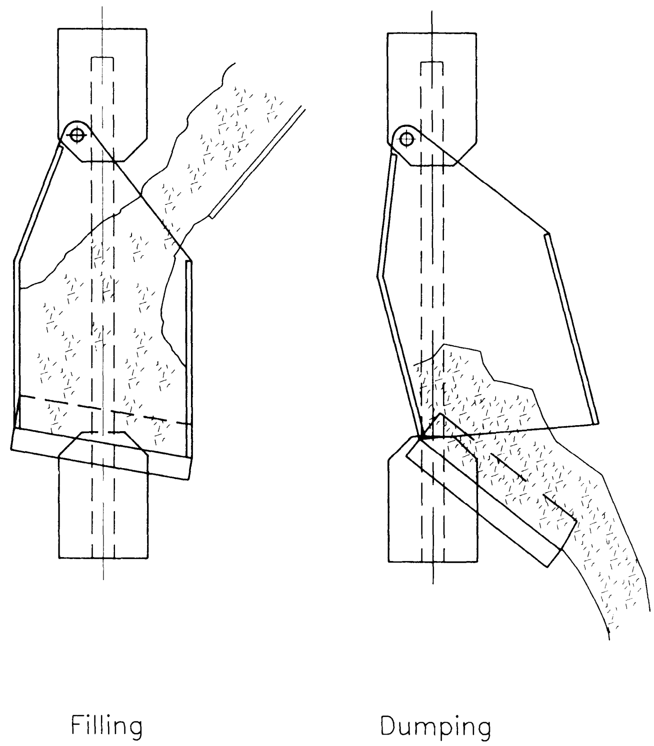


Fig. 17.5.8. Bottom dump skip, swing-out body (Penning, et al., 1977).

body to rotate around the shaft as the skip is being hoisted through the dump. The design of the tipping path is complex because the payload is being discharged as it moves along the path. The horizontal and vertical forces to be considered, according to Harvey (1973a), are dependent upon the rate at which the skip enters the dump, the inertia of the skip plus payload, the horizontal acceleration (which itself is dependent upon the angle of the tipping path), and the static forces.

The advantages of Kimberley skips are (1) there is less shaft spillage when handling fine or wet material, and (2) they handle larger pieces for a fixed cross-sectional area. The disadvantages are (1) the capacity is limited due to the height-to-width ratio, (2) the payload ratio is usually lower particularly with sticky material that tends to build up in the bottom, (3) they generally impose higher stresses on the headframe and bins, and (4) they require more headframe height to dump.

SWING-OUT BODY. The *swing-out body skip* also consists of a bail frame, a shaft, and the skip body. In this case, the skip body is suspended from the shaft and the skip dumped by means of rollers mounted on each side (at the bottom) of the body, engaging a scroll and causing the body to pivot away from the bail into the dump. At the same time, a discharge door, which forms the bottom of the skip, opens and the material is discharged (Fig. 17.5.8).

There are three common types of swing-out body skips. These are the Jeto, Saunders, and Rollamatic. The differences among these three types is the mechanism involved with the opening and closing of the door and the location of the door hinge. Safety latches have been used on all three types. This is to prevent the skip from opening in mid-shaft, particularly when traveling empty.

For bottom-dumping skips in general, the cross-sectional area is partially determined by the size of material to be handled.

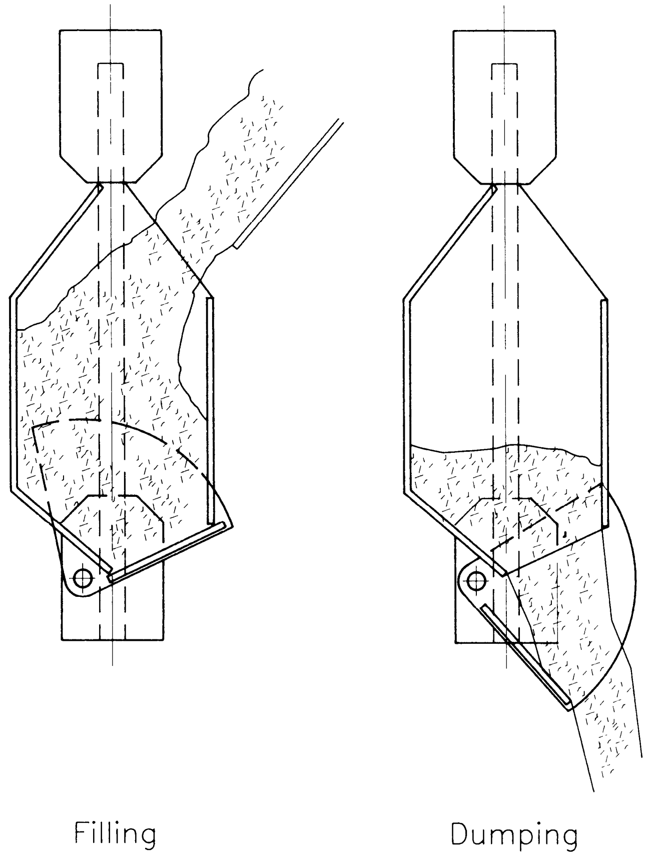


Fig. 17.5.9. Bottom dump skip, fixed body (Penning et al., 1977).

The door opening is sized to be at least three times the dimensions of the largest piece to be handled. This is to prevent possible bridging. Other factors to be considered are the length of the skip, the available space, and the hoist rope center-to-center distance.

The advantages of swing-out body skips are (1) they have more favorable payload ratio than the overturning type, (2) the tipping path is less complex than for overturning skips, and (3) the dumping distance in the headframe is less than for overturning type. The disadvantages are (1) muck build up at the hinge point in the door can prevent the door from closing and could place large forces on the headframe scroll or skip body during the closing cycle, (2) with wet material, there could be spillage or dribble from the door lip, and (3) the bottom door mechanism is more complex than the fixed body type.

FIXED-BODY. In *fixed-body skips*, the body of the skip remains fixed and does not move out of the confines of the skip compartment during dumping. Instead, a door at the bottom of the body swings open to form a chute and allows the material to discharge into the dump bin. This type of skip may or may not have a bail (Fig. 17.5.9).

There are several types of fixed-body skips. The most common are the Rolla-Chute, the Arc-Door, and Sala-type skip. The difference among these types is the mechanism involved with the opening and closing of the door. In the Rolla-Chute skip, the door is held closed by means of a door latch. When the skip enters the headframe, a latch track releases the latch, allowing the door to open and the material to discharge. With the Arc-Door chute, a roller is mounted on each side of the door. When the skip enters the skip dump, the roller engages a scroll, causing the door to swing open. In the Sala skip, the door is operated by means of air cylinders mounted on the skip body. Compressed air to operate the cylinders is provided through a yoke which is permanently mounted in the headframe and picked up by the skip as it enters the dump.

The cross-sectional area of fixed-body skips is determined by the size of the material to be handled. Since fixed-body skips can be built without a bail, the cross-sectional area of the skip can be increased by 10 to 15% when compared to skips with a bail for the same sized compartment.

The advantage of fixed-body skips are (1) the skip can load and dump through the guides, (2) the skip has no bail, thus allowing a larger cross-sectional area to be used for the skip body (this can reduce the overall length of the skip for a given capacity), (3) the only moving part is the door that forms its own dump chute, (4) they are suitable for fixed or rope guides, (5) a hinged floor at the top can convert the skip to a personnel and materials cage, (6) the skip requires less travel distance to open the door, (7) this type of skip places less stress on the headframe, skip body, and ropes during the dumping operation, and (8) there is less spillage at the door. The disadvantage of this type of skip is that its bailess construction requires adequate design and inspection during operation.

17.5.4.3 Skip Design Considerations

The process of skip design has been well documented by Penning (1977). When selecting a particular skip, the components to be considered during the evaluation process include (1) design methods, (2) factors of safety, (3) body, (4) door, (5) crosshead, (6) bail, (7) guide rollers, and (8) guide shoes.

DESIGN METHODS. Skip components may be designed on the basis of experience and practice or through analytical methods that define the loading being placed on the various components of the skip. The types of loading to be considered are (1) dynamic loading, caused by filling, hoisting, and dumping the skip, and (2) static loading. Factors of safety are usually specified by legislation.

BODY. The skip body is designed as a rectangular box with stiffeners. The type of material, its thickness, and the stiffener size and spacing are the variables to be examined. Particular attention must be given to the area on the back side of the skip that receives impact and abrasion from loading.

Skip bodies are usually lined with abrasion resistant (AR) material to increase its overall life. Two types of steel or elastometers have been used for skip liners. AR steels are members of the high-strength, low-alloy family of steels. Manganese steels are used because they have the ability to work-harden during service. Elastometers have the advantage of reduced weight, less ore sticking and buildup, corrosion resistance and ease of handling during replacement.

The use of aluminum is also increasing, particularly in drum hoist applications. Such skips have a lower mass and allow a higher payload to be hoisted.

DOORS. Both swing-out body and fixed-body skips have doors at the bottom of the skip. Doors must be designed to withstand the impact of large pieces falling on the floor during loading and the static loading of a full skip as well as abrasion during unloading.

With swing-out body skips, the door is hinged at the back, and there are several problems associated with leakage paths between the door and the body. These door leakage problems may be solved through the use of rubber sheets, flexible materials, or special modification of plates at the door. Fixed-body skips generally have fewer problems with leakage or door closure. One situation that may cause problems to both types exists when the skip is loaded on the side opposite the dump site. In this case, a vertical stop may be placed for the dump roller to react against. Doors are made and lined with the same material as used in the body.

CROSSHEAD. The *crosshead* fastens the bail to the rope. It must be designed to take all the skip loads, including acceleration, impact and service factors. The location of the crosshead above the skip is particularly important during loading. It should not interfere with the flow of muck from the loading pocket. The rope stretch in deep mines can be several feet and must be considered. It can be calculated from the following relation:

$$\text{rope stretch (ft or m)} = \frac{\text{muck load (lb or kg)} \times \text{rope length (ft or m)}}{\text{rope area (in.}^2\text{ or mm}^2\text{)} \times \text{rope modulus of elasticity (psi or Pa)}} \tag{17.5.37}$$

BAIL. During hoisting, the *bail* supports the body pivot, prevents the body from moving, and supports the door. At the dump, the bail is subjected to horizontal loading and supports the door and is thus subjected to both bending and axial loading.

GUIDE ROLLERS. The purpose of *guide rollers* is to keep the guide shoe off the guides. This in turn results in a smoother ride, reducing dynamic loading between the skip, guide, and buntons during travelling.

When selecting guide rollers, there is usually a conflict between a large-diameter tire, adequate shaft clearance, and the cross section of the skip. Left unresolved, the final design usually results in a small-diameter tire. Such tires need frequent replacement. The range of tire sizes used are from 6 in. (152 mm) to 14 in. (356 mm) in diameter. Tires are rated on a load-mile per hour basis.

GUIDE SHOES. *Guide shoes* are used on vertical conveyances to keep the main structure from being worn away by the guides. They are considered as expendable items to be replaced when they are too worn for further use. Clearance between the guide and guide shoe varies from ¼ to ½ in. (6.4 to 12.7 mm). Guide shoes can be made from many materials, such as cast iron, mild steel, hardened steel, AR steel, brass, or bronze. The amount of wear to be expected is directly proportional to the load on the guide shoe and inversely proportional to the hardness of the softer material.

17.5.4.4 Loading Pockets

At the *loading pocket,* ore and waste are transferred from the mine to the skip. The equipment and systems installed can vary from simple manually operated to complex and automated. There are two basic types of loading systems available. These involve measurement by weight or by volume.

The criteria to be considered in the design of all systems include (1) response time, (2) loading time, (3) accuracy of measuring devices, (4) operating and maintenance costs, and (5) amount of spillage.

Although details can vary, each system has certain common elements: (1) pocket chute with control gates, (2) measuring pocket, and (3) skip feed chute with control gate.

POCKET CHUTES. Pocket chutes are generally fabricated from steel with liners. They have an angle of inclination of 45 to 55°. Chute gates can be pneumatically or hydraulically controlled and may be guillotine, ball and chain, or radial type.

MEASURING POCKET. The measuring pocket construction is similar to that of the skip body. There are two types of fabricated pockets available, the swing-out body and the fixed-body. The fixed-body is preferred because it is faster operating, there are no large masses to be moved, and it can be easily automated.

The loading pocket is furnished with pneumatically or hydraulically controlled gates. The bottom of the loading pocket is usually inclined at an angle of 60° or greater to ensure rapid loading and complete clean out.

With balanced skip hoisting, two loading pockets are provided, one for each skip.

SKIP FEED CHUTE. A feed chute is required to direct the material from the measuring pocket to the skip. The angle of inclination of the feed chute is usually 60° or greater.

The clearance between the chute and the skip is approximately 3 in. (75 mm). It has been suggested that the ratio of the chute discharge width to the skip width should not exceed 0.7 for skips up to 4.25 ft (1.3 m) wide and 0.8 for skips up to 6 ft (1.8 m) wide. To reduce spillage, the sides of the chute should be slightly converged at the lip.

AUTOMATIC LOADING. Automatic loading of skips is used to improve the efficiency of hoisting systems and to reduce labor requirements. Efficiency is improved by ensuring that each skip receives the required load without overloading or underfilling. Manpower is reduced when automatic loading is used in conjunction with automatic hoisting. In order that automatic loading can be accomplished, a minimum of three control systems must be included (1) a means of measuring the weight and/or volume of material in the measuring pocket, (2) a means of opening and closing feed chutes to fill and empty the measuring pocket, and (3) a logic system to ensure that the above operations are carried out in the correct sequence to prevent such systems failures as double loading and dumping into an empty shaft. With automatic hoisting, the logic system extends to the operation of the hoist.

17.5.4.5 Spill Pockets

Despite the precautions taken to prevent spillage during the loading and dumping of the skips, a certain amount is inevitable below the loading pocket. This spillage may be confined to the hoisting compartment by lining these compartments down to a *spill pocket.* According to data collected by Souter (1973) from a number of mines in South Africa, the amount of spillage, expressed as a percentage of tons hoisted, varied from 1.5 to 5 %. The larger amounts of spillage were associated with manually operated systems handling uncrushed material.

There are two commonly used methods of handling spillage in modem hoisting systems. One method consists of installing deflectors in the shaft below the underwind position of the skip. These deflectors direct spillage into a pocket similar to the loading pocket. The spillage pockets are then cleaned as required. With drum hoisting, this system can be used at any elevation in the shaft. With a friction hoist, the spillage pocket must be located below the tail rope loops, and a second conveyance such as the cage must be used to handle spill. The second method of handling spillage consists of developing a ramp from the lowest elevation to the shaft bottom. In this system, any spillage falls directly to the shaft bottom, where it can be mucked out. Regardless of the system used, particular care is required with friction hoist systems to ensure that the tail ropes do not become entangled or affected by the build-up of spill in the shaft.

17.5.4.6 Cages

Cages are used primarily to handle personnel and materials entering and exiting from the mine. In some instances, cages are used to hoist cars loaded with ore and waste. The design and construction of cages are similar to those used for skips.

A cage is merely an enclosed box, opened by door(s), and suspended from a hoisting rope. The dimensions and capacity of the cage are determined by the quantity, volume, weight, and dimensions of the materials and/or items to be handled.

17.5.4.7 Counterweights

In some balanced hoisting systems, a *counterweight* may be used with either a cage or skip. Since counterweights serve no functional purpose other than to provide for balanced hoisting, their dimensions are quite variable. They are usually designed to fit the space available.

17.5.4.8 Safety Devices

In North America, different types of safety devices are used for drum hoists and friction hoists.

DRUM HOIST. With drum hoists, both excessive overwind and rope breakage can occur. Of these two types of accidents, rope breakage is considered the most serious, and provisions to bring the conveyance to a safe stop in the event of a rope break is required by legislation.

The most common safety device used is the *safety dog.* A pair of dogs is installed at the top of the conveyance at each guide. During travel, the tension in the hoisting rope keeps the dogs open. Should the rope break, or become slack, a large spring and series of levers cause the dogs to penetrate the guide and stop the conveyance.

FRICTION HOIST. In multiple rope friction hoist installations, the probability of all hoisting ropes breaking is small. Therefore, overtravel is considered to be more serious, and provisions to bring the conveyance to a safe stop in the event of overtravel are required.

The most common device used is *arrestor gear,* consisting of steel frames supported in the hoisting compartment and transported by the conveyance as it travels through the arrestor zone. these devices are located in the headframe and at the shaft bottom. They rely on friction between the device and the guides to decrease momentum.

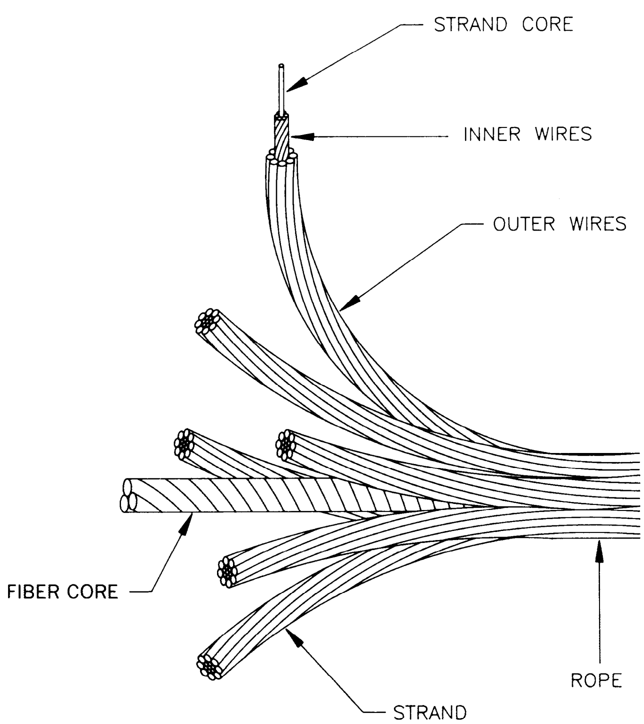


Fig. 17.5.10. Rope components (Anon., 1980a).

17.5.5 ROPES

The basic purpose of the *hoisting rope* is to connect the conveyance to the hoist. It is selected primarily on the basis of safety, compatibility, life, and costs. Safety requirements when hoisting personnel, or where persons may be endangered by hoists and their appurtenances, are usually determined by legislation. The life of a rope is usually expressed as the number of trips it will make; it is affected by rope construction, hoist and sheave dimensions, type of loading, shaft atmosphere, and maintenance. Costs of initial purchase, maintenance, and costs of rope changing including lost production are normally considered. In addition to hoisting ropes, the hoisting systems designer must be concerned with balance ropes (for friction hoist application), guide ropes, rubbing ropes, and rope attachments.

The type of rope construction commonly used for various hoisting applications are as follows:

Rope Use	Typical Rope Construction
drum hoist ropes	round strand—Lang lay flattened
	flattened strand 6 × 27 FC
	full locked coil
friction hoist rope	locked coil
	flattened strand
balance ropes	non-rotating 18 × 7 nylon core
	non-rotating 34 × 7 nylon core
guide and rubbing ropes	half locked coil
shaft sinking	non-rotating 18 × 7 IWRC
	full locked coil
slings	round strand 6 × 25 IWRC

In this segment, a general description of the various types of ropes, plus the factors to be considered to ensure correct application and selection, is presented.

17.5.5.1 Rope Construction

In the construction of most wire ropes, a number of individual wires are wound around a core to form a strand. The strands are then wound around a core to form the rope. Fig. 17.5.10 is an exploded view of a rope showing the components.

Some ropes are constructed without strands by winding consecutive layers of wire around the inner layers.

The factors to be considered in rope construction are (1) the wire, (2) strands, (3) cores, and (4) lay. By varying these factors, ropes with different characteristics can be constructed.

WIRE STRENGTHS AND PROPERTIES. Wire for manufacturing wire rope is available with varying strengths up to 360,000 psi (2480 MPa). It is thus possible to manufacture ropes of different strengths for the same diameter and the same construction. Generally speaking, as the wire strength increases, fatigue life and rope life tend to decrease.

The most common wire used in mining is improved plow grade 110/120. Wire is also available in different shapes with round, full-lock and half-lock being the most common. Galvanized wire is available for wire ropes and should be considered if ropes are being replaced due to corrosion.

STRANDS. Individual wires are twisted together to form *strands.* There are four common types of strands: (1) round strands, (2) triangular strands, (3) oval strands, and (4) flat strands.

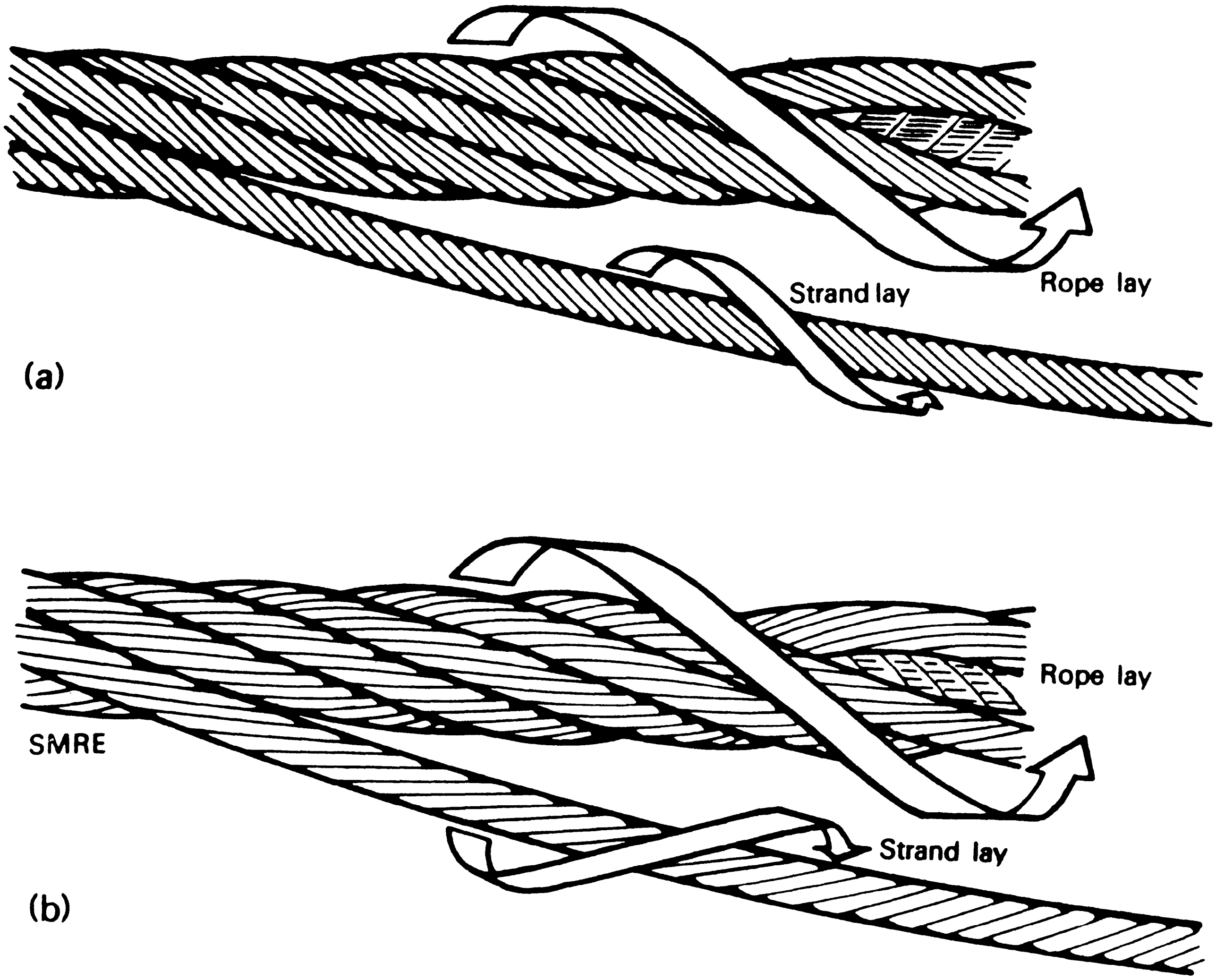


Fig. 17.5.11. Rope lay (Anon., 1980a). (a) Lang lay: wires and strands laid in same direction; (b) ordinary lay: wires and strands laid in opposite directions.

CORES. The purpose of the *core* is to take the internal compressive stresses. The largest of these stresses is due to the tendency of the rope to flatten when bent around a sheave or thimble.

Natural fiber cores (FC) such as sisal and hemp are suitable for most applications. Synthetic fiber cores such as nylon and polypropylene are used in conjunction with high-strength wires and in corrosive applications.

Independent wire rope cores (IWRC) are stronger than fiber cores. They add to the strength of the rope and reduce the stretch; fatigue life is reduced, however.

LAY. The *lay* of a rope is the manner in which the wires are twisted into the strand and the strands are twisted into ropes. There are three properties used to describe lay: (1) length (or pitch), (2) direction (right or left), and (3) type (regular or Lang).

Fig. 17.5.11 illustrates the direction of lay for regular lay and Lang lay, which are the usual lays used in mining ropes. The lay length is the distance from one crown to the next of a particular wire.

Right lay is usually used unless special circumstances require the use of left lay. Some drums are designed to take left-hand lay ropes and should be checked prior to purchasing. Fig. 17.5.12 shows how to select the lay of the rope required for a drum.

Regular lay ropes have the wires in the strands laid in such a manner that the wire crowns run approximately parallel to the rope direction. Such ropes have good resistance to kinking and twisting and are able to withstand considerable crushing and distortion. Regular lay should be used for slings and balance ropes.

With Lang lay, the wires are on the diagonal of the strand and are exposed for a greater length. This arrangement gives greater abrasion resistance and a more flexible rope. Lang lay is commonly used for hoisting ropes on drum hoists. Ropes with this lay slide into position on a drum better than regular lay ropes. Because of the tendency to untwist, these ropes should not be used with swivels or single-part hoisting without guides.

17.5.5.2 Hoist Ropes

The three types of ropes generally considered for hoisting ropes are round strand, flattened strand, and locked coil. Typical properties for these types of rope are given in Table 17.5.3.

ROUND STRAND ROPES. *Round strand ropes* consist of a number of strands, each consisting of wires, wound around a core. The rope designation (e.g., 6×27 FC) indicates the number of strands (6) and the number of wires (27) per strand; FC

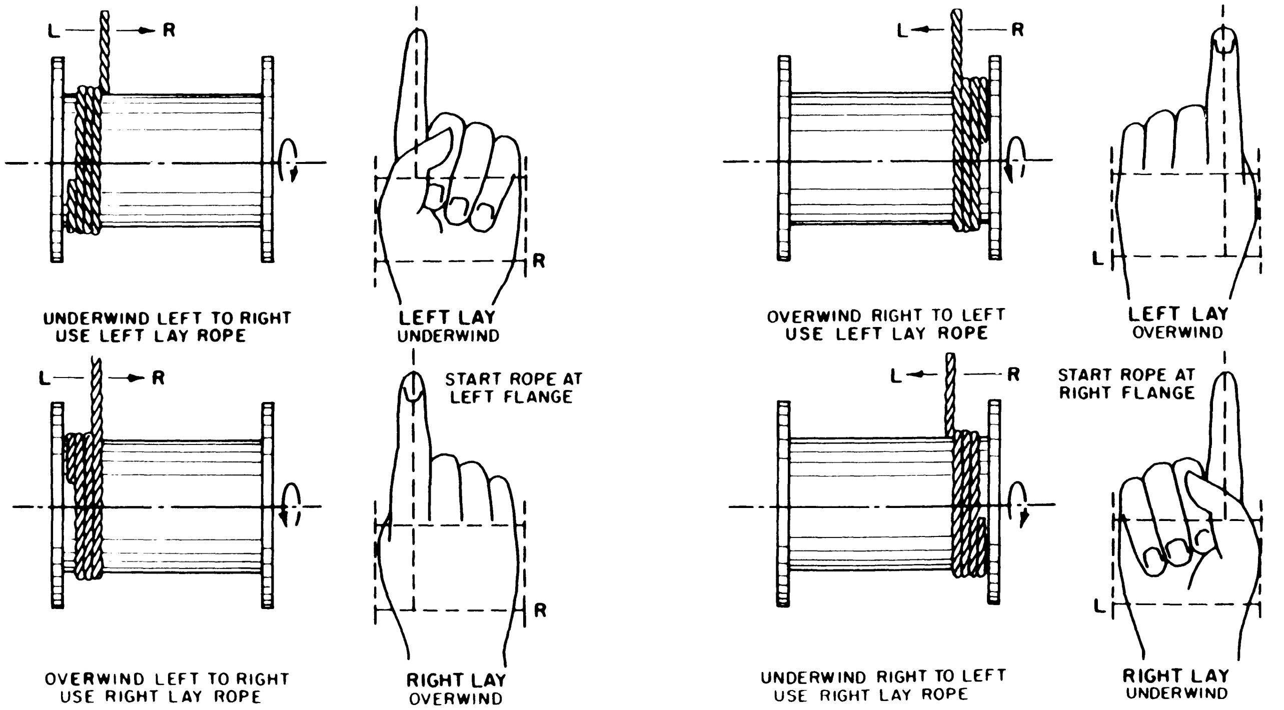


Fig. 17.5.12. Selection of correct direction of lay for a drum hoist ("Wire Rope Users Manual," 2nd ed., Committee of Wire Rope Producers, American Iron and Steel Institute, undated).

Table 17.5.3. General Wire Rope Data for Standard Tensile Wire, Nominal Breaking Load of 120 Long Tons/In.²

Rope	18 × 7 L.C. I.W.R.C. Spin-Resistant and Sinking Rope		6 × 27 F.C. Flattened Strand (Triangular)		Locked Coil	
Diameter (in.)	Weight (lb/ft)	Breaking Strength (1000 lb)	Weight (lb/ft)	Breaking Strength (1000 lb)	Weight (lb/ft)	Breaking Strength (1000 lb)
3/4	1.11	55.23	.97	57	1.31	69.4
7/8	1.42	74.04	1.33	78	1.85	94.0
1	1.84	95.85	1.74	103	2.45	123.2
1 1/8	2.33	121.21	2.22	132	3.08	156.8
1 1/4	2.80	148.65	2.74	163	3.75	192.6
1 3/8	3.38	179.44	3.32	197	4.53	233.0
1 1/2	4.02	212.53	3.95	235	5.25	277.8
1 5/8	4.69	249.00	4.66	276	6.24	324.8
1 3/4	5.62	298.47	5.37	319	7.29	376.4
1 7/8	6.18	331.19			8.46	432.4
2	7.08	375.79			9.67	492.8
2 1/8	8.19	434.85				
2 1/4	9.15	485.54				

Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 lb = 0.4536 kg, 1 long ton = 1.016 t.
Source: Wire Rope Industries Ltd.

stands for fiber core. The greater the number of wires in a strand, the greater the flexibility of the rope, but the smaller wires wear faster and are more easily damaged.

FLATTENED STRAND. The properties of round strand rope have been improved in *flattened strand rope* by making the strands triangular instead of round. The benefits of this are

1. An increase in the number of wires in contact with the sheaves and drums increases the contact area, thereby reducing the pressure and lessening wear on ropes, sheaves, and drums.

2. The rope has a greater crushing strength and can be used in multilayer winding.

3. The wire density in the rope is higher. Thus for the same diameter, more steel area is available, and the rope has a higher strength.

FULL LOCKED COIL. This type of rope is completely different from both round and flattened strand rope. The center or core of the *locked coil rope* consists of a concentrically laid strand of round wires. Around this core lies one or more layers of shaped wires, the outer layer always being interlocking. The shape of all shaped wires in a locked coil rope depends on the rope diameter and its end use.

Locked coil rope construction has some very definite advantages. In a locked coil rope, there is no initial twist in the rope so the wires do not tend to untwist under load. Locked coil ropes have a higher breaking strength than stranded rope for equal diameter and for the same nominal strength grade. Because of their smooth external surface, reduction in strength caused by frictional wear on drums or pulleys is much lower. Also, because of their design, locked coil ropes are less subject to rotation and stretch than stranded ropes.

17.5.5.3 Rope Selection

For the initial selection, four requirements should be considered: (1) strength, (2) resistance to being fatigued, (3) abrasion resistance, and (4) resistance to crushing or distortion. The choice should be made after correctly estimating the relative importance of each of the above requirements. Naturally, strength is the overriding concern.

Design considerations should provide economical and efficient rope service while maintaining the necessary degree of safety to both personnel and equipment. These objectives are best met by accepting the following guidelines:

1. Design the hoisting system with good rope life as an objective.
2. Specify ropes to be compatible with the required factor of safety to match the design of hoist, shaft, headframe, and sheave.
3. Design rope storage and handling procedures.
4. Design correct rope installation procedures.
5. Design correct and adequate maintenance procedures.
6. Design and institute an inspection procedure.

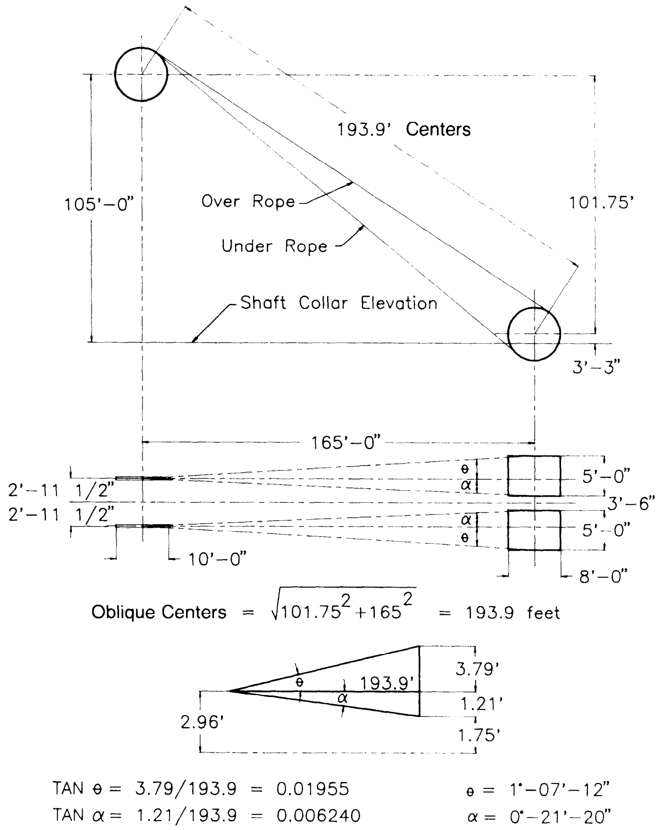
ROPE SELECTION FOR DRUM HOIST. This segment discusses the selection of the size of rope to meet operating conditions for drum hoists for vertical and slope applications. In the United States, the *Code of Federal Regulations* (CFR) (Anon., 1989) specifies minimum requirements for the following items that are relevant to drum hoisting: (1) factor of safety, (2) ratio of drum/sheave diameter to rope diameter, and (3) fleet angles. An example calculation of hoist rope selection concludes this discussion.

Factor of Safety—Mandatory requirements by the Mine Safety and Health Administration (MSHA) are covered in CFR. Sections 55/56/57.19-21 and 75/77.1431 for metal and nonmetal open pit mines, sand, gravel and crushed stone operations, metal and nonmetal underground mines, and underground coal mines, respectively, require the following minimum rope strengths, where L is maximum suspended rope length in ft:

winding drum ropes less than 3000 ft long: minimum rope strength = static load × (7.0 − 0.001L). (17.5.38)

winding drum ropes 3000 ft long or greater: minimum rope strength = static load × 4.0. (17.5.39)

Drum/Sheave Diameter—Mandatory requirements for drum and sheave diameters have now been removed from the regulations, but insofar as the old regulations are still common practice in the United States as well as Canada and Europe, they are repeated here.



Conversion factor:
1 ft = 0.3048 m

Fig. 17.5.13. Fleet angle calculation.

Drum and sheave tread diameters should be (1) not less than 60 times the hoist rope diameter for slope or inclined shaft applications, (2) not less than 80 times the hoist rope diameter if the hoist ropes are 1 in. (25 mm) in diameter or greater, or not less than 60 times the hoist rope diameter if the hoist ropes are less than 1 in. (25 mm) in diameter for vertical shaft applications, and (3) not less than 100 times the hoist rope diameter for locked coil ropes.

Fleet Angles—The mandatory regulation (Anon., 1989) states, "Fleet angles on hoists installed after Nov. 15, 1979, shall not be greater than ½° for smooth drums or 2° for grooved drums." (CFR 56.19-37). However, to increase rope life manufacturers of hoist ropes commonly recommend a maximum fleet angle of 1°20' and a minimum fleet angle of 0°30' to insure rope will crossback and start a new layer without piling. Computations of optimum total fleet angle are given in ANSI M 11.1 (Fig. 17.5.13).

Size Calculation—The procedure for sizing a rope for a drum hoist consists of the following steps:

Step 1. Tabulate operating conditions.

Step 2. Calculate personnel load.

Step 3. Calculate load with conveyance at collar. (Note: US practice only requires check of lowest level.)

Step 4. Calculate load with conveyance at lowest level.

Step 5. Select rope and check safety factor using ANSI M11.1 or CFR Title 30.

Step 6. Check other hoisting components.

Example 17.5.3. Given the following drum hoisting system:

Step 1. Assumed operating conditions.

Height, collar to head sheave	130 ft	(39.6 m)
Depth of lowest level	5000 ft	(1524 m)
Cage capacity	60 persons	
Cage capacity materials	20,000 lb	(9072 kg)
Cage weight	8000 lb	(3629 kg)
Weight of rope attachment	300 lb	(136 kg)

Select the proper hoisting rope and determine factors of safety SF.

Solution.

Step 2. Calculate man load.

60 persons @ 200 lb = 12,000 lb

Step 3. Calculate load with conveyance at collar.

(Not required in US practice.)

Assume 1-1/2 in., 6 × 27 flattened strand rope. Table 17.5.3 shows the properties of this rope.

Assume owner elects to employ factors of safety from 57.19a-21 of the *CFR* when hoisting materials.

Thus suspended load is

cage weight	8,000 lb
pay load	20,000 lb
rope attachments	300 lb
130 ft rope @ 3.95 lb/ft	514 lb
Total:	28,814 lb

Thus minimum breaking strength required = suspended load × SF = 28,814 lb × (7 − 0.001 × 130) = 197,953 lb.

Step 4. Calculate load with conveyance at lowest level.

suspended loads	
cage weight	8,000 lb
payload	20,000 lb
rope attachments	300 lb
5130 ft of rope @ 3.95 lb/ft	20,264 lb
Total:	48,564 lb

Thus minimum breaking strength = 48,564 × 4 = 194,256 lb

Step 5. Select rope and check safety factor.

From steps 3 and 4 above, it can be seen that the loading condition at the lowest level is critical, and that the minimum rope breaking strength required is 194,256 lb.

Referring to Table 17.5.3, it can be seen that 1½-in. diameter 6 × 27 flattened strand rope manufactured from 120 long tons/in.^{2} wire has an ultimate breaking strength of 235,000 lb.

For this rope the factors of safety are

		Actual	Required
@ collar	$SF = \frac{235{,}000}{28{,}814} =$	8.15	5.7 (not mandatory in US)
@ loading pocket	$SF = \frac{235{,}000}{48{,}564} =$	4.83	4.0

Rope is therefore deemed satisfactory.

Step 6. Check with other hoisting components.

The designer would normally check hoist drum storage capacity and fleet angles at this stage as well as rope projected pressure on the drum grooving. After considering the other components of the hoisting system, the designer may decide to reduce the weight or payload of the cage in order to use a smaller-diameter rope with higher tensile-strength wire. Alternatively, a decision may be made to use a larger-diameter rope and thus increase rope life.

Rope Selection for Friction Hoist. A friction hoist differs from a drum hoist in that the rope is not wound on and off the drum but passes around the drum, with the drum friction providing the force necessary to move the rope. Friction hoists may be mounted in the headframe tower directly over the shaft, or they may be mounted in a hoist house at ground level. The friction hoist drum is often referred to as a wheel.

A specific friction hoist application can have a number of acceptable hoist rope/tail rope alternatives. As the selected number of hoist ropes increases, their diameter can become smaller. This also means the hoist drum can have a smaller diameter, which affects the center-to-center distance of the compartments. Because of this, it is important that the hoisting system designer ensure compatibility with the other components of the hoisting system (i.e., shaft, headframe, conveyance).

Regulations and other calculations and considerations involved in rope selection for friction hoists are discussed here. An example that considers factors of safety and other criteria is then presented. When an overall hoisting system is designed by a hoist manufacturer, it is in the owner's best interest to carefully check the proposal as there have been instances where undue emphasis has been placed on the capital cost of the hoist at the expense of higher operating costs of the overall system. These trade-offs in system selection must be carefully weighed.

When determining a factor of safety for friction hoist ropes in North America, it is common practice to follow the Ontario (Canada) regulations:

$$\text{factor of safety} = 8.0 - 0.0005\,d, \text{ but not less than } 5.5 \quad (17.5.40)$$

where d is maximum length of rope in feet

Ontario (Canada) regulations require the following factors of safety that may be used as guidance for other ropes in a friction hoist installation:

1. Balance or Tail Ropes — 7.0 on weight of rope (static load)
2. Guide or Rubbing Ropes — 5.0 on weight of rope (static load)
3. Bending Ratio. The required bending ratios are the same as for drum hoists. The bending ratio may exceed the regulatory minimum but for locked coil construction does not normally exceed 140.

Wire Rope Industries of Canada recommends the following minimum bending ratios for locked coil ropes, although all such specifications have been removed from US CFR requirements:

Rope Size	Bending Ratio (Locked Coil Rope)
Less than 1 in. (25 mm)	100:1
1 to 1¼ in. (25 to 32 mm)	100:1
1½ in. (38 mm)	110:1
2 in. (51 mm)	120:1
Greater than 2 in. (> 51 mm)	130:1

In addition to the regulatory requirements for factor of safety, the following items should also be considered when selecting ropes for friction hoists: (1) T_1/T_2 ratio, (2) drum tread, (3) deflection sheaves, (4) differential rope loads, and (5) tail ropes.

T_1/T_2 Ratio—T_1/T_2 ratio is the ratio of the static rope tensions on each side of the drum, where T_1 is the weight of loaded conveyance plus weight of ropes, and T_2 is the weight of unloaded conveyance plus weight of ropes. The maximum recommended value for a particularinstallation is a function of the coefficient of friction of the tread material, of the acceleration and deceleration required, and of the angle of lap of the rope on the drum. A typical maximum value of T_1/T_2 is 1.5. This is also a concern for brake torque application. Should T_1/T_2 become too high, the rope will slip on the drum; when this happens, the coefficient of friction drops and control is lost. There will also be tread damage due to friction temperature rise and abrasion.

Drum Treads—Selection of an appropriate tread material for the drum rope grooves and tread pressure are important. As the rope passes over the drum, the load, and thus the strain in the rope changes. This change in strain tends to cause the rope to creep (as opposed to slip) on the drum. A resilient tread material such as polyurethane tends to flow with the change in strain and reduce creep.

Typically, the average tread pressure (i.e., the sum of the rope tensions/projected rope area on the drum) of 300 psi (2069 kPa) with lock coil, 250 psi (1724 kPa) with flattened strand, and 200 psi (1379 kPa) with round strand ropes on plastic treads will give good life.

Deflection Sheaves—Deflection sheaves are used when the pitch circle diameter of the friction hoist pulley wheel (sitting above the shaft) is greater than the center-to-center distance of the compartments. In addition to positioning the hoisting ropes, the use of deflection sheaves has the advantage of increasing the angle of contact of the ropes on the wheel, permitting a higher T_1/T_2 ratio before slippage will occur.

However, deflection sheaves have the disadvantage of requiring additional torque during the hoisting cycle, increasing the height of the headframe, and putting reverse bending into the ropes, which can reduce their life. One rule of thumb used to position the deflection sheaves is that the ropes traveling at full speed should take a minimum of 0.5 seconds to travel from the drum to the deflection sheave. This rule, it has been claimed, alleviates the fatigue problem due to reverse bending since it provides the requisite time for the rope strands and wires to adjust their position.

Differential Rope Loads—A great concern with multirope friction hoists is the equalization of loads between the ropes. The two most common causes of differential loading are a difference in rope length or in circumferential lengths of each rope groove on the wheel.

Analysis of this problem and procedures to identify and correct the effects have been well covered in the technical literature. In the initial design stage, it is important that steps be taken to reduce its effects. Since there are no hard and fast rules, the following rules of thumb have been developed in the industry:

1. The ratio of rope lengths (skip in dump to hoist drum)/(skip at loading pocket to hoist drum) should be greater than 0.015.

2. The total number of revolutions of the drum on a trip should be less than 100.

3. The number of ropes should be kept to a minimum. There will be less difficulty maintaining equal tension on a four-rope hoist than on a six-rope hoist. In addition, a larger wheel will be required for a four-rope drum having fewer revolutions per trip, lower rope bending stresses, and lower sensitivity to tension variations. This requirement may conflict with rope size choice, however.

4. Rope attachments should have both fine and coarse length adjustment.

Hoisting Speed—There are no relationships attributable to the effect of speed. However, as the speed of hoisting increases, any inadequacies in the hoisting system are magnified.

Tail Ropes—In friction hoisting, the use of tail ropes makes it possible to balance the head rope weight and achieve a lower peak horsepower than for drum hoisting. However, the use of tail ropes presents some additional problems that must be recognized.

1. *Tail Rope Orientation*

Any object falling down a mine shaft is affected by the angular velocity of the earth and will move toward the east wall. A tail rope behaves in a similar way. If the shaft compartments are arranged east—west, the tail rope loops will open and close freely. However, if the compartments are north and south, the ropes could foul the separators at the shaft bottom. Thus, if tail ropes are used, then the orientation of compartments should be considered. The above described action results from a physical principle known as the Corioli effect and is a function of the hoist speed.

2. *Tail Rope Loop Diameter*

A natural loop diameter exists for each type of tail rope depending upon its construction. This should coincide as closely as possible to the center-to-center distance of the conveyance. The bending ratio is defined as the natural bending diameter divided by the rope size. It is possible to specify different rope construction and number of tail ropes to satisfy the loop diameter requirements.

Rope Construction	Natural Bending Ratio
34 × 7 Non-Rotating	46:l
18 × 7 Non-Rotating	60:1
6 × 7 Round Strand—Reg. Lay	90:1

3. *Tail Rope Loop Dividers*

According to Delorme (1977), to avoid mishap and possible mechanical damage through entanglement, the balance rope loop area should be well controlled in the sump. One method is by constructing a loop divider. The balance ropes should be restricted quite closely in the direction of their lateral movement.

Early contact with the dividers between individual ropes will provide good control in the sump area and also assist in ensuring that adequate stability through the shaft is maintained. Longitudinal movement, on the other hand, should not overly be restricted. The dividers should be designed to avoid trapping any material from spillage. Some operators have found that wire rope covered with rubber hose and placed in the same configuration (as shown in Fig. 17.5.14) has proved most successful in this connection. The elevation of the loop divider for good results is one loop diameter plus approximately 3 ft (0.9 m) above the bottom of the loop. Some operators have placed such dividers at several elevations in the sump area to ensure good control.

Calculation—The design of a friction hoisting system, including the selection of ropes, involves a knowledge of costs, shaft design, headframe design, and other factors beyond the scope of this chapter. However, an example of rope selection is presented based on previously mentioned technical considerations. The procedure involved in the analysis is as follows:

Step 1. Tabulate data.
Step 2. Calculate rope tensions.
Step 3. Check factor of safety of head ropes.
Step 4. Check factor of safety of tail ropes.
Step 5. Check bending ratio.
Step 6. Check T_1/T_2 ratio.
Step 7. Check tread bearing pressure.
Step 8. Check loop diameter of tail rope.

Example 17.5.4. Given the following friction hoist system:

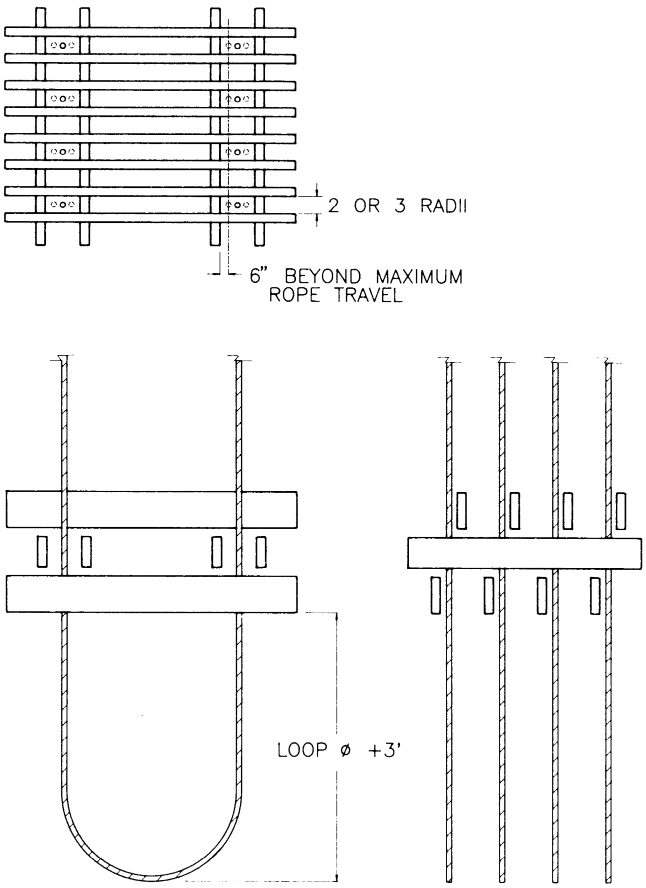


Fig. 17.5.14. Rope loop divider (Delorme, 1977).

Step 1. Assumed operating conditions.

a) Hoist wheel diameter	170 in. nominal	(4318 mm)
	172 in. new	(4369 mm)
	168 in. worn	(4267 mm)
b) Suspended loads		
Hoisting distance	1921 ft	(585.5 m)
Suspended ropes	2122 ft	(646.8 m)
Skip payload	105,000 lb	(47,628 kg)
Skip and rope attachments	136,000 lb	(61,690 kg)
c) Head ropes		
Type	6 × 27 flattened strand	
Number	6	
Diameter	2.125 in	(54 mm)
Ultimate strength	463,000 lb	(210,017 kg)
Weight/ft	7.8 lb	(3.53 kg)
d) Tail ropes		
Type	18 × 7 strand, non-rotating	
Number	4	
Diameter	2.625 in.	(67 mm)
Ultimate strength	493,000 lb	(223,625 kg)
Weight/ft	11.7 lb	(5.3 kg)

e) The difference between the hoisting distance (1921 ft) and the suspended rope length (2122 ft) is 201 ft. We will assume 100 ft of this is in the headframe and 101 ft is in the shaft bottom.

Select the proper hoisting rope and determine factors of safety.

Solution.

Step 2. Calculate rope tensions.

Wt/ft of 6 head ropes = 6 × 7.8 = 46.8 lb/ft
Wt/ft of 4 tail ropes = 4 × 11.7 = 46.8 lb/ft

Note: It is rare that a perfect balance is achieved, but a close match is desired.

The maximum rope tension T_1 occurs with the loaded skip at the dump.

T_1 = Rope from dump to drum 100 × 46.8	=	4680 lb
+ Skip weight		136,000 lb
+ Payload		105,000 lb
+ Tail ropes (1921 + 101) × 46.8	=	94,630 lb
		340,310 lb

T_2 = Rope from loading pocket to drum (1921 + 100) × 46.8	=	94,583 lb
+ Skip weight		136,000 lb
+ Tail ropes		4727 lb
		235,310 lb

Step 3. Check factor of safety of head ropes.

$$\text{SF} = \frac{\text{rope strength}}{T_1}$$

$$= \frac{6 \times 463{,}000}{340{,}310} = 8.16$$

This exceeds the requirement of 6.0 (according to CFR 57.19-21 and ANSI M11.1 standard for friction hoists at a depth of approximately 2000 ft).

Step 4. Check factor of safety of tail ropes.

From the calculation of T_1 the maximum weight of suspended tail rope is 94,630 lb.

$$\text{SF} = \frac{\text{rope strength}}{\text{suspended weight}}$$

$$= \frac{4 \times 493{,}000}{94{,}630} = 20.1$$

This significantly exceeds any regulations. This can be expected because the rope was selected on the basis of weight and not strength.

Step 5. Check bending ratio.

$$\text{Ratio} = \frac{\text{drum diameter (nominal)}}{\text{head rope diameter}}$$

$$= \frac{170}{2.125} = 80$$

This meets the requirements for good practice according to rope manufacturers and some codes that apply.

Step 6. Check T_1/T_2 ratio.

$$T_1/T_2 = \frac{340{,}310}{235{,}310} = 1.446$$

This is less than the 1.5 noted earlier as a typical maximum. The following should be borne in mind:

1. There is no code requirement for T_1/T_2.

2. A workable T_1/T_2 ratio is dependent on the coefficient of friction of the actual drum tread material and head ropes used, as well as angle of contact. The latter considerations enter with ground-mounted hoists that do not have as much contact as tower-mounted usually do.

3. Manufacturers apply a factor of safety to actual coefficients of friction to determine the T_1/T_2 for a particular application.

Step 7. Check tread bearing pressure.

$$\begin{aligned}\text{Average tread pressure} &= \frac{T_1 + T_2}{\text{projected area of ropes on drum}} \\ &= \frac{T_1 + T_2}{\text{no. ropes} \times \text{rope dia.} \times \text{drum dia.}} \\ &= \frac{342{,}736 + 235{,}431}{6 \times 2.125 \times 170} \\ &= 267 \text{ psi}\end{aligned}$$

Again there is no code requirement for tread bearing pressure, and acceptable pressures depend on tread material and rope construction. Varying the tread pressure changes the maintenance costs but not the safety of the installation.

Step 8. Check loop diameter of the tailrope.

Tailrope size	2.625 in.
Tailrope construction	18 × 7 NR
Bending ratio	60:1
Diameter of natural loop	= 2.625 × 60 = 157.4

This is marginally less than the drum diameter and is therefore satisfactory.

GUIDE AND RUBBING ROPES. Ropes used as guides and rubbing ropes may be of locked coil or half locked coil construction. These ropes are manufactured with a core of concentrically laid round wires covered with one or two layers of shaped wires. The outer layer of locked coil construction is always composed of interlocking wires of Z shape.

Some of the advantages of locked coil ropes as guides and rubbing ropes are as follows: (1) their smooth surface reduces friction, (2) they have a greater degree of rigidity, and (3) they have a higher breaking strength in relation to their diameter.

ADDITIONAL ROPE SELECTION CONSIDERATIONS. In addition to those factors covered in the previous sections, the following four items of importance should be considered in order to ensure that a particular hoisting rope will meet all of its hoisting requirements: (1) rope stretch, (2) rope torque, (3) rope harmonics and vibrations, and (4) rope sag. These are discussed in the following subsections.

Rope Stretch—When a skip is loaded at the loading pocket, the rope stretches. The amount of stretch, which has a static and dynamic component, has a number of significant consequences that must be considered in the design of the hoisting system. These are as follows:

1. The rope acts as a spring and helps absorb impact loads. The longer the rope, the greater the stretch, and the greater the amount of energy that can be absorbed. This is the reason why the codes permit reduction in the factor of safety with longer ropes.

2. When a hoisting system is being designed for balance hoisting using two skips, the amount of stretch must be considered when designing the overtravel allowance in the headframe. Under normal loading conditions, one skip will be positioned at the loading pocket after the skip in the headframe has completed dumping. If, however, a skip is not loaded, the stretch due to the payload will not take place. Should the empty skip be hoisted to the surface, it will be above the headframe dump (by the amount of rope stretch that did not take place) when the lower skip is at the loading pocket. For example, a 10-ton (9-t) payload on a 4000-ft (1219-m) long 1-in. (38-mm) rope would cause a stretch of about 4 to 6 ft (1.2 to 1.8 m).

3. Due to both the static and dynamic forces, the skip moves down during loading. Unless this movement is considered in the design of the loading pocket and the skip, considerable spillage could occur.

4. The static and dynamic rope stretch at the loading pocket must be considered in establishing the skip undertravel in the shaft bottom.

The formula for rope stretch or elongation in ft (m) is

$$\text{stretch} = \frac{PL}{AE} \tag{17.5.41}$$

where P is load in lb (kg), L is length of rope from the hoist to the loading point in ft (m), A is nominal rope cross-sectional area in in.^{2} (mm^2), and E is a constant for a given rope construction in lb/in.^{2} (Pa). E is analogous to Young's modulus. It is determined experimentally by rope manufacturers and takes into consideration the voids between the wires in the rope and the geometric change in the rope under load.

Values of E change as the rope stretches with use. Typical values are tabulated below:

Rope Construction	E, psi (kPa)
Full lock coil	14.3×10^6 (98.6×10^6)
Half lock coil	15.7×10^6 (108.3×10^6)
6 × 27 flattened strand	9.0×10^6 (62.1×10^6)
18 × 7 IWRC sinking	10.1×10^6 (69.6×10^6)
34 × 7 IWRC sinking	7.8×10^6 (53.8×10^6)
6 × 19 round strand, FC	9.0×10^6 (62.1×10^6)
6 × 19 round strand, WRC	9.9×10^6 (68.3×10^6)

The amount of dynamic stretch that takes place is proportional to the mass and velocity impact of rock feeding into the skip. These factors may be controlled (to reduce stretch) through loading pocket design. Dynamic stretch is commonly taken as equal to the static stretch produced by the load weight.

Rope Torque—Stranded ropes have a tendency to untwist. In drum hoisting, the rope is fixed at the drum, and at the conveyance the tendency to untwist is restrained by the guides and shoes. Consequently torque is developed in the rope. The counter torque exerted by the conveyance tends to produce wear on one side of the guides. When disconnecting the rope from the conveyance, the torque is released, and a dangerous situation develops if the end of the rope is not restrained and spins out of control.

Some mining companies are deliberately letting the torque out of the rope when they take rope end samples for testing. However, some experimental work, as reported by Dolan (1961), has shown that by allowing a rope to rotate, ultimate strength is lost.

One formula used for calculating the approximate amount of torque in a rope is

$$T = kPd \tag{17.5.42}$$

where T is torque in ft-lb, P is load in lb, d is nominal diameter of rope in in., and $k = 11 \times 10^{-3}$ ft/in. for Lang lay and $= 7 \times 10^{-3}$ ft/in. for regular lay.

Rope Harmonics and Vibrations—Vibrations in hoist ropes can be observed as "yo-yo-ing" at the collar, and in the case of drum hoists as ropes whip between the hoist and the head sheave. The causes of these vibrations are (1) jerking operations such as uneven braking, (2) the horizontal movement of the rope at the crossover points on the drum, and (3) the vertical movement of the rope at crossover points on the drum.

These vibrations cannot be avoided and normally do not cause problems. However, there have been cases where the frequency of the vibrations is a harmonic of the natural frequency of the hoist rope and severe and dangerous whipping has occurred. Such a situation is most likely with a long, flat span of rope between the hoist and head sheave. The following rules of thumb have been developed to assist designers to set up a good hoist and head sheave arrangement.

1. Keep the vertical rope angle as close to 45° as possible.
2. The slope distance along the rope, between the tangent points at the drum and the sheave, should not be an even multiple of the drum circumference.

The hoist designer should analyze the harmonic behaviors of the ropes and select the type of drum grooving which will avoid harmonic vibrations. This has been reviewed by Dimitriov and Whillier (1973). When unacceptable rope movements do occur, it is necessary to reduce hoisting speed or payload, or to install an intermediate tower with idler sheaves between the hoist and headframe.

Rope Sag—A rope between a ground-mounted hoist and a head sheave sags and takes the form of a catenary. With a heavy rope and a light load, the sag can be significant and cause problems at the sheave deck and the hoisthouse wall if not considered. Such a situation could occur during sinking a deep shaft when there is an empty bucket at the dump.

INCLINED (SLOPE) HAULAGE. The selection of ropes for an inclined shaft differs little from selection for a vertical shaft. The three major differences are (1) calculation of rope tensions, (2) sizing of sheaves, and (3) abrasive wear.

Calculation of Rope Tension—The tension in the rope is the sum of the gravitational, frictional, and accelerational forces. The gravitational force is the sum of the weights of the car(s), payload, and the rope multiplied by the sine of the angle of the incline. The friction force is commonly taken as 2.5% of the weight of the car and payload plus 10% of the rope load. To allow for acceleration, 10% is added to the gravitational and friction forces.

Sizing of Sheaves—Because of the space restrictions underground and the need for portability, deflection sheaves are commonly smaller than used elsewhere for hoisting ropes. Minimum *D/d* ratios are presented in Table 17.5.4.

Abrasive Wear—Abrasive wear is more likely to be a problem in a slope than in a vertical shaft and this potential problem should be addressed in rope selection. One approach is to select ropes with thicker wires such as 6 × 7. Some designers increase the factor of safety to allow more wear on the rope before it needs to be replaced.

Table 17.5.4. Minimum Ratio of Drum/Sheave Diameter to Rope Diameter

Rope Construction	Minimum Tread Diameter of Sheave or Drum
6 × 7 classification	72 × rope diameter
6 × 19 classification	45 × rope diameter
6 × 37 classification	27 × rope diameter
6 × 25 (Type B); 6 × 27 (Type H) 6 × 30 (Type G) flattened strand	45 × rope diameter
Rotation resistant	51 × rope diameter

Note: Conditions may exist where further reduction may be necessary. Under such conditions, service life would be affected and due precautions should be exercised. If possible, the hoist/sheave manufacturer should be consulted for guidance. Note that the Jan. 24, 1984, MSHA Standards do not define necessary *D/d* ratios.

17.5.5.4 Rope Testing Certification and Acceptance

It is common practice for the purchaser to require the manufacturer to certify that the rope will meet the nominal breaking strength. Certification usually includes a test report providing full particulars on the wires used to manufacture the rope as well as breaking strength in a failure load test to ultimate load. A test to destruction by an independent laboratory can be specified. However, such a test involves additional costs and is not normally specified in the United States.

17.5.5.5 Rope Operating Practices

The correct wire rope properly installed on well-designed equipment maintained in good working condition provides the foundation for satisfactory rope performance. In order to attain and sustain the required rope performance, it will be necessary that the rope be properly operated and adequately maintained. Well-developed rope operating practices should consider the following: (1) rope storage, (2) rope installation, (3) rope changing procedures, (4) rope operating practices, (5) equipment maintenance, (6) lubrication, and (7) inspection procedures.

ROPE STORAGE. Whether the rope must be stored temporarily or indefinitely, it should be indoors where it can be shielded from weather, corrosive fumes, and excessive heat that might dry the lubricant or fiber core.

ROPE INSTALLATION AND HANDLING. Ropes should be wound onto the drum under load. The direction of coiling should be determined by the accepted rule of thumb method, as shown in any rope handbook, and the individual turns should be tamped together using a copper hammer to avoid gaps between coils. Before cutting, the rope supplier or a technical handbook detailing correct cutting procedures should be consulted (Anon., 1980).

DRUM HOIST ROPE-CHANGING PROCEDURE. Rope changing procedures for drum hoists are relatively easy and are described in rope handbooks. Precautions must be taken, however, to ensure that adequate tensions are applied to provide correct spooling during installation.

FRICTION HOIST ROPE-CHANGING PROCEDURE. Rope-changing on friction hoists can be carried out using one of four methods.

1. *Method 1.* A rope-changing station is located midway in the shaft. Headropes are changed by chairing the conveyances at mid-shaft, where reels of new cable have also been positioned,

disconnecting both ends of the headropes and connecting one end of the old rope to one end of the new rope. The opposite end of the old ropes is connected to empty reels. All headropes are changed at one time by rotating the friction wheel and thus lifting the new ropes. The old ropes are wound on the empty reels. After the new rope has been hoisted into position and the old ropes wound on the spare reels, both ends of the new ropes are connected to the conveyance. Tail ropes are changed from the bottom of the shaft (see method 4).

2. *Method 2.* A portable rope winch located on the surface removes all head ropes one at a time, winding them on empty reels. New head ropes are lowered by this winch and installed on the conveyances. During this procedure, one conveyance is located at the top of the shaft, and one conveyance is located at the bottom of the shaft. Tail ropes are changed from the bottom of the shaft (see method 4).

3. *Method 3.* Both head ropes and tailropes are changed from the bottom of the shaft, using the hoist to raise the new ropes into position.

4. *Method 4.* In this method, the tail ropes are installed from the bottom of the shaft, following a five-step procedure: (1) locate no. 1 conveyance at the bottom of the shaft, (2) locate no.2 conveyance at the top of the shaft, (3) release the rope clamp on the bottom of no. 1 conveyance and start reeling the rope to be removed on a spare reel, (4) lower no.2 conveyance and remove rope clamp on the bottom of no.2 conveyance and complete winding up the end of the rope removed, and (5) install the new tail rope with the reverse procedure. The new tail rope will be hoisted into position.

The head ropes could also be installed by the use of an auxiliary rope-changing winch located at the collar elevation in line with the bending rotation of the hoisting ropes. The final method selected depends upon the amount of room available on the surface, orientation of hoist ropes, available space in the headframe, and personal preference.

ROPE OPERATING PRACTICES. Mine ropes are operated and maintained correctly when the following conditions are met:

1. Drum and sheave groove contours are correct.
2. Filler, starter, and riser strips are used on the drums where necessary.
3. Rope crossovers on the drum are prevented.
4. Shock loadings and overloads are avoided.
5. Vibration stresses are kept to a minimum.
6. Peening and abrasive actions are minimized.
7. Rope cutbacks are appropriately completed.
8. Cage-end attachments are refastened at regular intervals.
9. The rope is lubricated at regular intervals with the correct lubricant.

EQUIPMENT MAINTENANCE. Maintenance of equipment that comes in contact with the rope is important and should not be overlooked during the design or operating phase. Some items to be considered are the following:

1. Tight sheave and drum grooves will pinch the rope, restricting wire and strand movement.

2. Incorrectly designed rope crossovers on drum hoists will crush or flatten the rope, induce the core to pop, and severely scrub wires. This can be prevented through the correct design of filler, starter, and riser strips on the drum flange.

3. Faulty bearings can cause vibration stresses, which in turn cause fatigue fractures that are most likely to concentrate at the end attachment.

4. Corrugated deflection sheaves, drums and rollers can cause high unit pressures, aggravated wear, excessive rope vibration, and early wire fatigue. Correct measuring and trimming of tread diameters will prevent this problem.

LUBRICATION. Atmospheric moisture, shaft water, and chemical fumes are a few of the agents that will cause corrosion. This frequently leads to wire breakage and premature removal of mine hoist ropes. In mines with very wet and corrosive shafts, it is distinctly advantageous to apply a lubricant that will penetrate the interior of the rope and that contains a corrosion inhibitor. Lubricants with water-displacing properties are also available and are useful at end attachments.

INSPECTION PROCEDURES. In order to achieve optimum rope performance, an inspection procedure that requires findings to be recorded, fully diagnosed, and translated into appropriate action should be instituted.

The presence of the foregoing not only allows the operator to estimate the remaining safe working life of the rope but can also pinpoint the source causing such damage to the rope. For example:

1. Marked reduction in diameter may be due to a loss of rope's elasticity, excessive abrasion, or corrosion.

2. Broken wires may be due to worn out sheave, or drum grooves, tight oversized sheaves or gappy coiling.

3. "Kinking" and "birdcaging" of working ropes generally result from a sudden release of load.

The term "kinking" in rope use describes the snarling back of a rope on itself in the form of a loop resulting in a bend in the rope when straightened out, while "birdcaging" is applied to the springing of strands away from the core or inner strands.

A thorough evaluation of all facts obtained through routine inspections, examinations and tests, and compliance with statutory mining acts is needed to decide how early a rope should be discarded for the sake of safety, and how late can it be removed for the sake of economy.

17.5.5.6 Rope Attachments

Wire rope end attachments are as important as the rope to which they are fastened. Therefore, the selection of the correct type of attachments should be based on an understanding of how they affect ultimate rope serviceability and efficiency.

There are only two ways to attach something to a wire rope: (1) forming a loop in the rope, and (2) attaching a fitting to the rope. Loops are made either by splicing the rope to itself, or by use of clamps or wedges. Fittings secured directly to wire rope can be applied by cold forming (snagging) of the metal in the fittings, by pouring a liquid material such as molten zinc, or by a wedging arrangement. In this section, some of the most commonly used attachments are described.

FACTOR OF SAFETY. The factor of safety for rope attachments is not specifically covered in CFR *Title 30* (Anon., 1989). However, it is considered good engineering practice to design the rope attachments to be stronger than the rope, that is, the rope will fail before the attachments. British codes require a factor of safety of 10 based on static loads.

When an attachment is made to a rope, the attachment commonly weakens the rope locally. The efficiency of an attachment is expressed as the percentage of strength left in the rope after installation of the attachment. The efficiencies to be expected from the various types of attachments when correctly installed are presented in Table 17.55

CAPPELS. Cappels are rope fittings attached directly to the rope. These types of fittings consist of a pair of wedges grooved to suit the particular rope diameter and interlocked to ensure complementary movement. A number of bands driven over the diverging exterior surfaces of the cappel limbs provides the initial compressive force to ensure that the wedges grip the rope. The limbs' internal surfaces have been machined to fit the external surfaces of the wedges. A safety block is fastened on the end of

Table 17.5.5. Rope Attachment Efficiencies

Attachment	Size, in.	Efficiency, %
1. Thimble and clips	all	90–100
2. Thimble and clamps	all	90
3. Thimble and mechanical splice	less than 1	92–95
	$1\frac{1}{8}$–$1\frac{7}{8}$	90–92
	larger than 2	90–98
4. Thimble and hand splice	$\frac{7}{8}$–$2\frac{1}{2}$	75–85
5. Spelter or resin socket	all	100
6. Wedge socket	all	90
7. Swaged socket	all	95–100
8. Wedge cappel	all	100

Conversion factor: 1 in. = 25.4 mm.
Source: Anon., 1981b.

the rope, protruding beyond the bottom of the wedges. Several types of cappels are used for hoist ropes including the wedge and loop.

THIMBLES. In North America, most mine hoist ropes on drum hoists are terminated with a wire rope thimble and U-bolt cable clamps. With this type of attachment, a loop is formed in the rope and the thimble placed within this loop. The free end of rope is then clamped to the long end to secure the thimble.

Thimbles are made of cast steel, whereas the clamps, shackles, and pins are made of 1.5% manganese steel.

SWIVELS. When winding with balance ropes, the lay length tends to shorten as each conveyance in turn approaches the bottom landing. It is for this reason that swivels are provided in order to allow the ropes to spin and regain their normal length. Shaft conditions and loadings influence the type of swivel used.

ATTACHMENT MAINTENANCE AND INSPECTION. It is desirable that all types of attachments be installed as recommended by the manufacturer. If thimbles and clips are to be used, then the manufacturers' recommendation for the correct number of clips, amount of turn back, clip positions, and correct torques should be followed closely. If a wedge-type connection is being applied, then care should be taken to properly clean the rope surface. Frequent applications of cement dust and subsequent steel brushing will remove all the surface lubricant.

In some jurisdictions, statutory mining regulations requires that all types of attachments be inspected both visually and nondestructively on a regular basis. In other parts of the world, removal criteria for attachments have been established.

17.5.6 SHAFTS

A *shaft* is a vertical or inclined primary opening in rock that provides access to and serves various levels of a mine. Primary openings are those which are considered to be permanent and require a high degree of safety. Chapter 17.4 discusses design and construction details.

17.5.6.1 Shaft Design Procedure

A suggested procedure for the design of shafts follows. It is important that the steps be followed in the given sequence. It should be noted that the process is iterative and involves working through the process several times before an optimum design is achieved. The shaft design procedure involves the following steps: (1) define purpose of the shaft, (2) identify location and determine inclination, (3) determine the number of hoists required, (4) determine the size of conveyances and compartments, (5) determine the arrangement of compartments, (6) determine the exterior shape, (7) design interior members (guides, buntons, etc.), (8) design shaft lining, (9) check ventilation characteristics, (10) determine ground stabilization and temporary ground support, (11) determine shaft collaring method, (12) determine shaft sinking method, and (13) evaluate and modify starting at item 1.

17.5.6.2 Purpose of Shaft

One of the first items to be examined when designing a shaft is to identify its intended purpose. Shafts usually fall into one of the following categories: (1) production (ore and waste handling), (2) service (personnel and materials handling), (3) ventilation (upcast or downcast), (4) exploration (for defining mineral deposits), and (5) combination of the above.

17.5.6.3 Location and Inclination

LOCATION. Generally, the location of a new shaft at the mine site is determined after establishing the following: (1) mine surface layout, (2) location, dip, and extent of the ore body, (3) number of working levels to be considered, (4) location of ore and waste handling facilities, (5) water collection sump requirements, (6) safety and stability of the shaft pillar, and (7) the future planned expansion of the shaft. The bottom of the shaft, where possible, must also be in a stable formation and should be able to facilitate any planned, future redeepening. The location of the shaft must also avoid adverse surface features and should satisfy surface layout plus logistics of surface and underground ore and waste handling facilities.

INCLINATION. The dip of the ore body is the main factor involved in deciding to sink either a vertical or inclined shaft. A secondary factor is the relative ground strength and geologic formations to be encountered by the proposed shaft.

The main advantages associated with vertical shafts are (1) hoisting speeds are greater, (2) shaft maintenance costs are lower, (3) sinking can be carried out faster, and (4) sinking can be achieved in almost any type of ground.

Inclined shafts (slopes) are generally associated with inclined (dipping) ore bodies where the length of crosscutting to reach the ore body from a vertical shaft becomes longer with the increase in depth of the shaft. They have the advantage of minimizing development to reach the ore from the shaft and are used frequently in coal mines.

17.5.6.4 Number of Hoists

The number of hoists (and conveyances) required to meet hoisting demands has a major impact in the design of a shaft (see procedure, 17.5.3.5).

17.5.6.5 Compartment Size

The cross-sectional area of a particular compartment (horizontal area) is dependent upon its use. In order to determine the most appropriate size, it is necessary to list the items to be transported in the compartment, determine their approximate dimensions and weight, indicate their direction of flow and their approximate quantities. Another important factor to be considered is the size of the shaft sinking bucket. After having determined the size of compartments for cage, counterweight, and skips, the remaining area is then divided to accommodate ventilation, manway, and pipe facilities.

SIZE OF SKIP COMPARTMENT. The skip capacity is determined by such variables as the hoisting capacity, material density, lump size, and vertical height available in the headframe. Generally speaking, the skip dimensions and cross-sectional area are determined by the skip supplier. Good design practice allows the following clearances:

Skip compartment size:

face to face on guides = outside dimension of skip guide shoes + 1 in. (25 mm)

front to back = outside dimension of skip + 3 in. (75 mm)

SIZE OF CAGE COMPARTMENT. Cages are generally used to transport personnel and materials. The following clause of CFR *Title 30* (57.19-66 Mandatory) applies in the United States:

> "In shafts inclined over 45°. . . . Each person shall be provided a minimum of 1.5 ft^2 (0.14 m^2) of floor space."

The cage capacity is determined by the cage supplier based on a knowledge of the following factors (1) the amounts of material to be transported, (2) the dimensions and weights of the major pieces of equipment to be lowered and raised, (3) the number of personnel to be lowered and raised per shift, and (4) type of conveyance preferred (i.e, single deck, double deck, combination skip/cage, etc.)

After establishing the cross-sectional area for the cage, the cage compartment is sized the same way as the skip compartment to provide similar clearances.

SIZE OF MANWAY COMPARTMENT. From CFR *Title 30* (57.11-37 Mandatory): Ladderways constructed after November 15, 1979 shall have a minimum unobstructed cross-sectional opening of 24 by 24-in. (610 by 610 m) measured from the face of the ladder.

(57/11-41 Mandatory): Fixed ladders with an inclination of more than 70° from the horizontal shall be offset with substantial landings at least every 30 ft (9.1 m) or have landing gates at least every 30 ft (9.1 m).

In addition to the above, it is necessary to provide adequate clearance at the landing platform to allow a person to move from one ladder to the next offset ladder. Generally, an additional floor area of 36 by 24 in. (914 by 610 mm) is provided.

SIZE OF COUNTERWEIGHT COMPARTMENT. The counterweight compartment can be of any size or shape as long as it provides sufficient space and clearance for the safe passage of the counterweight with the same clearances as for skips and cages.

SIZE OF SERVICE COMPARTMENT. There are no hard and fast design rules for sizing the services compartment except that the cross-sectional area of the compartment should be adequate to provide the following: (1) separate locations for communications (signaling) and power lines, (2) space for locating air, water, pump, backfill lines, (3) clearance for pipe fittings, and (4) space for pipe column brackets.

17.5.6.6 Compartment Arrangement

The arrangement of the compartments in relationship to one another depends upon the surface layout, underground layout, and the type and size of hoist to be used. Because these design details are not known precisely during the initial design stages, one must first make a "best estimate" and be prepared to change the arrangement as the final design evolves.

On surface, the orientation of the compartments in relation to the hoist and headframe is of major importance. Good rope practice for drum hoists requires that the location of the hoist in relation to the shaft and headframe be in line to provide acceptable rope fleet angles.

For friction hoists, the center-to-center distance on the compartments is a major consideration for determining the size of the drum, the necessity for deflection sheaves, and the type of rope construction for the tail ropes.

17.5.6.7 Exterior Shape

The exterior shape of a shaft is established by considering ground stability factors. A circular shape provides better resistance to deformation by lateral pressure. In fair to competent formations, an opening excavated in this form is generally self-supporting. In weak formations, the circular shape is adaptable to a variety of lining materials, concrete, steel segments, or cast-in-place concrete. Competent formations can be excavated to a rectangular shape and be generally self-supporting. An elliptical shape offers better support than a rectangle but not as good as a circular one. The geotechnical considerations leading to the selection of an exterior shape are beyond the scope of this discussion, but their importance in the overall design process cannot be overstated (see Chapter 10.5). Also shafts may have different exterior shapes at different depths, reflecting the different strata encountered.

17.5.6.8 Interior Members

Guides and buntons are the main vertical and horizontal structural elements in a mine shaft conveyance system. The primary function of these members is to facilitate running of the shaft's conveyances. Their characteristics influence not only speed of operation and amount of maintenance required, but also production and ventilation costs. Therefore, the choice of sets and guides is of great importance to achieving operational cost savings.

SHAFT GUIDES DESCRIPTION. Shaft guides are used in vertical shafts to keep the skips, cages, and counterweights in proper shaft position. The two types of guides in use are (1) fixed guides (wood, rail, and steel), and (2) rope guides.

Wooden Guides—Wooden guides are supported by the horizontal sets installed in the shaft. They are favored where conveyances require safety devices. This is due to the fact that the decelerating characteristics of safety dogs on wood are more reliable than on steel or rail. When hoisting on wooden guides, a speed of 2200 fpm (11.2 m/s) with a medium-sized skip is the practical upper limit. Wooden guides are subject to changes in dimensions due to changes in moisture content. They are also subject to wear to a greater extent than steel guides. Wooden guides generally are not used in shafts where fire hazards exist, such as in coal mines.

Rail Guides—Rail guides are common in inclined shafts. These guides vary in size, with the larger rails giving a wider bearing surface for the wheel. The steel rail provides a harder surface but presents a smaller wear area; this creates more wear and maintenance on conveyance guide shoes and wheels.

Steel Guides—Structural steel tubes (SST) sections are becoming more popular as guides for production shafts where higher speeds and heavy loads require a fairly rigid and stable section. With steel guides, the conveyances are usually equipped with adequately sized guide roller assemblies to minimize wear between the guides and shoes on the conveyance. When hoisting on steel guides, a speed of 3000 fpm (15.2 m/s) is a practical upper limit.

Rope Guides—Rope guides are also used for guiding conveyances in vertical shafts. These are generally used with multirope hoisting systems where the effect of hoist-rope torque is minimal or nonexistent. Half locked coil ropes are used for this service. Rope guides do not require any intermediate support, thereby eliminating the steel sets required for fixed guides. Since rope guides are not able to withstand the horizontal forces of loading and unloading, the conveyance must be supported by other means in these areas. Fixed guides are used at the extremes of travel, and a retractable guiding system should be considered at intermediate levels. There are several schemes by which rope guides are suspended and tensioned, and the selection of any one of these will depend on the particular installation.

Because rope guides eliminate the need for horizontal buntons, the ability of the shaft to handle airflow is enhanced. In a large shaft, with multiple conveyances traveling at the same time, the aerodynamic effects can cause unexpected rope deflections. These effects should be studied during the design stage. Since rope guides are the smoothest of all guides, hoisting speeds up to 4000 ft/min (20 m/s) are feasible.

SHAFT BUNTON DESCRIPTION. Shaft buntons are horizontal members used to divide the shaft into compartments. They also support and carry the shaft guides, pipes and cables. Buntons can be of any cross-sectional shape and can be installed in a shaft at any spacing. However, the correct design and layout of buntons and their installation at a suitable spacing minimizes the air resistance of shafts and hence reduces ventilation operating costs.

GUIDE AND BUNTON STRUCTURAL DESIGN. Although there are no universally accepted loading parameters for the design of the shaft members, buntons and guides may be designed using empirical static loadings, generally accepted in the industry, and discussed in the next segments.

Bunton Loading—The static loads considered acting on set members are

1. The known weight of steel to be supported.
2. The vertical friction load on the guides transferred to the set members. The friction load is assumed equal to 3% of rope end load, where rope end load is calculated by adding the weight of conveyance, the attachments, and the payload in the conveyance.
3. A horizontal load transferred from guide members to set members assumed equal to 10% of rope end load.

Guide Loading—The static loads considered acting on the guide members are as for buntons. As mentioned earlier, there is no general consensus on these loading parameters and they have been challenged by several authors with diverging opinions (Bently, 1973; Backerberg, 1970; Van Wyk, 1961).

17.5.6.9 Shaft Lining

Shaft lining may be comprised of shotcrete, cast-in-place concrete, or steel and cast iron tubbing. When a lining is required, the selection depends on a number of design considerations that relate to the strata being excavated. For further discussion, see Chapter 17.4.

SHOTCRETE. Sprayed concrete, 5 in. (125 mm) or more in thickness, may be used to control immediate raveling and weathering of the shaft walls. To date, shotcrete has not been used extensively to provide long term support, nor as the only means of lining in shafts.

CONCRETE. The use of concrete for permanent support is becoming more common. Shafts may be fully lined or partially lined with "concrete rings" with areas of open ground between the rings. Common spacing used for ringed shafts is 4-ft (1.2-m) high concrete rings with a 4-ft (1.2-m) open area between, resulting in an 8-ft (2.4-m) set interval. Concrete is generally placed 12 in. (305 mm) in thickness, with a minimum of 10 in. (254 mm).

STEEL OR CAST IRON TUBBING. Although the use of concrete as a lining material is becoming more common, there are certain geologic conditions where steel or cast iron tubbing is a more appropriate material. Concrete is suitable when water pressures are less than 250 psi (1724 kPa) and when the stress field is uniform and compressive in nature. Steel is more suitable for a completely water-tight lining and is capable of withstanding tensile stresses. These stresses could arise if the shaft is subjected to bending loads caused by subsidence or buckling loads caused by a non-uniform horizontal stress field.

Steel lining can be butt-welded to give permanent watertightness and has a higher buckling safety factor than tubbing. Steel is particularly useful as a lining for shafts sunk by drilling methods. Cast iron tubbing consists of several segments bolted to form a ring and requires a lead gasket to be placed between segments.

DESIGN CONSIDERATIONS FOR SHAFT LINING. The design requirements for lining a structure is governed by the required duty of the shaft and environmental conditions in which it is constructed. Careful consideration of the following points should be made in the design of structural lining: (1) precise behaviour of the rock mass in situ, (2) self-supporting action of the rock itself, (3) effect of separation of the lining due to certain vertical and horizontal movements of the surrounding rock formation, (4) construction of watertight linings, and (5) economics of the lining structure.

The design of steel or cast iron linings is a much more complex process, since the buckling characteristics of an elastically embedded hollow cylinder with external pressure must be evaluated. Although the details of such a design are not discussed here, several reports on the subject are available in the technical literatures (Ostrowski, 1972; de la Vergne and Cooper, 1983; DeHart, 1983).

17.5.6.10 Ventilation Characteristics

The shaft is generally the largest single contributor to total mine air resistance when it is equipped with rigid guides for hoisting purposes. The resistance consists of two components: (1) frictional resistance, caused by viscous drag in the thin air layer at the shaft walls and the periphery of the vertical elements (guides, pipes, cables, ropes, etc.) of the shaft; and (2) shock resistance, due to the transverse sets (buntons and dividers), completely immersed in the airstream, thereby obstructing the flow, and recurring at intervals throughout the shaft.

These transverse elements create turbulence in the airstream with corresponding loss of flow energy. The turbulent waves below buntons and conveyances often contribute very much more to the total shaft resistance than the skin friction of the shaft walls. The forces caused by the buntons are parallel to the airstream, and proportional to velocity pressure, frontal area, surface texture, and shape. Because the frontal area of all the buntons in a shaft is likely to be more than the frontal area of the cages and skips, the aerodynamic design of the bunton is of importance in the design of a mine shaft.

A reduction in the shaft resistance can be achieved through (1) improving the aerodynamic effect of horizontal obstructions such as buntons, (2) minimizing the frontal area of the set members, and (3) increasing the spacing between the horizontal obstructions. Practical experience has indicated that there is little ventilation advantage to be gained from spacing greater than 15 ft (4.6 m).

General treatment of mine ventilation theory and practice is offered in Chapters 11.6 and 11.7.

17.5.6.11 Ground Stabilization and Temporary Support

Table 17.5.6 identifies three types of ground support problem encountered in shaft sinking and their corresponding solutions. For a detailed discussion of ground control, see Chapter 10.5.

Table 17.5.6. Ground Support Problems Encountered in Shaft Sinking

Problem	Solution
Ground is too weak or wet to allow any type of excavation	Pre-grout, freeze, drill and pump water wells.
Ground requires support during excavation	Rock bolts, screen, shotcrete, limit amount of unsupported ground.
Shaft requires permanent protection to allow hoisting	Install timber sets, steel sets, pour concrete lining either partial or rings, install steel or cast iron liner.

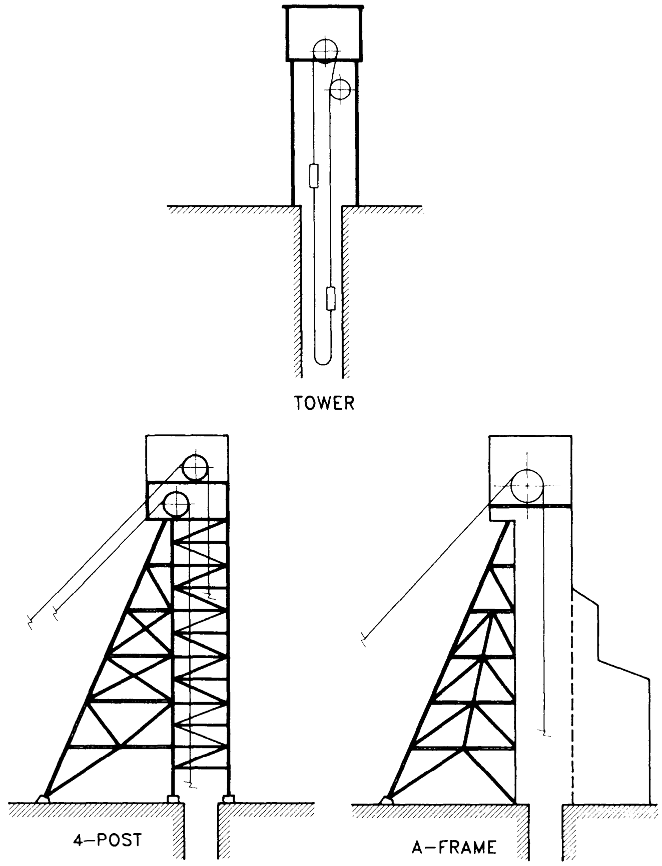


Fig. 17.5.15. Types of headframe structures.

17.5.6.12 Shaft Collar

Shaft construction begins with the excavation and forming of the shaft collar. The collar assists in forming the desired shape for the initial length of the shaft and provides the necessary stability and alignment for the remaining length of shaft to be excavated. Completion of the collar also allows erection of the headframe, sheave deck, and hoisting facilities for the sinking operation. The collar acts as a barrier to prevent water and soil from entering the shaft.

Collar construction methods are beyond the scope of this chapter but include steel sheet piling cofferdam, drilled interlocking concrete piles, steel walings with sheeting, drop caisson, drilling, pregrouting and sinking, open cut and dewater with well points, and deep well pumping.

17.5.6.13 Shaft Sinking

Following completion of shaft collaring, the actual shaft sinking is carried out usually by one of the following methods: (1) conventional methods of drill-and-blast or pilot raising and slashing, or (2) mechanized methods of raise boring or blind shaft boring. These are discussed in Chapter 17.4.

17.5.7 HEADFRAMES

The basic purpose of a headframe installed over a shaft is to support the sheave wheel over which a hoisting rope passes for raising or lowering conveyances. The construction of a headframe is also necessary to allow dumping of hoisted materials aboveground. Structurally, there are two types of headframes: (1) headframe structures with backlegs such as an A-frame, and (2) four-post and six-post and headframe structures of tower form (Fig. 17.5.15).

17.5.7.1 Construction Materials

Headframes may be constructed of timber, steel, or concrete. Modern trends to high-capacity hoisting have necessitated the construction of very large headframes, which effectively preclude the use of timber as a construction material.

COMPARISON—STEEL VS. CONCRETE. Butler and Schneyderberg (1981) outlined the principal criteria that should be used as a basis for a realistic evaluation of the merits of concrete and steel for a headframe. A summary of their arguments is given in Table 17.5.7.

Although the arguments given appear to favor concrete headframes, it should be noted that both concrete and steel can produce economically viable, structurally compatible, and visually appealing headframes for any new development.

Advantages of concrete structures include (1) effective use can be made of space within the structure for tower-mounted friction hoists, (2) when used in conjunction with ground-mounted drum hoists, the hoist may be located at any position around the headframe, (3) maintenance costs are low, (4) high damping effect of the massive concrete structure on shock and vibration, (5) simplicity of collar layout and interconnection with service buildings, (6) provision for future expansion, and (7) durability and resistance to corrosion. Disadvantages of concrete include (1) lengthy construction period, (2) higher capital cost in comparison with a steel headframe, and (3) no salvage value.

For steel structures, advantages include (1) lighter foundation loads, (2) adaptability to minor changes, (3) salvage value, (4) capital cost saving in remote areas, (5) can be pre-fabricated into transportable lengths for on site assembly, and (6) speedy construction. The main disadvantage of steel is that it is prone to corrosion, and thus not suitable as a production headframe for corrosive ore (e.g., salt and sulfides) unless the steel is treated.

17.5.7.2 Headframe Design Considerations

A successful design must be based on an overall integrated hoisting system design, which involves a critical examination of the impact of at least the following list of items: (1) loads, (2) foundations, (3) mine services, (4) sinking provisions, (5) equipment monitoring facilities, (6) conveyance handling and rope handling, (7) heating and ventilation, and (8) others, such as local mining regulations.

Table 17.5.7. Design Considerations for Concrete and Steel Headframes

Consideration Criterion	For Concrete Structures	For Steel Structures
Foundation	Requires sound bedrock	Lighter steel headframe can be erected on non-corrosive weak ground formations.
Construction	Slipformed concrete tower can (in favorable weather) cut construction time.	Can be erected in all weather.
Accommodation of shaft sinker	Can accommodate temporary sheave deck to facilitate shaft sinking.	Cannot facilitate installation of temporary head gear without extensive modification to the permanent structure.
Additional facilities	Provides sufficient enclosed space to house hoist's electrics, control room, and ventilation equipment.	Steel structure usually not capable of carrying extra loads.
Stiffness and damping effect	Concrete structure is stiffer to resist vibration during high speed.	Steel structure is more prone to vibration unless design is based on sophisticated dynamic studies.
Plant layout damping effect	Flexible to logistics, stiffer to resist vibration during high speed hoisting and in strong wind environment.	Steel structure is more prone to vibration unless design is based on sophisticated dynamic studies.
Adaptability to changes and future additions	Due to high load-bearing capability, concrete towers tend to be more flexible for future planned expansion.	Steel headframe, although amenable to minor changes, requires costly modification for major expansion.
Reclamation	Concrete headframe has no salvage value.	Steel headframe can be relocated and reused and has good salvage value.
Corrosion, durability and maintenance	Virtually immune to corrosion; can be made air-tight to isolate sensitive equipment from outside corrosive atmosphere.	Susceptible to corrosion, thus incurs higher maintenance cost; requires periodic repainting and cladding replacement.
Aesthetics	Visually more appealing to most people.	On unbroken flat horizons, large structure is often an aesthetic problem.
Cost	Depending on location and availability of the material, concrete headframe can be less costly.	In remote locations a steel headframe construction is usually cheaper.

Source: Butler and Schneyderberg, 1981.

LOADINGS. All headframes should be designed to withstand a combination of loads, comprised of the following: (1) dead load consisting of the weight of the headframe, sheave wheels, ore bins, and contents; (2) live load from hoisting at maximum capacity; (3) breaking loads of the ropes when the conveyance is stopped by the crash beams or if it is jammed in the shaft (breaking stresses introduced by an overwind are usually transferred to the structural system, often through an arrestor gear, and thus the supporting structure must be critically analyzed with a thorough understanding of the arrestor mechanism); (4) wind load, the intensity of which depends on the location, height, and shape of the structure; (5) snow load; and (6) temperature and seismic stresses, taken from published data relating to the geographic area of the shaft location.

FOUNDATION. Simply stated, the foundation should be capable of withstanding and absorbing stresses imposed by the structure and associated loads.

MINE SERVICES. The design of the headframe must suit surface layouts for providing the essential mine services. When possible, it should be located close to the shaft collar house, waiting area, lamproom, first-aid room, hoist house, maintenance shop, changing house, and administrative offices. The type and design of headframe must also take into account the shaft internal arrangement, position of the hoist with respect to the shaft, and the position of the sheave steel. Facilities for several key concerns listed must be considered and adequately developed during the conceptual design stage: (1) arrangement of skip dumping area in relation to ore and waste storage and handling facilities, (2) clearance required for installation, maintenance, and removal of conveyances, (3) lifting equipment required for maintenance of conveyances, and (4) access required for maintenance of conveyances, dump areas, etc.

In the case of headframes for drum hoists, provision must be made for (1) lifting equipment and access for maintenance of sheaves, (2) access and anchorage facility to support the ends of hoist ropes, when doubling down for pretensioning drum end coils, and (3) sub-collar loading access for double deck cages.

According to Cook and Werner (1966), the design of headframes for friction hoists should address the following: (1) supporting facility at predetermined elevations for hoist or balance ropes and conveyances or counterweights during installation and changes; (2) clearance for installation of conveyance arrestor gear; (3) provisions for handling shaft ventilation air and for sealing machinery rooms; (4) arrangement of MG sets close to hoist motors to minimize runs of heavy bus bars; (5) lifting equipment access during head and tail rope changing; (6) support, accommodation, inspection, removal, and replacement facilities for rope guides where used; (7) ventilation to specific machines and areas, controlling moisture, dust, and temperature where required; (8) accommodation of sinking arrangements; (9) accommodation of all services for air, water, communications, and power lines to the shaft and to the headframe-mounted equipment; (10) the accommodation of stairwells and elevators; and (11) lighting and lightning protection.

SINKING PROVISIONS. After the decision has been made to sink a shaft, considerations should be given to the type of surface sinking plant (such as headframe and hoist) that is to be used for shaft sinking. In particular, it should be determined whether the sinking plant is to be temporary or is to make use of the permanent headframe and/or hoist.

Temporary Headframe—Generally, a temporary headframe is used during shaft sinking by the contractor. These headframes may be a wood or steel structure in modular form. Some steel headframes may be fabricated from hollow structural steel, which combines strength with light weight, and facilitates easy erection and dismantling. Headframes of this type are portable and have a high resale potential.

Design criteria for a temporary sinking headframe are the same as for a permanent headframe. Its height depends upon the method of rock disposal and size of bucket. The sinking headframe should embody features for dumping buckets or skips, for protecting personnel on the surface and in the shaft from falling pieces of rock while dumping, and minimizing work of top men in dumping buckets and removing broken rock. The design of temporary headframe structures should accommodate an adequate-sized storage bin and sufficient clearance for a rock disposal vehicle.

Permanent Headframe—Current trends favor installation of a permanent headframe for the sinking operation, as it tends to be more economical and time saving. The permanent main skeleton can be designed to facilitate erection and installation of the sinking sheaves at a suitable elevation. The structure can then be finished and equipped during the shaft sinking operation. The use of the permanent structure as a sinking headframe can make the permanent hoisting facilities available at an earlier date following completion of the shaft sinking program.

EQUIPMENT MONITORING FACILITIES. Two main hoisting incidents, which can be detrimental to personnel and equipment and require monitoring, are (1) overtravel at the shaft extremities (i.e., sump, headframe), and (2) overspeed during the duty cycle. In order to protect against these incidents, it is mandatory that monitoring and safety devices be installed in the headframe.

Operation of all hoists are monitored mechanically by controllers, such as the Lilly hoist controller. These are powered off the hoist drums and check hoisting speed, overtravel at each end of the wind, and deceleration in the end zones. Additionally, track limit switches and conveyance arrestors must be incorporated. The function of track limit switches is to provide independent overtravel protection in the headframe.

Ultimate protection against overwind is provided for all friction hoists by conveyance arrestors. These are installed at the ends of travel in each hoisting compartment. The headframe arrestors are designed to retard an ascending conveyance from 70% of full speed at an acceleration of 0.9*g*. Shaft arrestors are designed to retard an empty descending conveyance from full speed at 2*g*.

Where applicable, the headframe should accommodate installations of a crash beam and catch gear. With drum winders, the crash beam is located below the sheaves in the headframe. In the event of an excessive overwind, it stops the conveyance from traveling beyond this point. Should a conveyance be involved in an overwind situation in which the hoist ropes breaks, the conveyance and tail ropes will fall down the shaft, causing damage. To prevent this, catch gear should be provided in the headframe to "catch" the conveyance.

CONVEYANCE AND ROPE HANDLING. In the design and layout of a headframe, provisions must be made for the handling and changing of conveyances, hoisting ropes, and rope guides (if applicable). Friction hoisting often requires the use of very large skips and cages. Because of their size, the service and handling of these conveyances, and the tying off the head and tail ropes associated with them, considerable problems can arise. In the friction hoist headframe, these can be facilitated through the installation of an adequately sized overhead traveling crane. In the drum hoist headframe, handling and changing of conveyances and ropes are relatively easy and are usually achieved through the use of mobile cranes, rope blocks, and tugger hoists.

HEATING AND VENTILATION. In warm climates, the headframe may be left open. However, in colder climates, it is necessary to design the headframe as an enclosed structure with adequate provisions for a heating system. In tower-mounted friction hoisting systems, with all of the hoists, motors, and controls, etc., enclosed in the headframe, it is necessary to provide a space ventilation system, supplying filtered air for both cooling and heating.

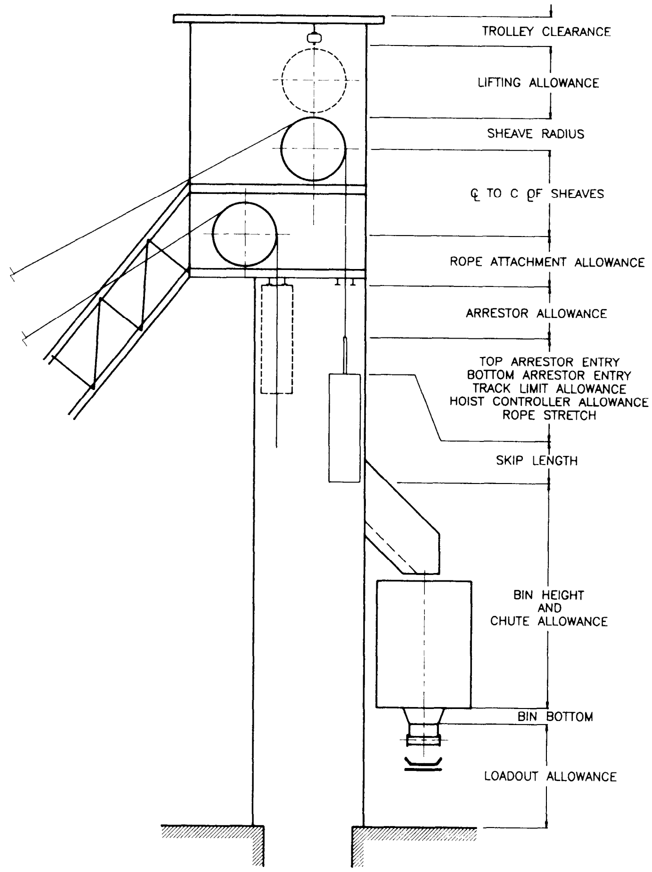


Fig. 17.5.16. Vertical general arrangement of headframe.

MINING REGULATIONS. Mining operations, including the design and construction of the headframe and shaft facilities, are subject to local, state, and federal regulations that relate to, among other things, safety of permanent structures and environmental protection. These regulations affect the design and construction of mine facilities.

HEIGHT. Due to today's increasing public awareness of the aesthetic impact of industrial development, the height of the headframe should be considered. This is especially true for locations in flat terrain where any large structure can be seen for long distances. The height of a headframe is generally based on (1) storage bin capacity requirements for ore and waste, (2) clearance for loading facility under the storage bin, (3) arranging that the resultant force from headframe loadings falls within the backleg post, and (4) providing minimum clearance for conveyance overtravel beyond its normal travel to the headsheaves. Of the above, the determination of the minimum clearance for overrun distance is the principal design consideration. This minimum clearance is obtained after establishing the value of three separate allowances (1) operating allowance, (2) rope stretch allowance, and (3) stopping allowance (Fig 17.5.16).

Operating Allowance—An operating allowance is required to allow for variations in repeatability of the final stop position of the conveyance in the headframe. In multilevel hoisting with drum hoists, this tolerance compensates for differences in coiling and rope stretch. In the case of friction hoists, the allowance compensates for creep during resynchronization and also allows a reasonable gap for rope adjustment due to permanent rope stretch.

Rope Stretch Allowance— This is included to avoid contact with the track limit switch when empty conveyances are hoisted. The rope stretch allowance generally includes an additional l-ft (0.3-m) margin as well as the extra travel length during brake deadtime and because of the effect of deceleration.

Stopping Allowance—A stopping allowance is to ensure that clearance remains between the top of the conveyance and the first obstruction in the headframe.

Calculations for Minimum Clearance—In an article describing hoisting plants at International Nickel Co., calculations for minimum headframe clearances were presented for both drum hoists and friction hoists (Albert, Cameron, and Gullick, 1975). These calculations are summarized in the following.

In computations for minimum clearances, two assumptions on hoist speed are made: (1) for rope stretch allowance, the hoist speed is 30 fpm (0.15 m/s); and (2) for the stopping allowance, it is assumed that the speed has been reduced to 300 fpm (1.5 m/s).

For drum hoists, the minimum clearance from the top of the hoist rope capping, in its lowest fully dumped position, to the rim of the headsheave is given by (in English units):

$$\begin{aligned}\text{clearance} &= \text{operating allowance} + \text{overtravel to Lilly limit}\\ &\quad + \text{rope stretch allowance} + \text{stopping allowance}\\ &= 5 + 1 + \frac{PL}{AE} + \frac{V_1^2}{2a} + V_1 t + 1 + \frac{V_2^2}{2a} + V_2 + 10 \end{aligned} \tag{17.5.43}$$

where P is maximum rated payload in lb, L is maximum hoisting distance in ft, A is equivalent area of rope in in^2., E is rope elastic modulus in psi, $V_1 = 0.5$ fps, $V_2 = 0.5$ fps, a is hoist deceleration rate in fps^2, and t is brake deadtime in sec.

For friction hoists, the minimum clearance from the top of the skip in its lowest position to the underside of the arrestor gear is given by (in English units):

$$\text{clearance} = 5 + 1 + \frac{PL}{AE} + \frac{V_1^2}{2a} + 0.5 + \frac{V_2^2}{2a} + V_2 + 7.5 \tag{17.5.44}$$

The required arrestor travel distance must be added to the above clearance in order to calculate the total clearance to the headframe crash beam. The arrestor travel length is given by (English units):

$$L = \frac{V^2}{2a} + H + \text{safety margin} \tag{17.5.45}$$

where L is arrestor travel length in ft, V is conveyance speed at time of entry in fps, a is maximum declaration rate of 0.9g (= 28.98 fps^2), H is height of arrestor gear (assume 4 ft), and the safety margin is 5 ft.

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Section 18 Underground Mining: Self-Supported Methods

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Chapter 18.0
INTRODUCTION

CHRISTOPHER HAYCOCKS

Self-supporting, open stoping underground mining methods are some of the earliest, starting with the prehistoric flint mines of Europe and the Egyptian gold mines of Nubia in the time of the pharaohs. When the early miner created an opening, it had to be inherently stable and self-supporting to enable him to continue. Therefore, as mining progressed beyond simple gophering, self-supported open stoping methods were devised to meet the needs of individual ore bodies. Timber for support appears in mines as early as 1000 BC, but this was for no more than local ground control. The ingenuity of mining engineers through history has so refined open stoping methods that today they produce more tonnage than any other mining system.

Self-supported open stoping finds its application in moderate to strong rocks that exhibit little jointing or fragmentation. Good back, hanging wall, and footwall conditions that require no more than incidental support are an essential part of these mining methods. Open stoping methods find applications in all inclinations of ore body from flat to vertical, and where ore movement may be either mechanical or under the influence of gravity. Table 18.0.1 demonstrates the range of ore bodies suitable for open stoping, with particular reference to their dip and the deposit thickness.

Most of the flat-dipping applications use ore pillars for overall support and ground control that may be recovered under some conditions after completion of normal stoping practices. In steeper ore bodies, rib, crown, and sill pillars used to block out the ore may be recovered subsequent to the completion of stoping. Open stopes are frequently filled to facilitate pillar recovery. Two of the methods, vertical crater retreat (VCR) and shrinkage stoping, utilize fragmented ore for temporary support in the stopes. In these two cases, the stope is not strictly open until after drawdown.

Table 18.0.1. Applications of Self-supporting Methods

Stoping Method	Ore Body Dip	Primary Support	Ore Body Thickness	Stoping Efficiency (TMS)*
Stope and Pillar, Room and Pillar	$<35°$	Ore Pillars	<300 ft (90 m)	30–70
Breast Stoping	$<35°$	Artificial Pillars	<12 ft (3.6 m)	1–5
Under/Overhand	$>45°$	None	<8 ft (2.4 m)	1–3
Shrinkage	$>45°$	(Broken ore)	>4 ft (1.2 m)	3–10
Sublevel	$>45°$	None	>20 ft (6 m)	50–70
Vertical Crater Retreat	$>45°$	None (Broken ore)	>40 ft (12 m)	>35

* TMS = tons/miner (employee)-shift

Self-supported stopes in ore bodies over 20 ft (6 m) thick are primarily high production and are amenable to mechanization, which makes them very efficient. The development of large-diameter (6- to 8-in., or 150- to 200-mm) down-hole drills has had a major impact on some open stoping because of their extreme accuracy over distances up to 400 ft (120 m). Innovative applications of these down-hole drills has significantly changed the layouts of the classical sublevel stope, making it much more productive while reducing costs. VCR stoping has also been a major beneficiary of this equipment. Most open stoping has also been a major beneficiary of trackless mining, particularly load-haul-dump units.

Chapter 18.1
ROOM AND PILLAR MINING

IAN FARMER

18.1.1 INTRODUCTION

Bullock (1982a), quoting previous data, showed that room and pillar mining together with stope and pillar mining accounted for most of the underground mining in the United States. He estimated that 60% of noncoal minerals (about 80 million tons or 70 Mt) and 90% of coal (about 290 million tons or 260 Mt) were obtained by room and pillar methods, and it is unlikely that things are radically different today. The method is cheap, highly productive, easily mechanized, and relatively simple to design. Ultimately, and particularly with increasing depths, mechanized longwall methods will make greater inroads into both coal and noncoal mining. But longwall requires major capital investment and development costs, and even now design is difficult, and success not always certain. In particular, longwall is inflexible. The rapid advance rates required to provide an adequate return on capital mean that all except very minor geologic faults must be avoided. Thus quite large areas of reserve are not minable using longwall methods, and they often give much lower overall recovery than retreat room and pillar mining, which is highly flexible.

The *room and pillar mining* method is a type of open stoping used in near horizontal deposits in reasonably competent rock, where the roof is supported primarily by pillars. Ore—or more commonly, coal—is extracted from rectangular shaped rooms or entries in the ore body or coal seam, leaving parts of the ore or coal between the entries as pillars to support the hanging wall or roof. The pillars are arranged in a regular pattern, or grid, to simplify planning and operation. They can be any shape but are usually square or rectangular. The dimensions of the rooms and pillars depend on many design factors, which will be considered later. These include the stability of the hanging wall and the strength of the ore in the pillars, the thickness of the deposit, and the depth of mining. The objective of design is to extract the maximum amount of ore that is compatible with safe working conditions. The ore left in the pillars is usually regarded as irrecoverable or recoverable only with backfill in noncoal mines. In this case backfill costs or the potential loss of valuable resource may be a limiting factor in room and pillar mining at greater depths. In coal mining, pillars are, ideally, recovered by retreat mining, allowing the roof to cave, thus relieving stress and reducing the likelihood of bumps.

The applications of pillar mining have been discussed by Hamrin (1982) and Hittman Associates (Anon., 1976) among others. Suitable conditions include ore bodies that are horizontal or have a dip of less than 30°. A major requirement is that the hanging wall is relatively competent over a short period of time, or is capable of support by rock bolts that are used extensively in room and pillar mining. The method is particularly suited to bedded deposits of moderate thickness (6 to 20 ft, or 2 to 6 m) such as coal—the main application—salt, potash, and limestone.

18.1.2 DESIGN OF PILLARS

18.1.2.1 Pillar Stress

Much of the following discussion is based on geomechanics theory presented in Chapter 10.5 and other chapters of Section 10.

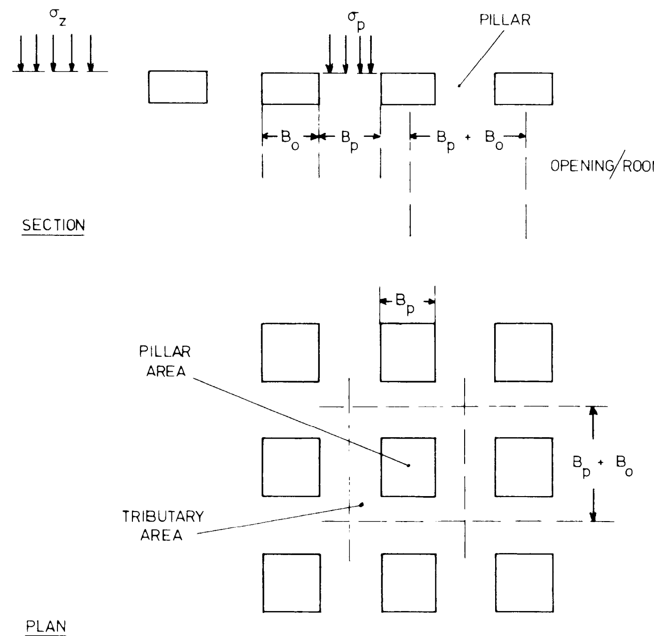

Fig. 18.1.1. Section and plans of rooms and pillars with widths and dimensions for simple analysis.

Despite the simplicity of the structure, and the detailed knowledge of rock behavior obtained over the past few years, pillar design has changed very little during the present century. It is based on the assumption that the stress in a pillar is evenly distributed and equal to the original vertical geostatic stress divided by the pillar area/original area ratio; and that pillar failure occurs when this stress exceeds the compressive strength of the pillar rock. It would be a naive assumption for any engineering structure in any material. It is particularly so in the case of pillars with high width/height ratios in a jointed, brittle material such as rock.

The major recent work on stresses acting on pillars has been carried out by Coates (1981). He started with the simplest and traditional statement of average pillar stress, known as the tributary area method. This assumes that each of the pillars left during excavation supports all the overlying strata that are "tributary" to their location. Then the average pillar stress σ_{pa} for square pillars with rooms of consistent width is

$$\sigma_{pa} = \sigma_z \frac{(B_p + B_o)}{B_p} \qquad (18.1.1)$$

where B_p and B_o are width of the pillar and room, respectively (Fig. 18.1.1), and σ_z is the geostatic or premining stress acting normal to the plane of excavation. If this is horizontal, then

$$\sigma_z = \gamma z \qquad (18.1.2)$$

where γ is rock average unit weight and z is depth to the mining horizon. This can be stated more simply for the common case of rectangular or irregular shaped pillars in terms of the extraction ratio R, where $R = \dfrac{B_o}{B_o + B_p}$ is the ratio of the area extracted to the total area of the orebody mined. Since $1 - R = \dfrac{B_p}{B_o + B_p}$, Eq. 18.1.1 can be more generally stated,

$$\sigma_{pa} = \sigma_z \left(\frac{1}{1 - R}\right) \tag{18.1.3}$$

This approach assumes that the mined area is extensive and shallow, that the mined rock is horizontally stratified, and that the pillars are equidimensional. It specifically ignores the relative extent and depth of the mined area, the stress component parallel to the plane of mining, the relative deformation properties of pillar, roof, and floor rocks, and the positions of the pillars in the mining zone. Taking some of these into account, Coates (1981) obtained a more general solution, principally for deep, long, mine pillars but applicable generally, by solving the statically indeterminate net deflection of the roof and floor rocks resulting from mining. Then the solution for average pillar stress becomes

$$\sigma_{pa} = \sigma_z \left\{ \frac{\left[2R - K_o \dfrac{H}{L}\dfrac{(1 - 2\nu_w)}{(1 - \nu_w)} - \dfrac{\nu_p}{(1 - \nu_p)} K_o \dfrac{H}{L}\dfrac{E_w}{E_p}\right]}{\left[\dfrac{H}{L}\dfrac{E_w}{E_p} + 2(1 - R)\left(1 + \dfrac{1}{N}\right) + 2\dfrac{RB}{L}\dfrac{(1 - 2\nu_w)}{(1 - \nu_w)}\right]} \right\} \tag{18.1.4}$$

where H is seam height; L is the extent of the mined area; K_o is the ratio between σ_h and σ_z or the coefficient of geostatic stress; and E_w, E_p, ν_w, and ν_p are the elastic constants of the wall (roof and floor) and pillar materials.

This is a two-dimensional elastic solution in plane strain and requires, strictly speaking, a length/width ratio of about 3 or more to be applicable. An analytical three-dimensional approach is not feasible, although finite element and boundary element methods (see for instance Tang and Peng, 1988) can be used to give a numerical solution.

Coates' (1981) approach is helpful in that it can be used to illustrate simply several of the fundamental characteristics of strata and geometry that affect pillar stresses. Some of these are illustrated in Fig. 18.1.2. For instance, as the E_w/E_p ratio rises (Fig. 18.1.2a), so the pillar stress is reduced from a magnitude close to $4\sigma_z$ (the extraction ratio has been chosen as 80%) to a level of $0.5\sigma_z$ for $H/L = B/L = 0.1$. This illustrates the bridging effect of the stiffer roof and floor layers and the tendency to transfer stress to the side abutment. Similarly, as L is decreased (Fig. 18.1.2b), the pillar stress is reduced from a maximum magnitude of $4\sigma_z$ to zero and $H/L = 0.4$ for a E_w/E_p, ratio of 6. Again this can be attributed to bridging at low spans. As a further illustration (Fig. 18.1.2c), using fixed values for E_w/E_p, H/L, B/L, there is considerable variation between the tributary area calculation (Eqs. 18.1.1 and 18.1.4) for stress at increasing extraction ratios.

It should be emphasized that this is used as an illustration, and that measurements of *average* pillar stresses are very infrequent. In fact, a review of the literature shows virtually no reliable measurements of average stress, principally because such measurements are difficult to obtain. One of the more interesting sets of data is by Orawecz (1977) from work in South African

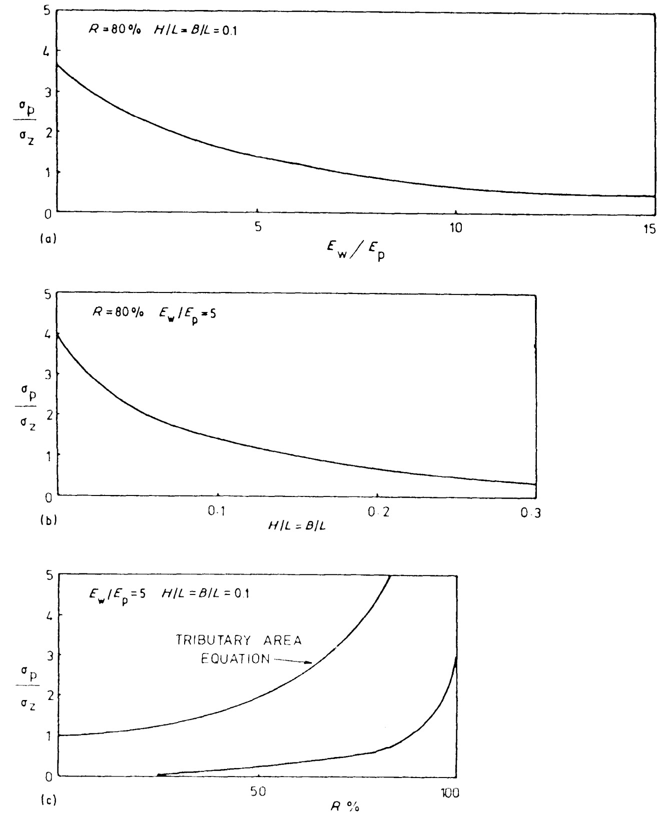

Fig. 18.1.2. Estimates of pillar stress σ_p as a proportion of vertical stress σ_z based on the variables in Eq. 18.1.4, putting $K_0 = 1$, $\nu_p = \nu_w = 0.33$, and N large, so that

$$\frac{\sigma_p}{\sigma_z} = \frac{(2R - 0.5H/L) - (0.5H/L \times E_\omega/E_p)}{H/L \times E_w/E_p + 2(1 - R) + RB/L}$$

coal mines. He describes two case histories in which surface settlements and underground displacements were measured using leveling and anchors in boreholes drilled from the surface to the seam level and below. The seams were at average depths of 131 ft (40 m) and 223 ft (68 m). The purpose of the measurements was to test an analog model, and satisfactory simulation allowed computation of pillar stresses from observed seam deformations.

The pillar geometries and data on the mining and instrumentation layouts are illustrated in Figs. 18.1.3 and 18.1.4 together with the pillar stresses σ_{pa}, computed from seam deformations in Figs. 18.1.3c and 18.1.4c. These are quite close to the pillar stresses σ_{pa} computed from the tributary area equation (Eq. 18.1.1). In these cases, the E_w/E_p and H/L ratios were, respectively, 3 and 0.01 and 2 and 0.05, and it can be seen from Fig. 18.1.2 that such a result would be expected. It is interesting to note the reduced pressure on the pillars adjacent to the ribside, and also the relatively low level of the abutment stress. The former would be expected; the latter is rather surprising and implies some weakening of the abutment.

The concept of average pillar stress is not a good one, since pillar stresses are not evenly distributed. This can be illustrated simply by stress analysis. A simple two-dimensional boundary element program, developed by Bray and Hocking among others, is included in Hoek and Brown (1981). This can be used, after modification, to calculate stresses around an opening or openings in a homogeneous, isotropic, linearly elastic material, under con-

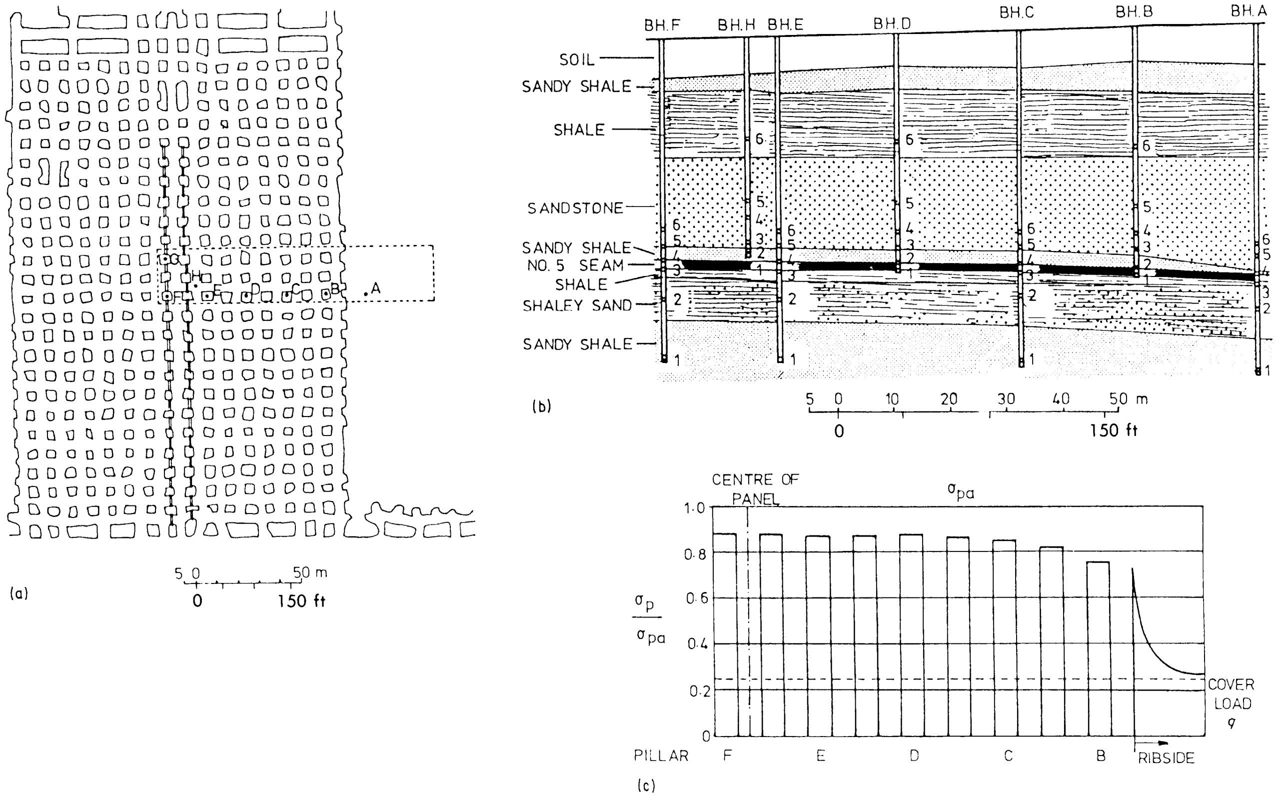

Fig. 18.1.3. Estimation of pillar stress σ_p as a proportion of pillar stress σ_{pa} computed from tributary area theory from experiments by Oravecz (1977) in No. 5 seam at Colliery A., South Africa. Data: average depth to mid-seam 40.3m; seam height 1.5m; pillar width 5.2m; room width 5.5m; percentage extraction 76.4%; panel width 176.2m (est.); deformation modulus, seam (est.) 1.54 GNm^{-2}; deformation modulus strata (est.) 4.43 GNm^{-2}; Poisson's ratio (est.) 0.15. Conversion factors: 1 ft = 0.3048 m, 10^6 psi = 6.894 GNm^{-2}.

ditions of plane strain in an infinite medium subjected to various combinations of uniform field stresses or external loadings. Typical solutions are given in Hoek and Brown, and the solutions for square and rectangular openings in a uniform stress field are reproduced in Fig. 18.1.5. Although the boundary conditions may be a little extreme for room and pillar mining, a simple example of how these computed stress distributions can be used in pillar design is given in Fig. 18.1.6. This takes the stress distribution in Fig. 18.1.5b and assumes initially two square rooms of dimension a at a distance 4a apart. Then the minor principal stress or confining stress in the pillar between the two can be projected on to a graph of minor principal stress against pillar width, to give the minor principal stress distribution and the average minor principal stress. This can be computed for pillars of any width (see Fig. 18.1.6c), and the resultant distribution can be used to compute the ultimate pillar strength using the strength envelope of the rock or coal in the form,

$$\sigma_{1f} = \sigma_{cf} + K_p\sigma_3 \tag{18.1.5}$$

Then σ_{1f} can be compared with the pillar stress σ_{pa} computed from the tributary area Eq. 18.1.1 to obtain an estimate of safety factor.

18.1.2.2 Pillar Strength

There is a large literature on pillar strength, much of it empirical. The most complete work is by Salamon and Monro (1967), and the best summaries by Bieniawski (1981) and Tsur-Lavie and Denekamp (1982). For detailed coverage of pillar strength theory, see Chapter 10.5.

The basic problem with pillar strength is that in a brittle rock, strength is dependent upon the size, and to a lesser extent, the shape of a test specimen. This means that the conventional method of pillar design, relating rock strength to pillar stress through a factor of safety $(FOS = \sigma_{cf}/\sigma_p)$, is unacceptable in brittle rocks, although it may be acceptable in more ductile rocks. The reason for this is evident: if failure occurs in a brittle manner, the strain energy stored in a pillar will be released from a volume onto a shear or tensile failure plane, where it will be distributed as surface energy per unit area of fracture surface; a constant for a particular rock. This is the basis of the Griffith failure criterion and is explained in Farmer (1985). Since energy is proportional to the square of stress, this means that strength will be inversely proportional to the square root of the dimension of the rock specimen, an observation confirmed experimentally by Bieniawski (1981) and Singh (1981) for various rocks including coal. In terms of pillar σ_{pf} and rock σ_{cf} strength, this can be expressed

$$\frac{\sigma_{pf}}{\sigma_{cf}} = \left(\frac{L_s}{L_p}\right)^{1/2} = \left(\frac{V_s}{V_p}\right)^{1/6} = \left(\frac{V_s}{V_p}\right)^{0.17} \tag{18.1.6}$$

where L and V represent dimension and volume, respectively, and the subscripts s and p refer to the laboratory specimen for strength testing and the pillar, respectively. In the ductile case, the energy is not transferred onto fracture surfaces but evenly distributed in the specimen or pillar. Then the exponent ap-

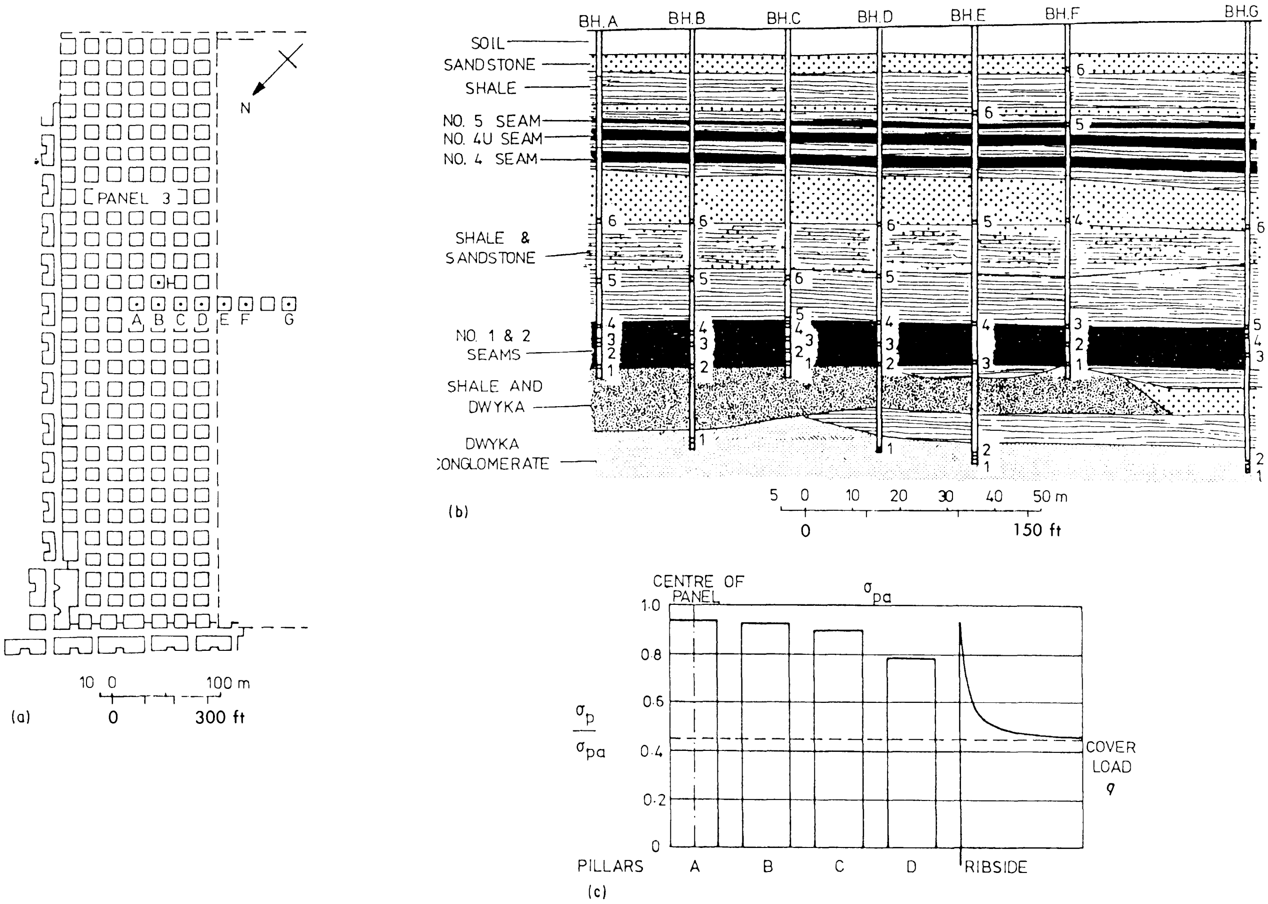

Fig. 18.1.4. Estimation of pillar stress σ_p as a proportion of pillar stress σ_{pa} computed from tributary area theory, from experiments by Oravecz (1977) in No. 2 seam at Colliery B., S. Africa. Data: average depth to mid-seam 66.7m; seam height 5.5m; pillar width 13.7m; room width 6.1 m; percentage extraction 52.1%; panel width 144.8m; deformation modulus, seam (est.) 3.92 GNm^{-2}; deformation modulus, strata (est.) 6.27 GNm^{-2}; Poisson's ratio (est.) 0.15. Conversion factors: 1 ft = 0.3048 m, 10^6 psi = 6.894 GNm^{-2}.

proaches unity. Thus, in the case of wide pillars, and pillars in pseudo-ductile rocks such as rock salt, Eq. 18.1.6 can be modified.

The relevance of Eq. 18.1.6 can, however, be confirmed by the empirical work of Hardy and Agapito (1977) on oil shale pillars in western Colorado. They proposed a general pillar formula which is recommended for all brittle rocks—that is, where the pillars fail in tension or shear—in the form,

$$\frac{\sigma_{pf}}{\sigma_{cf}} = \left(\frac{V_s}{V_p}\right)^{0.118} \left[\left(\frac{B_p}{H_p}\right) \Big/ \left(\frac{B_s}{H_s}\right) \right]^{0.833} \tag{18.1.7}$$

where B and H are pillar and specimen width and height, respectively. There are, of course, limitations for this approach, one of which would probably be the pillar width/height ratio. If this is less than 1, and particularly if the rock is ductile, the volume exponent will increase.

For the record, although the above method is strongly recommended, it is useful also to include the conventional representations of pillar design equations, often called the Holland-Gaddy (Holland, 1964) equation in the United States, which take the form,

$$\sigma_{pf} = \sigma_{cf}\left(a + b\,\frac{B}{H}\right) \tag{18.1.8}$$

$$\sigma_{pf} = K\,\frac{B^{\alpha}}{H^{\beta}} \tag{18.1.9}$$

In this case, σ_{cf} is uniaxial compressive strength of a cube of specified dimension; a and b are dimensionless constants, usually chosen so that $a + b = 1$ and $\sigma_{pf} = \sigma_{cf}$ when $B/H = 1$; α and b are dimensionless constants; and $K = f(\sigma_{cf})$ is a constant so that $K = \sigma_{cf}$ when $\alpha = \beta$ and $B = H$. There is a reasonable agreement about constants a, b, α, and β in Eqs. 18.1.8 and 18.1.9. Some representative values from early times to more recent are quoted in Table 18.1.1, principally for coal mines. All of the constants are effectively shape factors. The basic problem is that σ_{cf} in either equation is essentially the laboratory value, and a factor of safety, usually not included in the equation, is needed to allow for size effects and ensure safe design. Quoted values of this "safety factor" are difficult to find. Wilson (1983) suggests 5 for coal, but incorrectly recommends 1 for strong massive unjointed rock and 6 to 7 for weak rock—quite the reverse of the probable actual values. Where the economic success or failure of an operation depends on correct estimation of extraction ratio, a more accurate approach is required and Eq. 18.1.7 is recommended as a starting point. This represents a safety factor of 4 to 5 for most rocks and pillar shapes.

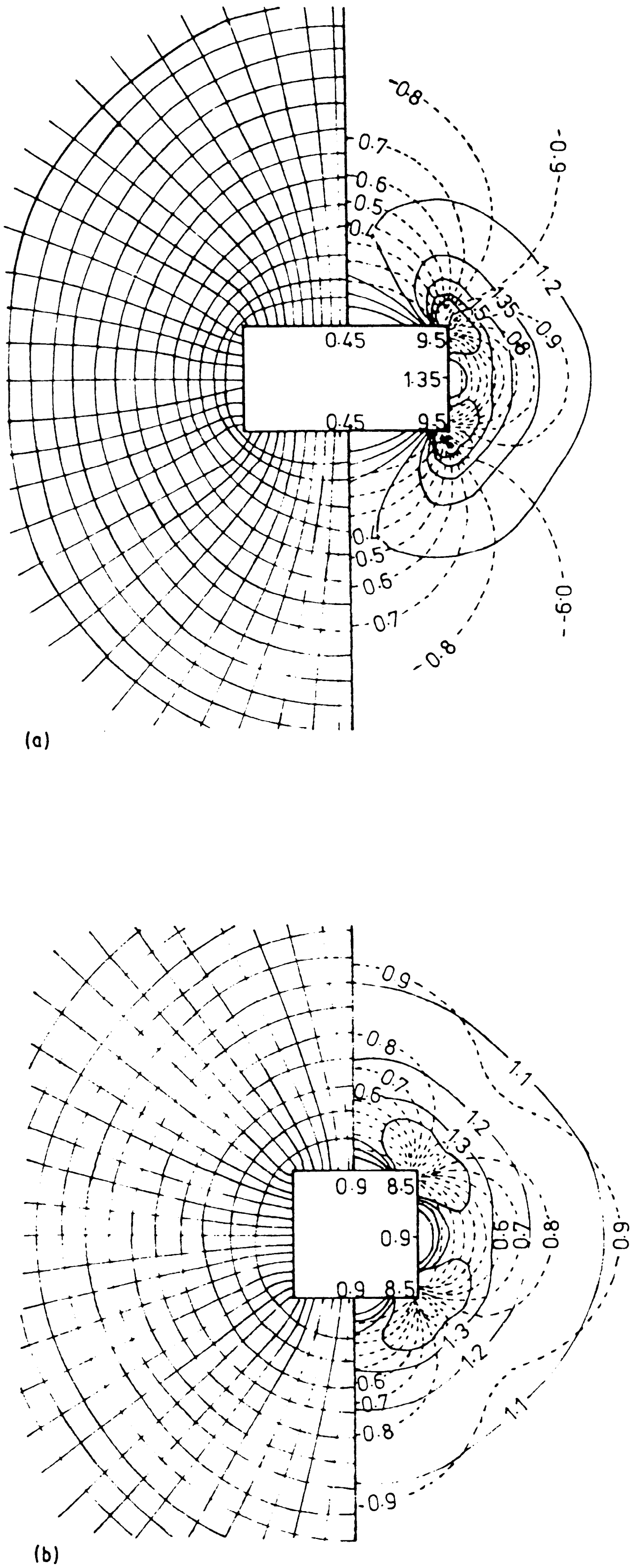

Fig. 18.1.5. Principal stress trajectories (LHS) and contours (RHS) of the ratio of major principal stress to applied stress (solid line) and minor principal stress to applied stress (dotted line) for (a) a rectangular and (b) a square opening in an infinite medium subject to a uniform stress field (using Bray and Hocking's two-dimensional boundary element analysis, in Hoek and Brown, 1981).

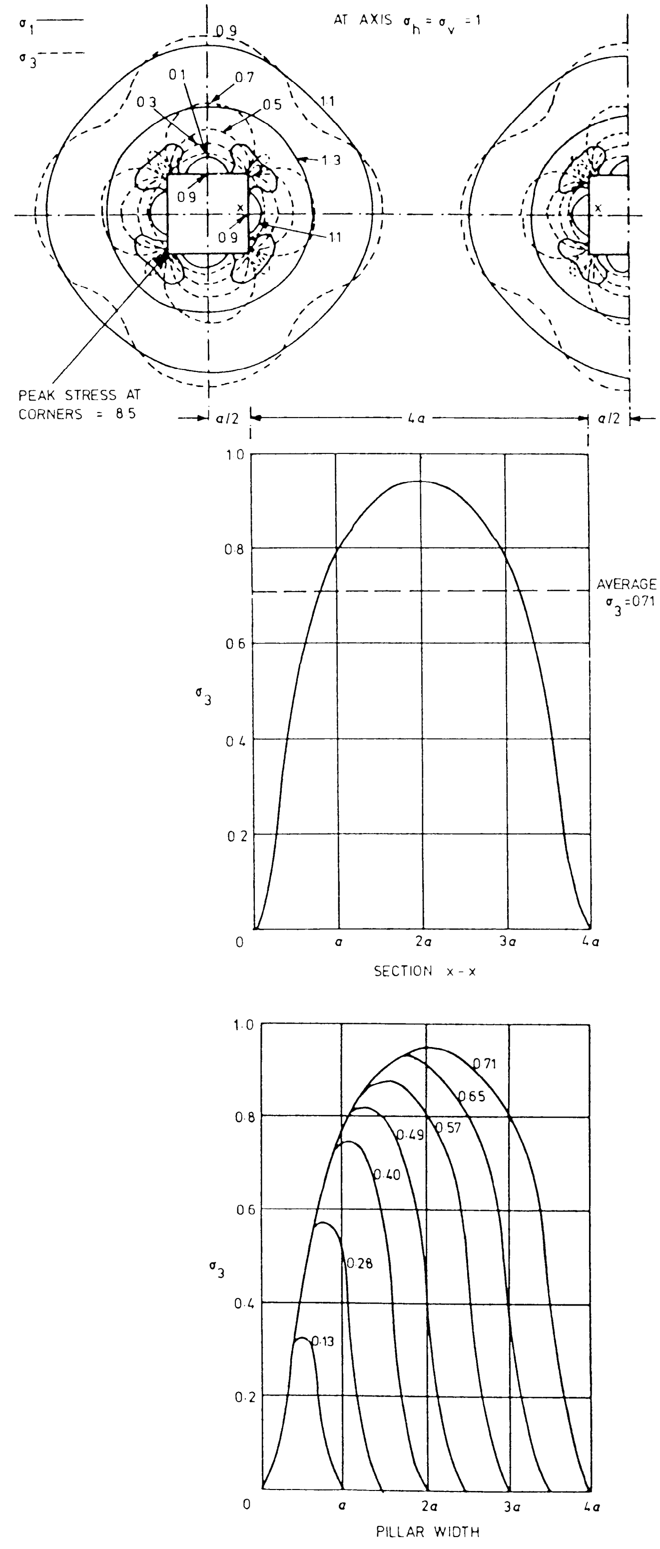

Fig. 18.1.6. (a) Contours of major (solid line) and minor (dotted line) principal stress around two rooms of dimension a separated by a pillar 4a in width, and (b) plotted to give minor principal stress (expressed as a proportion of applied stress) distribution in the pillar, and (c) relation between minor principal stress expressed as a proportion of uniform applied stress and pillar width—for pillars of varying width. Values for average σ_3 are given for each curve.

Table 18.1.1. Constants in Equations 18.1.7–18.1.9

Source	a	b	α	β	K MNm^{-2}	Comments
Bunting (1911)	0.7	0.3	—	—	—	Laboratory data
Obert, Windes, and Duvall (1946)	0.78	0.22	—	—	—	Laboratory data
Bieniawski (1968)	0.64	0.36	—	—	—	In situ—S. Africa
Van Heerden (1974)	0.70	0.30	—	—	—	In situ—S. Africa
Wang, Skelly and Wolgamott (1977)	0.78	0.22	—	—	—	W. Virginia mines, United States
Sorensen and Pariseau (1978)	0.69	0.31	—	—	—	Statistical—United States
Greenwald, Howarth, and Hartmann (1939)	—	—	0.5	0.83	—	In situ—Pittsburgh mines, United States
Streat (1954)	—	—	0.5	1	—	Statistical—S. Africa
Holland (1964)	—	—	0.5	1	—	Statistical—United States
Salamon and Monro (1967)	—	—	0.46	0.66	—	Statistical—S. Africa
Bieniawski (1968)	—	—	0.16	0.55	—	Statistical—S. Africa
Hazen and Artler (1972)	—	—	0.5	0.5	—	Statistical—United States
Zern (1926)	—	—	0.5	0.5	—	Empirical—United States
Morrison, Corlett and Rice (1975)	—	—	0.5	0.5	—	In situ—Canada
Greenwald, Howarth, and Hartmann (1939)	—	—	—	—	9.3	Originally in psi for B, H values in inches-Pittsburgh, United States coals
Salamon and Monro (1967)	—	—	—	—	9.1	Originally in psi for B, H values in feet-S. Africa
Bieniawski (1968)	—	—	—	—	6.9	Originally in psi for B, H values in feet-S. Africa
Jenkins and Szeki (1964)	—	—	—	—	12.4	Originally in psi for B, H values in feet-Britain
Wagner (1974)	—	—	—	—	11.0	B, H values in meters. Based on in situ tests-S. Africa

Conversion factor: 10^3 psi = 6.894 MN m^{-2}.

18.1.2.3 Barrier Pillar Design

Room and pillar mines are usually developed in a series of rectangular panels separated by barrier pillars. There is no specific design method for these pillars, but where the roof is not caved or where pillars are left in place, design of barrier pillars assumes greater importance. Fig. 18.1.2 shows that pillar stress is not necessarily evenly distributed, and where the roof and floor rocks are stiffer than the pillar rocks, stress with be transferred to an abutment. There is also the probability that deterioration—or overmining—of highly stressed pillars may lead to a reduction in load capacity of individual (or groups of) pillars, and transfer of load to other pillars that may lead to progressive failure. This is one of the most common causes of extensive pillar collapse (Mottahed and Szeki, 1982, describe a total mine collapse), and barrier pillars can control this.

Wilson (1983) analyzed this problem and suggested, for coal mines, barrier pillar widths of 1/10th of the working depth, but his approach, although applied to room and pillar workings, was designed principally to reduce entry damage in longwall entry chain pillars. A more satisfactory approach may be to consider pillar yield. Hudson, Brown, and Fairhurst (1971) in a series of tests on marble, which can be repeated on coal, showed that a pillar behaved in a yielding rather than a brittle manner if its height/width ratio was less than 1/3. The implication is that below this ratio, a pillar will deform rather than fracture, resisting rapid collapse. A yielding, barrier pillar of 3 to 4 times the excavation height can, therefore, be recommended, particularly at greater mining depths.

18.1.3 SUPPORT OF ROOMS

18.1.3.1 Rock Bolts

The key to design of rooms is support. This invariably means the use of rock bolts in room and pillar mining. At present, over 100 million bolts per year are installed in US mines. There are various types of rock bolt, and the type and method of installation can have a significant effect on performance. Classification of rock bolts into types is difficult. Conventionally, there are two methods, either as (1) grouted (usually fully grouted) or (2) mechanically anchored (usually point-anchored) bolts. A list of available bolt types from Peng and Tang (1984) is given in Table 18.1.2. A point-anchored bolt is usually tensioned; a fully grouted bolt is usually untensioned. A mechanical anchor can be installed easily, but is unreliable over a period of time; a resin bolt requires precision in installation-whether point or fully grouted-and usually has better long-term characteristics. The theory of rock bolting is developed fully in Chapter 10.5.

Conventional rock bolts are made from ⅝-in. (16-mm), ¾-in. (19-mm), l-in. (25mm), or l¼-in. (32-mm) steel rebar with an approximate yield force, respectively, of 7 (6), 9 (7.5), 17 (15), and 26 tons (23 tonnes). Normally, the installed bolt tension is 50% of this load. Steel bearing plates at the hole collar are usually 6 in. (150 mm) square and ¼ in. (6 mm) to ⅜ in. (9.5 mm) thick and are flat or bell-shaped with a center hole. The main function is to distribute stress to the rock at the collar through a nut threaded on to the top of the bolt, and tensioned through a drill chuck. Angle or spherical washers are used to create a uniform bearing surface. To prevent falls of rock between bolts-an important factor in weaker rocks-mesh or bench bars are placed behind the anchor bearing plates. For long-term installations, shotcreting is essential.

Bolts are usually considered temporary supports. At bolt forces close to working load, they are, like all rock stress systems, prone to deterioration with time. At differential roof deformations, greater than 1 to 1½%, they usually cease to function, although performance can be improved with shotcreting. The reduction or change in capacity with time is not well documented and relies to a great extent on ground conditions. A particularly useful recent paper by Signer and Jones (1990) illustrates the changing reinforcement loads on fully grouted bolts during roof deformation and illustrates their very flexible response to deformation.

In the case of mechanical bolts, installation is invariably accompanied by reduction in tension with time. This was investi-

Table 18.1.2. Types of Roof Bolt

Types of bolt	Types of anchor	Suitable strata type	Comments
Point-anchored bolt (tensioned)	Slot-and-wedge	Hard rock	Used in early stages
	Expansion shell	Medium-strength to soft rock	Most commonly used in US
	Resin grout	All strata especially for weak rock	Increased usage recently. Resin length less than 24 in. (0.6 m)
	Combination shell and resin anchor	Most strata	Resin lengths greater than 24 in. (0.6 m)
Full-length-anchored bolt (nontensioned)	Cement grout	Most strata	Disadvantage: (1) shrinkage of cement (2) longer setting time
	Resin grout	All strata	Increased use recently, especially for weak strata
Special bolts: Yieldable bolt	Expansion	Medium-strength rock	An expansion-shell bolt with a yield device
Pumpable bolt	Resin	Weak strata	Complex in installation
Helical bolt	Expansion shell	Most strata	In experimental stage
Split set	Full-length friction	Weak strata	Cheap but needs special installation equipment
Roof truss	Expansion shell	Adverse roof	Recommended for use at intersections and/or heavy pressure areas
Cable sling	Cement anchor and full-length friction	Weak strata	Substitute for timber, steel, or truss support
Lateral force system	Full-length 2-piece steel	Soft strata	Applying full-length lateral force (compression) to the strata
Swellex bolt	Full-length holding	Water-bearing strata	Using high-pressure water to swell the steel tube thus holding the rock

Source: Peng and Tang, 1984.

gated by de la Cruz (1964) and Parson and Osen (1969) among others and was attributed principally to slippage of serrations on the anchor shell, rock deformation and rock breakage at the anchorage and collar, and ground movement following excavation. In addition, dynamic vibration due to blasting is a major cause of tension loss. This means that constant monitoring and retensioning of bolts is needed if long-term installation is required. Conversely excessive bed separations can lead to bolt head failure, which is not found in grouted bolts.

It has been claimed that fully or point-grouted resin or cement anchors give improved performance, both long and short term, and there is some evidence for this. Franklin and Woodfield (1971), in a series of experiments, showed that reliance on bond rather than friction means that the force take-up is much quicker, and by extrapolation, the possibility of slippage is much less. There remain dangers associated with faulty installation, excessive annulus thickness, and poor bonding in wet holes, which in practice can make resin grouting less attractive.

The action of bolts is best described through the typical theoretical stress distributions around the openings illustrated in Fig. 18.1.5. In both cases, the surface of the opening is subjected to compressive tangential stress and zero radial stress. Further away from the surface, both the radial and tangential stresses approach the primitive stress levels in the rock mass undisturbed by excavation. The tangential compression stresses are high at the corners and in the sidewalls, but low in the roof and floor. This condition is exacerbated as the height/width ratio of the rectangular room is reduced further. This will have two effects: there will be crush at the corners and possibly squeeze in the sidewalls, and the reduced radial compression will allow sag of the roof and uplift of the floor. The most important of these is the reduced roof compression—particularly if combined with bedded and jointed strata—which will create conditions for bed separation or release of blocks from the roof strata.

Rock bolts are the cheapest and most obvious way of maintaining stability in such circumstances. Provided that the rocks are suitable for an anchorage location, are not subject to swelling or slaking, and there are no high pore pressures or water flows, then bolts have two main functions acting either singly or as a pattern. These are to maintain the stability of sagging roofs, particularly in weaker stratified rocks, and to restrain blocks in well-jointed or blocky rocks where release surfaces daylight in the exposed roof. The former application is principally for roof support in room and pillar mining in stratified rocks. This is the most common use of rock bolts, and it can be improved by variations such as trusses or slings (see, for instance, Seegmiller, 1990). The latter application is principally in civil engineering works, such as tunnel and cavern construction, and occasionally in slopes, where quite-large-capacity anchors are often used.

18.1.3.2 Support Design

Where a bolt is used to restrain a single block in the roof of an entry, the volume and hence the weight of the block and where necessary its direction of sliding can be determined by stereographic analysis of the kinetics of sliding. This method is outlined in Farmer and Shelton (1980) and in Farmer (1985). Methods of support based on the common requirement that bolt spacing should be half the bolt length are discussed in the same sources.

In coal mining, the design of bolts is usually based on Panek's (1962a, b) analysis. The most simple assumption for design purposes is to consider a sagging roof plate or beam of thickness L, span B, and length X, supported by rows of bolts with separation a between rows and spacing S. Then the bolt tension force P to support the roof will be given by:

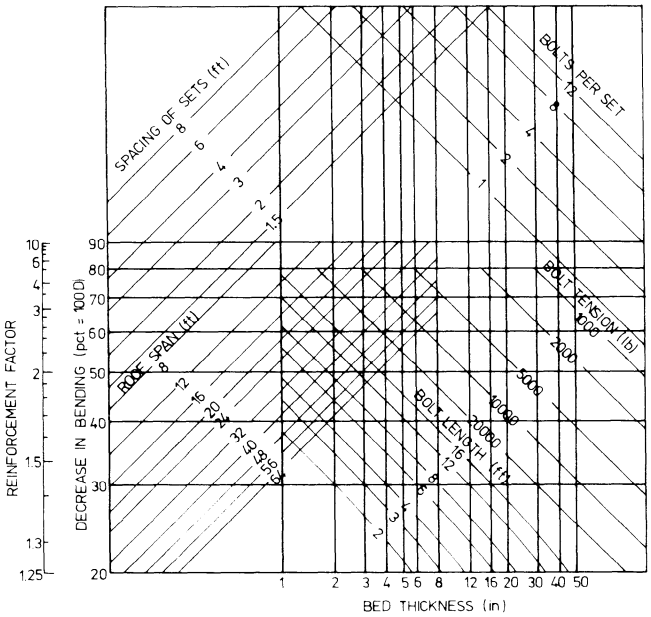

Fig. 18.1.7. Nomograph to determine the friction effect for bolting in mine roofs. $\mu = 0.7$, $\sigma = 24.6 kN/m^3$. Conversion factors: 1 in = 25.4 mm, 1 ft = 0.3048 m, 1000 lbf = 4.448 kN. (After Panek, 1962b.)

$$P = \frac{\gamma BXL}{\left(\dfrac{x}{\alpha} + 1\right)\left(\dfrac{B}{S} + 1\right)} \tag{18.1.10}$$

where γ is unit weight of the roof rock.

This equation, suggested by Obert and Duvall (1967), is valid if the roof above the excavation is completely suspended by bolts. For an assumed bolt load, it can also be used to estimate spacing and the number of rows. It represents the upper limit of bolt force since it ignores the important supporting effect of the abutments. It also ignores the interaction of a series of roof beds.

A more accurate approximation can be obtained by considering the effects of friction between beds and also by considering the roof span as a series of thin beams, fixed at each side of the opening. Panek (1962a,b; 1964) in a series of seminal papers considered this condition both experimentally by centrifugal testing and analytically, and developed the nomograph illustrated in Fig. 18.1.7, which has been used extensively in mine design. It is explained in detail by Panek and McCormick (1973) in the *SME Mining Engineering Handbook.* The basic variable is a reinforcement factor RF that is used to evaluate the interbed friction effect due to bolting. The roof is considered as a series of beds of equal thickness, of the same material, and without bonding between them. The bolts are assumed normal to the beds and tensioned to give normal compressive loading across the beds. Then

$$RF = \left(1 + \frac{\Delta\sigma_f}{\sigma_{fs}}\right)^{-1} \tag{18.1.11}$$

where $\dfrac{\Delta\sigma_f}{\sigma_{fs}}$ is the *decrease* in bending stress from frictional resistance induced by bolting, expressed as a ratio of the maximum bending stress in the unbolted strata, and is given by the empirical equation:

$$\frac{\Delta\sigma_f}{\sigma_{fs}} = \frac{3}{8}\,\mu(aB)^{-0.5}\left[\frac{B}{S}\,P\left(\frac{L}{t} - 1\right)\cdot\frac{1}{\gamma}\right]^{0.33} \tag{18.1.12}$$

where μ is the interbed coefficient of friction, a is spacing between rows, B is span, S is bolt spacing, t is average roof layer thickness, P is assumed bolt tension, and L is assumed equal to bolt length or supported thickness. For typical thin-bedded mine roof strata, *RF* should be greater than 2, and bolt spacing must by law be less than 5 ft (1.5 m). Spacings of 4 ft (1.2 m) are more common. Based on Eqs. 18.1.11 and 18.1.12, Panek's well-known nomogram (Fig. 18.1.7) allows rapid estimation of *RF* for a bolted roof, and forms a basis for rapid rock bolt pattern design.

18.1.3.3 Roof Caving

Although roof caving is not strictly speaking related to support, the mechanics are similar and it can be considered here. Caving is an important part of strata control in all mining operations. Correctly carried out, caving relieves stresses on abutments, barrier pillars, and chain pillars and improves overall mine stability. The need to cave the roof successfully determines the width of a room and pillar panel, as it does the width of a longwall face.

Cavability is a difficult concept. It is usually expressed in terms of a pressure arch, a circular, parabolic, or rectangular zone in the rock above an opening in two dimensions (see Fig. 18.1.5a) that has low radial compression stress, and where the rock sags and ultimately collapses under self weight at a critical unsupported span. This process is assisted by the presence of joints and weaknesses, which is why elastic analysis leaves a certain amount to be desired. The basics of computation of fracture onset in a roof span, analogous to the beam, plate, or "cracked arch," have been considered, with little success, by Obert and Duvall (1967) and Wright (1973). A better approach may be Terzaghi's (1946) arching theory, based on shear resistance in a frictional material above a bin hopper (the unsupported roof), and similar empirical methods that are summarized in Farmer (1985). An outline of this is given in Table 18.1.3. If a bulking factor of 1.1 is assumed for most layered rocks (Gorrie and Scott, 1970), then for caved strata to bulk sufficiently to support upper layers, the *span B* must be such that 1.1 $xB = xB + M$, where M is the excavated (or in the case of coal, seam) thickness, or

$$B = \frac{M}{0.1\,x} \tag{18.1.13}$$

where x = 0 *(good) to 2 (poor)* depending on the rock quality in Table 18.1.3. Obviously, a hard and intact rock is not cavable. For a massive, moderately jointed rock, a span in excess of 20 times the excavated thickness (i.e., 200 ft or 60 m, for a 10 ft or 3 m, thick excavation) would be required.

18.1.4 METHODS OF ROOM AND PILLAR MINING

18.1.4.1 Hard-rock Mining

Room and pillar mining takes place in sections or panels, which are usually rectangular and regular in plan. It is important

Table 18.1.3. Relation Between Cavability and Rock Classification Systems

Terzaghi classification	Rock behavior and possible causes of instability	Approximate stand-up time	Deere classification	Rock breakage height, m
1 Hard and intact	Stable excavation unless induced stress greater than rock strength	Many years	Excellent: RQD 90–100	0
2 Hard stratified and schistose	Bed separation with time; surface spalling	1 year		0.25 *B*
3 Massive, moderately jointed	Immediately stable. Detachment of blocks, progressively releasing further blocks	1 week	Good: RQD 75–90	0.5 *B*
4 Moderately blocky and seamy	Immediately stable. Detachment of blocks, progressively releasing further blocks	1 week	Fair: RQD 50–75	0.7 *B*
5 Very blocky and seamy and shattered	Immediately fairly stable. Surface dilation of rock due to rapid block detachment	1 day	Poor: RQD 25–50	1.5 *B*
6 Completely crushed	Local roof falls during excavation. Rapid peripheral dilation	1 hour	Very poor: RQD 0–25	2 *B*
7 Sand and gravel	Immediate collapse	0		
8 Squeezing: moderate depth	Rapid yielding and deformation		Squeezing and swelling ground	

Note: *B* is excavation span. RQD is rock quality designation.
Conversion factor: 1 ft = 0.3048 m.
Source: After Terzaghi, 1946; Deere, 1963.

here to differentiate between hard-rock and coal mining. In hard-rock mining of horizontal ore bodies, the method is very similar to open stoping (see Chap. 18.2). In many cases, ore grade control may be the primary requirement in mine design, and ground control and ventilation secondary considerations. This may lead to an ad hoc room and pillar design with irregular-shaped, nonrecoverable pillars of low-grade ore. In coal mining, ventilation and ground control are major factors, and this requires carefully designed room and pillar panels isolated from the rest of the mine and with a controlled ventilation system. It may also require plans for retreat pillar mining and caving.

Hard-rock room and pillar mining is effectively a method of open stoping (stope and pillar mining) at a low angle to the horizontal, excavating rooms and leaving supporting pillars. Where mineral values vary, the method is similar to the old "gophering" method of mining where random excavations followed highly mineralized zones. Where mineral values are consistent, the mine layout can be regular. The method differs from most hard-rock mining methods in that gravity flow is limited, and ore must be loaded in the excavation where it has been blasted and transported from that point. In large operations, this involves trucks and loaders or load-haul-dumps (LHDs), although slushers may also be used.

There are various methods of room and pillar stoping. The most common are full-face *slicing* or *breast stoping* and *multiple slicing or bench and breast stoping,* illustrated in Figs. 18.2.5 and 18.2.7 (see Chapter 18.2). In the former, the rooms are opened to their full vertical height with no mineral or economic value left in the roof or the floor. Probably the reasonable safe limit for full-face slicing is 25 to 35 ft (8 to 10 m) depending on drilling and support equipment, and beyond this, multiple slicing is used. In the United States, most coal, trona, and potash deposits are mined in one slice. Limestone, lead, and zinc mines use multiple slicing. In multiple slicing, the face is divided into a breast or brow, which is the top slice, and a bench (or benches), which is the bottom slice (or slices). It is quite common for mining to be organized so that there is simultaneous mining on the breast and one or two benches (Fig. 18.2.5). Multiple-slicing is usually done from the top down as illustrated (underhand stoping), but mining from the bottom up (overhand sloping) is equally feasible provided a layer of broken ore is left as a working platform. Overhand stoping is, however, more dangerous since new roof is continually exposed, whereas underhand stoping can be carried out under an undisturbed, supported roof.

18.1.4.2 Coal Mining

The basic unit in room and pillar coal mining is the panel that defines the area of the mine to be worked and ventilated. In the panel, there are two main phases in which the rooms are first developed, isolating the pillars, to the extent of the panel. Then the pillars may be extracted in a reverse direction. Conveyor belts, LHD transports, and services are extended with the room advance and are taken up during retreat pillar extraction. Room advance and pillar extraction can be carried out separately, or at the same time, or the pillars can be left in place. Kauffman, Hawkins, and Thompson (1981) describe four primary methods of production room and pillar mining—principally for coal mines—although they may be adopted for any mining operation. The methods are illustrated in Fig. 18.1.8 and may be summarized as follows.

1. *Panel advanced on entry set; rooms only extracted on retreat* (Fig. 18.1.8a). Here a group of entries, or an entry set, just large enough (usually three or four) to handle the necessary ventilation, haulage, and other support services, is developed, usually in the center of the panel, to the full panel length, connecting through to the return airway gas bleeder system in the case of a coal mine. Then production rooms in sets of four or five are driven in both directions as the equipment is retreated from the panel. No pillar extraction is carried out.

2. *Full panel advanced on rooms; pillars extracted on retreat* (Fig. 18.1.8b). Here a full width panel with 10 to 12 entries is developed off the panel neck to the full panel length, connecting through to the return airway entries and chain pillars to establish a bleeder system. Pillars are then extracted in retreat until the full panel in mined. It is essential, as will be shown, to maintain a pillar line for caving, and this is established either at an angle (conventional and continuous mining) or parallel to the line of retreat (continuous mining).

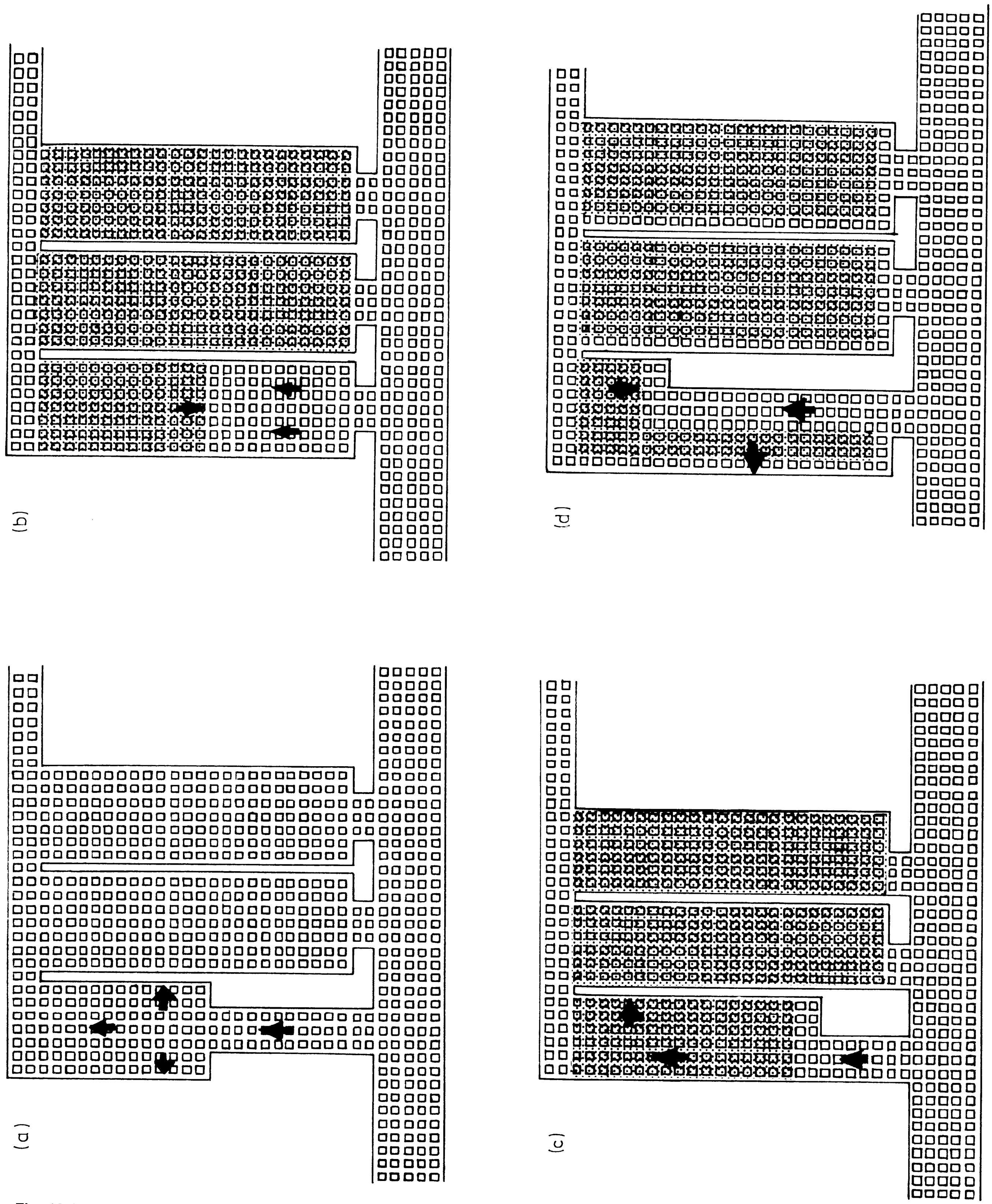

Fig. 18.1.8. Typical panel room and pillar coal mining methods. (a) Panel advanced on entry set; rooms only extracted on retreat. (b) Full panel advanced on rooms; pillars extracted on retreat. (c) Panel advanced on entry set; rooms developed and pillars extracted on retreat. (d) Panel developed on entry set; rooms developed and pillars extracted during advance and retreat. (After Kauffman, Hawkins, and Thompson, 1981.)

3. *Panel advanced on entry set: rooms developed and pillars extracted on retreat* (Fig. 18.1.8c). Here a panel entry set (three to five entries) large enough to handle ventilation, haulage, and support services is developed to the full panel length, usually on one side of the panel, although it can be in the center. After establishing a bleeder system, production rooms are developed to the side of the entry set in groups of three or four, then production and chain pillars are extracted using flat or angled pillar lines. Because of the limitation on the number of working faces, this method is only suitable for continuous mining.

4. *Panel developed on entry set; rooms developed and pillars extracted during advance and retreat* (Fig. 18.1.8d). In this method, rooms are developed and pillars extracted on one side of the panel entry set as the panel is advanced. When the entry set reaches the panel limit, and a ventilation bleeder system is established, the rooms on the other side of the entry set are developed, and the resultant pillars are extracted together with the entry set chain pillars in retreat. The pillar line can be flat or angled; the method is only suitable for continuous mining.

Kauffman, Hawkins, and Thompson (1981) consider the advantages and disadvantages of each of these methods related to the more desirable features of room and pillar mining, and these are worth repeating as they highlight the fundamental principle of this type of mining. Desirable features are listed below, and the methods not conforming are mentioned.

1. Active working places should not be near a caved area, since the increased pressures associated with caving increase the likelihood of roof falls. This is a drawback in the case of methods 3 and 4 above.

2. The length of time that openings are maintained should be a minimum. The loosening of roof bolts referred to above and exposure of roof and pillar sides to oxidation and moisture will cause deterioration. The exposure time is largest in the case of methods 1 and 2.

3. Ideally, solid coal should be retained on at least one side of the panel entry to reduce pressures on chain pillars during advance development. This is not the case in method 4.

4. Work places should be concentrated in a limited area. This reduces the area of direct supervision and improves management of the operation. This is not the case in method 2.

5. The tonnage produced between take-ups of belts and services should be maximized, and haul distances should be minimized to reduce nonproductive time. Arguably this is lowest in method 2, highest in 1, 3, and 4.

6. The ventilation system should operate with the minimum number of diversions during mining. The most difficult method to ventilate is method 4.

7. The bleeder system should be easy to establish and maintain in order to reduce ventilation. This is most difficult in the case of method 4.

8. The maximum amount of reserves should be recovered. Ore or coal left in the panel is lost and reduces the overall economics of mining. This is obviously a drawback with method 1.

18.1.4.3 Multiple Layer Room and Pillar Mines

A type of pillar mining that is common but not widely discussed is multiple-layer pillar mining where close vertical separation of pillars may lead to stability problems in roofs and floors. The applied mechanics approach to design is considered by Obert and Duvall (1967), and the main factors can also be identified from Figs. 18.1.5 and 18.1.6.

The main design approach must be to reduce stress concentrations in the roof. It is therefore logical to position pillars above pillars since the lower pillar will provide the better support for the upper pillar. Similarly the rock thickness between the mined layers must be sufficient to avoid excessive stress concentrations. This will depend on local conditions, but it can be seen from Fig. 18.1.5a that in the case of a rectangular excavation a roof thickness of twice the room height would be advisable.

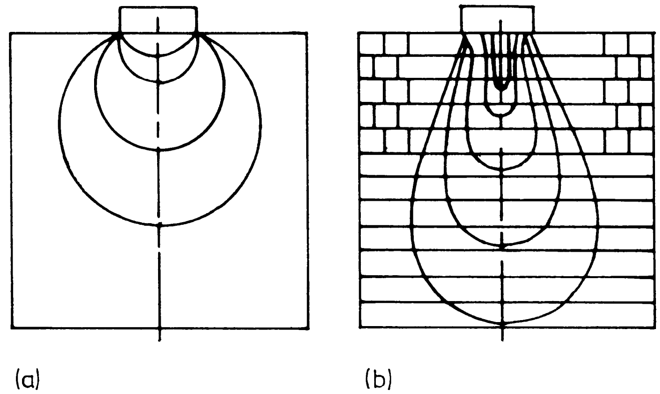

Fig. 18.1.9. Increase in major principal stress beneath a pillar in (a) homogeneous rock and (b) stratified rock. (After Gaziev and Erlikhman, 1971.)

Peng (1986) considers the particular problem in some detail, using the approach devised by Gaziev and Erlikman (1971) who demonstrated, using photoelastic models, the effect that layers of increasing or different modulus could have on the stress distribution beneath a foundation element (Fig. 18.1.9). The unavoidably high stress concentrations under pillars leads to Peng's particular recommendations for multiseam room and pillar mining:

1. *The upper seam is mined out prior to mining the lower seam.* High abutment pressure under upper seam pillars and abutments is the interaction problem most likely to be encountered in the lower seam. The design guidelines applicable to these conditions are (a) no pillars should be left unmined in the upper seam, (b) small pillars should be left in the upper seam if partial extraction is practiced, (c) pillars in the upper and lower seams should be columnized, (d) entries should not be driven under high stress zones such as abutment zones, and (e) longwalling might be the best alternative for the lower seam if pillaring is practiced in the upper seam with a few remnant pillars left.

2. *The lower seam is mined out prior to mining the upper seam.* Subsidence will be the most troublesome interaction effect. Caving induced by the lower seam mining might disrupt mining operations in the upper seam if seam separation is small. The design guidelines applicable to these conditions are (a) do not drive entries in the tensile zone of the subsidence trough, (b) reduce subsidence or arching effects by reducing opening width and extraction ratio, (c) columnize pillars, and (d) backfill the lower seam.

3. *Mining of the upper and lower seams is carried out simultaneously* with development and pillaring being kept in advance in the upper seam. Possible interaction problems are pillar stress concentrations. The design guidelines applicable to these conditions are (a) columnize pillars, and (b) keep the face of the upper seam ahead of the lower seam face by a minimum distance equal to the product of interburden thickness and the angle of draw.

18.1.4.4 Yielding Pillars

A major concept in pillar mining—although it has greater application in chain pillar design for longwall mining—is that

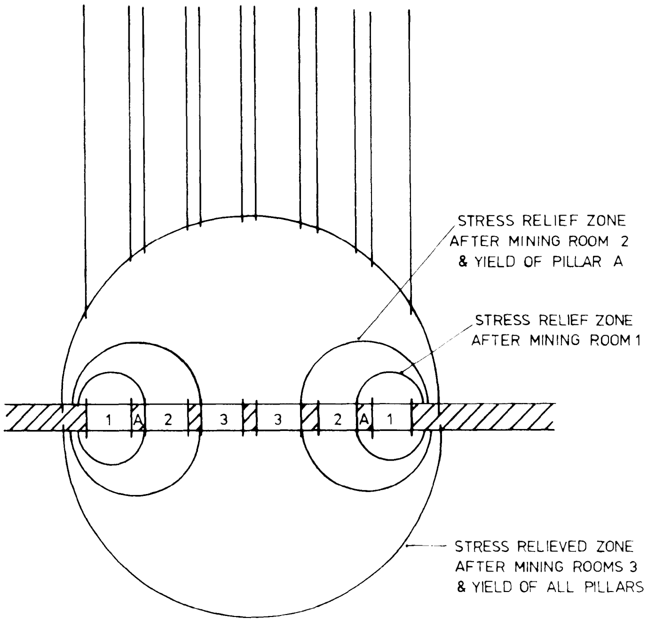

Fig. 18.1.10. A yield pillar layout for a six entry system, illustrating the development of a stress-relieved zone.

of yield pillars. A major application has been in deep potash mines, but it is important in any application where a combination of stress and rock conditions can leads to bumps, bursts or excessive deformation.

Yield pillars are pillars that are designed to yield as soon as they are isolated, so that they transfer most of their overburden pressure to the abutment pillars of the panel. This prevents the buildup of high roof and floor pressures at the edges of the pillars at the center of panel, and should ensure improved roof conditions in most rooms at the expense of the outer rooms. The detailed mechanics of yield pillar design are explained by Serata (1983), although the method has been used—often in an ad hoc way—for many years.

Fig. 18.1.10 illustrates a typical layout for a six-entry system. The outer entries are driven first, as rapidly as possible, and the adjacent entries immediately afterwards, leaving a yield pillar. Yielding of this pillar should concentrate stresses in the abutment pillar, creating a pressure arch that will lower the vertical stresses on the remainder of the panel while damaging the outer room and abutment edge. The inner entries can then be driven in stress-relieved ground. Pillar extraction, by outside lifting (see 18.1.6.2) from the four protected rooms can then be used to complete the mining process. With suitable ground conditions, this method can be adapted to a greater or lesser number of entries. Even where pillar extraction is not considered desirable or feasible, use of the yield pillar approach allows a much higher rate of extraction than conventional tributary area design, and reduces the likelihood of bumps, bursts, and other roof falls.

An alternative approach to high extraction, used in salt, potash, and trona deposits and sometimes called the time-control technique (Serata 1983), involves rapid single-, double-, or triple-entry extraction using a "Christmas tree" or chevron approach (Fig. 18.1.11). This is designed for use in weak ground, and the objective is to excavate as much ore as possible very rapidly in a controlled way, using secondary yielding pillars to protect the central access entry, and using as little support as possible over a short time period. This method is not feasible in coal mines

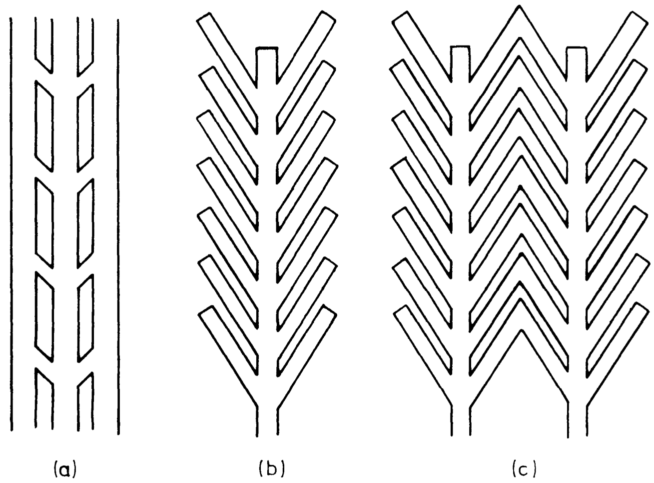

Fig. 18.1.11. Rapid development or time-control layouts used to obtain high productivity in weak deposits at depth: (a) 3-room yield pillar, (b) Christmas tree, and (c) chevron. (After Serata, 1984.)

but has been very successful in deep potash deposits where 50% extraction ratios are possible.

18.1.5 PRODUCTION METHODS—NONCOAL

18.1.5.1 Production Cycle

It is necessary to differentiate between coal and noncoal production methods. This has been done very ably by Bullock (1982 b,c,d), utilizing a US Bureau of Mines-commissioned report by Dravo Corporation (Anon., 1974) on noncoal mining and an EPRI report by Hittman Associates (Anon., 1976) on coal mining. The difference arises from three main factors:

1. *Strength*, which means that the weaker coal can usually be cut by continuous miners.
2. *Scale*, where US coal seams are generally thinner than noncoal deposits.
3. *Gas*, where coal mines are gassy and noncoal mines are usually gas free. Thus noncoal mines are usually mined by drilling and blasting off the solid in large working excavations; coal seams are undercut and blasted or continuously mined in relatively small excavations.

There are three basic types of room and pillar mining cycles, which are illustrated as flow diagrams and element interaction bar charts in Fig. 18.1.12. For *hard-rock ore bodies*, the basic cycle (Fig. 18.1.12a) is similar to hard-rock tunneling with four main elements: (1) mark out and drill blastholes, usually in a wedge pattern; (2) charge, blast, and ventilate to remove blast fumes; (3) introduce mucker and muck and load; and (4) scale the face and walls and bolt the roof where necessary. There is considerable complexity in the interaction among these elements that make up a basic critical path. In order to estimate the cycle time, it is necessary to determine unit loading and drilling rates and task times for these elements and also to estimate how subsidiary elements and tasks such as haulage and ventilation takeup may impinge upon the critical path in a badly organized mine.

18.1.5.2 Panel Development

A panel layout for a typical room and pillar mine in a noncoal mine is illustrated in Fig. 18.2.3 (see Chapter 18.2). The

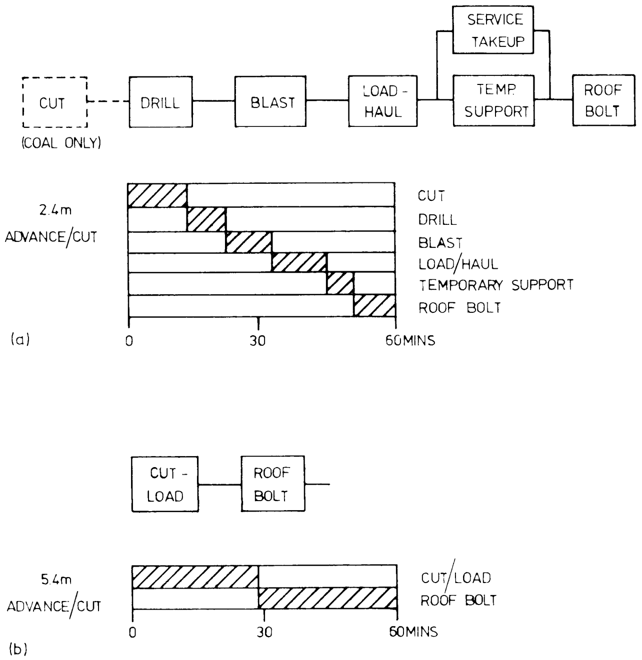

Fig. 18.1.12. Flow diagrams and element interaction bar chart for (a) conventional room and pillar and (b) continuous mining. Conversion factor: 1 ft = 0.3048 m.

excavation height is about 15 ft (4.5 m), and the normal stoping practice is to drive a single development drift about 35 ft (10.5 m) wide a distance of about four or five rooms into the ore body. This will serve as the main haulage drift. Pillars are then marked out on the drift walls and rooms driven between them.

To drill and blast the initial drive when the only exposed or free face is the drive face, some form of cut pattern is used. This is known as the “face round” or “swing” and in a 15- by 35-ft (4.5- by 10.5-m) face will comprise 60 to 70 holes (see Chapter 9.2) of about 1½ in. (38 mm) to a depth of 10 to 12 ft (3 to 3.6 m). If more than one face is exposed, a group of holes may be drilled at a low angle to the free face in what is known as a “slab round” or “slabbing” or “slashing”. This requires less explosive and less drilling than a single face. The most common form of face round is a wedge or V, cut although burn cuts can also be used.

Drilling is carried out with jumbo-mounted hydraulic drills; loading is usually by gathering arm loader, although in modern mines, trackless LHD vehicles are used to carry the load to a transfer raise where it is reloaded into trucks or conveyors. Some typical productivity figures for this type of operation are given in Table 18.1.4.

18.1.5.3 Cut and Fill Pillar Mining

Where the roof can be caved, as in coal mining, high levels of extraction can be obtained by retreat mining. Where the roof is stronger—as in most non-coal mining—the pillars are normally left as semi-permanent support. In high-grade ores at depth or where roof conditions are poor, loss of from 25 to 50% of the ore body may be unacceptable and, in this case, backfill may be considered. Placing of backfill can most easily be carried out using a form of slot and pillar, rather than room and pillar mining where parallel panels or drives are developed across the strike and then filled in an alternating sequence. A particular arrangement at a gold mine in Washington, designed to reduce subsidence in this case, is described by Tesaric, Seymour, and Vickery (1989) and Brechtel (1987). Slots or rooms (Fig. 18.1.13a) were excavated in 50-ft (15-m) vertical intervals by multiple benches 24 ft (7.3 m) wide and 24 ft (7.3 m) high. They were mined and filled in an alternating sequence from footwall drifts. The cemented fill was dumped from dumper trucks and allowed to settle at its angle of repose. At the top of the ore block, the backfill was rammed tight using a plate mounted on a LHD. It is a relatively simple system of mining that can be adapted for any room and pillar configuration. Completed stopes range in height from 30 to 130 ft (9 to 40 m), depending on their location in the ore zone. The fill comprised 55% minus 2-in. (50-m) river gravel, 40% alluvial sand, and 5% cement. A more useful mix might utilize tailings, which are often pozzolanic and require little or no cement.

A very radical approach to backfill pillar mining has been suggested by Dixon (1990). Called spiral slot and chamber mining, it is presented as a total extraction method for strata-bound deposits in a horizontal plane. The ore body is mined (Fig. 18.1.13b) in a continuous, flat, but not necessarily circular, concentric spiral pattern. There are three operations—top heading, benching, and backfilling from radial crosscuts with a chamber width of 30 ft (9 m)—the initial slot spiral being followed by a chamber spiral. The slot is backfilled with cemented fill the chamber with mine tailings or sand. Several potential benefits are claimed. The pattern should induce more even and favorable stress distributions than conventional layouts and should be more amenable to automation, leading to reductions in bursts, better strata control, and improved productivity.

One problem with backfilling, apart from the major logistical one, is the cost of cement, and in some cases its availability. Mitchell (1989) suggests using geogrid reinforcements as an alternative, and this is probably feasible. However, most silicates have some pozzolanic properties and it may be that added cementing agents, particularly in bulk fills, are unnecessary.

18.1.6 PRODUCTION METHODS—COAL

18.1.6.1 Panel Development

In coal mining, blasting off the solid is illegal, principally because of the danger associated with blown-out shots in an environment where explosive gases may be present. Where blasting is used, a horizontal cut is formed, generally in the face. This is usually a bottom cut in thin seams, a center cut in thicker seams. A vertical center cut may also be used. A similar approach may be used in rock salt and potash mining. The presence of the cut creates a free face for blasting and reduces both the amount of explosive needed and the possibility of blow-outs. A typical cutter jib is 9 to 12 ft (2.7 to 3.6 m) long, and the picks are arranged to cut a 6-in. (150-mm) slot or kerf. The cutter jib is sumped into the center of the face and moved to each side to complete the undercut. The basic cycle of operations (Fig. 18.1.12a) thus requires one more element before drilling.

Cyclic systems are usually referred to as *conventional room and pillar mining.* Much more productive and much commoner in mechanized mines is *continuous mining.* This is particularly important because, as the name implies, it reduces the number of unit operations and hence the cyclic element (Fig. 18.1.12c). There are several types of continuous miner, but they all combine the same basic elements—a cutting head above, or combined with a gathering arm loader, and attached to a short armored conveyor—so that the only delays on the critical path are for

Table 18.1.4. Typical Productivity of Noncoal Room and Pillar Mines (1970s Data)

Type of Rock	Location	No. of Mines in Sample	Range of Production 1000 tonnes/day	Range of Productivity tonnes/employee-shift	Average Productivity tonnes/employee-shift
Dolomite (Lead/Zinc)	New Lead Belt, MO	8	1.6–4.5	19.2–59.0	32.6
Cherty Limestone (Zinc)	E. Tennessee	5	1.4–3.1	12.5–26.6	17.8
Shale & Sandstone (Copper)	N. Michigan	1	6.2	—	14.6
Limestone	Various	45	1.1–8.7	31.8–136.1	74.5
Phosphate	Utah	1	2.2	—	52.2
Shale & Potash	N. Mexico	6	3.6–8.3	22.0–75.0	42.8
Salt	Various	10	1.6–10.8	14.3–63.6	36.9
Sandstone	Pennsylvania	1	2.4	—	73.7
Salt (Trona)	Wyoming	3	3.2–5.6	27.9–63.1	41.1

Conversion factor: 1 ton = 0.9072 t.
Source: Bullock, 1982a.

ventilation and roof support. The reduction in the number of unit operations means that for efficient operations, a much smaller number of faces can be worked continuously. This is illustrated in Fig. 18.1.14, which shows typical plans for (a) conventional development of a six-pillar, seven-entry room and pillar panel, developed as in Fig. 18.1.8b as full panel, and (b) continuous miner development of a four-pillar, five-entry set for rapid development of a panel (see Fig. 18.1.17c or d).

In Fig. 18.1.14a, the conventional method, initial development is on 20-ft (6-m) rooms with 60- by 50-ft (18- by 15-m) pillars. Twenty feet (6 m) is the maximum room width under the Federal Coal Mine Health and Safety Act. The advance per cut is planned to be 10 ft (3 m). Unit operations have been described in detail by both Stefanko (1983) and Bullock (1982). The basic elements of the plan can be seen and can be described in terms of the working cycle in Fig. 18.2.3. Cut 1 (entry 7) is being loaded; entry 1 has just been loaded, and the loader, typically a gathering arm loader with integral armored flight conveyor on caterpillar tracks, has moved from this entry to cut 1. The blasted material is loaded into a rubber-tired shuttle car, which transports it to the feeder belt where it is dumped. There are usually two shuttle cars, and, depending on whether they are cable reel electric or diesel, they follow the same or separate paths. Cut 2 (entry 6) is being charged and prepared for blasting. Cut 4 (entry 4) is being drilled, and cut 5 (entry 3) is being undercut. Cut 6 (entry 2), after loading is being bolted, and cut 7 (entry 1) is being prepared for bolting and other service move-ups. The continuing sequence following this cycle can be seen in the numbered cuts in Fig. 18.1.14a.

The efficiency of the operation and the overall productivities and advance per shift depend on the time taken for each of the elements in the cycle, the way in which they interact, and the speed with which equipment can be moved from one entry to the next. Typical times, from the same sources as Fig. 18.1.14a, are:

Cutting cycle	11–16 min	2 employees
Drilling cycle	8–12 min	1 employee
Blasting cycle (6–8 holes)	8–12 min	1 employee
Load/haul cycle	12–20 min	3 employees
Temporary support cycle	5–10 min	1 employee
Roof bolt cycle	8–15 min	2 employees

Since these cycles are concurrent, the overall cycle time will be the longest part of the cycle, probably the load/haul cycle, which is discussed later. Ideally, it should be possible to complete a cycle in 12 to 20 minutes and to complete 24 to 40 cycles in each 8-hour shift. This may be compared with Table 18.1.5, which includes actual data. Bearing in mind the time lost in shift changes, traveling, and breakdowns, the actual achievement is remarkably good.

Continuous mining is noncyclic and utilizes a smaller crew, and the results (Table 18.1.5) are better in comparable situations. The major advantage is that the reduction in cycle time reduces the number of entries that need to be driven in order to maintain output to 3 or 4, since the only separate operations needed are cutting and roof support. Where these can be combined, and shuttle cars eliminated by extensible conveyors, then 100 ft (30 m) of continuous driving can be obtained to isolate a pillar side before moving the machine. The method then becomes very productive.

18.1.6.2 Methods of Pillar Extraction

Four basic pillaring methods are described in detail by Kauffman, Hawkins, and Thompson (1981). These are split and fender, pocket and wing, outside lift, and open ending. Each of these methods is illustrated in Fig. 18.1.15, and their basic characteristics are summarized in the following.

1. *Split and fender* (Fig. 18.1.15a) is the most commonly used pillar extraction method in the United States. The basis of

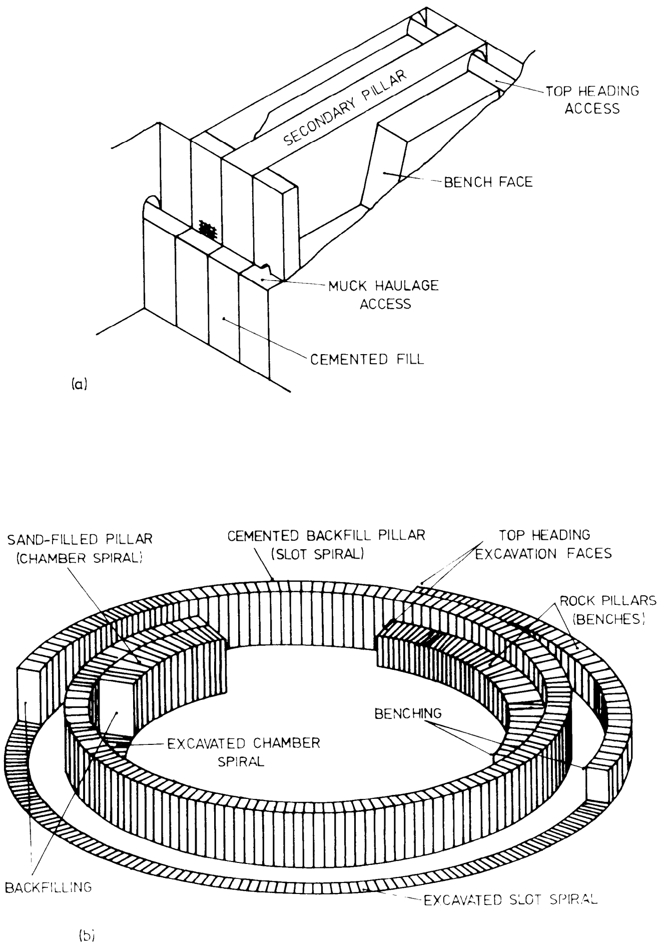

Fig. 18.1.13. (a) Slot and fill mine layout (after Brechtel, 1987). (b) Spiral slot and fill mine layout (after Dixon, 1990).

the method is to mine through the pillar center parallel to the longer side, creating a split and a fender of coal on each side of the split. Before mining, breaker posts are placed at all openings to the gob, and roadway posts are placed to reduce roadway widths to 16 ft (4.8 m). Turn posts and breaker posts are used in the split for additional support, and roof bolts are installed, as in room development, to support exposed roofs. The split is the same dimension as the rooms in the original panel, and the width of the fenders is usually fixed so they can be wholly extracted by the continuous miners without additional support. This effectively determines the maximum pillar width. Under most conditions, the minimum fender width is 8 ft (2.4 m), and the maximum about 13 ft (3.9 m). The split width can range from 10 ft (3 m) to 20 ft (6 m), giving a range of pillar widths from 26 ft (7.9 m) to 46 ft (14 m). Wider pillars can be extracted using multiple splits, but this reduces the simplicity, and the pocket and wing technique is more suitable. There is no limit on pillar length. The method usually involves mining two or more pillars simultaneously. Fig. 18.1.15a illustrates a double pillar sequence with 1 to 7 and 16 being split operations, the remainder fender operations. Support comprises roof bolts in the split. Breaker posts are installed after extraction of splits 7 and 16; turn posts are set across the split before the cut to extract each fender segment. Each fender sequence is therefore extracted from beneath supported roof. Ventilation is difficult, involving quite complex brattice curtain erections at critical points. The process is, however, simple and can be adapted to all thicknesses from 40 in. (1 m) to 25 ft (7.5 m) and to all equipment from simple loaders to continuous miners. The method is generally not suitable for large pillars and fragile roofs.

2. *Pocket and wing* (or pocket and fender, Fig. 18.1.15b) is a single pillar extraction method used mainly in northern West Virginia. Two working places are extended in the pillar leaving wings or fenders to support the roof. It can be easily adapted to large pillars and allows concentration of working places in a pillar, and hence, rapid extraction. Ventilation and haulage are also easier. It is not as efficient as the split and fender method and is used primarily where mining at depth requires large pillars for roof control. The method is not suitable in bump conditions.

3. *Open ending* (Fig. 18.1.15c) is a method similar to pocket and wing, but the mining sequence is taken along the sides of the pillars, breaker posts being extended at the pillar edge. It has limited use; ideally, the roof should be competent enough to span the opening, but brittle enough to break off or cave beyond the breaker posts.

4. *Outside lifts* (Fig. 18.1.15d) are rarely used except to extract narrow pillars rather like the fenders of the split and fender method. The method is used in shallow mines that allow the safe use of small pillars. A variation is in deep pillar extraction, particularly in bump prone areas where residual small yielding pillars are desired. Such a plan permits rapid extraction. The method can also be used for ad hoc partial extraction of pillars where bolting is not needed.

18.1.6.3 Mobile Roof Support (MRS)

Retreat pillar mining is highly productive. Supply, power, haulage, and ventilation systems are established during panel development and knowledge of roof and water conditions obtained. It is also dangerous, particularly where the roof does not cave in a predictable manner, and where the seam is prone to bursts, floor heave, and crushed pillars. The prime factor in improving safety is successful roof control through correct design of pillars, including yield pillars and supports such as posts, cribs, and roof bolts. These supports have the disadvantage that they act in a passive way. Technology from longwall mining, where active waste edge shields give an added dimension to roof control, was urgently needed, and this has been supplied through mobile roof supports (Thompson, 1983) developed initially by the US Bureau of Mines and improved by Fletcher (1990).

These are approximately 14 ft (4.2 m) by 6 ft (1.9 m) wide. Specific design features (Fig. 18.1.16) include continuous miner-type crawler drives with variable speed motors, hydraulically operated plows, 600-ton (540-t) total load capacity (through four cylinders each with 150-ton, or 135-t, yield load capacity), lemmiscate-linked canopy with free-floating support cylinders, and a heavy-duty rear shield. The supports are self contained apart from a cable reel electric power source, and are operated through a hand-held radio remote controller.

The mobile roof supports are typically used in two-pair configurations (Fig. 18.1.17), each pair being located between a solid pillar and the pillar being extracted. After each miner cut, they are advanced one unit at a time until there is just enough space remaining for the miner at the last cut.

18.1.7 VENTILATION

18.1.7.1 Bleeder Systems

Ventilation is particularly important in coal mining, and provisions of the 1969 Coal Mine Health and Safety Act affect

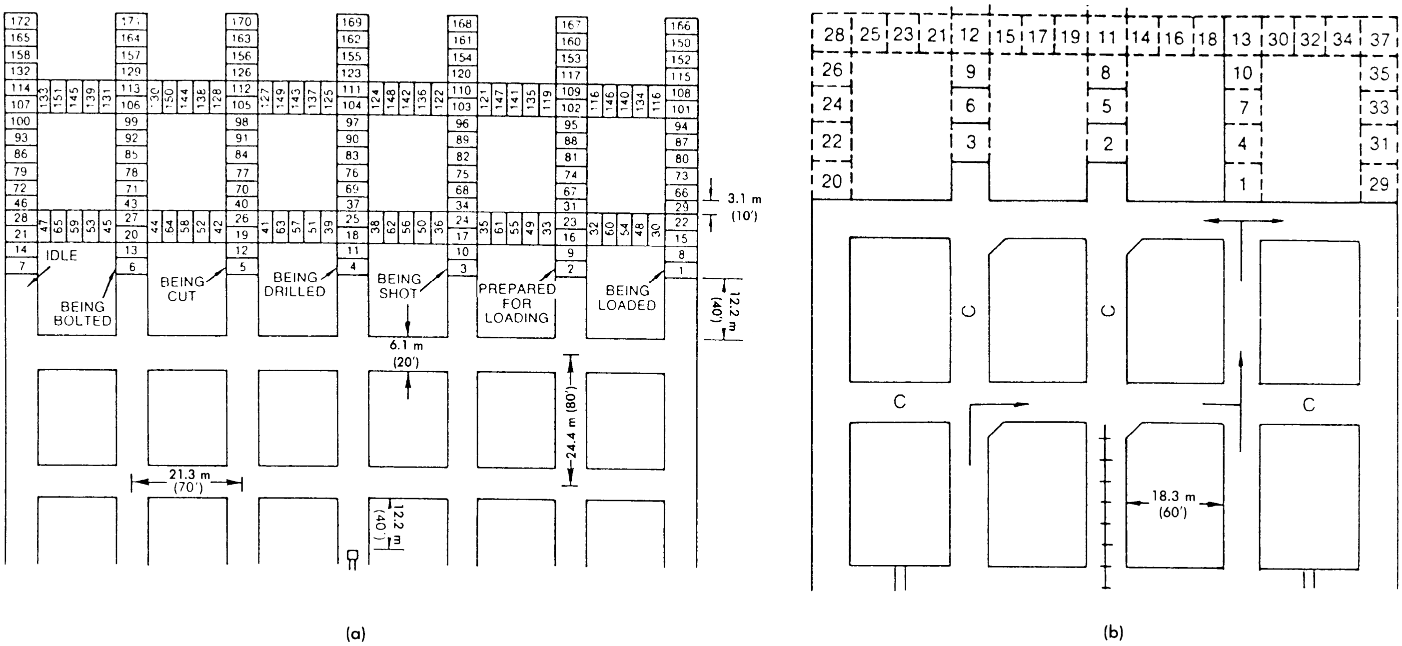

Fig. 18.1.14. (a) United operations in conventional room and pillar mining, showing the cut sequence of a seven-entry plan. (b) Mining sequence for a five-entry continuous mining operation (after Anon., 1976).

Table 18.1.5. Comparisons of Productivity Estimates for Conventional (A) and Continuous Miner (B) Room and Pillar Mining

Production Data	Conv. A	Cont. B	Conv. A	Cont. B	Cont. A	Cont. B
Seam height, ft (m)	4 (1.2)	4 (1.2)	5 (1.5)	5 (1.5)	6 (1.8)	6 (1.8)
Width, ft (m)	20 (6.1)	20 (6.1)	20 (6.1)	20 (6.1)	20 (6.1)	20 (6.1)
Depth of cut, ft (m)	8 (2.4)	18 (5.4)	8 (2.4)	18 (5.4)	8 (2.4)	18 (5.4)
Tons/cut	23	53	29	65	34	78
Cuts/shift	20	8.7	20	8.3	18	8.2
Tons/shift	464	454	581	544	617	635
Work min/shift	400	400	400	400	400	400
Face Crew Employee (mins/cut)						
Continuous miner	—	1 (31)	—	1 (33)	—	1 (33.5)
Loading machine	2 (30)	1 (31)	1 (30)	1 (33)	2 (34)	1 (33.5)
Shuttle cars	2 (30)	2 (62)	2 (30)	2 (66)	2 (34)	2 (67)
Cutting machine	2 (30)	—	2 (30)	—	2 (34)	—
Drill	1 (13)	—	1 (13)	—	1 (15)	—
Shooting	1 (17)	—	1 (17)	—	1 (20)	—
Roof bolt machine	2 (30)	2 (72)	2 (30)	2 (72)	2 (30)	2 (76)
TOTAL FACE CREW	10 (150)	6 (196)	10 (150)	6 (204)	10 (168)	6 (210)
Employee min/shift at the face	3000	1700	3000	1700	3024	1720
Employee min/ton	5.32	3.07	4.25	2.59	4.01	2.21
Tons/employee-shift	46.4	75.6	58.1	90.7	61.7	105.8

Conversion factor: 1 long ton = 0.996 t.
Source: Anon., 1977.

ventilation of room and pillar mines. A major provision is the requirement for bleeder entries and systems. *Bleeders* (Kauffman, Hawkins and Thompson, 1981) are entries surrounding an area being mined or which has been mined out. The purpose of bleeder entries is to bleed methane and other explosive gases from the gob area and into the main mine return airways, using a controlled filter of intake air. Bleeder entries should be maintained for access and examination. Only in areas liable to spontaneous combustion is sealing of caved areas permitted.

The practice of bleeding requires that a differential air pressure be maintained between the intake and return airways across a gob so that gases flow into the return airway. Where there

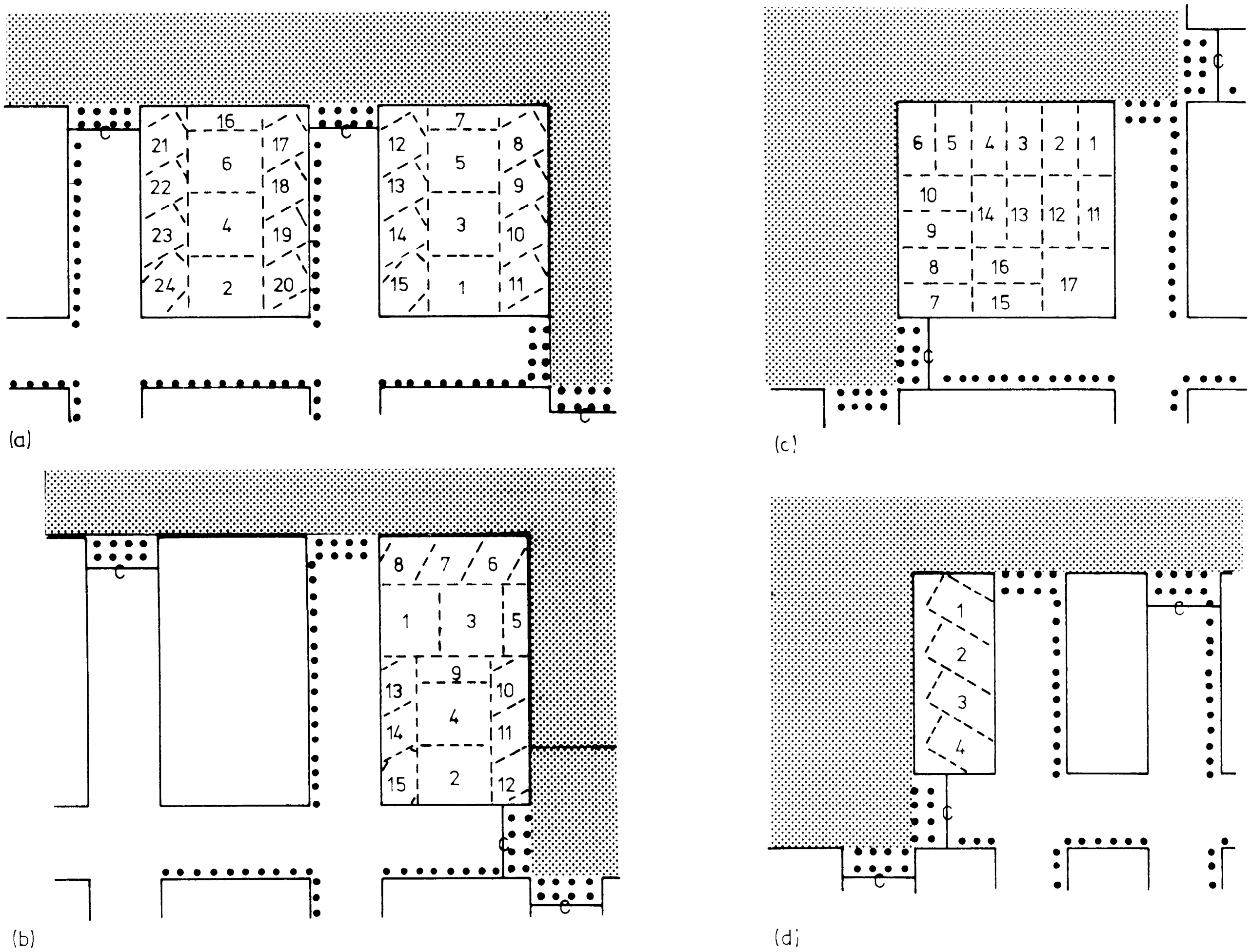

Fig. 18.1.15. Overall cut sequence for pillar extraction during retreat mining: (a) split-and-fender method, (b) pocket-and-wing or pocket-and-fender method, (c) open ending method, and (d) outside lift method (after Kauffman, Hawkins, and Thompson, 1981).

has been pillar extraction beneath a blocky roof, there will be sufficient flow through the caved area to bleed. A laminated roof may seal the caved area and an entry through the gob may be needed.

In order to simplify bleeding, it is sometimes desirable to remove barrier pillars during retreating to connect panels and to allow bleeding of an extended area of the mine. During development, it is better practice to bleed individual panels.

For details of mine ventilation theory and practice, see Chapters 11.6 and 11.7.

18.1.7.2 Section Ventilation

As discussed in Chapter 11.7.2, each working section of a coal mine should be ventilated with a minimum of 9000 cfm (4.25 m^3/s) of air to the last open crosscut. At least 3000 cfm (1.42 m^3/s) must reach each working face where coal is being mined. The air must contain more than 19.5% oxygen and less than 0.5% carbon dioxide.

During development, air going into the section is directed to the face by means of curtains across entries, line curtains, or auxiliary fans. This air is then directed to the main return. While rooms are being driven and pillars extracted, air going to the section is usually split at the working place with some going to the section return and some to bleed the gob.

Blower or exhaust fans in coal mines must be capable of delivering or exhausting 3000 cfm to or from the working face. Exhaust fans have the advantage that they can remove dust, fumes, and gas from the working area more efficiently than blower fans, particularly if the tubing is close to the continuous miner head.

A typical exhaust ventilation layout for a coal room and pillar system is illustrated in Fig. 18.1.18. This is a system employing line brattice curtains and is described by Stefanko (1983). The line brattice is essentially a space divider or temporary partition made of an impervious material that is installed and maintained carefully and kept as close to the face as possible. Its purpose is to guide the airflow through the face area and last open crosscut and into the return. Brattices were formerly (and to some extent still are) made of untreated jute, but nylon-reinforced plastics and similar materials are more commonly used today.

The line brattice is installed so as to split the heading longitudinally and thus provide an inlet as well as a return from the face to the last open crosscut. Since the mining machine must have room to maneuver on one side of the brattice, it is not practical to split the entry evenly, so a wide side is provided for the machine. The air may be brought up the narrow side and, after it sweeps by the face, returned on the wide side. Arrange-

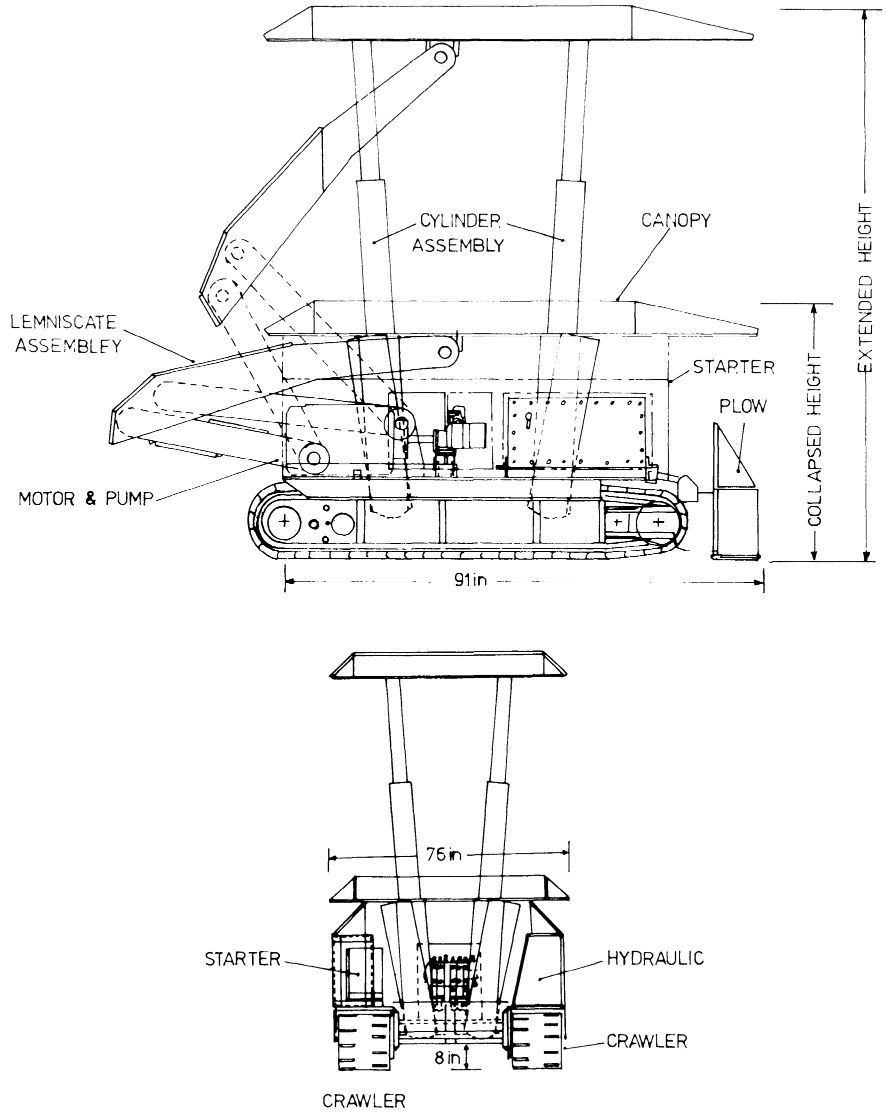

Fig. 18.1.16. Design features of the mobile roof support. Courtesy of Fletcher Mining Equipment Co. Conversion factor: 1 in. = 25.4 mm.

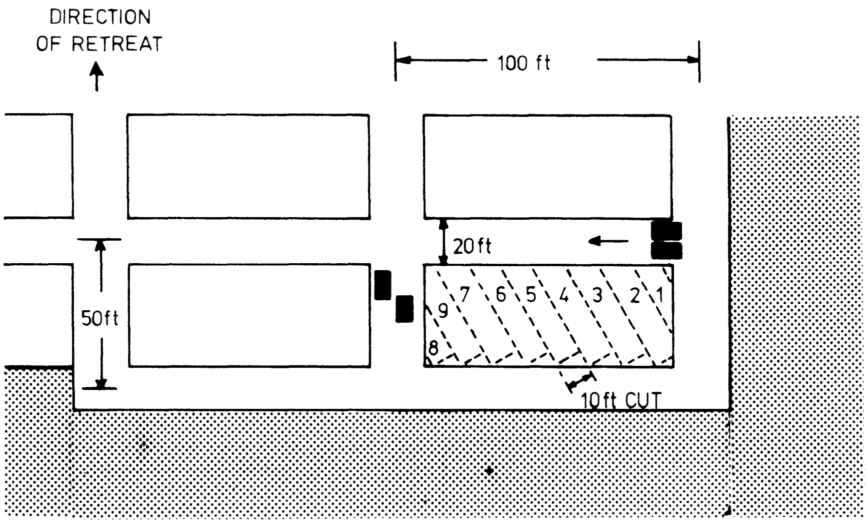

Fig. 18.1.17. A pillar extraction method using two pairs of mobile roof supports. Courtesy of Fletcher Mining Equipment Co. Conversion factor: 1 ft = 0.3048 m.

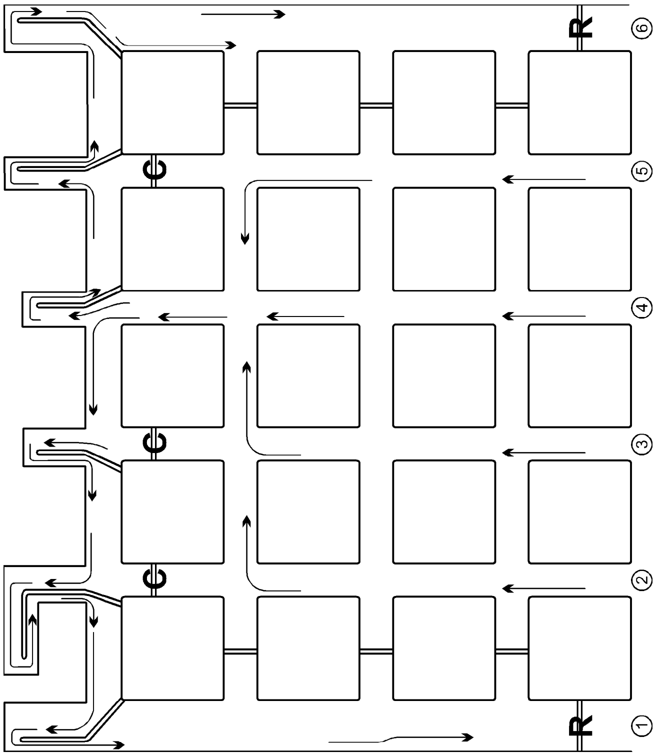

Fig. 18.1.18. An exhausting line brattice ventilation layout for a six-entry room and pillar panel (after Stefanko, 1983).

ments may be made to blow or force the air into the heading or to exhaust it. The exhaust system illustrated in Fig. 18.1.18 is more commonly used since the operators work in fresher dust-free air. Exhaust fans can be used to replace line brattices at the face, and these are effective in controlling dust.

18.1.8 SYSTEMS ANALYSIS

The room and pillar method is particularly suitable for simulation modeling and systems analysis. Particular approaches are recommended by Bullock (1982c) and Stefanko (1983), but the most sophisticated model is that of Manula and Suboleski (1982). In this *Handbook*, there is coverage of systems analysis in Chapters 8.3 and 9.4.

Simulation modeling to optimize productivity requires a detailed analysis of the mining process and the way in which the mining variables interact with and affect the selection of mining methods and equipment. Mining variables include seam height, floor quality, roof quality, methane quantities, coal hardness, depth, and presence of water. Functional relations between these and unit operations are the basis—together with observations from underground studies—of planning and simulation. For instance, in a typical cycle, *cutting* and *continuous miner* operation will be affected by the seam thickness, floor quality, water presence, and particularly the strength of the coal. *Drilling* will be affected by coal hardness, because more holes will be required to break the coal, and *blasting* will similarly take longer if more holes are used. *Roof bolting* will be affected by roof quality, and *ventilation* by methane quantities. Layout and pillar size will be determined by depth, and this will affect loading and hauling.

Loading and hauling are unique in that they are affected by layout and design and are to a large extent independent of mining variables apart from weak or wet floors. They are also the major factor affecting conventional and continuous miner cycle times. Manula and Suboleski (1982) and Bullock (1982c) illustrate how *total cycle times* may be predicted through a simple mathematical model. Thus loader cycle time LCT for a cut is given by:

$$\mathrm{LCT} = \mathrm{LT} + \mathrm{COT} + \mathrm{WSC} + \mathrm{MISC} \qquad (18.1.14)$$

where $\mathrm{LT} = \mathrm{T}/(\mathrm{LR})$
$\mathrm{COT} = (\mathrm{N} - 1)\ (2 \times \mathrm{COD}/\mathrm{SPD})$
$\mathrm{WSC} = [(\mathrm{N} - 1)/2]\ (2 \times \mathrm{HD}/\mathrm{SPD} + \mathrm{DT}) - (\mathrm{NO} - 1) \times (\mathrm{CAP}/\mathrm{LR}) + (2 \times \mathrm{COD}/\mathrm{SPD}) - (\mathrm{NO} - 1) \times (\mathrm{CAP}/\mathrm{LR}) + (2 \times \mathrm{COD}/\mathrm{SPD}) - (\mathrm{NO} - 1) \times (\mathrm{CAP}/\mathrm{LR}) + (2 \times \mathrm{COD}/\mathrm{SPD})$
$(\mathrm{N} = \mathrm{T}/\mathrm{CAP})$

and where LT is the time loader spends loading, COT is the time the loader spends waiting for cars to travel, WCS is the time the loader spends waiting for the car to arrive at the change point after the other car has cleared the change point, MISC is the time to check for connect and disconnect water hose, hand curtain, tram, etc., T is the weight of coal in the cut, LR is the mean loading rate, N is the number of shuttle car loads in the cut, COD is the change out distance one-way, SPD is the mean shuttle car speed, CAP is the mean shuttle car payload, HD is the distance from dump to change point, DT is the mean dump time of the shuttle car, NO is the number of shuttle cars in use (normally equals two), [ ] indicates truncation of the number to the next lowest integer, and { } indicates raising of number to next highest integer.

Analysis of Eq. 18.1.14 can give several indications about production improvement methods. They are relatively insensitive to loading rate increases alone since a decrease in LT is partially offset by an increase in WSC, all other factors remaining equal. Increasing the number of cars in use at one time will decrease WSC but will have no effect on COT (in practice, COT is often increased in the case of three cars, since one of the cars may be forced to change out farther from the face than the other two); and the greatest sensitivity is experienced with a change in shuttle car payload, since this affects N, which in turn affects the value of COT and WSC. The payload also affects WSC directly.

Time studies and simulations of room and pillar mining systems indicate that *change-out time* can represent from 15 to 25% of the available time for production. (This is defined as the shift time less travel, face preparation, scheduled meetings, breakdowns, lunch, servicing, etc.; that is, the time in which the units and men otherwise are actually capable of coal production.) In general, available time for production will range from 175 to 300 min/shift with an "average" value at 225 min. Thus, 30 to 60 min could be saved if suitable continuous haulage units were available. It must be recognized, however, that not all of this time will be additional loading time. In general, this time will be distributed proportionally among the remaining loading and hauling activities.

Additional time is lost in those cuts where the car cannot go back to the change point at or prior to the time it is cleared by the previous car. The maximum distance from the dump to the change point at which an additional wait will not be encountered can be calculated by balancing the load and change-out time with the haul and dump times:

$$\mathrm{WAIT} = \mathrm{O} = (\mathrm{TTD} + \mathrm{D} + \mathrm{TFD}) - (\mathrm{N} - 1)\ (\mathrm{L} + \mathrm{T1} + \mathrm{TO}) \qquad (18.1.15)$$

where $TI = SE \times COD$
$TO = SL \times COD$

$TTD = SL \times HD$
$TFD = SE \times HD$

Then

$$(SL + SE) \times HD + D = (N - 1)\ (L + (SL + SE) \times COD)$$

or

$$HD_{max} = (N - 1)\ L - D + (N - 1)\ (COD)\ SL + SE$$

where L is loading time, TTD is the time to travel from the change point to the dump, TFD is the time to travel from the dump to the change point, D is mean dump time, N is number of haulage units in service, SE is mean travel time unloaded, SL is mean travel time loaded, TI is the changes out time unloaded, TO is the change-out time loaded, COD is change-out distance, and HD is the distance from dump to change point.

These are examples of simple approaches to planning. A fuller listing is given in Manula and Suboleski (1982), and elsewhere in this *Handbook.*

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Chapter 18.2
STOPE AND PILLAR MINING

CHRISTOPHER HAYCOCKS

Stope and pillar is probably the oldest underground mining method, having been developed by flint miners in Europe some 6000 to 8000 years ago (Temple, 1972). Remains of many of these workings are still visible today in the great chalk beds. They show that these ancient underground miners had learned the fundamental rules of stope and pillar mining: (1) to leave sufficiently large ore pillars for roof support, and (2) to limit the width of openings to minimize the possibility of roof falls. The occasional skeletons of miners crushed by falling roof prove that these lessons were not learned without cost. In its modern form, stope and pillar mining remains essentially unchanged except for the effects of mechanization and vertical expansion to accommodate more massive deposits. Stope and pillar differs from room and pillar mining in that room and pillar is a term commonly reserved for coal mining, which has regular pillars normally developed in panels, and thin deposits of usually less than 8 ft (2.4 m). Stope and pillar is also known as breast stoping, breast and bench stoping, board and pillar, stall and pillar, and panel and pillar (Hartman, 1987).

18.2.1 GENERAL FEATURES

Stope and pillar mining is characterized by

1. Irregularly shaped and sized pillars left for support (Fig. 18.2.1), which may require little or no planning. In the extremes, the method is often little more than a variation of gophering (Peele, 1941) or breast stoping, where only artificial pillars are used for roof control. Most mines, however, specify minimum pillar dimensions to meet overburden support requirements, and the dimensions must take into account the ultimate pillar height, which is a major factor in pillar strength.

2. Typically flat ore bodies of large horizontal extent dipping less than the angle of repose. Below this angle of about 30 to 35°, the ore will not flow under the influence of gravity; this is the major difference between stope and pillar and the higher-angle stoping methods such as cut and fill, shrinkage, and sublevel. Cascade stoping developed for the Zambian Copperbelt mine of Mufulira was designed as an intermediate stoping method for ore bodies dipping at the angle of repose or in the range of 25 to 45° (Kelly, 1969). Massive ore bodies can be mined by the stope and pillar method by mining in a series of horizontal slices or passes from or near the top down, such as at the lead-zinc deposits at Austinville, VA, and the Bonne Terre Mine, MO, where pillars up to 300 ft (90 m) high were left (Wykoff, 1950).

3. Competent rock masses. These are very desirable as this is an open method where the stopes or rooms normally stay open during the life of the mine. Strengths may vary from 50,000 psi (345 MPa) to 4000 psi (25 MPa), and structural integrity of the roof, pillar, and floor components should ideally be excellent, unless increasing quantities of secondary support are to be used. There are numerous examples of successful stope and pillar mining in less than competent rock, such as some areas of the new lead belt in Missouri and the uranium mines of New Mexico and Wyoming (Bullock, 1982). Under these conditions, roof stability is of primary importance, secondary support is frequently used, and continual barring down or scaling may be essential. Uncontrolled pillar failure is to be avoided at all costs, as the failure can be progressive to surrounding pillars and jeopardize the entire mine. Occasionally, pillar recovery is possible under good ground conditions and where mine stability can be carefully controlled.

4. Definite depth limitations due to the load capacity of the pillars. Increasing pillar size to combat increasing vertical load is counterproductive as extraction decreases and can range from 25% in deep salt mines to a maximum of 95% in selected lead mines (Bullock, 1982; Prugger, 1984). Absolute depths to which the method can be taken depend on site-specific conditions, principally rock mass strengths, but typically 3000 ft (600 m) is a normal limit. In the deeper operations pillar sizes become so large that extraction falls to uneconomic limits and alternative methods must be considered. The exception to the depth limitation is the special case of breast stoping, which permits total stope closure and is used successfully at depths over 12,000 ft (3600 m).

5. Absence of surface subsidence, except in the very long term when massive roof or pillar failure has occurred, or in the case of breast stoping. Considering the fact that the oldest underground mines in existence are stope and pillar, it is possible to design successfully for long-term stability.

6. Stope and pillar mining is normally classified as a large-scale method in terms of total production, and is quite versatile and flexible enough to meet a wide range of production requirements.

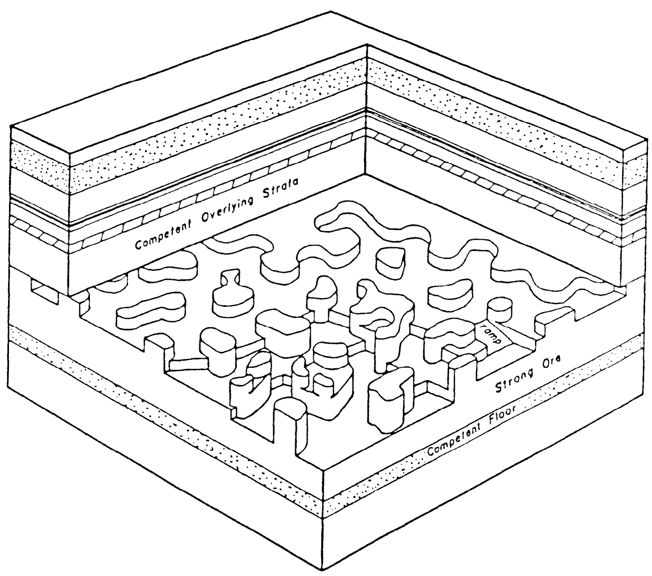


Fig. 18.2.1. Stope and pillar mining with random pillars (Clark and Caudle, 1961).

18.2.2 CONVENTIONAL STOPE AND PILLAR

18.2.2.1 Development

The basis of the stoping system is straightforward since, in most forms, the method requires no specific stope development unless the ore body is very irregular and a separate haulage level is driven. Under these conditions connecting orepasses, ventilation ways, and access raises will be necessary. Most mines hoist from a single level, and many utilize declines to facilitate movement of rubber-tired equipment underground.

18.2.2.2 Stoping

Stope and pillar mining is typically cyclic, consisting of the following unit operations (Hartman, 1987):

$$\text{DRILL} \rightarrow \text{BLAST} \rightarrow \text{SUPPORT} \rightarrow \text{LOAD} \rightarrow \text{HAUL} \rightarrow \text{DUMP}$$

In softer materials such as trona or halite, mining machines may be used to replace the drill-blast cycles. Drilling is usually carried out by self-contained, diesel-powered, trackless jumbos, but in smaller operations, hand-held jacklegs may be used. Under very soft-rock conditions, electrical-powered auger drills may be utilized. Holes are drilled horizontally using wedge or V cut rounds, although burn cuts may be used under some conditions such as in the harder limestones and dolomites (Fig. 18.2.2). Slabbing or sliping rounds are then typically used to expand the initial face cut around the proposed pillar (Fig. 18.2.3). Faces, or breasts, may be carried up to 30 ft (9 m) in height with the majority of conventional equipment, and hole depths of 16 ft (4.8 m) are suitable for most conditions. The predominant explosive is ANFO under dry conditions, with slurries, water gels, and dynamites as alternatives. Blasting may be carried out during the shift or at shift change times, depending on the size of the mine. Good fragmentation plus limiting the throw of the round are important in stope and pillar mining. Cleaning a large area and secondary blasting can seriously disrupt the production cycle. Barring down or scaling the back using cherry pickers is done immediately following blasting.

Mining takes place on the advance with ground control factors rarely, if ever, allowing for complete extraction of pillars, although substitution of artificial pillars has been tried under some situations (Reed and Mann, 1961). The use of artificial pillars was technically successful, provided the pillar could be prestressed; otherwise, roof deformation to load the pillar caused significant roof failures (Chellson, 1941). Unfortunately artificial pillar systems have all proved uneconomical to date in conventional stope and pillar systems, although they are an essential part of breast stoping.

Depending on the shape and thickness of the ore body, there are three mining options available: single pass, multiple pass, and advancing multiple bench.

SINGLE PASS. In thinner, less than 25 to 30 ft (7.5 to 9 m), regular ore bodies mining can take place in a single first pass. In the oil shales, attempts were made to single-pass mine, carrying benches up to 60 ft (27m) high, but were abandoned due to the high cost of the specialized equipment required (Hustrulid, Holmberg, and Pesce, 1984).

MULTIPLE PASS SYSTEM. The multiple-pass system is suitable for irregular ore bodies over 30 ft (9m) thick, such as some of the lead-zinc deposits of Missouri and some underground limestone mines where the face would be too high to be mined in a single full-face slice. In the multiple-pass system, the first pass is also used in part for definition of the ore body. Mining typically starts near the top of the ore body at a convenient mining height. By starting near the top of an irregular ore body, the first pass can be driven on a regular upgrade. The top of the ore body can then be mined out by upwards, or overhand, stoping by operating the drilling and support equipment on a shrinkage pile (Fig. 18.2.4). Under these conditions, holes are drilled horizontally into the brow or overhand bench either by hand or using a jumbo. When mining is complete, the shrinkage piles provide excellent stockpiles, although considerable capital can be tied up in a pile until it is available for processing. Another major advantage to starting at the top is that the back can be scaled and secured with secondary support where necessary while it is within easy reach. The underlying ore is then mined in a series of subsequent slices using either vertical or front benching. Vertical benching requires wagon type drills in addition to drill jumbos. This process is also known as multiple slicing (Fig. 18.2.5). Under these conditions, ore handling and equipment movement can become complex, and some mines resort to underlying haulage levels to simplify haulage and reduce costs.

ADVANCING MULTIPLE BENCH. This system is ideal for thick regular ore bodies such as oil shales, salt mines, and some limestones where the upper and lower boundaries of the ore body are regular and well defined. In these ore bodies, mining commences at the upper ore body contact with the lead pass and is then followed by a succession of benches in echelon (Fig. 18.2.5). Equipment moves between the benches via a series of ramps, and vertical bench blasting can be used in all passes except for the uppermost.

SLABBING AND PILLAR RECOVERY. To improve or optimize extraction, limited pillar robbing, or slabbing, may be permissible in some operations, provided ground control considerations are met. Slabbing is normally performed toward the end of the life of the mine and indeed may be the only option available when total pillar removal is not possible. Slabbing in the Old Lead Belt, under ideal conditions, permitted total extraction to increase from the first pass 85 to 95% (Bullock, 1982). The ore body was sufficiently narrow to permit arching to occur, and the slabbed pillars carried little load. The earliest recorded case of mining litigation involved illegal pillar robbing in the Athenian stope and pillar silver mines at Laurion in Greece. In 383 BC, a mine owner named Diphilios was sentenced to death and his fortune confiscated for directing his slaves to illegally rob pillars for quick profit (Rickard, 1974).

Pillar removal generally requires exceptionally good ground conditions. With low mining heights, caving that follows pillar removal will normally swell and support the main roof. With higher mining heights this is not possible; therefore either the roof must be self-supporting during pillar removal, or fill must be placed to support the roof. Whatever the situation, a number of ingenious systems have been developed for pillar removal, which can be particularly difficult in the case of very high pillars. Pillars are typically shot from the top down, although there are cases on record of mining commencing at the bottom when it was thought the pillar would separate cleanly from the back (Bullock, 1982; Reed, 1959).

18.2.2.3 Loading and Haulage

Trackless front-end loaders (FELs) and load-haul-dumps (LHDs) are normally selected for loading the sizes and capacities of which depend upon truck capacity, stope height, and size of the largest ore fragments. These units are used in 75% of hard-rock mines (Bullock, 1982). Slushers and gathering arm loaders are also used in soft rock. Boom-type shovels are used in some operations but require a large amount of clearance and are rarely the first choice of modern operators. To facilitate loading, sec-

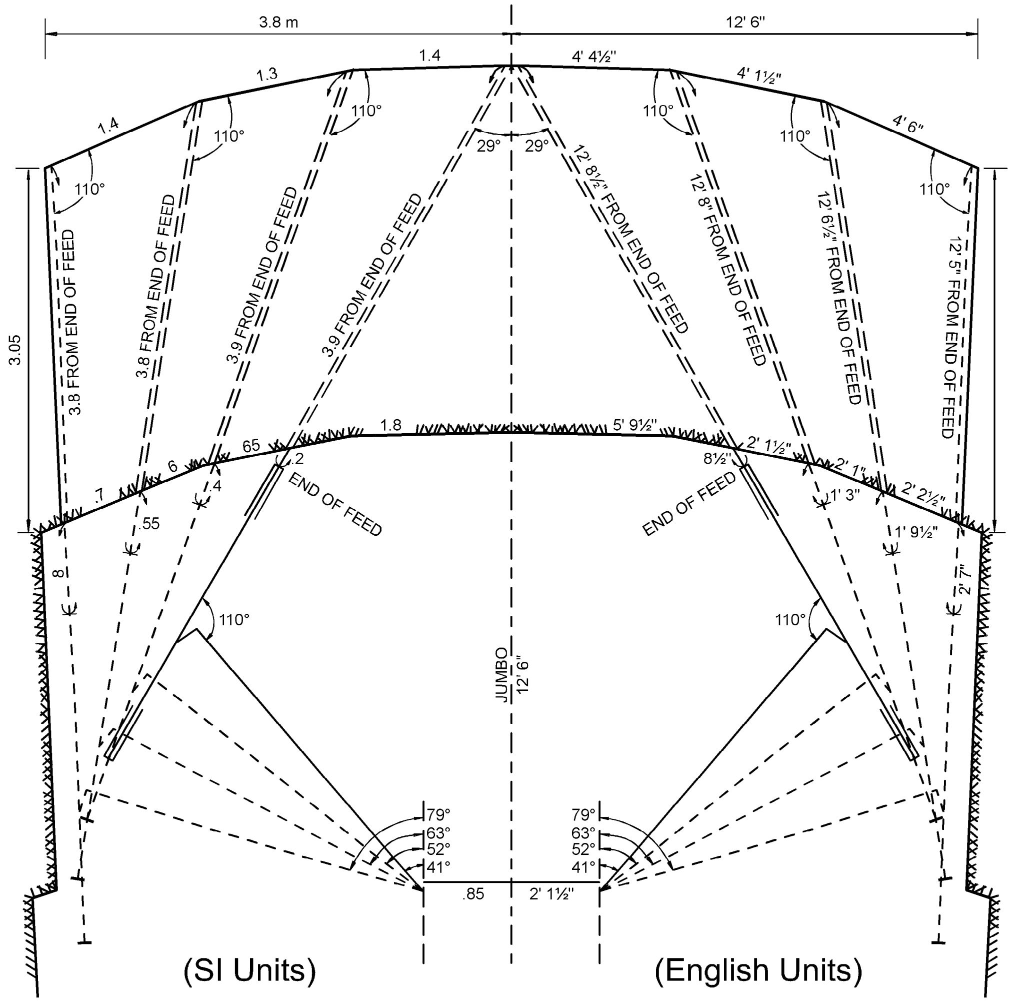


Fig. 18.2.2. Typical V cut as used in dolomites or limestones (Bullock, 1982).

ondary breakage is carried out where essential by blasting, impact hammer, or drop ball.

Since most stope and pillar mines are of large horizontal extent, ore must be transported considerable distances to the adit or shaft. Conventional diesel-powered trucks up to 100 ton (90 t) in capacity are used, the precise size depending on mining height and clearances. Truck haulage is used more often than rail haulage due to its flexibility and versatility. Rail haulage can be used where gradients can be guaranteed suitable. Belt haulage is also used under some conditions, particularly in soft ore bodies such as halite or trona where small fragments are produced, otherwise primary crushers must be installed near the faces. Part of the haulage cycle can be carried out using LHDs or slushers, but these are generally auxiliary to truck or rail haulage.

Selection of equipment combinations for a particular mine is necessarily site specific. Numerous computer programs now exist to facilitate such selection (Klemme and Mousset-Jones, 1984; Zambas and Yegulalp, 1973).

18.2.2.4 Ground Control

Specific design criteria for roof spans, pillar dimensions, support requirements, and floor stability are covered elsewhere in this *Handbook* (see Chapter 10.5). Support is typically installed outside of the normal production cycle as needed. Secondary ground control normally consists of roof bolts for both back and pillar control. Bolt lengths and types depend on local ground conditions and many operations have no artificial support except in the area of portals. Cherry pickers are used for scaling and barring down, which is done immediately after blasting and is an integral part of the production cycle. Less routine scaling may be performed periodically along roadways if ground conditions require. Under multiple-horizon mining conditions, columnization of the pillars is the most widely accepted ground control practice. This involves careful control of pillar sizing and location and is another practice that dates back to the silver miners at Laurium about 400 BC.

Where pillars have been excessively robbed, are inherently weak, or weakened by blasting, stabilization has been facilitated by pillar wrapping using wire ropes or encasing the pillar in shotcrete (Wykoff, 1950). These systems are designed to increase horizontal stresses in the pillar and thereby increase their resistance to shear failure. Horizontal bolting, sometimes with wire mesh, has also proved beneficial. Unless the panel width is small enough to permit arching, failure of under-designed pillars can throw additional load on surrounding pillars. This can in turn cause these pillars to fail, producing a domino effect, sometimes with disastrous results for the entire mine.

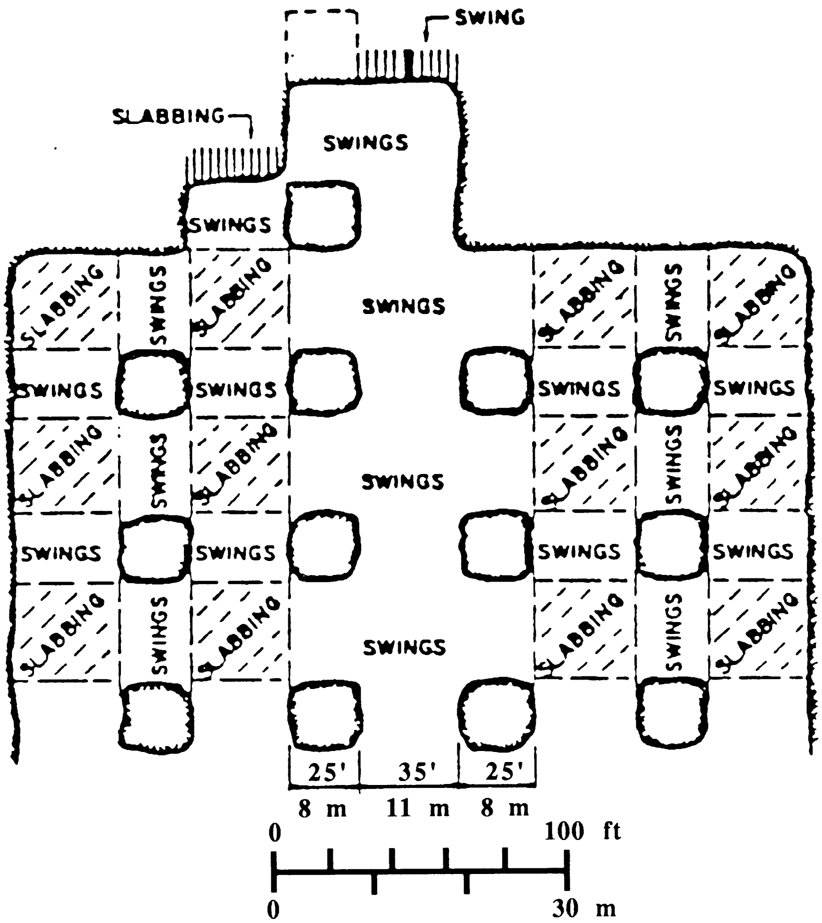


Fig. 18.2.3. Slabbing rounds driven off burn cut heading rounds (Bullock, 1982).

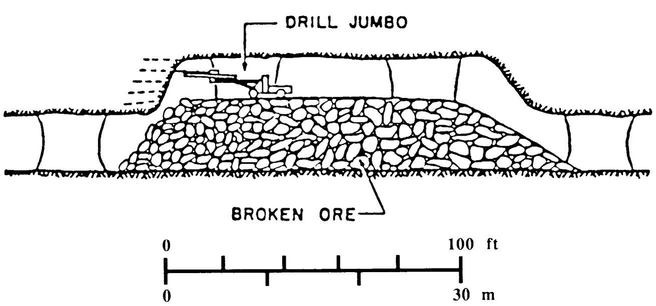


Fig. 18.2.4. Overhand stope and pillar mining (Bullock, 1982).

18.2.2.5 Ventilation

Ventilation is generally fair to good in this stoping method as the mines rely on their large volume to dilute exhaust gases and dust. If methane or toxic gases are present, the mine is subject to strict coal mine regulations that guarantee adequate airflow. Mines typically have dedicated exhaust and intake airways, but control of airflow underground is usually difficult due to large areas between pillars. Stoppings are rarely used, although brattice curtains and ventilation doors are employed only under exceptional conditions. Airflow near the face is normally effected using booster fans or air ejectors (Hartman, 1987).

18.2.2.6 Water

Water problems can be troublesome in some tabular operations as mining follows the ore body and openings cannot always be driven upgrade. Even small or shallow synclines or dips in the ore can serve to collect water, which then must be continually pumped away from the face. Portable compressed air or electritally powered pumps are used, and low points in the mine are selected to function as sumps with more permanent pumping arrangements. These pumps are frequently floating to combat significant variations in water level as water flow cannot readily be controlled in this stoping system.

18.2.3 DIPPING ORE BODIES

Conventional stope and pillar mining becomes impossible as the dip of an ore body increases beyond about 20% grade since equipment cannot travel updip. Numerous variations of stope and pillar have been developed to meet the needs of dipping ore bodies; for instance, the bar pillar system (Christiansen and Scott, 1975), and Elliot Lake stope and pillar (Hedley and Grant, 1972). After review of the numerous variations, two basic versions of stope and pillar can be identified for ore bodies dipping up to 30°, one suitable for rail haulage and the other for conventional trackless mining (Hamrin, 1980).

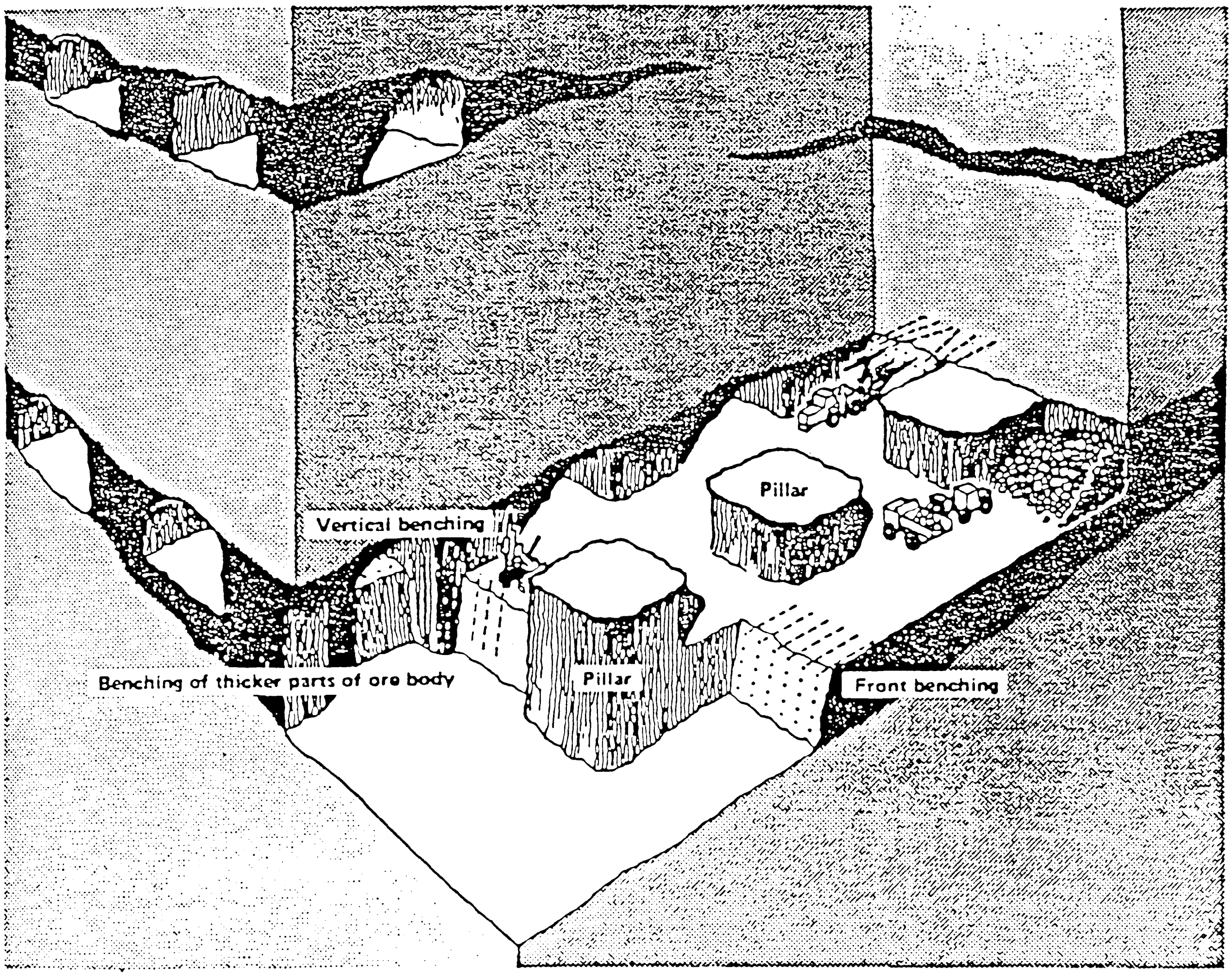


Fig. 18.2.5. Stope and pillar mining using vertical and front benching. (Hamrin, 1980. By permission: Atlas Copco.)

18.2.3.1 Step Stope and Pillar

Step stope and pillar mining utilizes trackless haulage with access drifts running transversely across the dip at an angle suitable for the equipment (Fig. 18.2.6). Ore extraction is made from a series of stope drifts that run horizontally following the strike of the ore body working from the top down. Pillars are sufficiently narrow updip to allow drilling and loading equipment to operate and extract broken ore. Stopes are cut successively downdip, each stope slice having an approximately horizontal floor and being stepped in the middle to the second half of the stope. Crosscuts are also mined with horizontal floors for passage of equipment. This results in the footwall being stepped downdip except where it is cut by equipment roadways.

18.2.3.2 Inclined Stope and Pillar

Inclined stope and pillar can operate efficiently at angles up to 30°. This method can utilize rail haulage, which is limited to near-horizontal grades. Development starts by driving a series of haulage drifts along the footwall following the strike of the ore body. Spacing of the haulage drifts is selected to allow operation of a single slusher to pull all the ore from immediately below the upper haulage level to the level below. In practice, this is typically up to 500 ft (150 m), although production is severely curtailed with longer scrapes. Stoping commences by mining updip from a haulage level using hand drills or jumbos where possible and then scraping the ore down dip into mine cars (Fig. 18.2.7). Scraper cutouts are made on the downdip side of the haulage drifts to accommodate slusher winches. At higher dips, the method is particularly labor intensive and therefore has low stope efficiency.

18.2.4 BREAST STOPING

Breast stoping is a unique variation in stope and pillar mining in that 100% of the ore in the stope is extracted, and no natural pillars are left for support. Instead, artificial pillars are left for roof control with the understanding that stope closure will ultimately approach 100%. While numerous variations of this method exist, it probably finds its most classic usage in the South African gold mines. Other mines, such as the Consolidated Denison, Canada (McCutcheon and Gutterer, 1960), the Conglomerate Mine, Michigan (Vivian, 1931), and the Radon mine in Utah (Love and Knoerr, 1956) have used the method. Breast stoping is confined to flat tabular ore bodies, dipping less than the angle of repose. Typically, the mining height is less than 8 ft

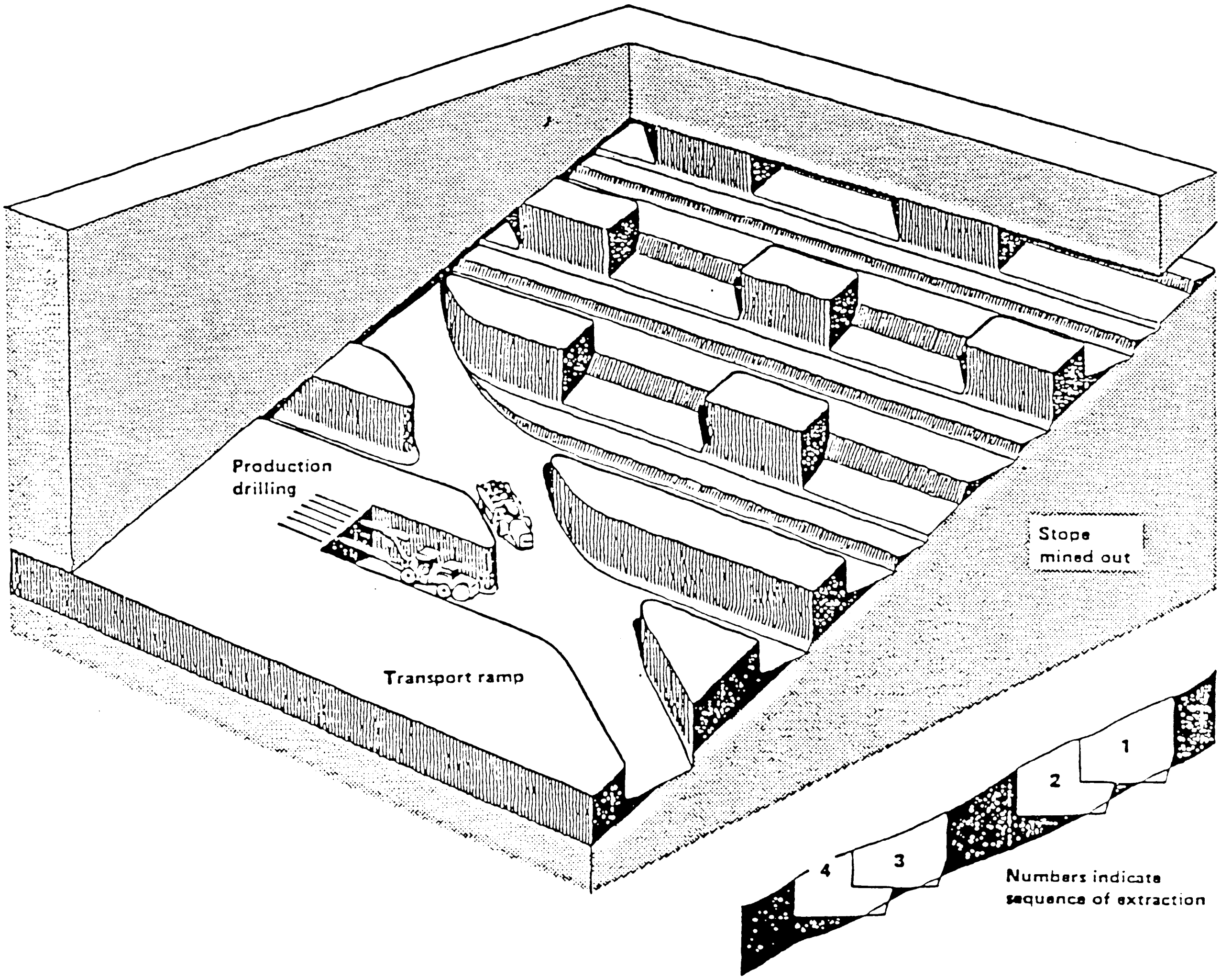


Fig. 18.2.6. Step stope and pillar in an inclined ore body. (Hamrin, 1980. By permission: Atlas Copco.)

(2.4 m) to permit controlled stope closure without caving of the back.

The choice of breast stoping vs. conventional stope and and pillar mining depends on (Haycocks, 1973):

1. The depth of the ore body in relation to the strength of the ore. Under very deep conditions, extraction ratios would be so low with stope and pillar systems as to be uneconomical.

2. Ground conditions sufficiently poor so as to make stope and pillar mining exceptionally hazardous due to back failures and rock bursts.

3. High cost of ground control under deep and poor ground conditions.

4. The deposit is not amenable to longwall mining as practiced in coal mines.

These conditions are all met in the South African gold mines where high stresses due to the great depth and the strong rock burst-prone ore make leaving conventional pillars impractical.

Ore is generally drilled using hand-held jackhammers. Four-ft (1.2-m) holes are placed perpendicular to the face and blasted electrically. The muck is then cross-scraped down the face to stope drifts where it may either be cross-scraped again to ore passes or deposited directly into rail cars. In soft nonabrasive ores, flight conveyors may be used at the face. Immediate face support is achieved using friction or hydraulic props that also support lagging boards placed to control fly rock. A variety of layouts has been devised to meet the needs of individual mines, such as the dip-gully method, the track-and-gully method, and the herringbone system (Fig. 18.2.8). Faces are carried as straight as possible where bursting occurs to reduce the incidence and may be up to 4000 ft (1200 m) long (Fig. 18.2.9). Packs are made of a variety of materials, but timber cribs filled with waste rock are typical. No effort is made to support the overburden, but rather to control closure and minimize stress buildup at the face. As with all stope and pillar methods, ventilation can present serious problems. Unless lagging or pack walls are built behind the face, booster fans are the only real way to provide air movement along the face.

Stope efficiencies in breast stoping are very low, as the method is invariably labor intensive. Average figures for the South African gold mines run 1 to 5 tons (0.9 to 4.5 t)/underground employee-shift (Anon., 1963). The Radon mine in Utah achieved 7.6 tons (6.8 t)/underground employee-shift, which is excellent for this stoping system.

The method is seriously disrupted by faults in the ore; but under many conditions, roof control is excellent, and almost 100% of the ore is extracted. Resuing is also possible with breast

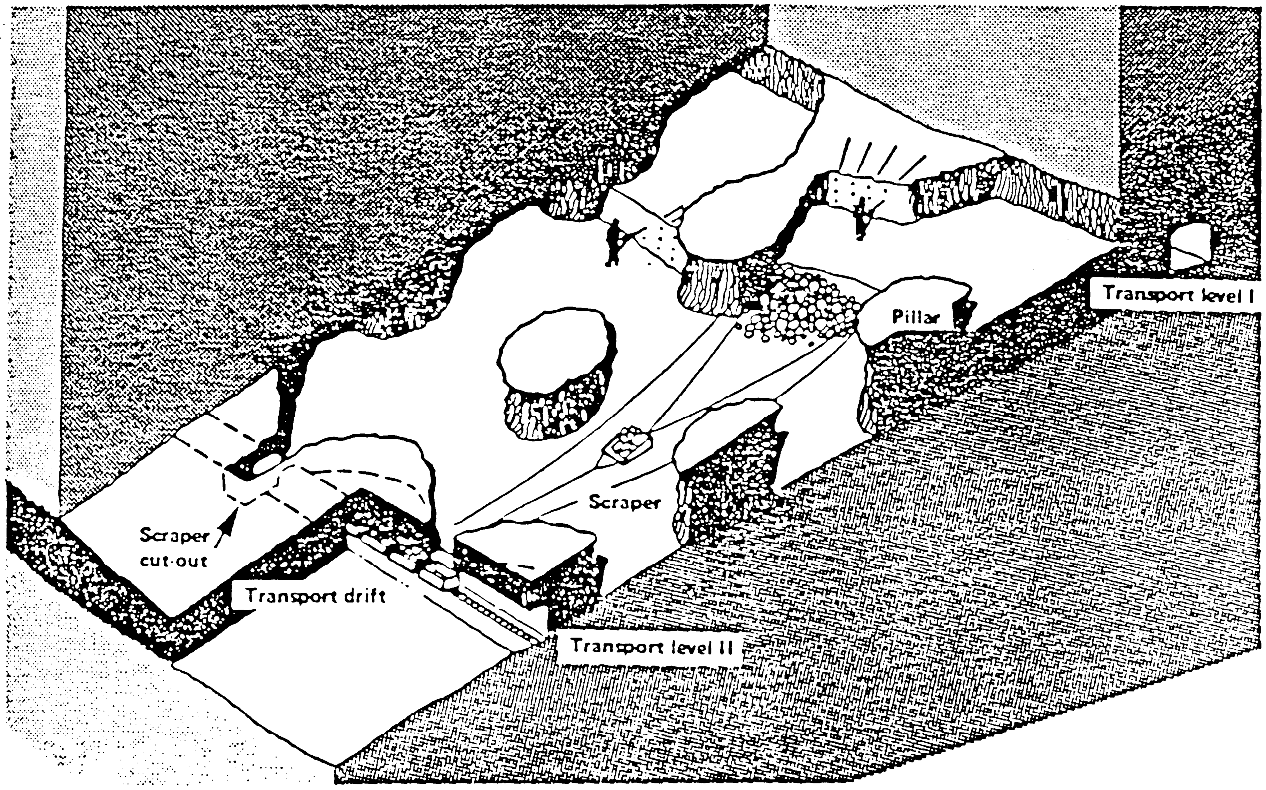


Fig. 18.2.7. Inclined stope and pillar operation in inclined ore using slushers and hand-held drills. (Hamrin, 1980. By permission: Atlas Copco.)

stoping, which can significantly improve the economics of some ore bodies.

18.2.5 ADVANTAGES AND DISADVANTAGES

18.2.5.1 Advantages

1. Most systems can be equipment intensive and are available for computerized optimization in the selection and utilization of equipment and therefore exhibit fairly high stoping efficiencies.
2. The method is somewhat selective. Lean ore can be left in pillars, or the area worked around.
3. Since no development is required with most applications of the method, production can start virtually immediately.
4. Multiple working places can be operated, and therefore production rates can be high if desired.
5. Most applications of the method lend themselves to mobile, self-contained trackless haulage and equipment with its high productivity potential.
6. The method is not labor intensive but may require extensive skills in equipment operation and maintenance.
7. Production rates vary from 50 to 70 tons (45 to 63 t)/employee-shift.
8. Dilution is low under most mining conditions, unless it is necessary to take a parting or roof or floor material for ground control reasons.
9. Surface subsidence should not be a factor in this method, if the mine is adequately designed.

18.2.5.2 Disadvantages

1. Up to 30% or more of the ore may be left in the form of pillars, which are usually not recoverable.
2. Stope ventilation is frequently difficult.
3. The openings can require continual maintenance when ground control conditions are minimal.
4. Water problems can create major difficulties when mining down dip, particularly in undulating deposits.

18.2.6 CASE STUDY: BUICK MINE

The Buick mine is an underground stope and pillar operation located in the Missouri Lead Belt in Iron County, some 120 miles (200 km) southwest of St. Louis (Anon., 1990; Osborne, 1990). The ore body was discovered in 1960, and the mine has been in operation since 1969. Production is approximately 5100 tons (4640 t)/day, from two shifts, with an overall efficiency of 75 tons (68 t)/employee-shift (includes underground mine and maintenance crews). The ore body has remaining reserves of some 17.8 million tons (16.2 M t) containing 5.7% lead, 1.3% zinc, and 0.08% copper; and it averages 14 ft (4.3 m) in thickness with an area of 14.5 million ft^2 (1.34 M m^2) and dips 2° to the north.

18.2.6.1 Geology

The ore body lies in the 42-mile (68-km) long Viburnum trend of the southeast Missouri lead-zinc belt. Mines along this belt account for over 90% of US lead production. Actual miner-

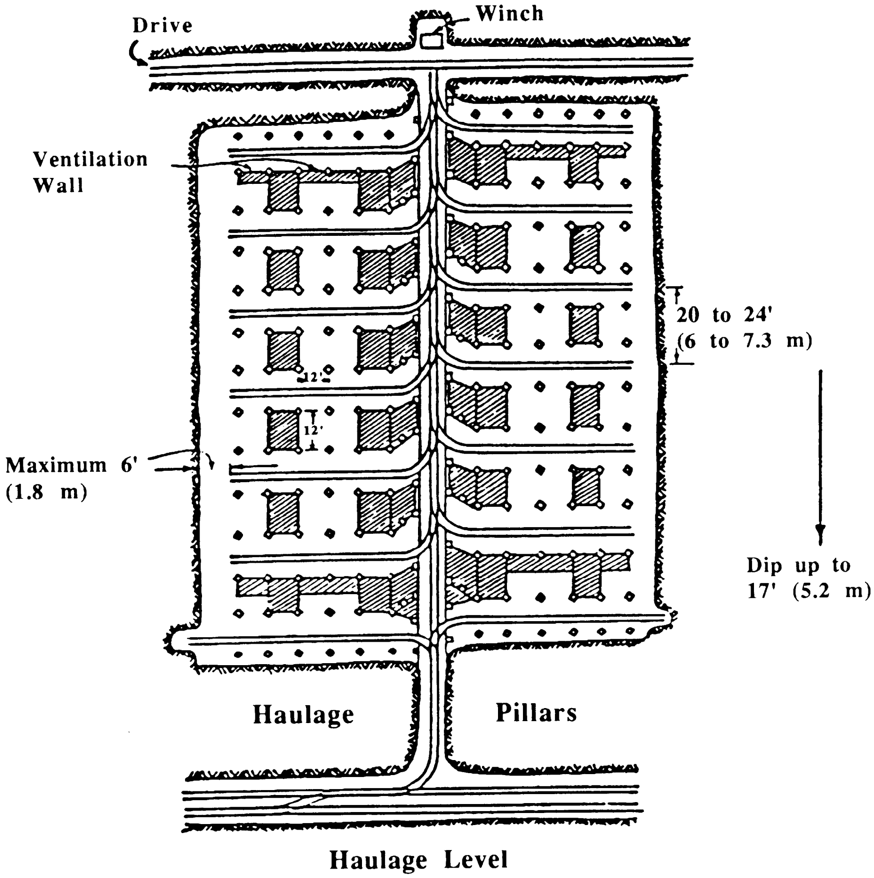


Fig. 18.2.8. Herringbone stope. Beck et al., 1961. (By permission: South African Institute of Mining and Metallurgy.)

alization is located in the upper 100 ft (30 m) of the Bon Terre formation, consisting of dolimitized sandstones and mudstones overlain by shales. Two distinct types of ore bodies are mined in the Buick mine, the breccia and the blanket. The breccia ore zones are up to 100 ft (30 m) in thickness, 250 ft (75 m) wide, and contain three mineralized zones. The blanket ore body is in the same stratigraphic position as the breccia but is only 8 to 22 ft (2.4 to 6.7 m) thick; it is, however, up to 1500 ft (455 m) wide and 5000 ft (1525 m) long (Fig. 18.2.10).

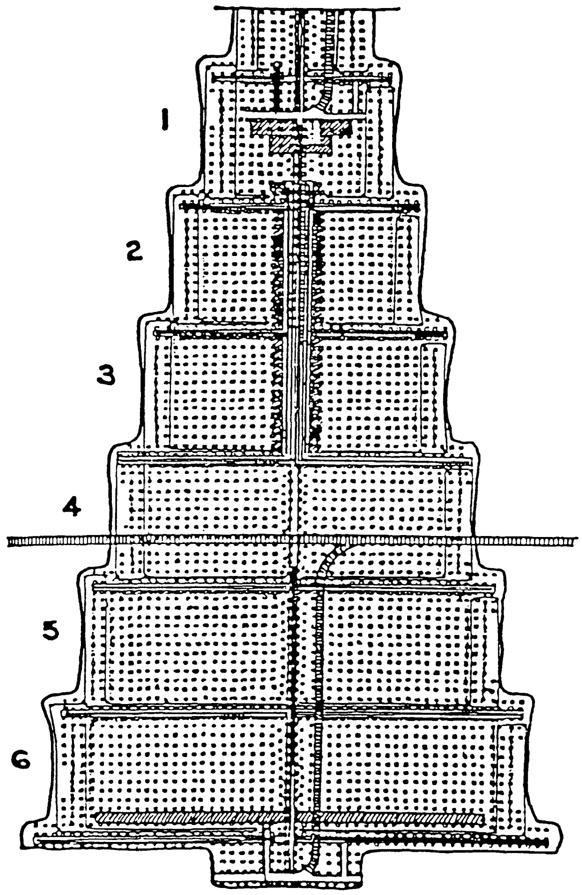
18.2.6.2 Mining Methods

The ore is mined in a series of passes, depending on the thickness of the ore body. The blanket ore body varies from 10 to 22 ft (3 to 6.7 m) in thickness and can be mined in a single or "first pass." First-pass mining accounts for 80% of the mine's production and enables the ore bodies to be fully defined. Openings and crosscuts are 32 ft (10 m) wide, and all pillars are a minimum of 28 by 28 ft (8.5 by 8.5 m). Stoping is carried out using 75-hole breast rounds, with 12-ft (3.6-m) 1-3/4-in. (41.3-mm) diameter holes. These are drilled with either Joy or Atlas Copco two-boom hydraulic jumbos equipped with electric motors and diesel tramming. Blasting is conducted using either an ammonium nitrate-fuel oil mix (ANFO) primed with Trojan stingers and 60% gelatin dynamite in the burn, bottom, and wet holes. Initiation is carried out using non-electric L-P series caps, 15-grain primacord, and one electric cap. Each blast produces 500 to 550 tons (455 to 500 t) and is shot at the end of each shift. Secondary blasting is usually not a problem, but a portable rock pick is available. The primary crusher is located underground and is a 60-in. (1.5-m) gyratory.

The thicker breccia ore body, which varies from 8 to 100 ft (2.4 to 30 m) in thickness, is mined in a series of passes (Fig. 18.2.11). A first pass driven near the top defines the ore body and, when complete, preparation can then be made to remove the remainder of the ore. Since the first pass is normally driven below the top of the ore body to ensure it stays in ore, the back is mined first. An up-hole cut is drilled and blasted with the broken ore providing footing for subsequent mining. Subsequent mining is done by drilling and blasting the breast. As many passes as needed are carried out until the top of the ore body is reached. After removal of the blasted ore, the third pass is carried out by blasting benches which have been drilled horizontally with 1-3/4-in. (44.5-mm) holes. In various parts of the mine, horizontal thicknesses of no less than 15 ft (4.6 m) are left between pillars to give added strength to the pillars. Benching using vertical holes was abandoned as the blasts caused excessive pillar damage. At the Buick mine, approximately 10% of production comes from third-pass mining, while all three passes extract 78% of the ore. When projected plans to recover high-grade pillars are carried out, total recovery is expected to be about 90%.

18.2.6.3. Loading and Hauling

Loading is carried out using two Caterpillar (Cat) 980C and Wagner ST8 8-yd^3 (6-m^3) FELs. The 980C loaders do most of the truck loading, with the ST8s used in headings less than 14 ft (4.3 m) in height. The truck fleet consists of two 50-ton (45.5-t) Kiruna beds or two 50-ton (45.5-t) CTC Lo-Pro wagons powered by Cat 631-E diesel tractors, or a 40-ton (36.4-t) Athey wagon powered by a Cat 631D diesel tractor. All rubber-tired vehicles are serviced and repaired underground in a shop facility,



Dip 25°; Length of longwall 900 ft (275 m); Stoping width 39 in. (1 m)

Fig. 18.2.9. Longwall breast stoping operation. (Beck et al., 1961. By permission: South African Institute of Mining and Metallurgy.)

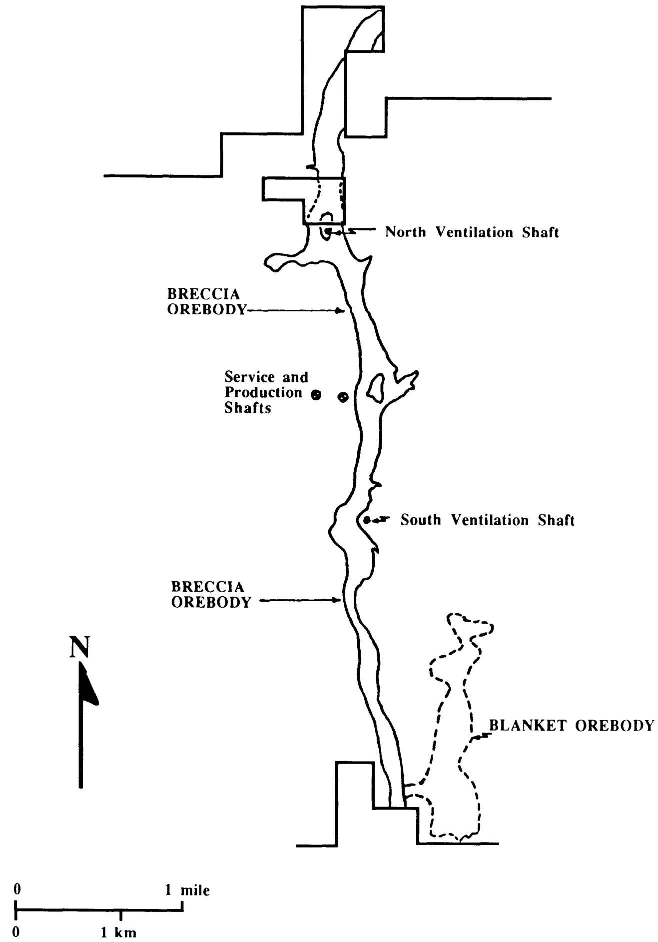


Fig. 18.2.10. Buick mine ore body (Anon., 1990).

with full spares available. The truck fleet averages 13,000 mi (21,000 km)/month. Trucks travel on specific haul roads and feed a 3000-ton (2700-t) central orepass. The orepass feeds the primary crusher, which reduces the ore to –6 in. (150 mm). The ore is then carried by conveyor to one of three 1000-ton (910-t) ore storage bins that feed the 12-ton (11-t) automatic skips. Shaft capacity is 400 tph (365 t/h) from high-speed friction hoists.

18.2.6.4 Drainage and Ventilation

Water flows by gravity through 6-in. (150-mm) drainage holes to the bottom level and then to the service shaft where it is pumped to the surface. The mine has six 500-hp (375-kW) centrifugal pumps, and the water is either used by the mill or directed to the tailings pond.

Ventilation needs are met by pulling 800,000 cfm (283 m^3/s) through centrally placed production and service shafts, then through the production level, down raises to the haulage level, and out through two ventilation shafts located to the north and south. The 12-ft (3.6-m) diameter south shaft has an 800-hp (597-kW) surface fan and the 8-ft (2.4-m) north fan a 500-hp (375-kW) surface fan. To facilitate airflow underground, eight 50-hp (38-kW) and 75-hp (58-kW) booster fans are used.

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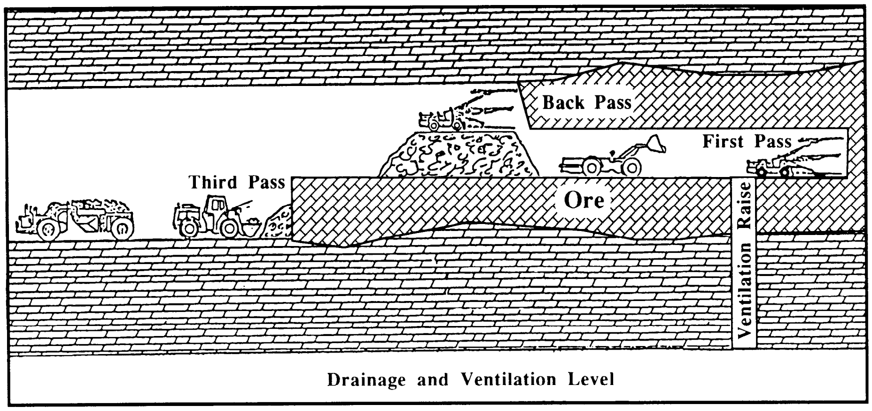

Fig. 18.2.11. Multiple-pass mining in the breccia ore body (Anon., 1990).

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Chapter 18.3
SHRINKAGE STOPING

JACK HAPTONSTALL

18.3.1 INTRODUCTION

Shrinkage stoping is a vertical, overhand mining method whereby most of the broken ore remains in the stope to form a working floor for the miners. Another reason for leaving the broken ore in the stope is to provide additional wall support until the stope is completed and ready for drawdown.

Stopes are mined upward in horizontal slices. Normally, about 35% of the ore derived from the stope cuts (the swell) can be drawn off ("shrunk") as mining progresses. As a consequence, no revenues can be obtained from the ore remaining in the stope until it is finally extracted and processed for its mineral values.

The method is labor intensive and cannot be readily mechanized. It is usually applied to ore bodies on narrow veins or ore bodies where other methods cannot be used or might be impractical or uneconomical. The method can be easily applied to ore zones as narrow as 4 ft (1.2 m), but can also be successfully used in ore widths up to 100 ft (30 m).

Logically, the broken ore should be free flowing and not pack in the stope. Neither the ore nor adjacent country rock should contain undue amounts of clay or other sticky material to cause the ore to hang together in the stope and either be difficult or impossible to draw. Additionally, the ore should not readily oxidize, which may cause the broken pile to re-cement itself, and consequently "hang up." Oxidation can also have an adverse effect on ore dressing. Ore should be fairly continuous along the strike of the vein or ore body in order to avoid mining extensive amounts of waste as dilution from the stope back. However, small waste areas may be mined around and left as random pillars.

Consideration must also be given to the plunge or rake of the ore body, especially where the entire ore body may be mined as a single stope (Fig. 18.3.1 rather than as pre-established stope panels with defined vertical end lines. A stope with a shallow plunge or rake (< 50") may be very difficult to mine by shrinkage methods because the ore moves away too quickly from the pre-developed extraction system (Fig. 18.3.2). Additionally, stopes where ore abruptly extends for great distances beyond stope end lines are also difficult to mine because they often require much additional development work to the stope extraction system (Fig. 18.3.3), especially raising.

18.3.2 DEVELOPMENT AND PREPARATION

Sites for shrinkage stoping are generally developed by drifting in the vein or ore zone on two levels, spaced vertically 100 to 600 ft (30 to 180 m) apart. After a viable ore body has been established, the next phase consists of driving one or more raises to establish vertical ore continuity and also to provide ventilation and access to the stope (Fig. 18.3.1)

Raises may be driven conventionally, with Alimak-type raise climbers, or by raise boring machines. Drifting for shrinkage development is normally done by conventional drill-and-blast, track or trackless methods.

Stopes may be prepared with extraction raises on 25- to 30-ft (7.5- to 9-m) centers over the length of the ore shoot; each raise is fitted with a chute, normally of timber construction. Extraction raises are belled out and "hogged over" as the undercut for the start of the first stope cut. This type of preparation is still used but on a very limited basis.

Another method of preparing a stope is to blast down at least two backs of the ore zone, clean up the broken ore, and install stull timbers or timber sets in the drift below the stope. Timber chutes, or even "chinaman" chutes, are installed at approximately 25-ft (7.5-m) intervals as part of the timbering.

A more common method of preparing stopes in modern operations is to drive an extraction drift parallel to the ore body development drift, about 25 to 50 ft (7.5 to 15 m) in the footwall of the ore body. Subsequently, drawhole extraction crosscuts are driven from the footwall drift into the ore drift on 25- to 50-ft (7.5- to 15-m) centers. The back of the ore body is then blasted

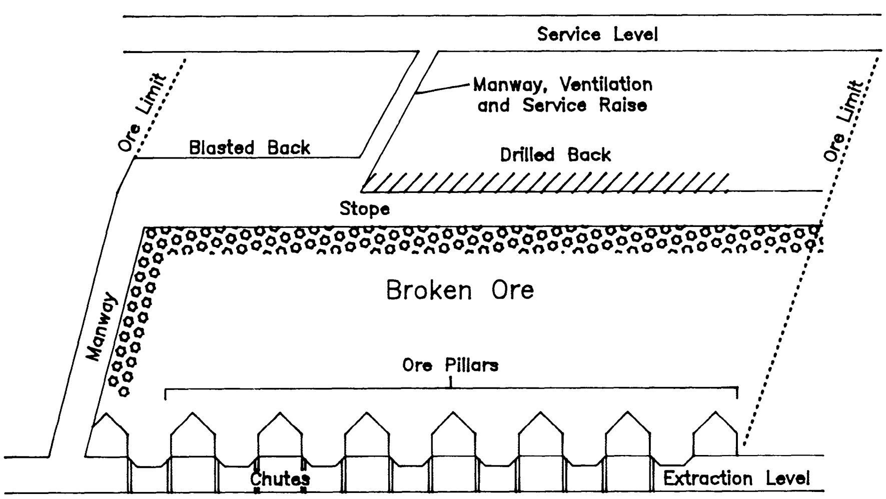

Fig. 18.3.1. Longitudinal section–typical shrinkage stope.

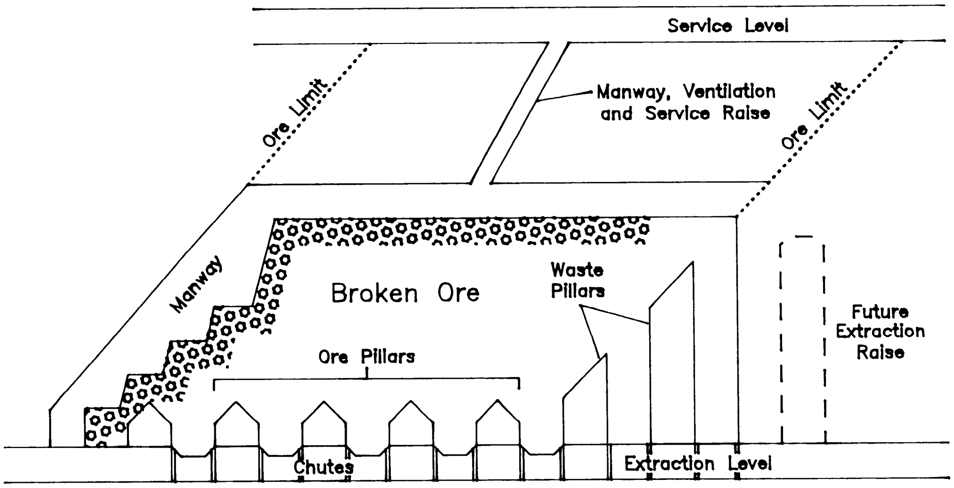

Fig. 18.3.2. Longitudinal section—shrinkage on shallow-raking ore body.

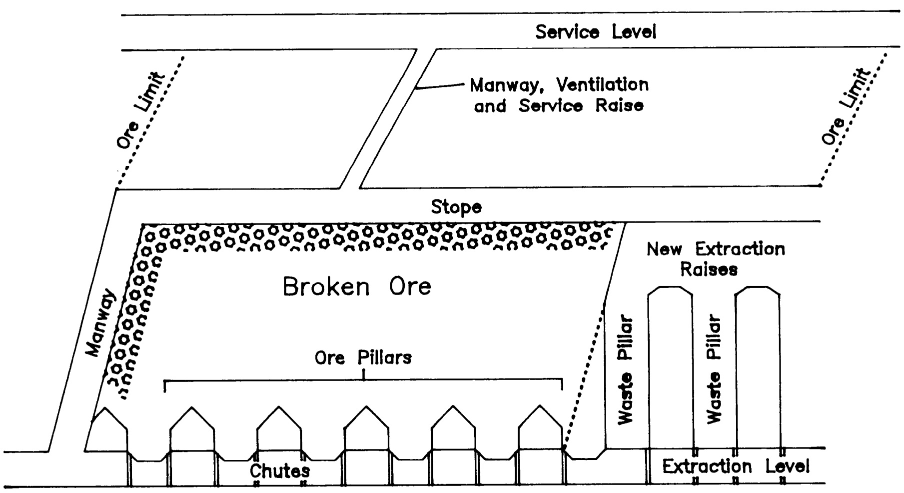

Fig. 18.3.3. Longitudinal section—shrinkage on irregular ore body.

down, and the swell is extracted through the drawholes, either with rail-mounted mucking machines or load-haul-dumps (LHDs) (Fig. 18.3.4).

18.3.3 STOPING OPERATIONS

Once a shrinkage stope has been established, manways are usually installed in the raise from the next level. A manway and service way is normally constructed on one or both end-panels of the stope. Often a timber slide is installed in one of the manways for hoisting and lowering materials into and out of the stope; hoisting is often accomplished with a single-drum air hoist installed in the level below the manway. Once the manways, ventilation raises, and service ways have been established for a stope, mining can commence.

Drilling of a shrinkage stope back is accomplished with hand-held stopers or jacklegs, although mechanized drill wagons or stope jumbos may be used in wider stopes. Back stoping is the normal mode of operation, but breasting down is also common. Up-holes are generally 1.8 to 2.4 m (6 to 8 ft) in length. In most cases, all holes are loaded and a complete back is blasted at a time. Breasts are drilled with a 8- to 10-ft (2.4- to 3-m) horizontal holes and normally blasted once per shift.

Holes are loaded with ANFO products or water gels and even with slurry blasting agents. Initiation is commonly with non-electric methods, but electric blasting is also practiced.

After a cut has been blasted in a stope, drawdown of the 35% swell is necessary, after which the muck pile must be leveled to facilitate drilling of the next cut. Leveling of the pile can be done by hand shovels in the case of small stope, with 2- or 3-drum slushers in larger or longer stopes, and even with LHDs in large stopes. After leveling, drilling of the next stope cut, raising of the manways, and so forth are done to continue the mining cycle.

Variations for the establishment of openings for manways, ventilation raises, or service ways may include installation of strategically placed timber cribbed openings, steel culverts or rings, or timber sets within the broken ore area. These installations may be very desirable during the mining phase, but may create safety problems and nuisances with the collapse of the materials used to construct these openings. Pinning, stulling, or wedging these installations to the stope walls may prevent their destruction during drawdown; materials from a destroyed manway may be drawn down with the broken ore into the chutes or drawholes and cause hangups.

A stope should have strong, self-supporting walls to permit the application of shrinkage stoping. Dilution through scaling of

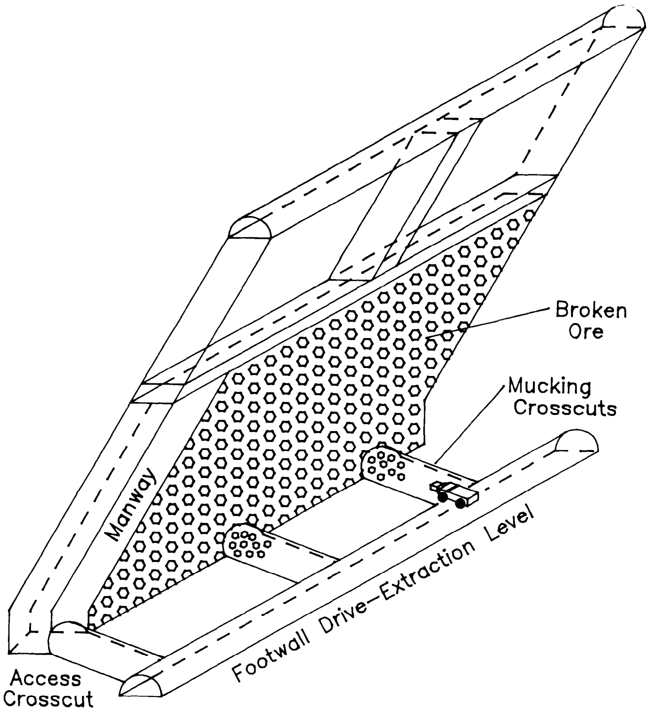

Fig. 18.3.4. Typical shrinkage stope with LHD extraction.

walls can preclude use of the method. Good mining practice coupled with state and federal regulations may dictate a least a minimum ground support program.

Wall and back support may be accomplished by leaving random or even systematic pillars. Pillars left in ore zones may be drilled off and blasted upon drawdown of the stope. Traditionally, timber stulls fitted with plank headboards have been installed to support suspicious slabs or areas of bad ground of stope walls. Horizontal stulls and cribbing are also used to support loose areas of stope backs; however, the timbers may be subsequently buried in the muck pile upon shooting the next stope back, and may become a hazard or, at the very least, a nuisance, upon drawdown of the stope. Rock bolting has evolved into the preferred mode of wall and back support. Mechanical as well as grouted types of bolts are used. Correct installation of bolts in the walls of narrow shrinkage stopes may be difficult because of the lack of room to drill the bolt hole perpendicular to the stope wall as well as to install the required length of rock bolt.

Sampling of narrow shrinkage stope backs is usually done by either taking a channel or chip sample by hand or through mechanical means. Sampling is usually done at a systematic interval (say, 5 ft or 1.5 m) along the entire back, ends, and in some cases, the ribs of the stope after every stope cut. In wider stopes, drill sampling of the back and ribs can be done. The drill sample may criss-cross the stope back on a predetermined pattern. Drill cutting samples are collected in a sample bag through a hose and funnel or other device.

18.3.4 STOPE DRAWDOWN

One of the most dangerous jobs in a mine is the drawdown of shrinkage stopes, especially where the ore contains sticky material to hang up between the stope walls. Hung-up stopes must either be washed down with water, bombed down with explosives, picked down by miners (a practice not recommended), abandoned, or re-mined. In any case, a hung-up stope is a costly and dangerous problem, and shrinkage stoping should not generally be used where the ore has a tendency to hang up.

Stopes should usually be drawn down systematically, drawing the pile evenly so if the stope walls do peel or slough, the waste remains atop the pile and does not trap broken ore rilled above the pile. Once a stope drawdown is started, the operator's control over the walls, pillar recovery, etc., is minimal and in most cases, the re-entry of miners into a stope under active drawdown would be considered too great a safety hazard to risk.

Stopes can be drawn down from strategically placed chutes or from drawpoint crosscuts. Haulage from the stope extraction points may be done with rail equipment or LHDs and/or trucks. Chutes should be robustly designed and constructed to avoid destroying them through blasting of large slabs in them. Stopes may also be extracted through slusher trenches developed below the stope.

18.3.5 VARIATIONS AND APPLICATIONS

Variations of shrinkage stoping include inclined shrinkage and longhole shrinkage. Recovery of large pillars may be done by shrinkage methods.

One example of mines that employed shrinkage as a primary stoping method is the Homestake mine at Lead, SD. Fifty-foot (15-m) wide "bull pen" shrinkage stopes were developed transversely across the full width of the great Main Ledge ore body. Stopes were mined over a timbered sill where strategic china chutes were constructed for ore extraction. Stopes were mined over the sill for about 70 vertical ft (20 m) to within 30 ft (9 m) of the next level. Twenty-five-ft (7.6-m) wide pillars were left between stopes, which along with the crown pillars, were subsequently extracted with square-set stopes. Homestake abandoned this type of shrinkage stoping just before World War II.

A second example is the Idarado mine located near Ouray and Telluride, CO. Stopes were mined along the veins and the full width of the veins, which varied from 5 to 25 ft (1.5 to 7.6 m). Stope panels were generally 400 ft (122 m) long and were prepared over a slusher trench developed about 20 ft (6 m) over the back of the main level drifts. A series of pockets and raises on 25-ft (7.6-m) centers were developed from the slusher trench and the pockets "hogged over" to form pillars between the trench and the first cut of the stope. Ore was extracted from the stope, slushing from the pockets to a chute in the center of the stope. Stopes were normally mined from level to level or about 200 ft (60 m) along the dip.

A variation of the above was practiced at the Morococha and Casapalca mines of the Cerro de Pasco Corp. located in the central Andes mountains of Peru, South America. Stopes in these mines were prepared over the main development level driving 25-ft (7.6-m) raises on 25-ft (7.6-m) centers and "hogging" out from the raises to form the first stope cut at about 16 ft (5 m) over the level. Each raise was then fitted with a timbered chute for ore extraction.

In all cases, a raise was first developed through each ore block or stope panel for ventilation and service. Manways were either carried as cribbed raises in the stope or, in the case of Idarado, as boreholes 10 ft (3 m) in the footwall of the vein. In the vein mines, drilling was accomplished with either stopers or jacklegs, while at the Homestake, drilling was done with bar-and-column mounted Leyner-type drills.

Some variations of shrinkage stoping include inclined shrinkage, longhole shrinkage, and construction shrinkage.

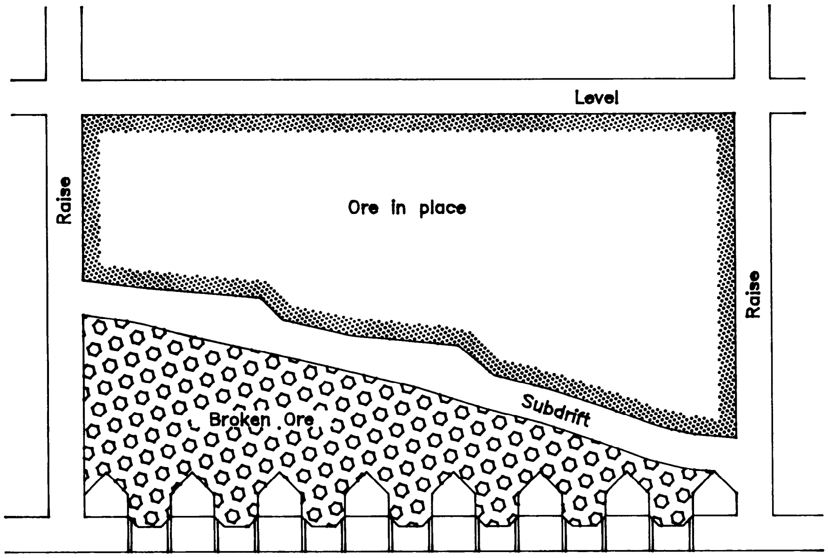

Fig. 18.3.5. Shrinkage stope, Rosiclare, IL.

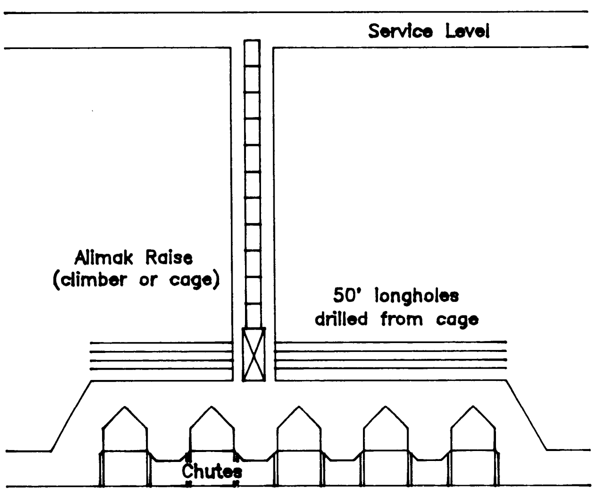

Fig. 18.3.6. Longhole shrinkage stoping.

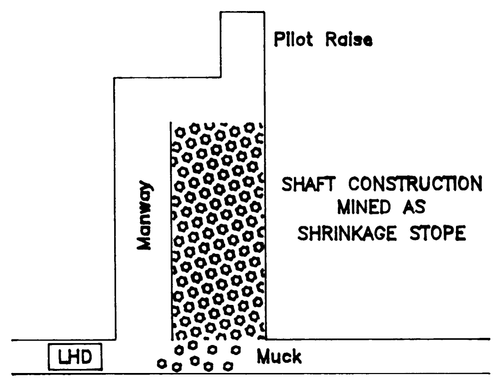

Fig. 18.3.7. Construction shrinkage stoping by conventional method.

Inclined shrinkage refers to a rill stoping adaptation where multiple faces or benches for drilling are carried along the back of the stope as it is mined upward (Fig. 18.3.5). Stopes are developed conventionally over pillars and chutes or over timber sets fitted with chutes on centers of about 25 ft (7.6 m). The advantage of carrying the stope in benches is that multiple faces can be drilled in a given shift where it is desirable to drill the stope with airleg-type drills rather than stopers.

Longhole shrinkage (Fig 18.3.6) is developed conventionally as described previously. The exception is that drilling of the stope is done from vertical raises driven through the ore zone on 50- to 100-ft (15- to 30-m) centers. Raises can be developed with raise climbers or through cage raising techniques. The raise climber or the cage becomes the entry and exit vehicle as well as the platform for drilling and loading. Parallel longholes are drilled along the strike of the ore body and loaded from the raises. Initiation normally is done from a safe area on the service level above the stope.

Shafts, winzes, or large break raises for blasthole or sublevel caving stopes may be developed through shrinkage methods (Fig. 18.3.7). In many cases, this is done as described in the longhole variation. Given a large enough opening, conventional shrinkage stoping of a shaft or raise may be justified.

18.3.6 CASE STUDY: LA LIBERTAD MINE, PUEBLO NUEVO, DURANGO, MEXICO

The small La Libertad mine was brought on-stream in July 1977 by Minas de San Luis, S.A., a 51% Mexican-owned company (Haptonstall, 1980). The mine was developed entirely for shrinkage stoping.

GENERAL. La Libertad was essentially a virgin silver-gold deposit in a very rugged location in the Mexican Sierra Madre mountains. The only previous mining done in the area was a small tonnage extracted from the outcrop of the principal Santa Rosa vein in the 1930s. Production ceased in the mine in 1985 due to political strife in the area. The total investment to bring La Libertad on stream was about $3.5 million (in 1975 dollars).

GEOLOGY. The ore deposit occurred in quartz veins hosted in tertiary rhyolite intrusives and tuffs. The principal oreshoot on the Santa Rosa vein is 1150 ft (350 m) long, 450 ft (145 m) high, and on average 20 ft (6 m) wide. Dip of the vein is 70°W.

ORE RESERVE. Mine commenced with 193,800 tons (176,200 t) averaging 11.5 oz/ton (400 g/t).

MINING METHOD. Shrinkage stoping.

EQUIPMENT. 1-yd^3 (0.8-m^3) LHDs, 2-drum air slushers, stopes and jacklegs, on-highway trucks.

PRODUCTIVITY. 7.7 tons (7.0 t)/employee-shift in stope.

18.3.7 SUMMARY

Under most economic evaluations, the labor intensity of shrinkage stoping precludes its widespread application in modern mining situations. However, it may be the only possible method applicable in the case of a mine in which the ore bodies occur in very narrow veins and cannot be stoped by other methods. Shrinkage may also be used in special situations where small ore blocks cannot be extracted economically any other way or in conjunction with other stoping methods.

18.3.7.1 Parameters

The following is based on Boshkov and Wright (1973), Lucas and Haycocks (1973), Morrison and Russell (1973), and Lyman (1982):

1. Ore characteristics: requires strong ore, non-oxidizing ore, ore that does not pack or stick together, and ore that does not spontaneously combust.
2. Host rock characteristics: requires strong to moderately strong walls.
3. Deposit shape: almost any shape but should have uniform dip and boundaries.
4. Deposit dip: greater than angle of repose (> 45°), and preferably steeper than 60°.
5. Deposit size: narrow to moderate width (3 to 100 ft, or 1 to 30 m); length minimum of 50 ft (15 m) to unlimited panel stopes on long strike lengths.
6. Ore grade: moderate to high.

18.3.7.2 Features

The following is based on Morrison and Russell (1973), Hamrin (1982), and Lyman (1982):

Advantages.

1. Small to moderate production rates.
2. Gravity drawdown of stope.
3. Simple method, especially for small mines.
4. Low capital investment, some mechanization possible.
5. Ground support of ore and walls minimal.
6. Stope development moderate.
7. Good ore recovery (75 to 100%) low dilution (10 to 25%).
8. Reasonable selectivity possible.

Disadvantages.

1. Productivity low to moderate, 3 to 10 tons (2.7 to 9 t)/ employee-shift in stopes.
2. Mining costs moderate to high.
3. Labor intensive, mechanization limited.
4. Difficult working conditions, especially in narrow and/ or short stopes.
5. About 60% of ore tied up in stope until completed.
6. Ore can pack, oxidize, or spontaneously combust in stopes.
7. Risk of loss of stope during drawdown if not properly controlled.

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Chapter 18.4
SUBLEVEL STOPING

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Sublevel stoping, also known as blasthole or longhole stoping, is an open stoping, high-production, bulk mining method applicable to large, steeply dipping, regular ore bodies having competent ore and rock that require little or no support. The method is reported to have originated in the Michigan iron mines in 1902 (Peele, 1941) and was originally devised as a shorthole bench and trail system. Production ranges from 15 to 40 tons (14 to 36 t)/employee-shift, and individual stopes may produce in excess of 25,000 tons (22,700 t)/month. Sublevel stoping currently accounts for some 9% of US and 3% of world noncoal production (Lawrence, 1982). The method is often selected as an alternative to sublevel caving when dilution levels must be kept to a minimum. In the Zambian Copperbelt, dilution was reduced from 30 to 20% by changing from sublevel caving to sublevel stoping, although recovery dropped from 90 to 80% because of difficulties in pillar recovery (Misra, 1983).

Sublevel stoping is very development intensive, although the cost of development is compensated by the fact that much of it is done in ore. It is currently limited to steeply inclined ore bodies where both ore and country rock are competent and broken ore flows under the influence of gravity. In the 1930s, it was used at the Roan Antelope Mine at dips as low as 12 to 30° with slushers to move the ore (Peele, 1941). More recently, the Cascade method, which is a direct modification of sublevel stoping, has been used at dips as low as 25°, also by utilizing scrapers to move the ore (Kelly, 1969). Ore bodies should be regular, because the method is not selective.

Production drilling is accomplished using longhole equipment. New developments in sublevel stoping involve the utilization of large-diameter DTH (down-the-hole) drills that, because of their directional accuracy, are revolutionizing the mining method (Pandey, 1984). The large, highly mechanized drilling equipment used restricts the minimum-width ore body that can be mined, while high development costs associated with sublevel stoping require that a high production rate be maintained. Efficient use of large-scale blasting makes sublevel stoping one of the lowest-cost underground mining methods available. Sublevel stoping also finds widespread application for pillar recovery in cut and fill and other types of mining methods (Irvine, 1982; Bharti, Lebl, and Cornett 1983). A variation of sublevel involves blasting the ore in horizontal instead of vertical slices with long stope holes drilled from raise climbers (Robertson, Vehkala, and Kerr, 1989).

18.4.1 ORE BODY TYPE

The typical ore body required for successful sublevel stoping must be regular, large, strong to fairly strong, and competent, and the wall rock must be self-supporting. Rock strengths vary widely and can be compensated for in the design, but typically range from a minimum of 8000 psi (55 MPa) with no upper limit. The dip of the ore body footwall must be such that it exceeds the angle of repose of the broken ore, which permits gravity flow of blasted ore through to drawpoints and chutes.

Ore bodies are typically a minimum of 20 ft (6 m) wide to afford efficient application of longhole blasting. Ore bodies less than 20 ft (6 m) wide realize a higher cost per ton of ore because of lowered production per blast, and as widths go less than 5 ft (1.5 m), hand drilling and overhand and underhand methods must be resorted to. In wider weaker ore bodies, transverse stoping may be practiced to afford additional stope support (Matikainen, 1981). No upper size limit exists for ore bodies mined using this method. However, in large ore bodies, support pillars must often be left in place during the entire mining cycle on that level. These pillars are usually recovered after adjacent completed stopes have been backfilled (Boshkov and Wright, 1973; Hamrin, 1982).

Longhole drilling and large-volume production blasting require ore bodies that are even and fairly well defined. Stope boundaries must be regular, because irregular ore bodies and those containing large waste inclusions cannot easily be avoided. Waste from irregular ore bodies and inclusions dilutes the ultimate grade of mined ore and thus increases the cost per ton of ore produced. A smooth ore-to-footwall contact allows for easier flow of blasted ore to drawpoints and chutes. Rock must be structurally competent and self-supporting as large openings may be left unfilled for extended periods of time. Additionally, repeated shocks from large blasts require an ore of high compressive strength and minimal structural weaknesses such as joints, faults, and bedding planes.

Failures resulting from collapse of incompetent ground cause excessive dilution, loss of sublevels, and large chunks blocking drawpoints, and necessitate reconditioning of stopes. Small, localized ground failures result in ground movement and displacement, and cracking of blastholes. This in turn makes loading of blastholes difficult and in some cases necessitates extensive redrilling of a block (Morrison and Russell, 1973; Mitchell, 1981; Lawrence, 1982). Sublevel stoping is used to depths of 3000 ft (900 m) (Misra, 1983).

18.4.2 STOPE DEVELOPMENT

18.4.2.1 Stope Layouts

Mine development normally starts from a shaft sunk in the footwall to avoid any subsequent caving effects from the stopes. The ore body is divided vertically by driving crosscuts and haulage levels every 150 to 400 ft (45 to 120 m). Access raises driven in the ore body are used to further subdivide the ore body into blocks for stoping. A collection system is constructed, during which time the stope block is all or partially undercut. Sublevels are driven through the proposed stope block every 30 to 180 ft (10 to 55 m) (Fig. 18.4.1), and more than one sublevel may be used on each level, depending on the width of the ore body. Some typical dimensions for a wide variety of ore bodies are shown in Table 18.4.1.

Stoping is carried out by blasting vertical slices into an expansion slot (Fig. 18.4.2) the height and width of the proposed stope, usually by enlarging a slotting raise with longhole drilling and blasting. Shrinkage stoping has also been used to form the starting slot that may be developed at the end or middle of the stope. Slotting has been shown to be a very expensive part of the stoping operation, accounting for 20 to 30% of total stoping costs (Matikainen, 1981).

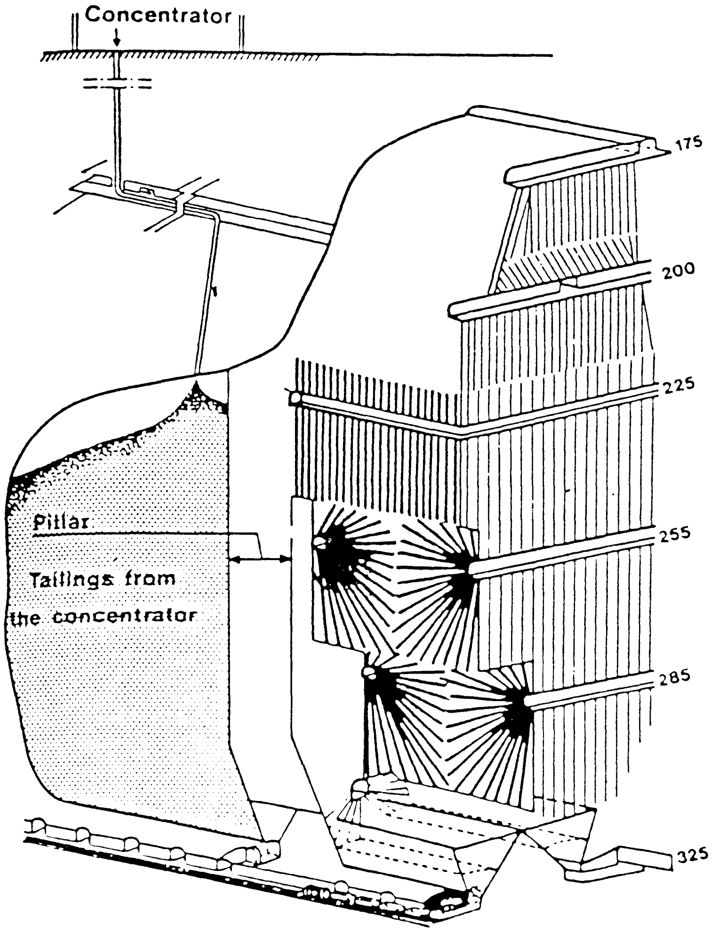


Fig. 18.4.1. Sublevel stoping at Outokumpu Oy Vihanti mine showing the relationship between development and production drilling (Matikainen, 1981).

Table 18.4.1. Sublevel Stoping Basic Dimensions

Mine	Ore Body	Stope Dimensions, ft				Pillars, ft	Haulage, Interval, ft
		Width	Length	Height	Sublevel		
Kidd Creek (Belford, 1981)	Massive base metal sulfide	79	98	299	98	70–98	397
Torman (Matikainen, 1981)	Massive limestone	148–164	328–492	328	49–164	148–164	—
Rio Tinto (Botin and Singh, 1981)	Massive sulfide	66	66–164	131–236	131–236	41	174–276
Mt. Isa (Goddard, 1981)	Bedded sulfide	82–164	98	410–820	66	82	574–984
Burra Burra (McNaughton, 1929)	Massive sulfide	39	328	164	43	43	197
Luanshya (Mabson and Russell, 1981)	Bedded sulfide	39	39	115	36	16–32	164–230

Conversion factor: 1 ft = 0.3048 m.

Stopes are typically contained by a crown pillar, which protects the level above, rib pillars, and a sill pillar through which the ore collection system is cut (Fig. 18.4.3). Substituting filled stopes for rib pillars has been achieved successfully under some conditions (Fig. 18.4.4) (Belford, 1981). Pillars are typically removed through large-scale blasting when stoping is exhausted, and up to 100% of the ore can be recovered under ideal conditions. Stopes can be filled where caving is unacceptable or must be controlled to reduce surface effects or minimize underground stress effects. Filling of stopes also facilitates pillar removal under some conditions.

18.4.2.2 Stope Extraction Systems

The base of the stope may be slotted horizontally prior to production blasting with the base of the slot being a collection trough or series of 45° cones. These are cut through the sill pillar to collect the ore. Alternative to this, no separate slot may be used, and stope blasting will expose the collection system as the stope face retreats. Excavation of the troughs or cones is typically carried out by drilling from the proposed drawpoint and mining upwards using small (2.16 to 3.70 in., or 55 to 94 mm) longholes (Fig. 18.4.5).

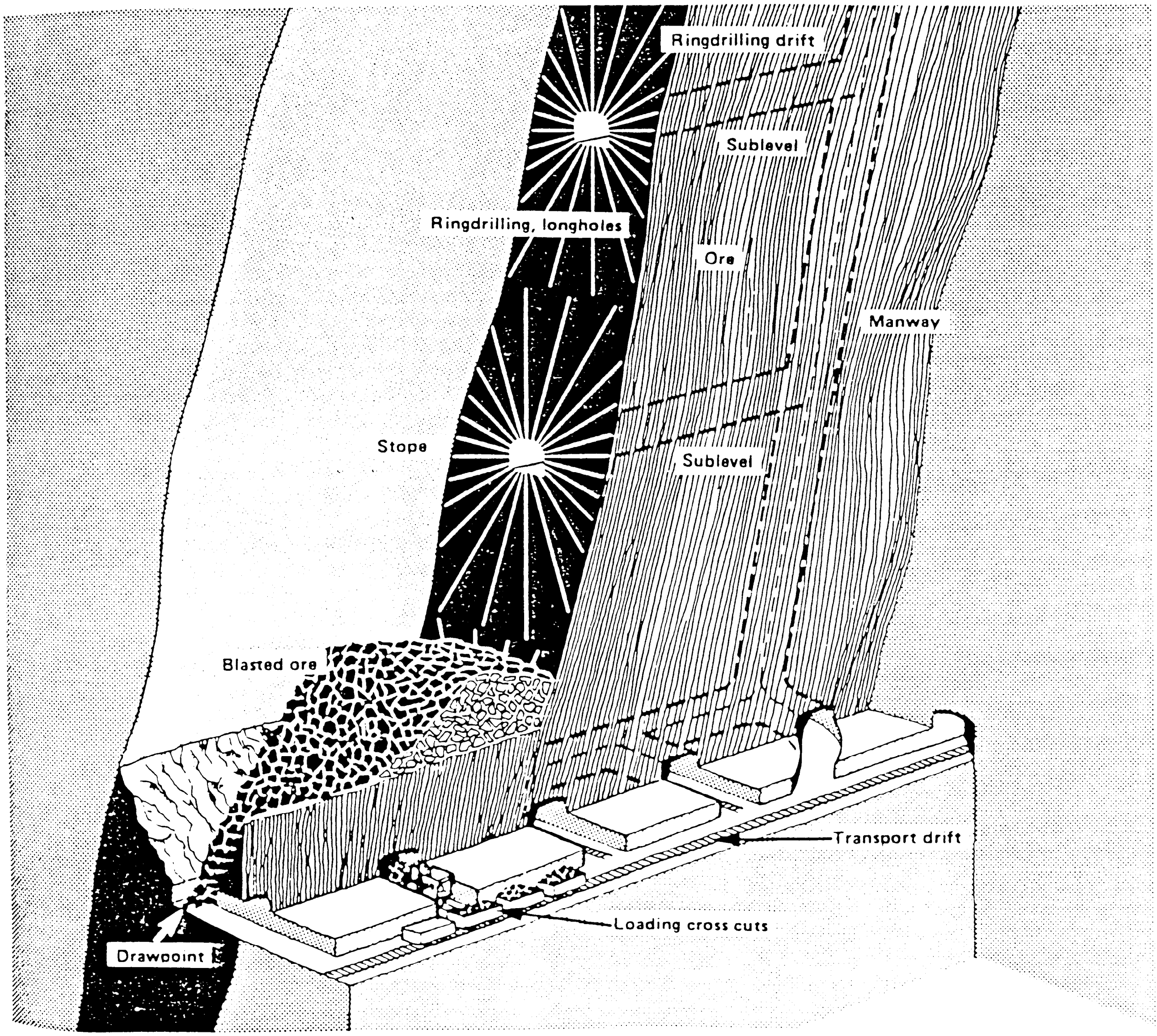


Fig. 18.4.2. Production blasting in a sublevel stope showing ring drilling patterns and loading system (Hamrin, 1982).

Seven ore handling systems are available to remove broken ore from the stope, depending on the needs of the various mining operations (Haycocks, 1973a):

1. Ore may be directed from the cones through finger raises to a grizzly level and then directly into mine cars (Fig. 18.4.6). In this case, secondary blasting is confined to the grizzly itself. This is a system usually selected for very large stopes where the high cost of development is offset by its longevity.

2. Ore can be drawn directly into the finger raises or chutes in cases where it breaks very fine, and then loaded directly through the boxhole to mine cars.

3. In order to minimize the number of boxholes, a single boxhole and chute feeding into a scram or slusher drift may be used. Up to ten drawpoints feed into the slusher drift that will pull ore to a grizzly.

4. This method is similar to (3) above but utilizes a chinaman and drops ore directly into the mine cars (Fig. 18.4.7). The method does not have as much storage capacity as the previous system.

5. The more modern system utilizes trackless rubber-tired load-haul-dump units (LHDs) loading directly out of drawpoints that lie below the cones (Fig. 18.4.8). The machines then transport the ore directly to orepasses. Remote-controlled LHDs have proved most successful for loading out drawpoints under exposed conditions, particularly near the end of the stope life (Bharti, Lebl, and Cornett, 1983).

6. This method utilizes a track-mounted system loading into a rail haulage system (Fig. 18.4.9).

7. Ore is loaded directly from drawpoints onto pan feeders (Lawrence, 1982).

Sill pillar thickness, which is usually the distance between the base of the stope and the haulage level, depends on the ore handling system selected but generally varies between 30 and 70 ft (9 and 21 m).

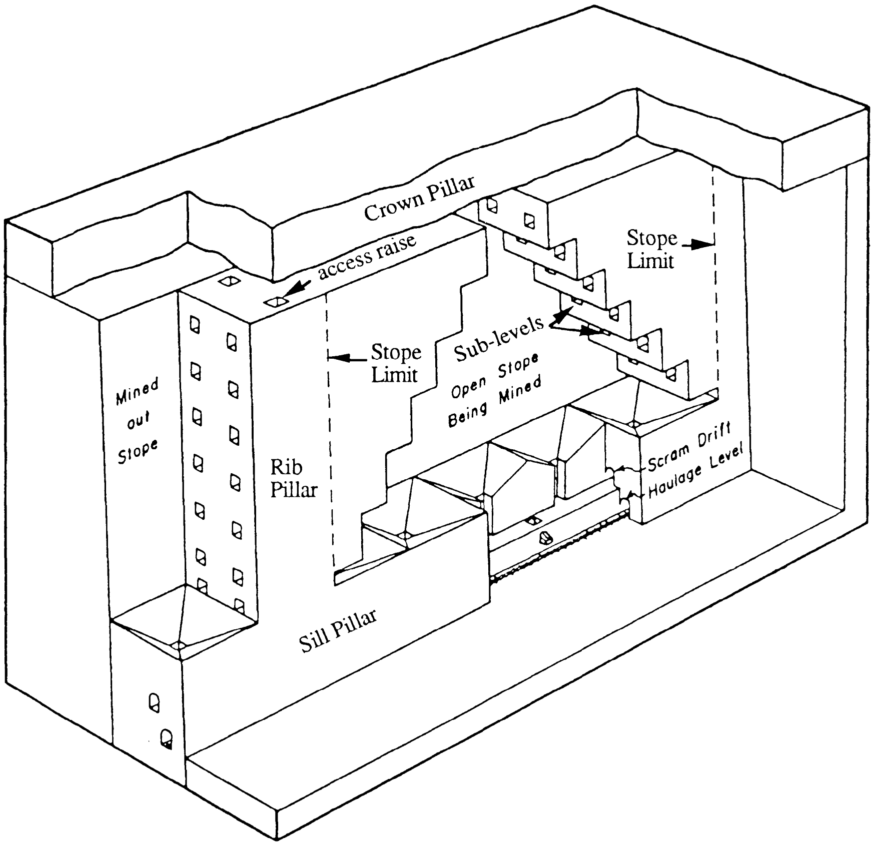


Fig. 18.4.3. Pillar system around a sublevel overhand stope (Clark and Caudle, 1961).

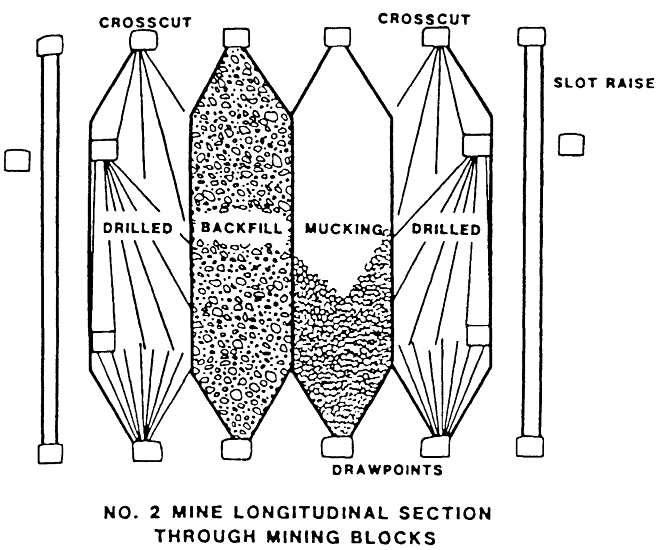


Fig. 18.4.4. Cemented backfill used as alternative to pillars (Belford, 1981).

18.4.2.3 Stoping Action

Stoping typically commences by blasting the lowermost faces first, then moves progressively upwards from the overlying sublevels (Fig. 18.4.3). The overall stope face is carried overhand to facilitate ore flow in the stope. The faces may be carried vertically where the ground is very weak. An alternative to this was implemented at the Burlington mine where the overall stope face was carried out underhand to improve stability (Fig. 18.4.10). Significant ore loading problems resulted (Fletcher and Evans, 1973).

18.4.3 PRODUCTION DRILLING

Consideration must be given to a number of factors before production drilling can commence. Drillability of the ore, required fragmentation size, hole size, hole length, orientation and spacing of drillholes, and desired drilling accuracy must be established. The above factors contribute to the selection of drilling equipment and design of production drilling patterns. Typically, production drilling is accomplished using mechanized longhole drilling. Recent developments in drilling technology include high-efficiency column-and-arm longhole drills, and DTH drill rigs have revolutionized sublevel stoping. These units incorporate electric over hydraulic drive and feed systems, high-pressure pneumatic DTH hammers, or rotary-percussion drilling systems. This type equipment is capable of drilling holes up to 9 in. (229 mm) indiameter to depths of 400 ft (120 m). Alignment errors of holes exceeding 400 ft (120 m) is generally less than 2%. Drilling accuracy can have important economic implications for any proposed mining operation (Almgren and Klippmark, 1981).

Many drill rigs have the capability of drilling a 360° ring of holes. This capability allows for up-hole and down-hole drilling, which in turn results in lower development times and costs. The best drilling efficiency, however, results when a pattern of vertical parallel holes can be drilled from a drill sill developed in the stope (Fig. 18.4.11).

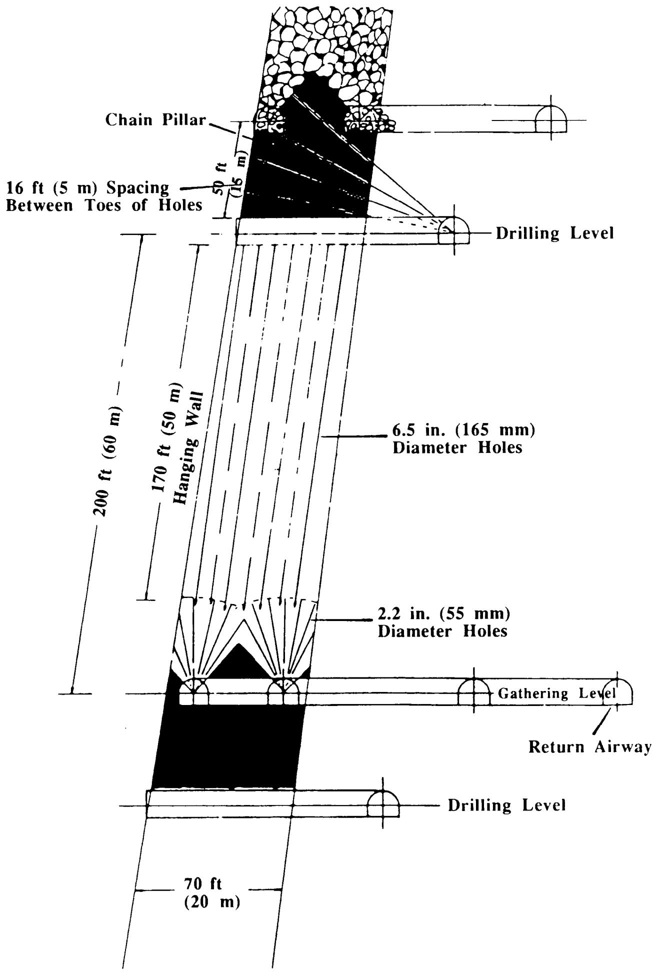


Fig. 18.4.5. Use of small-diameter holes for cutting collection cones (Mabson and Russell, 1981).

Holes patterns, whether parallel or in rings, are drilled from sublevels that are 60 to 400 ft (18 to 120 m) apart (see Fig. 18.4.1). Generally, steeply dipping deposits are drilled with parallel holes, whereas ring drilling is usually adaptable to massive ore bodies. With the introduction of up-hole and down-hole rigs, the drilling sill no longer has to be developed to the full width of the ore contact. A fan-shaped array of holes drilled from a narrow drill sill can effectively delineate the required width of the ore body. This is probably the oldest method, predated only by the now obsolete overhand and underhand shorthole bench and trail (Hamrin, 1982).

The second parallel hole system evolved from the old shorthole bench and trail method and utilizes 6 3/4-in. (170-mm), large-diameter, accurate DTH drills (Fig. 18.4.11). This method is much more efficient than ring or fan drilling and results in consistently acceptable fragmentation. The distance between sublevels can be extended to 150 to 180 ft (45 to 55 m) (Hamrin, 1982). This system also uses a starting slot cut to the full height of the stope for expansion of the initial blast. The problems with this system are the necessity of carrying the bench and trail, which must be continually developed as the stope advances, and the possibility that large charges could cause stope wall deterioration.

Drill patterns and hole sizes are frequently established by applying a preselected powder factor to the size of the ore block to be mined. Hole spacings vary from 20 by 20 ft (6 by 6 m) at Kidd Creek with 7 7/8-in. (200-mm) blastholes (Belford, 1981), to a burden of 5 to 7.5 ft (1.5 to 2.3 m) and a spacing of 7 to 8 ft (2.2 to 2.5 m) at Finnish operations using 2-in. (51-mm) blastholes (Matikainen, 1981).

18.4.4 PRODUCTION BLASTING

Selection of appropriate explosives is related to factors such as required fragmentation, hole size, spacing and burden, hole condition, water inflows, allowable blast sizes, and hardness of the ore. ANFO, water-gels, emulsions, and heavy ANFOS in bulk or packaged form can all be used for blasting sublevel stoping holes. The selection of explosive type is largely related to economics. Presently, ANFO is the least expensive form of explosives. ANFO can be free- poured in down-holes or pneumatically loaded in up-holes. The technology exists today to load upholes up to 6-in. (150-mm) diameter with ANFO. This situation requires the use of specially constructed collar plugs and cascading techniques.

In areas where water inflow is a problem, ANFO cannot be allowed to remain in the hole for periods of more than a few hours. In this type situation, watergel, emulsion, or a blend of the two is used. These products are waterproof, high-density, and typically high-velocity explosives. Again, these products can be loaded in either down-holes or up-holes.

A number of techniques can be used to load holes with explosives. In ring blasting, a slot or raise is developed in the footwall of the stope. This allows for either column loading of a hole or decking of a hole. A column-loaded hole is simply loaded from toe to collar with explosive, while a decked hole is loaded with alternating decks of explosive and an inert stemming material. The decision to load a hole with either a column charge or decked charge is based on considerations such as allowable powder factors, hardness of the ore, uphole or downhole condition, fragmentation requirement, and in some cases blast vibration considerations (Fig. 18.4.12). In the last 15 years, some mine operators have begun using spherical charges to blast large-diameter down-holes. This method requires that a charge height equal to six borehole diameters be maintained within the hole. An inverted crater is created when the charge is detonated. Craters from each hole are designed to overlap each other, effectively allowing for the blasting of a slice across the entire horizontal extent of the stope. Timing and sequencing of blasting is achieved by either electric or non-electric delay detonators. These detonators can be placed either in the hole or at the collar of each hole. Recently, electrical sequential blasting machines have been used in underground stope blasting to achieve greater timing flexibility and detonation accuracy.

The timing sequence of a blast is critical to the success of the blast and resultant fragmentation. Typically, a blast is sequenced so that one ring of holes is fired at a time. Rings are blasted in succession into the opening created by the blasting of the previous ring. Proper blast timing is also critical in minimizing the effect of excessive blast vibration and concussion effects underground.

Secondary blasting is necessary if primary blasting has failed to sufficiently fragment the ore to a size that can be dealt with by ore removal and handling equipment. Secondary blasting can take two forms. The first is called blockholing and involves drilling holes into oversize boulders and loading these holes with explosive. The second method requires that a small shaped charge be placed on top of the site to be blasted. The shaped charge is detonated, shattering the boulder.

Secondary blasting is an expensive, time-consuming exercise. Close engineering control on primary drilling and blasting will

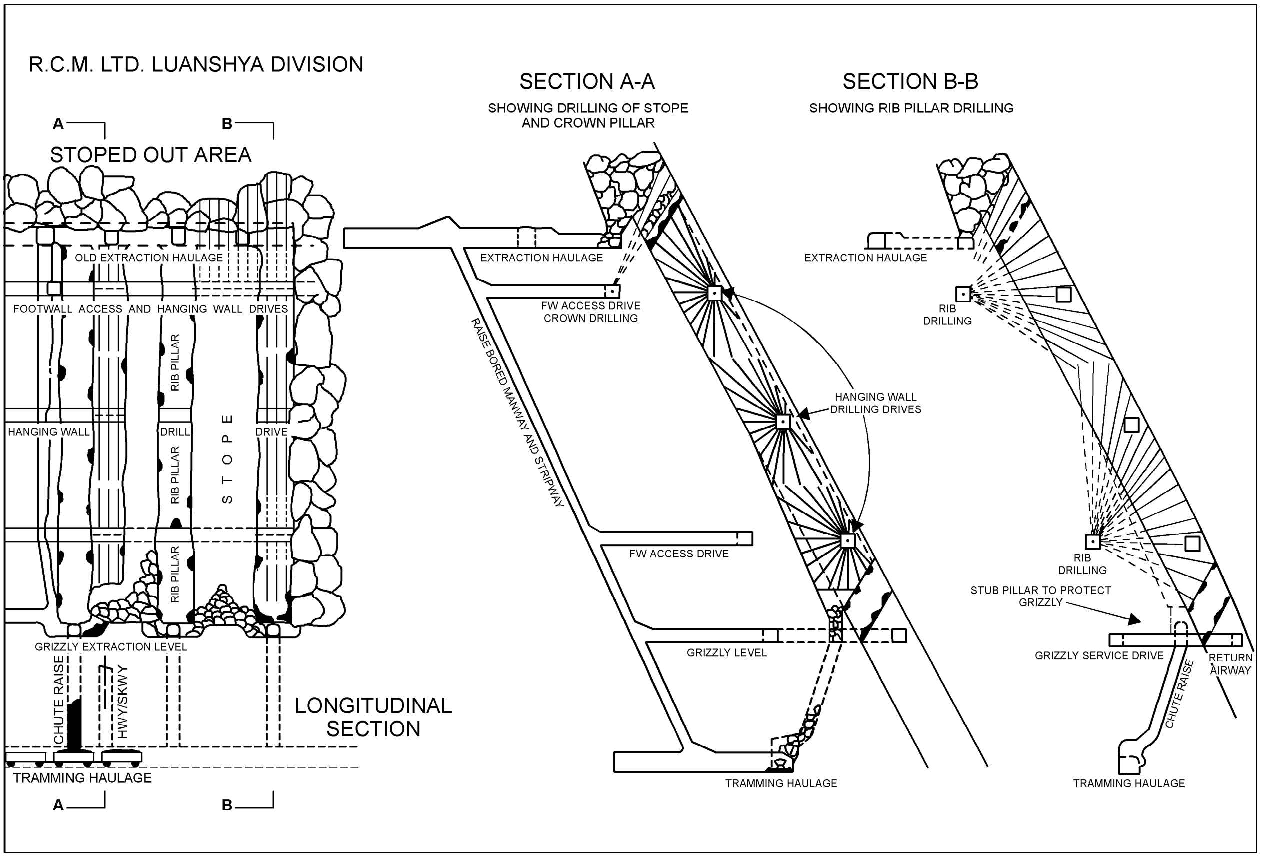


Fig. 18.4.6. Grizzly and orepass collection system (Mabson and Russell, 1981).

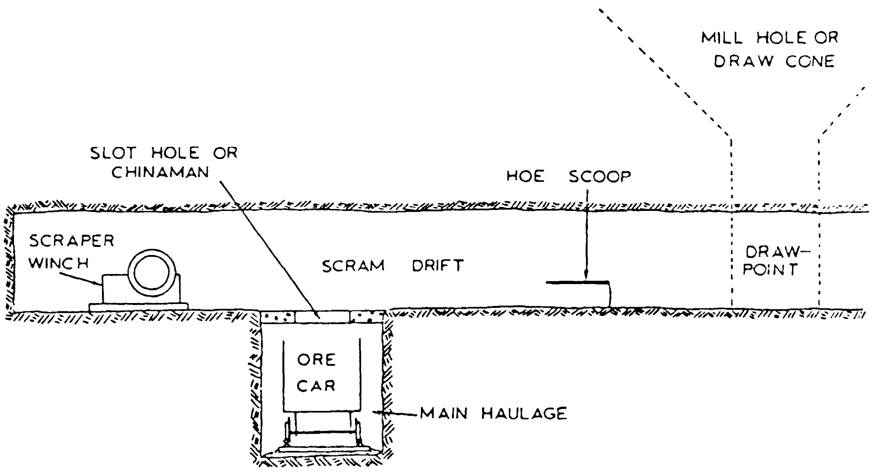


Fig. 18.4.7. Slot hole method of ore handling (Haycocks, 1973a).

minimize the frequency of secondary blasting. Analysis of a large sublevel operation has shown that blasting costs were less than 10% and drilling costs more than 30% of total costs. Optimization of the stoping layout is now possible using computer simulation (Chatterjee and Just, 1981). Dynamic programming methods have also been applied successfully to grade control and scheduling for sublevel stoping (Dowd and Elvan, 1987).

18.4.5 WASTE FILLING

The large openings created by sublevel stoping typically require that some sort of backfilling program be practiced. Backfill includes uncemented rock and sandfill, cemented rock fill, cement hydraulic tailings fill, and a high density tailings or alluvial fill. Backfilling allows for future recovery of support pillars.

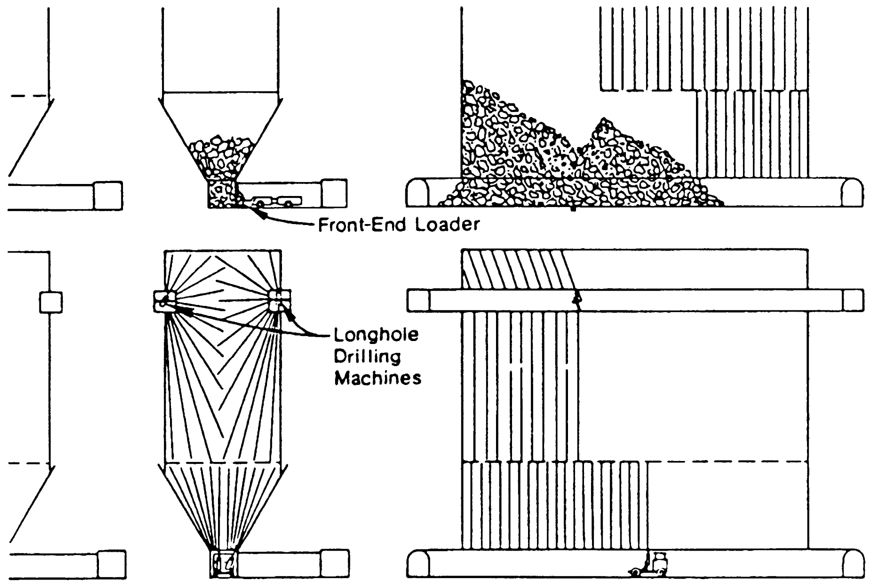


Fig. 18.4.8. LHD drawpoint system of ore extraction (Irvine, 1982).

Recovery of pillars permits up to 90% recovery of ore (Fig. 18.4.13). Backfilling also minimizes the occurrence of subsidence and allows for redistribution of stresses created by the mining cycle. This in turn minimizes the frequency of rock bursting. Backfilling has also been used successfully to eliminate rib pillars between the stopes. In this case the backfill contains sufficient cement to form a self-supporting unit (see Fig. 18.4.4). Cementing the backfill is not always economical, in which cases pillar recovery may not be practical, and the fill is used to control surface movement (Matikainen, 1981).

18.4.6 GROUND CONTROL

Ground control is typically minimal with this system because the rock must be inherently strong, although different types of bolts are used for local support, which may or may not include mesh (Fig. 18.4.14). Regional stresses have been shown to affect stope design and dimensions and are an important consideration. The large open stopes can serve to concentrate high horizontal stresses and cause severe deterioration in development openings in close proximity to the stopes (Goel and Page, 1981).

Cable bolting has proved very successful for full stope wall support, using cables up to 150 ft (45 m) long (Fig. 18.4.15). Cables are normally installed from the blasting sublevels during blasthole development, and resin anchored bolts have proven most successful although cement grout is also used (Goddard, 1981). A variety of cables have been used, including old hoisting cable and low-relaxation strand cable (Matikainen, 1981).

18.4.7 SAFETY CHARACTERISTICS

The sublevel stoping method is a very safe mining method by virtue of design. Typically, miners work only under conditioned ground that has been secured by means of rock bolts, cable bolts, and artificial supports. Miners are not required to work on top of broken ground. In addition, the method is such that mining is scheduled to retreat from unsupported or previously mined areas. The introduction of mechanized equipment has also yielded significant gains in safety. LHD units can be operated remotely by means of radio controls in areas where ground is unsupported. Much of the modern, sophisticated drilling equipment allows the operator to operate the equipment remotely from the collar location of the hole.

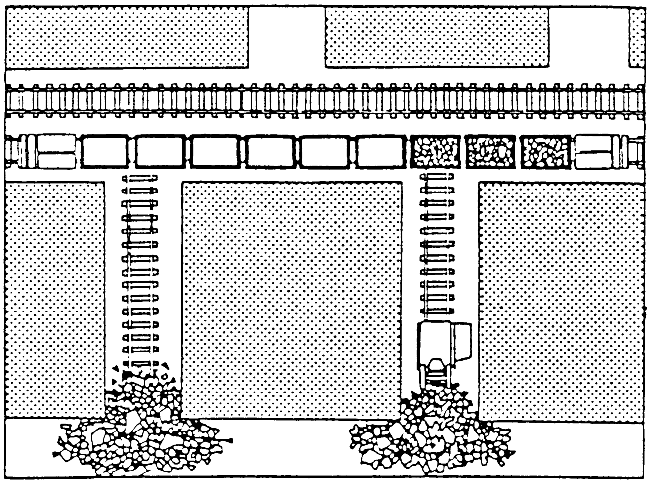
Being a bulk mining method, sublevel stoping requires that large-volume blasts be taken to maintain productivity levels. Such large-volume blasts are usually initiated remotely from a refuge station or surface location. This procedure ensures that no operating personnel are exposed during blasting. Additionally, it ensures that detonation gases have had time to be exhausted by the ventilation system before workers are reintroduced underground.

The large drifts, multiple accesses, and internal raise and winze system allow for a very efficient ventilation system that keep air clean and working conditions comfortable.

18.4.8. ECONOMICS

Sublevel stoping is an inherently high-production, low-cost method and is frequently selected as a primary underground method when surface mining of a deposit is no longer economical (Hedberg, 1981). The key to minimizing costs is mechanization, using as large-capacity machines as the ore body will permit in terms of production capacity and opening size. The utilization of large-diameter DTH machines can reduce total development compared to smaller-diameter longhole drills that are limited to hole lengths less than 90 ft (30 m) by accuracy restrictions.

A breakdown in mining costs for a typical operation is shown in Fig. 18.4.16. This figure clearly shows the development-intensive nature of sublevel stoping, with development accounting for one-third of total mining costs (Lawrence, 1982). Labor cost typically average 40 to 50% of total stoping costs (Matikainen, 1981). Cost analyses are now possible for sublevel systems, involving such factors as stope sizes, drilling and blasting parameters, number of drill points, and loading machines (Chatterjee and Just, 1981). Detailed analyses of stope potential and mining costs can be handled in a sophisticated and precise manner (Pugh and Rasmussen, 1982).



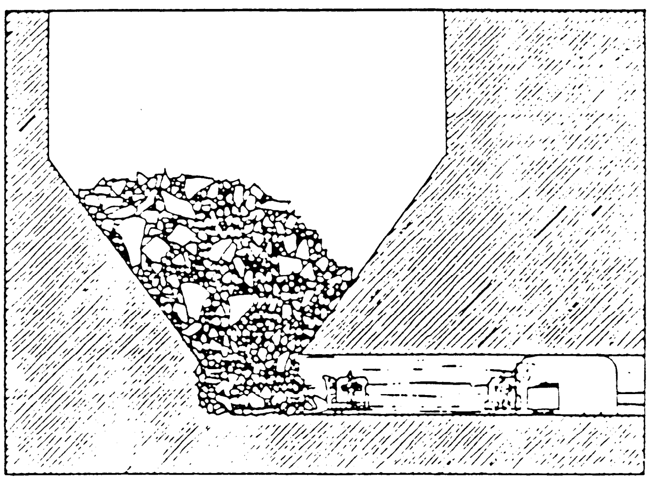


Fig. 18.4.9. Track-mounted loading system for sublevel stope (Bjorkstedt, 1981).

18.4.9 ADVANTAGES AND DISADVANTAGES

18.4.9.1 Advantages

1. Sublevel stoping is very amenable to mechanization, and therefore stoping efficiencies are high, running up to 110 tons (100 t)/employee-shift in larger stopes (Takata, Nanko and Izawa, 1981).

2. The method has a moderate- to very high-production rate, with individual stope outputs running as high as 25,000 tons (22,700 t)/month.

3. The method is safe and, apart from driving the sublevels, easy to ventilate, particularly where weekly blasts are used.

4. Ore recovery can be high, in excess of 90%, when good pillar recovery is possible. Dilution is generally low and can be contained below 20% for most operations.

5. Stopes can be drilled in advance of blasting as equipment is available.

6. In big operations, blasts can be carried out periodically, such as once a week, by very efficient highly trained crews, thus improving blasting efficiency.

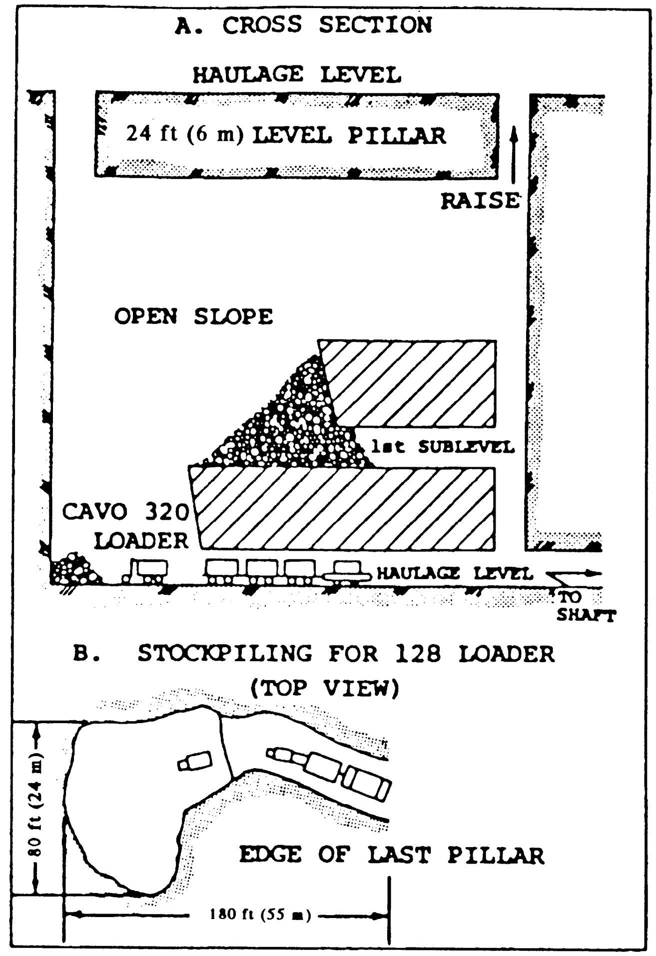


Fig. 18.4.10. Sublevel faces carried underhand (Lawrence, 1982)

7. Ore can be drawn off immediately once primary stope blasting commences.

18.4.9.2 Disadvantages

1. The method is very capital intensive, requiring a large amount of development before production can begin.

2. The method is nonselective and requires most of the ore body to be taken. Variations in footwall or hanging wall are difficult to accommodate.

3. The method becomes very inefficient at lower dips where dilution may be expected to increase.

4. Secondary blasting fumes may leak back into the stope if excessive secondary blasting is necessary.

18.4.10. CASE STUDY: SUBLEVEL STOPING AT KIDD CREEK MINES

The Kidd Creek (Belford, 1982) ore body is a massive base-metal sulfide with horizontal dimensions at the surface of 525 by 2075 ft (168 by 670 m). Initial mining was by open pit, which supplied ore until 1977 when it reached a depth of 680 ft (219 m). At this point, a second phase of underground mining was initiated. Careful planning was essential to ensure sending an even grade of ore to the concentrator. Two mines were commenced with the No. 2 mine below the No. 1 (Fig. 18.4.17).

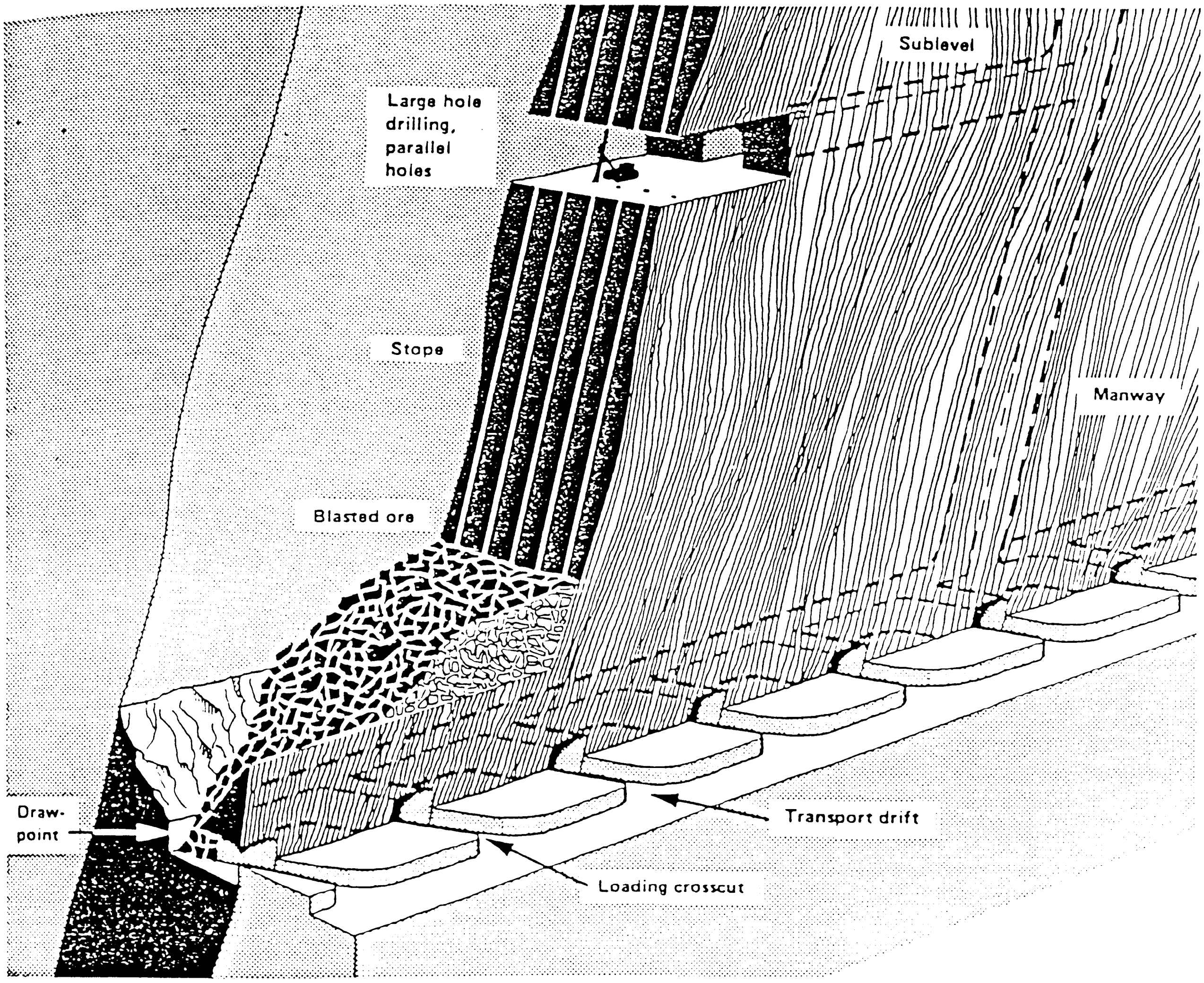


Fig. 18.4.11. Parallel hole system of stope drilling from a drill sill (Hamrin, 1982).

18.4.10.1 Mining Method Design: No. 1 Mine

Sublevel stoping was selected as the mining method, with stopes averaging 55 to 75 ft (18 to 24 m) wide, 95 ft (30 m) long, and 280 ft (91 m) high. Precise stope dimensions depend on local conditions. Rib pillars are 75 ft (24 m) wide and sill pillars approximately 95 ft (30 m) thick. The main footwall haulageways are being developed at 375-ft (121-m) intervals with crosscuts being driven through the ore body to a hanging wall drive. The ore extraction system from the stope consists of a series of drawpoints on 22-ft (7-m) centers driven from the haulage to crosscuts. Ore and waste passes are developed in the footwall.

Sublevels are driven on 95-ft (30-m) vertical intervals and are tied to the footwall ramp system. The sublevels are designed primarily to give access for longhole drilling in the stopes, but have also been used for production to facilitate draw control and ore blending.

Ventilation poses serious problems because of low winter temperatures. Air must be preheated prior to moving it underground, or the broken ore can freeze causing major hangups. Water sprays are used to control dust at drawpoints.

18.4.10.2 Stoping

Stoping commences by boring a raise from the uppermost sublevel to the undercut drill drift (Fig. 18.4.18). The base of the stope is undercut in the shape of a trough using Gardner-Denver DH123 2 1/8-in. (54-mm) diameter drills. The starting raise is enlarged using longhole drilling, and the fan stoping holes are drilled using Robbins 11D rotary drills drilling 7-7/8 in. (200-mm) diameter holes on a 20- by 20-ft (6- by 6-m) pattern. Blasting is done using water gels and ANFO with nonelectric methods of initiation. Initial results show damage to surrounding excavations from blasting the large-diameter charges. The method was resolved by breaking the charge down to a maximum of 350 lb (160 kg) per delay. Secondary breakage is achieved primarily through drill-blast. Hydraulic rock breakers have obvious advantages in that they do not interrupt the loading cycle, but they are bulky and have difficulty operating in the extraction drifts.

18.4.10.3 Equipment

Primary mucking equipment is Wagner ST8 scoop trams powered by GMC V8 diesels. The fleet is designed to work at

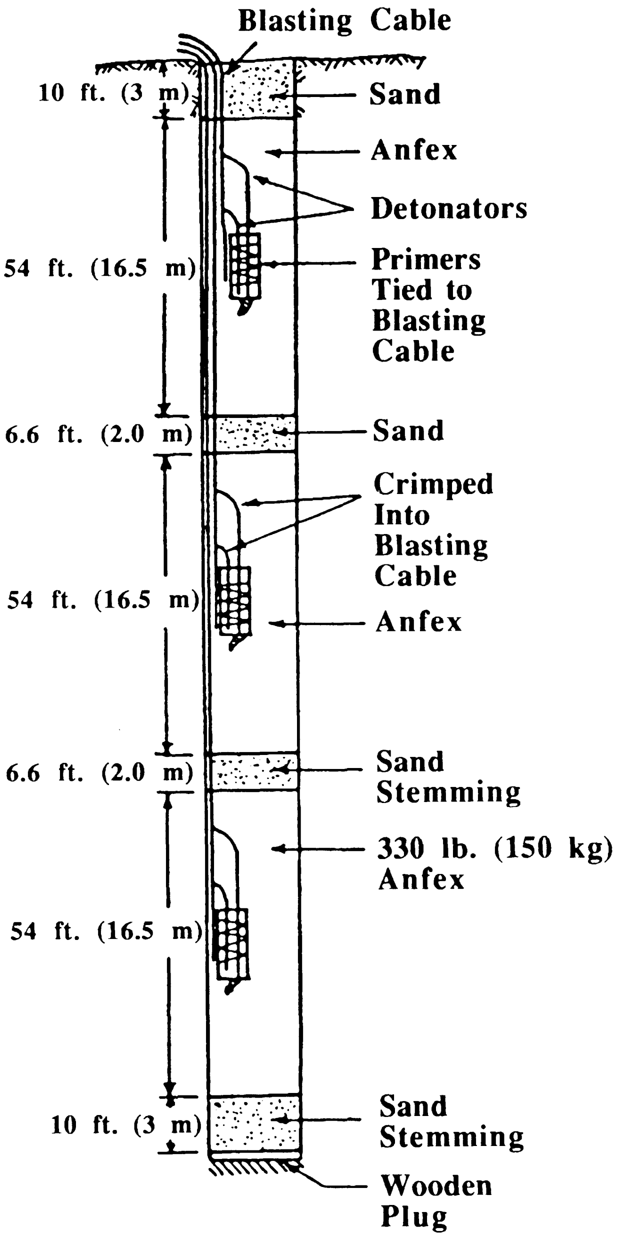


Fig. 18.4.12. Explosives layout in a large-diameter hole showing spacers (Pandey, 1984).

67% capacity with two out of three units being operated. To facilitate equipment utilization, roadway floors are concreted to an average depth of 10 in. (250 mm). This results in significant benefits in the life of the equipment, although roadways themselves must also be maintained and cleaned periodically.

18.4.10.4 Backfill and Ground Control

Stopes are backfilled upon completion primarily using cemented rock fill, but also occasionally sand fills. Design strengths of 1000 psi (7 MPa) are achieved, and fills are carried to the stopes using a fleet of haulage trucks. Both aggregate and cement slurries are delivered by conveyors and boreholes to the mixing stations, and success of the system is based on the efficient mixing achieved.

Ground conditions at Kidd Creek are normally very stable; however, permanent long-life excavations such as haulageways and ramps are supported primarily by grouted rebar bolts. Shotcreting is done in some special areas, such as underground shops and shaft stations. Cable bolting has been carried out in some geologically weak shear zones.

18.4.11. UNDERHAND AND OVERHAND STOPING

Sublevel stoping is ideal for ore bodies or stopes that are over 20 ft (6 m) in thickness. Widths below this create major problems for longhole blasting, and when widths approach 6 ft (2 m) or less, only shorthole methods of stope blasting are practical. At this point, underhand or overhand onto stulls are possible methods if shrinkage stoping is undesirable. Historically, such methods have been widely used in such areas as the Cornish tin and copper mines, Kolar gold field, and gold mines of Colorado. Today the methods are last-resort type because of the inherent limitations in mechanization, which result in low productivity and high cost. At the Geevor Mine in West Cornwall, England, a combination of underhand and overhand methods was used in the same stope (Haycocks, 1973b).

18.4.11.1 Major Development

The ore is typically blocked out by driving haulage levels in the ore 100 to 200 ft (30 to 60 m) apart vertically, the closest spacing serving to allow more detailed sampling of the ore body. Raise/winze connections are driven, again in ore, typically 200 to 250 ft (60 to 80 m) apart horizontally. Both underhand and overhand stoping typically use crown, rib, and sill pillars for protection of the stope, although stulls may be used under some conditions as an alternative to both crown and sill pillars. Box holes are cut in the sill pillar every 15 to 30 ft (5 to 10 m).

18.4.11.2 Underhand Stoping

Underhand stoping typically starts from a raise driven through the center of the projected stope (Fig. 18.4.19). Ideally a stope drive is carried below the crown pillar, and the face is benched underhand for shorthole blasting. A second stope drive is carried some 10 to 15 ft (3 to 5 m) above the haulage level, and cones are cut into the floor of this drift. The face is benched in a series of benches 6 ft (1.8 m) high and 4 ft (1.2 m) wide, with the overall face dipping approximately 60°. After blasting, the ore falls down the starting raise to a box hole. As the face moves backwards, new box holes are exposed. Stoping requires a considerable amount of physical effort on the part of the miners, as benches must be cleaned by hand prior to drilling, and drills and explosives must be hauled into the stope. Good hanging wall and footwall are essential in the method to prevent scaling, and thereby prevent material falling onto miners working below. Scaffolding could be used to bar down the base of the crown pillar. Stulls are occasionally left for support in underhand stopes and roof bolts are also used where necessary. In the Kolar gold field, cut granite blocks have also been used.

18.4.11.3 Overhand Stoping

In the overhand stope (Fig. 18.4.20) development is the same as for underhand, but stoping now commences from the base of the proposed stope, and the face is carried overhand at a dip of about 45°. Once the initial blasts have been carried out, stulls are installed that collect shrinkage material as stoping progresses. The miners use ladders to move between the stulls and work up the shrinkage piles to gain access to the face for

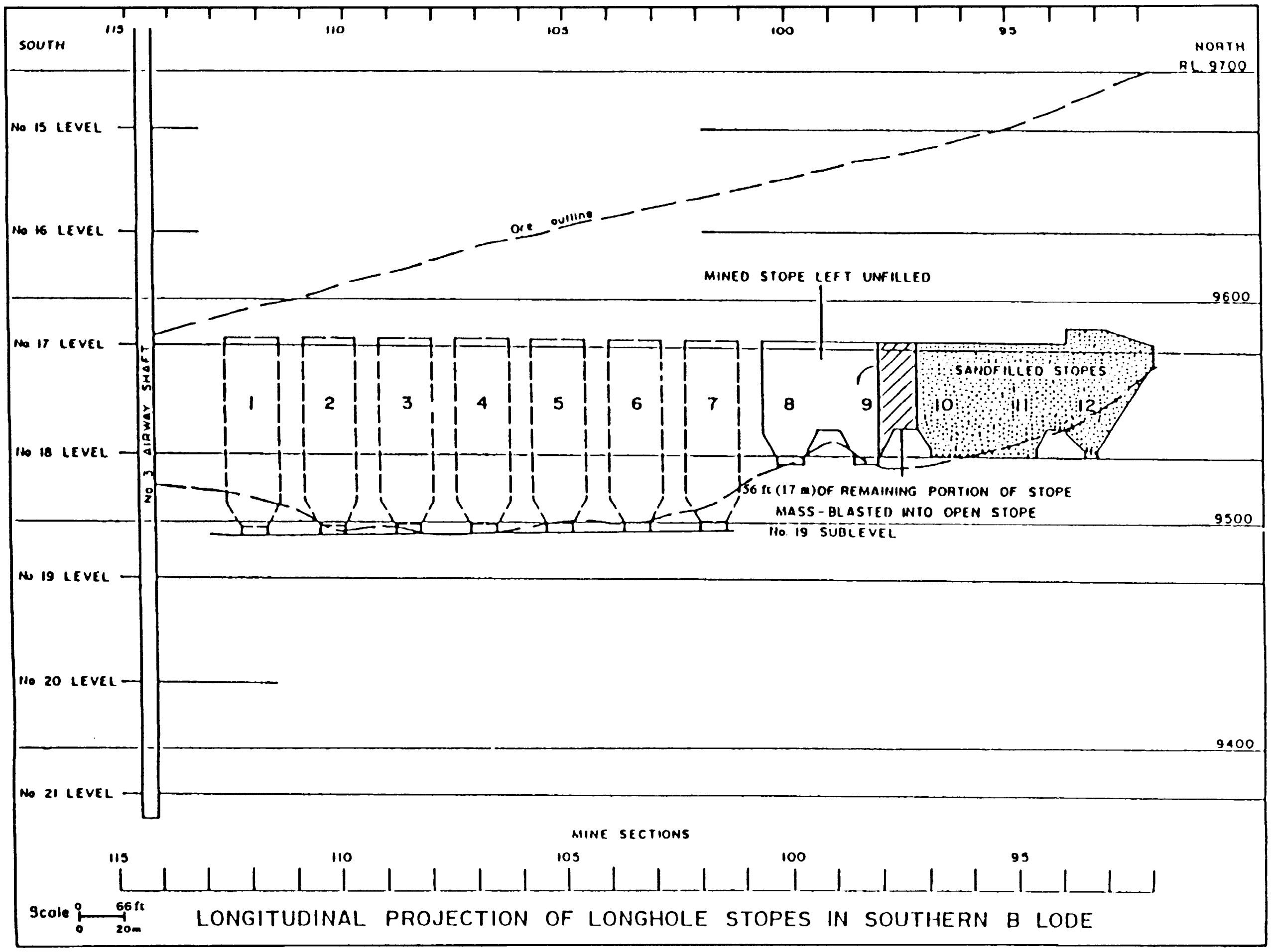


Fig. 18.4.13. Sand filling of completed sublevel stopes prior to pillar recovery (Smaniotto, Solomons, and Tillmann, 1981).

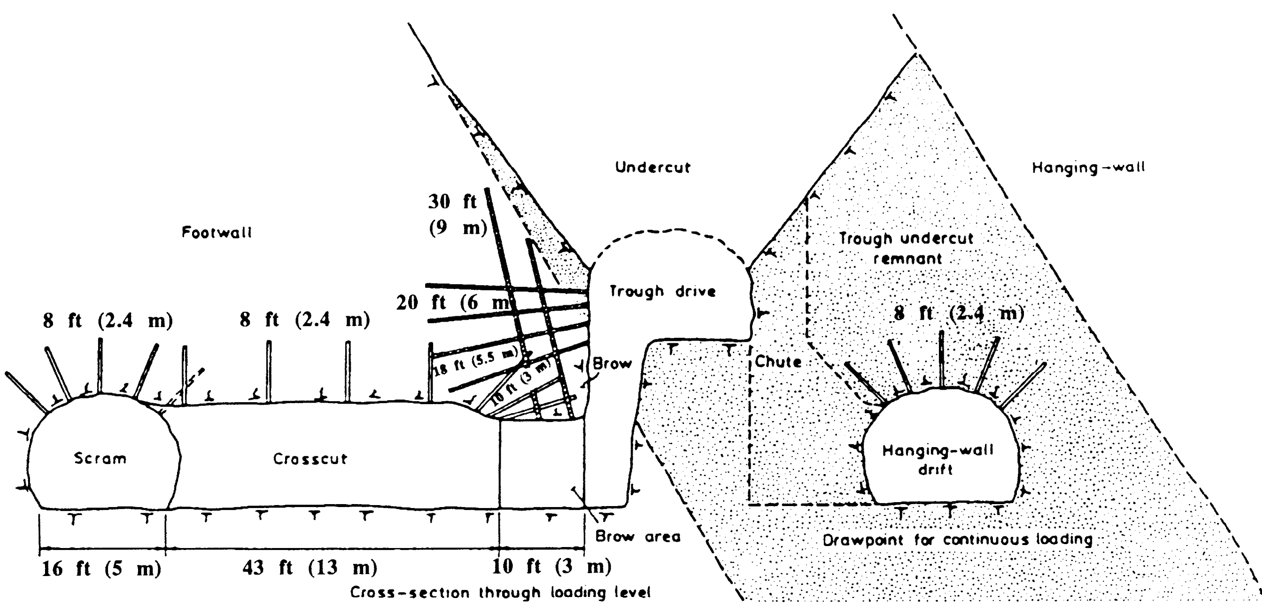


Fig. 18.4.14. Support of loading crosscut showing brow reinforcement (Maw, 1986).

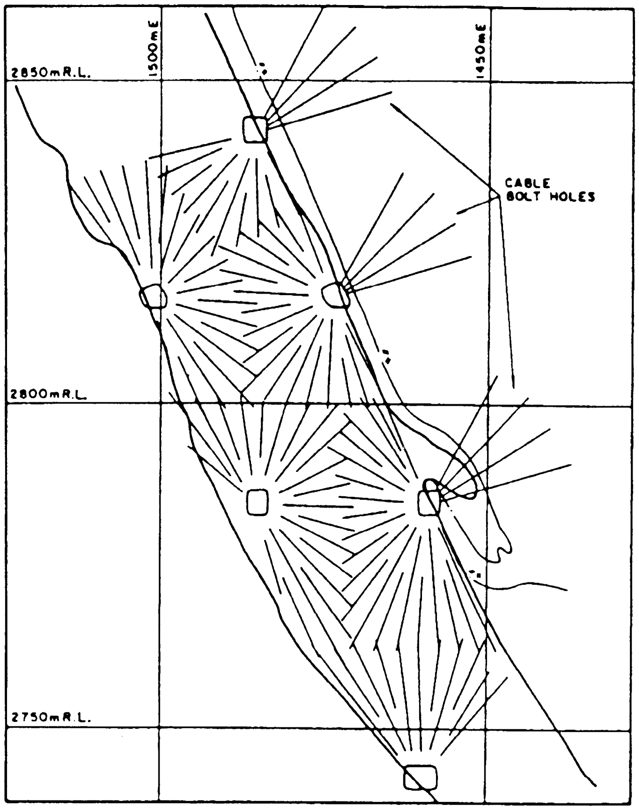


Fig. 18.4.15. Cable bolting holes for hangingwall reinforcement drilled from stope sublevels (Goddard, 1981).

Development	30%
Load & Haul	20
Supervision & Service	14
Stoping	11
General	6
Hoisting	5
Power	3
Stope & Fill	1
Crushing & Conveying	10

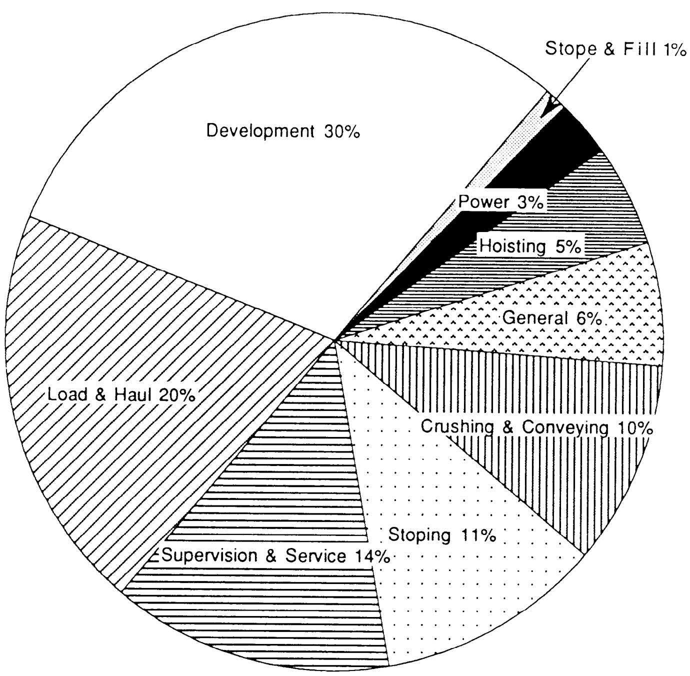


Fig. 18.4.16. Cost distribution for typical sublevel stope (Lawrence, 1982).

drilling and blasting. Again the method requires considerable physical effort on the part of the miners to haul drills into the stope, although ventilation and maintenance of the back is much easier with this method than with overhand.

18.4.11.4 Advantages

1. Both underhand and overhand stoping require very little timber for support, and major development is driven in the ore.
2. Neither method ties up significant quantities of ore in the stope as does shrinkage.
3. Ventilation is good with overhand stoping because air can be directed across the stope face.
4. Maintenance of the back is very good with overhand stoping because the miners work in close proximity to the stope face.
5. The methods are very viable for narrow steeply inclined ore bodies where shrinkage stoping is undesirable.

18.4.11.5 Disadvantages

1. Both underhand and overhand are high-cost low-productivity methods with production rates running 1 to 3 tons (0.9 to 2.8 t)/employee-shift.
2. Underhand stoping has poor ventilation, and leaves the miner exposed to falls of rock from the stope walls or crown pillar.

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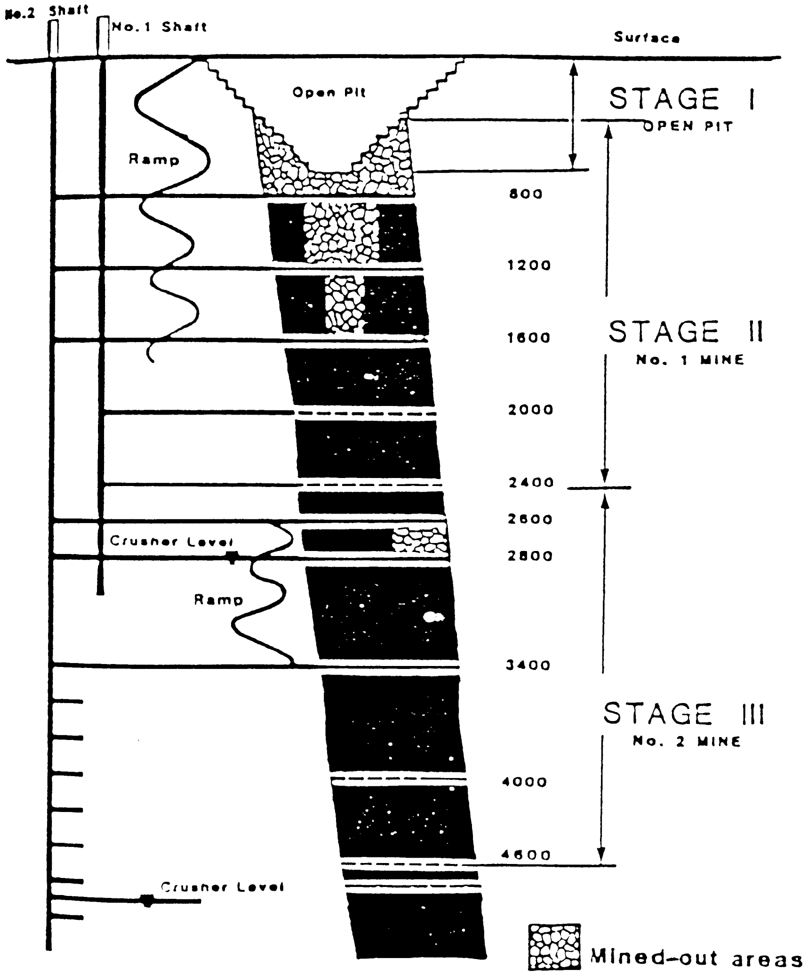

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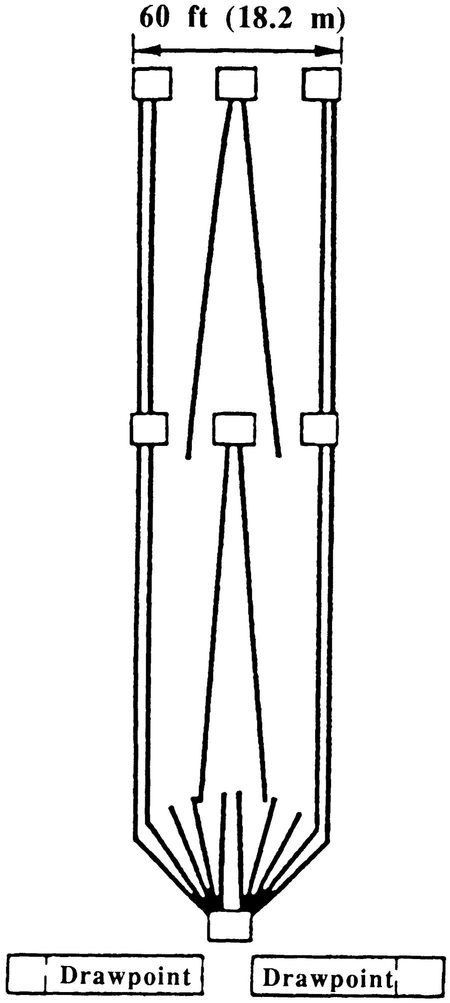

Fig. 18.4.18. Stope blasting patterns in the Kidd Creek No. 1 mine (Belford, 1981).

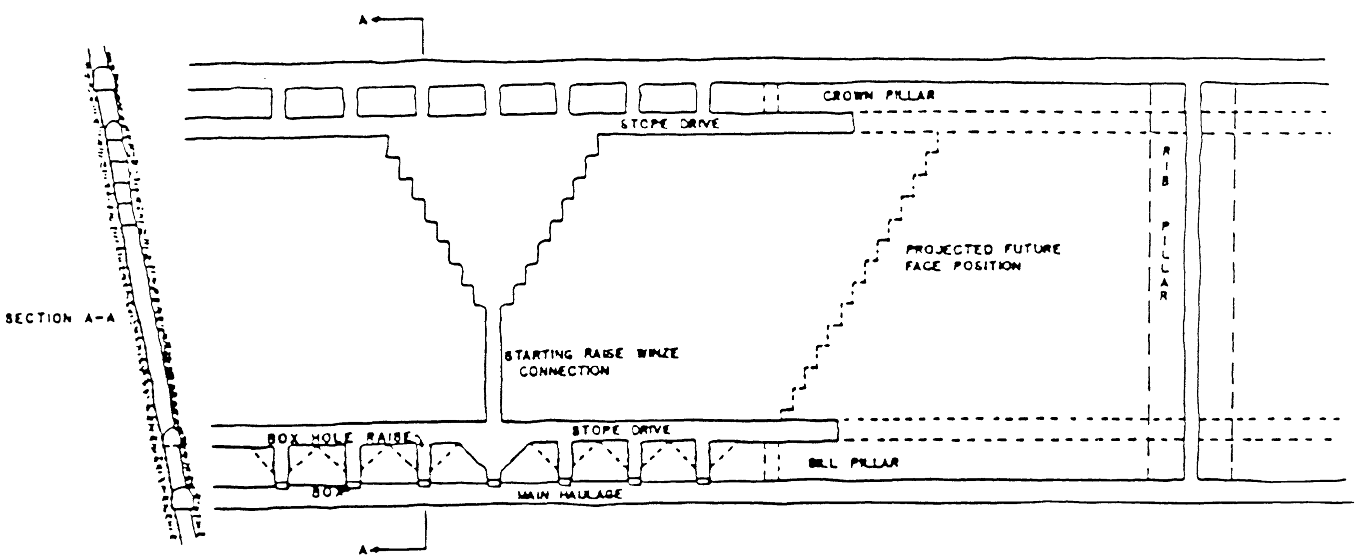

Fig. 18.4.19. Typical underhand stope (Haycocks, 1973b).

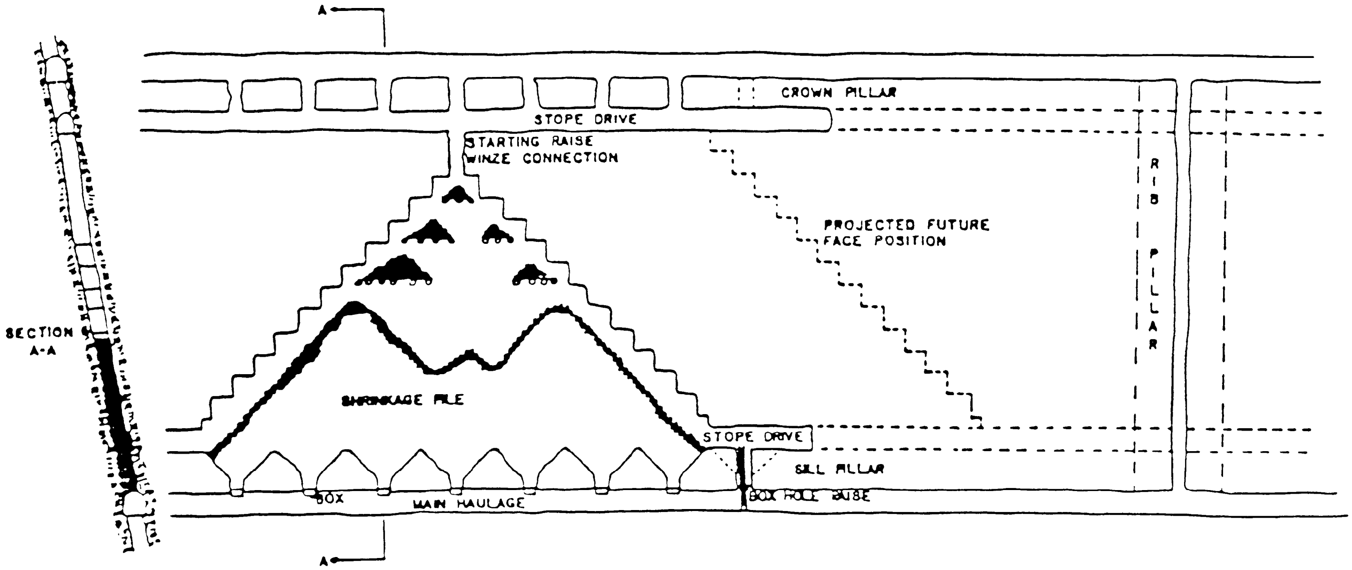

Fig. 18.4.20. Typical overhand stope showing combination of shrinkage and stulls for face access (Haycocks, 1973b).

Chapter 18.5
VERTICAL CRATER RETREAT MINING

KELLY OSBORNE AND VERN BAKER

VCR (vertical crater retreat) mining is a horizontal, flat-back variation of sublevel stoping using spherical crater charges to break the ore. It is the only patented stoping method (Livingston, 1973). Blasting is carried out at the base of vertical holes, making horizontal cuts and advancing upward (Mitchell, 1981). Shrinkage can be utilized in the stopes for wall support. Because the method requires less development than sublevel stoping, it has the potential for lower costs and is finding increasing application not only for pillar recovery but also for primary stoping (Lang, Roach, and Osoko, 1982).

18.5.1 GENERAL METHODOLOGY

The first step in VCR mining is to define a block (stope, panel, section, pillar) of ore that is amenable to this type of mining. The initial criteria are dip and plunge, as the ore must be able to flow down to drawpoints under the influence of gravity. The second factor assessed is ore block shape and consistency. The ore block must have a shape that can be basically defined from two sills spaced a significant vertical distance apart. Once an ore block is defined, the blasting characteristics of the rock can be assessed. The assessment can be done theoretically or tests on similar ore blocks can be done for empirically developing the data (see 18.5.2). Selection of hole size and drilling system can be made once blasting characteristics are defined. The extraction system is often predefined by existing extraction techniques in use at the mine; if not, then ground control, block size, production needs, and availability of equipment will determine the size and layout of the mucking scrams and haulage system (VCR is a bulk method, and high rates of production are common).

Once the design is completed, the top sill and bottom sill are cut. The vertical separation between the sills is dependent upon ore consistency, drilling accuracy, accessibility, and hanging wall competency. After the sills are cut, any secondary ground control measures (such as cable bolts) necessary are installed, and the stope is drilled. The holes are drilled from the top sill down to the bottom sill, usually with a down-hole hammer to minimize deviation. With the drilling complete, the stope should be ready to blast. Blasting is done to take off vertical slices of ore progressing from the bottom sill to the top sill. During blasting, only enough broken ore is mucked from the stope to open up the volume necessary for successive blasts. This keeps the open stope full of broken rock to support the walls until blasting is completed (basically, a form of shrinkage stoping without miners having to crawl in on top of the pile of broken rock each cut). Many variations of the blasting sequence have been used successfully at different operations.

Once blasting is complete, the ore block is a large inventory of broken rock that can be extracted as fast as the extraction system will allow. After the ore is extracted, the bottom sill accesses can be closed off, and the stope can be backfilled from the top sill to maintain rock stability in the area.

18.5.2 VCR THEORY

C. W. Livingston (1973) derived equations modeling rock breakage from crater blasting. The basic theory is that a spherical charge placed at an optimal distance from the back of a stope will break a maximum volume of rock in the shape of an inverted crater. The theory correlates placement of a given charge weight in varying blasthole depths (height above back) to the volume of rock broken by its detonation. The actual charge size and placement is dependent upon the size of the hole and the density of the explosive as well as the characteristics of the rock. Charge geometry can alter blasting results—as the length of the charge increases relative to the diameter (defined by hole diameter), the blasting results adhere less rigorously to the theoretical results. Testing determined that a ratio of explosive column length L to hole diameter D of 6 or less ($L/D \leq 6$) will work similar to a spherical charge. Thus either higher-density explosives or larger-diameter holes will reduce the ratio for the same charge weight. Practice has shown that in most rock types, high-density/high-energy slurry explosives provide superior results. A good explanation of the mathematical equations and a description of a practical test is provided by Orr (1983).

18.5.3 ADAPTIVITY

Originally used to take out large pillars where only limited access was practical, VCR is now being applied to many situations. Some very large blocks have been mined in successive stages with VCR where adjacent ore blocks are filled with cemented backfill once mined out to provide an adequate wall for the next stope to mine against. Some very small blocks that would not support extensive development have been mined with VCR.

Effectively using VCR means adapting it to the specific situation. Many variations on the basic blasting sequence are also being used, the most common being multiple-face crater blasting where the early shots in a round are used to open up a vertical free face for successive shots to utilize. Another common blasting variation is to utilize a mass blast that may break up to 50% of the ore block's tonnage in a single round. Mucking methods have varied from slusher extraction to mechanical continuous muckers, although extraction by load-haul-dump (LHD) is most common.

18.5.4 ADVANTAGES AND DISADVANTAGES

VCR stoping possesses many of the best features of sublevel and shrinkage stoping, with which it shares some parentage. Advantages include:

1. Bulk, high-capacity mining method with good recoveries.
2. Efficient stoping method that is very susceptible to mechanization and can have productivities in excess of 35 tons (32 t)/employee-shift.
3. Offers good wall support during the stoping phase using shrinkage.

4. Safe method with miners working under fully supported roof that can be adequately ventilated.

There are also disadvantages:

1. Requires extensive diamond drilling, pre-stope planning, and development lead-time for maximum effectiveness.

2. Ore is tied up in the stope until final drawdown, which represents lost income and can tie up significant funds.

3. Some ores that are mineralogically unstable may be subject to break-down, causing subsequent difficulties in mineral dressing.

18.5.5 CASE STUDY: VCR MINING AT HOMESTAKE MINING COMPANY

The Homestake gold mine is a deep, underground gold property with a production of approximately 1.9 million tons (1.7 Mt)/year at 0.175 oz/ton (5.5 g/t). The mine has 41 active levels ranging from the 800 level (feet below collar) to the 8000 level. Levels are spaced 150 ft (45 m) vertically. Two main shafts serve for access and ore skipping from the 5000 level and up. Two major winzes provide access and skipping from the 7400 level and 8000 level to the 4850 level. Ramp systems have been developed in major ore zones; however, no ramps access the surface. The mine has been in continuous production (except for World War II) since 1877.

Declining grade and productivity in the 1970s led Homestake's management to investigate bulk mining methods. Some success was achieved with blasthole stoping; however, it was limited to areas of highly competent rock. Investigations led to testing of vertical crater retreat mining. VCR showed promise for several reasons: (1) minimum development was required, (2) broken ore can be retained to support the hanging wall until final draw down, (3) reduced safety concerns, and (4) high productivity. VCR mining was tested in 1978 and by 1983 was providing approximately 50% of the mine's production.

18.5.5.1 Geology

The Homestake mine is located in the Northern Black Hills of South Dakota. The predominant rock structures are three Precambrian metasediment formations, the Poorman formation (pf), the Homestake formation (hf), and the Ellison formation (ef), going from oldest to youngest. All formations have been highly folded into a series of synclines and anticlines, plunging 10 to 80° southeast and dipping 50 to 80° northeast. Major folds within the mine's structure are known as "ledges." Specific ledges contain the eight major ore trends now defined. Ore zones exist in many sizes and shapes within the ore trends. Ore bodies range from 10 to 100 ft (3 to 30 m) wide, 30 to 500 ft (9 to 150 m) in length, with vertical extension of 300 ft (90 m) or more common. Ore bodies are located within the Homestake formation, and gold is erratically associated with quartz, arsenophyrite, and phyrhotite. The Homestake formation ranges in strength from 30,000 psi in upper levels to 19,000 psi in lower levels.

18.5.5.2 Cratering Theory Applied to the Homestake Mine

In 1977, crater testing began at the Homestake mine. Original testing was done with 4-in. (100-mm) diameter holes drilled into the back at various lengths. Based on research, crater tests comparing water gel slurries to ammonium nitrate and fuel oil (ANFO) found high-energy metallized slurries gave superior results (Grant, 1964). Therefore, a high-density (1.36) aluminized slurry was used for initial testing. Results from these initial tests were tabulated, and empirical values for the rock at Homestake were given to many of Livingston's constants.

In applying test results to the basic mathematical relationships, the explosive length-to-charge diameter ratio (L/D) became a factor. Homestake found an L/D ratio of 6:1 to give the best fragmentation and still provide data comparable to true spherical charges. Testing and refinement produced a charge depth of 6.75 ft (2.0 m). Analysis of Canadian operations successfully applying VCR showed that 6 1/2-in. (166-mm) diameter holes were effective. This was a fairly standard bit size. Down-hole drills were available on the market for drilling the large 6-1/2-in. (166-mm) holes. Down-hole tests made in 1978 were successful.

18.5.5.3 Defining and Evaluating an Ore Body

Ore bodies at the Homestake gold mine are defined by diamond core drilling from main levels, spaced 150 ft (45.7 m) vertically. Drill setups are spaced on 50-ft (15.2-m) intervals along strike (Fig. 18.5.1). An array of angled holes is drilled in a vertical plane to define the ore in each section. All core is visually logged and defined by assay value. Sections are drafted of the vertical planes encompassing holes from each setup, and from these sections, estimates are made of the grade and tonnage of an ore body.

Each ore body is evaluated or reevaluated every six months and then assessed to see how it best fits into the overall mining sequence. The ore body is evaluated for its adaptability to VCR mining (Fig. 18.5.2). This is the cheapest and most productive mining method and is applied whenever possible. Once the mining method for an ore body has been selected, a rough schedule for mining is developed.

Several physical constraints determine the practicality of using VCR mining for an ore body. Hanging wall and footwall dips are a primary concern as are ore body consistency, thickness, plunge, and rock competency. Flat hanging walls tend to increase dilution. Footwalls of less than 55° dip tend to have large amounts of broken ore hanging-up on them. This not only hinders ore recovery, but may also build up, causing blasting problems by allowing little or no relief for sequential blasting. Ore consistency and thickness usually cause problems with blasting and dilution if not considered. Ore widths less than 16 ft (5 m) do not crater blast well, but they may be slab blasted if an area can be cratered nearby for relief. Waste horses in an ore body either complicate blasting or vastly increase dilution (sometimes both). Thickness is important as narrow stopes tend to have higher dilution factors because more hanging wall is exposed per ton of ore. Flat plunges can result in excessive waste mining during the development stage. The need to get the drilling sill (top sill) over the down-plunge ore necessitates waste mining. Also increased drilling is required per ton of ore due to the amount of waste material that is drilled to get to the flat-plunging ore. Almost every physical constraint can be and has been overcome; however, forcing the VCR method on an ore block not suited to VCR will be counter-productive.

18.5.5.4 Stope Development

PRELIMINARY DESIGN OF A STOPE. This segment is called "preliminary design" because all the components of a stope as discussed are constantly modified as new information becomes apparent during actual mining of the sills and necessary drifts.

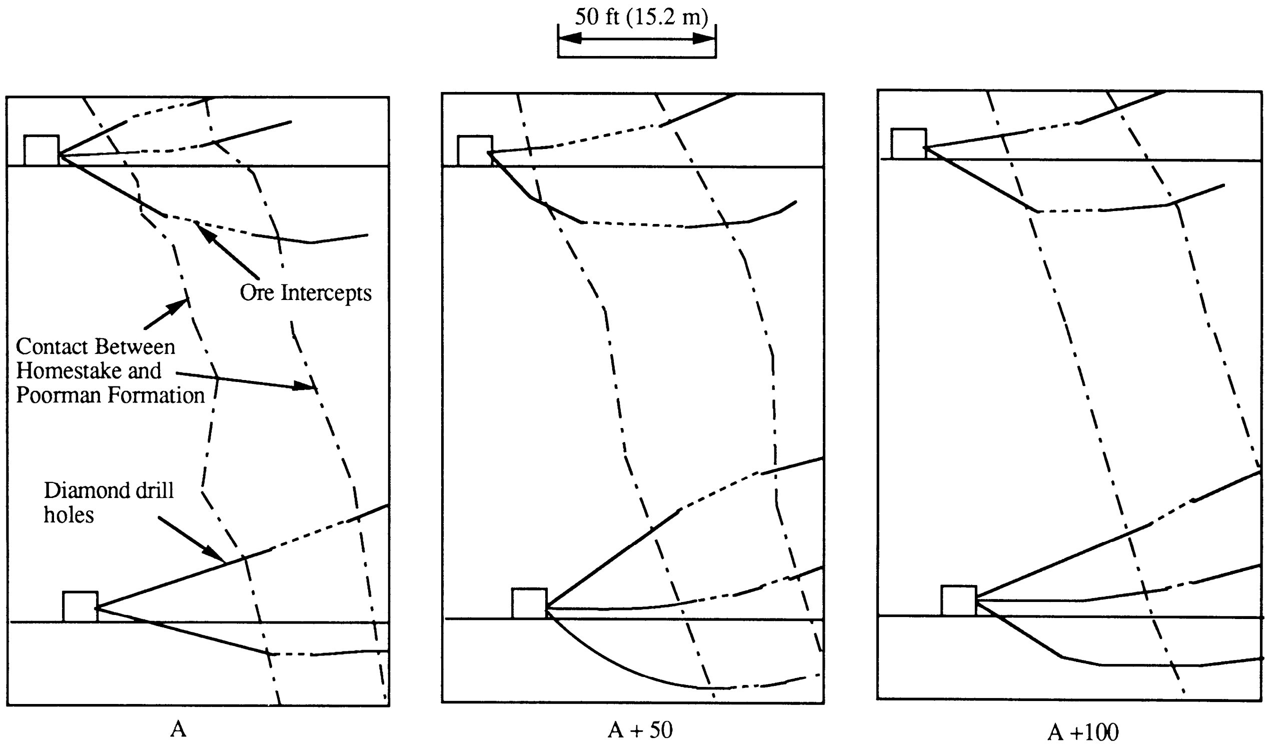

Fig. 18.5.1. Vertical cross sections with diamond drillholes and ore intercepts.

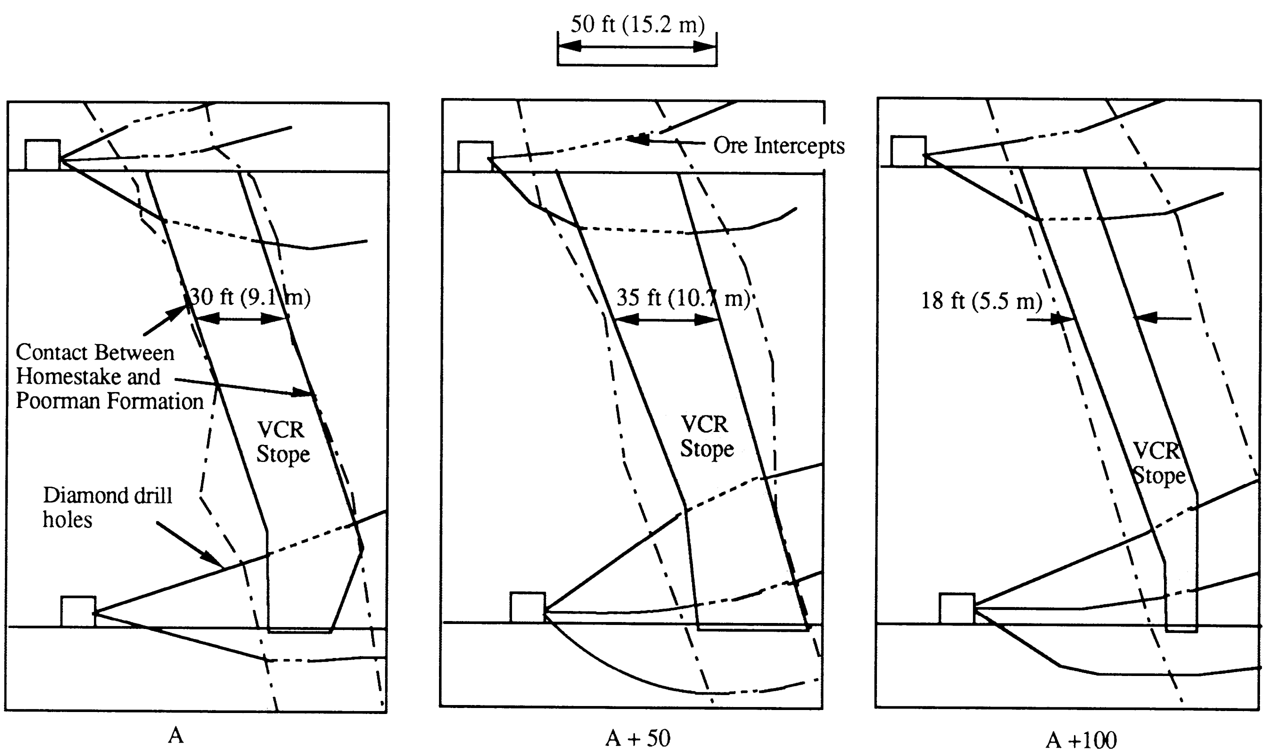

Fig. 18.5.2. Expected VCR stope outlines from plotted diamond drill intercepts.

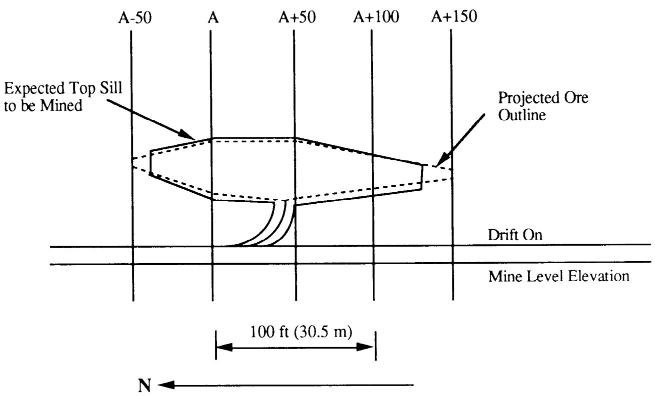

Fig. 18.5.3. Top sill layout of projected ore from vertical sections and expected sill to be mined.

to 50 ft (9 to 15 m) along strike. Ground conditions determine scram spacing (smaller spacing allows higher extraction without remote mucking, but leaves smaller pillars). Extraction boreholes are usually planned in conjunction with LHD mucking. A borehole is designed to access a haulage level 150 or 300 ft (45 or 90 m) lower. A power chute is built at the bottom of the borehole to regulate rock flow into granby rail haulage cars. Car loading by LHDs is sometimes used but is much less productive than a borehole system (Fig. 18.5.5).

CAVO muckers are usually used on smaller stopes in remote and/or hard-to-ventilate areas. The extraction system is designed for side-loading granby cars. The main haulage is track and driven 30 to 40 ft (9 to 12 m) from the bottom sill along strike. Scrams are driven as closely spaced as possible to maximize extraction because remote mucking is usually not feasible. Drifts and scams are driven 9 by 9 ft (2.7 by 2.7 m) arched. Scrams should be as perpendicular as possible to the main haulageways to facilitate side loading.

Some large ore bodies have necessitated panel mining systems. The basic difference is that adjacent stopes (panels) must

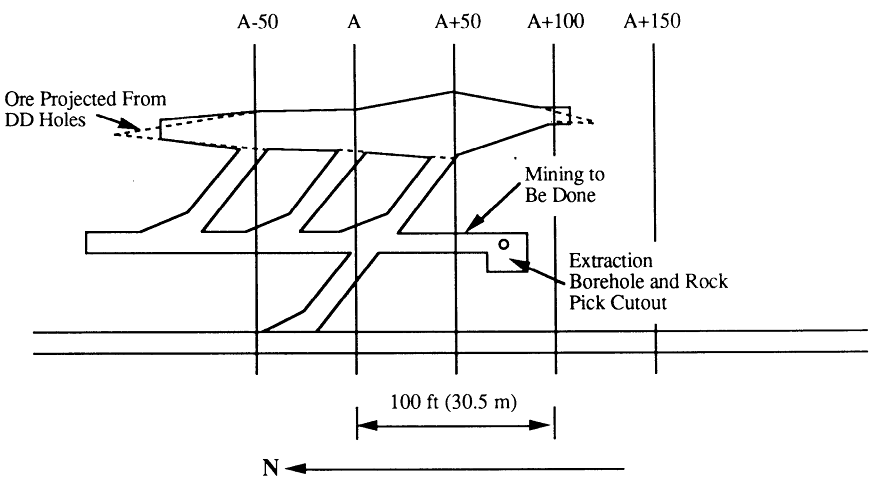

Fig. 18.5.4. Bottom sill layout on projected ore with mining to be done for scrams and borehole dump.

This is a big key in making VCR mining work, particularly with irregular ore bodies.

Once it is decided that an ore body will be mined by VCR, geologic data are used to develop and lay out the plan views and projections at the top and bottom sill elevations (Figs. 18.5.3 and 18.5.4). The layouts are designed to minimize development and optimize the drilling and blasting of the stope. If ground conditions or stope size dictate it, pillars of unmined ore may be left in the top sill. Once the expected sills are laid out, top sill access is usually designed as the minimum drifting necessary to reach the ore zone and planned as a 9- by 9-ft (2.7- by 2.7-m) or 10- by 10-ft (3- by 3-m) arched drift depending upon the LHD size used by the development crew. Sometimes ore bodies are stacked, and the top sill for one stope becomes the bottom sill for another stope. In this case, top sill access is usually designed to be integrated into the extraction system of the next stope. The bottom sill mucking system is very important as it determines the productivity of the stope. Mucking systems are designed in the footwall almost exclusively because of drawdown problems with hanging wall scrams. Whether a LHD unit or a CAVO mucker is used makes a large difference in mucking configuration.

LHD extraction is used predominantly. With LHDs (2 and 3 1/2 yd^3, or 1.5 and 2.7 m^3), accesses and drifts are usually driven 10 by 10 ft (3 by 3 m) arched. The main haulageway is usually driven along strike with 40 to 50 ft (12 to 15 m) separating it from the bottom sill. Scrams are driven into the sill every 30

be sequenced and backfilling must contain cement to provide strength. Low hanging wall competency has forced reduced stope heights in certain areas of the mine. One hundred vertical feet (30 m) between top sill and bottom sill is becoming almost as common as 150 ft (45 m). This has greatly reduced hanging wall failure.

DEVELOPMENT SEQUENCE. Top sill development consists of driving an access to the ore body and then cutting out the ore on that horizon, together with any waste mining necessary to form the drilling sill. The sill is now ready for ground control or down-hole drilling. The drilling sill must be cut at least 11 ft (3.4 m) high and 15 ft (4.5 m) wide for drill mast clearance and drill maneuvering.

The extraction system is developed with the bottom sill, which is accessed via the designed mucking system. The ore zone is accessed as soon as possible so that it can be cut out and compared to original estimates and design modification made as necessary. If the ore extends below the sill, but only by a small amount, the development crew may mine a 12-ft (3.6-m) cut out of the bottom of the sill. Once done, the crew will drive the extraction system and place the waste into the area just mined. As the crew mines the drifts, any major slips or ground control problems are noted and plans are changed when necessary. The development crew usually mines the bottom sill cut 10 ft (3 m) high. Often another cut is made above the sill cut. This helps

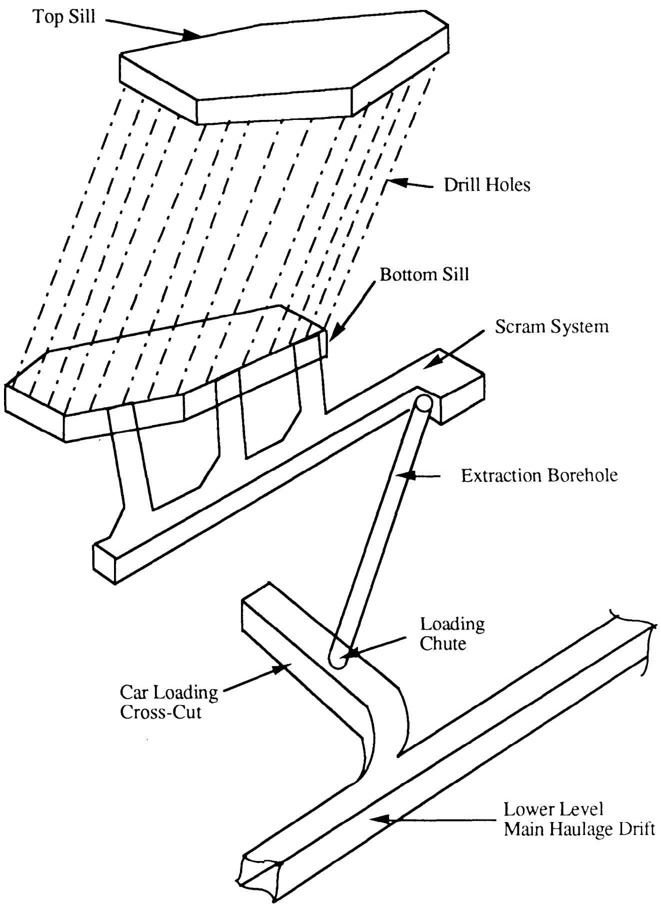

Fig. 18.5.5 Generalized schematic of VCR mining method.

minimize blast damage to scrams and also reduces the length of the down-holes to be drilled, thereby improving accuracy.

During the development stage, the preferred equipment is a 3.5-yd (2.7-m) Wagner diesel LHD and a two-boom, pneumatic hammer, diesel jumbo (MJM20B or GD Minibore). When a crew so equipped is not available, a crew outfitted with jacklegs and a Wagner ST2D (or JC220E) will do the development. When necessary, l-yd^3 (0.75 m^3) LHDs are used, or even slushers and track muckers. The extraction borehole is an important part of many stopes. This is usually drilled and reamed by a Dresser Super 480 or Dresser 500 machine to 6 or 7 ft (1.8 or 2.1 m) diameter.

Mining the cutout for a borehole machine that will drill and ream an extraction borehole is usually necessary and often a priority. Borehole drilling and then the installation of the power chute on the bottom of the borehole often cause delays to production, and close attention must be paid to these steps.

18.5.5.5 Exploitation

GROUND CONTROL. There are several stages of ground control in the development and extraction of a VCR stope. First, pillars may be designed into the sill cut to keep widths minimized and the back stable. Second, is the immediate ground control of each round blasted by the development crew. Here 5- and 8-ft (1.5- and 2.4-m) long bolts are installed (often in conjunction with mats) to secure the immediate back and walls. Cable bolts are the most common form of secondary reinforcement. Cable bolts are used to stabilize the back and hanging wall of the top sill during the blasting and mucking cycle. This helps to maximize the safety of blasters and to minimize dilution. Bolt length varies from 30 to 60 ft (9 to 18 m). Bolt length is determined from sill size, ground conditions, and additional mining plans for ore above the top sill. Bolts are twin 5/8-in. (16-mm) wire rope cables installed parallel in a 2 1/4-in. (57-mm) diameter hole and grouted with a type 2 cement. The standard pattern of cables is 10 by 10, with 10 ft (3 m) between rows and 10 ft (3 m) between bolt points.

Sequencing of mining by paneling a large stope is done to minimize the amount of open hanging wall at any given time. This has been effective in minimizing hanging wall slough. It does necessitate delays in stope production for backfilling of panels. Research into the rock mechanics of VCR stopes at Homestake by the US Bureau of Mines led to a rule of maintaining a 4:1 length-to-width ratio as the maximum amount of hanging wall to be opened by any single panel or stope.

During the mining cycle, several things can be done for ground control. The use of controlled blasting techniques along the hanging wall helps to minimize hanging wall fractures. Also keeping the stope as full of rock as possible to support the hanging wall during blasting seems to be effective.

Specialized equipment for ground control is limited to cable bolting equipment: a ringdrill and grout pump. Ringdrills are equipped with full rings and BBC 120 drills. Grout pumps are small auger-style pumps used to pump grout (Portland cement) into cablebolt holes.

DRILLING. The drill crew is provided with a map of the sill with the grid overlaid on it. Surveyors mark the location of each row (numbered) and reference strike lines (lettered). The drill crew will then drill plug holes at each location indicated by survey. (Plugs are wooden hole plugs, driven into drilled holes, and spads inserted.) Strings are pulled across rows to line up the drill mast and along strike lines to locate holes.

Ore bodies and sill configurations vary widely, and patterns are adapted to each stope. General guidelines are to utilize an 8- by 8-ft (2.4- by 2.4-m) pattern (Fig. 18.5.6), to minimize breaking into the hanging wall, and to minimize drill footage. Situations often force (or allow) departures from standard such as employing 10- by 10-ft (3- by 3-m) patterns, the need to splay holes into the hanging wall to recover ore, or utilizing a high density of drill footage near the top sill due to the changing geometry of the ore body between the sills. Holes that are drilled from the top sill and drilled through to the bottom sill cut are called "breakout" holes. Three breakout holes per row to start the blasting is the accepted minimum. Accuracy is a necessary part of the drilling. The grid developed in the top sill provides the basis for accurate drilling. Once a drill is lined up on the collar location of a hole, the inclination indicator mounted on the mast is used. The inclination indicator is set for the specified angle of the hole, a green light then indicates when the mast is correctly aligned, and a red light indicates improper alignment.

When the drilling is nearing completion, the location of holes in the bottom sill is checked. If there appears to be a problem, surveyors may be asked to check actual vs. planned hole breakthrough locations. If major deviation has occurred, then additional holes may be needed to cover the stope correctly.

Once set up, drilling proceeds fairly rapidly. Many crews can average 200 ft (60 m)/shift. Tons blasted per foot of hole drilled (total drilling) runs about 3.5 (11.5 t/m). Stope drilling is done with down-hole hammer drills and 6 1/2-in. (165-mm) diameter bits. The Homestake mine has four Gardner Denver ATD 3100 crawler carriers and two Mission Megadrill 6200 units, each equipped with TRW Mission masts and rotating head

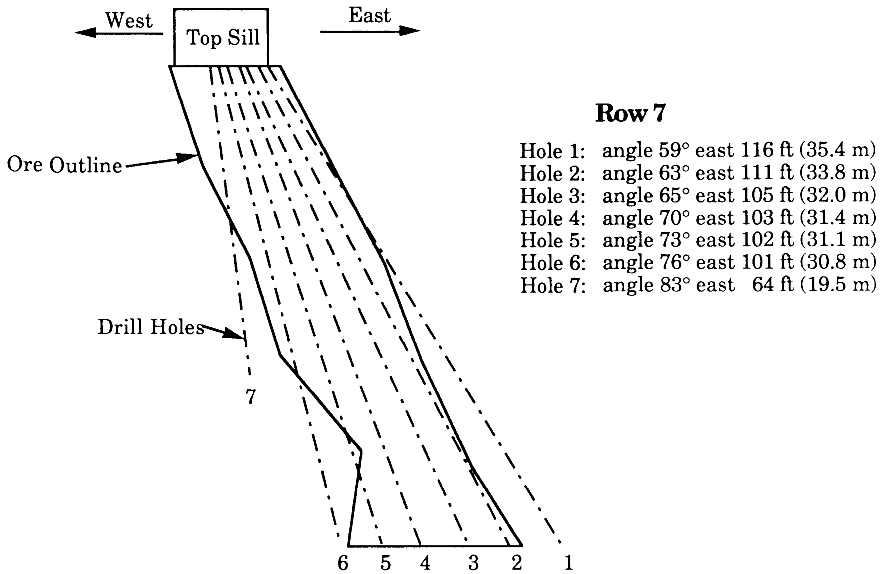

Fig. 18.5.6. Cross section of standard row of VCR pattern layout.

assemblies. The GD carriers utilize air-powered hydraulics and air tramming motors, while the Mission carriers have diesel- and electric-powered tramming motors and hydraulics. Compressors are necessary to boost mine air 100 psi (690 kPa) to provide 250-psi (1720-kPa) air to the down-hole drill. Each compressor unit consists of two model AOL 100 booster compressors driven by two 460-volt, 40-hp (30-kW) electric motors. A carrier combined with a compressor comprises one down-hole drill unit.

BLASTING. Blasting of VCR stopes is performed by a two-man crew monitored by an engineering technician. These technicians order necessary blasting supplies and do the design work for the blast. Blasting crews are on a production bonus system that rewards them according to tonnages broken. Supplies are usually delivered to the top sill by a LHD and then distributed by hand. Sometimes a Bobcat can be provided to the blasters for handling powder and sand within the top sill.

Prior to blasting, each hole is measured and recorded. Holes are measured by attaching a 1 1/2-in. by 1 1/2-in. by 3-ft (38- by 38- by 914-mm) piece of mahogany to the end of plastic coated 1/4-in. (6.4-mm) cable. The cable is approximately 150 ft (45 m) in length. The cable is stamped every 5 ft (1.5 m) from the end with the corresponding distance on the stamp. Each hole is measured by lowering this device down the hole until it reaches the floor or rock pile on the bottom sill. Next the cable is pulled up until the mahogany stick hits the bottom of the blasthole. In the case of nonbreakout holes, a steel weight may be substituted for the mahogany if the hole is wet.

Once all holes have been measured, it is necessary to estimate the volume of void available to determine if there is sufficient room for expansion of the rock from the blast. An area of the stope is selected which is equivalent to a volume of at least one-third the volume of the stope. This is called the slot and is the first section of the stope to be blasted. When cratering the slot, a 7-ft (2.1-m) average distance from rock pile to back of stope is the accepted minimum open space to allow successful blasting.

The next step is determining where to block the blasthole. Blocking is the process of securing two wedges at the desired location near the bottom of the blast- hole. The explosives are placed on top of the blocking. Angles of the blasthole determine where the hole is to be blocked. Generally accepted values for blocking heights are as follows (Andrews, 1988):

Hole Angle	Blocking Height (above hole bottom)
80–90°	4 ft (1.2 m)
57–79°	5 (1.5)
50–56°	6 (1.8)
Less than 50°	7 (2.1)

Loading the hole can begin now that blocking height and hole distance are available. A 12-grain primacord is measured according to desired blocking location, and a wooden wedge is tied to the primacord. The primacord is marked to correspond to the blocking location. Once the mark on the primacord reaches the hole collar, a second wedge is dropped down the hole and forms a friction lock with the first wedge at the desired blocking location. Next small rocks and drill cuttings are dropped down the hole to seal the space around the wedges. The end of the primacord is tied to a rock on the top sill until needed.

Holes in the slot are loaded with two 30-lb (13.6-kg) cartridges of water gel. The cartridges are cut lengthwise and one dropped down the hole. Next a sliding primer with the appropriate delay is assembled. The primacord is fed through the cord well of the booster, which is slid down the hole on the top of the explosives. The final 30 lb (13.6 kg) of explosives is placed in the hole on top of the booster. The completed powder column is approximately 4 ft (1.2 m) in height (Fig. 18.5.7).

Top stemming consists of a sand column of 7 ft (2.1 m). The sand stemming is topped off with 15 ft (4.5 m) of water. The break holes are shot first, and the remaining holes are capped in a concentric pattern. The slot will continue to be shot in this fashion until within 25 to 30 ft (7.5 to 9 m) of the top sill. This crown pillar will require double-decking two explosive charges in each hole. Delay time between the double decks is 25 to 50 ms.

MASS BLASTING. Typically, the initial slot in a VCR stope is one-third of the stope's volume. Once the initial slot has been blasted, the next step in the stope sequence is to prepare for the

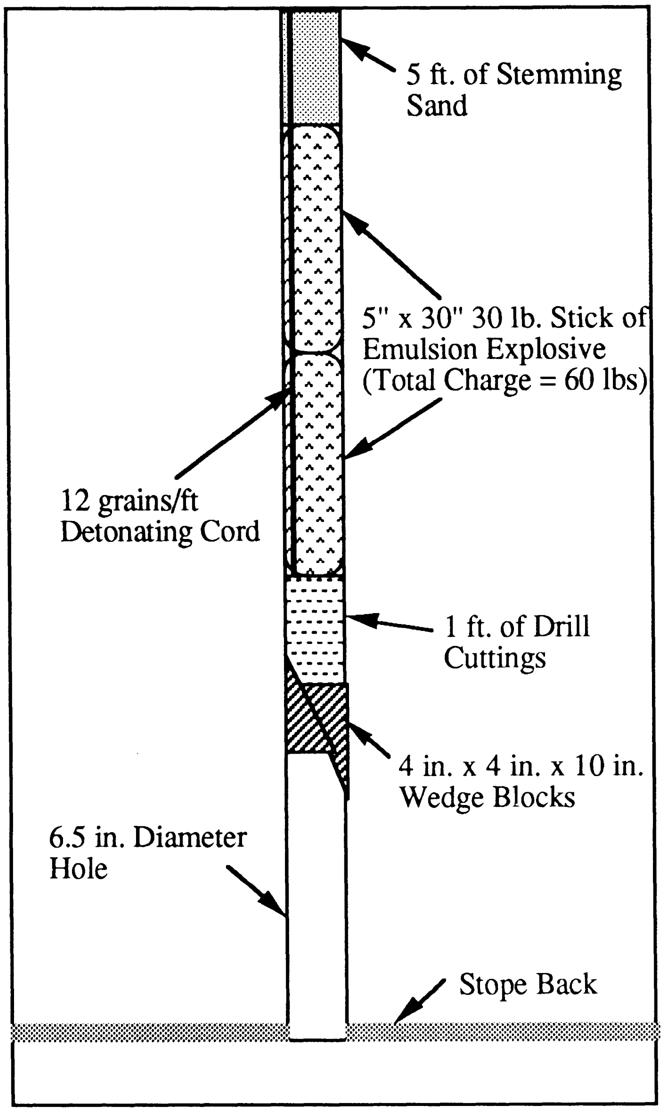

Fig. 18.5.7. Typical VCR cross section of a charged hole. Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 lb = 0.4536 kg.

mass blast. The mass blast consists of shooting all the remaining holes in the VCR stope.

All holes available for the mass blast should first be measured. This will aid the blasting technician in determining the number of decks to shoot per hole. The following is a guideline at the Homestake mine (Andrews, 1988):

Number of Decks	Length of Hole
1	30 ft (9 m) or less
2	31 to 55 ft (9.3 to 16.5 m)
3	56 to 130 ft (16.8 to 39 m)
4	131 ft (39.3 m) or more

To stay within the maximum per delay requirement of 500 lb (226 kg), a column height of ANFO cannot exceed 40 ft (12 m). Explosives with higher densities are adjusted accordingly. All holes in the mass blast are loaded with ANFO unless water is a problem in nonbreakout holes. To keep the powder factor minimized, some upper decks may be eliminated from the design due to closer spacing of holes at the collar. Some holes may be 2 ft (0.6 m) to 4 ft (1.2 m) apart at the collar. Powder factors are ideally less than 1.1 lb/ton (0.5 kg/t). Seven to ten feet (2.1 to 3 m) of sand are used to stem between the decks and at the collar of the hole.

After determining the number of decks and column height of explosive, proper down-hole delays must be designed. All surface and down-hole delays are nonelectric initiators. In holes where multiple decks are used, a delay of 50 ms is used between decks. On double-deck holes, the bottom deck is a 450-ms delay and the top deck is a 500-ms delay. Triple-deck holes have a 400-ms delay on the bottom deck progressing to a 500-ms delay on the upper deck. Single-deck holes are delayed using a 500-ms nonelectric initiator. This type of delay pattern ensures a free face for each hole to break to. All holes in the shot will be delayed down-hole with this pattern.

Proper selection of delay intervals between rows is important to minimize duplication of firing times and reduce vibration. Ideally, all surface delays should be activated (hot) prior to detonation of the blastholes. To minimize the risk of cutoffs, the surface trunkline is covered with sand.

MUCKING. Mucking of VCR rock is usually done with Wagner 2- and 3.5-yd^3 (1.5- and 2.7-m^3) diesel LHDs, Jarvis-Clark 2-yd^3 (1.5-m^3) electric LHDs, and one Wagner 3.5-yd^3 (2.7-m^3) electric LHD; all are used in scram mucking at various places in the mine. In isolated areas, stopes may be mucked with rubber-tired CAVO 320 overshot muckers. The LHDs dump their rock into an orepass. Installed at this location is a steel grizzly on a 22- by 36-in. (0.56- by 0.91-m) pattern. Any oversize rock that cannot pass through the grizzly is broken by rockpicks where available, or otherwise slanted grizzlies are developed on a 22-in. (559-mm) pattern to handle the oversize.

In conjunction with the LHD mucking the stope, a grizzly and a rockpick are used to eliminate oversize rocks before they reach the skipping system. Two types of rockpicks are in use, Kent Pneumatic hammers, and Allied Hydraulic hammers. Both are pedestal mounted adjacent to a grizzly at a LHD dump point. When a LHD unit is not available, CAVO 320s have been used to direct load haulage cars. CAVOs are effective in remote and/or poorly ventilated areas. However, LHD production is significantly better and used most often.

During initial blasting, selective mucking is done to draw down the broken ore and provide enough relief for continued blasting. Oftentimes, various scrams may be mucked to provide the proper relief for the next blast. This is another reason why measuring drillholes prior to a blast is important: it provides the production foreman a profile of the existing muck pile.

Sampling the scrams is done twice a shift by the LHD operator. The information obtained from these samples is used to determine if a stope is above cutoff grade. If not, especially near final clean down, management can decide if a stope has reached its economic limit. Sample information is also used in determining the unit cost of the stope.

To keep VCR stope productivities high during the mucking cycle, two areas are focused upon: equipment availability and scheduling mucking as often as possible. In major mining areas, equipment availability increases with the accessibility of stopes via the extensive ramp systems.

Extra care and precautions must be taken when the VCR stope reaches its final stages of mucking. As the stope is mucked out, more and more of the hanging wall is exposed (this is where high productivities can pay off by minimizing exposure time), thus increasing the potential for hanging wall failure. As scrams are opened up, the chance of large boulders rolling into scrams increases, which increases the danger to the LHD operator.

BACKFILLING. When ore drawdown from a stope is completed, the backfilling process begins. Several factors affect this process: availability of waste material, the need to dispose of mill tailings, time available for backfilling, and the availability of a crew to wall off the bottom sill and scrams.

Walls are constructed in bottom sill scrams or accesses to keep backfill material from intervening with any level or stoping operations in the area. Walls are usually built by forming an 18- or 24-in. (457- or 610-mm) thick concrete wall. The concrete is placed into the forms by pumps. Drain pipes with "weeping tiles" (devices that allow water to pass but not sand) are installed in each wall to allow the stope to drain.

Sand is usually run into the empty stope once the walls are built to bed the walls and protect the drain lines. Unless an empty stope is very inaccessible, waste rock will be dumped into it. If sufficient time and waste are available, the stope will be completely filled with waste rock, and then sand will be placed to fill the voids. If time is short, sand will be placed concurrently with waste dumping until the stope is full. When a stope is a panel of a larger stoping area, it may be necessary to add cement (20 : 1) to the sand to provide a more competent backfill to mine against. In panel sequencing, six months is usually scheduled for water to drain off and fill material to solidify.

Backfilling is done with both mine waste and mill trailings. A sand system is in place that delivers the sand portion of mill tailings to the mine for backfilling purposes, via 5-in. (127-mm) rubber-lined pipe. Waste is usually placed back into a stope by a LHD or by dumping cars into a borehole that feeds into the top of the mined out stope.

18.5.5.6 Productivity and Costs

VCR mining has evolved into the most productive stoping method employed at the Homestake mine. Direct mine labor for VCR mining had a productivity of 35.1 tons (31.8 t)/employee-shift in 1988. By comparison, this shows a 30.5% higher productivity than the next best alternative, mechanized cut and fill (MCF) mining, which has a productivity of 26.9 tons (24.4 t)/employee-shift. Currently, 55% of total mine production is from VCR stopes.

Productivity of a VCR stope is directly related to the stope size. Large stopes have the tonnages to support the high mucking rates for extended time and therefore tend to have higher productivity rates. A small-block VCR program is being utilized for stopes from 10,000 to 50,000 tons (9000 to 45,000 t) in size. These smaller VCR stopes have been marginally successful; however, very tight control of each phase is necessary.

The Homestake mine as a whole has been improving productivity since the 1970s. VCR mining has been a major part of the effort (Fig. 18.5.8). Overall Mine Department productivity has risen from 5.94 tons (5.39 t)/employee-shift in 1979 to 10.38 tons (9.42 t)/employee-shift in 1988. Remarkably, this productivity increase has coincided with a greater average depth of mining. VCR mining has a 48% cost advantage over MCF in direct mining costs. This is attributable to higher productivity and less ground control.

For a discussion of the cut and fill method, see Chapter 19.1.

18.5.5.7 Safety

A major advantage of VCR mining is safety, based on minimizing the exposure of miners to ground that has not been secured by rock bolting (Fig. 18.5.9). Most VCR stopes are extracted with only two areas needing to be rock bolted, the top and bottom sills. In any type of overhand cut and fill, 13 to 16 lifts would have to be rock bolted. The bolting cycle is reduced by as much as 88%, comparing VCR to conventional cut and fill operations. Properly designed VCR stopes have good drainage, good visibility, stable ground, smooth corners, and are well ventilated. This leads to higher productivities and safer operations.

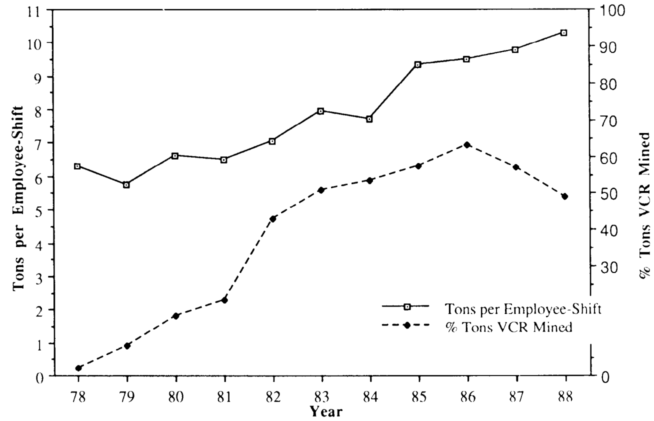

Fig. 18.5.8. Productivity per year compared to percentage of tons mined by VCR mining method at Homestake (1986 corresponds with replacement of remaining labor-intensive methods with mechanized cut and fill).

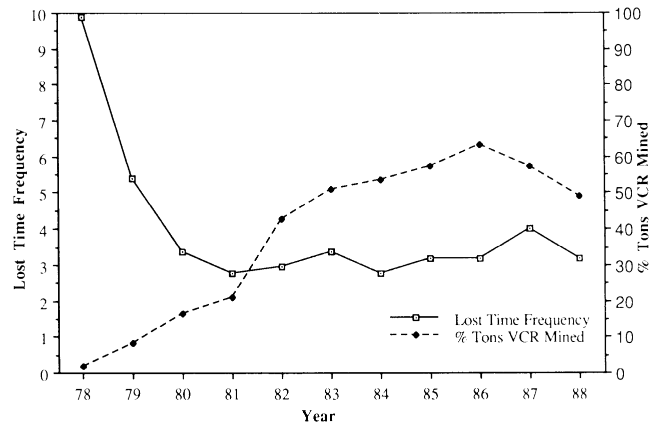

Fig. 18.5.9. Frequency of lost-time accidents compared to percentage of tons mined by VCR mining method at Homestake.

18.5.5.8 VCR Refinements

To adapt to a variety of different ore bodies, VCR mining at the Homestake mine is continually modified to accommodate each situation. Stopes have been successfully mined with VCR ranging from 15,000 to > 150,000 tons (14,000 to > 140,000 t).

Ongoing at the Homestake mine is research into improved drilling patterns, blasting techniques, etc. Tests are frequently done on different types of explosives. Stemming material has progressed from bagged sand carried in the top sill by hand to bulk sand that is delivered by LHDs or bobcats. Sampling programs on the drillhole cuttings (samples taken from cuttings of each 6 ft or 1.8 m of hole as rods are changed) were implemented to further delineate the ore bodies and in some instances, increase the actual tonnage of the VCR block. Drilling accuracy has improved along with refinement of top sill grid layout and locating of breakout holes on the bottom sill.

The VCR process at the Homestake mine has continually evolved with advances in technology and better understanding of crater blasting. Refinements have been made in stope design, equipment selection, blasting products, extraction systems, stope sequencing, and even in the backfilling process. Maintaining a cost-effective mining system demands continuing technical evolution.

Several imperatives of VCR mining can be generalized: 1) use as big and efficient equipment as possible to increase productivity; 2) adapt the mining plan according to the actual physical geometry of the ore body as it is accessed; 3) do not force VCR mining onto an ore body that has physical restraints limiting VCR applicability; and 4) integrate geology, engineering, supervision, and miners in planning and replanning of the stope.

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Section 19 Underground Mining: Supported Methods

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Chapter 19.0
INTRODUCTION

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19.0.1.1 Section Organization

In underground stopes that require more than minimal support, backfill is almost universally used. The term *cut and fill stoping* is used in a broad sense in this section, which describes underground mining methods requiring support. Support in stopes can be classified as local or short-term and general or long-term. Local support for the back of a stope, for example, can be accomplished with rock bolts or timber stulls. General support for the walls of the stope is provided by backfill. Local support provides a safe working place for the stope miners. General support, provided by backfill, preserves the access way to the stope and prevents large scale dilution of ore.

Supported methods are a set of cut and fill stoping methods using backfill for general support and various local support methods.

This section is subdivided into three chapters:

19.1 Cut and Fill Mining
19.2 Excavation Techniques
19.3 Backfilling Methods

19.0.1.2 Definitions

In order to assist the user of the *Handbook* to understand the terminology used in this section, definitions of certain key words follow. Terminology varies in different parts of the world, but the definitions presented will aid the reader who may be unfamiliar with the terminology or who has a different terminology.

Back—The roof or overhead rock surface of an underground opening.

Breast—The vertical end or face of a horizontal cut. The breast is a mining face that is generally as wide as the ore body and as high as the cut height.

Caps—Round or square timbers generally greater than 8 in. (203 mm) in diameter, placed perpendicular to the vein for wall and back support. Caps are part of a timber set.

Chute—The loading arrangement that utilizes gravity flow in moving broken rock from a higher level to a lower level. A gate is used to control flow. Chute often refers to the wooden tube extending upward into the stope.

Concrete Admixture—An additive to concrete or backfill that modifies the setting time or handling characteristics of the mix.

Crosscut—A nominally horizontal tunnel, generally driven at right angles to the strike of a vein.

Culm—Waste from coal cleaning operations.

Cut (and Fill)—That part of the cut and fill mining cycle that includes drilling, blasting, slushing, mucking, and ground support. A rapid excavation technique may be substituted for drilling and blasting.

Cut—The volume of the ore body that is mined and filled in one cut and fill mining cycle.

Cyclone—A conical-cylindrical device with no moving parts, used to size or dewater mill tailings (or other slurries). The slurry is fed peripherally near the top end. Coarse material moves downward on the outside to the apex, and fine material moves up the center to the vortex.

Drift—A nominally horizontal tunnel, generally driven parallel to or coincident with a vein.

Dwidag bolts—A German-made, coarsely threaded rock bolt. The threads are present the full length of the bolt.

Girts—A timber, as part of a timber set, that braces caps or posts. Girts are placed parallel to the vein.

Heading—The working face of a drift, crosscut, or ramp. In timber sets, a heading is a bundle of wooden boards placed between a cap and the wall rock.

Grizzly—A grating, usually made of steel bars or rails, with square or rectangular openings for sizing coarse ore and waste.

Load-Haul-Dump (LHD) units—Rubber-tired, front-end loader-transports designed for efficient tramming of broken rock.

Manway—The compartment of a raise or shaft that is equipped with ladders and landings.

Mucking—The process of loading and transporting broken ore from the point where it is excavated to an orepass or haulage vehicle. Mucking generally involves the use of a wheel or track-mounted digging vehicle such as an LHD or an air-powered overshot loader.

Posts—Timbers that are placed vertically. When used individually for roof support in a flatly dipping ore body, they serve the same functions as stulls. When part of a timber set, posts support caps vertically.

Pugmill—A horizontal bed mixer with a pair of counter-rotating axles that carry multiple paddles; also called a paddle-mixer or twin-screw mixer.

Raise—An opening driven upward generally on the vein. A raise is an accessway to a stope. A raise consists of one or two chutes, a manway, and a timberslide.

Ramp—An inclined underground tunnel generally driven downward with LHD equipment.

Slushing—The process of moving broken ore from the point where it is excavated to an orepass, using a winch-driven, rope-drawn scraper bucket.

Sparging—Clearing a pipeline or other container by blowing with air and water.

Square Set—A prismatic set of timber comprising a cap, a girt, and a post.

Stope—An opening in which excavation of ore takes place. A stope may be any shape.

Stulls—Generally round timbers less than 8 in. (203 mm) in diameter, placed perpendicular to the vein for wall support. In a flat-dipping deposit, the same members are called posts.

Timberslide—A compartment in a raise used for hoisting materials. The hoisting bucket used generally slides on the footwall.

Tramming—The haulage of broken ore with a train, truck, or LHD from a working place to an orepass.

Uppers—In stope mining, the method of drilling blastholes in the back to excavate the ore.

Chapter 19.1
CUT AND FILL STOPING

FRED W. BRACKEBUSCH

19.1.1 GENERAL

19.1.1.1 Description of Cut and Fill Cycle

A description of cut and fill mining is necessary to avoid confusion with similar practices that are part of other mining methods.

Cut and fill stoping is an underground mining method. Open cast (strip) mining actually uses a cut and fill cycle, but surface mining is not discussed in this section. In the narrowest terms, *cut and fill stoping* includes methods in which a single excavation pass (cut) is completed and backfilled before another cut is made. Open stoping methods often use backfill, but these are not considered as cut and fill mining methods. For example, vertical crater retreat (VCR) stopes are routinely backfilled at the Homestake mine in Lead, SD, after completion of mining the entire stope block.

Cut and fill mining is primarily utilized for steeply dipping vein deposits and large, irregularly-shaped deposits. However, backfill is being used to a certain extent in the flatly dipping South African gold reefs.

A listing of the work elements involved in completing one cut and fill mining cycle is useful in describing the method. The cycle begins with the first round after backfilling on the previous cycle. The drill-blast-mucking-ground support cycle is a subcycle of the cut and fill cycle. Table 19.1.1 lists typical major work elements for a cut and fill mining cycle.

A much more detailed list of work elements in the cut and fill mining cycle is presented in Table 19.1.2. One must consider the entire daily or shift cycle, including travel time, lunch, and the blasting schedule.

19.1.1.2 Variations of Cut and Fill Methods

Following is a short list and description of various methods that are considered to be variations of cut and fill methods.

OVERHAND CUT AND FILL STOPING

General—Horizontal cuts, 6 to 15 ft (1.8 to 4.6 m) in height, are mined advancing away from an access point. The excavated ore falls and rests on the backfill placed during the previous cut and fill cycle. As cut and fill cycles are completed, the stope is advanced upward. This means that an ore block must be developed from the bottom.

Unsupported Backs—In competent ore, the stope back may require little or no support. An occasional rock bolt or stull may be required to support a loosened slab.

Rock-bolted Backs—The most common method of back support in overhand stopes is the installation of rock bolts on a pattern after each blast. This minimizes the extent of unsupported back that men must work near or under. Rock bolts prove to be a nuisance when mining the next cut, however, because blastholes may be drilled into an installed rock bolt, and rock bolts have to be manually removed from the broken ore.

Timber Stull Back and Rib Support—If backs and ribs are in poor condition, and breasting must be used, installing timber sets after each blast provides support for the back and ribs, preventing dilution and increasing safety for the stope miners. Generally, the advance in a blast is the span of one timber set, or about 6 ft (1.8 m). Before mucking out, a timber set is installed

Table 19.1.1. Major Work Elements of the Cut and Fill Mining Cycle

1. Drilling
2. Blasting
3. Mucking/Slushing
4. Ground Support
5. Cleanout
6. Raising Up
7. Preparation for Backfill
8. Backfilling

Note: Work elements 1–4 are cycled until the cut excavation is completed.

Table 19.1.2. Detailed List of Work Elements in the Cut and Fill Mining Cycle

Shift Cycle Elements
Travel time
Communications/supervision
Receive tools and supplies
Workplace inspection
Work on elements in progress
Wait for blasting time
Wait for equipment repair
Lunch

Mining Cycle Elements
Slusher setup
Equipment inspection/servicing
Slushing/mucking
Drill setup
Rock bolting
Installing timber supports
Drilling blastholes
Drill teardown
Clean blastholes
Make primers
Charge blastholes
Fire round

Cleanout/Raise-up Elements
Slushing/mucking
Bulkhead construction
Place fill delivery pipeline
Drill setup
Drill blastholes in raise
Make primers
Charge blastholes
Fire round
Place timber sets
Raise crib chutes
Raise equipment to next floor
Backfilling

by first heading in a cap from wall to wall at the proper location. The headings allow the cap to be tightly installed. Another function of the headings is to absorb wall closure without causing the cap to be crushed in compression. Because wood fibers in the heading are at right angles to the pressure, and because wood is compressible when confined at right angles to the fibers, the cap is protected until wall closure has compressed the headings to the fullest extent. A bulkhead is built from cap to cap to support the back. The bulkhead should contact the back tightly to prevent caving. Posts are placed under the cap near each heading and supported from a cap from the previous cut and fill cycle. Girts are placed from cap to cap and post to post along the walls to provide longitudinal support. Space between posts, at right angles to the vein, must be allowed for clearance of mucking equipment. Often when scrapers are used for mucking, boards are nailed on the lower part of posts along the walls to prevent broken ore from getting lodged in the spaces between posts along the walls, thus reducing the cleanout effort. The bulkhead from the previous cut becomes the floor for the current cut.

Square Set Timber Support—The square set stoping method is used for wide stopes where caps cannot reach from wall to wall. Square sets are prismatic timber sets, generally made from sawn timbers with mortise and tenon joints.

POST-PILLAR STOPING. The post-pillar method is used for wide ore bodies with a greater vertical extent than can be mined with the room and pillar method. As each horizontal cut is mined and backfilled, pillars are maintained to support the back. Pillars become tall and slender, but the backfill prevents buckling. Ore competency should be good to prevent pillar and back failure. Post-pillar can be classified as an overhand stoping method.

UNDERCUT AND FILL STOPING. Undercut and fill, or underhand cut and fill, mining consists of mining horizontal cuts as is done in overhand mining, but the direction of the stoping is downward rather than upward. As each horizontal cut is mined, it is backfilled with cemented backfill. Mining of the next cut proceeds beneath the cemented backfill.

DRIFT AND FILL STOPING. Drift and fill is a modern replacement for square set stoping. For wide ore bodies with moderate or poor ore competency, a horizontal cut is mined as a series of drifts. When a drift is mined, it is backfilled with cemented backfill, and another drift can be excavated beside the backfilled drift. The cemented backfill provides the back support adjacent to drifts.

19.1.1.3 Excavation Methods

BACK STOPING (UPPERS). When back stoping, enough open space is allowed between the backfill surface and the stope back to drill vertical or steeply inclined blastholes into the stope back in an upward direction. Back stoping can be done only with overhand or post-pillar stoping, with rock-bolted or unsupported backs. Ribs should be competent to avoid dilution and safety problems.

BREASTING-DOWN (UP). This excavation method employs a vertical working face and horizontal or slightly inclined blastholes. Backfill must be placed to nearly fill the void of the preceding cut, allowing only enough space (1 to 3 ft, or 0.3 to 1 m) for expansion space for effective blasting. After each round is blasted, mucking and ground support activities are completed. Breasting is generally used in poor rock conditions often with timber support. In undercut and fill stoping, rounds are breasted upward instead of downward.

DRIFTING. On the first cut or on any cut where breaking room is not provided, drift rounds are blasted using a burn cut or other type of drifting cut.

BENCHING. Benching or the drilling of vertical holes from the top down can be done in undercut and fill stoping.

MECHANICAL. Mechanical excavation methods could be utilized with any of the stoping methods. At this time, the use of mechanical stoping technology is not well developed.

19.1.1.4 Environmental Regulations

In the United States, mine backfilling operations are regulated as Class V Injection Wells under the Safe Drinking Water Act, *CFR* Title 40, parts 124, 144–146 (Anon., 1989). Mine backfilling is subject to inventory, assessment, and reporting requirements. The US Environmental Protection Agency was required to report to Congress on backfilling practices and environmental effects, which was done in September 1987. At present (1988), Underground Injection Control (UIC) regulations are in a confused state and subject to change.

19.1.2 HISTORY

The cut and fill mining method evolved from open stoping with occasional stulls for support (*stull stoping*), to fully timbered square set stopes with backfill (*square set stoping*), and finally to the general abandonment of timber with the use of rock-bolt systems and consolidated backfill.

Gardner and Vandenburg (1933) describe the reasons for development of square set stoping:

"Prior to 1860, open stope methods with stulls for casual support accounted for most of the mineral production from underground mines. Attempts to mine large ore bodies that required artificial support resulted in disastrous cave-ins and loss of life. The art of mining received a tremendous impetus following the introduction of methods for mining the bonanza ore body which was discovered in 1860 in the Ophir mine on the Comstock lode, Virginia City, NV. The ore body was phenomenally rich and eventually proved to be 122 m (400 ft) high, 27 m (90 ft) wide, and 98 m (320 ft) long. The mining of this large high-grade ore body taxed the best engineering talent of the time; the problem was finally solved by Philip Deidesheimer.

"According to Eliot Lord in 1882:

'At the 50-foot level the vein of black sulphurets was only 3 or 4 feet thick, and could be readily extracted through a drift along its line, propping up the walls and roof when necessary by simple uprights and cap-posts. As the ledge descended the sulphuret vein grew broader until, at the depth of 175 feet, it was 65 feet in width, and the miners were at a loss how to proceed, for the ore was so soft and crumbling that pillars could not be used to support the roof as in coal mines. They spliced timbers together to hold up the caving ground; but these jointed props were too weak and ill-supported to withstand the pressure upon them, and were constantly broken and thrown out of place. The dilemma was a curious one. Surrounded by riches, they were yet unable to carry them off and their mass of black sulphurets bade fair to become a white elephant on their hands. The Ophir Company began to wish themselves less fortunate, as their miners narrowly escaped burial day after day in their attempts to stope out the ore.

'A young mining engineer, Philip Deidesheimer, was in charge of a quartz mine in Georgetown, El Dorado County, Calif., in the autumn of 1860, when this serious check to the development of the lode occurred. At the request of William F. Babcock, a trustee of the Ophir Company, Mr. Deidesheimer left his California mine and crossed the mountains with a letter from the directors of the Ophir Company, authorizing him to inspect the workings of their mines and make such changes in the method of timbering as should seem to him expedient. After examining the vein he designed, in the course of a few weeks, a system of timbering which proved to be exactly adapted to the requirements of the work. Experiments which he had previously made in California gravel and quartz mines furnished the outline of his plan. This was to frame timbers together in rectangular sets, each set being composed of a square base, placed horizontally, formed of four timbers, sills, and crosspieces from 4 to 6 feet long, surmounted at the corners by four posts from 6 to 7 feet high, and capped by a framework similar to the base. The cap pieces forming the top of any set were at the same time the sills or base of the next set above. These sets could readily be extended to any required height and over any given area, forming a series of horizontal floors, built up from the bottom sets like the successive stories of a house. The spaces between the timbers were filled with waste rock or with wooden braces, forming a solid cube whenever the maximum degree of firmness was desired. By adjustments of these sets the ore bodies along the line of the lode were extracted with comparative ease and security.' "

Note that waste backfilling had already been introduced in the Ophir mine. Gardner and Vandenburg (1933) go on to describe the general adoption of waste backfill in square set stoping.

"In the earlier years of square set mining the erroneous idea was held that timber in heavy ground could support the mass of rock between the back of the opening and the surface. Round timbers 457 to 914 mm (18 to 36 in.) diameter and square timbers of 457 x 457 to 610 x 610-mm (18 x 18 to 24 x 24 in.) size were employed in square set stopes. Cave-ins were frequent occurrences on the Comstock, in Butte mines, on the Mother lode, and on the Homestake belt, in stopes where filling was not used. As it was recognized that any timbering short of solid cribbing could not withstand the slow movements of heavy ground, the sizes of square set stopes were restricted, and the timbering was followed closely by waste filling. This practice made possible the use of smaller timbers."

Waste to be used for backfill in stopes could originate in mine development headings. Often this source was not adequate, or timing of development waste availability did not coincide with the time of need of backfill in stopes. Thus it was common practice to drive headings into the walls of the stope simply to generate waste that could be easily moved into the stope for backfill.

In the 1940s and 1950s, hydraulic sandfilling was introduced largely to replace waste backfilling in stopes. Sand backfill was less labor intensive, and a reliable supply of sand could be produced by the milling operation. The use of timber declined from the 1950s to the present because of the adoption of rock-bolting systems and consolidated backfill.

In the late 1980s, square set stoping was not practiced at any major mine in the United States. Timber support is still used for access raises and for ore passes within backfilled stopes.

The mechanization of overhand cut and fill stopes with load-haul-dump (LHD) equipment has made it convenient to dispose of mine development waste as backfill in stopes. When broken waste is used as backfill, it is generally topped with hydraulic sandfill to fill the voids and provide a flat working surface for the succeeding cut.

19.1.3 REASONS FOR SELECTING CUT AND FILL METHODS

19.1.3.1 Ore Body Geometry

The shape of an ore body is important to the selection of a mining method. Cut and fill methods are almost always considered for steeply dipping veins. Most cut and fill stoping and development methods rely to some extent on gravity flow of broken rock, and thus veins with dips less than the angle of repose (< 45°) of broken rock must be mined using footwall or hanging wall development.

Irregular ore bodies, such as replacement ore bodies, are often candidates for cut and fill mining.

Whenever mining requires manned stopes with potentially high backs and walls, cut and fill mining is used to improve safety and wall support by limiting the working height of a stope. Backfill provides a working floor at a convenient elevation for mining activities.

19.1.3.2 Mining Selectivity/Grade Control

Because of the limited open volume of cut and fill stopes and the wall supporting function of backfill, very irregular surfaces can be followed precisely. Greater selectivity results in a higher-grade ore product, which is important to the economics of a mining operation.

19.1.3.3 Prevention of Caving and Dilution

The wall-supporting function of backfill aids in the prevention of massive wall failures in stopes. Limited open spans help prevent caving failure in the stope back. Dilution in overhand cut and fill mining frequently occurs when ore gets mixed with backfill during the mining process. In timbered stopes, ore can be lost in the nooks and crannies and beneath the timber floor. In waste-filled stopes, fine particles of ore become lost in the voids between waste fragments. Backfill can be cemented to minimize loss of ore values in the backfill. In undercut and fill mining, ore loss in backfill generally does not occur, but backfill can fall out of the back, diluting the ore.

19.1.3.4 Rock Quality Designation

The lower the rock quality designation of the ore and the quality of the enclosing wall rock, the more likely it is that cut and fill mining will be the favored method.

19.1.3.5 Prevention of Subsidence

Subsidence from caving methods and long-term failure of pillars in abandoned mines has grown to be a very expensive problem because of damage to surface structures. The use of backfill minimizes the amount of subsidence.

19.1.3.6 Waste Disposal

Where surface waste disposal is limited due to environmental factors or space limitations, backfilling can be utilized. Note that backfilling can serve this purpose not only for cut and fill mining but also for open stoping methods.

19.1.3.7 Stress Control in Bursting Mines

The stress near a longwall face or in pillars is reduced by tightly backfilling mined-out areas. Backfilling as soon as practicable after excavation is desirable. Backfill is compressed by the walls or roof until significant resisting forces develop. The strength of backfill is not necessarily as important to resist closure as is backfill density. The incompressibility of solids becomes the dominant factor in confined situations.

19.1.3.8 Ventilation

Ventilation can be effectively routed to the working face in cut and fill mining operations. The reduction of open space tends to reduce ventilation (quantity) requirements. Methane or radon gas emissions from mined-out areas can be minimized by backfilling. The disposal of combustible materials, such as wood, can be safely done when backfilling. The amount of wood exposed to air, and thus the risk of fire, is reduced when using cut and fill mining.

19.1.3.9 Cost

Cost is a primary consideration in the choice of a mining method (see 19.1.5). Cut and fill mining will generally be more expensive than open stoping. However, the previous criteria may rule out all methods except cut and fill, or the reduction in dilution may be financially attractive and justifiable.

19.1.4 CASE STUDIES

19.1.4.1 Overhand Cut and Fill Stoping

1. Sunshine mine—Kellogg, ID

The Sunshine mine uses a variety of overhand cut and fill mining methods, depending on ground conditions. Located in the Coeur d'Alene mining district of northern Idaho, the Sunshine mine contains narrow, steeply dipping veins containing copper-silver ore that cut Proterozoic quartzites and argillites. Mining width averages 5 ft (1.5 m), and the dips of the veins average about 70°. Ore shoots have short strike lengths, with most stopes being 100 ft (30 m) or less in total length. Mining is conducted from about 3000 to 5000 ft (900 to 1500 m) below the surface. The mine production rate is about 1000 tons (900 t) per day.

Where vein and wallrock are competent, mostly in the upper part of the mine, back stoping is conducted. Six-foot (1.8-m) high horizontal cuts are drilled with hand-held stoper drills using 1 1/2-in. (38-mm) bits. When back stoping, two or more raises in a stope are desirable because it is necessary to blast over the raises, thus blocking them temporarily. Back support consists of occasional rock bolts. About 7 ft (2.1 m) of vertical open space is required between the backfill and the stope back in order to operate the stoper drills. Most access raises are blind, that is, they do not extend above the elevation of the stope and are advanced concurrently with the stope. Access raises have three compartments: two orepasses and one timber slide/manway. Raises require an opening 9 ft wide by 20 ft long (2.7 m by 6.1 m), thus causing dilution because of the narrow vein width. Four round timber caps are used in sets 6 ft (1.8 m) on centers for raise support. Secondary access raises for a stope do not need orepasses, thus allowing a smaller raise to be used for access only.

In the deeper parts of the mine or where ground is poor, breasting-down stoping is practiced. Horizontal cuts are 9 ft high (2.7 m). Breast rounds are usually 6 or 8 ft (1.8 or 2.4 m) in length. Rock bolts and some stulls are used for back and wall support. Raises are similar to back stopes, but only one raise per stope is needed.

Average stoping productivity is about 10 tons (9 t) per employee-shift.

2. Homestake mine—Lead, SD

Located in the Black Hills of western South Dakota, the Homestake mine contains fairly wide, steeply dipping bodies of gold ore enclosed in metamorphic rocks. Mechanized overhand cut and fill stoping is one of the mining methods in use at the Homestake mine, where stoping is done at depths greater than 7000 ft (2100 m) beneath the surface (Smith, 1987). Cut and fill stoping is used in preference to VCR when (1) the dip is flat or variable, (2) the ore body is narrow or irregular, or (3) ground conditions are relatively poor for open stoping. (For a discussion of VCR mining, see Chapter 18.5.) Access to cut and fill stopes is by footwall ramps driven parallel to ore-body strike. Orepass boreholes are located in the footwall. Ventilation and waste transfer boreholes are located in the ore body. Horizontal cuts 13.5 ft (4.1 m) in height are excavated by breasting using pneumatic drill jumbos. Mine development waste is used for backfill to reduce waste hoisting. The mine waste is topped with hydraulic sandfill to provide a flat surface for mucking. Mucking is done with diesel-powered LHD equipment. Back support consists of cable bolts 60 ft (18 m) in length, installed on 10-ft (3.0-m) centers and short split-set and point-anchor bolts, 5 to 8 ft (1.5 to 2.4 m) in length, installed in a standard fixed pattern. Cable bolts need only be installed every three or four cuts, but the short rock bolts are installed every cut.

Average stoping productivity is about 29 tons (26.3 t) per employee-shift.

3. Galena mine—Wallace, ID

The Galena mine, located in the Coeur d'Alene mining district of northern Idaho, also uses overhand cut and fill methods. Narrow, steeply dipping (65 to 90°) veins containing copper-silver ore cut Proterozoic quartzites. Mining width is 5 to 6 ft (1.5 to 1.8 m) with some exceptions where vein width is as much as 12 ft (3.7 m). Ore shoots generally have short strike lengths, and most stopes are less than 150 ft (46 m) in length. Stoping is conducted from 3400 to 4900 ft (1030 to 1500 m) beneath the surface. The level interval is 300 feet (90 m). The mine production is 800 tons (725 t) per day.

Breasting down is the predominant excavation method at the Galena mine. Cuts are 9 ft (2.7 m) in height. The mining equipment most commonly used includes jackleg drills and slushers. In the occasional wide stope, an air-powered Cavo mucker may be used. Hydraulically placed mill tailings sand is used for backfill.

Average stoping productivity is 18 tons (16.3 t) per employee-shift.

19.1.4.2 Post-pillar Stoping

1. Blackcloud mine—Leadville, CO

The Blackcloud mine, located in the Leadville district, uses the post-pillar method or, as termed by the mine operators, random room and pillar/cut and fill stoping. The ore bodies are irregular replacement bodies of pyrite, sphalerite, and galena in dolomite near an igneous stock. A shale member forms the top limit of the ore bodies. Ore-body height ranges from 150 to 200 ft (46 to 61 m). The access shaft is 1500 ft (460 m) in depth. The mine production rate is 800 tons (725 t) per day.

Stoping is done using the breast-down excavation method with jackleg drills. Rock bolts are used for back support. Mucking is done with small electric LHD equipment and air-powered Cavo muckers. Pillars are left on a random basis, maximizing the use of waste or low-grade ore for supporting pillars. Pillars of ore are 10 to 12 ft (3.0 to 3.7 m) square. Hydraulically placed mill tailings support the slender pillars. The backfill has a high concentration of pyrite, which makes a dense fill.

Average stoping productivity is over 40 tons (36 t) per employee-shift.

19.1.4.3 Undercut and Fill Stoping

1. Lucky Friday mine—Mullan, ID

The Lucky Friday mine, also located in Idaho's Coeur d'Alene district, uses an undercut and fill method with longwall geometry. A steeply dipping vein containing lead-silver ore cuts Proterozoic quartzites. The mining width averages 7 ft (2.1 m), and the vein dips about 80°. A single vein with only minor branching has a strike length of about 2000 ft (600 m). Mining operations are being conducted about 5000 ft (1500 m) beneath the surface. Because of rock-burst problems, longwall geometry was adopted to eliminate mining of remnants.

A longwall geometry is used in laying out the mine, in that four or five stopes, all more or less at the same elevation, encompass the strike length of the ore body. All mine production, about 700 tons (635 t) per day, comes from these stopes, except for some remaining remnants in upper portions of the mine where overhand cut and fill stoping is used. Access to the underhand stopes is via ramps and sublevels located in the footwall quartzites. LHD equipment is used for mucking to rock passes that are also located in the footwall. Stoping is conducted beneath cemented backfill. Because wall closure tends to compress and fracture the backfill as mining progresses, a safety net of wire fabric is used to retain broken fragments of backfill. The stopes are about 500 ft (150 m) in length. Hand-held jackleg drills are used for drilling 1.5-in. (38-mm) diameter blastholes. The ore is breasted up because the backfill-ore interface acts as a free face for blasting. When a horizontal cut, which is 10 ft (3.1 m) in height, is completely mined, the stope is filled as completely as possible with a stiff, paste-like backfill consisting of mill tailings and cement.

19.1.4.4 Drift and Fill Stoping

1. Cannon mine—Wenatchee, WA

The mining method used for the B-North ore body at the Cannon mine does not fit exactly into the classifications used in this *Handbook.* The method, however, best matches overhand drift and fill stoping. Excavation methods include drifting, bench drilling or end slicing, and VCR raising. These methods are more commonly used in unsupported stopes. The B-North ore body is an epithermal gold deposit in Eocene volcanic rocks. Its approximate size is 400 ft (122 m) wide by 800 ft (244 m) long by 200 ft (61 m) high. The ore body is overlain by about 200 ft (61 m) of unconsolidated rocks. The mine has a circular shaft 620 ft (189 m) deep and a ramp for access. The production rate is 1500 tons (1360 t) per day.

Level development is done within the ore body because of poor wall rock competency. Drifts are driven on strike at vertical intervals of about 65 ft (20 m). Crosscuts 24 ft (7 m) wide are driven from the drifts on approximately 48-ft (15-m) centers to the footwall and hanging wall. Crosscuts from an upper level are directly above crosscuts of lower levels. Raises are driven between the ends of corresponding crosscuts. The raises serve as free faces for end slice blasting, as backfill conduits, and as ventilation ways.

Mining commences on the lowest developed blocks and proceeds upward in a generally overhand direction. Blastholes are drilled downward between corresponding crosscuts. Mucking is done with LHD equipment operating from the lower level. A cut is actually a slot about 24 ft (7 m) wide by 65 ft (20 m) high, driven transversely to the strike and extending from the footwall to the hanging wall. The slot or cut is backfilled with a cemented mixture of crushed rock and natural sands and silts. After backfilling of the slot, mining of the next higher slot can commence. Other slots on the same level are separated by slot pillars as defined by the crosscut development pattern. Of course, many slots can be mined concurrently on the same level.

As the mining proceeds on a level, alternating slots of backfill and ore result. The remaining slot pillars of ore are then mined in much the same fashion as the primary slots, but are backfilled with uncemented material including mine development waste.

Productivity for the entire mine department of 84 workers plus mine staff is 28 tons (25.4 t) per employee-shift.

2. Kristineberg mine—Sweden

The Kristineberg metal mine located in north-central Sweden uses overhand drift and fill and overhand cut and fill stoping methods. The deposits are enclosed in altered Precambrian volcanic and sedimentary rocks. Rock types are sericite/chlorite schists and quartzites with talc-rich rocks in the hanging wall of the stopes. The ore zones dip 45 to 70° and are as wide as 65 ft (20 m).

Overhand drift and fill stoping is used where the ore zone is wider than 16 to 20 ft (5 to 6 m), and overhand cut and fill is used in narrower parts of the ore zone or in good ground conditions. Backs are supported with cement-grouted rock bolts 9 ft (2.7 m) in length. Horizontal cuts are 13 ft (4 m) high. Cemented sandfill is used for drift and fill mining, and uncemented sandfill is used for cut and fill. Fill is transported hydraulically.

Footwall ramps provide access to the stopes for LHD equipment.

Stoping productivity exceeds 20 tons (18 t) per employee-shift.

3. Fukasawa mine—Japan

The Fukasawa mine in the Hanaoka area of northern Japan is an operation using the underhand drift and fill mining method. The base metal deposit being mined is a Kuroko complex sulfide volcanogenic ore body. The ore body is a flat-lying tabular-shaped deposit. The country rocks are clayey, soft, and fragile. Even though the depth is only 1300 ft (400 m), pressure is high because of the incompetent rock.

An inclined drift is first driven within the ore body. Horizontal gate drifts are driven from the inclined drift on 10-ft (3.1-m) vertical centers, thus dividing the ore body into stoping blocks or horizontal slices. From the gate drift, crosscuts are driven to the limit of the ore body in the footwall or hanging wall. As each crosscut is completed, it is backfilled before the adjacent crosscut is driven. Backfill is cemented to provide an artificial roof for the next horizontal slice to be mined beneath the backfill.

LHDs are used for mining, and the average stoping productivity is 25 tons (23 t) per employee-shift.

4. Hanaoka mine—Japan

The Hanaoka mine in northern Japan produces 2200 tons (2000 t) per day from the Matsumine Kuroko-type ore body. The stratabound deposit is 650 to 1300 ft (200 to 400 m) deep enclosed in volcanic pyroclastic rocks, which have been altered to clay. Ground pressure is heavy, and the ore body is irregular with zoning of economic minerals. The ore is relatively high grade (values of Au, Ag, Pb, Zn, and Cu).

A gate drift is driven between two chutes to define a horizontal slice 10 ft (3.1 m) high. Crosscuts are driven at right angles to the gate drift to the extremity of the ore body. A crosscut is backfilled with cemented fill before commencing an adjacent crosscut. When a horizontal slice is completely mined, the next lower slice is mined but with crosscuts at right angles to those in the previous slice. Only a few posts are required to support the backfill of the previous cut.

A stoping crew of three men can complete two rounds per shift during the excavation cycle, or a productivity of about 80 tons (73 t) per employee-shift.

19.1.5 MINING COSTS

19.1.5.1 Development

Development can be subdivided into primary development and secondary development. Secondary development is also known as *stope preparation.* Primary development includes:

1. Main access shaft or ramp
2. Secondary/escape shaft or ramp

3. Level development
4. Pumprooms, hoistrooms, and miscellaneous excavations

Secondary development includes:

1. Sublevels
2. Raises on vein
3. Footwall ramps
4. Raise preparations
5. Access crosscuts
6. Orepasses
7. All excavations in ore body prior to stoping

Development costs must be estimated for a given mine because each mine has unique characteristics. Also there are several development alternatives for each mine that should be evaluated and compared. The US Bureau of Mines *Cost Estimating System Handbook* (Anon., 1987), is a useful estimating guide for mine development. Cost case studies are provided in *Mineral Industry Costs* (Hoskins, 1982).

Generally, when a new mine is being developed, mine development is considered as a capital cost and is included in the return on investment calculations. The point to remember is that both primary and secondary development are investments and should be treated that way from an engineering point of view, no matter what accounting practices are actually used by the company. Primary development is usually concentrated during the initial development phase of a mine. Secondary development tends to occur uniformly throughout the operating life of a mine. Hence there is a tendency to consider secondary development as an operating cost, and benefits are not matched against investments to determine if investment criteria are met.

As an example of the capital nature of secondary development, consider the case of raise preparation. The purpose of raise preparation in vein mining is to support the wooden structure of a raise that provides access to a stope. However, raise preparation is made complex by the provision for chute lips and gates to control gravity flow of broken ore from the stope to train or truck haulage. In heavy ground, the raise preparation structure requires special timbering techniques in order to survive the duration of stoping operations, and repair costs usually mount with time. Since the primary reason that complex raise preparation is needed is to allow gravity flow of ore into train cars or trucks, one should consider much of the cost of raise preparation as an investment to make haulage efficient. Is that a worthwhile investment? Would it be better to use a simpler drawpoint technique with LHD mucking and loading of trucks or train cars?

Primary development, such as shaft sinking, requires special equipment and expertise. It is often advisable to contract the shaft sinking rather than acquire the equipment and expertise to do it in-house. In support of this point is the fact that most shaft sinking is done early in the life of a project when mine management has not been fully tested as an operating organization. After the initial shaft sinking activity, further sinking operations are often not needed for many years.

19.1.5.2 Exploitation

The individual mining methods that are included in the category of cut and fill mining are diverse, and so are the costs. Labor productivity is the most important independent variable affecting mining costs. The *Cost Estimating System Handbook* (Anon., 1987) is also a useful guide in preliminary studies for estimating exploitation costs for cut and fill methods.

Labor productivity for cut and fill stoping varies from 10 to 30 tons (9 to 27 t) per employee-shift in the case studies listed in the previous discussion (19.1.4). Naturally, mechanized stoping has a higher productivity than less mechanized methods. Underhand cut and fill and underhand drift and fill methods have high productivity if the mechanization is extensive.

The recommended method to estimate exploitation costs for a specific cut and fill mining operation is to build a spreadsheet model. The reader is referred to the various pertinent sections in this *Handbook* for aid in estimating costs.

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Chapter 19.2
EXCAVATION TECHNIQUES

WALTER A. PARONI

19.2.1 OVERHAND CUT AND FILL MINING

19.2.1.1 Description of Methods

BREAST STOPING. *Breast stoping* is a method of extracting a block of ore by mining successive horizontal slices (cuts) 5 to 15 ft (1.5 to 4.6 m) thick from the back of the stope. Mining progresses upwards from a lower level to an upper level. Horizontal holes 5 to 15 ft (1.5 to 4.6 m) deep are drilled in the breast. After the ore has been blasted the walls and back are scaled, and temporarily supported by timber and/or rock bolts. The broken ore is then removed by use of slushers, or load-haul-dump (LHD) equipment in mechanized stopes, through orepasses to the level below (Fig. 19.2.1) (Rausch and Stitzer, 1973; White, 1984). Orepasses are usually spaced 50 to 150 ft (15 to 46 m) apart, and mining from one orepass to another constitutes a cut. When the cut is completed, the orepasses are extended upward the height of the ore removed, the stope is backfilled, and another cut is mined.

POST-PILLAR STOPING. *Post-pillar stoping* is used to mine wide ore bodies that have a great vertical extent. Horizontal cuts are made by breasting-down with jackleg drills. Sill pillars are left on a random basis, and where possible, low-grade and waste areas are used for the pillars. Each cut is backfilled, and the pillars support the back. As the mining progresses upward, the backfill prevents the tall slender pillars from buckling (Fig. 19.2.2).

DRIFT AND FILL STOPING. *Drift and fill stoping* is used to mine wide ore bodies, exceeding 16 to 20 ft (5 to 6 m) in width, where the competency of the ore is moderate to poor. The mining block is divided into series of parallel drifts with vertical walls. A heading is driven along the hanging wall contact for access to the stopes. The mining starts by stoping a drift with a 12 ft (3.7 m) high cut. The stoped area is then filled with cemented sand fill, which provides the back support for the next drift. Then another drift is excavated adjacent to the backfilled drift. This sequence is repeated until the entire block is mined out on that level (Fig. 19.2.3).

A variation of this method is the *transverse stope and pillars* method (Waterland et al., 1982). The ore body is mined by dividing it into transverse stopes and intervening rib pillars. During the mining of the stopes, the rib pillars serve as abutments for the stope back and provide some overall support to the hanging wall. The stopes are filled with cemented sand fill, and the rib pillars provide partial support, or resistance, to closure between the hanging wall and footwall. Extraction of the rib pillars is usually done by undercut and fill methods.

Another variation is called *concrete pillar stoping*. This method was developed by the Outokumpu Company of Finland, and it has been the main stoping method at their Keretti mine since 1954 (Murray et al., 1982). The mining block is divided alternately into 20-ft (6-m) and 26-ft (7-m) wide parallel rooms with vertical walls. The 20-ft (6-m) rooms are stoped first, and they are backfilled with concrete. Then the 26-ft (7-m) wide ore pillars are stoped, and they are filled with classified tailings and natural gravel.

BACK STOPING. *Back stoping* is very similar to breast stoping in that a horizontal slice of ore 3 to 10 ft (0.9 to 3.1 m) thick is removed from the back of the stope. The major difference is that the blastholes are drilled vertically, or at a steep angle (50 to 70°), instead of horizontally. Usually many holes are drilled, for example, from one orepass to another, and the entire series of holes are loaded and blasted at the same time.

After the ore has been blasted, the walls and back are scaled and temporarily supported by stulls and/or rock bolts. The broken ore is then removed by use of slushers, or LHD equipment in mechanized stopes, through orepasses to the level below (Fig. 19.2.4).

The orepasses are usually spaced farther apart in back stoping, 100 to 200 ft (30 to 61 m) spacing, depending upon how straight the vein is. The orepasses can be either in the backfill or in the walls of the stope.

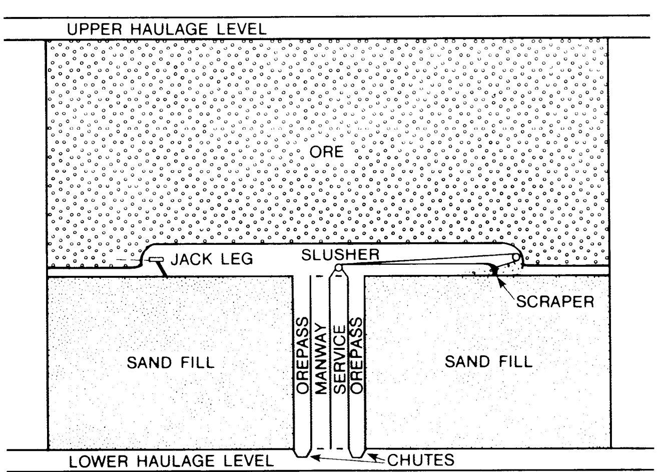

Fig. 19.2.1. Longitudinal section of a typical non-mechanized breast stope.

19.2.1.2 Application to Various Geometries

Overhand cut and fill mining is applicable to ore bodies that have considerable vertical extent and dip 50 to 90°. The method is suitable to ore bodies that meet one or more of the following conditions:

1. Require selective mining
2. Have weak walls
3. Have poor continuity
4. Other mining methods would produce excessive dilution

Post-pillar stoping is applicable to wide ore bodies that have a greater vertical extent than can be mined using the room and pillar method. The competency of the ore should be good in order to prevent failure of the pillar and the back.

Drift and fill stoping is applicable to ore bodies that exceed 16 to 20 ft (5 to 6 m) in width, and where the competency of the ore is moderate to poor.

Back stoping is particularly suited to steeply dipping, narrow-vein ore bodies.

Extensive sampling during each mining cycle permits the operator to change the size and shape of the stope as conditions

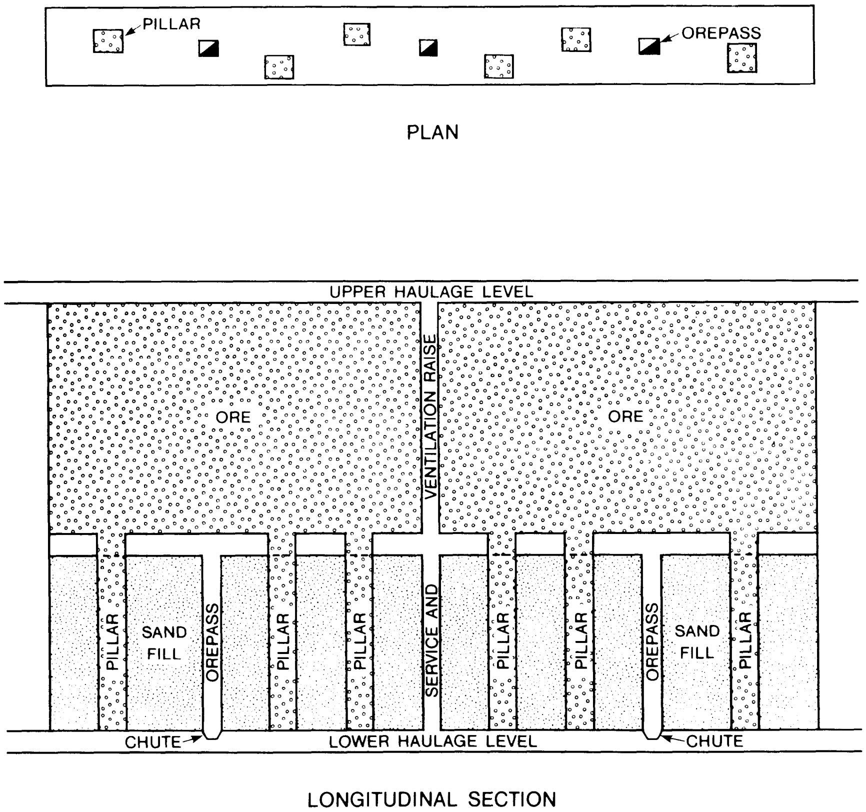

Fig. 19.2.2. Post-pillar stoping.

change. The major limitation is that the ore value must be high enough to support the high cost of overhand cut and fill mining.

19.2.1.3 Development

One common method of developing narrow veins is to drive crosscuts from a lateral in the wall rock when poor ground conditions exist, or when the vein is crooked or has poor continuity. The crosscuts are driven through the vein, and the raises are driven on the vein from the crosscuts. An initial drift is driven in the ore for the length of the stope, and it may be driven on the level or 20 to 25 ft (6 to 8 m) above the level. This serves as the undercut.

Another common method is to drift on vein and not drive laterals. This method is used by the Galena and Sunshine mines in the Coeur d'Alene district of Idaho.

19.2.1.4 Access

Access to the stopes is via raises or ramps in mines using LHD equipment.

Some mines drive raises from the lower level to the level above, or drill a borehole from the upper level to the lower level. In the Coeur d'Alene district, it is common to run the raises up to the initial drift, and then continue them upward as mining progresses.

Another method is to drive a three-cap raise in which one compartment serves as the manway and the other as the rock chute. Other mines maintain four, separate, closely spaced crib raises, surrounded by backfill: two for rock chutes, one for the manway, and one for the timberslide. These raises are usually constructed of square or hexagonal wood crib, or of round steel liner plate. The manway and timberslide also serve as ventilation conduits.

In mechanized stopes, ramps are driven in either the hanging wall or the footwall so that extraction raises may be installed. The inclination of the ramps varies from 1 : 5 to 1 : 10, depending on the mine. Short access inclines or declines are also driven into the stopes at regular intervals as the mining progresses.

19.2.1.5 Drilling

BREAST. Jacklegs are used for drilling in non-mechanized breast stopes. Normally, a 2-ft (0.6-m) starter steel is used to collar the hole, changing to 4-ft (1.2-m) and 6-ft (1.8-m) lengths to finish the hole. The drill steel may be either integral or tapered with a knock-off bit.

Rubber-tired or crawler-mounted, two-boom jumbos are used for breasting down in mechanized stopes. Either integral drill steel, or tapered drill steel with a knock-off bit, is used.

The number of holes drilled varies from eight for a narrow stope (42 in. or 1.07 m or less, in width) to 12 or more holes for wide stopes. It is important that all holes be drilled parallel to the walls in the vertical plane. In the horizontal plane, all except the top row of holes should be drilled parallel to the floor of the stope. The top row of holes should be inclined upward 2 to 3°.

UPPERS. Drilling upper holes is done with either stopers, jumbos, or crawler-mounted drifters.

In nonmechanized stopes, the drilling is normally done with stopers, using integral or tapered drill steel equipped with $1\frac{5}{8}$-in. (41-mm) bits. The holes are drilled vertical to a depth of 3 to 10 ft (0.9 to 3.1 m). The complete back of the stope is drilled out before blasting.

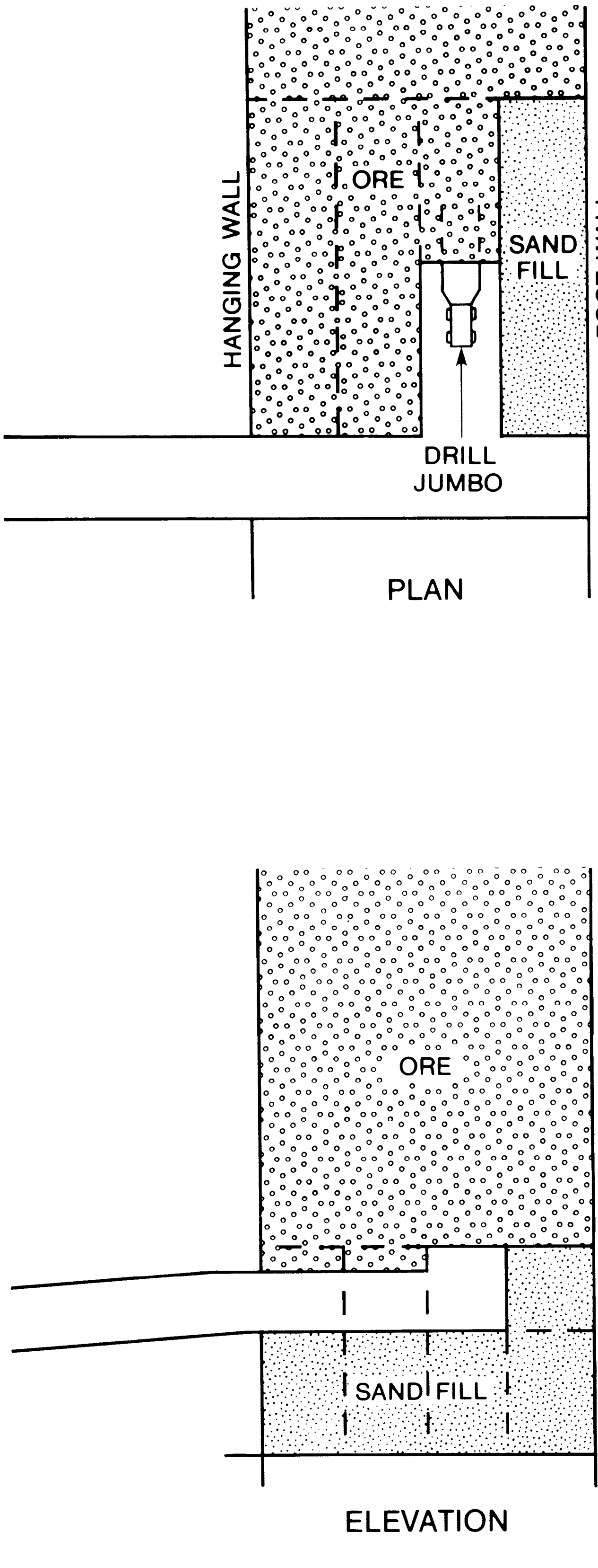

Fig. 19.2.3. Drift and fill stoping.

Homestake in Lead, SD, uses stope jumbos (Waterland et al., 1982). The jumbo is in the form of a tripod with the front of the feed shell mounted on a frame attached to two wheels, with the third leg of the tripod being formed by the end of the feed shell. The angle of the feed shell is preset to 55 to 60° from the horizontal. Presetting the angle ensures that all the holes will be drilled parallel and to the same height above the cemented sand fill floor. A 3-in. (76-mm) bore drill machine is mounted on the feed shell, and 3/4-in. (19-mm) hex drill steel with a 1-1/4-in. (32-mm) bit is used to drill the blastholes to a depth of 10 ft (3.1 m).

Two- or three-boom rubber-tired-mounted jumbos or crawler-mounted drifters are used for drilling upper holes in mechanized stopes. The holes may all be drilled vertical or at preset angle, and be 10 to 15 ft (3.1 to 4.6 m) in length.

The important thing is to drill all of the holes parallel to each other, and to have them all end at the same height above the floor of the stope.

If any ore is left on the periphery of the stope from the last cut mined, it is slabbed before the round is blasted.

19.2.1.6 Temporary Ground Support

Temporary ground support is provided by square sets, stull sets, stulls, and/or rock bolts.

SQUARE SETS. The use of square sets has declined in recent years due to the increasing cost of labor and material. This method is primarily used where the walls of the stope are weak, or where the ore body is too wide for stull timbering. The ore is excavated in blocks of approximately the same size, ranging from 5 to 5- by 7-ft (1.5- by 1.5- by 2.1-m) high to 6- by 6- by 8-ft (1.8- by 1.8- by 2.4-m) high. As soon as a block is removed, a square set is erected in the open space (Bishop, 1945; Haffner and Hoskins, 1973; Wilhelm et al., 1982; Young, 1946).

A square set is composed of a vertical post and two horizontal members, cap and girt, that meet to form a right angle. The ends of the three members are framed to give each a bearing against the other two at the corners, and they join with three other similar sets to form a complete timber support unit.

The cap is laid in the direction of the maximum lateral pressure, and it is the main load-bearing member. The girt acts as a stabilizer. The size of the timber is determined by prior experience in that ore body or in that district.

Sills are initially laid in trenches cut in the floor of the stope, and the posts are set directly on them. If no stoping is to be done from below, and if the ground is hard, the sills may be omitted, and the posts set directly in hitches in the floor or set on floorboards.

Through the years, many methods of framing square sets have been developed by various mining companies. A good reference is Peele (1941) in which 26 different methods are illustrated.

STULL SETS. Stull sets are applicable to ore bodies that dip 70° or more, have very heavy walls, and are not more than 20 ft (6.1 m) in width. The stull sets prevent movement of the walls on the mining floor until sand fill can be poured.

Stull sets were used at Hecla's Star mine in Burke, ID (Murray et al., 1982). The veins were 5- to 20-ft (1.5- to 6.1-m) wide and mining these veins caused plastic flow of the vein walls into the stope. The stull set consisted of two posts, a cap, and two squeeze headings. The caps were round timber, 12 to 24 in. (305 to 610 mm) in diameter, depending upon the span. The cap was supported by two posts extending to the footwall or the cap below. The posts provided rigidity rather than support, and were round timbers 8 to 12 in. (203 to 305 mm) in diameter. Girts were placed between the caps and between the posts to provide longitudinal support.

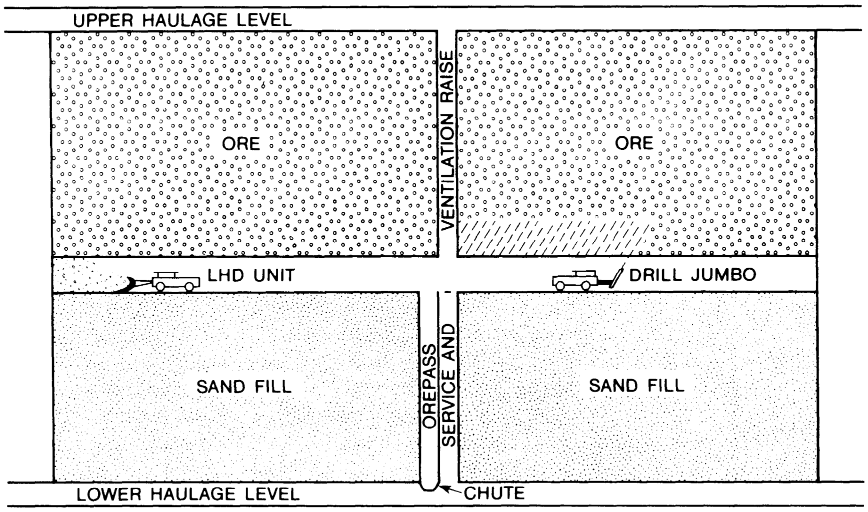

Fig. 19.2.4. Longitudinal section of a mechanized back stope.

The key to the system was the squeeze headboards composed of 3- by 12-in. (76- by 305-mm) lagging 24- to 36-in. (610- to 914-mm) in length. Two to four of the headboards were placed on each end of the cap and wedged tightly with wooden wedges. As the wall pressure increased, the squeeze headboards gradually collapsed but maintained maximum pressure.

Double lagging was placed on top of the caps, and where the back was high, cribbing was placed above the lagging.

STULLS. Stulls are used in narrow vein stopes where the span can be covered by a single timber. Stulls are applicable to stopes with weak walls or to stopes where the ore is irregular, which permits the removal of high-grade sections leaving the low-grade and barren sections behind. Stulls are usually the main means of temporary support in back stopes. Stulls do not carry the entire stress in the rock, so they are used only for temporary support until the stope can be backfilled.

Stulls are usually round timber, 6 to 8-in. (152 to 203-mm) in diameter, and with round timber 12 in. (305 mm) or more in diameter occasionally being used. Squeeze blocks are installed at one or both ends to prevent breakage of the stull in mines where continuous closure occurs (Young, 1946).

ROCK BOLTS. Rock bolts may be mechanically point-anchored (expansion-shell type), friction stabilizers (Split Set, Swellex), and grouted (resin or cement-grouted). Rock bolts are popular because they are inexpensive, relatively easy to install, efficient, do not reduce the cross-sectional area, and can be combined with other methods of ground support.

Expansion-shell type rock bolts tend to loosen shortly after installation so they have to be checked, and, if necessary, retorqued. This checking and retorquing procedure may have to be done daily for a week or more, depending upon the ground conditions and proximity of blasting. The use of expansion shell rock bolts has declined in recent years because MSHA requires that the torque of point-anchor bolts be checked when installed, and also periodically, which is time consuming and costly.

Friction stabilizers are easy and very fast to install. Their main advantages are that (1) the holding force acts over the full contact length of the bolt, (2) they have the ability to yield while retaining a full holding force, (3) anchorage increases with time, (4) plate loads are maintained, and (5) full contact around the circumference of the hole provides a greater resistance to displacement of the rock along the shear and bedding planes. The disadvantages are that they do not have the holding power of a resin-grouted rock bolt, and they do not hold as well as a resin-bonded bolt in hard, fractured ground.

The use of resin-grouted rock bolts is becoming more widespread because they provide instant support, the grouting length may be varied, and the resin cartridges are clean and easy to handle. The bolts may be plain rebar, threaded rebar, or cable. The disadvantages are the high cost, poor mixing of the resin and hardening agent under certain conditions, and the relatively short storage life of the resin cartridges.

Bolts grouted with bulk cement have good reinforcement properties, are protected well against corrosion, and the cement is inexpensive and readily available. The disadvantages are the long setting time, a mixer/grout pump is required, and it is not practical when only a few bolts are to be grouted.

19.2.2 UNDERCUT AND FILL MINING

19.2.2.1 Description of Method

Undercut and fill mining is a method of extracting a block of ore by mining successive slices (cuts) 6 to 15 ft (1.8 to 4.6 m) in height from the top down, and filling each successive cut with cemented sand fill. This method was developed in the late 1950s by Inco, Ltd., in the Sudbury district of Ontario, Canada, to deal with abnormal ground conditions encountered in pillar recovery. Many mines now use the undercut and fill method as their primary mining method (Murray, 1973; Murray et al., 1982; Rausch and Stitzer, 1973; Young, 1946). A vertical or steeply dipping vein is developed on levels 150 to 200 ft (46 to 61 m) apart with a system of lateral drifts on each level. Extraction crosscuts are driven through the vein on 100- to 300-ft (30- to 91-m) intervals. Orepass raises are driven or bored, in the ore body or wall rock, between the upper and lower crosscuts. In mechanized mines, ramps are driven for access to the stopes and orepasses.

The initial floor is established on the upper level. It is mined by driving a conventional drift round, and the opening is supported by square sets or by rock bolts and mats. The ore is removed by use of slushers, or LHD equipment in mechanized stopes, through orepasses to the level below. When the initial cut is completed, a timber and/or wire fabric mat is placed on the floor of the cut, and the opening is filled tightly with cemented sand fill; this method appears in Fig. 19.2.5

Access to the mining area on the next cut is achieved by carrying the manway and timber slide openings, or the access ramps, downward. Mining is then resumed on the next cut below

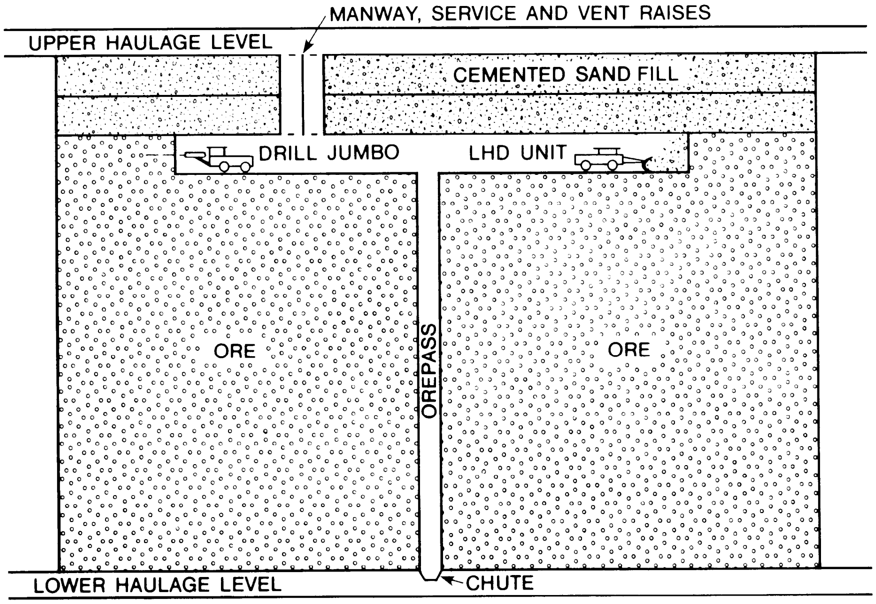

Fig. 19.2.5. Longitudinal section of an undercut and fill stope.

the mat. As the cut is advanced, the mat/sand fill covering is supported, if necessary, by round timber posts. Hecla's Lucky Friday mine in Mullan, ID, uses a net of wire fabric in lieu of posts. The net spans the two ribs and the floor, and it is held in place by rock bolts in the ribs.

The fill placed in each of the successive cuts must be competent in order for this method to be successful. There must be adequate cemented sand fill material available at the stope by having a dependable sand plant and distribution system.

19.2.2.2 Application to Various Geometries

Undercut and fill stoping can be used where either the wall rock or vein is too unconsolidated to permit safe mining by overhand cut and fill methods.

This method can be applied to ore bodies that vary in width and to those that dip from vertical to flat lying.

The method can be used to recover pillars in lieu of the labor-intensive and expensive square set method.

One advantage is that it eliminates the removing of a pillar of decreasing area, which supports horizontal stress, that is encountered in overhand cut and fill mining when advancing from a lower level to a mined-out upper level. In the deep mines of Idaho's Coeur d'Alene district, this method reduces rock-burst problems.

19.2.2.3 Development

In non-mechanized stopes, the laterals are driven in the wall rock parallel to the vein structure. Extraction crosscuts are then driven through the vein on 100- to 300-ft (30- to 91-m) intervals, depending upon the length of the vein, and the most efficient slushing distance for the mine.

Raises for orepasses are driven in the vein up to the upper level, or are bored in the vein from the level above. In mines where the ore is not competent, the raises are located in the wall rock. Loading chutes are then installed at the bottom of the orepass raises.

In a mechanized stope, a main ramp decline is driven in the wall rock from the upper level. These declines are driven at some predetermined distance in the wall rock, commonly 30 to 50 ft (9 to 15 m). Crosscut access ramps are then driven from the main ramp into the ore body.

19.2.2.4 Access

Access to the stopes is via winzes that are carried down as the mining progresses or via ramp declines in mechanized stopes. Access may also be provided via conventionally driven or bored raises that were completed during the development stage.

Some mines drive three-cap raises in which one compartment serves as the manway and the other as the rock chute.

19.2.2.5 Drilling

Drilling is done with either jacklegs or jumbos. Before the drilling commences, the face and walls are barred down, and the underside of the mat is checked to remove any loose sand fill.

The top row of holes is drilled flat and at a minimum distance of 2 to 3 ft (0.6 to 0.9 m) below the mat in order to prevent damage to the mat when the round is blasted. Also care is taken in drilling the lifter holes in order to avoid the necessity of popping (blasting) the bottom to set the posts and stringers. This also ensures a relatively smooth bottom for mucking.

Normally, a 6-ft (1.8-m) round is drilled, and in tight ground a burn cut is used.

19.2.2.6 Temporary Ground Support

Temporary ground support is provided by posts and/or rock bolts. As mining progresses, round wood posts are placed under the stringers of the previous cut. Rock bolts may also be used to supplement the posts.

In narrow veins, posts may not be required, and rock bolts are used to provide temporary support until the mat is constructed and the cemented sand fill is poured.

Another method is to span the two ribs and the floor with wire fabric, which is held in place with rock bolts.

The cemented sand fill supports the major portion of the load. The posts and mat have a double function as they carry the weight not supported by the sand fill, and also prevent any pronounced settling of the fill mass.

Table 19.2.1. Equipment Requirements

Type of Stope	Drilling Equipment	Mucking Equipment
Breast (Narrow Vein)	Jackleg	Slusher 1/2 yr^3 (0.4 m^3) LHD
Breast (Wide Vein)	Jackleg 2-boom Jumbo	Slusher 1–5 yd^3 LHD (0.8–3.8 m^3 LHD) Rubber tired Overshot Mucker
Post-Pillar	Jackleg	1–5 yd^3 LHD (0.8–3.8 m^3 LHD)
Drift and Fill	2-boom Jumbo	1–5 yd^3 LHD (0.8–3.8 m^3 LHD)
Back	Stoper Stope Jumbo 2- or 3-boom Jumbo Crawler mounted Drifter	Slusher 1–5 yd^3 LHD (0.8–3.8 m^3 LHD)
Undercut	Jackleg 2-boom Jumbo	Slusher 1–5 yd^3 LHD (0.8–3.8 m^3 LHD)

A detailed description of the permanent support system is given in Chapter 19.3.

19.2.3 ELEMENTS COMMON TO OVERHAND AND UNDERCUT METHODS

A comparison of equipment requirements for drilling and mucking with different stoping methods appears in Table 19.2.1.

19.2.3.1 Blasting

Each drillhole is first loaded with an initiating device such as an electric blasting cap or a nonelectric blasting cap, either of which may be a standard delay or a millisecond delay, along with a booster or a stick of powder. Then each hole is loaded with gelatin cartridges or ANFO.

19.2.3.2 Ore Removal

The ore is removed from a nonmechanized stope by means of a double- or triple-drum slusher. The slusher may be either electric or pneumatic powered. The ore is scraped to the stope orepass where it drops to the level below.

In mechanized stopes, the ore is removed from the stope by a diesel- or electric-powered LHD unit. The ore is dumped at the stope orepass for handling on the lower level, or the ore is transported directly to a main orepass.

As soon as a stope is mined out, all of the ore must be removed from the stope. This is called the clean-out cycle.

In stopes using slushers, the scraper should be played back and forth across the muckpile to keep it fairly level. Trenching or winrowing should be avoided. The clean out should progress from the back end of the stope towards the orepass. A minimum of sand should be taken in those stopes not using cemented sand fill.

The clean out usually progresses much faster in stopes using LHD equipment for mucking, as one area of the stope can be cleaned out entirely before progressing to the next area. The LHD operator should be careful not to dig too far into the sand if cemented fill is not used.

Whenever possible, any remaining slabbing should be done early in the clean-out cycle, and the walls should be trimmed down as the muck level is lowered. Where necessary for safety, and to prevent dilution, the walls should be supported by stulls or rock bolts in nontimbered stopes.

19.2.3.3 Orepasses

Orepasses can be unlined when they are located in the country rock. When they are located in the backfill, they are constructed of square or hexagonal wood crib reinforced with angle-iron wear plates, or constructed of round steel liner plate. Some mines are now lining orepasses with steel-fiber-reinforced shotcrete, and they are holding up very well. The choice of which lining to use depends upon the type of rock, the length of the orepass, and the estimated tonnage that will flow through the orepass.

Short unlined orepasses, 100 ft (30 m) or less in length, in hard tough country rock will probably stand up under the constant passage of ore or waste rock. Longer orepasses in country rock or those located in the ore should be constructed of wood crib, steel liner plate, or fiber-reinforced shotcrete.

The bottom of the orepass at the haulage level usually contains a loading chute that is lined with steel plate and equipped with a pneumatically operated gate. The ore is loaded into trucks or into trains containing 3 to 12 ore cars. The trucks or ore cars are then dumped at the main orepass, which is usually located at a haulage shaft.

Instead of using a loading chute, the ore may be dropped to an enlarged area beneath the orepass. The ore is then loaded into ore cars or trucks using either a LHD unit or an overshot loader, or the ore may be hauled directly to the main orepass by a LHD unit.

19.2.3.4 Ventilation

Fresh air normally enters the mine through a downcast shaft, and it is split to flow through the active work areas. The air is then exhausted through raises, laterals, and an upcast shaft. About 30 cfm (0.014 m^3/s) per person is required for normal ventilation, and where diesel equipment is used, the requirement is approximately 100 cfm (0.047 m^3/s)/hp. Hot mines will require more than the above-stated flows and possibly refrigeration equipment as well, depending upon the ambient temperature of the working place, which should be kept below 85°F (47°C) wet-bulb temperature.

Auxiliary ventilation for individual work places is usually provided by 10- to 20-hp (74.5 to 149-W) axial flow fans, and the air is conveyed to the area via 16- to 20-in. (410- to 510-mm) vent tubing, or 20- to 24-in. (510- to 610-mm) vent pipe. The Lucky Friday mine, Mullan, ID, uses 21- and 22-in. (530- and 560-mm) corrugated steel pipe that is buried in the sand fill. In the stope, a vent tube connected to the end of the corrugated pipe conveys the ventilation air to the work area.

Some mines drive ventilation raises at the extremities of each stope to exhaust the air, and these raises are equipped with ladders as well as compressed air and water lines. The raises can also be used for stope access and escape routes. Other mines do not drive raises all the way through from the sill level to the level above. These mines space their manways, timberslides, and rock raises 50 to 100 ft (15 to 30 m) apart, and the open manways and timberslides provide a conduit for the exhaust air.

In mechanized stopes, vent tubing is strung in the access ramp to convey the fresh air. The exhaust air is routed up a vent raise or out through an access ramp.

19.2.3.5 Crews

Non-mechanized stopes usually have a crew of two miners per shift. Back stopes may have three or four miners per shift, because normally a series of holes are drilled before blasting the round. The additional personnel speed up the drilling cycle and also decreases the time required for the mucking and raise up cycles.

Mechanized stopes usually have two miners per shift. Mt. Isa uses three miners plus a diesel operator in their large, wide stopes.

Stoping operations are usually conducted on a two-shift basis, but a few mines carry on operations on a three shift-basis.

Support crews consist of one or two nippers per level, depending upon the number of stopes. The nippers deliver supplies and haul ore and waste in small mines. In larger mines, the nippers only service the miners, and two-person crews load and haul the ore and waste.

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Chapter 19.3
BACKFILLING METHODS

WALLACE E. CRANDALL

19.3.1 WASTE FILL

19.3.1.1 General

The use of waste fill is probably as old as mining since it has always been valuable to a mine operator to dispose of development rock in mined-out stopes to avoid removing it from the mine. The primary purpose of waste fill or any type of fill, however, is to support mined-out openings so as to prevent caving, surface subsidence, pressure on the work place, or rock bursting. Another reason for using waste fill is simply that the waste material may be conveniently available to the mine and is the most economic method of filling mined-out openings that require support.

19.3.1.2 Sources

Sources of waste fill material include the following:

1. Surface quarries near the mine.
2. Underground development rock.
3. River gravel.
4. Glacial till.
5. Talus.
6. Dune sand.
7. Mine dumps.
8. Sink-float plant reject.
9. Coal preparation plant reject.
10. Smelter slag, fly ash.
11. Naturally broken fault material from an underground source far enough from the workings to be caved and drawn safely. Source materials may include such variations as:

a. Completely ungraded waste rock with little or no fines.

b. Graded and sized material containing cement and appropriate water content for greater strength.

c. Raw material containing sulfides (e.g., pyrite, pyrrhotite) or other minerals that help consolidate the fill.

d. Mixtures of waste rock or gravel and mill tailings to provide a better size gradation that will fill most of the voids in the mixture. Part of the slime fraction may be removed by cycloning for better handling and placement.

e. Mixtures of waste rock and smelter slag or fly ash that may have cementing or pozzolanic properties resulting in consolidation of the fill.

19.3.1.3 Preparation and Delivery Methods

In general, the waste material has to be sized according to the handling and placement method used. If the waste fill is excavated in a surface quarry, dropped by gravity through waste raises, trammed by rail to a stope raise, and slushed from the raise into the stope, then –12-in. (305mm) material is satisfactory. This method was in common use for many years in the mines of the Coeur d'Alene District, near Wallace, ID. A typical example is the Star mine in Burke. During the period from 1950 to 1960, –12-in. (305mm) waste fill was quarried from a surface pit and fed to the upper workings via a system of waste passes to the working levels. Development waste was also used where possible. In some stoping areas, waste fill was broken by driving crosscuts in the walls. A hydraulic sandfill system replaced waste fill in 1959.

If development waste from underground is used for stope fill, material –12 in. (305 mm) in size is convenient. If the material is handled by conveyor belt, width and particle size must be compatible.

Pneumatically placed waste fill is usually –1 in. (25 mm) in size, although particles up to 4 in. (102 mm) can be handled, depending on pipe size (Powell and Ruby, 1981).

Any waste fill can be improved and made stronger by sizing the fines and coarse material so more of the voids will be filled and a more compact fill achieved. Crushing and screening are necessary, and the increased cost must be balanced against the need for a stronger fill; that is, can adequate wall support be attained without the expense of sizing the fill? Most waste fill can be placed without special sizing gradation and still provide adequate wall support or protection from pressure on the workings. However, some excellent consolidated waste fills have been made by the addition of cement to a graded sand and crushed rock mixture to produce concrete of the desired strength. A detailed study of the strength of fill that can be achieved using coal mine washery refuse is provided by Knissel and Helms (1983). The material studied was sandy shale and sandstone in particles from 0.0039 to 3.94 in. (0.1 to 100 mm) in size. Uniaxial compression strength up to 580 psi (4 MPa) was achieved in 28 days with 10% cement.

A mine that is developed by rail haulage via a long tunnel might find it necessary to haul waste into the mine in the returning ore cars. Similarly, waste fill may be hauled into the mine by returning trucks in a trackless mine. Conveyor belts can be used to deliver waste fill to the stoping areas, and waste rock, gravel, and sand can be passed through boreholes or raises of various diameters from surface to the mining area. As a rule of thumb, however, the hole or pipeline diameter should be at least three times the largest particle to prevent plugging. Pneumatic delivery of waste fill is a field in itself and discussed later.

19.3.1.4 Placement Methods

Actual placement of the waste fill material in the stope may be accomplished in the following ways:

1. *Slushing* (see definitions, Chapter 19.1, part 19.0.1.2). The waste fill may be slushed directly from a waste raise to the stope area to be filled. Usually, the waste raise leading to the stope is supplied by rail car, front-end loader, or load-haul-dump (LHD) from a waste pass or transfer raise, which is the material source for the entire mining area.

2. *Slinger belt placement.* This system may use a high-speed belt and hopper supplied by truck or LHD. The material may also be placed by a slinger belt mounted under a hopper. Slinger belt systems provide good compaction and relatively strong fill if gradation of the material is proper.

3. *Stower placement.* Dry waste fill in the 3-in. (76-mm) range may be placed by pneumatic stower. The system utilizes a low-pressure blower and a rotary feeder that delivers the mate-

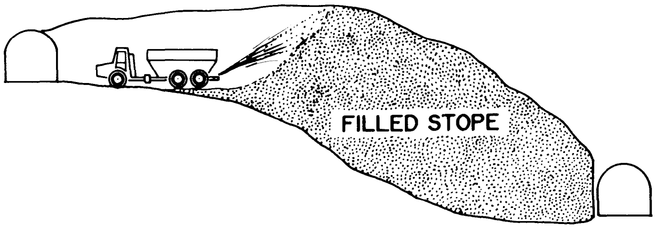

Fig. 19.3.1. Truck-mounted slinger belt at Keretti mine, Outokumpu Oy, Finland.

rial to the area to be filled via pipeline. See 19.3.2 on pneumatic filling methods for further details.

19.3.1.5 Advantages

Many times, an inexpensive source of waste fill available near the mine improves the economics of the mine. The preparation of waste fill is usually less expensive than other types of fill due in part to a simpler plant facility.

1. The use of underground development rock reduces the cost of disposal by removing it from the mine and also helps eliminate environmental problems on the surface.
2. The addition of cement or other binding agent can make a strong consolidated fill if necessary.

19.3.1.6 Disadvantages

Waste fill may result in a relatively loose, uncompacted fill that does not resist closure well if the material is not sized so that fines occupy most of the voids. Unconsolidated waste fill can result in unsafe caving or sloughing conditions if too much fill face is exposed. Overly wet fill in transfer raises can become hydraulic, resulting in dangerous surge problems.

19.3.1.7 Applications

Outokumpu Oy's Keretti mine in Finland uses a specially designed truck-mounted slinger unit to place a waste fill composed of 86% crushed rock, 5% cement, and 9% water (Fig. 19.3.1).

Asamera's Cannon mine in Wenatchee, WA., uses a concrete fill made of Columbia River sand and gravel and 5% cement in a very thick mixture (Argall, 1988). The sand, aggregate, and cement are transferred dry, by gravity, through boreholes to an underground mixing plant. Small bulldozers are used to level the low-strength concrete, providing flat working floors for successive cuts. Pillars are removed later and filled with 3% concrete fill or dry fill.

The Cannon mine is backfilling its overhand sublevel bench stopes at a rate of up to 480 tpd (435 t/day) with truck-placed cemented and uncemented backfill (Figs. 19.3.2 through 19.3.4). The high-strength, cemented backfill, consisting of sand, aggregate, and cement, is mixed underground by an automatic batching system operated by a programmable controller (Fig. 19.3.5). Batching is initiated by truck-mounted radio. Uncemented fill is delivered by the same system.

The need for a cemented fill strength of 1200 psi (8.3 MPa) was indicated by safety considerations and the expectations of full overburden loading of some cemented fill pillars during removal of the intervening secondary stopes. Because of this loading and in order to limit subsidence, both the cemented fill

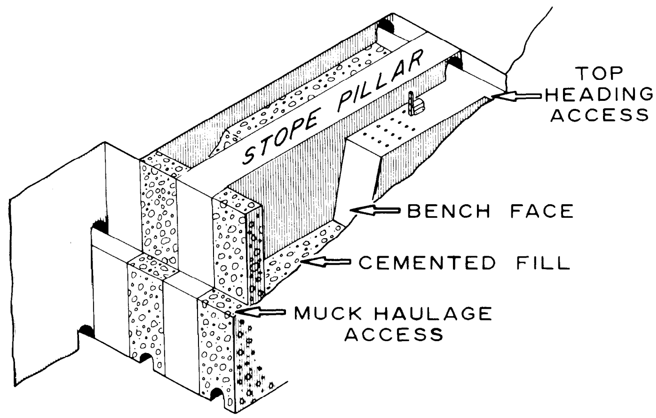

Fig. 19.3.2. Isometric drawing illustrating sublevel benching, cut and fill mining method, Cannon mine

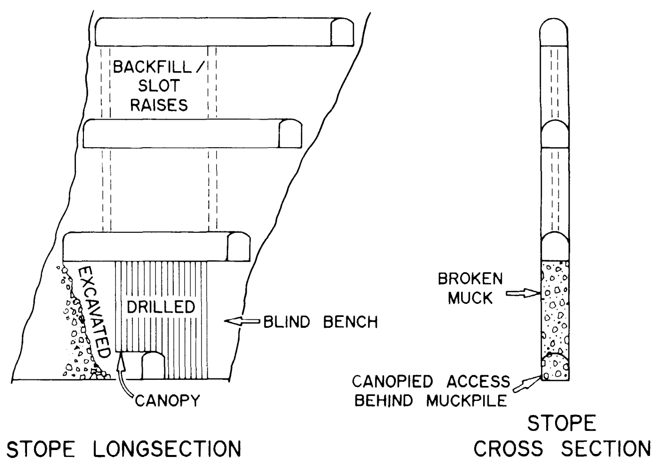

Fig. 19.3.3. Overhand/underhand bench, general arrangement, Cannon mine.

in the primary stopes and the uncemented backfill in the secondary is placed tight against the back at the top of the ore body. A mechanical jamming device has been developed in conjunction with a 5-yd^3 (3.8-m^3) LHD to drive the fill tight against the back.

Fill material is delivered from surface sources to the grizzlies, over the boreholes, leading to the underground mixing plant (Fig. 19.3.5). Initially, fill material was a mixture of 55% 3-in. (76-mm) aggregate, sand, and 5% Portland cement, but this resulted in segregation when dumped over a berm in the stope (Fig. 19.3.6). Subsequent use of aggregate with a maximum size of 2 in. (51 mm) has adequately relieved the problem.

Surface subsidence is minimized by the support offered either by in place pillars or the pillars created by high-strength fill. The voids (e.g., robbed pillars) that do not require cement fill are available as dump points for mine waste, eliminating the costly handling of waste both underground and on the surface.

Cemented backfill is the support means that allows complete pillar recovery with no perceptible surface subsidence. The cemented fill is designed to be strong enough to support full overburden load and provide stable exposed stope walls during extraction of secondary stope benches.

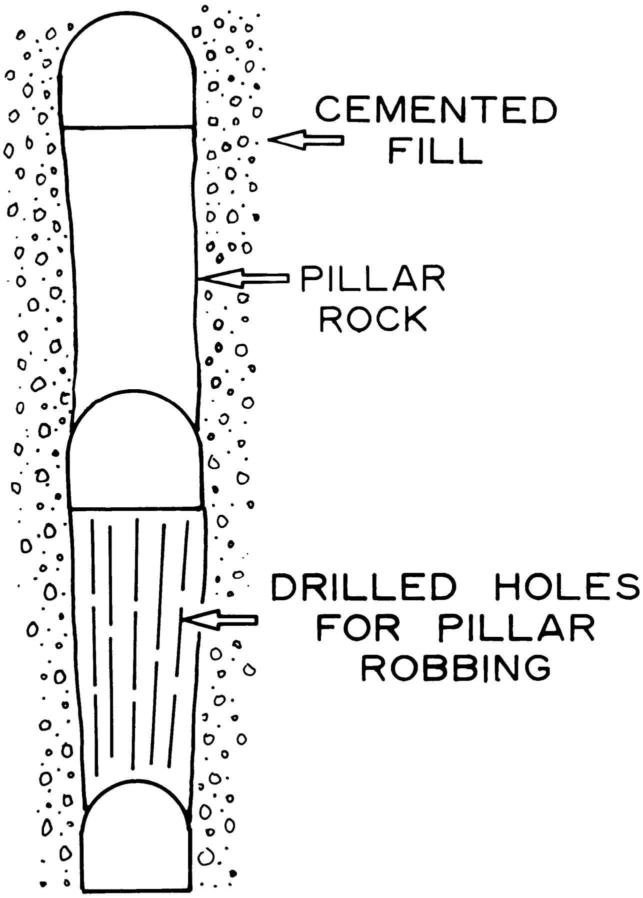

Fig. 19.3.4. Pillar robbing cross section general arrangement, Cannon mine.

The backfill mix is as follows:

Aggregate (2 in. to #4 mesh, or 51 to 4.7 mm)	55%
Sand	40%
Type I, II Cement	5%
Water : cement ratio	1 : 1
Water reducing admixture 6 oz/100 lb (3.8 g/kg) cement.	

The backfill mix provides an unconfined compressive strength of 1200 psi (8.3 MPa).

Cemented fill is batched by a pugmill underground. Sand and aggregate are dumped down boreholes into underground silos of approximately 3000 tons (2700 t) each. The sand and aggregate are fed into the pugmill by vibrating feeders. Cement is stored in a 350-ton (318-t) capacity surface silo and fed to the pugmill by a pipeline in a borehole from the surface. The cement flow is regulated by two vane feeders, one on the surface and one near the pugmill. The water-reducing admix is prepared on the surface and metered to the pugmill by a pump. All material feeders are controlled by a process controller. The ingredients are mixed and fed into a fill bin by a double-screw pugmill. The batch is fed into a truck by a clam gate. The pugmill batch cycle is initiated by the truck driver after his truck is loaded. The batch cycle requires approximately 4 minutes to process 10 yd^3 (7.65 m^3), equivalent to 18 tons (16 t). The trucks dump the cemented fill down a backfill slope with a 38° angle of repose. The low water content of the fill allows the trucks to drive on freshly placed fill (Guenther, 1988).

B.P. Minerals, Ltd. operates the Greens Creek mine on Admiralty Island west of Juneau, AK. They use a drift and fill mining method in ore bodies averaging 8 to 15 ft (2.4 to 4.6 m) in thickness (Steffen, Robertson, and Kirsten, 1984). Since the grind in the mill is 95% –200 mesh (0.074 mm), the tailings are too fine to make a satisfactory fill. Less than 15% of backfill requirements could be met with appropriate material recovered by cycloning. Wall rocks are friable argillite and tuffite, which would result in 25% of development waste rock being discarded as fines, aggravating the disposal problem. Further degradation would occur in pumping, all of which makes a conventional hydraulic fill unattractive. Pneumatic delivery has also been discarded due to high cost and inflexibility. There will be an accumulated stockpile of development waste available, which will be sufficient for 5 years of backfilling. The system planned will use this available waste with 3% (30 : 1) cement and slinger belt placement, utilizing either slinger belts loaded with an LHD unit or with a truck-mounted slinger belt loaded directly from the truck body. This system is expected to give adequate-strength fill for a 15-ft (4.6-m) high free-standing face.

The Meggen mine in western Germany has a total extraction drift-and-fill system that uses a clean 2-in. (51-mm) maximum size aggregate from a heavy media separation plant together with 3 to 5% cement to produce an effective backfill. The cleaned aggregate at Meggen is deficient in fines but reaches an adequate strength (Steffen, Robertson, and Kirsten, 1984; Rohlfing, 1983).

Fill is placed with a skid-mounted or truck-mounted stowing machine (Figs. 19.3.7 and 19.3.8). Only dry placement methods are in use with rockfill out of development drifts (5%) and slinger belt stowing with cleaned rockfill (95%). Slinger belt stowing replaces pneumatic stowing, which results in rather high overall backfill costs.

The compaction of fill material by the slinger belt system is comparative to pneumatic stowing and provides a surplus of compressive strength compared, for example, with rockfill placed by LHD units. The complete filling of a mined-out room in a flat-lying deposit can only be achieved with pneumatic or slinger belt stowing.

The use of slinger belt stowing superseded the earlier use of pneumatic placement because of higher performance and lower overall costs of this method (Rohlfing, 1983).

An example of the use of dry sand fill in Canadian mines is described by Twidale (1962). Two of the oldest gold mines have used dry sandfill in the past. These mines are deep, and fill is required in a multitude of small vein-mining operations. The practice of sand filling existed before hydraulic systems came into use, and it is not considered economical to change the material and methods. The pit-run sand is mined by mechanical shovel and transported some 3½ miles (5.6 km) to the mine site by aerial tramway. The fill is stored in near-surface stopes and fed to a sand pass that connects the mining horizons to a total depth of nearly 6000 ft (1800 m). Sand and gravel fill is placed in similar fashion to fill empty shrinkage stopes less than 1000 ft (300 m) below the surface that have been emptied of ore and where stability is consequently necessary to prevent ground movement. A near-surface stope is used as a storage bin and connected to the sand pass through control gates.

19.3.2 PNEUMATIC FILL

19.3.2.1 General

The disposal of mine tailings, development rock, and other unwanted refuse material underground in mines by using pneu-

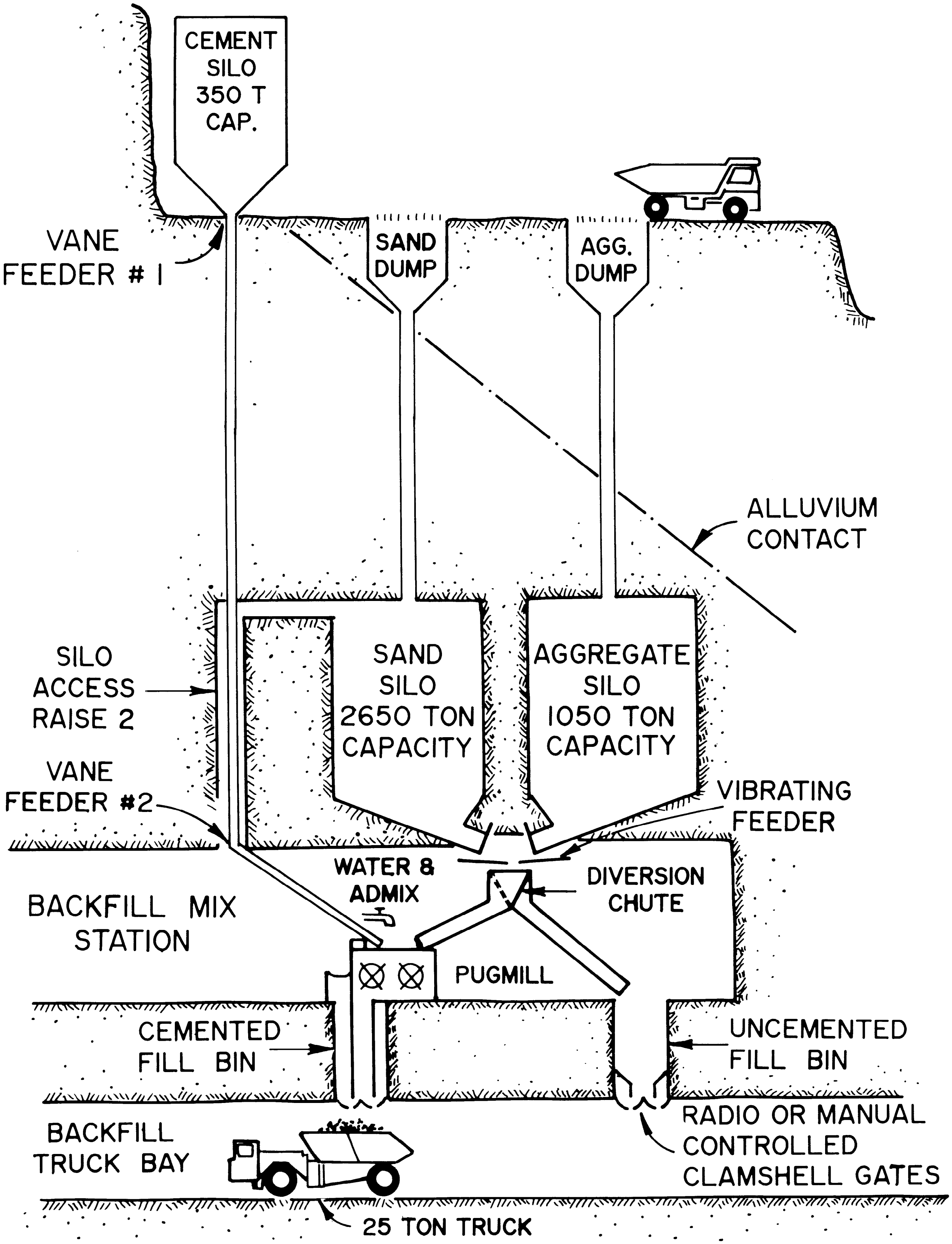

Fig. 19.3.5. Backfill plant schematic, Cannon mine.
Conversion factor: 1 ton = 0.9072 t.

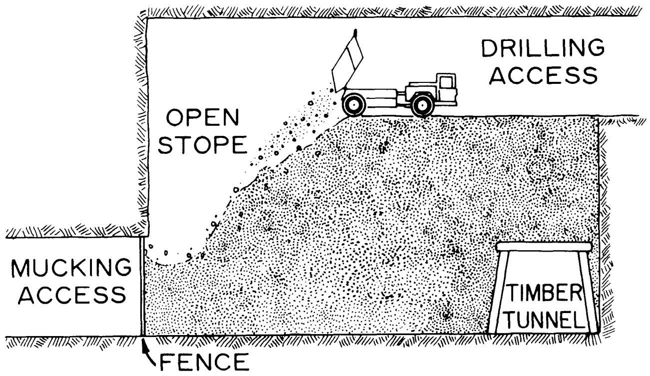

Fig. 19.3.6. Backfill placement, Cannon mine.

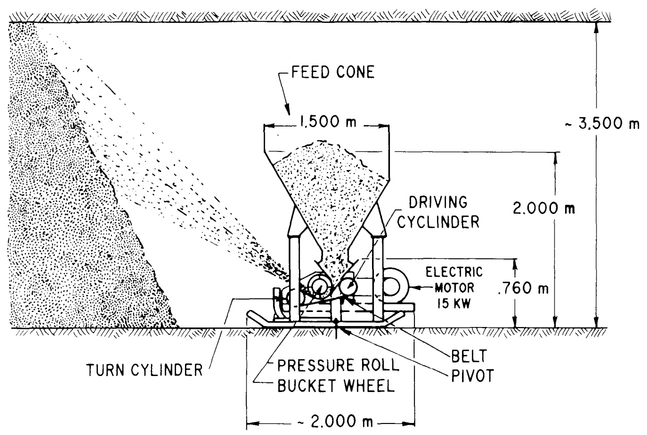

Fig. 19.3.7. Slinger belt stowing machine, Meggen mine, West Germany (*courtesy:* Rohlfing, 1983). Conversion factor: 1 ft = 0.3048 m.

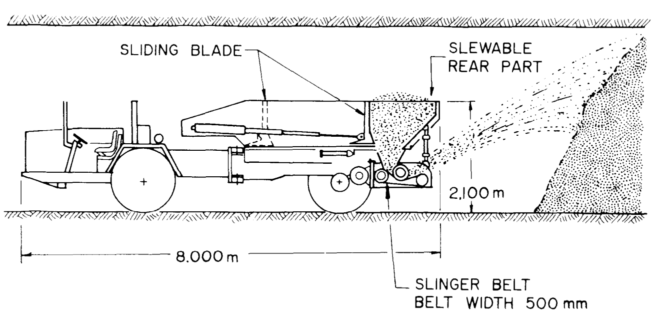

Fig. 19.3.8. Slinger belt truck, Meggen mine, West Germany (after Rohlfing, 1983). Conversion factor: 1 ft = 0.3048 m.

matic conveying techniques has been practiced in the coal industry for many years. Similar equipment has been utilized for backfilling hard-rock mines and is currently being applied in a South African gold mine where the development rock is being stowed in mined-out stopes to reduce the incidence of rockbursts. On a limited scale, pneumatic conveying has been applied for backfilling uranium mines, and one mine in New Mexico has used this technique for eight years (Powell, 1980). The method was also used at the Billie mine in Death Valley, CA, and at the Apex mine near St. George, UT.

Pneumatic conveying can overcome many of the problems that result from hydraulic filling of flat-lying stopes, in that voids can be completely filled with a high degree of compaction, heavy bulkheads are not necessary, and development rock can be added to the pneumatic system for disposal underground, thus eliminating the hoisting cost to the surface. Cement can be added to the wetted material to form a high-strength concrete (Powell, 1980).

Although pneumatic conveying has several advantages over hydraulic fill, there are two major disadvantages: (1) the excessive power required for long horizontal distances; and (2) as a result of the high air and material velocity in the long pipeline, a wear problem that necessitates frequent rotation of the pipelines and eventual replacement.

Where long pneumatic pipelines require high horsepower and high velocity with resultant wear, a combination hydraulic-pneumatic system has been proposed by Leon Gregory but not yet used (Powell, 1980). The hydraulic fill material would be delivered to the mining area by pipeline or borehole using pumps or gravity, as required. Then the slurry would be dewatered, development waste added as well as cement, if desired, and the mixture delivered pneumatically to the stope to be filled. Main pipelines and fittings would be manufactured from abrasion-resistant steels with a hardness of up to 700 Bn (Brinnell). Elbows would be made with replaceable liners. (Such high-quality steel would be required only in the more permanent installations.) In the stopes themselves, mild-steel pipelines could be utilized and, when worn, patched and finally rejected. Rubber hose could also be used for elbows and for the final placement section in the stope. A properly graded fill could result in a relative density of 79 to 83% due to the high impact of larger particles driving the fines into the voids.

19.3.2.2 Sources and Materials

Sources of materials for pneumatic filling are much the same as those used for waste fill. Again, a low-cost, convenient, and accessible source with a simple preparation plant may make a mine requiring fill economic while other more sophisticated systems would not be commercial. Typical sources may include the following:

1. Deposits of overburden such as talus, sand and gravel, glacial till, dune sand, or river gravel.
2. Mine dumps.
3. Mine development waste.
4. Sink-float reject.
5. Coal preparation plant refuse.
6. Smelter slag.
7. Mill tailings or mixtures of mill tailings and waste rock.

Cement can be added with appropriate moisture to any of the above for greater strength and stability. The primary difference from ordinary waste fill is in the methods of preparation and delivery.

19.3.2.3 Preparation, Delivery Methods, and Equipment.

Dense-phase pneumatic handling techniques are used for such fine-grained materials as cement, rock dust, pulverized coal, and chemicals that are readily flushed. Air at high pressure is usually provided by a reciprocating compressor. Pneumatic conveyors of the dilute-phase type are used to handle the materials used in mine backfilling. Air supplied by a fan or positive-displacement blower is used to transport the material at relatively low pressures, usually ranging from 5 to 20 psi (34 to 138 kPa)

(Powell, 1982). The particles are suspended and carried in the airstream.

Dilute-phase pneumatic conveying has been applied in coal mines since the 1920s to convey rock or mine refuse into underground workings. Its use is mainly for roof support in those areas where coal has been extracted. It has a secondary application in that it does reduce or eliminate the disposal of rock waste on the surface of mines. In more recent years, the same technique has been applied for hoisting coal out of mines through vertical shafts. Similarly, pneumatic conveying has been used to hoist shaft or raise-boring cuttings vertically up a bored shaft (Powell, 1982).

It is recommended that the pipeline diameter should not be less than three times the largest particle size, though in practice in the mining industry, 8-in. (203-mm) I.D. pipes are commonly used even though there is an occasional lump of 3 in. (76 mm). Rubber-lined pipes are not recommended for pneumatic conveying systems due to the high friction factor and sharp particles of rock slicing the rubber. As the material is abrasive, and rapid wear can take place in the pipeline, consideration is given to abrasion-resistant steel pipes where the inner face is hardened to 500 Bn in order to reduce the wear. There is an increase in cost compared to mild steel pipe, but it is justifiable from an economic point of view, providing that the wall thickness does not have to be increased. This, of course, makes the pipes much heavier, increasing the labor cost for laying and recovering the pipelines. When handling sand, plastic and fiberglass pipes have been used, although these wear more rapidly than steel pipes and are used for the last sections in the pipeline only where they are to be constantly coupled and uncoupled. The detrimental costs in high wear rates is offset by the savings in labor, with one man able to carry a 10-ft (3-m) length of plastic pipe compared with the requirement of two men for an equivalent length of steel pipe (Powell, 1982).

The use of early-bearing compact stowing is considered particularly appropriate for support beneath objects or areas that are subject to mining damage. The most compact possible stowing pack with the lowest possible pore volume can only be obtained in the presence of a grain-size distribution curve of the stowing material that largely coincides with the Fuller curve for concrete aggregates. It is of particular importance that the stowing material contains a sufficient amount of fine and finest grains to allow the formation of the most densely compressed spherical packing. Flotation slurries with their particularly high clay content are suited to this purpose. Electrostatic filter ash from power stations with its long-term hydraulic binding effect can also be used for admixture (Voss, 1983).

In order to keep dust levels at a minimum at the delivery point, it is essential that neither too little nor too much moisture be present in the feed. Too much moisture can result in fog carrying dust, just as too little moisture results in airborne dust. A moisture content averaging 8 to 10% for material with a good size distribution range will usually result in the highest density of fill. Water is generally injected through a wetting ring near the discharge nozzle, especially if cement is used. Too much moisture in the delivery line may result in build-up. Water added at the head end of the system helps to lubricate the pipeline but must be controlled carefully.

A pneumatic conveying system consists essentially of an air supply, an in-feed arrangement, a pipeline with necessary elbows, and a discharge. A heavy-duty pneumatic conveying system also consists of these essentials but with several important differences that are necessary to accommodate the large abrasive material. Component parts are described below.

The air requirement for a heavy-duty pneumatic system, say, for example, conveying 200 tph (181 t/h) of refuse into a coal mine, will require from 3000 to 5000 cfm (85 to 142 m^3/ min) of air. With a pipeline diameter of 8 or 10 in. (200 or 250 mm), pressure requirements for the air supply will be in the region of 15 psi (104 kPa) and, allowing for some reserve, should have a maximum pressure available of 18 psi (124 kPa). Such an air supply is most economically provided by a positive-displacement blower, of which there are several types and makes available.

This method of producing pressurized air is probably the most economical, and in most instances, blowers of this size can be direct coupled to an electric motor operating at 1750 rpm. Blowers of this type are simple in construction and easy to maintain. Provided they are selected to operate at a known height above sea level, so that there is always sufficient quantity of air flowing through the blower to dissipate the heat of compression, they will have a long life. They have one inherent disadvantage, in that the air produced is pulsating, and, close to the blower itself, the noise is objectionable. To overcome this problem in the confined space of a mine, the blower and electric motor are enclosed in an acoustically insulated enclosure provided with the necessary access doors. The components are mounted on a substantial fabricated steel base with sufficient rigidity to prevent strain on either the motor, blower, or coupling when located underground in the mine on an uneven floor. The mine air is often polluted with dust, which could in time affect blower operation by wearing the rotors. This increases the air gaps and lowers the efficiency. The air is carefully filtered before being drawn into the blower by means of a panel filter that can be readily changed when necessary. The starter for the motor is mounted on the same skid-base, which reduces the amount of electrical connection work the mine electricians have to undertake when these units are frequently relocated.

Being a positive-displacement machine, the blower supplies air at a pressure that increases according to the demands on it, and if the pipeline becomes choked with material or becomes crushed from a roof fall, the pressure continues to increase until the electric motor is switched off by the overcurrent relays. To ensure that physical damage does not take place, there are several protection devices built into the blower units, including the following:

1. Pressure relief valve set at 18 psi (124 kPa).
2. Rupture disc.
3. Series of pressure-sensing switches:
   a. To reduce infeed.
   b. To shut down the motor if pressure continues to increase.
   c. To shut down the motor if air temperatures become too high.
   d. To provide electrical overload tripouts on the motor.

Reciprocating compressors can be used as an air supply and were used for many years in Europe to supply air to pneumatic backfill systems in coal mines. However, mine compressed air is usually supplied at a pressure of 90 to 100 psi (621 to 690 kPa), which is much higher than that required in a pneumatic conveying system of the dilute-phase type. Pressure has to be reduced to approximately 20 psi (138 kPa) through an orifice plate or pressure-reducing valve. In European coal mines, these compressors were already available for powering, by compressed air, other machinery in the mine, such as coal cutting machines and drills. They were, therefore, readily available for operating the backfill equipment, which was most often done on a nonproduction shift when the other machinery would not be in use in the mine. The positive-displacement blower has, however, proven to be the most economic source of pressurized air for pneumatic conveying of coal or rock in the mining industry.

There are several methods of introducing material into the pressurized airstream from atmosphere, namely, lock gates, pressure vessels, and rotary valve feeders. The rotary valve feeder has proven to be the best infeed device for handling large-sized abrasive materials. Occasionally, this type of feeder becomes jammed by foreign material that cannot be sheared. Provision of a fluid coupling or a shear pin prevents damage. It is often found that a piece of rock, especially shale or the weaker sandstones, will break after being subjected to two or three blows. Arrangements are made so that when the feed wheel is stopped suddenly, it immediately reverses for a short distance and then turns forward again. Several repetitions will often shear the rock or move it to the end pocket, allowing the feeder to continue turning.

These feeders are made of Ni-hard castings of 700 Bn hardness for wear and abrasion resistance. The stator halves are also adjustable so as to compensate for wear. Wear on end housings is reduced by pressurizing them with clean air. The feeder unit is assembled on a skid base that also accommodates the pipeline inducer where the material is accelerated into the airstream. A rotary valve feeder capable of handling 200 tph (181 t/h) weighs 6000 lb (2720 kg). A skid-mounted power center contains a hydraulic oil supply for the drive motor of the rotary feeder, auxiliary air for the end housings, backup controls for the system, a variable volume hydraulic pump that allows the operator to adjust the speed of the rotary feeder, a hydraulic relief valve, oil tank, filters, forward/reverse valve, and pressure sensing devices for providing optimum load conditions.

A control panel is attached to the unit by a 50-ft (15-m) umbilical cord, allowing the operator access to all functions of the pneumatic system.

Pipelines generally are made of specially hardened steel, but also can be of mild steel, plastic, or fiberglass depending on the specific application required. They should be kept as straight as possible and also should be turned regularly to even out the wear that occurs primarily on the bottom of the pipeline. Elbows are designed with replaceable hardened steel bricks on the outer radius. These together with an impact section and an accelerator assure that the direction of flow is changed smoothly.

As the material flows along the pipeline, the larger particles tend to bounce along the bottom, with the fine particles being carried in the airstream and the intermediate particles in between. As the various sizes fall out of the airstream, saltation takes place. The material then forms slugs, which results in inefficient flow. To overcome this problem, devices called kickers are installed that direct the material back toward the center of the airstream, alleviating the problem.

For mine backfill systems, the material is ejected from the open pipe at high velocity and the air allowed to flow away freely. It is this property of a pneumatic conveying system that makes its application attractive for backfilling mine refuse in mine galleries, due to the compaction achieved when the particles impact on each other at the face of the placed material. This not only results in the maximum volume of material being confined in the minimum of space, but also helps to stabilize the support pillars and roof, which will reduce collapse and subsidence at the surface (Powell, 1982).

19.3.2.4 Advantages

Advantages of the method include the following:

1. Improvement of the face climate. The air delivered by a pneumatic system may be valuable for ventilation purposes.
2. In uranium mines, better control of radon gas emission is achieved. There is less air leakage through a tightly packed fill.
3. In coal mines, better control of methane gas may be achieved by better overall control of ventilation due to the high density fill.
4. Reduction of surface subsidence is possible, with as little as 13% convergence achieved in some European coal mines (Voss, 1963).
5. Reduction of internal mine damage, that is, damage to the mine structure. Bumps or bounces in coal mines and rock bursting or excess pressure on the workings can be controlled by high-density, early-bearing, pneumatically placed fills.
6. Use of underground development waste saves the cost of delivery to the surface and reduces the environmental problems of surface storage.
7. Problems of excess water in hydraulic fills are eliminated.
8. High-velocity impact yields high-density fill with good strength.

19.3.2.5 Disadvantages

1. If the pneumatic system is located underground (perhaps due to the great distance to surface), it will be necessary to provide a transportation system from surface for the fill material.
2. Dust from highly siliceous waste can be a problem in the stoping area being filled and may require additional ventilation.
3. Initial capital costs may be high due to the need for specially hardened steels in feeders, pipelines, elbows, fittings, and nozzles.
4. Operational costs are high due to the requirement for large volumes of compressed air and high wear on components.
5. Handling and placing of heavy pipeline and fittings can increase operational costs. However, it may be shown that the labor cost associated with the lighter pipelines may more than offset the added pipe cost (Soderberg and Corson, 1976).
6. High noise level at the blower requires special sound insulation.
7. Pneumatic filling is not well adapted to small crooked veins in cut and fill operations, for these reasons:
   a. The number of elbows causes increased wear, reduced velocity, and decreased density in place.
   b. Short time duration of filling compared to the cost of setting up and filling.
   c. Difficulty in producing a level working floor for clean-up purpose (Soderberg and Corson, 1976).
8. For an underground installation in a small mine, the equipment is large and difficult to move and set up. A semi-permanent installation is almost mandatory.
9. The system may be restricted to surface installations due to the size of equipment and accessibility of materials. This may result in excessive power use and high cost of delivery due to long lines.

19.3.3 HYDRAULIC FILL WITH DILUTE SLURRY

19.3.3.1 General

The first use of hydraulic fill is thought to have been in 1864 at Shenandoah, PA, when breaker waste and culm were slushed into old mine workings to protect the foundation of a church (Lightfoot, 1951). Shortly afterward, the method was applied to several eastern coal mines for area fill to control subsidence. In 1884, hydraulic backfilling was used to control a mine fire in the Schuylkill region of Pennsylvania.

The idea was taken to Europe by German engineers and developed, successfully, at the Myslovitz colliery in Upper Silesia in 1901. From there it was applied to several mines throughout Europe. The practice of area filling was taken to South Africa in 1909 to the Village Gold mine in the Transvaal.

At the start of World War I, mill tailings were being used for mine backfill on a limited scale in the Cripple Creek district of Colorado. By the end of the war, the method was introduced by the Anaconda Co. in Butte, MT, to fight mine fires. Since 1917, more than 7½ million tons (6.8 Mt) of mill tailings have been placed hydraulically in the Butte mines.

During the 1920s, many world mining districts used fill material placed as a slurry. The Matahambre mine in Cuba was one of the first to make extensive use of hydraulic backfilling as an integral part of their mining method. They also pioneered in the use of rubber-lined pipe as a transporting system.

Homestake Mining Co. was one of the first in the United States to exploit simultaneous hydraulic backfilling as a part of the mining cycle. In 1932, the company developed a method of handling its washed cyanide tailings in pipelines and introduced the sand portion for stope fill.

After World War II, many mines in the western United States and in Canada applied hydraulic backfilling methods, and developed mining cycles based on the use of classified mill tailings. This progress was due in part to the increased experience within the minerals industry in the handling of solids in liquid suspensions. The perfection of rubber-lined slurry pumps and the expanded technology of materials handling in pipelines, together with the application of wet cyclones for dewatering and desliming, led to economical applications in the mining field (Stewart, 1958).

Hydraulic fill may be described as a dilute slurry varying generally from 35 to 75% solids. There are, however, variations above and below these limits. The slurry may be pumped to the stoping area by various types of pumps or delivered by gravity flow.

A survey by the Fill Subcommittee of the Canadian Advisory Committee on Rock Mechanics (CACRM) indicated that hydraulic fill prepared from mill tailings is the most common material in current use. The 10,830,000 tons (9,824,976 t) of fill placed in reporting mines in 1971 consisted of 8,240,000 tons (7,475,328 t) of hydraulic mill tailings, 1,370,000 tons (1,242,630 t) of rock, 870,000 tons (789,264 t) of hydraulic alluvium and 350,000 tons (317,520 t) of miscellaneous material including dry sand, slag, float, etc. Some 228,000 tons (206,840 t) of cement was used for floors and fill stabilization. Ninety percent of the ore involved was produced by mining methods using concurrent filling, and 70% of this ore came from overhand cut-and-fill methods. No fill was used primarily for subsidence control, although this and fire control are important applications of hydraulic filling in various parts of the world.

Wall support was recognized by all mines as the most important function of fill. Use as a working floor followed, with void filling and tailings disposal, as the third major use.

All mines indicated stabilization of the operating stope and adjoining pillar as a prime function of fill within the wall support category. Stabilization of access openings and boundary pillars is the next important function in small and medium mines, while large mines identify the stabilization of peripheral stopes and ore zones as a primary requirement. Control of surface subsidence was listed as the least important (Dickhout, 1973).

19.3.3.2 Sources

Of the several possible sources for hydraulic fill sands, the simplest is mill tailings, when the mine and mill are integrated. Alluvial sand or gravel deposits can also be considered as a source of fill material, when the mill is located at a distance from the mine, or when mill tailings are unsuitable.

Conventional backfill materials also include crushed mine waste, rock, glacial till, smelter slags, mixtures of mill tailings and –l-in. (25-mm) gravel, crushed open pit material, dune sand, and consolidated mill tailings, using cement or other binding agents. Mill tailings may be classified or unclassified, depending on the grind in the mill. The greatest fill strength is achieved when the material size gradient results in the maximum filling of voids between particles.

19.3.3.3 Preparation and Materials

Table 19.3.1 reports on a survey of plant operating conditions and describes typical hydraulic fills from 6 different mines. Several of these are no longer in operation, but the table illustrates the spread of classified mine tailings that can be used.

Typical screen analyses, for classified mill tailings, as used for backfill in the mining industry are shown in Figs. 19.3.9 and 19.3.10.

The following comments on the basic characteristics of various hydraulic fills are adapted from a paper by Dickhout (1973).

Corson in 1966 showed that unconfined compressive strength increases with increased cement use (Fig. 19.3.11). He also found that inclusion of minor amounts of dispersant yielded a significant increase in bearing strength of the leaner sand-cement mixes. Although this seemed important, there is no known application of the use of dispersants for this purpose in industry.

McCreedy and Hall, in 1966, reported further that fill, to which Portland cement in ratios of 1 : 30 to 1 : 20 had been added, behaves as follows:

1. Strength increases with age.
2. Strength increases significantly with increase in pulp density.
3. The percolation of acid mine water through the fill does not seem to affect the strength.
4. Using acid mine water to transport the cemented fill does not seem to affect the ultimate strength.
5. The various special Portland cements do not seem to produce any substantial differences in strength, as compared to conventional type 1 cement.
6. A Proctor needle can be used for determining the in situ compressive strength of the cemented fill.

Rawlings, Toguri and Cerigo, in addition, reported in 1966 that:

1. Compressive strength of cemented fill decreases with increased moisture content.
2. Permeability decreases with increased cement content.
3. Permeability decreases as curing progresses.

The Canada Cement Co., in cooperation with International Nickel Co. and Falconbridge Nickel Mines, carried out monumental studies confirming much of the earlier work, and introducing some new findings. Luka and Weaver in 1970 reported that:

1. Varying the fineness of Portland cement does not produce great changes in the strength of the cemented fill.
2. The addition of fly ash, to economize on the amount of cement required, is not particularly attractive.
3. The grain-size distribution of the tailings is significant.
4. Flocculants have beneficial effects on strength and percolation rate, and in reducing losses in the drainage water.
5. Curing temperatures of between 50 and 85° F (10 and 29° C) do not have much influence on the ultimate strength.

Further findings reported by Corson in 1970 include:

1. Compressive strength increases with increased cement content, but varies according to the properties of sand used. In lean mixes, the uniformity of the tailings material is the most significant factor.
2. Compressive strength increases more rapidly when dry cured than wet cured, which is the usual condition in the mine.

Table 19.3.1. Survey of Plant Operating Conditions

	Screen Analysis								Maximum Pulp Transportation Distance, ft (m)		
	Mesh	% Wt	Mineral Composition	Pulp Density %	Dry Solids Sp Gr	Percolation Rate, In./hr (mm/h)	Pipeline Size, I.D.	Friction Loss	Horizontal	Vertical	Pumps
American Smelting & Refining Co., Galena Mine, Wallace, ID	−35 −48 −65 −100 −150 −200 −325 −325	0.80 2.80 7.20 14.80 17.40 20.20 19.60 17.00	Siderite, quartzite, some pyrite	68 to 74	3.2	2 to 3.06 (51 to 71)	3 in. (76 mm) pipe some $2\frac{1}{2}$ in. (63.5 mm) rubber-lined	No data	1780 (543)	2380 (725)	None (gravity system)
Calera Mining Co., Cobolt, ID	−65 −100 −200 −325 −325	3.0 10.0 25.0 22.0 40.0	Biotite, schist phyllite, quartzite, minor pyrite and pyrrhotite	54 to 60	3.9	No data	4 in. (102 mm) wood stave and 4 in. (102 mm) steel pipe	14.2 ft (4.3 m) of head/100 ft (30.5 m) in nominal 4 in. (102 mm) pipe at 400 gpm (25 L/s), 58% solids	2800 (853)	230 (70)	None (gravity system)
Day Mines Inc.	−65 −100 −150 −200 −200	5.0 18.0 20.0 17.0 70.0	Quartzite, quartz, siderite, ankerite, sericite, chlorite, minor pyrite				3 in. (76 mm) nominal steel pipe	No data	2500 (762)	750 (229)	None (gravity system)
Anaconda Co., Butte, MT	−35 −48 −65 −100 −200 −400 −400	5.7 15.0 30.6 24.3 28.3 4.7 1.4	Quartz, feldspar, sericite, minor pyrite	65 to 70	2.9	4 to 8 (102 to 203)	4 in. (102 mm) nominal steel pipe (rubber-lined) $3\frac{1}{2}$ in. (89 mm) 5 in. (127 mm) wood stave pipe on surface	4.9 psi/100 ft (34 kPa/30.5 m) in 5 in. (127 mm) wood stave pipes at 70% solids, 310 gpm (19.6 L/s), 5.07 fps (93 m/min).	1100 ft (335 m) (2 in. pipeline) 4000 ft (1219 m) (3 in. pipeline)	250 ft (76 m) 3800 ft (1158)	Lexington 40 hp A.S.H. C frame* 1170 ft (357 m) horizontal in 5 in. (127 mm) wood stave pipe *Allen-Sherman-Hoff, centrifugal.
Homestake Mining Co. Lead, SD	−80 −100 −150 −200 −200	2.8 10.1 20.7 28.0 38.4	Cummingtonite, biotite, quartz, 7 to 8% sulfides	50 to 65	3.0	2 (51)	6 in. (152 mm) steel pipe (rubber-lined) 5 in. (127) (rubber-lined) $4\frac{1}{2}$ in. (114 mm) 2600 to 4100 level 4 in. (102 mm) steel pipe in stopes	No data	2460 (750)	745 (227)	None (gravity system)
Kerr-Addison Gold Mines Virginiatown, ON	−65 −100 −150 −200 −200 −325 325 −10	0.2 0.8 6.2 24.1 31.7 37.0 1.0	Silicates and carbonates of Ca, Mg, Fe	55 to 60	2.88	3.5 (89)	6 in. (152 mm) pipe (rubber-lined) $5\frac{9}{16}$ in. (141 mm) Flexible hoses on each level, $5\frac{9}{16}$ in. (141 mm) Extra heavy black pipe on levels $3\frac{53}{64}$ in. (97 mm)	Approximately 21 ft/100 ft (6.4 m/31 m) level lines at 60% solids, 86 dry tons/hr (78 t/h) total volume 352 gpm (22 L/s)	1480 (451)	316 (96)	None (gravity system)

Source: Adapted from Stewart (1958).

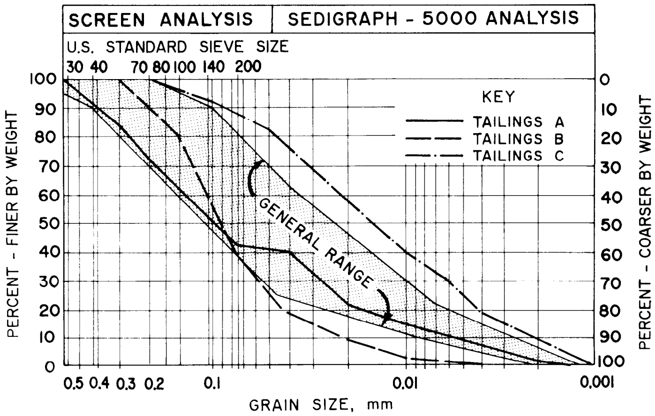

Fig. 19.3.9. General range of US metal mine tailings (adapted from Soderberg and Busch, 1977).

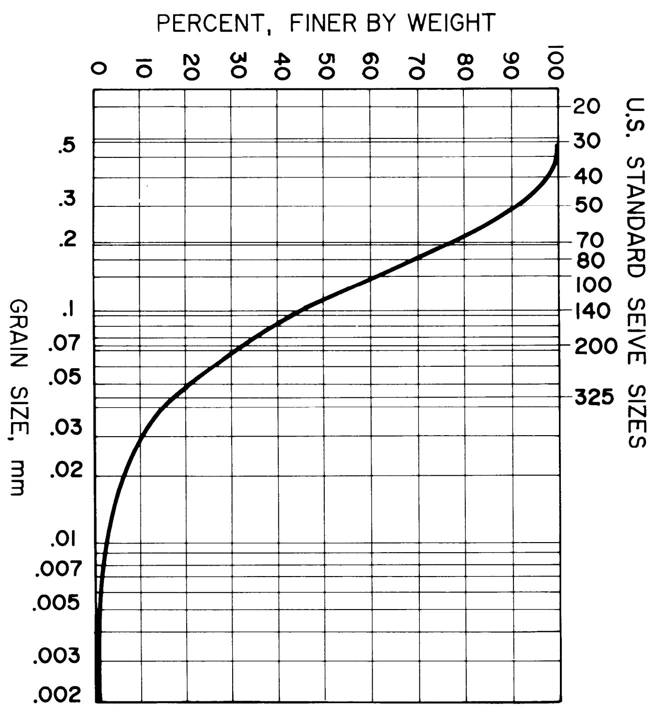

Fig. 19.3.10. Grain size distribution of a moderately well-graded fill (Dickhout, 1973).

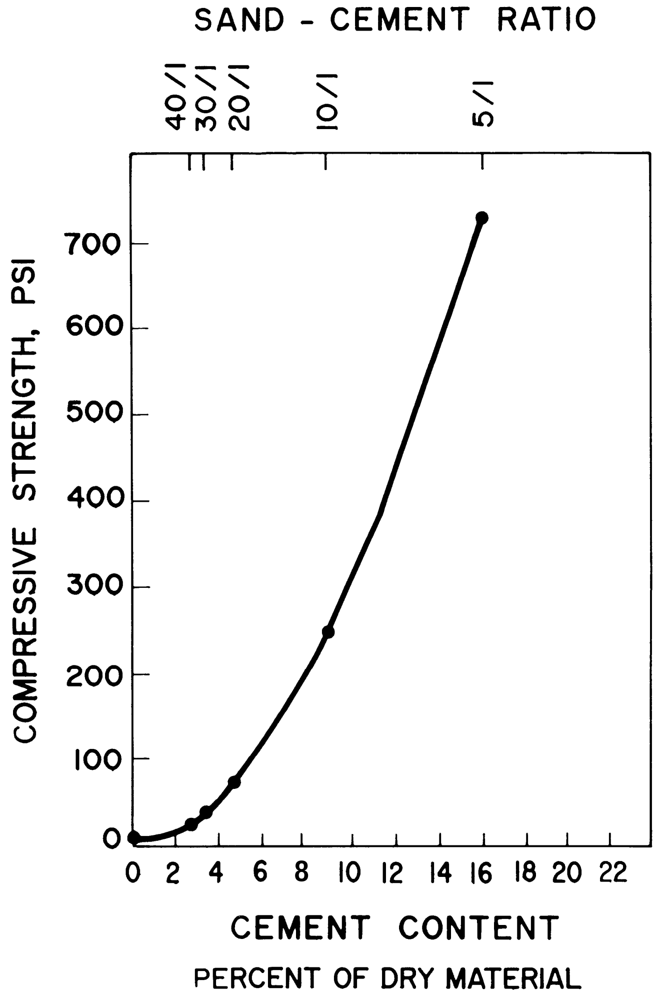

Fig. 19.3.11. Unconfined compressive strength vs. cement content (Dickhout, 1973). Conversion factor: 1 psi = 6.9 kPa.

3. Compressive strength is significantly higher in all sand-cement ratios, as density is increased by approximately 10 lb/ft^3 (160 kg/m^3).

4. Triaxial tests showed strengths increase with increases in either cement content or confining pressure. At a sand to cement ratio of 40 : 1, the finer material exhibited much higher cohesion than the well-graduated material, whereas the well-graded material had the larger angle of internal friction and was capable of supporting a higher normal stress.

Thomas, in 1971, published a comprehensive report on cemented-fill practice at Mount Isa, and again confirmed many previous findings as well as introducing new data, which included:

1. Moisture content decreases with increased cement content.
2. Void ratio decreases with increased cement content.
3. Compressive strength decreases with increased void ratio.
4. Compressive strength increases with increased fines addition.
5. At Mount Isa, ground copper reverberatory furnace slag, acting as a pozzolan, greatly increases the strength of cemented fill in proportion to the amount of slag added.

Nicholson and Wayment, in connection with vibratory compaction of mine hydraulic fill in 1967, reported that:

1. Probe-type concrete vibrators give effective compaction.
2. Density of fill increases with percentage solids by weight of fill poured.
3. Moisture of about 15% is optimum for vibratory compaction.
4. Fills with a uniformity coefficient of around 5 yield the lowest final void ratio after compaction.

Coates and Yu-in in 1969 further reported that the in-place density of most fills can be increased by modifying the grain-size distribution, and that by so doing, the following results were obtained:

1. The volume of stope tilled by 1 ton (0.9 t) is decreased, thereby increasing filling costs.

2. The stiffness of the fill is increased.

3. The percolation rate is decreased.

4. The amount of cement required for consolidation is decreased.

Corson, in 1971, reported on the field evaluation of hydraulic backfill compaction at the Lucky Friday mine. Measurements taken in adjacent stopes, one containing compacted fill and the other normal fill, showed the following:

1. The density was increased from 100 lb/ft^3 (1602 kg/m^3) to 104 lb/ft^3 (1666 kg/m^3) by compaction.

2. Pressures and closures, taken in both stopes, continue to increase with time.

3. Somewhat less strain was recorded on the 7th floor of the stope containing the compacted fill, than on the same floor of the stope containing the regular fill.

4. Pressure build-up was slower in the regular fill (Dickhout, 1973).

19.3.3.4 Applications

The hydraulic sandfill system at the Galena mine, near Wallace, ID, originally used unclassified mill tailings. Slimes were decanted from the top of the storage tanks, initially, but the method has been changed at present to cyclone classification.

The following comments are adapted from a paper by Suveg (1970).

The mill tailing was pumped at 20% solids from the mill by two 6-in. (152-mm), rubber-lined centrifugal pumps in series, driven by 25-hp (19-kW) motors through a 6-in. (152-mm) line, partly steel pipe and partly wood stave pipe, for a horizontal distance of about 850 ft (259 m) at a static head of 115 ft (35 m) into a surge tank. The tank had three conversion outlets and the tailings could be accumulated in either of the three 14- by 18-ft (4.27- by 5.49-m) steel tanks. Each tank initially had a 6-in. (152-mm) overflow pipe for decanting the slimes. The plant handled sand at a rate of 20 tph (18.2 t/h) at 70% solids.

Agitation was provided by both compressed air and by a 4-ft (1.22-m) diameter propeller-type agitator, driven at 110 RPM by a 30-hp (22-kW) reversible motor. Compressed air agitation was provided by eight peripheral nozzles and four bottom nozzles.

Compressed air at 100 psi (690 kPa) was introduced into the bottom nozzles and allowed to agitate for 10 minutes to free the propeller, then the agitator was started and the side nozzles turned on. After thorough agitation, the pour was started.

An independent telephone system was installed from the sandhouse to the stopes for communication. The sand was poured through a 3-in. (76-mm) standard pipeline. In the shaft, to minimize wear and maintenance, the 3-in. (76-mm) pipe was lined with ¼-in (635-mm) rubber. Sand presently flows by gravity, a vertical distance of up to 4900 ft (1494 m). Before a pour is started, the line has to be cleared. High-pressure water was injected for 30 seconds, followed by compressed air until the water reaches the stope. The sandman notifies the sand house operator and then bleeds back the line. After the line was cleared, water was introduced again into the line followed by sand. The pour was started at 70% solids and finished at 65%. One hour was required to pour one tank. Two tanks were poured in sequence. An average stope required 5 to 6 tanks of sand during each filling cycle. A sand pour had a percolation rate of 6 in./hr (152 mm/h), and this made it possible to blast the next cut on the following shift.

After the mining cycle was completed, the stope was cleaned out and a crib chute raised. Five-inch (127-mm) × 8-in. (203-mm) cribbing was used. The chute was burlapped and the stope filled (Suveg, 1970). Cement capping was used for several years, but was dropped about 10 years ago as being too expensive and ineffective due to loss of cement in the overflow from the stope.

In most flotation concentrators in the Coeur d'Alene mining district, the grind is fine enough to produce an excess of slimes in the sandfill. The slimes may be decanted but can also be removed by cycloning, as was practiced at the Star, Lucky Friday, Sunshine, and Bunker Hill mines. If too much slimes (– 325 mesh or 0.044 mm) are left in the fill, a layer of sticky clay will be left on the fill surface that is difficult to work on. An average 20 to 22% of –325 mesh (0.044 mm) material has been found to be satisfactory for a well-graded fill without excess slimes.

During the period from 1960 to 1985, the Lucky Friday mine near Mullan, ID. used the following system:

Tailings containing 71% solids were pumped from the mill with 4- × 4-in. (102- × 102-mm) rubber-lined centrifugal pump with a 20-hp (15-kW) motor through 902 ft (275 m) of 5-in. (127-mm) standard pipeline at 60-ft (18.3-m) static head. Two hundred forty gpm (15.2 L/s) was pumped at a velocity of 5.3 fps (1.62 m/s). This pump was started manually by mill personnel, but was shut down by remote control from the stope where sand was being poured.

Sand from the mill pump was delivered to a small constant-head receiving tank located over the main storage tank near the No. 2 shaft. A 3- × 3-in. (76- × 76-mm) rubber-lined centrifugal pump was fed by this tank and provided the pressure to operate two 10-in. (254-mm) diameter cyclones in parallel. The cyclone underflow containing 22% –325 mesh (0.044 mm) sand fed directly into the tank. Cyclone overflow was returned to the mill tailings line. The 18-ft (5.49-m) deep by 17½-ft (5.33-mm) diameter storage tank held 120 tons (109 t) and was excavated out of solid rock. It has a concrete bottom and a small raise connects to the manway compartment of the shaft. It is equipped with an agitator having a 48-in. (1.22-m) diameter, rubber-covered impeller, driven by a 30-hp (22-kW) motor at 95 rpm. Sand slurry was poured at 68 to 70% solids. The sand fill system was remotely operated. Four signals were transmitted each way over a single two-conductor, type 50, size 16 cable, utilizing 380 to 9900 cycle signals and transistor-operated relays. The receiver and transmitter units were developed to company specifications. The four signals from tank to control panel operate lights that indicate the following:

1. Storage tank full.
2. Storage tank empty.
3. Agitator operating.
4. Main slurry valve open.

The four-signal control panel to the tank was switch-operated and performed the following functions:

1. Close the solenoid valve controlling water pressure to the jacket of the main sand valve.

2. Operate the 2-in. (51-mm) solenoid valve controlling air to purge sand lines.

3. Operate the 2-in. (51-mm) solenoid valve controlling water to flush sand lines.

4. Close a system shutdown switch that operates a solenoid controlling pressure to an air cylinder on the slurry diversion funnel in the mill. This also initiates a timed automatic flushing cycle that clears the line from mill to storage tank.

The primary purpose for automation was to reduce the number of personnel required to operate the plant to one employee/shift and to give better control of pouring. This remote control system was in use from 1959 to about 1980 and has now been replaced by a high-density, consolidated-fill system (see 19.3.4.5).

A 2½-in. (62-mm) Victaulic standard-weight pipe with ¼-in. (6.35-mm) rubber lining to 2-in. (51-mm) I.D. was used for transporting the sand down the vertical shaft. Sand has been transported up to 5000 ft (1524 m) vertically and 2000 ft (610 m) horizontally with this system. Pouring rate was 40 to 50 tph (36 to 45 t/h), and the tank was filled at a rate of 16 tph (14.5 t/h). The stopes were untimbered and from 5 to 15 ft (1.5 to 4.6 m) wide.

The latest addition to hydraulic fill practice (1962 to 1970) was consolidation of the fill by cement addition. The first work involving cement addition was done by International Nickel Company followed by the Mayflower mine in Utah and the Homestake mine in South Dakota. Hecla Mining Company conducted experimental work at the Lucky Friday mine in 1962, but it was not until early 1964 that a decision was made to use cement-stabilized fill on a regular production basis in the mining system. The Lucky Friday operation was then equipped for developing a cement capping on top of the regular hydraulic sand fill. Pilot test work was successful and a system was installed. For many years, most operating companies in the Coeur d'Alene mining district used cement capping in their stopes. Cement capping provided substantial savings. Experience with hydraulic sand fill shows that 8 to 10 in. (200 to 250 mm) of sand are removed in conjunction with ore in the slushing cycle during the final clean out of the stopes. This indicated that the recycled sand amounted to 3.6% of the mill tonnage. Cement capping stopped this loss and increased the tonnage of ore put through the mill by an equal amount. Cement capping was placed at a ratio of 1 part type I Portland cement to 7 parts sand. The capping was placed at an average thickness of 6 in. (150 mm). This provided a good clean-up surface for the overhand mining method used. The first floor on each level was poled over with 6-in. (150-mm) diameter timber and filled to within 18 in. (0.46 m) of the back with 1 : 20 ratio sandfill for consolidation. This provided a strong back to mine under from the level below.

In actual practice, some additional, hidden savings that appear to be quite substantial, were encountered. They are

1. Speed-up of stope clean-out after each cut.
2. No ore is lost in sand.
3. No sand in chutes to cause hang-ups.
4. Miners and supervisors devote more effort to obtaining level stope backs.
5. Sandmen are more particular about leveling out the sand pours.

The bulk cement was stored in a 10-ft (3-m) diameter by 20-ft (6-m) high cement silo equipped with an 8-in. (203-mm) by 8-in. (203-mm) rotary feeder and variable speed drive controlling the feed. The cement was fed into a fluidizing cone where it was slurried and then pumped through a 2-in. (51-mm) line into the 3-in. (76-mm) sandfill line in the shaft.

At the Galena mine, near Wallace, ID, an 8 : 1 sand to cement ratio was used for capping. The cemented sand has been found to have several applications in addition to capping stopes:

1. Stabilization of backfill in underhand mining.
2. Consolidation of the fill material in the initial pour of the first stope cut, to permit pillar removal.
3. Consolidation of fill material in intermediate stope drifts, or in any areas that will eventually have the ore mined from underneath.

The mining of irregular ore bodies or paralleling vein splits has also been improved with the use of cemented sand.

To summarize, cement is used in hydraulic sand fill mainly for two specific applications: (1) floor capping and (2) consolidation of fill material. These applications have resulted in substantial savings, improved grade control, and increased flexibility of mining methods (Suveg, 1970).

At Homestake Mining Company's Bulldog Mountain mine in Creede, CO, an undercut and fill mining method using LHD units was developed to mine a section of vein lying 50 to 80 ft

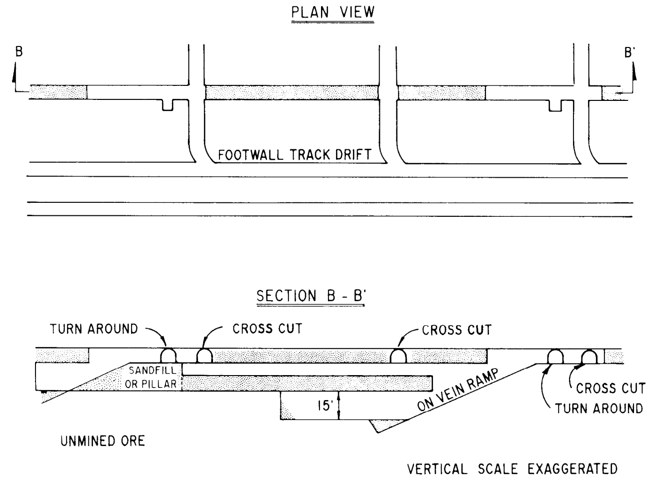

Fig. 19.3.12. Idealized on-vein LHD-UCF, no level below (Simonson and Logsdon, 1983).

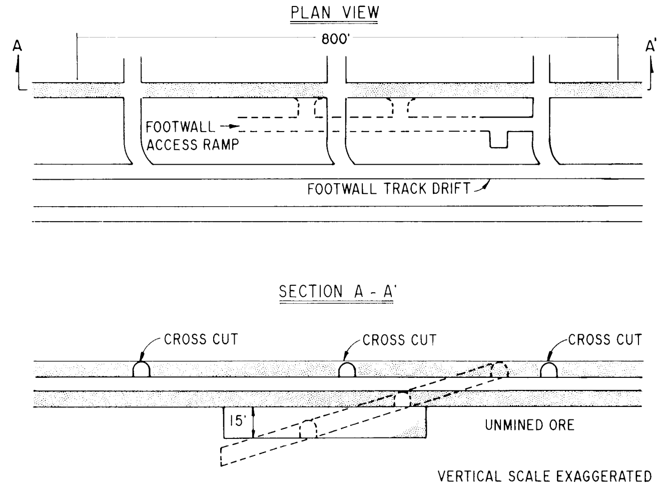

Fig. 19.3.13. Idealized off-vein LHD-UCF, no level below (Simonson and Logsdon, 1983).

(15 to 24 m) below the 9000 level. The underhand method was used because this block of ore did not justify driving a new level 150 ft (46 m) below the 9000 level. The original mining method at the Bulldog was ascending cut and fill but was changed to undercut and fill in the early 1970s due to the unconsolidated nature of the vein (Simonson and Logsden, 1983).

Both on- and off-the-vein ramping systems were developed, as shown in Figs. 19.3.12 and 19.3.13. Stoping cuts vary from 12 to 16 ft (3.7 to 4.9 m) in height, depending on the competency of the walls.

After a stope cut had been mined to its endline, it was prepared for sand backfilling. The stope floor was leveled with 6 to 12 in. (152 to 305 mm) of broken ore. This amount of ore was generally left as mining proceeded to maintain a smooth roadbed. In sandfill preparation, it was left to act as a cushion for the next lower stope cut. Round timbers 6 to 8 in. (150 to 200 mm) in diameter were then laid from wall to wall on 7-ft (2.1-m) centers. These timbers were set in hitches that had been installed in the walls and/or secured with $\frac{3}{8}$-in. (9.52-mm) cable

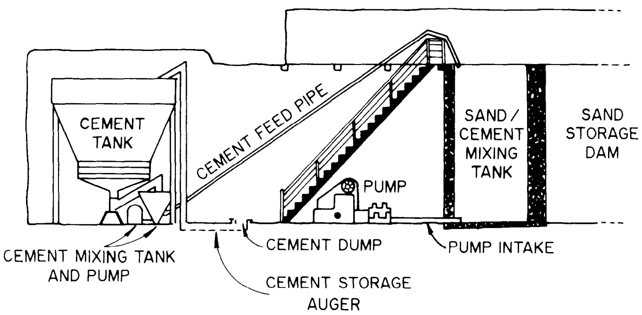

Fig. 19.3.14. 9700 sand plant. Bulldog Mountain mine (Simonson and Logsdon, 1983).

attached to an eye pin installed 4 ft (1.22 m) up the wall. (Often it is only necessary to hitch or cable tie the timber on the hanging wall and simply wedge the timber to the footwall.) Unmilled round timbers have recently replaced more expensive 8- by 8-in. (203- by 203-mm) milled timber in the sandfill preparation.

Chain-link wire was laid over the timber for the length of the stope. The wire was nailed to the timber and stretched along the floor between the timbers. Care was taken to get the timbers and wire on the floor. This allowed the cemented sand backfill to be poured with no danger of slabs hanging below the wire and timber on the next cut below. If the wire was stretched across the tops of the timber, as was previously the method, uncemented sand was poured first to fill this 8-in. (203-mm) gap. The sand then harmlessly fell out below the wire on the next lower cut. However, substantial ore dilution resulted from this layer of sand. The wire should be on the floor for improved grade control and safety.

A simple burlap covered bulkhead was constructed at the stope end. The bulkhead was designed to be as water tight as possible in order to avoid losing cement and sand through it. Clear water was decanted over the bulkhead and through the burlap. The hanging of 2-in. (51-mm) plastic pipe with distribution valves and hoses every 20 ft (6.1 m) completed the stope preparation. Backfilling began with a 3-ft (0.9-m) layer of cemented sand, followed by 4 to 5 ft (1.2 to 1.5 m) of sand. The desired cement-sand mix was 15% cement by weight. Through experience and testing, 15% cement was determined to safely provide enough strength to the mix. When the cement drops below about 10% by weight, its addition does not add appreciable strength to the sand. The strength is low due to low solids. The 4- to 5-ft (1.2- to 1.5-m) layer of sand was added to control wall movement, and add weight to the cemented sand layer, so as to eliminate lifting from blasting.

When the amount of cement in the backfill drops below the desired 15% by weight, failures in the stope backfill began to occur. Backfill failures cause safety problems, production losses, and grade dilution. The control and consistency of cement addition at the sand plant is most critical to the overall quality of the stope backfill. When failures began happening regularly, problems at the sand plant were apparent. The former auger system, conveying the dry cement to the mixing chambers was inadequate for mixing cement with the sand slurry. The system was inconsistent due to variabilities in the cement moisture and temperature, causing extreme variations of the feed rate. A steady, adequate flow of cement could not be maintained. To eliminate this problem, the auger was removed and replaced with a cement slurry pump. The dry cement is fed directly from the storage tank to the pump. A small amount of water is added to slurry the cement. This slurry is then pumped to the mixing chambers for addition to the sand slurry (Fig. 19.3.14). Cement flow control to the pump is from a variable speed auger in the bottom of the cement storage tank. The cement pump then maintains a steady, correct flow of cement to the sand mix.

The undercut and fill system at the Garpenberg Mine in Sweden is typical of Boliden's cut and fill mines. Ninety percent of the annual production is by undercut and fill. The ore body is irregular in shape with some areas that can be mined with drift and fill methods. The method of filling is as follows:

First about a 12-in. (0.3-m) layer of ore using the bucket on an LHD is leveled off. On top of this is placed ordinary building polyvinyl chloride (PVC) sheeting, which is followed by a layer of welded wire netting with about a 6- by 6-in. (15- by 15-mm) grid. The first pour is made 6 ft (1.8 m) high, and the area to be poured is bulkheaded off with a timber wall to retain the cemented fill. This wall is later extended to the roof for the completion of the pour, and the bulkhead is equipped with drainage water outlets. The drainage outlets are sealed with planks, as filling progresses, and the fill in progress has relatively clear decant water. The first 6-ft (1.8-m) pour is made with a ratio of 1 part cement to 4 parts sand, and is poured in one steady stage so as to achieve a uniform quality without stratified layers if possible. Then the remaining 8 ft (2.4 m) is sanded with a weaker cement ratio of 1 : 10 in the second stage. The pours are made as close to the roof as possible. This second pour can be interrupted if necessary. The lower 6 ft (1.8 m) is very strong, averaging 700 psi (4.83 MPa). The upper 8 ft (2.4 m) is softer, but the entire mass is a very strong beam. As the next floor is mined under this cemented fill, 5.2-ft (1.6-m) fully grouted rock bolts are placed in the lower 4.6 ft (1.4 m) of the fill to make sure it is bolted together to form a beam. The broken ore cushion serves to prevent damage to the sandfill, and the PVC sheeting serves to separate ore from sand. The wire netting strengthens the lower surface of the cemented fill in tension.

Ninety percent of the annual production at Garpenberg mine is through downward cut and fill mining.

The sandfill mixing plant is located on the surface in the mill, and mixing is a continuous process during the filling cycle (Fig. 19.3.15). It is automatically controlled and regulated. The capacity of the system is about 125 tph (113 t/h), and the water content is held at 30% plus or minus 1%. The fill is delivered hydraulically, and at 70% solids is about as thick as it is possible to handle. There are two 5-in. (127-mm) diameter feed lines, one of which is a reserve line with outlets at all of the mining levels. Cemented sand is delivered by means of a pump to the different stopes, and down to the current depth of 1641 ft (500 m) by way of horizontal transport drifts. The general sand plant is similar to that at other Boliden mine properties, with a bin for sand that has been filtered to about 15% moisture, a bin for cement, and a supply of water. These three items are mixed together in a paddle-type mixer, and delivered to the mine either by pump or gravity. Magnetic flow-meters control the mixture. Sand is fed from a bin by belt-feeder, and cement by an auger to the paddle-type mixing chamber. Weightometers on the belt control the amount of sand and cement, and the quantity of water supplied is measured with a water meter. The system is quite simple and easily automated. The slurry is pumped with a German-made concrete pump. There is little head against the pump, but an even flow rate is desired.

19.3.3.5 Advantages

When extensive use of hydraulic fill began in 1929 and 1930, the waste fill then in use was gradually replaced. Hence the advantages listed here compare hydraulic fill to waste fill in general. Advantages of hydraulic fill include the following:

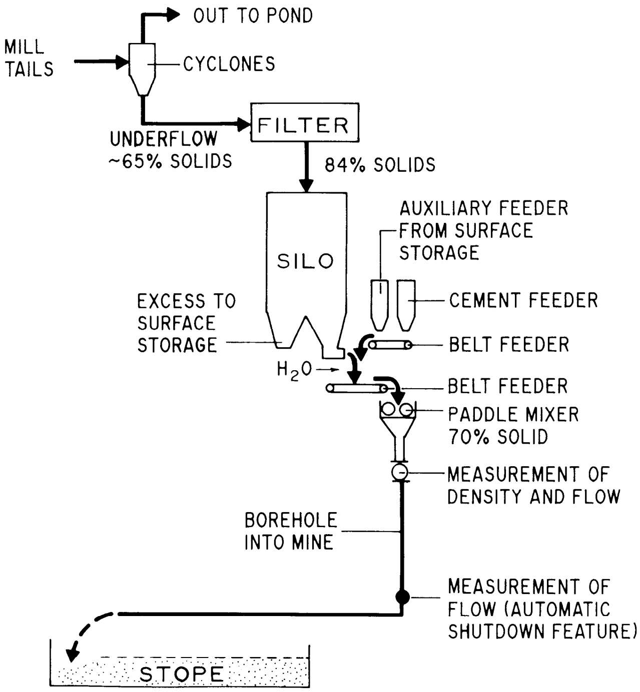

Fig. 19.3.15. Typical Boliden sand fill system (Brackebusch, 1984).

1. Better wall support is achieved as the hydraulic fill will flow tightly against irregularities in wall and back.
2. Overall pressure on the workings is reduced when the improved support reduces wall closure. This effect is enhanced when fill material is well graded so smaller particles fill voids. Addition of cement or other binder, of course, provides even greater strength.
3. Better ventilation control as much tighter seals are achieved.
4. When used with cement as a floor capping in overhand cut and fill stoping, a firm clean-up surface is provided. More fine ore is recovered, and little or no sand is recycled through the concentrator.
5. A good working surface is provided for mechanized mining equipment.
6. Surface storage and environmental problems are reduced.

19.3.3.6 Disadvantages

Hydraulic fill methods can have some disadvantages including the following:

1. If density is not maintained in the 65 to 70% range, excess water causes carry-over of slimes at the bulkheads, causing housekeeping problems, slime-filled sumps, wear on pumps, muddy haulage ways, equipment wear, and general disposal problems of slimes underground.
2. Excess water results in weak, stratified cemented sand fill and waste of cement when it washes over dams and bulkheads. If decant water containing cement gets into an ore-filled raise or pocket, plugging, hang-ups, or washouts will result.
3. There is a possibility of hydraulic failure if fill is not drained properly.
4. Slimes in sumps and shaft bottoms are difficult to remove.
5. Too dense a slurry can result in plugged pipelines.
6. High velocity in long, vertical pipelines can cause excessive wear. In deep mines, such as those in the Coeur d'Alene district of northern Idaho, ceramic-line-diameter reducers have been successfully used to overcome this problem.

19.3.4 HIGH-DENSITY BACKFILL

19.3.4.1 General

Development of high-density, consolidated backfills began in the 1970s and 1980s. Several considerations have led to the development of high-density backfills, consolidated or otherwise.

1. The excess water and slimes, associated with most hydraulic fills, cause wall deterioration in some mines with weak ground. Apart from this, there was a continual clean-up problem in ditches and sumps, as well as increased wear on pumps.
2. In deep mines with high wall pressures, a stronger, denser consolidated fill was advantageous to resist closure and reduce rock bursting.
3. In mines where high fill exposures were encountered in mining adjacent to the fill, a stronger consolidated fill was advantageous.
4. High-density, consolidated fill allows wider spans and a safer, stronger back in undercut-and-fill mining.
5. High-density fills allow the use of a greater proportion of slimes, thus reducing or eliminating the volume that must be impounded on the surface.
6. When cement is used, the low water content in high-density consolidated backfills results in maximum strength of fill, especially when the water content is just sufficient to hydrolize the cement. A water to cement ratio between 0.4 and 0.5 water to 1 part cement generally gives the greatest strength for the least cement.
7. Turbulent, high-speed transportation in pipelines may degrade a soft fill material and result in excess slimes in the fill. High-density pumped fills move at a much slower rate in the pipelines, reducing this problem and also reducing pipeline wear.

19.3.4.2 Sources of Material

It is entirely possible to "manufacture" a well-graded fill from numerous sources such as those described earlier for hydraulic or pneumatic fills. However, ordinary mill tailings are the most commonly used material today. This is primarily because the tailings are often readily available, fit the need for a graded fill, and also need to be disposed of underground to reduce environmental problems on the surface. Coarse rock, gravel, or development waste from surface sources, plus cement or other binders, may be added to prepare an even stronger fill than a straight tailings and slimes fill. The size range of coarser material will depend on whether the fill must be pumped with material of less than –1 in. (25 mm) or handled by gravity. In the latter case, particle size should be no larger than 1/3 the diameter of the pipeline used.

19.3.4.3 Preparation of Fill

The amount of fill preparation depends on the degree of support desired. If there is little or no wall pressure, and the requirement is simply to fill up the space, then total tailings with little attention paid to gradation might be used. If, in an overhand cut and fill system, a strong clean-up floor is desired, a graded fill with sufficient cement to provide a 600- to 700-psi (4.1 to 4.8 MPa) concrete may be necessary. In the case of the Lucky Friday mine, in the Coeur d'Alene mining district of northern Idaho, the wall pressure has been measured at 10,000 psi (69.0 MPa). The fill must be well graded, and 6% Portland cement is used.

Even solid concrete with a compressive strength of 3000 to 6000 psi (20.7 to 41.4 MPa) would not resist the wall closure until all voids are closed up. The mining method is being changed from overhand cut and fill to underhand longwall cut and fill. A system of 6- by 6-in. (152- by 152-mm) welded wire mesh and rock bolts, within the fill itself, is used to provide a strong, safe back as minor cracking of the 300- to 500-psi (2.1- to 3.5-MPa) cemented sandfill occurs.

It is apparent that sandfill must be designed to fit the requirements of the mine involved. Applications are presented in 19.3.4.5.

19.3.4.4 Delivery Methods

Because of the high density of the prepared fill, a positive displacement concrete or mud pump must be used to transport this fill material by pipeline. There are several existing systems at shaft-accessed mines, where the preparation plant and pump are located on the surface, with the pump and/or gravity supplying the pressure to place the fill in the stope. A secondary pump may be necessary underground if horizontal distances are too great.

The preparation plant can also be located underground with materials delivered separately to the site. Cement can be delivered pneumatically by pipeline or in bulk. Mill tailings with or without gravel can be delivered as a dilute slurry and dewatered prior to mixing. Although no instance is known, pneumatic delivery of tailings, sand, gravel, and cement should be possible.

Cement can be added either at a surface preparation plant, an underground preparation plant, or near the end of the delivery line in the stope. Adding the cement near the end of the line as a slurry prevents problems with line plugging and allows the line to be left full of high-density fill (82 to 87% solids) for hours or even days if desired. Only a slight increase in pumping pressure is required to start the material moving again after a shutdown. The high proportion of –325 mesh (0.044 mm) material in the fill makes the mixture thixotropic and provides a lubricating layer on the walls of the pipeline, reducing friction. Segregation of such material does not occur in the line or after placing in the stope.

19.3.4.5 Applications

Some of the earliest work on thickened backfill was begun in 1972 at Cambridge, Ontario, Canada. The following comments are adapted from a paper by Wayment (1978).

At this time, studies began toward the development of continuous or nonexplosive underground hard-rock mining systems. This led to the conclusion that a complementary technology had to be developed involving a support system that would fit the proposed mining system. Consequently, a new type of centrifuge or dewatering machine was developed that used either classified or unclassified mill tailings for feed at an optimum density of 60% solids by weight (Fig. 19.3.16). The centrifuge was found to operate most efficiently on unclassified tailings. Underflows from 76 to 85% solids, by weight, were achieved using various tailings. The centrifuge overflows contained from 5 to 15% of feed material that had to be either settled and stored underground or pumped out with the mine water. A second stage of centrifuging reduced the size of the overflow particles to –20 μm.

In the test process, 3% or more cement was added to the centrifuge underflow material. This lowered the water content by 1 to 2%, and consequently, the material exhibited considerable cohesion and behaved very much like a stiff mortar.

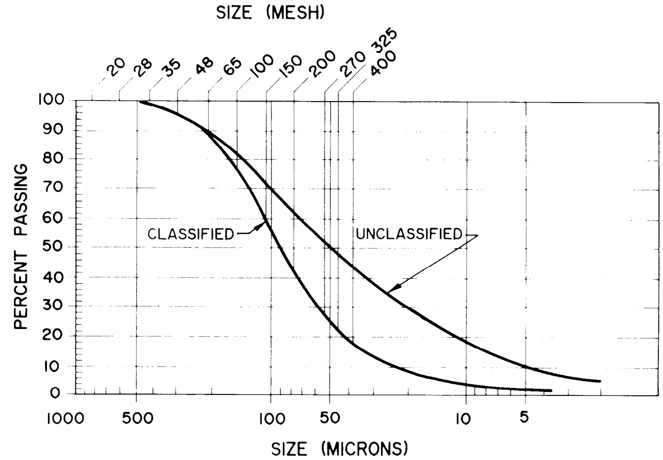

Fig. 19.3.16. Particle-size distribution, baseline tailings (nickel-copper) (Wayment, 1978).

PSI (M Pa)

SAND/CEMENT RATIO / MATERIAL	5:1	10:1	20:1	30:1	MIXING PULP DENSITY % SOLIDS BY WEIGHT
MINE A	931 (6.42)	316 (2.18)	143 (.986)	109 (.752)	74
MINE B	1679 (11.6)	436 (3.01)	178 (1.23)	111 (.765)	77
MINE C	3150 (21.7)	700 (4.83)	349 (1.30)	188 (1.30)	82

CYLINDRICAL SPECIMENS:
4 INCHES LONG, 2 INCHES DIAMETER (100 × 50 mm)
PREPARED ON VIBRATING TABLE
CURED IN PLASTIC BAGS

Fig. 19.3.17. Unconfined compressive strengths, 28-day (Wayment, 1978).

In the fall 1976, a semi-portable surface test plant was built at Anglo American's Western Holding, Ltd. mine in South Africa. Satisfactory underflow material was produced from these mill tailings. Two centrifuge units were then placed underground for initial trials. Cement was added to the underflows and used to fill an operating stope.

A wide range of unconfined compressive strengths for various cement mixtures, with centrifuge underflows from three different tailings, is shown in Fig. 19.3.17. Considerably higher strengths were achieved than are normally obtained, when equivalent amounts of cement were added to unthickened tailings with excess water. As in concrete technology, strength increases as the water-cement ratio decreases. In concrete work, a weight ratio of water to cement of 0.4 to 0.5 : 1 produces a strong concrete. With materials as fine as tailings with vastly increased surface area, the increase in strength is even more pronounced. A difference can even be detected between classified and unclassified tailings, with the latter producing the higher strength fill for a given amount of cement (Wayment, 1978).

The following comments are adapted from a paper by Patchet and Currie (1982), describing the Anglo American installation at Western Holdings, Ltd.

The system uses slurry transport, dewatering centrifuges, and positive-displacement placing pumps. The fill material ranges from 76 to 79% solids by mass and is placed tightly

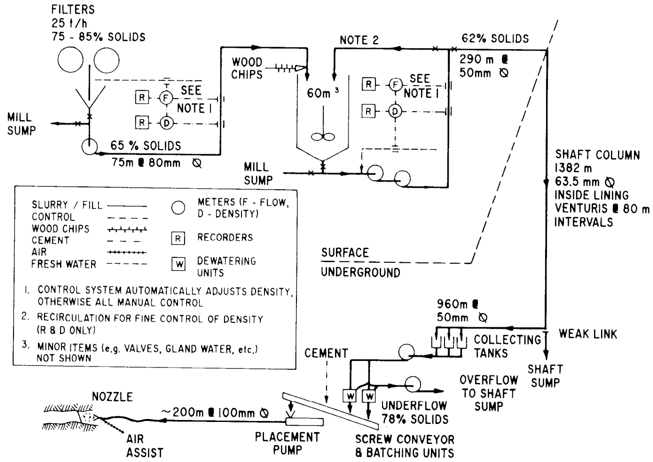

Fig. 19.3.18. Schematic flow chart of the prototype system (Patchett and Currie, 1982). Conversion factors: 1 ft = 0.3048 m, 1 gpm = 0.0631 L/s.

between hanging walls and footwalls, without shuttering or confinement, by use of air-assisted nozzles.

The system is cost-competitive with conventional support systems on a direct replacement basis. All the operation, basic repair, and maintenance are performed by typical mine labor. The system has an acceptable level of mechanical availability. The use of reduction plant tailings (RPT) and a slurry transport system using physical dewatering was chosen for the following reasons:

1. RPT are readily available on all mines.
2. RPT are produced as a matter of course, and do not incur any additional preparation costs.
3. RPT produce an excellent fill material provided the water content and density are controlled.
4. The transportation of RPT is proven, simple, low in cost, and entirely separate from all other transport and handling systems. It is readily suited to the placing of relatively small quantities of fill to a number of sites simultaneously.
5. The storage of slurry is relatively simple and cheap.
6. Dewatering centrifuges are fairly simple to operate, and provide a well-graded fill material of low water content. Positive dewatering eliminates decant or drain-water problems into stope faces, gullies, and boxholes.
7. Cement is added at the latest possible stage, which greatly reduces the possibility of setting in pipelines.
8. Pneumatic placement by means of hose and nozzle eliminates the need for shuttering on the face, ensures a uniform tight fill, allows placement at the earliest opportunity, and does not interfere with other operations on the face.

The trials and operation of the system lasted about 13 months, and during that period, some 1500 tons (1360 t) of fill was placed in old stopes. Sufficient data and operating experience were available to permit design and construction of pre-production equipment. A schematic flowchart of the system is shown in Fig. 19.3.18.

An examination of the fill some 12 months after its placement indicated that it does not slump, shrink, or crack, and a subjective assessment of conditions of old stopes in the area was that the fill area had closed slightly, but not nearly to the extent of the surrounding pack supported areas. Conclusions drawn from the prototype trials were

1. Slurry of controlled density can be prepared in a simple and easy manner.
2. The transportation of slurry is simple and effective.
3. Wear in slurry lines is negligible, even on sharp right-angle bends.
4. The ceramic venturis in the shaft column worked as designed and did not show any indications of wear.
5. The dewatering centrifuges worked acceptably well, but wear was a problem. A complete redesign of the centrifuge rig was indicated, both to simplify its construction and to improve its performance characteristics.
6. The placement pumps worked, but some fairly substantial changes were required to ease their repair and maintenance and improve their reliability.
7. Fill material could be placed without shuttering or other containment.

Preproduction trials followed in which four centrifuge and pump sets were installed on four stope panels with one common feed and waste system.

The system began operating in March 1980 with the installation of the first unit, and by June 1981 all four units were installed and 22,000 tons (19,960 t) of fill had been placed.

The existing shaft column was used, but the ceramic venturis have been removed. The column is rubber-lined, schedule 80, flanged pipe with a final internal diameter of 2.5 in. (63 mm.). It is run in a fully flooded condition, and the static head that develops at the bottom is used to drive the slurry approximately 1970 ft (600 m) to the storage facility.

The placement pump is an extensively modified positive-displacement concrete pump. The fill material is deposited in the hopper, and a small amount of cement is added by the use of a small belt conveyor. The hopper-agitator produced acceptable mixing. The material is then pumped to the desired placement location in the stope panel. Each site can serve several panels, and semi-permanent 4-in. (100-mm) lines are run to each. Relatively rapid changes can be made simply by changes between lines, at the dewatering site. The initial length of each line is high-pressure, lightweight, concrete placement pipe, but nearer the ends, as pressure permits, standard victaulic pipe is used. At the end of each line, a length of 2-in. (50-mm) hose is used with an air injection nozzle. Compressed air is added at the nozzle to break up and to impart momentum to the fill material, thus ensuring a dense backfill that fills all the irregularities, particularly in the hanging wall.

The slurry side of the system worked extremely well. The pipelines and shaft column presented no real problems. Initially, some difficulties were encountered with the glands of centrifugal pumps where little or no gland water can be used for reasons of density control. However, the selection of suitable packing materials, along with hard-coated shaft sleeves, has alleviated the problem. Agitation of the flat, low-slurry storage dams with compressed air has proved unsuccessful, and it appears that mechanical agitation is preferable. Air agitation does work, and would probably be successful in a vertical storage facility such as a bored raise.

The dewatering units have given a variety of problems but represent a significant advance over the prototype. Repair and maintenance have been eased, and the operation simplified. A considerable number of further design changes to both the centrifuge and the rig have been recognized. Most of the changes are in detail rather than in concept, but they should yield an improvement in mechanical performance equal to that achieved over the prototype when they are implemented on the next generation of equipment.

The placement pumps were completely redesigned after the prototype trials. As a result, repair and maintenance are much easier and faster. In both the dewatering rigs, the fill system is competitive with current support techniques on a direct replace-

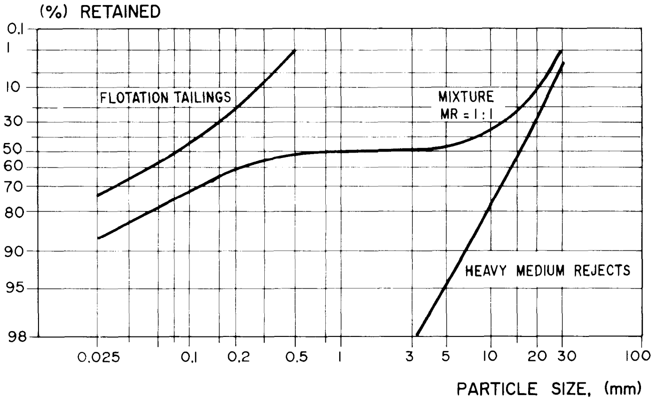

Fig. 19.3.19. Size distribution of pumped fill mixture (Lerche and Renetzeder, 1984). Conversion factor: 1 in. = 25.4 mm.

ment basis. The cost benefits would be greater at a lower stoping width, where a lower percentage filling by area is possible.

Trials of the production equipment result in these conclusions:

1. The use of slimes to produce a fill material of acceptable density and moisture content is possible.
2. The preproduction system described is capable of operating at acceptable placement rates in a production environment.
3. All the mechanical aspects of the preproduction system are acceptable. A variety of design improvements have been recognized, which should further increase the reliability by a significant amount.
4. The system can be operated and basically maintained using unskilled labor. The amount of supervision required is reasonable and acceptable.
5. The management of a fill system requires careful study if acceptable levels of capital expenditure and utilization are to be achieved.
6. The use of fill is cost-competitive with conventional support on a direct replacement basis (Patchet and Currie, 1982).

Preussag of West Germany published a paper in 1984 (Lerche and Renetzeder, 1984) describing the pumping of a high-density, concrete-like mixture of fine tailings and coarse rejects at the Grund mine. The system was developed between 1978 and 1983 to provide a satisfactory fill from available materials for cut and fill mining. Wall rocks are incompetent at the Grund mine, and sublevel caving methods in use during the seventies resulted in lower ore grades due to dilution.

Hydraulic filling was not considered due to degradation of shale particles from turbulent flow in the delivery lines. Excess slimes caused insufficient drainage, and desliming on the surface presented a storage problem.

Development then began on a high-density, low-turbulence, piped system similar to those used for conveying of concrete in the construction industry. The following decisive advantages of the pumped fill method were seen:

1. The fill materials available were fine and coarse mill tailings 0 to 0.012 in. (0 to 0.3 mm), and heavy media rejects from 0.12 to 1.18 in. (3 to 30 mm) in size (Fig. 19.3.19). The size range is almost complete so the packed fill material attains a density of 1.76 tons/yd^3 (2.1 t/m^3), resulting in excellent supporting properties.
2. The low moisture content of 12 to 15% means there is no excess water in the fill to weaken the natural rock bond, and there is no seepage of mud into the development workings and stopes.
3. Residual water is bound by cement. Owing to the low water content, addition of 3% cement (near the end of the pipeline) is sufficient to obtain a uniform compressive strength of 290 psi (2.0 mPa).
4. Because of these fill properties, mining methods can be better adapted to the conditions of the deposit (parallel slicing, underhand stoping, etc.), and the need for temporary ground support such as timbering is reduced or eliminated.
5. Stabilized rock conditions, owing to the high quality of the fill, leads to a reduction of ore dilution.
6. In conveying the fill material, a highly consistent mixture is formed that can be pumped uninterruptedly from the mill to the stope over a maximum of 6500 ft (2000 m) of pipeline, depending on pump pressure (294 to 1911 psi, or 2.0 to 13.2 MPa) and/or hydraulic profile, at rates of 26 to 40 yd^3/hr (20 to 31 m^3/h).
7. A flow velocity of up to 2.28 fps (0.7 m/s) is used, that results in minimal pipe wear.
8. The proportion of ultra-fine grains (< 0.001 in. or 0.025 mm), amounting to 10 to 15%, has a major influence on the pumpability of the mix. Under pressure, a lubricating film is formed at the wall of the pipe that reduces friction and saves energy.
9. The ultra-fine particles show thixotropic properties that provide a high degree of protection against settling out, so that the mixture may be left standing in the pipeline over several days and still remain pumpable.
10. The disposal of mill tailings in the workings, including slimes, clearly alleviates the problem of surface disposal of waste.
11. In favorable circumstances, the requirement for tailings disposal on the surface may be fully eliminated with concomitant cost and environmental benefits.

The pumped-fill process characteristically involves a high-density suspension of heterogeneous mixed solids that are conveyed in a pipeline. At the Grund mine, the fill materials available are substantially dewatered and mixed in the ratio in which they arise (approximately 1 : 1) to form a concrete-like mixture. To avoid blockages, the volume of fine material must always exceed the theoretical pore-volume of the coarse fraction. In addition, the consistency must be regulated so as to insure plug flow. A certain degree of segregation of the slime fraction occurs in the pipeline under pressure whereby the slimes tend to migrate to the low-flow region immediately adjacent to the side walls. This is clearly a welcome rheological effect.

Preparation of the fill on surface is shown in Fig. 19.3.20 and involves the dewatering of the heavy-media rejects to 2 to 3% moisture and filtration of the flotation tailings to 18 to 20% moisture.

The composition of the mix is controlled by belt-weighers (10 and 13), defined in 5-second cycles by the flotation tailings flow. The required quantity of coarse material is regulated by a variable feeder.

Final mixing is currently done by a twin-screw mixer (16). This mixture contains, basically, 12% moisture, but minor amounts of water may be subsequently added to achieve the best consistency for the particular requirements, dictated largely by conveying distances.

The surface pumping station (Fig. 19.3.21), houses a twin-piston concrete pump (3) with a power output of 215 hp (160 kW) and designed for delivery pressure of 1838 psi (12.7 MPa). Ahead of the pump is a mixer (1) and provision for measuring the pulp consistency (2).

The flexibility of a hydraulic transport system allows the fill to be brought directly to the working place, even in the confined

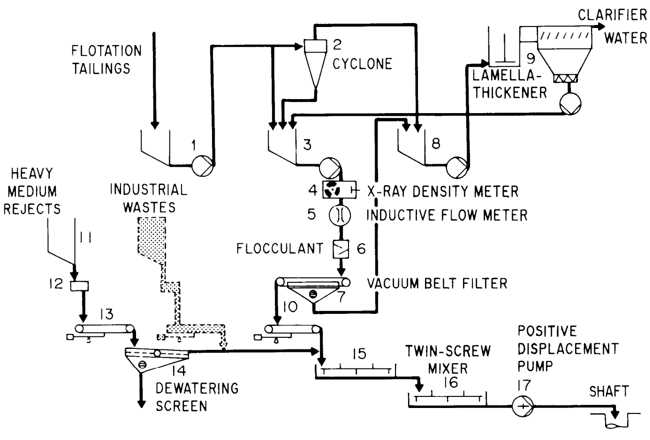

Fig. 19.3.20. Preparation of pumped fill (Lerche and Renetzeder, 1984).

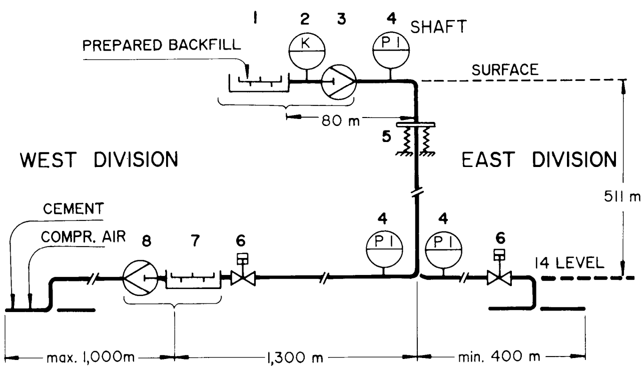

Fig. 19.3.21. Pumped-fill pipeline layout, surface and underground (Lerche and Renetzeder, 1984). Conversion factor: 1 ft = 0.3048 m.

conditions occurring in the cut and fill stoping of narrow veins. The cement to bind the fill and the contained water is injected immediately ahead of the pipe discharge as a slurry. The cement requirements are dictated by the moisture content and mining conditions. In general, 3 to 4% by weight is sufficient to yield a fill strength of 218 to 290 psi (1.5 to 2.0 MPa). The higher value is needed only when underhand stoping is in progress.

It is planned to replace the wet-dosing of cement by a pneumatic system. The dry method results in a stiffer mix that has a steeper angle of repose and facilitates the recovery of pipes and temporary supports.

To insure thorough mixing of the cement, a small injection of compressed air is made near the discharge point. This has the added function of increasing the throwing range and simplifying the task of packing the fill well up to the roof or backs.

A considerable amount of engineering, laboratory, and other test work was done by Preussag to determine the appropriate water content for mill tailings and coarse heavy media reject in order to arrive at the optimum of 12% moisture. This was determined to give maximum fill strength with a minimum cement use of 3%. It is apparent that similar test work must be done on fill material available at any mine in order to arrive at the lowest possible cement use for the fill strength desired.

Test work with pumped pipeline loops is also necessary to determine the pipeline sizes, pumping equipment, pumping rates, and fill density to design the optimum plant for a particular mine.

A high-density, consolidated fill system has also been developed at Lucky Friday mine near Mullan, ID. The following comments are adapted from papers by Brackebusch and Lautenschlauger (1988), and Noyes, Johnson, and Lautenschlauger (1988).

The Lucky Friday mine is developed by a 6200-ft (1890-m) deep primary shaft and a 5300-ft (1615-m) deep secondary shaft. The vein is near vertical, S-shaped, and has a strike length of approximately 2400 ft (732 m). Vein width averages 8 ft (2.4 m) with the maximum width less than 30 ft (9.1 m). Wall rocks are argillites, quartzites, and siltites of the Proterozoic age, Belt Supergroup. Within this group, the most productive horizon is the Revett-St. Regis stratigraphy. The Lucky Friday vein is a replacement vein containing the sulfide minerals galena, sphalerite, and tetrahedrite. The gangue minerals are quartz and siderite.

Historically, the mine has been developed on 200-ft (61-m) level intervals with drifting on the vein in early years, and by a lateral and crosscut system in recent years. Prior to 1958, waste fill was used and after that a hydraulic sandfill system with mill tailings. Mining was by overhand cut and fill stoping with horizontal slices of about 8 ft (2.4 m.). The Lucky Friday mine is currently in transition from the above system to an undercut and fill system using longwall geometry. During the period 1983 to 1986, a high-density, consolidated-fill system was developed to meet the requirements of this underhand method.

The Lucky Friday vein is enclosed in the brittle quartzites of the Revett formation. The vein strikes nearly parallel to the strata and dips steeply, ranging from 70° southerly to vertical.

The Coeur d'Alene District is located in a region of high horizontal stress, the maximum horizontal stress being about 1.5 times the vertical. This stress is nearly perpendicular to the plane of the vein at Lucky Friday. The high, horizontal stress field and the brittle quartzite rock together make the Lucky Friday a burst-prone mine. Bursting became common as the mine approached 3000 ft (914 m) in depth, and burst incidence has generally increased to the current stoping depth of about 5000 ft (1524 m). Failures are associated with stress concentrations caused by overhand stoping. The remnant of stope pillar between the stopes back and the mined-out area above has extremely high stress concentrations, due to magnification of the already high concentration of horizontal stress. Review of South African mining practice on the Witwatersrand indicated that the energy release rate associated with rock bursting could be reduced by the use of an undercut and fill system, using high-density, consolidated fill with a longwall configuration of stopes. It was apparent that the backfill system was the most important part of the undercut and fill mining system. Considering the weak wall rocks, rockbursting problems, the need for an underhand, longwall-type mining method with improved safety, and the many problems associated with dilute hydraulic sandfill in a deep mine, the selection of high-density consolidated fill for underhand mining was a logical conclusion.

In 1983, a formal research program was begun to test the new mining system called LFUL for Lucky Friday Underhand Longwall. A pilot stope was developed on the 5100 level using a footwall ramp and orepass with diesel LHD units for haulage. After testing and research on backfill system design, a new backfill plant was to be built.

A three-way cooperative agreement was made between Hecla Mining Co., the US Bureau of Mines Spokane Research Center and the University of Idaho College of Mines and Earth Resources. The latter two agencies performed much of the test work and assisted in many phases of the underground test stope.

Principal tests performed to aid in designing the backfill plant included the following:

1. Dewatering tests in a pilot scale dewatering plant using a thickener and disk filter. About 200 tons (188 t) of tailings containing 14% moisture were made. The size distribution was approximately:

Fraction	% Retained
50 mesh	1
70	3
100	8
140	13
200	18
325	22
−325	35

2. A surface pipeline test loop was set up. The line was 600 ft (183 m) long, using 4-in. (102-mm) standard pipe. A concrete pump such as is used in the construction industry pumped a tailings-cement mixture around the loop at rates from 30 to 60 yd^3 (23 to 46 m^3)/hour with slumps varying from 9 to 10 in. (22.9 to 25.4 mm) to 3 to 4 in. (7.6 to 10.1 mm). Pump pressures varied from 200 to 600 psi (1.4 to 4.1 MPa).

A model stope was set up on the surface and pumped full of cemented backfill. It was apparent that a hard, consolidated backfill could be produced from a tailings-cement with low-slump mixtures.

3. Flow of the high-slime tailings in bins and silos was tested and found to be a problem. Some compromise was, therefore, made with the amount of slimes included. Visual examination of different mixtures indicated those that would be a problem due to thixotropicity, the characteristic of converting from a solid when handled.

This work indicated that storage in a bunker rather than a silo was less risky. Also the decision was made to classify about 60% of the tailings slurry feed in order to facilitate the handling of dewatered tailings by conventional means.

4. Pneumatic conveying systems were reviewed for underground delivery but it was planned to use an existing 6-in. (152-mm) pipe column in the shaft in order to reduce costs. This line would not deliver the desired rate of 125 tph (113 t/h). In addition to the size limitation of the 6-in. (152-mm) shaft column, a pneumatic system was not tested because of the higher cost of conducting adequate tests, the concern due to higher energy costs, and because of abrasion due to high flow velocities.

Fig. 19.3.22 is a flowsheet of the consolidated fill plant. Two independent functions of the plant are:

1. Classifying and dewatering tailings.
2. Weighing, mixing, and pumping backfill.

Mill tailings are delivered to the dewatering section by centrifugal pump. About 40% by weight of the tailings go directly to the thickener where flocculent is added. The remaining 60% is cycloned with the overflow going to tailings disposal. Thickener feed and backfill have the following approximate size distribution since no further solids are removed:

Fraction	% Retained
70 mesh	15
100	17
140	15
200	13
325	13
−325	27

The thickener overflow is clear water, and the underflow is 60 to 65% solids, which is pumped to a vacuum drum filter. The filter produces fill material at 12% moisture that is then conveyed to a bin storage area. During the research stage, this bin was 20 ft (6 m) deep by 30 ft (9.1 m) wide by 40 ft (12.2 m) long with provision to increase the length for more storage as desired.

The dewatering section operates when the flotation concentrator is operating. The backfilling section of the plant is an intermittent process that is operated when stopes need to be backfilled.

During tests, the fill material from the bunker was fed to the mixing section by a front-end loader. Later, as the system became fully operational, a remotely operated multiple bucket reclaimer was installed, and the bunker capacity increased to 3000 tons (2123 t). Cement is conveyed from a silo to a separate weigh hopper, feeding into the mixer. A process control computer controls the conveyor to the weigh hopper, based on desired batch weights. The conveyors are automatically started and stopped during the preparation of batches.

The weighed portions of sand and cement are dropped into a high-intensity, paddle-type concrete mixer, where water is added under computer control. After a 2 yd^3 (1.5 m^3) batch is mixed, it drops into a pump hopper. The pump is a 125-tph (113-t/h) capacity concrete or sludge pump. The entire weighing-mixing-pumping process is controlled by the computer. Operating strategy is to produce a consistent, defined mix. The pump operating rate is controlled manually, and the computer provides mix to the pump as required by the pumping rate.

An 8-in. (203-mm) pipeline about 200 ft (61 m) in length connected the concrete pump to the 5100-ft (1554-m) long, 6-in. (152-mm) vertical line in the Silver Shaft. An 8-in. (203-mm) line about 2000 ft (610 m) in length, connected the shaft line on the 5100 level to a point near the pilot LUFL stope. The line used in the stope was 6 in. (152 mm) in diameter.

Mix design involved testing of a range of relatively liquid to stiff mixtures to obtain a given backfill strength of approximately 300 psi (2.1 MPa) unconfined compressive strength. A range of mixtures can be used; however, the stiffer the mix, the less cement content is required to yield the desired compressive strength. A more liquid mix requires a much higher cement content to yield the desired compressive strength. The liquid mix was termed the "Garpenberg" style after Boliden's Garpenberg mine in Sweden, which used a similar mix. Table 19.3.2 shows some of the specifications and properties for the different mixes.

A continuous range of mixtures between the stiff and Garpenberg styles can be produced using intermediate values of the specifications shown in the table, but all yield similar compressive strengths. As the mix becomes more liquid, there is a greater tendency for segregation of cement in the stope. Precisely controlled, high-density slurry avoids this problem.

After the completion of the new consolidated fill plant, initial pumping tests were conducted. The first testing involved calibration and observation of the equipment in operation. After the plant was checked out and problems corrected, loop pumping tests commenced. A pipe loop 463 ft (141m) in length consisting of 6- and 8-in. (152- and 203-mm) pipe was included with the new plant.

Besides the need for equipment operational data, two basic reasons for the loop tests were to determine friction factors and to determine the effects of time on fluid and pumping characteristics.

The results of the friction factor tests were particularly interesting:

1. Pressure loss due to pump hammer was greatest near the pump and decreased at greater distances from the pump. Pump hammer loss was greater with stiff mixtures, but effects were

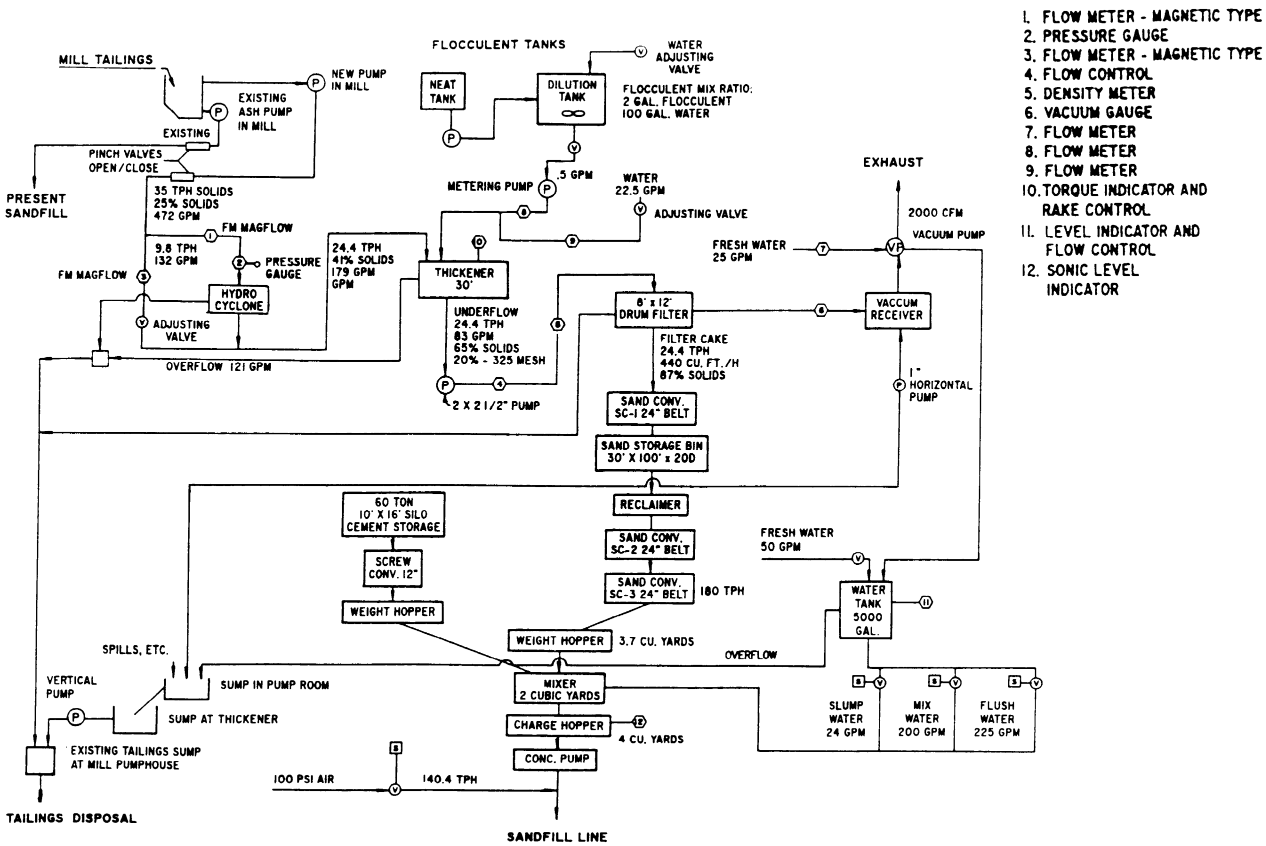

Fig. 19.3.22. Lucky Friday mine, underhand longwall consolidated-fill flowsheet (Brackebusch, 1988). Conversion factors: 1 ft = 0.3048 m, 1 ton = 0.9072 t, 1 yd^3 = 0.7646 m^3, 1 gpm = 0.0631 L/s, 1 psi = 6.895 kPa, 1 gal = 3.785 L.

Table 19.3.2. Mix Specifications and Properties

Specification/ Property	Very Stiff	Mod. Stiff	Garpenberg
Cement content	4%	5.7%	18%
Solids density	83%	82%	74%
Slump, in./mm	3/76	9/229	12/305
Minimum pipeline dia., in./mm	5/127	4/102	2/51
Cured strength, 28 day, psi/ MPa	300 to 440/ 2.07 to 3.03	300 to 500/ 2.07 to 3.45	300/2.07
Water-to-cement ratio	4.25	3.85	2.31

present for shorter distances away from the pump than with liquid mixtures.

2. With stiff mixtures, the pressure loss per foot in the 8-in. (203-mm) line was equal to or greater than the pressure drop in the 6-in. line (152-mm).

3. As pumping rate increased, using a stiff mixture, pressure drop increased in the 6-in. (152-mm) line, but was stable in the 8-in. (203-mm) line.

In the time duration test, a quantity of mix was pumped around the loop for two hours. The slump increased from 6 in. (152 mm) to 8.5 in. (216 mm), probably due to the time effects of wetting the surfaces of particles.

It was very evident from these initial loop tests that complex relationships exist when pumping thixotropic mixtures, but the loop testing showed that the desired mixes could be pumped at a reasonable pressure.

After the loop tests were completed, shaft pumping tests commenced. Backfill material was pumped down the 5100-ft (1554-m) vertical shaft pipeline and across the 5100 level to the LFUL stope. The total distance was about 7000 ft (2130 m), with about 2000 ft (610 m) being nearly horizontal.

A problem developed immediately, with mix falling at high velocities down the shaft pipeline and damaging the pipeline at the 90° curve on the 5100 level station. An impulse vessel was installed on 5100 level to absorb the energy of the falling mix, and at the same time, the shaft line was vented at the shaft collar. Problems with rapidly falling slugs of mix stopped with the venting of the shaft line, thus making the impulse vessel unnecessary. The presence of air in the line was cushioning the falling material, whereas before the pipeline was vented, the air was pushed out of the open pipeline in the stope, thus generating a vacuum in the shaft pipeline.

Pipeline failures in the 5100-level shaft station area were violent and caused considerable concern about the safety of the system. It was apparent that the pipeline had not been assembled and anchored properly. A test was conducted using a concrete pump on the 5100 station, which would be an inherently safer system because of the reduction in stored energy of a vertical pipeline containing a column of the backfill mix. The pipeline was opened up on the 5100 station and routed into the hopper of a portable construction-type concrete pump. Vertical flow with this setup was not a problem except that flow was intermittent and often overflowed the hopper of the concrete pump. However, the pump was unable to sustain the pressure needed to force the mix across the 2000-ft (610-m) horizontal distance of the 5100 level to the LFUL stope. The required pressure was about 500 psi (3.45 MPa) with 8-in. (203-mm) slump mix. This caused damage to the pump seals, which were then quickly eroded, disabling the pump. Line cleaning proved to be a problem for horizontal lines in the initial testing. Foam balls proved unsuccessful, as they tended to ride up over the material in the pipeline. Line cleaning worked best by pumping water immediately behind the mix. After the water emerges from the end of the pipeline, the line can be sparged with air and water to scour out any remaining mix in the pipeline. The sparging process consists of pumping water in slugs into the pipeline with the concrete pump while accelerating the water slugs through the pipeline with air pressure.

The results of these tests led to major modifications to the pipeline used for pumping stiff mixes which are described below. Meanwhile, the Garpenberg style backfill was being placed in the LFUL stope using the mine's existing 2- and 3-in. (51- and 76-mm) sand slurry pipeline, but supplied from the new consolidated backfill plant.

Because friction in pipes proved not to be as great as had been thought, smaller pipes could be used, thus simplifying pipe support and facilitating cleaning of the line. The pipeline in the 5100-station area was changed from 6-in. (152-mm) unlined steel to a 4-in. (102-mm), schedule 80, polyurethane-lined steel pipe for 150 ft (48 m), and the line was braced substantially. The impulse vessel was changed from 8-in. (203-mm) unlined steel to 5-in. (127-mm), schedule 80, unlined steel to the stope area. Four-inch (102-mm), schedule 40, unlined steel pipe was used for distribution to the stope. The stope distribution line has subsequently been changed to 4-in. (102-mm), schedule 40 steel. On the surface, the short pipeline between the backfill plant and the shaft was changed from 8-in. (203-mm) unlined steel to 4-in. (102-mm) unlined steel. The new pipeline configuration was designed to handle both stiff and Garpenberg styles of mix.

Certain plasticizers increase slump, at a given water content of a mix, and retard the initial set of the mix. Higher slump reduces pressure loss in pipelines. With the smaller pipelines to be used for both stiff and Garpenberg styles, it was decided to use an admixture with a resulting mix of 8 to 9 in. (204 to 229 mm) of slump for the stiff style.

The cement content of stiff style mix was set at 5% by weight of dry solids, with the goal of obtaining 300 psi (2.07 MPa) unconfined compressive strength in 28 days.

The Garpenberg style mix was not changed from that shown in Table 19.3.2.

After successful testing was completed for both stiff and Garpenberg styles of mix, the consolidated backfill plant was expanded to full-scale capabilities. This consisted primarily of enlarging the storage bunker and installing a reclaimer. The reclaimer drags buckets across the face of the sandpile and loads a conveyor that feeds the weigh hopper.

The backfill plant can produce up to 150 tons (136 t) per hour of operation. Typical results are shown in Table 19.3.3.

Flow in the shaft pipeline is smooth for Garpenberg style, but is intermittent for the stiff style. Thus flow rate into the stope with the stiff style also fluctuates. Apparently, slugs of fill build up in the vertical line in the presence of air "bubbles". Some of the air rises back up the vertical line to the vent, and some is entrapped and is emitted in the stope.

In placing of backfill in the stopes, the original objective was to fill the stope completely with consolidated backfill with no air space at all between the new lift of fill and the previous roof. While this may be a good objective, it has not been attained in practice. Actual practice leaves about an 18-in. (0.457-m) space that amounts to 15% of the mined-out void. With a stiffer backfill mix, the unfilled space can be reduced somewhat.

Table 19.3.3. Typical Operating Conditions

	Garpenberg Mix	Stiff Mix
Consistency, slump, in./mm	12/305	9/209
Pumping rate, strokes/min	12	10
Yd^3/hr (m^3/h)	110 (84.1)	92 (70.3)
Dry tons/hr (t/h)	118 (107)	122 (111)
Pump pressure, psi	150 to 200	175 to 225
Pump pressure, MPa	1.0 to 1.4	1.2 to 1.6
5100 station pressure, psi	300 to 400	700 to 800
5100 station pressure, MPa	2.1 to 2.8	4.8 to 5.5
Velocity, 4-in. line, fps	9.5	7.9
Velocity, 4-in. line, m/s	2.9	2.4
Velocity, 5-in. line, fps	6.1	5.1
Velocity, 5-in. line, m/s	1.9	1.6

Preparation for backfill consists of:

1. Spreading a layer of 12 to 18 in. (0.305 to 0.457 m) of broken ore evenly on the floor of the stope.

2. Laying 6- by 6-in. (152- by 152-mm) welded wire fabric along the long axis of the stope.

3. Driving 6- and 8-ft (1.8- and 2.4-m) Dywidag rock bolts into the broken pore to a minimum depth of 6 in. (152 mm). Bolts are installed on 3-ft (0.9-m) centers.

4. Attaching an 8- by 8-in. (203- by 203-mm) plate to the top of each "standing" bolt to serve as an anchor.

5. Installing a 4-in. (102-mm) steel fill delivery pipeline that extends to the face of the stope and connecting the stope line to the main fill delivery line. The same fill delivery pipeline can be reused by installing fittings that are buried in the layer of broken ore.

6. Building and sealing a wooden bulkhead at the entrance to the stope.

After backfilling the stope, a new underhand cut can be mined beneath the new layer of backfill starting as soon as 5 days after backfilling. As mining progresses, the vein walls close, and there is a tendency for the fill to fail in compression. Wire fencing is fastened to the protruding lengths of Dywidag bolts as they are exposed by blasting. The wire fencing and wire mesh serve to support loosened fragments of backfill.

Conclusions drawn from the design, testing, installation, and operation of this high-density, consolidated fill system are as follows:

The development of a high-capacity, consolidated backfill system at the Lucky Friday mine was successful and necessary as part of a new, underhand, longwall mining method for vein mining. While much remains to be learned about flow of thixotropic mixes (paste mixes), pipeline transportation of high-density backfill into a deep shaft and across long horizontal distances has proven completely workable.

The consolidated backfill, consisting only of fine mill tailings and cement, has proven to be an effective roof in stopes when lightly supported.

Precise control is required in the backfill preparation and pumping plant, with a process control computer being necessary. Pipelines must be engineered in detail, and good quality workmanship in connecting and fastening pipes is required. More highly skilled operators and better supervision are needed than with previous systems. These needs are all indicative of a higher level of technology.

Because of the many advantages of using a stiff backfill, more underground mines will probably develop similar systems and undoubtedly will improve the technology. The potential advantages include reduced cement usage, higher backfill strengths and densities, improved rockburst control (when used as a part of a mining system designed to reduce bursting), reduced mine dewatering requirements, fewer abrasive solids in the mine dewatering system, high solids flow rates, simpler sand bulkheads, reduced tailings impoundment requirements, and as a result of all of the above advantages, lower costs and higher productivity (Brackebusch, 1988).

19.3.4.6 Advantages

Obviously, there are many advantages to a high-density fill system compared to any of those discussed previously.

1. If desired, a high-density fill system can utilize all mill tailings including slimes. This eliminates or reduces surface impoundment requirements and associated environmental problems.
2. The system can also utilize waste rock or gravel up to 1 in. (25 mm) for a stronger fill. Underground waste removal can be reduced.
3. As described by Lerche and Renetzeder, (1984), a high-density pumped (or gravity) fill system can also be used to dispose of many other undesirable waste products.
4. A well-graded, high-density, consolidated fill produces a strong fill that allows underhand cut and fill, parallel slicing or drift and fill methods. Recovery of pillars against high fill faces, as in post and pillar mining, is also feasible. The method is useful in overhand cut and fill mining with cemented fill capping or no consolidation at all.
5. High-density fill (85% solids by weight) with a water-cement ratio that binds all or most of the available water, results in the strongest possible fill for the least cement.
6. Excess water (from a hydraulic fill) that increases wear on mine pumps and results in difficult clean-up problems is eliminated.
7. Using the slime fraction of the tails has marked benefit on pumping or gravity flow of the mixture. Slimes migrate to the pipe wall and act as a lubricant. However, sufficient water must be present to allow this flow between grains.
8. If cement is added near the end of the delivery line (approximately 30 ft or 10 m), the thickened slurry can be left in the pipeline for several days, with no start-up difficulty. This is due to the thixotropic nature of the sand-slime mixture.
9. Low pipeline velocity in a thickened fill system reduces pipe wear.
10. Use of timber or other support methods is reduced with a strong fill.
11. A stronger fill, used in conjunction with a longwall mining system, reduces rock bursting problems in deep mines that are prone to bursting.
12. Better wall support reduces dilution and improves grade control.

19.3.4.7 Disadvantages

Possible disadvantages of a high-density, consolidated fill system include the following:

1. The preparation plant is more expensive and more sophisticated than other methods. More precise control is required.
2. More highly skilled operators are required.
3. Depending on requirements of the individual mine, fill costs may be higher due to cement addition.

19.3.5 UNIQUE OR UNUSUAL FILL SYSTEMS

19.3.5.1 General

Several unusual fill systems are noted in the literature, which bear out the theory that in order for fill to be economic, the operator must utilize the material available at the least cost.

19.3.5.2 Frozen Fill (Archibald and Nantel, 1986)

Although no frozen tailings material has ever been used to backfill openings in Canadian mines (as of the date of this writing), this material may be required in the future to perform the support functions of underground mines located in zones of permafrost, notably in the high Arctic. At present, only one Canadian mine, the Polaris mine, situated on Little Cornwallis Island, is investigating the use of frozen backfill.

The potential for the use of this material may increase as future mines are discovered in permafrost areas.

19.3.5.3 Consolidating Fill with Non-cement Binders

Several mines have been using slag and rock fill for some years with an additive that cements the particles together in a solid mass (Twidale, 1982).

One mine uses granulated copper smelter slag with pyrrhotite concentrate as cementing agent. The slag is granulated by dumping it molten into a tank of water. The fine material is recovered with a clam-shell excavator. The granulated product is mixed with coarse slag (– 4 in. or 102 mm) and sent underground by a fill pass. At the first level, the slag is transported by a conveyor to the main fill pass where pyrrhotitic concentrate, in slurry form, is added. This pyrrhotite is a waste product from the flotation concentrator. The main fill pass delivers the fill, as required, at the working horizons as had been done previously with sand and gravel.

The reaction that consolidates the fill is by oxidation of the pyrrhotite to form a hydrous ferric oxide, which is the cementing agent. Heat and sulfur dioxide are given off. When a large mass is poured into an empty stope, the surface temperature reaches 100° F (38° C), the internal temperature being higher. When used in 6- to 8-ft (1.8- to 2.6-m) layers in a cut and fill stope, the heat effect of the reaction is dissipated in normal ventilation and is not felt. The sulfur dioxide is dissipated and is hardly noticeable.

Mineral admixtures are described by Plavsic (1984). Three specialty materials, (1) fly ash, (2) slag, and (3) silica fume, are classified as mineral admixtures. The first two materials exhibit many similarities in performance. The same characteristics that apply to slag also apply to fly ash. (One important exception concerns references to the pozzolanic behavior of fly ash; slag is not a pozzolan and does not exhibit those properties).

Fly ash is a finely divided residue resulting from the combustion of powdered coal in electrical power plants. Two types of fly ash are currently being produced, Class F and Class C.

When the coal is bituminous, the fly ash is a classic pozzolanit substance, that is, a siliceous or siliceous and aluminous material that in itself possesses little or no cementitious value, but when mixed with lime and water forms cementitious compounds. This fly ash is called Class F. Subituminous coal and lignite tend to produce a material containing appreciable amounts of lime as well as silica so that fly ash may be cementitious as well as pozzolanic. This type of fly ash is called Class C.

Blast furnace slag comes from the steel industry and may be used to replace cement as fly ash does. If the slag is granulated or quickly cooled in water, it may replace more cement than fly ash. If the slag is activated by sodium hydroxide, it may be used as the only cementitious material in concrete (or backfill).

Fly ash and slag should not be used in excessive amounts. When too much cement is replaced by these materials, the strength of the concrete or fill can be materially reduced. Because fly ash and slag can vary considerably in their effect on concrete mixtures, they should be tested before use in combination with the specific cement and aggregates being used.

Silica fume, also a pozzolan, is a byproduct of electric arc furnace production of metallic silicon and ferrosilicon alloys. It is an extremely fine material with an average particle size two orders of magnitude smaller than Portland cement. Chemically, it is nearly pure silica.

More water is required in the mixture when silica fume is used to replace cement. There are potential difficulties in handling silica fume because of its fineness, but these can be overcome by handling it in a slurry or in compacted form.

When used in quantities of 5 to 10% of Portland cement, it can easily produce concretes with compressive strength exceeding 15,000 psi (103 MPa).

It should, therefore, be possible to produce much stronger backfill, using less cement, by substituting such admixtures as those described above. However, little work appears to have been done in this field to date to determine the value of admixtures in backfilling materials.

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Section 20 Underground Mining: Caving Methods

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Chapter 20.0
INTRODUCTION

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Caving mining methods that are based on a planned caving of rock above and/or at times surrounding the material being mined can be classified in three broad categories: longwall mining, sublevel caving, and block caving. Each of the three methods involves a distinct technology as well as art of operation. Each requires a relatively large, regular, and predictable ore body. As a rule, the effect of mining results in some form of impact or change on the surface.

Operationally, underground caving methods are characterized by high productivity as well as a high percentage of extraction of the mineral being mined. A key element for the productivity of this form of mining is the ability to standardize the various elements of work taking place underground. Therefore, workmen may perform a specialized role rather than function as all-around miners. Further, the nature of the mining methods lend them to a high level of mechanization. Operations and manpower can be centralized into a relatively small area where supervision can be concentrated and therefore be more effective. Productivity as well as safety benefits thereby.

The caving method is unique in that the exploitation openings are deliberately destroyed by the process of mining. Rock mechanics principles focus on the ensurance and the propagation of the cave as well as the continuance of the mine openings designed to extract the broken ore. As a rule, production operations are designed to maintain a steady, continuous flow of ore from the extraction areas. This steady flow rather than an intermittent flow tends to promote good caving action as well as reduce hangups.

As a class of mining, underground caving accounts for about 15% of US mineral production. Because of the relative advantages of the caving class vs. other classes, for example, low mining cost, high productivity, and high recovery, its use is growing.

Chapter 20.1
LONGWALL MINING

S.S. PENG AND H.S. CHIANG

20.1.1 INTRODUCTION

Modern longwall coal mining was introduced to the United States in the early 1960s. The *longwall mining* method has the simplest system layout and provides continuous production and full potential for automation, which can further improve productivity and personnel health and safety. The panel layout is simple and conducive to good ventilation, and crews always work under protective supported roof. Therefore, it is much safer than the room and pillar mining method and complies easily with current US laws. Since the longwall system with full caving leaves a lesser amount of residual pillars than other mining methods, coal recovery is higher and surface subsidence is relatively uniform and complete.

In the United States, longwall mining is mainly employed in the Appalachian coalfield, although more longwalls are expected to be installed in the Western and Illinois coalfields in the near future. Depth of overburden ranges from 200 to 2700 ft (60 to 820 m) with the majority between 400 and 1200 ft (120 to 370 m). Seam thickness ranges from 43 to 156 in. (1.1 to 4.0 m), but mostly between 48 and 120 in. (1.3 to 3.0 m). All coal seams are nearly horizontal and extracted in a single slice, although one inclined seam and one thick seam using multislicing longwall mining are being operated in the West.

Longwall panels are blocked out by the panel entries that are excavated in-seam on both sides of the main entries (Fig. 20.1.1). Longwall mining in the United States is of the retreating type. The immediate entries on both sides of the panel are called the head entry and tail entry. The *head entry* is used for the passage of intake air and the transportation of coal, personnel, and supplies, while the *tail entry* is used for the passage of the return air.

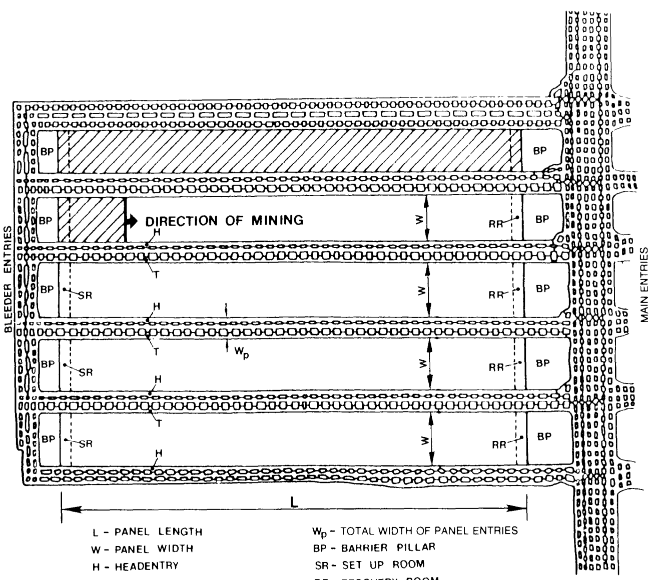

Fig. 20.1.1. Typical US longwall panel layout (Peng and Chiang, 1984. By permission from John Wiley & Sons, Inc., New York).

Fig. 20.1.2 is a cutaway view of a longwall face. Coal at the face cut by the shearer or plow is loaded onto the armored flexible conveyor (AFC) and transported to the head entry T-junction. Then coal is transferred from the AFC onto a stage loader, which in turn empties onto the entry belt conveyor. Powered supports are used to support the roof along the whole face. The AFC and powered supports are advanced hydraulically after each cutting cycle of the shearer, and the roof strata behind the supports are permitted to cave immediately after advance of the support. The extracted area between the rear edge of the supports and faceline is called the *face area* or working face, while that behind the roof supports is called *gob*. The method of roof control is classified as a roof caving method.

The panel entries are maintained by roof bolting as in room and pillar mining. But in the tail entry, one or two rows of cribs are erected to strengthen the entry support. At the head entry T-junction area, about 100 to 500 ft (30 to 150 m) outby the face, one or two rows of temporary supports are generally set to increase support density to cope with the moving front and side abutment pressures.

20.1.2 PANEL LAYOUT DESIGN

Panel layout design includes the determination of panel size, entry width, and the size and number of chain pillars (Fig. 20.1.1).

20.1.2.1 Panel Size

Panel width and panel length are usually determined by experience, based on the size and shape of the coal reserve, geologic conditions if known, location of surface structures, and the capacities of the transportation, ventilation, and power equipment that can be supplied. In the United States, the panel width *W* varies from 400 to 960 ft (120 to 293 m) center to center with an average of 666 ft (203 m). Most panels are wider than 600 ft (180 m), and a few are 900 ft (275 m) or larger. The panel length *L* varies from 3000 to 14,000 ft (610 to 4268 m) with the most common lengths from 4000 to 6000 ft (1220 to 1830 m).

If the panel width is substantially less than 400 ft (120 m) then the method is referred to as *shortwall mining* (Hartman, 1987). In application, it is intermediate between longwall and room and pillar mining (Chapter 18.1).

It is obvious that panel width is very important, because a wider longwall will increase the production of coal. A large number of factors influence the panel width, which can be divided into economic and technical.

ECONOMIC FACTORS. From an economic point of view, increasing panel width will reduce the number of panels in a mine reserve, resulting in (1) reduction of the development cost of panel entries, especially for a multiple entry layout; (2) an increase in the recovery rate of coal due to fewer chain pillars; (3)

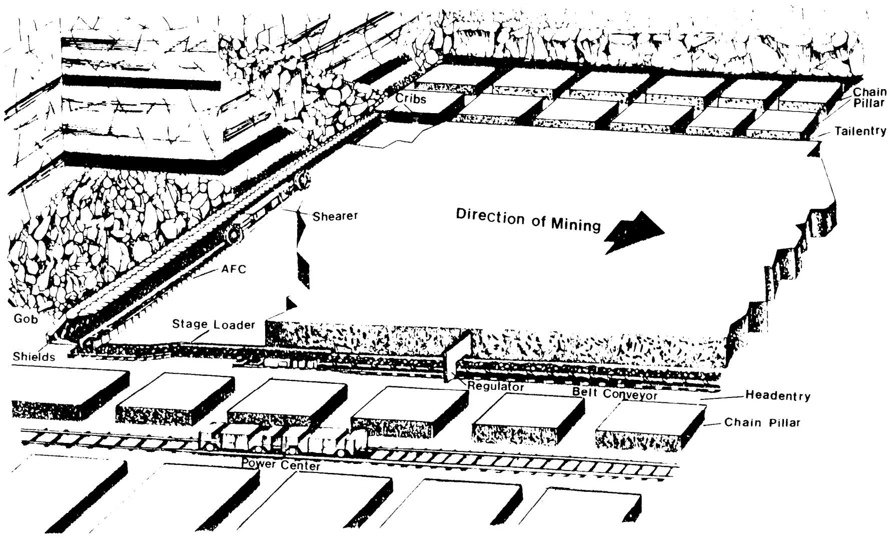

Fig. 20.1.2. Typical longwall face in cutaway view (Peng and Chiang, 1984. By permission from John Wiley & Sons, Inc., New York).

an increase in the production of coal, but, if the panel width exceeds a certain range (e.g., 1000 ft or 300 m), further increases in panel width have less influence on coal production; (4) reduction of the number of face moves and an increase in the minable coal per move, but with corresponding longer move times because more equipment has to be moved for longer distance; and (5) an increase in the total capital investment of the face equipment (i.e., more units of support, longer and stronger face conveyor, etc.).

TECHNICAL FACTORS. US longwall mining technology has proven that panel widths up to 1000 ft (300 m) are not only feasible but also achievable with high production. However, some important technical factors should be considered. (1) The panel width is primarily limited by the power required and structural strength of the face conveyor. (2) Roof control is now playing a less significant role in panel width. But, under unstable immediate roof and unidirectional cutting pattern conditions, increasing panel width increases the roof exposure time, resulting sometimes in local roof falls between the faceline and the tip of the canopy. It will reduce the support advance rate and, consequently, results in decreased overall shift production. (3) Operational and management considerations play a more important role in wider longwall faces. For example, in order to keep a normal shearer cutting, the faceline, powered supports, and the face conveyor should maintain three separate straight lines. If the panel is wider, more time is needed to adjust the faceline, supports, and conveyor, especially at both ends of the face.

20.1.2.2 Total Width of Panel Entries

The width of the panel entries W_p varies from 100 to 350 ft (30 to 110 m), depending on the number of panel entries. In the United States, three- and four-entry systems are commonly used in longwall panel layout, although a two-entry system is often used in the West. However, three entries is the minimum and thus the most common number of entries required by law for developing panel entries: one for the intake air escapeway, one for the return air escapeway, and one for the belt conveyor in neutral air. In a four-entry system, a neutral track entry is maintained, and two entries are served by intake air so that sufficient air quantity can be provided to the face from the main entries. In high-production retreating longwall mining, a fast entry development rate is required to support the continuous longwall production. Therefore, the number of panel entries is an important factor when the panel layout system is being designed.

20.1.2.3 Chain Pillar Size

In the three-entry system, two rows of equal-sized square chain pillars are commonly used. Under this condition, the following expression applies (Peng, 1986):

$$W_{pc} = \left\{ \frac{E_i}{E_c}, \frac{E_m}{E_c}, \frac{H_m}{H_c}, \frac{E_f}{E_c}, h, \sigma_c, d, W, \frac{H_i}{H_c}, W_o, \sigma_h \right\} \tag{20.1.1}$$

where W_{pc} is the width of square chain pillar; E_i, E_c, E_m, and E_f are the Young's modulus of the immediate roof, coal, main roof, and floor, respectively; H_i, H_c, and H_m are the thicknesses of the immediate roof, coal (mining height), and main roof, respectively; h is seam depth; σ_c is minimum uniaxial compressive strength required for a specifically sized coal pillar to remain stable under a certain roof condition; d is half-panel length; W and L are the panel width and panel length, respectively; W_o is width of the entry, and σ_h is in situ horizontal stress. Eq. 20.1.1 is for subcritical or semi-critical panels. If the panels are critical or supercritical in size, $L = (1.2 \sim 1.4)$ h.

Fig. 20.1.3 is a nomograph constructed by using a three-dimensional finite element technique in a series of parametric analyses to verify the quantitative relationship of Eq. 20.1.1.

Example 20.1.1. To demonstrate the use of this nomograph, follow the dotted line a-b-c-d-e-f-g-h to determine the pillar size W_{pc}, given the following conditions:

$\sigma_c = 1000$ psi (6.895 MPa); $E_m/E_c = 0$; $E_i/E_c = 0$; $L = 5000$ ft (1524 m); $h = 500$ ft (152 m); $E_f/E_c = 1$; $W = 400$ ft (122 m)

The value of W_{pc} read from Fig. 20.1.3 is 61 ft (18.6 m).

When designing a new panel without roof property information, it is recommended that the values of E_i/E_c, E_m/E_c, and

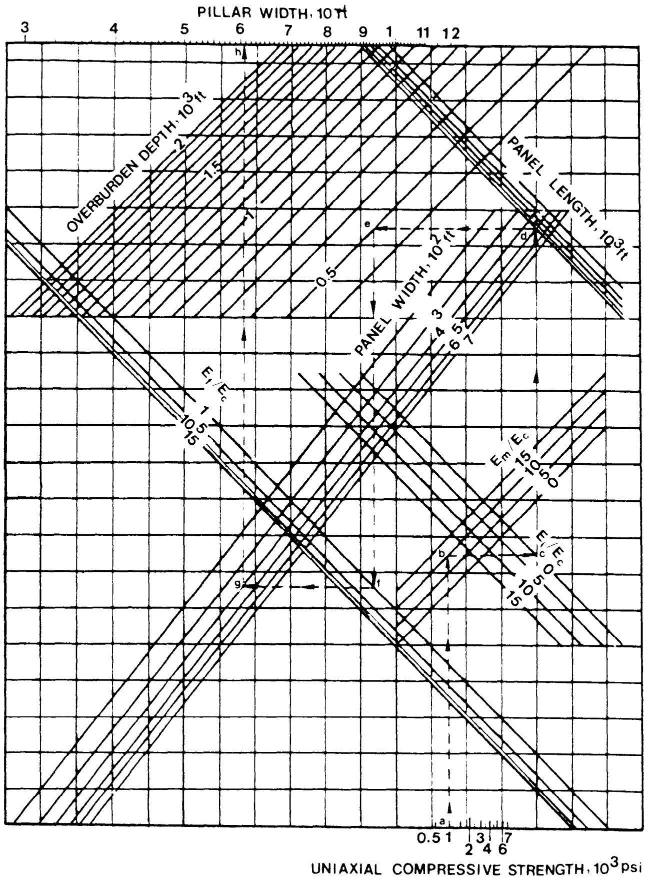

Fig. 20.1.3. Nomograph for determining chain pillar size (Peng, 1986: By permission from John Wiley & Sons, Inc., New York). Conversion factors: 1 ft = 0.3048 m, 1 psi = 6.895 kPa.

E_f/E_c should be assigned as small as possible. For a safe design, E_i/E_c and E_m/E_c are assumed to be zero. Since the parameter E_f/E_c in the formula is in logarithmic form, it cannot be zero. Therefore, it is assumed to be 1.

20.1.2.4 Barrier Pillars

In order to protect the bleeder entries and the setup room, where all of the face equipment is located initially, a barrier pillar between the bleeder entries and the setup room, ranging in width from 200 to 500 ft (60 to 150 m), is developed depending on the seam depth. Sometimes, if the bleeder system contains a sufficient number of entries, no barrier pillar is developed, and the setup room is developed as part of the bleeder system. Also a barrier pillar from 300 to 500 ft (90 to 150 m) (sometimes less than 200 ft or 60 m) in width is left between the recovery room and the main entries for protection of the mains.

20.1.3 SELECTION OF MINING EQUIPMENT

20.1.3.1 Selection of Cutting Machine

The *shearer loader* and the *plow* are the two major types of cutting machines used on longwall faces. Major factors to be considered in selecting the cutting machine are the mining height, seam structure, roof bonding strength of the coal, and the cutting/plowing resistance. However, in US longwall mining, the shearer loader is almost exclusively used (95%). Only five plow systems (5%) are employed, and those where the seam thickness is below 50 in. (1.27 m). Major parameters for selecting the shearer are (1) type, (2) dimension, (3) haulage speed, and (4) power capacity.

TYPES OF SHEARER. Nowadays, three types of shearer loader are used on US longwall faces: *double-ended ranging-drum* (DERS), *single-ended ranging-drum* (SERS), and *single-ended fixed-drum* (SEFS) *shearers.* The vast majority of these are DERS, which are used to extract coal seams from 58 to 156 in. (1.47 to 3.96 m) in thickness. Only two SERS units are employed simultaneously on a longwall face with a 60-in. (1.52-m) mining height. Two SEFS units are used simultaneously to extract a thin seam of 48 in. (1.23 m) thick, while another two units of an older SEFS type with a rope haulage system have been employed since 1968 in two thin seams 50 and 54 in. (1.27 and 1.37 m) thick, respectively. However, DERS is used predominantly when the seam thickness exceeds 60 in. (1.52 m) because it can cut the whole seam height in one trip and in either direction of travel, thereby ensuring high production with rapid face advance and shortening of roof exposure time. Recently, the in-web shearer has gained some favor due to interest in extracting thin coal seams. It moves on the floor in front of and along the face conveyor. In this case, the leading drum cuts the coal and makes sufficient space for the passage of the shearer's body.

There are two cutting patterns for the shearer: unidirectional and bidirectional. A unidirectional cutting pattern is the most commonly used in US longwall mining in order to reduce the shearer operator's and shield mover's dust exposure. Nowadays, due to adoption of automation and remote control technologies, the shearer operators and shield movers can always stay on the fresh-air side of two major dust sources, the shearer and moving shields. Therefore, in order to increase coal production and reduce the unsupported roof exposure time, a bidirectional cutting pattern is favored for increasing longwall production.

In *unidirectional cutting,* the common practice is that when the shearer travels from tail entry to head entry, the leading (or the headend) drum is raised and cuts coal while the trailing (or tailside) drum is free or cuts the floor coal depending on seam height. On the return trip from head entry to tail entry only cleaning of loose coal is performed. In *bidirectional* (or bi-di) *cutting,* a web width of coal is cut in both directions of travel. It must be noted that in bi-di cutting, each cutting trip involves two face-end operations to turn the shearer around; whereas in unidirectional cutting, only a tailend turn-around operation is needed, and in some cases no turn around is required depending on the method selected.

DIMENSIONS OF SHEARER. The shearer comes in different models with varying dimensions and power available, depending on the manufacturer. In selecting the shearer, the mining height H_c should first be considered for properly selecting the dimensions of the shearer, which includes the diameter of the cutting drum D, body height H_b, ranging arm length L_a, body depth B, and swing angle (above the floor) a of the ranging arm (Fig. 20.1.4). The relationship among those parameters can be expressed by:

$$H_c = H_b - \frac{B}{2} + L_a \sin \alpha + \frac{D}{2} \qquad (20.1.2)$$

Fig. 20.1.5 is a nomograph to verify the quantitative relationship of Eq. 20.1.2. However, for the double-ended ranging-drum shearer, the maximum mining height should not exceed twice the diameter of the cutting drum D.

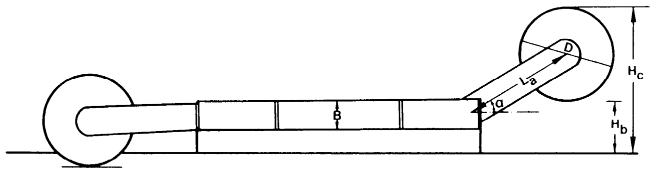

Fig. 20.1.4. Components for determining shearer dimensions and mining height.

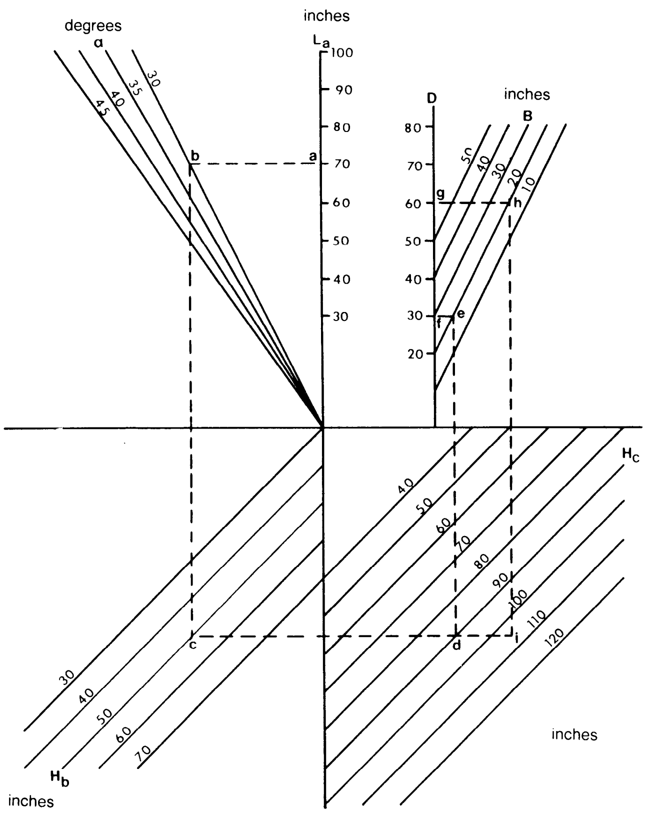

Fig. 20.1.5. Nomograph for determining shearer dimensions and mining height. Conversion factor: 1 in. = 25.4 mm.

Example 20.1.2. Use the nomograph shown in Fig. 20.1.5 to select the drum diameter *D*, given the mining height and major dimension of a shearer:

H_c = 90 in. (2.29 m); H_b = 50 in. (1270 mm); B = 20 in. (508 mm); L_a = 70 in. (1778 mm); α = 30°

Solution. Follow the dotted line a-b-c-d-e-f in Fig. 20.15. The minimum value of *D* required under the given conditions is 30 in. (762 mm). In practice, in order to reduce the loading resistance of the drum and to increase the loading efficiency, the drum diameter is selected larger than the minimum value. Usually, the drum diameter is 75 to 80% of the mining height; i.e., in this example, *D* = 60 in. Thus, following the dotted line g-h-i in Fig. 20.1.5, the cutting height of the shearer can attain 105 in.

Three web widths are currently in use: 30, 36, and 40 in. (762, 914, and 1016 mm) with the most common one being 32 in. (813 mm). Drum rotational speeds are 36, 44, 58, or 63 rpm, depending on the manufacturer, with the most common one being 44 rpm. Drums rotating less than 36 rpm have poor loading capacity. Conical bits account for 75% while radial bits cut 25%.

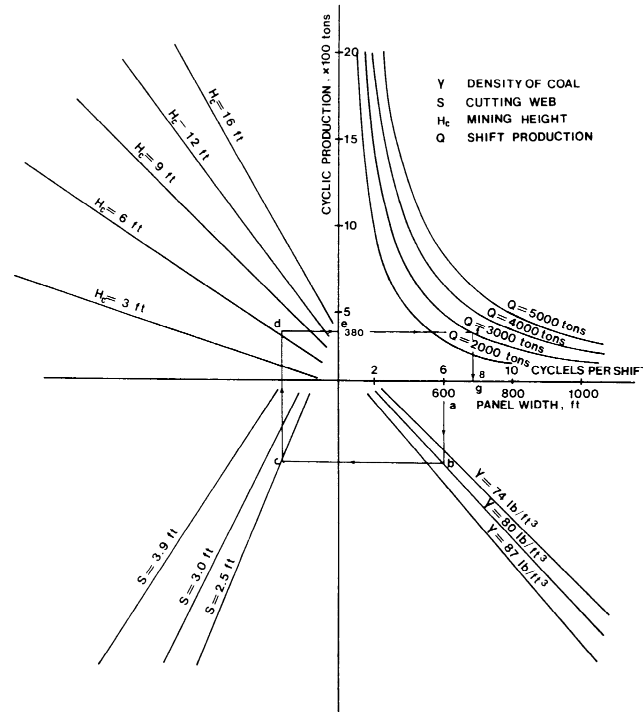

Fig. 20.1.6. Nomograph for determining longwall production. Conversion factors: 1 ft = 0.3048 m, 1 lb/ft^3 = 0.01602 t/m^3, 1 ton = 0.9072 t.

HAULAGE SPEED OF SHEARER. The haulage speed of the shearer ranges from 13 to 82 fpm (4 to 25 m/min). In order to increase coal production, higher haulage speeds (40 to 59 fpm, or 12 to 18 m/min) are now in common use.

The shearer hourly production Q_{she} in tons/hour, cycle production Q_c tons/cutting cycle, and its cutting cycle time T_{cut} in minutes can be determined, respectively, by:

$$Q_{she} = 60\ H_c\ s\ V_s\ \gamma\ C \tag{20.1.3}$$

$$Q_c = W\ H_c\ s\ \gamma\ C \tag{20.1.4}$$

$$T_{cut} = 60\ k\ Q_c/Q_{she} \tag{20.1.5}$$

where H_c is mining height in ft or m; *s* is cutting web in ft or m; V_s is haulage speed of the shearer in fpm or m/min, γ is weight per unit volume of coal (generally 74.8 to 87.3 lb/ft^3, or 1.20 to 1.40 t/m^3); *W* is panel width in ft or m; *C* is the loading coefficient of the shearer (generally 0.90 to 0.95); and *k* is the coefficient of time utilization, generally 1.1 to 1.4, depending on the amount of time required for shearer stopping/reversing at both ends of the face and some stoppages occuring along the panel. Fig. 20.1.6 shows a nomograph to determine longwall production.

Example 20.1.3. Use the nomograph shown in Fig. 20.1.6 to determine the cycle production and the required number of cutting cycles per shift. If the panel width *W* is 600 ft (180 m), the weight per unit volume of coal γ is 80 lb/ft^3 (1.28 t/m^3), the cutting web of the shearer *s* is 2.5 ft (0.76 m), and the mining height H_c is 6.5 ft (2.0 m), then follow the arrow to determine

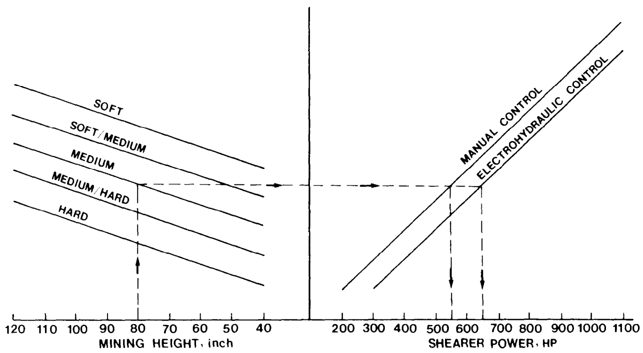

Fig. 20.1.7. Nomograph for selecting shearer power. Conversion factors: 1 in. = 25.4 mm, 1 hp = 0.7457 kW.

the cycle production, Q_c = 380 tons (340 t). Thus, if the planned shift production of coal is 3000 tons (2700 t), 7 to 8 cutting cycles should be completed.

POWER CAPACITY. Power consumption of the shearer reflects the total electrical load during shearer operation, which is affected by a series of operation-related factor, such as haulage speed, drum diameter, width and rotational speed, cutting depth and bit conditions, mining height, and seam hardness. Under normal conditions, mining height, seam hardness, and haulage speed are major factors in selecting the power capacity of the shearer. Based on a recent survey of the shearers employed in US longwall mining, a reference chart is constructed for selecting shearer power under different conditions (Fig. 20.1.7).

Example 20.1.4. Use the chart shown in Fig. 20.1.7 to determine the power required of the shearer under the following conditions: mining height is 80 in. (2 m), and seam hardness is medium. Follow the arrow to select the power required: N = 550 hp for manual control, or N = 650 hp for electrohydraulic control.

20.1.3.2 Selection of Armored Face Conveyor

TYPES OF CONVEYOR. There are four types of chain strands for longwall armored face conveyors: *single center* (SCS), *double center* (DCCS), *double outboard* (DOCS), and *triple-chain strand* (TCS). In US longwall mining, the double chain strand DCCS, with chain size of 1.18 and 1.34 in. (30 and 34 mm) and the single center strand with chain size of 1.50 and 1.65 in. (38 and 42 mm) are most often employed. In comparison with DOCS, the double center strand provides a larger loaded cross section, more flexibility, less chain wear, and easier maintenance. Lately, chain strength has been improved considerably, and the single center strand is gaining more acceptance. As production increases in longwall faces, a chain size of 102 in. (2.6 m) or less becomes insufficient.

Recently, in order to meet the demand of high-production longwalls, wider pans are being used. More than 40% of them are in the range of 37 to 39 in. (945 to 1000 mm).

CAPACITY OF AFC. The carrying capacity of the AFC, Q_{afc} in tph (t/h) can be determined by the relations:

$$Q_{afc} = q_c\, V_c = 60\, A_{max}\, \psi\, \gamma_c\, V_c \tag{20.1.6}$$

$$q_c = A_{max}\, \psi\, \gamma_c \tag{20.1.7}$$

where q_c is the weight of coal loaded per unit length of AFC; A_{max} is the largest loaded crosssection of the pans line, which depends on the construction and width of the pan, type of chain strand, angle of repose of the broken coal, and height of spill plate; ψ is a loading coefficient, with ψ = 1 when it is fully loaded but generally 0.65 to 0.90, depending on the seam inclination, hardness and size of the broken coal, and the structural features of the pans and spill plates; γ_c is weight per unit volume of the loaded coal, generally γ_c = 55 to 76 lb/ft^3 (0.88 to 1.22 t/m^3); and V_c is chain speed, usually 250 fpm (76.2 m/min) or less but recently some designs of 300 fpm (91.5 m/min) have become available to meet the demand of a wider face and higher production of coal.

Table 20.1.1. Coefficients of Moving Resistance for Chains

Chain Strand Type	f_c	f_n
SCS	0.40–0.60	0.25–0.40
DOCS	0.60–0.80	0.20–0.35

* f_n is positive when the chain is moving upslope while f_n is negative when the chain is moving downslope.
** when the floor and subsequently the AFC are undulating, f_c and f_n may be increased accordingly.

POWER REQUIREMENTS OF AFC. In determining the power requirements of the AFC, the moving resistances R_L and R_E for loaded (upper) and empty (under) sides of the AFC, respectively, and haulage force at drive sprockets F_S should be determined first by:

$$R_L = (q_c f_c + q_n f_n)\, L_{afc} \cos\alpha \pm (q_c + q_n)\, L_{afc} \sin\alpha \tag{20.1.8}$$

$$R_E = q_n\, L_{afc}\, (f_n \cos\alpha \pm \sin\alpha) \tag{20.1.9}$$

$$F_S = K_b\, K_S\, (R_L + R_E) \tag{20.1.10}$$

where L_{afc} is length of AFC in ft or m; q_c and q_n are weights of loaded coal and chain, respectively, per unit length of L_{afc} in lb/ft (kg/m); f_c and f_n are coefficients of moving resistance for loaded coal and chain unit, respectively (reference values of f_c and f_n are listed in Table 20.1.1); a is angle of face slope in degrees; K_b = 1.1 is a resistance factor considering the additional resistances due to curvature and bearings at the sprockets; and K_S = 1.1 is another resistance factor for the snaked portion of the AFC.

Finally, the maximum power required for the AFC can be determined by:

$$N_{max} = \frac{1.15\, F_S\, V_C}{C_{ele}\, \Phi} \tag{20.1.11}$$

where F_S is haulage force in lb or kg, V_c is chain speed in fps (m/s), Φ = 0.80 to 0.83 is transmission efficiency of the gearbox and fluid coupling, C_{ele} is a conversion coefficient = 550 hp units and = 102 kW units, and 1.15 is a safety factor.

The minimum power required for the AFC when the shearer just starts is

$$N_{min} = \frac{1.15\, q_n\, f_n\, L_{afc} \cos\alpha\, V_c}{C_{ele}\, \Phi} \tag{20.1.12}$$

Thus the AFC power required for longwalls with shearer can be determined by:

$$N = 0.58\, K_{afc} \sqrt{N^2_{max} + N_{max}\, N_{min} + N^2_{min}} \qquad (20.1.13)$$

where K_{afc} = 1.15 to 1.20 is a safety factor. Thus the total power required for the AFC can be determined. However, if the floor is soft and/or floor heave occurs, additional power, up to 35% of the total, should be added according to the site conditions.

Example 20.1.5. Use Eqs. 20.1.3 to 20.1.13 to select an AFC. The carrying capacity of the face conveyor Q_{afc} should be compatible with that of the shearer, that is,

$$Q_{she} = Q_{afc} \qquad (20.1.14)$$

Given the panel conditions as follows:
- Panel width W = 600 ft
- Mining height H_c = 6.5 ft
- Coal seam is approximately horizontal, $\alpha \cong 0°$
- Weight per unit volume of coal γ = 80 lb/ft^3

Major specifications of the shearer are given as follows:
- Cutting web s = 2.5 ft
- Loading coefficient C = 0.92
- Haulage speed V_s is 30 fpm on average; this is compatible with the electrohydraulic controlled advance rate of the shield supports (if the advance cycle time for each support is 10 seconds, the advance rate will be 6 shields/min or 30 fpm).

Solution. Using Eq. 20.1.3, the production of coal by the shearer is

$$Q_{she} = 1076 \text{ tph}$$

Major parameters of SCS type AFC are known as follows:
- Chain size is 30 × 108 mm
- Weight per unit length of chain q_n = 12.9 lb/ft
- Pan width B_P = 32.76 in.
- Maximum loaded cross section A_{max} = 3.0 ft
- Loading coefficient ψ = 0.85
- Weight per unit volume of loaded coal γ_c = 63 lb/ft^3
- Coefficient of moving resistance of loaded coal f_c = 0.5
- Coefficient of moving resistance of chain, f_n = 0.3

Using Eq. 20.1.6, the weight per unit length of loaded coal is determined,

$$q_c = 161 \text{ lb/ft}$$

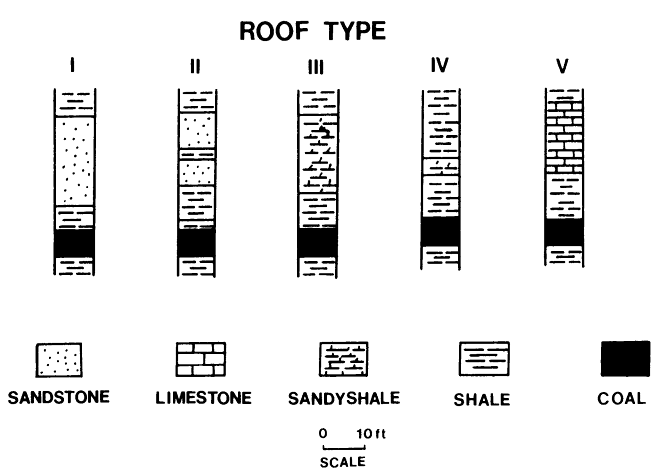

Fig. 20.1.8. Classification of roof strata (Peng, Hsiung, and Jiang, 1987). Conversion factor: 1 ft = 0.3048 m.

Use Eqs. 20.1.7 and 20.1.14 to determine the chain speed of AFC:

$$V_c = 3.74 \text{ fps}$$

Major factors selected for the AFC drive unit:
- Resistance factors K_b = 1.1 and K_S = 1.1
- Transmission efficiency Φ = 0.81
- Conversion coefficient C_{ele} = 550
- Safety factor K_{afc} = 1.2

Use Eqs. 20.1.8 to 20.1.12 to determine the maximum and minimum power capacities required for the AFC:

$$N_{max} = 618 \text{ hp}$$

$$N_{min} = 23 \text{ hp}$$

Finally, the total power for the AFC operated under the conditions as given above can be determined by Eq. 20.1.13:

$$N = 438 \text{ hp}$$

20.1.4 ROOF CONTROL

In US longwall mining, *shield supports* are predominantly used for roof control, accounting for 98% while four-leg *chock supports* are responsible for only 2%. Among shields, the majority (67%) are the two-leg type, while four-leg shields and chock-shields account for 32% with only one panel employing six-leg shields. The most important element in successful longwall mining is good roof control, which requires complete understanding of the behavior of the overburden strata for proper selection and application of powered supports.

20.1.4.1 Classification of Roof Strata

According to the thickness and uniaxial compressive strength of the immediate roof, the ratio of the thickness of the immediate roof to the mining height, and the thickness and tensile strength of the main roof, roof conditions can be classified into five types under US coal seam conditions (Fig. 20.1.8) (Peng et al., 1987):

Type I. The main roof consists of a thicker and stronger sandstone, and the immediate roof is very thin. Normally, the main roof can overhang a large distance in the gob, causing severe periodic weighting.

Type II. Features for Type II are similar to those of Type I, except that the main roof of sandstone in Type II is thinner, or that it can cave separately if multiple-layered.

Type III. The thickness of the immediate roof is one to two times the mining height, and the main roof consists of a thicker sandy shale.

Type IV. There is no main roof, or the thickness of the immediate roof is larger than four to six times the mining height. Under these conditions, there is no or only a small effect of periodic weighting; consequently, the support load requirement is small.

Type V. The main roof is a thick jointed limestone that can sag gradually and form an equilibrated semiarch. Thus, as a main roof, the effect of the limestone is very small, and the required support capacity is smaller.

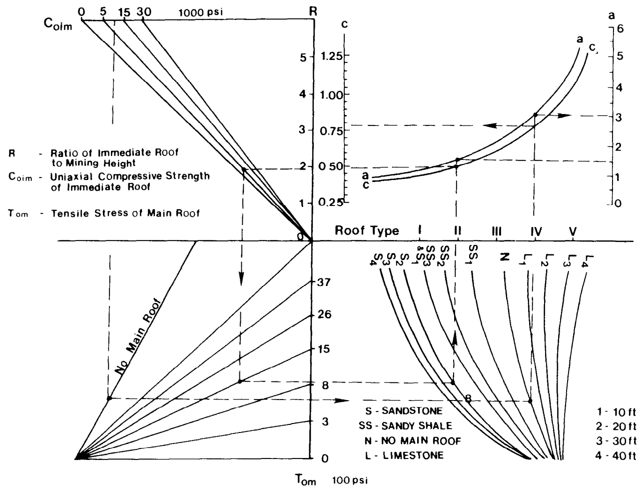

Fig. 20.1.9. Nomograph for determining constants a and c (Peng, Hsiung, and Jiang, 1987). Conversion factors: 1 ft = 0.3048 m, 1 psi = 6.895 kPa.

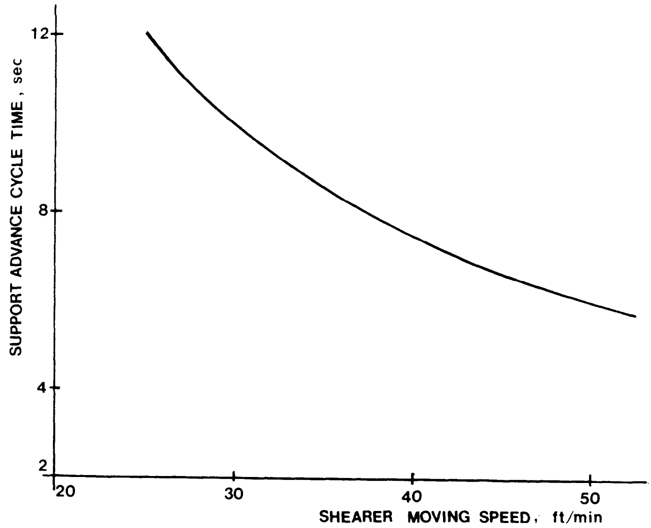

Fig. 20.1.10. Support advance cycle time required to achieve a given shearer moving speed. Conversion factor: 1 fpm = 0.3048 m/min.

20.1.4.2 Determination of Support Capacity

Based on the characteristics of the roof, a statistical model for describing the interaction between the roof strata and support has been developed (Peng et al., 1987):

$$\Delta q = 2q_s^2 \, e^{-2q_s} \tag{20.1.15}$$

where Δq is the effective increment of load density in tons/ft^2 (i.e., $\Delta q = \Delta P \eta / A$; $q_s = P_s \eta / A$); P_s is the setting load in tons; ΔP is the load increment from support setting to final load immediately before the support is released for advance in tons; η is the support efficiency; A is the canopy area of the support in ft^2; and a and c are constants related to roof conditions.

Based on this model, the following formulas can be used for calculating the setting load, load increment, and the yield load P_y:

$$P_s = 3.2 \; A / c \; \eta \tag{20.1.16}$$

$$\Delta P = 0.417 A \; a / c^2 \; \eta \tag{20.1.17}$$

$$P_y = P_s + \Delta P \tag{20.1.18}$$

Values of the constants a and c can be determined using the nomograph shown in Fig. 20.1.9. Thus the required support setting and yield capacity can be calculated using Eqs. 20.1.16, 20.1.17, and 20.1.18.

20.1.4.3 Selection of Support Type

Two-leg shields are most suitable for weak immediate roof while four-leg shields and chock shields are excellent for medium and strong roofs. A weak immediate roof refers to roof that caves immediately behind the shields, while in medium roofs the immediate roof is relatively thin and overhangs slightly. A strong roof refers to a massive immediate roof that overhangs considerably into the gob.

20.1.5 LONGWALL AUTOMATION AND REMOTE CONTROL

Since 1984, automation and remote control technology have been rapidly adopted in US longwall mining (Chapter 22.2). Now there are 72% of all longwalls equipped with electrohydraulic control, most of which are equipped with the option to expand to a shearer-initiated control system (Broussard and Palowitch, 1979; Wood, 1982; Nelson and Bessinger, 1989).

20.1.5.1 Automation and Remote Control of Support Advance

High-production longwall mining requires faster support advance. This is achieved via the electrohydraulic control system, which also provides uniform and consistent support and conveyor advance, reduces the size of the face crew, and reduces the exposure of shield men to respirable dust. The systems employ two basic modes of operation: floating batch control and shearer initiation. The floating batch control is most commonly used in US longwall panels (Peng, 1987). The adjustable size of the batch ranges from two units to full face. Usually, 8 to 30 units are recommended, because within this range a shield man can keep sight of the shearer. In a floating batch control system, several modes can be operated simultaneously, depending on the area of the face and cutting pattern adopted (e.g., bidirectional electric adjacent autosequence control, bidirectional electric adjacent manual control, and in-support hydraulic manual control). Generally, the cycle time for support advance of the latest models ranges from 6 to 10 seconds. However, the advance cycle time should be compatible with the shearer's moving speed, which ranges from 35 to 82 fpm (10 to 25 m/min). For example, a shearer speed of 35 fpm (10 m/min) requires a cycle time of less than 8.6 seconds (Fig. 20.1.10). Consequently, the shearer speed is always limited by the support advance rate, especially under bad roof conditions.

Shearer initiation is an electronic control system that issues commands for the automatic operation of support cycling and AFC advance at fixed distances from the shearer. In this system, a microprocessor console located in the head entry controls all of the support functions and another electronic unit monitors and indicates the shearer's position along the face. The initiation system controls the supports and AFC to be advanced at a predetermined distance behind the shearer. Using this system, the number of shield men can further be reduced to one for the whole face, but the shearer speed is very much limited by the support cycling time because the supports are advanced conventionally one by one.

In order to solve the problem of constraints on shearer speed, the softwares of the electrohydraulic control and shearer initiation systems have been improved so that:

1. Two adjacent supports can be advanced simultaneously, if the immediate roof is stable, or
2. Two odd supports advance first behind the shearer and then the even number of two adjacent supports, if the immediate roof is unstable, or vice versa. In this case, the snaked portion of the AFC will be longer.

In this manner, the shearer speed can be doubled, and coal production can also be increased proportionately.

20.1.5.2 Automation and Remote Control of the Shearer

The latest technology of automation and remote control for the shearer consists of remote control by the operators, cutting horizon control of the drums, face alignment control with support advance and AFC advance, and roll control of the shearer body.

Remote Control by the Operators. The main purpose of this technology is to protect the operator's safety from falls of coal and rock fragments and health from exposure to airborne dust. Using a shortwave radio system, the operator can be sited in a safe position on the intake side of the shearer about 10 to 40 ft (3 to 12 m) or farther from the face to control all shearer functions and observe the operation of the shearer. Remote control of the shearer is now commonly employed in US longwall faces.

Cutting Horizon Control. The function of horizon control is to maintain the cutting drums of the shearer to cut the seam at a predetermined distance from the coal-roof and/or coal-floor interface, so that only the coal is extracted or a fixed thickness of roof coal is left unmined. Numerous sensors have been developed and applied for horizon control system; they are divided into three groups: (1) coal thickness indicators that measures the thickness of the coal left on the roof and/or floor after a cut, (2) surface recognition sensors that differentiate between coal and rock and also indicate the coal-rock interface, and (3) roof height followers that indicate the position of the existing roof.

Data obtained by the sensors have to be transmitted to the control system for evaluation and processing, and then the commands from the control system are issued to the hydraulic system to adjust the positions of the drums.

Face Alignment Control. The irregular shape of the coal face generally is caused by irregular shearer cutting, while irregular shearer cutting is usually caused by unequal advance of both AFC and supports. In order to maintain a straight faceline, an optical instrument called the coalface surveyor has been developed for monitoring face alignment. The surveyor unit is mounted on the shearer and emits a pulse of low-intensity infrared light after every 4 in. (100 mm) of travel of the shearer. The light beam is reflected back along its initial direction by the retroreflectors, which are mounted on every fifth shield, and focused onto a linear photodiode array. The angle at the shearer subtended by adjacent two retroreflectors is continuously measured and transmitted, along with the distance traveled by the shearer, to the main processor located at the head entry. The main processor computes the face profile and displays it, interfacing with the AFC advance control system.

Roll Control. Roll of the shearer is a common event in longwall operation, which is usually caused by undulating floor, soft bottom coal and floor, and the inherent instability caused by the offset of the drum from the supporting shoes, especially with longer ranging arms. Two kinds of instruments have been developed for monitoring shearer roll, the tilt transducer and inclinometer.

1. *Tilt transducer.* It measures the inclination of the shearer based on the desired orientation of the AFC, which may not be horizontal. The measured data are transmitted to an automatic adjustment control system, and the shearer angle necessary to reach the desired orientation is computed by the control unit. Then the shearer position is reoriented by adjusting the underframe steering jacks.
2. *Inclinometer.* It is an integrated part of the roll control system and detects the angles of shearer roll by gravitational acceleration. The vibrational accelerations along the sensitive axis will be transferred into roll signals, and the control system rotates the shearer about its longitudinal axis by using the hydraulic actuators to adjust the shearer position to the desired orientation.

20.1.5.3 Status of Implementation of Longwall Automation and Remote Control

Control of shields using the electrohydraulic control system is a routine operation. In this system, no more than two shield operators are required for a longwall face up to 900 ft (270 m) in width, and in some faces only one operator is sufficient.

Shearer initiation has been successfully demonstrated in operational faces, but the practice of advancing shields one by one slows down the shearer's cutting speed. Therefore, if shearer initiation is to be widely used, some techniques of improving the shield advance time such as those mentioned earlier must be perfected.

All shearers are equipped with remote control either by radio or umbilical cord. The first level of automation is the use of nucleonic sensors to detect the roof coal thickness and then direct the drum to cut and leave a predetermined roof coal thickness. In more advanced levels, the drum cutting height at all locations along the face for one cutting cycle is stored in a computer. These data are used to direct the second cutting cycle during which the roof sensors continue to collect a new set of data. If there is any discrepancy between the 2nd and the 1st cycle, the 2nd cycle data will replace the 1st cycle ones and be used for the 3rd cycle setting. This process repeats for all subsequent cutting cycles.

It must be noted that current automation technology is not universally applicable. The roof must be fairly good without large roof falls or steps in unsupported areas, and the floor must not be too soft and sharply undulating.

20.1.6 LONGWALL FACE MOVE

A longwall face move is a major job when one panel has been completed and all the longwall equipment is transferred to the next panel for setup. During a longwall face move, production is interrupted, resulting in a decrease in overall longwall output and productivity and in an increase in the operating cost per ton, in addition to an interest increases due to the large capital investment. For example, with an $8 million investment and a 20% return on investment, a longwall move of 4 weeks causes a loss of $135,000, in addition to labor expense of $200,000 for the move, and the interest and depreciation costs of the equipment (Adam et al., 1982). The key to reducing the loss from a longwall move is to shorten the move time, which ranges from four days to four weeks. The major factors in reducing longwall move time are good planning and coordination, proper tools and equipment, and effective and efficient operations (Peng and Chiang, 1984; Adam et al., 1982).

A longwall face move includes the following steps: pre-move preparation, move preparation, move and transportation, and installation and completion.

20.1.6.1 Pre-move Preparation

At least one month before the last panel is completed, a number of tasks for preparing for the move should be started and performed. The major ones include (1) preplanning, (2) arrangement of supplies, tools, and equipment, (3) preparation of the transportation route and communication system, (4) location of transfer points, storage areas, and repair shops, (5) provision of communication system, and (6) manpower organization and training.

20.1.6.2 Move Preparation

Move preparation includes preparation of the recovery room and inspection and maintenance of face equipment.

PREPARATION OF RECOVERY ROOM. Usually, the last 5 to 10 cuts from the termination point of a longwall are used to prepare a safe and efficient *recovery room.* This consists of determination of the roof control method, room size, and room installation.

Roof Control Methods—Methods of roof control in the recovery room depend on the immediate roof conditions. Generally, steel wire meshes over canopies by bolting are commonly used under either weak roof or stable roof conditions. However, under weaker roof conditions, in order to strengthen the supporting capacity of the wire meshes, wire rope and/or steel I-beams are laid parallel to the faceline on top of the wire meshes at regular intervals, comparable to the center-to-center distance of the supports. Sometimes, cribs and single posts are also used as temporary supports.

Room Size—Usually, the height of the recovery room remains the same as the normal mining height. Sometimes, the room height at the panel end may be increased to facilitate equipment recovery. The effective width of the recovery room is the lineal distance between the faceline and the front of the base plate of the support. It must be sufficient for shield removal.

Room Installation—For example, crib blocks needed during the support withdrawal operation must be delivered and placed in the recovery room or close to the room. Rails and winches must be installed, if rail transportation is selected.

INSPECTION AND MAINTENANCE OF FACE EQUIPMENT. A careful and comprehensive inspection of all face equipment must be planned and performed before the longwall face reaches the termination point. Based on the inspection, usually some maintenance and repair work can be performed at the face and completed prior to terminating the face during downtime and maintenance shifts, and some maintenance and repair work should be scheduled for completion during the face move at the underground repair shops. Any equipment that is to be employed for the next panel but can only be overhauled on the surface requires a special schedule with the highest priority.

20.1.6.3 Face Move

Face-move procedures depend on the transportation route and methods, available spare equipment, and maintenance requirements. The detailed move procedures vary from mine to mine and time to time. As an example, the sequence of removing and transporting the major equipment for a shearer face consists of (1) entry belt conveyor, (2) power center, (3) stage loader, (4) face conveyor chain units, spill plate, head drive, and cable handler, (5) shearer loader, (6) communication system, cables, and water lines, (7) face conveyor linepans and tail drive, (8) face lighting system, and (9) shield supports.

The setup room should be well prepared: the face side rib must be straight, the floor smooth and the roof well supported.

The installation sequence of the face equipment in the setup room depends on the withdrawal and transporting sequence from the recovery room and the amount of spare equipment available. Generally, the installation sequence at the new face is approximately in reverse order as the old face is removed, but some installation operations can be performed simultaneously because the equipment can be entered from both head and tail entries. However, all installation should follow a critical path flowchart and/or Gantt chart, according to the face move plan.

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Chapter 20.2
SUBLEVEL CAVING

RUDOLF KVAPIL

20.2.1 INTRODUCTION

Sublevel caving is a mass mining method based upon the utilization of gravity flow of the blasted ore and the caved waste rock. As with any other mining method, sublevel caving has advantages and disadvantages that must be carefully considered and evaluated for mine design and planning.

Major advantages of sublevel caving are the following.

1. Safety. Because all the mining activities are executed in or from relatively small openings, sublevel caving is one of the safest mining methods. Drifts are the primary working places. They are distributed in a uniform pattern on all levels. Normally, the maximum dimensions of the sublevel drifts are about 17 to 20 ft (5 to 6 m) wide and 12 to 13 ft (3.7 to 4.0 m) high. Transportation drifts can have the same cross section, or the height may be increased to about 15 ft (4.5 m) when trucks are loaded in the transport drifts. The stability and safety of such drifts in competent rock can be easily controlled by smooth blasting alone or by a combination of both smooth blasting and shotcreting. In less competent rock masses, stability can be achieved by a combination of smooth blasting, shotcreting, and rockbolting.

2. Mechanization. As shown in Fig. 20.2.1, when using the sublevel caving method, major mining activities can be broken down into four groups of unit operations: (1) drifting and reinforcing, (2) fan drilling, (3) production blasting (i.e., fragmentation), and (4) ore drawing, loading, and transportation. Because of the repetitive nature of the mining system, almost all mining activities can be standardized. This means that a high degree of mechanization is possible. In modern sublevel caving, cross sections of the drifts and tunnels are sufficiently large to allow the introduction of large trackless mining equipment. The advantages of a trackless system can then be broadly utilized not only for direct mining but also for all services.

3. Flexibility. Standardization and specialization of mining activities and equipment on separate levels (lower level or levels in development, upper level or levels in production mining) together with a trackless haulage system creates a high degree of flexibility. This allows a rapid start-up of mining and flexibility when making production rate changes.

4. Work Organization. This method lends itself to good work concentration, organization, and working conditions. Normally, on the lower levels, various phases of development are underway. Upper levels are in various stages of extraction. Therefore, the work can be easily organized into a system which excludes interference between mining activities.

In summary, safety of mining (in small-dimension openings), good flexibility, work organization, and high mechanization using large modern mining equipment provide very good working conditions. The method also facilitates a high work concentration and rationalization of separate specialized mining activities. Therefore, mining by sublevel caving can be effective and relatively inexpensive.

Major disadvantages of sublevel caving are

1. There is a relatively high dilution of the ore by caved waste, especially when higher recovery is requested.

2. All ore must be drilled and blasted in order to obtain a coarse material suitable for extraction by gravity flow.

3. Various types of ore loss can occur. When the extraction limit (that point yielding the maximum acceptable amount of dilution) is reached, the remaining highly diluted ore represents an ore loss. Some ore is lost in passive zones located on the level of extraction between the active zones of the gravity flow. A small part of the ore forming these passive zones can be recovered together with ore extraction on the lower sublevel, but some undiluted ore located in passive zones above the plane of the footwall is lost. In general, these losses are larger as the inclination of the ore body and the footwall is reduced.

4. A relatively large amount of development is required. This includes transport drifts, usually located in the footwall waste rock on each sublevel, and sublevel drifts, which connect the active mining areas in the ore body to the transport drifts. The sublevel drifts are partially in ore and partially in the waste rock. The waste rock length increases as the inclination of the ore body and footwall decreases. The system also needs orepasses, used to transport the ore or waste from the separate sublevels downward to the main haulage level, and these openings are located in waste.

5. Mining generates progressive caving in overlying rock and results in subsidence and damage to the surface.

6. To maximize ore recovery, minimize dilution, and achieve a high efficiency of mining by sublevel caving, good data regarding the gravity flow parameters for the blasted ore and the caved waste are of utmost importance. The exact type and amount of data required depend upon the purpose and needs of the mine design. For the first feasibility study, it may be sufficient to utilize the data from other sublevel caving operations with similar conditions and circumstances. For any higher level of mine design and planning, it is clear that more exact data, including analytical and experimental analyses up to full-scale in situ testing, are necessary.

Gravity flow principles and design guidelines for the application of the sublevel caving mining method are presented in the segments that follow. Although somewhat simplified, they should provide a basis for mine planning and operation. The gravity flow principles described can be effectively applied to other mining situations, with some modification. Also steeply dipping formations such as coal seams can be effectively mined by modified sublevel caving.

In sublevel caving, all ore must be drilled and blasted in order to utilize the gravity flow of fragmented ore, which after blasting forms so-called “coarse material”. The objective is to obtain by minimum drilling and blasting a coarse material with a fragmentation that will permit gravity flow and undisturbed ore extraction.

The coarse material is characterized by different size distribution and shape. The size distribution and shape of fragments depends on the density of the drillholes and on “blasting/tectonics” of the ore. Coarse materials can be very heterogeneous. With certain simplification, we could distinguish (with respect to their behavior) four basic types (Fig. 20.2.2). Type I shows coarse material with large “spherical” pieces of more or less the same size and shape. Type II represents a material almost of the same size but different shape. Type III indicates a coarse material composed of large fragments, chippings, and sand. Type IV

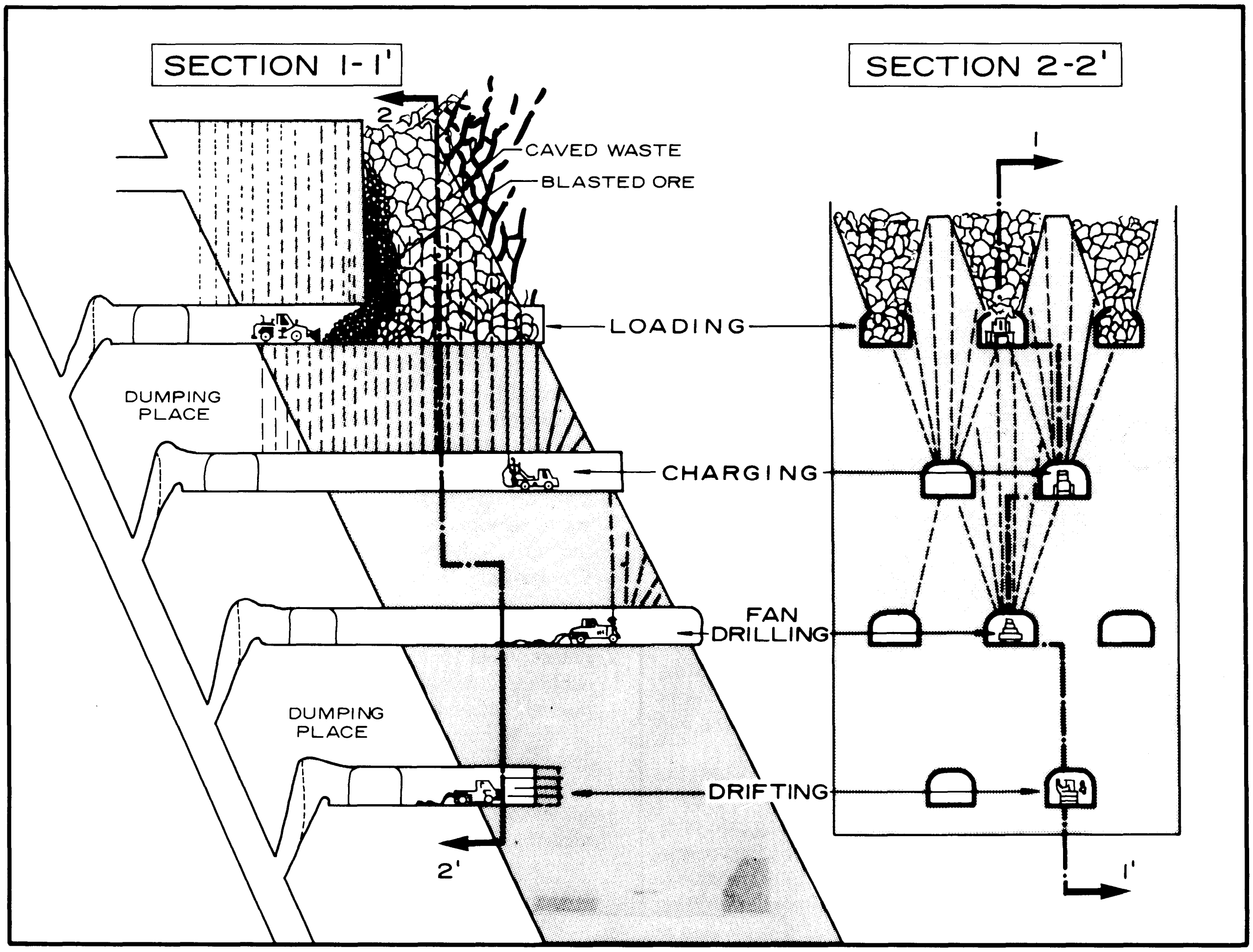

Fig. 20.2.1. Typical schematic of sublevel caving visualized on a simplified transverse section 1-1′ and longitudinal section 2-2′ (after LKAB information brochure, 1989).

represents a coarse material that is characterized as a mixture of large blocks, medium-sized fragments, chippings, sand and/or rock powder, earth or clay components, etc. The coarse material, Type III, and especially Type IV, could change considerably the mechanical properties and behavior depending on the percentage of fine particles (sand, clay, etc.) and on the presence of water. If water is present, then especially in coarse material, Type IV, the fine particles could create plastic and sticking components. This could change very unfavorably its properties and behavior.

Naturally, the coarse materials could have very different properties. The most simple characterization is shown in Fig. 20.2.2, indicating the angles for gravity transportation of coarse material. Up to about 40° the coarse material requires mechanical transportation. With larger angles of inclination, gravity transportation could be used in open chutes or in orepasses. In Fig. 20.2.2, GF represents the range of gravity transportation applicability, where the range A is for open chutes for transportation of Types I and II only, while the Types III and IV (in inclination range B) must be transported in steep passes. Type III requires inclination BI, and Type IV, especially when wet, needs very steep passes in the inclination range BII (Fig. 20.2.2). Almost all blasted ores (and waste) are similar to Types III and IV. The composition, properties, behavior, etc., of these materials could be very different because they could consist of very large boulders down to the finest particles.

20.2.2 PRINCIPLES OF GRAVITY FLOW

In order to construct efficient bins or silos, it is necessary to know the principles of *gravity flow* the stored materials. Undisturbed operation of a bin is a function of the gravity flow, of the material.

Gravity flow of blasted ore and caved waste in sublevel caving is a process much more complicated than flow in bins. The geometry, dimensions, and even operational constraints of sublevel caving cannot be selected at random, but must be planned respecting the laws that govern the gravity flow of coarse materials (i.e., the gravity flow of blasted ore and caved waste rock). Because of the heterogeneity of coarse materials and a great number of other factors and conditions, the gravity flow of these materials is a very complex process.

It must be emphasized that although flow as used in gravity flow means an uninterrupted movement of coarse material, it is a completely different process from the flow of liquids. Hence knowledge of the flow of liquids cannot be applied to gravity flow of coarse materials. The basic laws and principles of gravity flow are independent of fragment size; they are the same for coarse material as for small granular material (i.e., small gravel or sand). Geometrical parameters are different in some materials and under varying conditions.

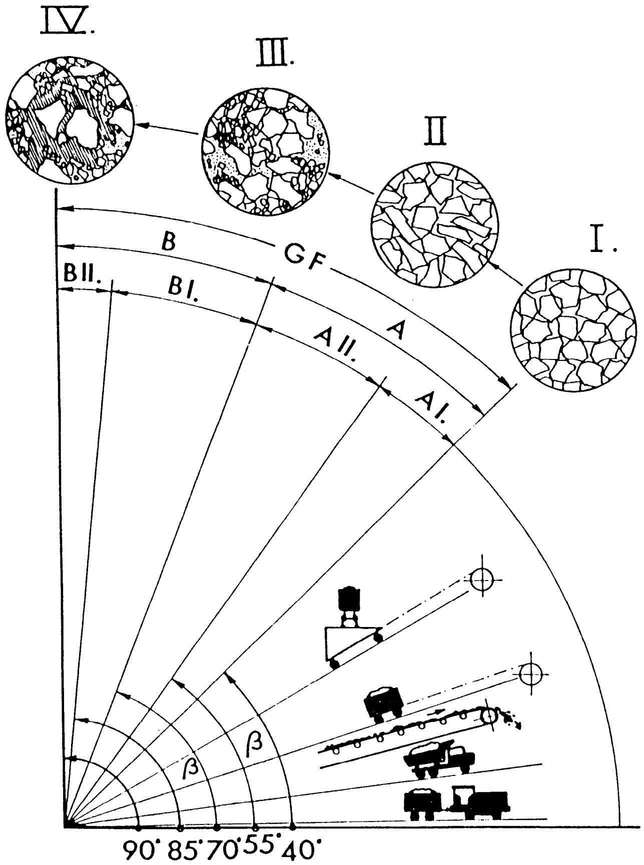

Fig. 20.2.2. Four basic types I through IV of coarse material and their mobility as a function of inclination of chute or orepasses.

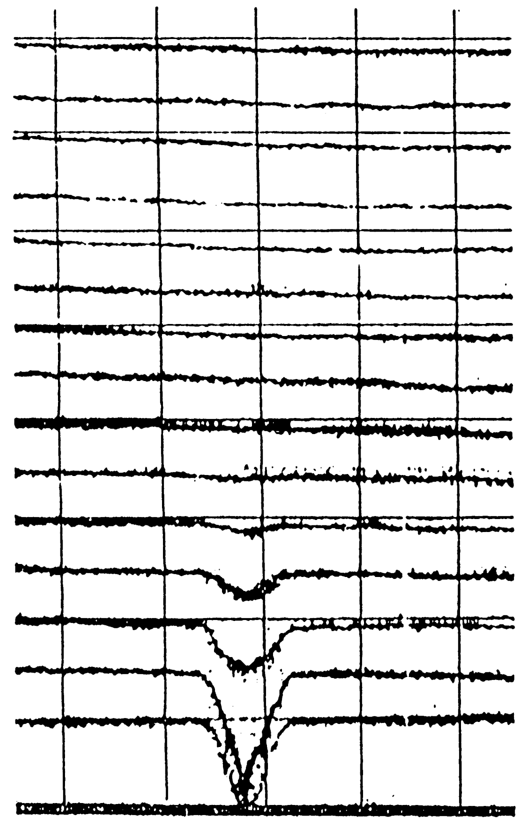

Fig. 20.2.3. Model of gravity flow; beginning of material extraction initiates the gravity motion of sand filling, visible by curvature of originally horizontal black line.

20.2.2.1 Factors That Control Gravity Flow

Gravity flow is usually demonstrated with a very simple vertical glass model with horizontally layered white and black sand filling, as shown in Fig. 20.2.3. For model research, we can use more sophisticated models, such as having the fill materials arranged like a black and white checkerboard (Fig. 20.2.4).

It has been found that gravity flow in a bin has a rather complex form, as is shown in Figs. 20.2.5 and 20.2.6 (especially when the involved height of the granular or coarse material is small). This form is even more evident from the model of a bin shown in Fig. 20.2.7, where the apparently low cohesion of sand is caused by a small percentage of water. (Note: As is evident from Figs. 20.2.5 and 20.2.6, the inclination of the bin bottom has no influence on the gravity flow of its fill materials in the short term. The bottom must be much steeper in order to influence the flow in the bin or silo.)

For simulation of the gravity flow principle used by sublevel caving (or block caving), the most simple model was used. This model represents a vertical section of a bin or a sublevel through the axis of its extraction opening (or sublevel drift) on the bottom. The extraction opening in the model bottom has minimum dimensions, but it is sufficiently large for a fluent and uninterrupted flow of material. (For simulation of certain sublevel caving conditions, the model is continuously refilled during extraction). Figs. 20.2.8 through 20.2.10 represent successive phases of material extraction.

The deflection of the originally horizontal thin black layers indicates the *active zone,* that is, the zone with gravity motion of the material. Because the motion is caused by the gravity, the axis of the active zone is vertical. The material with an intact white and black pattern is stationary and forms the so-called *passive zone.*

As simply shown on Fig. 20.2.8, when the individual black lines at the boundary of gravity motion are connected, one obtains a form similar to an ellipse. The real shape, as has been demonstrated on the models in Figs. 20.2.5 through 20.2.7 is not elliptical, but the elliptical shape is introduced for simplification and also, because at greater heights, the active zone is very similar to an ellipse. In reality, this ellipse is a vertical and axial section of an axisymmetric body, namely, that of an elongated ellipsoid of revolution whose geometry can be explicitly defined by eccentricity. This simplification enables application of simple numerical analysis.

It should be emphasized that the demonstration of gravity flow depicted in Figs. 20.2.3 through 20.2.10 is incomplete and can lead to erroneous conclusions because gravity flow is even more complex.

To avoid errors, it must be understood that what is being demonstrated by these successive phases is only the consequence of the extraction of a certain material volume from the model. The elliptical zones of the gravity motion can now be seen, but only the gravity motion of that material still remains in the model.

After the extraction of a certain volume, the remaining material replaces this loss by its loosening. Although it is evident that a certain relation exists between the extracted material and the loosening, only the visible zone of the *ellipsoid of loosening* can

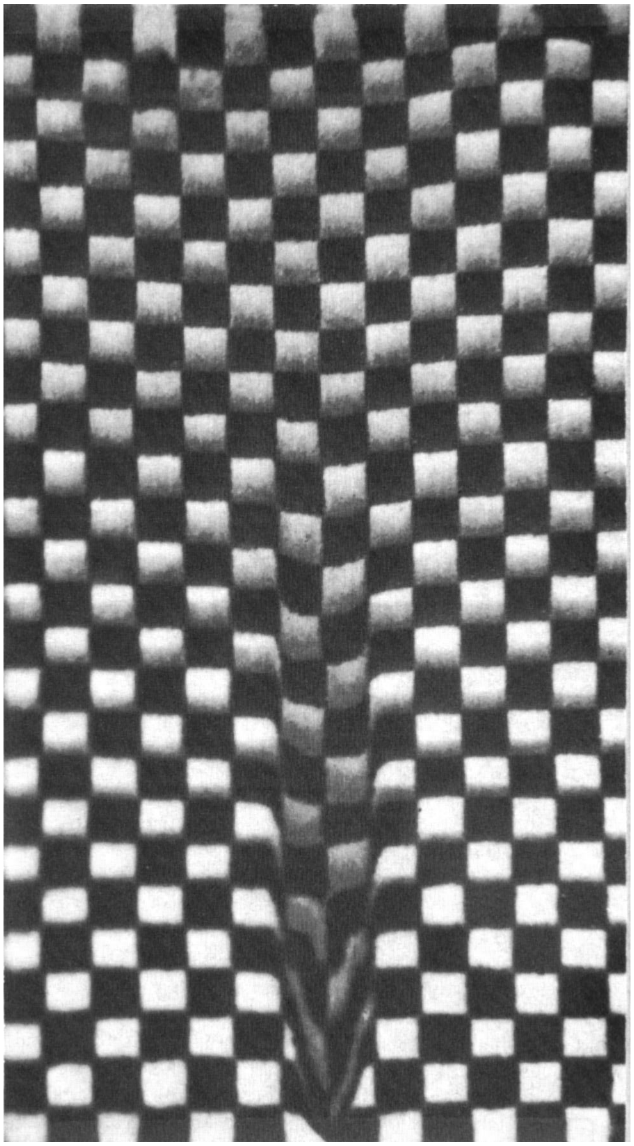

Fig. 20.2.4. Beginning of gravity motion in a model with checkered (black and white) sand (Kvapil, 1955).

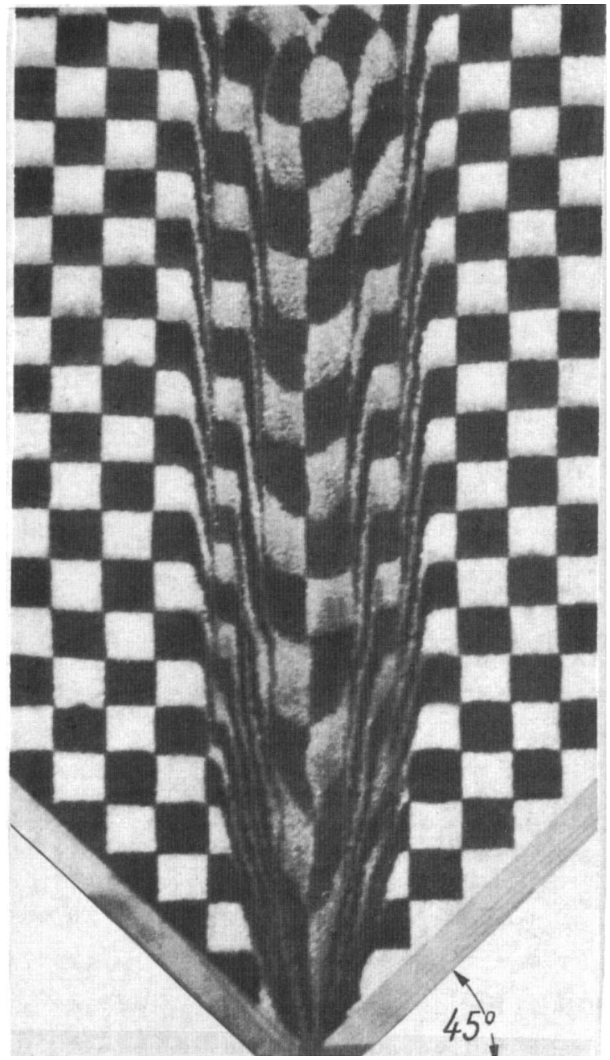

Fig. 20.2.5. Model of gravity flow shape in a bin with 45° bottom inclination (Kvapil, 1955).

be defined by the models shown above. The size of the ellipsoid (and its eccentricity) increases as more material is extracted. Depending upon material properties and the acting conditions, the volume of the ellipsoid of loosening can be about 14 to 16 times larger than the volume of extracted material.

From observing the pattern of deflected black lines (or checkered pattern in different phases of extraction), it is not possible to define either the location, shape, dimensions, or volume of the zone from which the material, now lying below the discharge opening, has been extracted. Extraction of a certain volume of material results in the development of a certain ellipsoid of loosening. Under these conditions, there is momentarily no motion at the boundary of the ellipsoid of loosening, and the greatest particle velocity is in the center of the discharge opening. The result of an analysis to determine the velocity distribution on levels E-E′ down to A-A′ is shown in Fig. 20.2.11. This provides an axonometric representation of the envelope surface of velocity vectors from $V = 0$ to $V_1 > V_2 \ldots > V_5$ For better visualization, the velocity vectors are not constructed in a true direction, but perpendicular to the axial section of the ellipsoid, with zero velocity on the boundary.

From Fig. 20.2.11, one can easily derive the zones of the same particle velocity V_1 defined on the boundary. This is shown in Fig. 20.2.12. The line which connects the particles of the same velocity forms an elliptical figure in a vertical section, and in space an ellipsoid of the same velocity. Evidently, the shape of the gravity flow zones is controlled by a specific distribution of the velocity of motion, resulting in ellipsoids of the same velocity. Therefore, not only does the zone of loosening have the shape of an ellipsoid (boundary of the ellipsoid of loosening has the momentary velocity = 0), but so also does the zone from which the discharged material was extracted. This zone, similar to the elongated ellipsoid of revolution, is called the *ellipsoid of extraction.*

20.2.2.2 Simplified Theory

The existence of the extraction ellipsoid can be demonstrated by using several different methods. One possibility is a three-

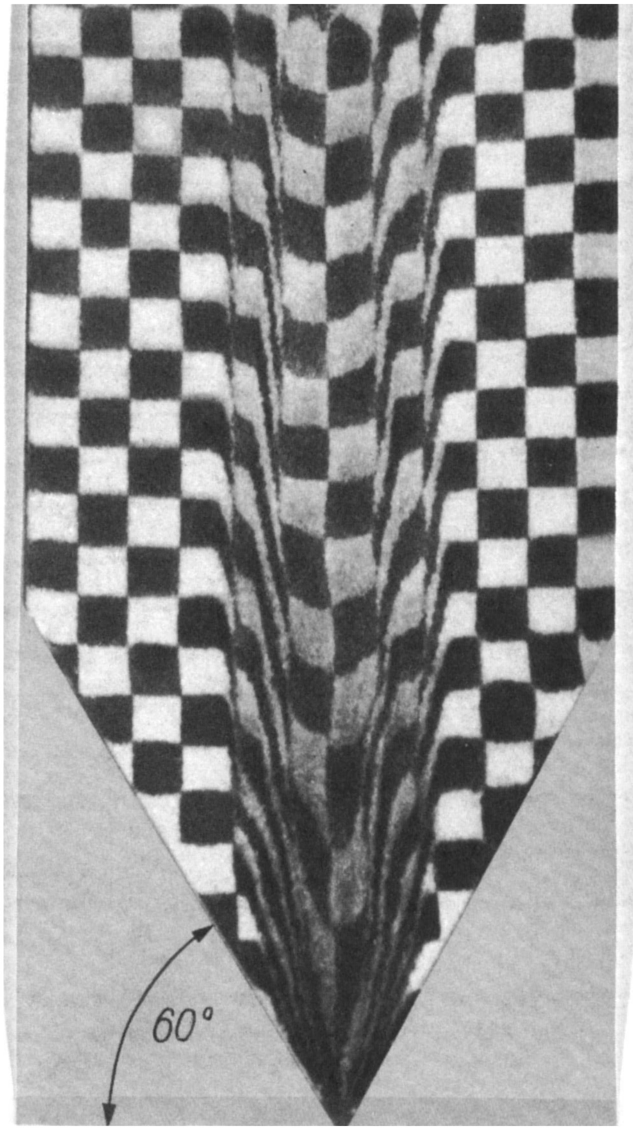

Fig. 20.2.6. Same case as in Fig. 20.2.5, but bottom of bin is inclined by 60° (Kvapil, 1955).

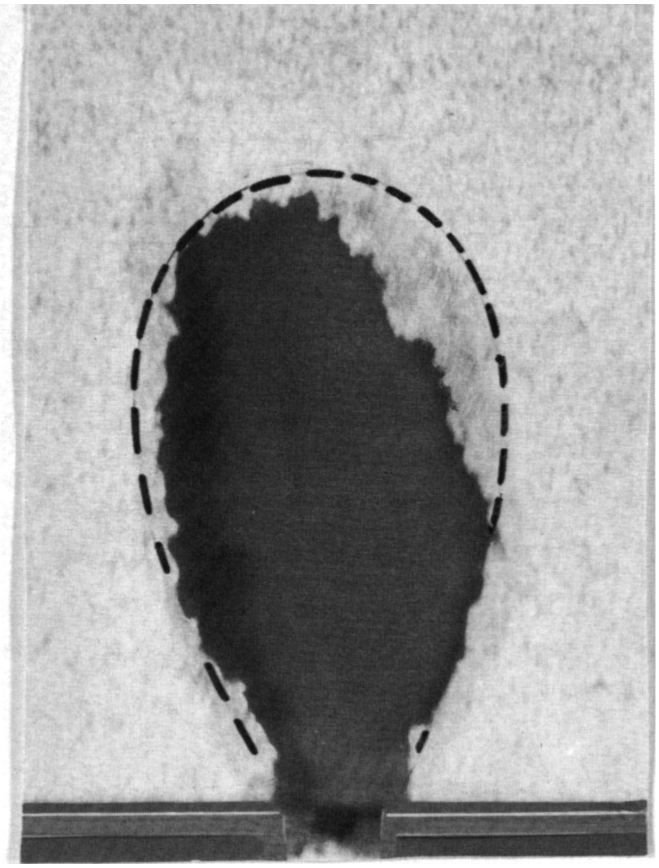

Fig. 20.2.7. Exaggerated shape of gravity flow visualized by a cavity in sand model with apparent cohesion (Kvapil, 1955).

dimensional model in which markers are located in a certain pattern in the granular material. When the original location of the marks is known (defined by space coordinates) before the extraction, the extracted markers together with the extracted material will define the extraction ellipsoid.

This method of using markers has been applied to determine the extraction ellipsoid (and also the entire active zone) in bins and silos. In order to ensure that the markers will correctly follow the gravity flow of stored material, they were constructed from rubber rings and picked out on a screen below the outlet. In sublevel and block caving tests, the markers were made of pieces of compressed air hoses (rubber), longitudinally cut and marked inside with a short-half-life radioactive color in order to determine their coordinates. These markers were fixed in the drillholes by cement grouting, and then detected in the muck using a Geiger-Müller radioactivity counter (Kapil, 1954). This system was also used to project and realize full-scale tests of sublevel caving in Grängesberg, results of which were later published (Janelid, 1975).

Probably the most instructive evidence of an extraction ellipsoid is obvious from Fig. 20.2.13. Phase a represents the model with a clearly marked ellipsoid of extraction. The eccentricity of the extraction ellipsoid was determined on a separate model with the same granular material. Apex point N is at a distance h_n above the discharge opening. Note that h_n represents the height of the extraction ellipsoid. As evident from phases b, c, and d (Fig. 20.2.13), increased extraction results in a successive decrease in the volume of the extraction ellipsoid and an increase in the volume of the loosening ellipsoid. When the apex point N arrives at the outlet, the entire volume of the extraction ellipsoid has been discharged and forms a cone below the discharge opening.

The gravity flow mechanism is shown diagrammatically in Fig. 20.2.14a, b, c, and d, which is identical to the model test of Fig. 20.2.13.

From Fig. 20.2.14d, it is evident that, with complete extraction of the extraction ellipsoid, the original horizontal plane n passing through the apex point N is deflected downward forming the outflow funnel 1, N, 2. Points 1 and 2 intersect the ellipsoid of loosening at the height h_n, which is equal to the height of the extraction ellipsoid. It means that the outflow funnel diameter 1, 2 is equal to the horizontal section of the ellipsoid of loosening measured at the height of the apex point (N) of the extraction ellipsoid. The volume of the outflow funnel is the same as the volume of the extraction ellipsoid.

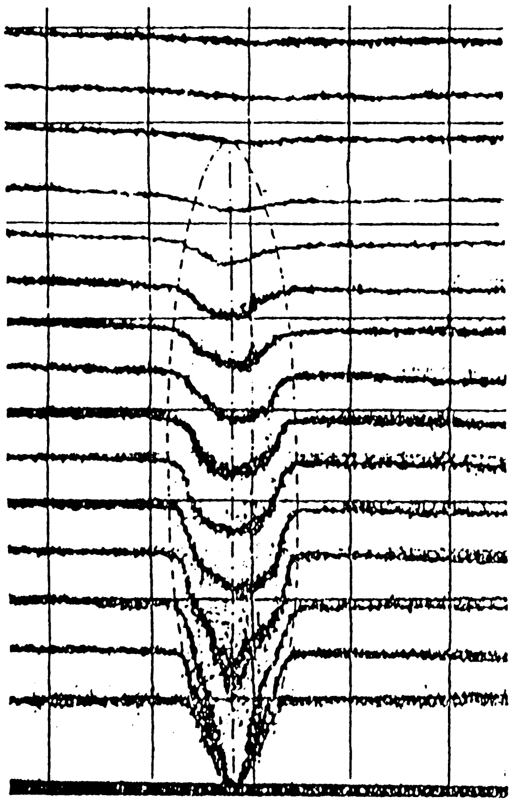

Fig. 20.2.8. Successive phase of material extraction from model shown in Fig. 20.2.3. Boundary of gravity motion forms a zone similar to an ellipsoid. It is an ellipsoid of loosening that is shown in the model by an ellipse.

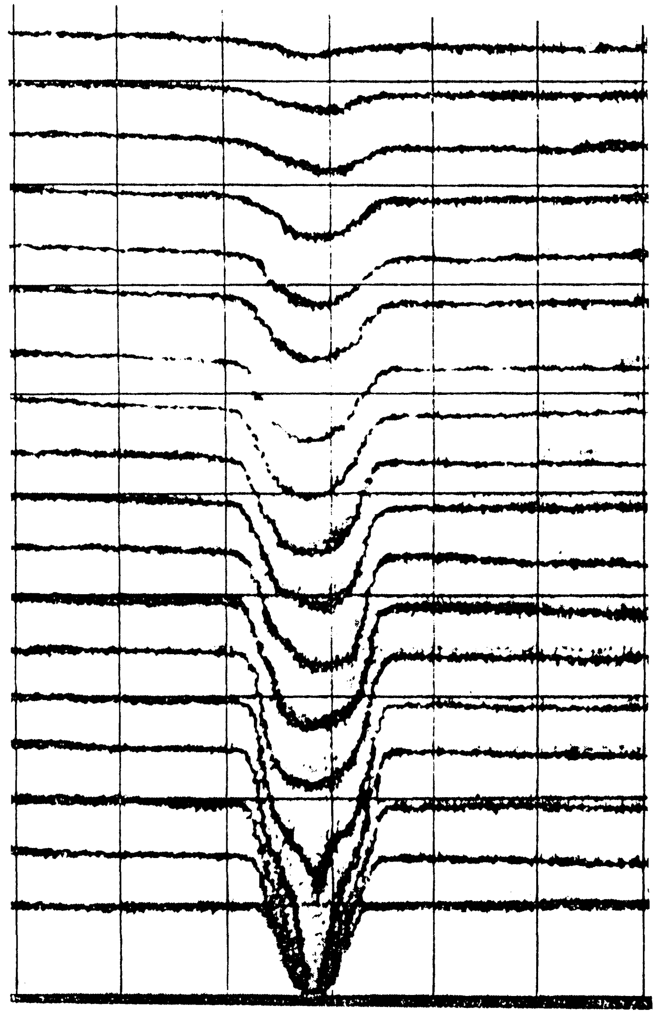

Fig. 20.2.9. Gravity motion created by successive phase of extraction.

As shown in Fig. 20.2.15, for a certain extraction height h_n, the mechanism of gravity flow is defined by explicit relationships between the discharged material, the ellipsoid of extraction, the outflow funnel, and the ellipsoid of loosening.

Using the designated nomenclature (V_c is volume of discharged material, E_E is the extraction ellipsoid, V_{EE} is volume of the extraction ellipsoid, h_n is height of the extraction ellipsoid, E_L is ellipsoid of loosening, V_{EL} is volume of the ellipsoid of loosening, h_L is height of the ellipsoid of loosening, F is outflow funnel, and V_F is volume of the outflow funnel), the characteristic relations for gravity flow can, by introducing certain simplifications, be expressed by the following formulas:

$$V_{EE} \approx V_C \tag{20.2.1}$$

that is, the volume of the extraction ellipsoid is the same as the volume of discharged material,

$$V_{EE} \approx V_C \approx V_F \tag{20.2.2}$$

that is, the volumes of extraction ellipsoid, discharged material, and outflow funnel are the same,

$$E_{EL} \approx 15\ V_{EE} \approx 15\ V_C \approx 15\ V_F \tag{20.2.3}$$

that is, the volume of the ellipsoid of loosening is about 15 times larger than the volume of the extraction ellipsoid, the volume of discharged material, or the volume of the funnel.

Assuming approximately the same eccentricity of the ellipsoid of loosening and the extraction ellipsoid, and knowing that $V_{EL} \approx 15\ V_{EE}$, then

$$h_L \approx 2.5 h_n \tag{20.2.4}$$

Hence the height h_L of the ellipsoid of loosening is about 2½-times greater than the height h_n of the extraction ellipsoid.

These relationships, which are generally valid, form the basis for understanding, explaining, and justifying the different phenomena involved in gravity flow.

Consider, for example, the situation in which the lower part of the material with height h_n is the ore, denoted by R in Fig. 20.2.15, and the upper part is the waste rock W. As is evident from Fig. 20.2.15, the maximum volume of pure ore that can be discharged without any waste rock (i.e., without any dilution) is only that from the extraction ellipsoid with height h_n. Any additional material discharge will result inevitably in a rapid increase in dilution because the outflow funnel, equal to the discharged extraction ellipsoid of height h_n, is now filled by waste rock.

For the same extraction height h_n, the extraction ellipsoid can have a different volume in different materials because the

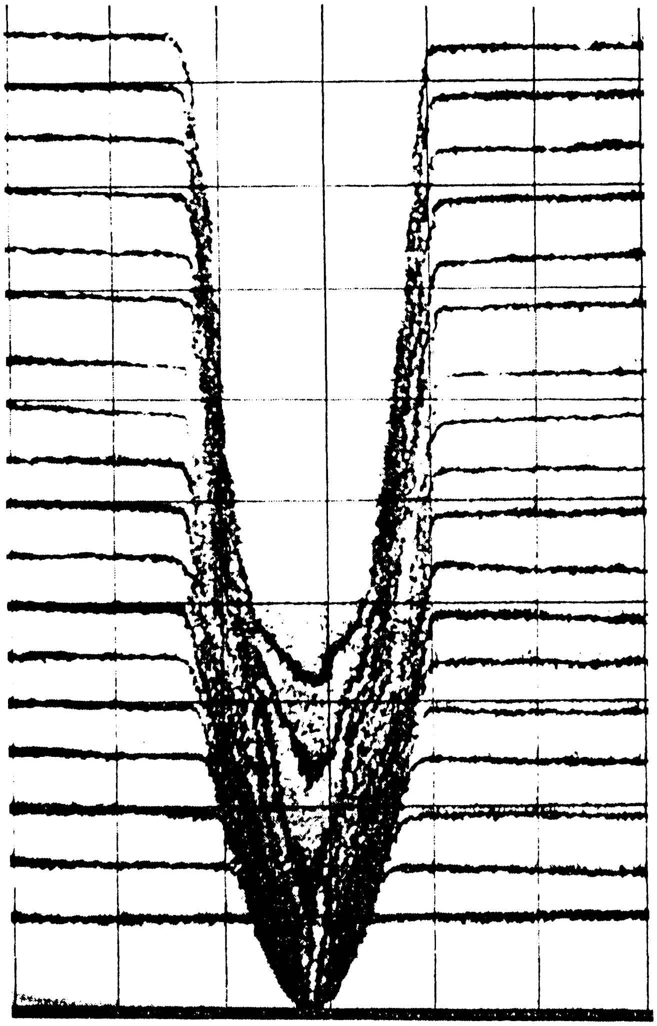

Fig. 20.2.10. Gravity motion created by much later successive phase of extraction in Fig. 20.2.9. Because the model is filled simultaneously while undergoing extraction, this figure shows the flow of white sand into the active zone.

eccentricity ϵ of the ellipsoid of extraction (and loosening) depends upon the material.

The formula for eccentricity of the ellipsoid is

$$\epsilon = \frac{1}{a}\sqrt{(a^2 - b^2)} \tag{20.2.5}$$

where a is the semimajor axis and b the semiminor axis of the ellipsoid (see also Fig. 20.2.15). Naturally, the greater the eccentricity, the slimmer the ellipsoid of extraction (or loosening) and the smaller the volume.

It is well known that in fine material (cement, fine sand, etc.), the active zone is very slender, and in coarse material, the active zone is broad. This is shown in Fig. 20.2.16a for very fine material, in Fig. 20.2.16b for granular material, and in Fig. 20.2.16c for coarse material. The eccentricity of the ellipsoid of extraction and loosening also increases with the height of the ellipsoids. This effect, which is relatively small in sublevel caving, has a much greater importance in block caving because of the very large block heights.

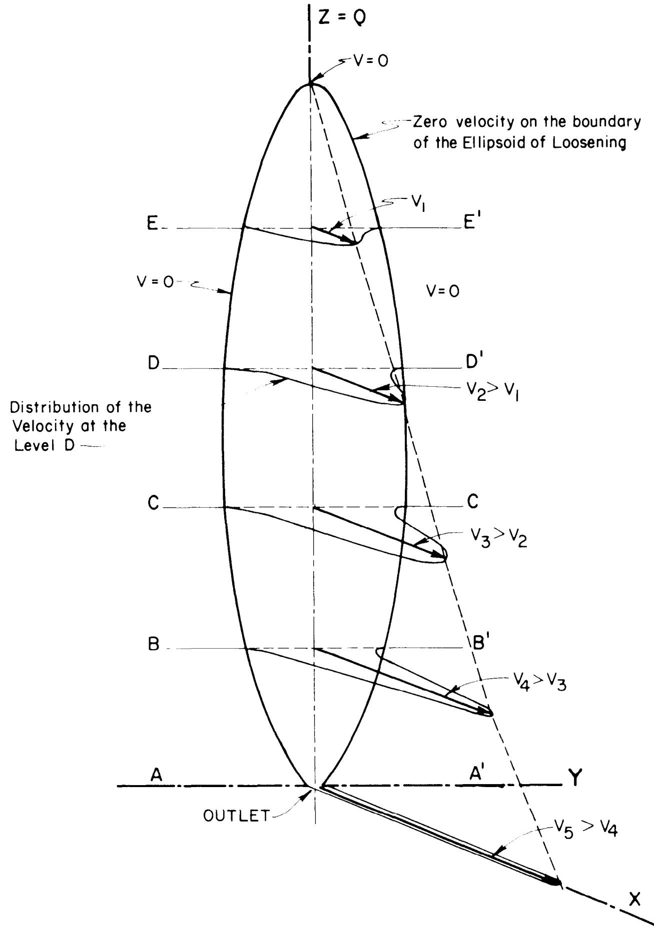

Fig. 20.2.11. Distribution of velocity in an ellipsoid of loosening (or extraction).

The eccentricity depends not only upon the size of the particles (and the gravity flow height) but also on many other factors, such as shape or form of the particles (spherical, irregular, etc.), surface roughness of the particles (smooth, rough), angle of friction (small, large), density (high, low), percentage of components with a lubricating effect (high, low), extraction or discharge rate (fast, slow), and material properties of the particles (strength, moisture, etc.).

All of these factors result in a certain behavior which can be expressed in terms of the mobility of granular or coarse materials. Greater material mobility results in smoother gravity flow and higher eccentricity of the ellipsoids. This means that in materials with high mobility, the ellipsoids are slender and in materials with low mobility, the ellipsoids are broad and contain large volumes.

Although an absolute scale is not yet available, the mobility of materials [classified between the limits of high (= very good) to low (= poor mobility)] can be used as a qualitative comparison of the flow behavior. From this indication of the flow behavior, one can deduct the approximate geometry of the ellipsoids. Fig. 20.2.17 shows the shape and eccentricity of the ellipsoid of extraction and loosening as a function of material mobility.

20.2.2.3 Specific Application to Sublevel Caving

For a vertical sublevel front, the sublevel drift forms a vertical opening, located in the plane of the sublevel front. Therefore,

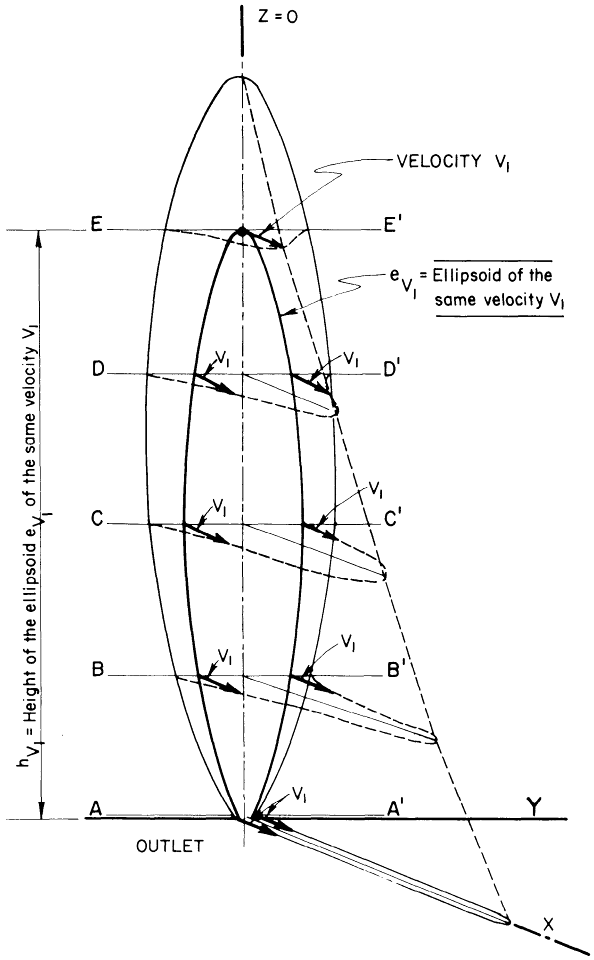

Fig. 20.2.12. Ellipsoid of same velocity.

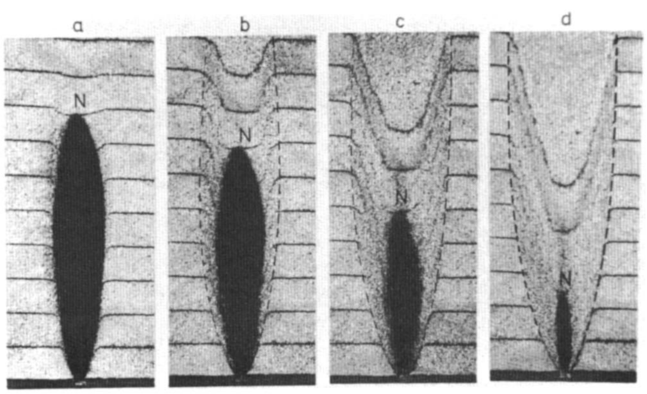

Fig. 20.2.13. Ellipsoid of extraction: (a) represents the model prepared for extraction; (b), (c), and (d) depict successive phases showing the flow of (black) material from the extraction ellipsoid and simultaneous development of the ellipsoid of loosening.

on a vertical section A-A′ as shown in Fig. 20.2.18, the geometry of sublevel caving is similar to a bin with discharge openings located not on the bottom but in the vertical wall of the bin. As is evident from the model in Fig. 20.2.19, the gravity flow zone is cut off by the vertical bin wall but is otherwise unchanged. It means that the vertical wall cuts off the ellipsoid of extraction and loosening as shown in Fig. 20.2.20.

The axis of gravity flow on this section deviates from vertical by a certain angle Δ. This angle increases as the friction along the vertical wall increases. Neglecting this deviation, one can assume that the vertical wall with vertical discharge opening cuts off half of the ellipsoid of extraction and loosening. Naturally, when this half of the ellipsoid is inscribed in a prism, then its volume is 50% of this prism. Therefore, in sublevel caving, if half of the extraction ellipsoid is inscribed in the body of blasted ore as shown in Fig. 20.2.18, then a maximum of 50% of the blasted ore volume can be extracted without dilution.

For the foregoing description of the gravity flow to be representative, the material must be discharged through a minimum size opening sufficiently large for the mass of solids to flow through the opening. In sublevel caving, the width of vertical discharge is formed by the width of the sublevel drift, which is greater than the width of a minimum size opening.

A greater opening width produces certain changes in the gravity flow that can be utilized for better ore extraction. These changes in the gravity flow pattern are clearly visible in the model shown in Fig. 20.2.21. To better demonstrate this effect, the model was constructed with an exaggerated opening width (in relation to the size of granular material). The central part of the gravity flow zone above the discharge opening moves downward more or less as a column; therefore, the black and white checkered pattern of the model filling is almost unchanged. This kind of gravity movement is called *mass flow*.

In sublevel caving, the opening at the level of the sublevel drift roof has the shape of a slot. Its theoretical length is equal to the roof width of the horizontal drift. In such a situation, the extraction zone (which has the same function as the extraction ellipsoid in a minimum size opening) no longer has the shape of an elongated ellipsoid of revolution, but forms an ellipsoid approximately similar to that shown in Fig. 20.2.22a. Simply, one can assume that this extraction ellipsoid consists of three parts as shown in Fig. 20.2.22b. Gravity flow consists of a partial mass flow (B) that only occurs on contact with the vertical plane of the sublevel front, while the remaining zones of the ellipsoid (A) undergo normal gravity flow. It means that mass flow occurs only in the zone that is marked as B in Fig. 20.2.22a and b.

It is clear that as the width of extraction opening increases, the width, and therefore the volume of the extraction zone, will increase. At the same time, a wide extraction opening will also result in an increase in the volume of undiluted extracted ore, because the outflow funnel will reach the extraction opening after the extraction of more undiluted ore than with a minimum size opening. This effect is shown in vertical section along the plane of the vertical front of the sublevel in Fig. 20.2.23. Here Fig. 20.2.23a represents the situation for a minimum size opening and Fig. 20.2.23b, the situation with a large width extraction opening.

It must be emphasized that the effective width of the extraction opening depends not only upon the width of the sublevel drift, but also upon the shape of the sublevel drift roof. When the roof is arched as shown in Fig. 20.2.24a, the slope of blasted ore will form a cone in the drift. Ore extraction from any place on the periphery of the cone toe will induce the flow of ore, which will follow the surface lines of the cone. Therefore, the cone will form a very narrow width outflow opening in the zone of the apex of the arched roof. Such a situation is unfavorable for sublevel caving because the effective extraction opening width will be very small. The gravity flow of blasted ore in the slice and flow of waste will have the pattern shown in Fig. 20.2.23a.

When the roof is horizontal (or slightly arched, see Fig. 20.2.24b), then the blasted ore will form a prism in the sublevel

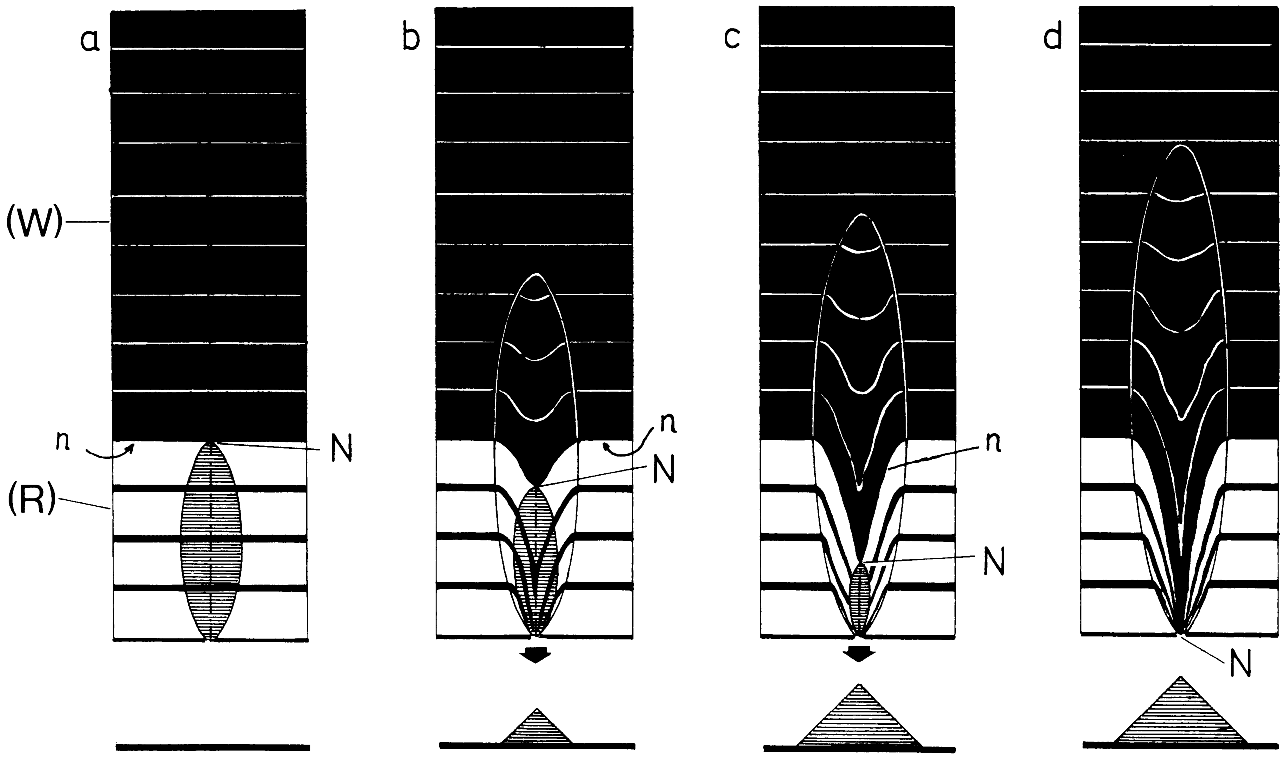

Fig. 20.2.14. Principle of gravity flow in an ellipsoid of extraction and loosening, identical to Fig. 20.2.13.

drift. Extraction from the slope toe of this prism will have a parallel flow on the slope, and therefore almost the total width of the sublevel drift can be utilized as an effective width of the extraction opening. Naturally, such a situation is favorable for sublevel caving because the effective width of extraction will be large and the gravity flow of ore (and waste) will have the pattern similar to Fig. 20.2.24b. Fig. 20.2.25 indicates the approximate effective extraction width as a percentage of the width of the sublevel drift w_d as a function of the roof shape.

Correct ore extraction demands not only a large extraction width, but also a satisfactory thickness or depth of the outflow zone. This depth depends upon how deep the loader can dig into the stope. Naturally, when the digging depth is small, the depth of the outflow zone is also small, and only a small part of the sublevel drift height is utilized for extraction.

According to Rankin's theory, the maximum stress trajectories in the slope of the bulk material are neither vertical nor perpendicular nor parallel to the slope inclination, but are inclined away from the vertical by $\beta = (90° - \phi)/2$ (see Fig. 20.2.26). In this figure, ϕ is the natural angle of repose, h_D is height of the sublevel drift, point 1 represents the theoretical slope toe, point 3 is the point at the edge that is formed by the intersection of the drift roof and sublevel front, point 2 represents the intersection of trajectories from point 3 with the floor of the sublevel drift, and X is the distance between points 1 and 2. With material extraction from the toe, the slope angle will approach the theoretical stable limit defined by the trajectory 3-2. This is the theoretical situation in which the slope 3-2 will have a stability coefficient equal to 1.0, which means that such a slope is just ready to fail.

Logically, in order to utilize the complete height h_D of sublevel drift for extraction, the digging depth must be

$$X = h_D \cot\phi - h_D \tan \frac{90° - \phi}{2} \tag{20.2.6}$$

The digging depth of a loading machine, usually about 3 to 4 ft (1 to 1.3 m), is much smaller than the theoretical depth X. This means that only a certain upper part (e) of the sublevel drift height will be utilized for normal extraction (Fig. 20.2.27). The remaining lower part, although not utilized for a normal extraction, also has a function: allowing the extraction of blocks with dimensions larger than the depth of the outflow zone on the level of the sublevel drift floor.

Even in normal extraction, the slope in the sublevel drift does not have a constant angle. The slope can vary between the limits, defined in Fig. 20.2.26 as points 1 and 2. It means the slope toe (point 1) can be close to point 2. In some cases, point 1 will be identical with point 2, and the slope will be very steep. The closer the slope is to the plane 3-2 (see Fig. 20.2.26) the more dangerous is its abrupt failure. Therefore, for safety reasons, it is useful to define the limiting angle of plane 3-2 and to induce the slope failure before this limiting condition occurs.

Extraction can be stopped by arching the material over the extraction zone. Most of the time, arching occurs over the outflow depth of the extraction zone (see Fig. 20.2.27) because this is the smallest dimension of the extraction opening. Although arching can have many variations, one can distinguish the following three basic types: (1) arching caused by a group of blocks that forms an arch structure, (2) arching caused by a compacted material (including the effect of apparent cohesion and/or plasticity), and (3) arching by a combination of these two.

When ore extraction is interrupted, the blasted ore (and caved waste) starts a process of settlement that can result in some compaction of the material. This occurs primarily when

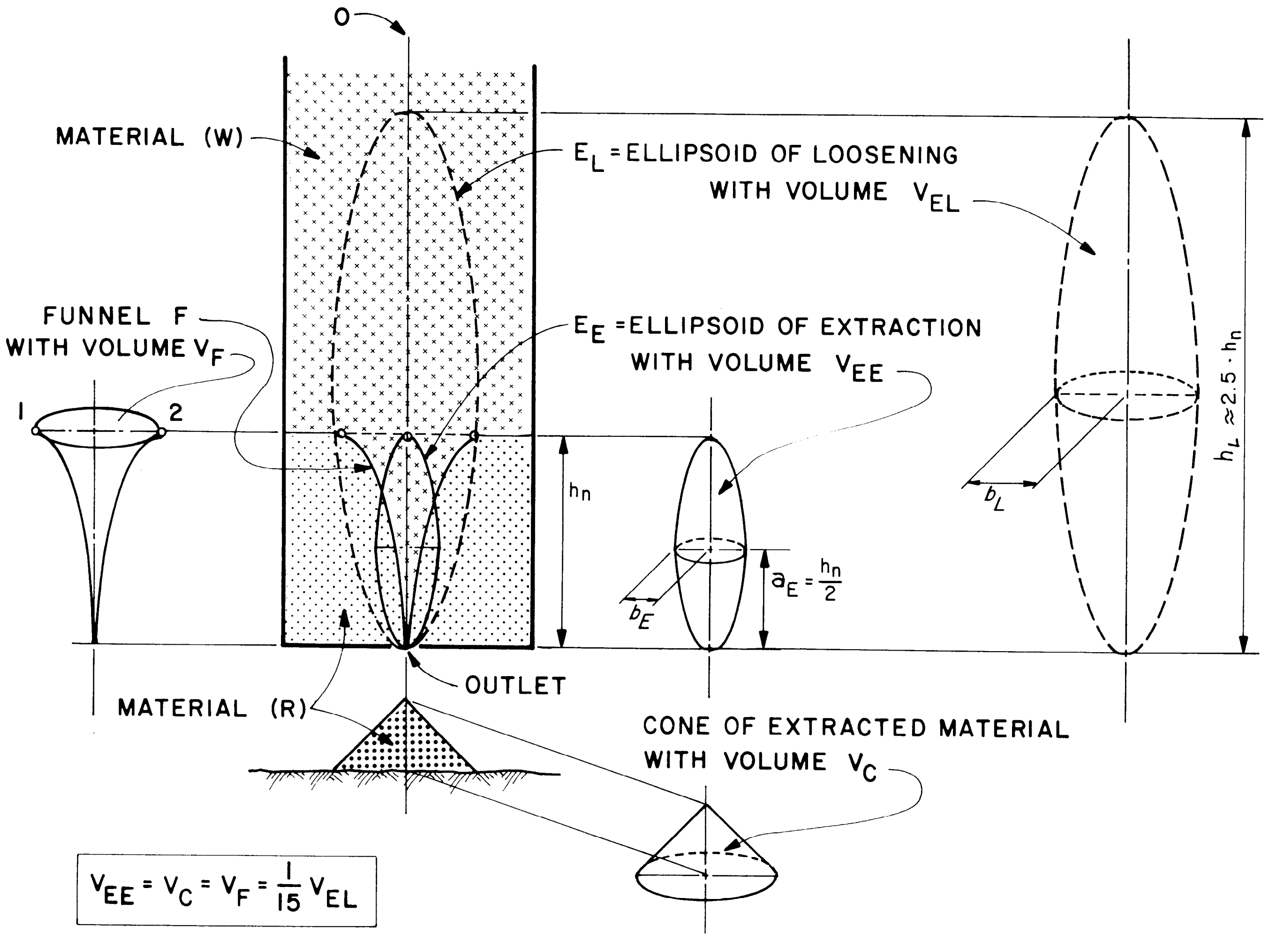

Fig. 20.2.15. Relationships in gravity flow mechanism.

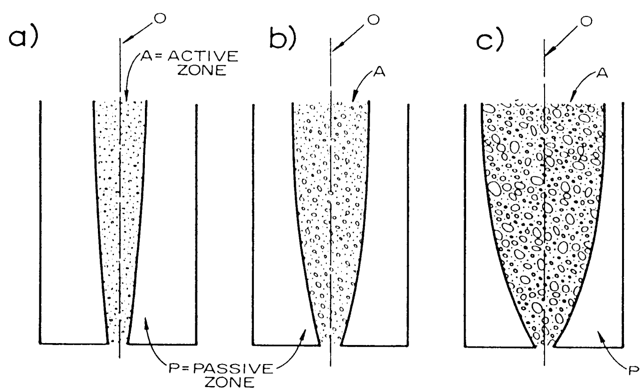

Fig. 20.2.16. Principle of particle or fragment size influence on shape of active (gravity flow) zone: a = fine material, b = granular material, c = coarse material.

plastic components and/or fine particles and water are present (see coarse material Type IV in Fig. 20.2.2). Arching (mainly of the second type) is more frequent after holidays or after any longer interruption of the flow of extraction muck. In materials with a relatively high rate of settlement and compaction, it is best to prevent this interruption. In practice, this means that total ore extraction from blasted slices of a sublevel should be completed before any long interruption.

The best method to avoid or minimize arching caused by large blocks is through controlled drilling and blasting, that is, to control the fragmentation.

Another question is the safe and efficient removal of the different types of arches. It is impossible to define a universal method that will satisfy all types of arching under varying conditions and circumstances. It can only be recommended that before any removal work commences, it is very important to understand the type, structure, and probable behavior of the arch and to define the critical zone or spot whose displacement (or excavation) will induce the failure of the arch. Otherwise, the work can be unsuccessful and even very dangerous.

As has been mentioned, the extraction ellipsoid (for a minimum-sized opening) does not have the shape of an elongated ellipsoid of revolution whose upper half is exactly identical to its lower half. Rather, the real shape has an upper half wider than the lower half (Fig. 20.2.28). For fine materials, the difference between the real and simplified shape is very small and can be considered negligible. The difference starts to be more pronounced in coarse materials and for extraction openings with widths greater than the minimum size. Lower mobility of coarse material, an apparent interlocking of the large fragments, together with partial mass flow in the central zone, results in a steepening of the lower part. Here the loosening and gravity

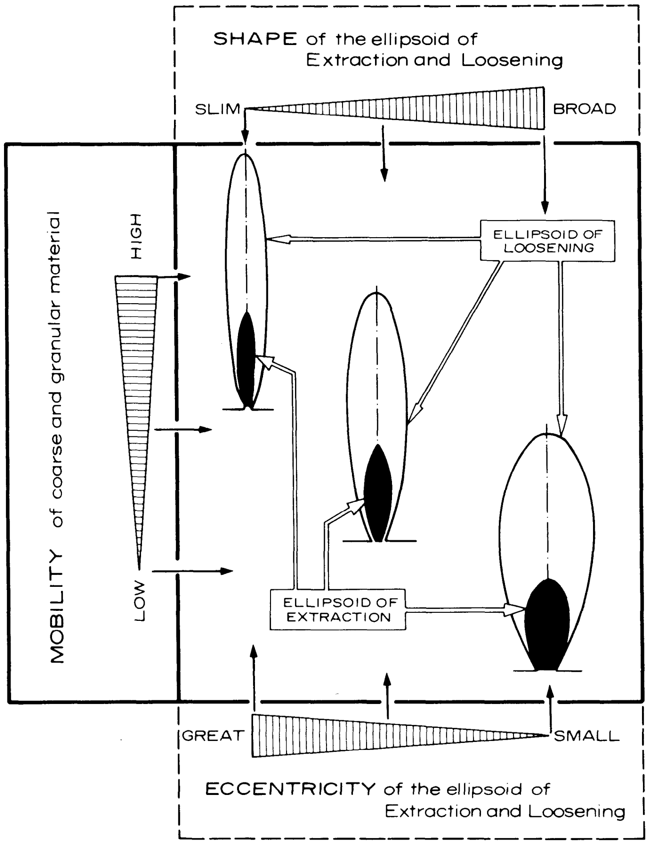

Fig. 20.2.17. Shape (and eccentricity) of ellipsoid of extraction and loosening as a function of material mobility.

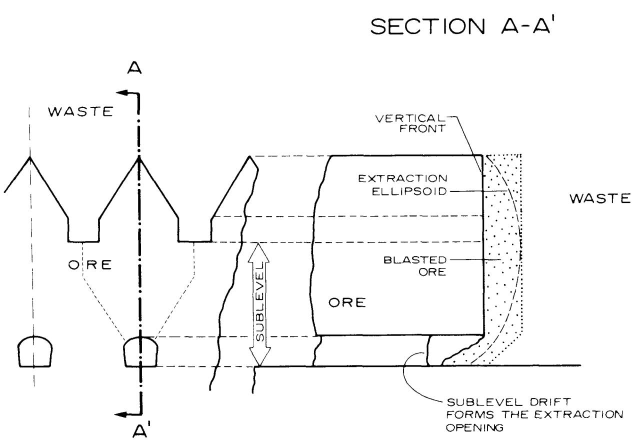

Fig. 20.2.18. Simplified geometry of traditional sublevel caving.

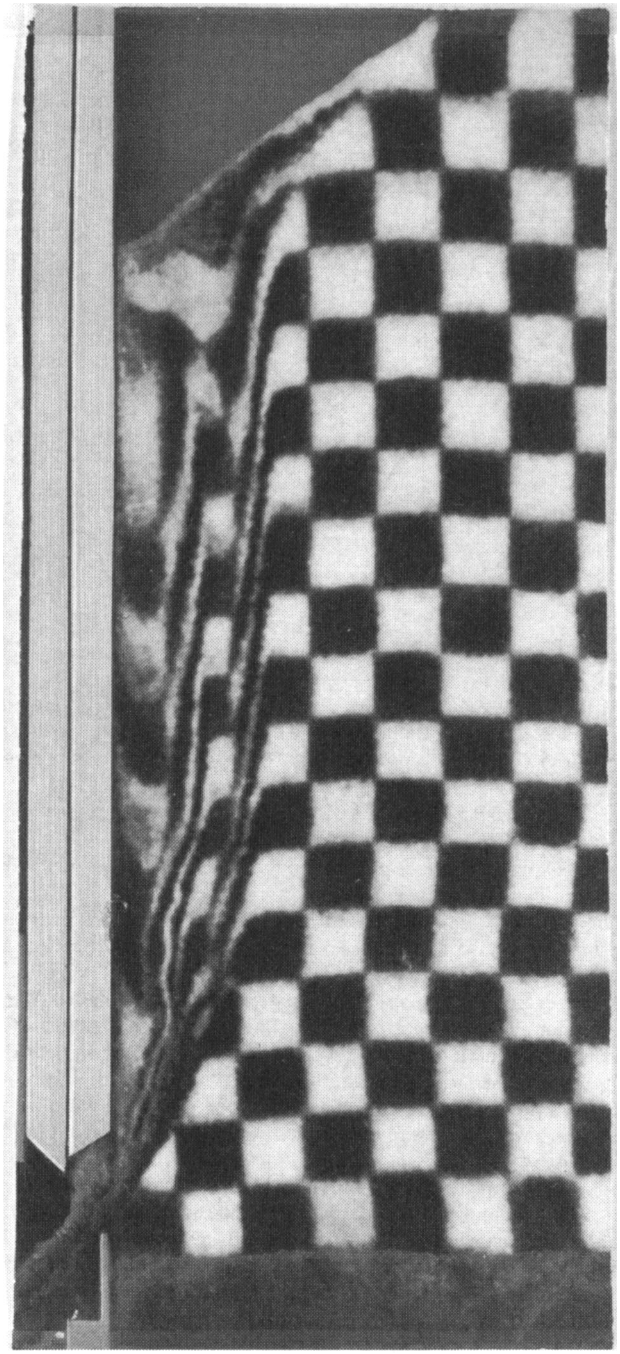

Fig. 20.2.19. Model of gravity flow when discharge opening is vertical, characteristic of sublevel caving (see vertical section A-A′ in Fig. 20.2.18).

motion is smoother because it is directed toward the extraction opening. The material in the upper part is remote from the opening, has lower loosening and lower mobility, and therefore needs a wider zone for gravity flow (see Fig. 20.2.29).

The change of shape is even more evident in sublevel caving because, in addition to the coarse and blocky material and greater width of the extraction opening, there are some additional factors due to ore blasting. In sublevel caving, the vertical slice of the ore is blasted against the caved waste. The final swell of the blasted ore varies over the height of the slice. Naturally, the greatest swell and loosening is in the lower part because a certain volume of the blasted ore falls and flows into the sublevel drift. This lower portion of the slice also has finer fragmentation due to the closer spacing of the drillholes in the fan drilling pattern. (Even with a parallel drilling pattern, the lower part of the blasted slice contains a higher percentage of small fragments and

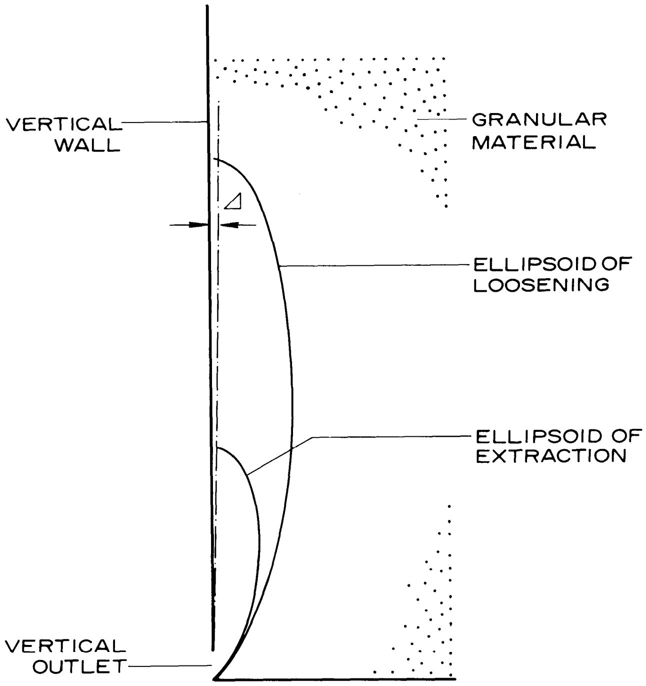

Fig. 20.2.20. Schema of ellipsoid of extraction and loosening when material is extracted through vertical outlet (sublevel drift) located in bottom of vertical wall. The vertical wall cuts off approximately half of both ellipsoids.

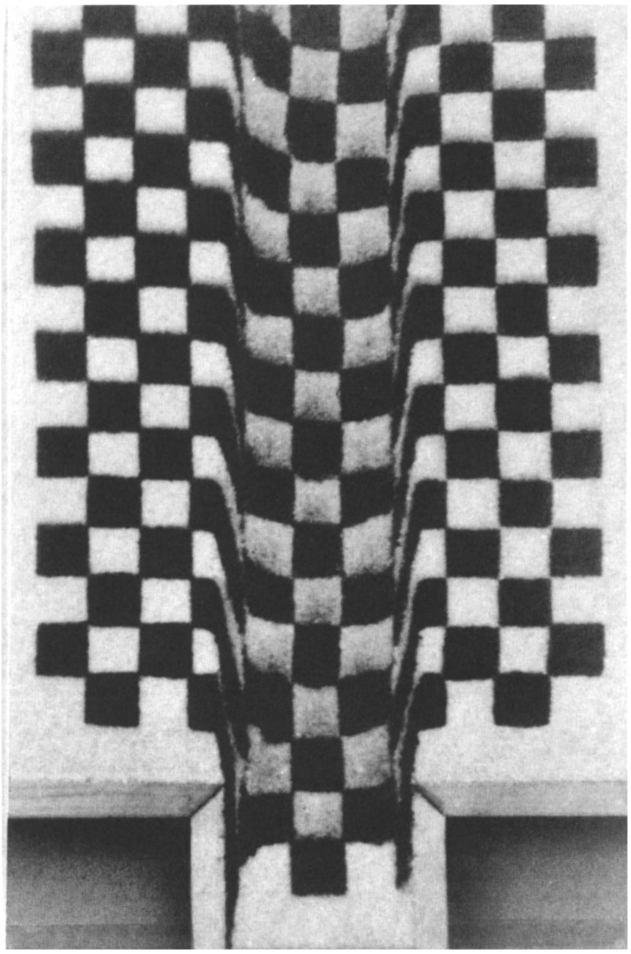

Fig. 20.2.21. Model of mass flow of material caused by exaggerated width of the extraction opening.

particles than the upper part, since they can penetrate downward into the cavities between the larger fragments.)

The swell and mobility of blasted ore in the slice decrease upward (away from the sublevel drift), and the fragments (and blocks) are bigger because of the larger spacing of drillholes in the upper part of the fan drilling pattern. In the upper parts, the confinement of the blasted ore in the slice is higher than in the lower part, which certainly results in a lateral prestressing of the blasted ore. An additional factor that can decrease the mobility of the upper parts is the penetration of a certain volume of overlying waste between the blasted slice and the ore front at the moment of blasting when, for a short while, the blasted slice is separated from the sublevel ore front. This can also be combined with a certain back break in the remaining ore front of the sublevel. It can be said that the blasted ore in the slice has the highest mobility in the lower part and the mobility decreases in the upward direction. The smaller the mobility, the greater the width necessary for gravity flow.

The extraction ellipsoid will be slimmer in the lower section with its maximum width being realized somewhat above half of its height, as is shown in Fig. 20.2.30a.

The shape of the extraction zone (in vertical section A-A′, perpendicular to the sublevel front) is constructed in Fig. 20.2.30b. Because the surface of the sublevel ore front is very rough, the friction along this plane is high, and the ellipsoid axis will deviate from vertical. The deviation *e* increases with increasing friction. Determination of this deviation is difficult, but it can be assumed that at a sublevel height of about 37 ft (11 m), the horizontal distance between the apex of the extraction ellipsoid and the vertical plane representing the sublevel front can be about 3 ft (1 m) or even more, depending on sublevel height.

Blasted ore in the slice usually represents a very heterogeneous material. Even with perfect drilling and blasting, the variations induced by so-called blasting tectonics can result in the creation of zones with large fragments or zones with fine material. Big blocks of ore (or waste) occur at random, mainly in the upper part of the slice. The migration of these large blocks or even pockets of fine ore (or waste) will cause disturbances in the gravity flow. Many other disturbing factors can also enter, for example, interruption of ore extraction from the slice, that can result in settlement and compaction of the blasted ore; discontinuous failure and caving of waste rock masses that can change the stresses acting on the slice of blasted ore; progressive failure and caving of other kinds of rock types at depth that can produce another type of fragmentation of caved waste than experienced in the upper zones; and increased size of the progressive failure zone and increased area of subsidence leading to the collection of water, that can cause undesirable changes in the properties and behavior of the blasted ore and therefore difficulties in ore extraction, etc. It is evident that because of these factors, the extraction ellipsoid in sublevel caving is not regular as might be expected with a homogeneous coarse material extracted without these disturbing influences.

20.2.2.4 Laboratory Studies and In Situ Testing

Research directed toward understanding the shape of the extraction ellipsoid in sublevel caving has been executed both on

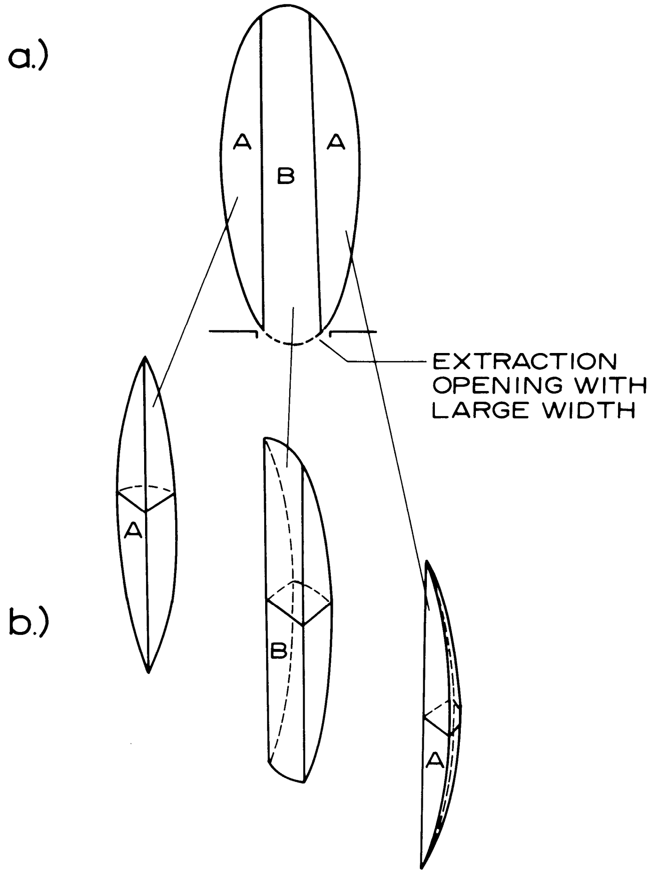

Fig. 20.2.22. Simplified shape of extraction zone created by sublevel caving and caused by large extraction width of the sublevel drift.

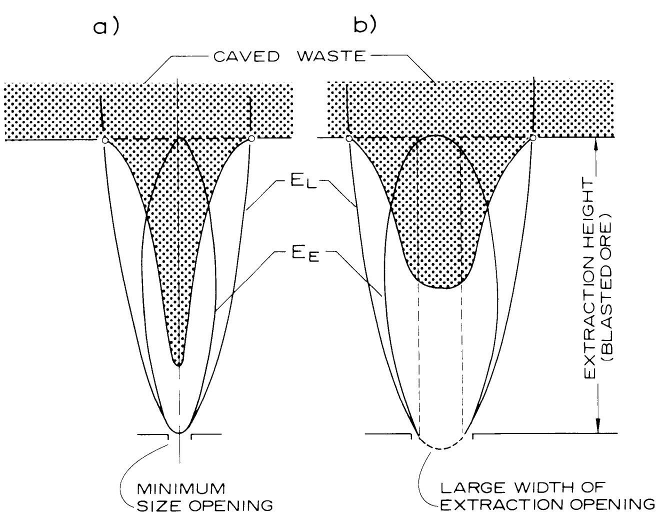

Fig. 20.2.23. Influence of extraction width on gravity flow pattern: a = small opening width, and b = large opening width.

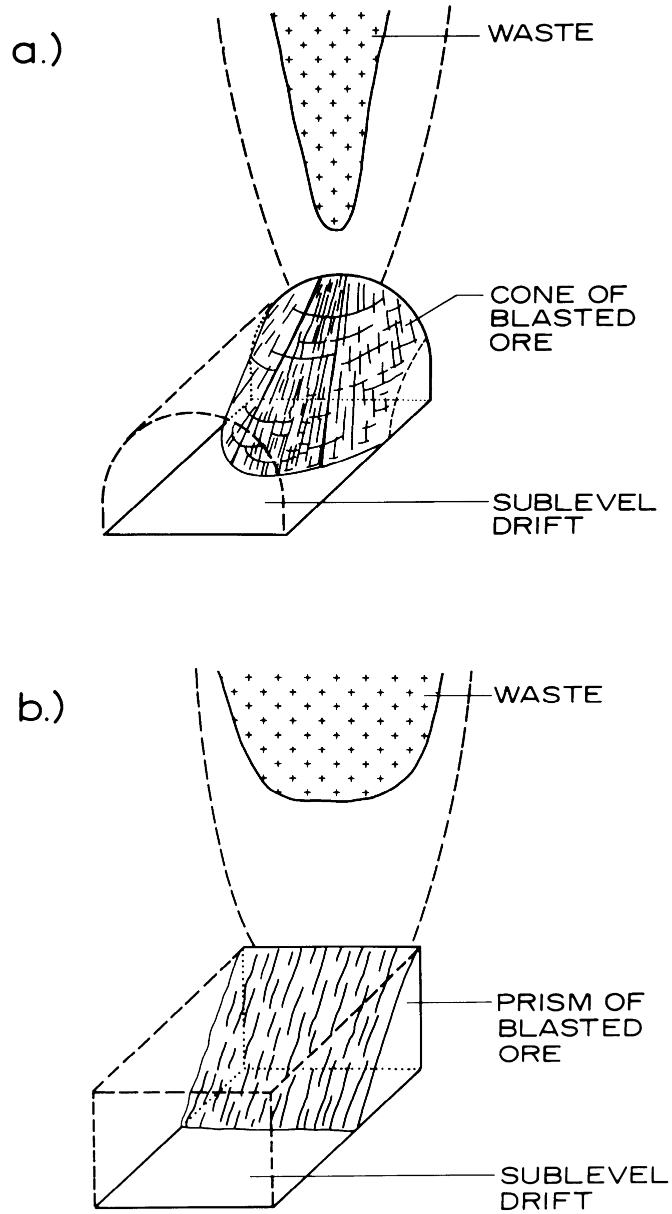

Fig. 20.2.24. Gravity flow pattern as a function of the shape of sublevel drift roof: a = arched roof, and b = flat roof.

models in the laboratory and full-scale in situ testing. Each of these approaches has certain advantages and disadvantages. In the laboratory, even very sophisticated models cannot simulate natural conditions correctly. In the field, two types of testing may be necessary.

The first type of in situ testing is based on the utilization of a large number of radioactive markers, located in drillholes in the ore slice before blasting and fixed into position by cement grouting. After blasting, the markers are dispersed in the rubble, and their exact location is not known. The second type is based upon the location of markers in the fragmented ore, that is, in the blasted ore of the slice. This method is difficult, laborious, and very expensive, and therefore only a few markers can be safely installed in blasted ore.

Analysis of laboratory experiments and of in situ tests suggests two shapes for the extraction ellipsoids in sublevel caving. The generalized geometry of both shape types is shown in Fig.

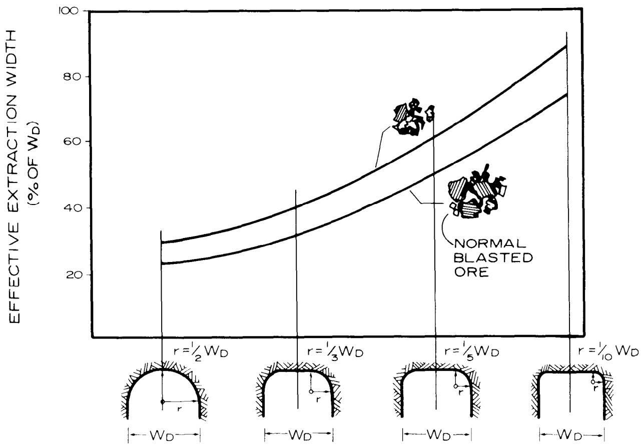

Fig. 20.2.25. Approximate effective extraction width as a function of the sublevel drift roof shape, expressed in a percentage of the sublevel drift width.

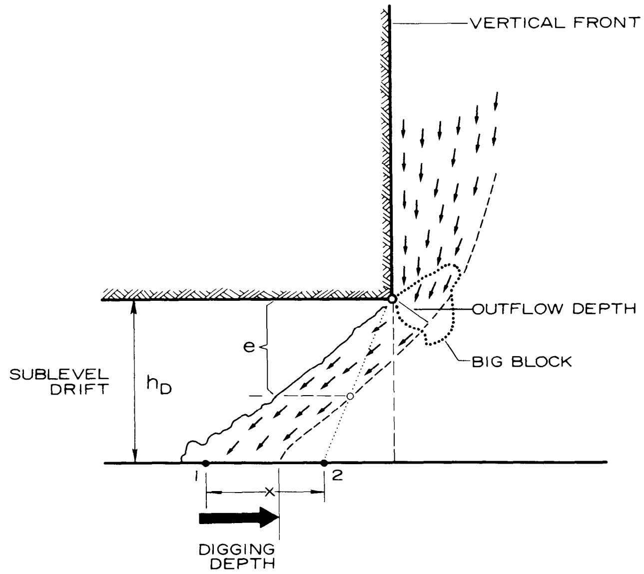

Fig. 20.2.27. Approximate outflow depth into the sublevel drift as a function of digging depth.

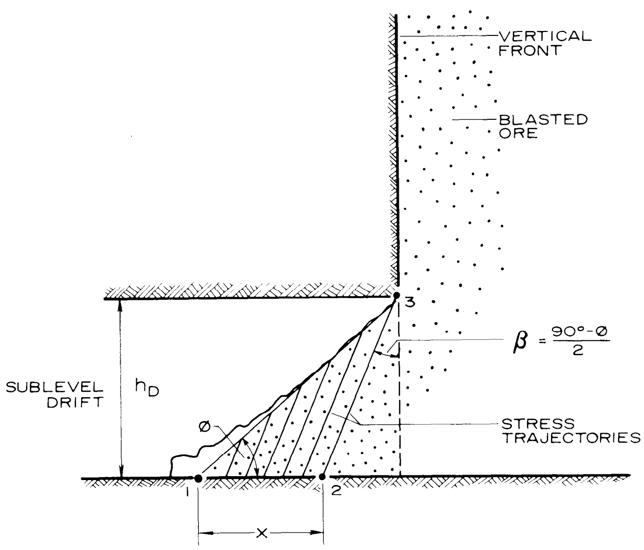

Fig. 20.2.26. Mechanics of the slope from fragmented material in the sublevel drift.

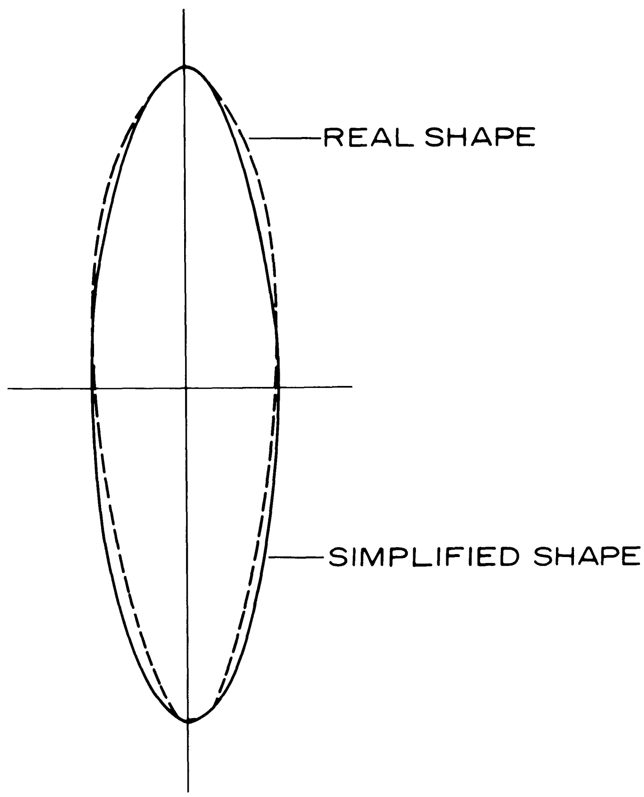

Fig. 20.2.28. Principle of the difference between simplified (idealized) and actual shapes of ellipsoid of extraction in a uniform granular material having a minimum size extraction opening.

20.2.31. Although the differences are relatively small, it is possible to distinguish that type A has a more rounded shape while type B is more angular. The cause of a type-A or a type-B shape is still questionable since the data from models (with limited precision) and from in situ tests (small number of tests) are still insufficient for explicit solution.

In both shapes, the maximum width W_T of the extraction ellipsoid is about 2/3h above the floor of the sublevel drift, where h is extraction height (see Fig. 20.2.31). The maximum depth d_T (ore thickness) of the extraction ellipsoid perpendicular to the sublevel front also occurs at the same height. Because of the large width of the extraction opening, the structure of the extraction ellipsoid is analogous in principle to that shown in Fig. 20.2.22.

For these reasons (disturbance factors and difficulties in testing), the exact shape of the extraction ellipsoid in sublevel caving is still not explicitly defined. As shown in Fig. 20.2.32, which represents the full-scale in situ test at Grängesberg, the shape of the extraction ellipsoids is very irregular depending on acting disturbances. This shape changes with the extraction height, that is, with a volume of extracted material. Even so, the contour shape is very similar to the type A shown in Fig. 20.2.31. Although knowledge of the shape is sufficient for engineering

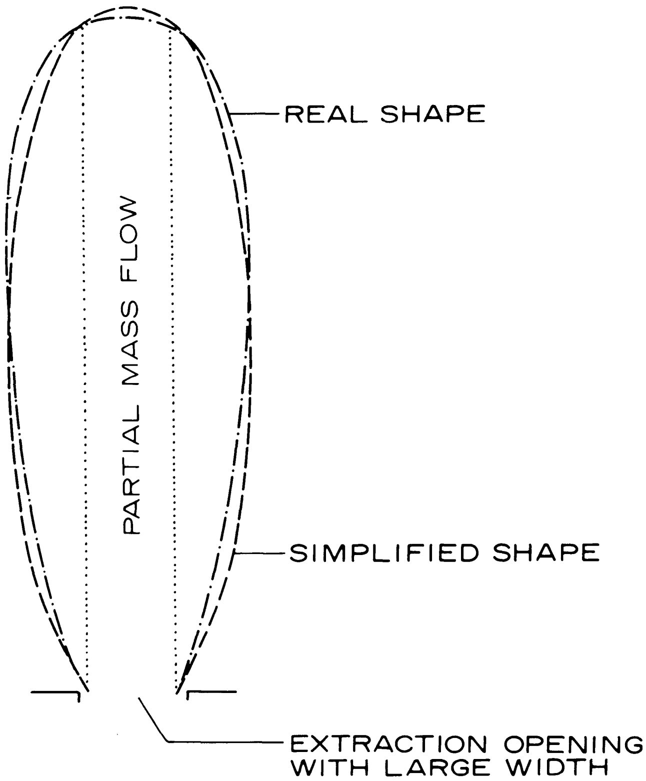

Fig. 20.2.29. Simplified and actual shape of an ellipsoid in granular material having a large-width extraction opening.

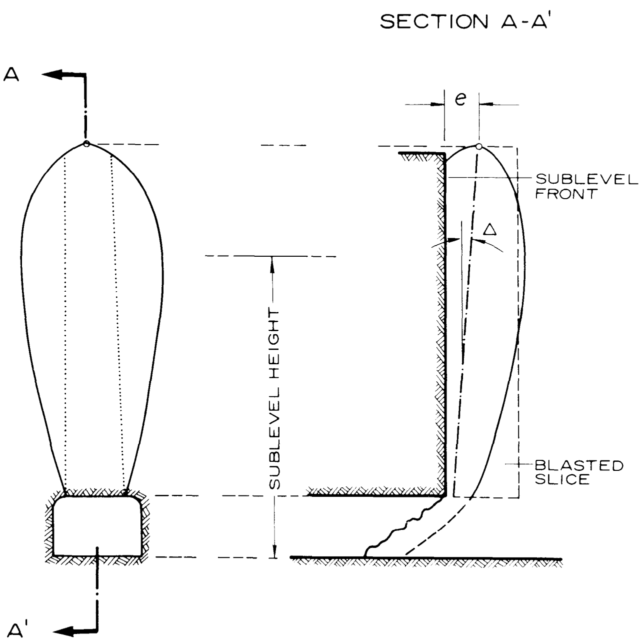

Fig. 20.2.30. Principle of actual shape of extraction zone in sublevel caving.

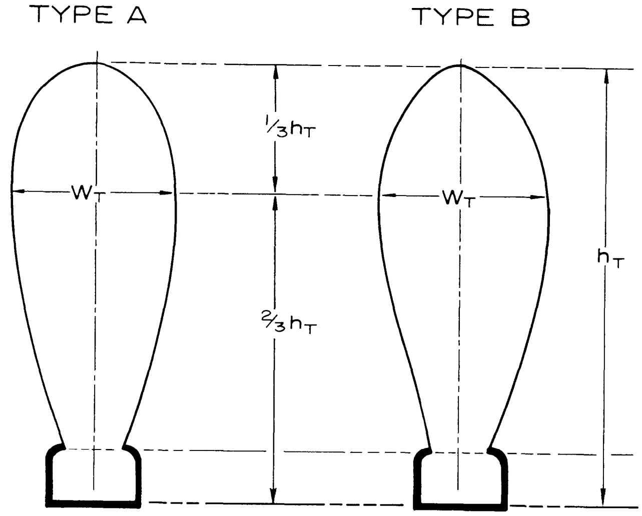

Fig. 20.2.31. Generalized geometry (type A and B) of extraction zone.

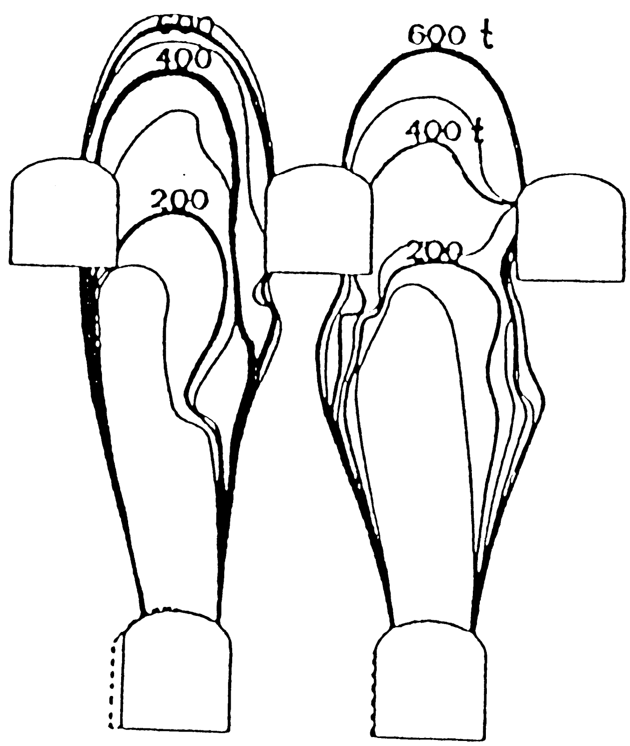

Fig. 20.2.32. Full-scale test of sublevel caving at Grängesberg mine (Janelid, 1973). Conversion factor: 1 ton = 0.9072 t.

practice, it is still insufficiently precise for the development of an explicit theory.

20.2.3 PRACTICAL DESIGN GUIDANCE

The main question in the design of a sublevel caving mining method is the determination of a mining geometry that will satisfy as much as possible the parameters of gravity flow. This means that one needs first to determine the width and thickness of the extraction ellipsoid for a certain extraction height.

These parameters can be determined by in situ tests, but usually test data are not available in time for the mine design.

Presently, no explicit method for making engineering calculations of these parameters is available. Because of the heterogeneity of coarse materials and the complexity of factors involved in gravity flow, certain empirical formulas will be introduced here that can serve as guidelines for the approximate determination of gravity flow parameters and for the geometry of the method.

20.2.2.1 Dimensions of the Extraction Ellipsoid

Coarse material is actually a mixture of different-sized particles. A small proportion of fine particles and fragments decreases, often markedly, the effect of the coarse fragments. As a result, the gravity flow zone in coarse material is sometimes surprisingly narrow.

Since the eccentricity of the ellipsoid increases with its height, the greater the height, the slimmer the flow. (This is very well known in block caving. With large block heights, the gravity flow above a single drawpoint can form a chimney with vertical walls.) With the same fragmentation, the gravity flow of a high-density material (i.e., blasted iron ore) will be slimmer than the flow of a low-density material (i.e., blasted copper ore). The width of the gravity flow depends also upon the size of the extraction opening. (See also Figs. 20.2.22 through 20.2.25 and 20.2.29.)

To exclude the variable factor of different extraction opening sizes, the data from analytical and model research, in situ tests, and observations in sublevel caving operations were used to determine an approximate theoretical width W' of the extraction ellipsoid, assuming extraction of material through a minimum size opening. Assuming a normal fragmentation of the blasted ore, the theoretical minimum size opening is about 6 ft (1.8 m). For high-density blasted ore (iron ore), the approximate theoretical width W' of the extraction ellipsoid is shown in Fig. 20.2.33 as a function of the extraction height h_T. In traditional sublevel caving, the total extraction height h_T in the ore is normally between 70 ft (20 m) and about 100 ft (30 m). The theoretical width W' of the extraction ellipsoid, corresponding to the different total extraction heights h_T within these limits, is shown in Fig. 20.2.24 for low- and high-density blasted ore. The effective extraction width a (see also Fig. 20.2.25) is usually larger than the width of a minimum size opening (6 ft or 1.8 m), and therefore, the total width W_T of the extraction ellipsoid in a sublevel caving operation will be bigger than that shown in Fig. 20.2.34.

A very approximate value of the total width W_T and total depth d_T of the extraction ellipsoid for a given height h_T can be calculated (in meters) using the following empirical formulas:

$$W_T \approx W' + a - 1.8 \tag{20.2.7}$$

$$d_T \leq W_T/2 \tag{20.2.8}$$

where W' is total extraction height h_T (see Fig. 20.2.34), and a is the effective width of the extraction opening dependent upon the shape of the sublevel drift roof (see Fig. 20.2.29, expressed as a percentage of W_D.

20.2.3.2 Vertical Spacing of Sublevel Drifts

The sublevel caving extraction drifts should be located in a checkered pattern conforming to the pattern of gravity flow.

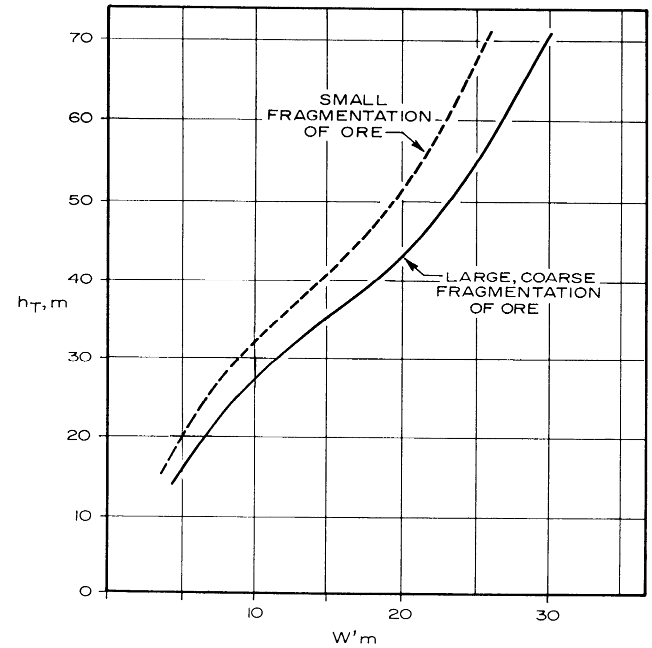

Fig. 20.2.33. Theoretical approximate width W' of a very large extraction ellipsoid for high density ore as a function of its height.

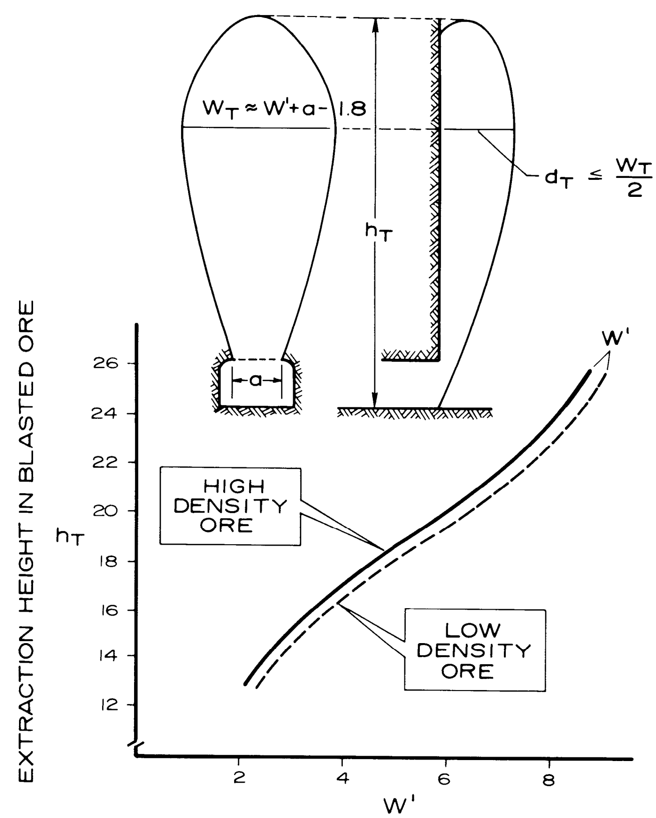

Fig. 20.2.34. Approximate width of the extraction ellipsoid (in traditional sublevel caving) in high- and low-density ore.

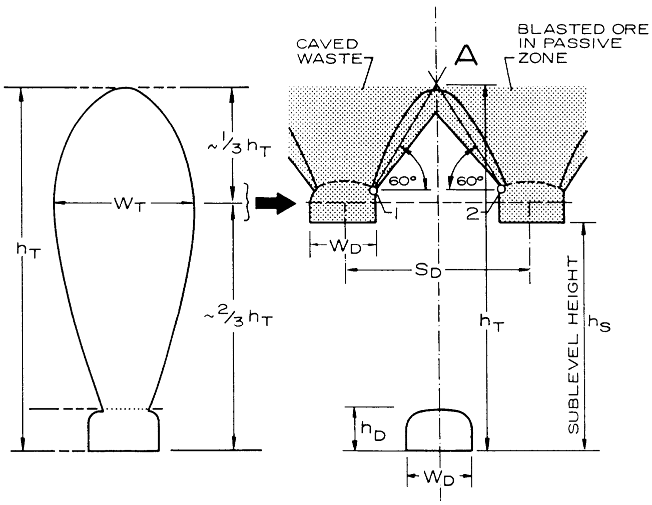

Fig. 20.2.35. Vertical location of sublevel drifts conforming to pattern of gravity flow.

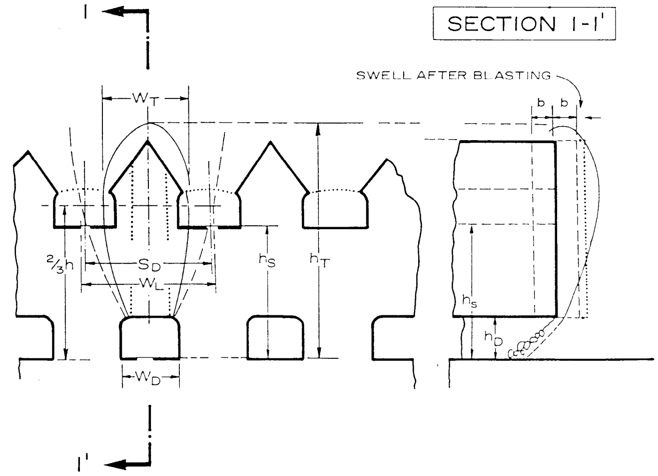

Fig. 20.2.36. Simplified sublevel caving geometry.

Using traditional sublevel caving methods, in vertical locations (see Fig. 20.2.35), sublevel drifts should be located in the zone where the extraction ellipsoid has a maximum width W_T. This occurs at about $2/3h_T$. In principle, this location indicates approximately the height h_S of the sublevel (see Fig. 20.2.35).

After ore extraction, an intact (not blasted) ore pillar having a triangular shape (transverse section) formed by the limiting inclined drillholes of the fan drilling pattern remains between the sublevel drifts on the upper level. This pillar is usually covered by blasted ore remaining in a passive zone above the pillar after ore extraction. The thickness of this passive zone of blasted ore can be large or small depending upon the inclination of the limiting drillholes in the fan drilling, the width W_D of sublevel drifts, the spacing S_D of sublevel drifts (see Fig. 20.2.35), and the properties and gravity flow parameters of the coarse material that control the shape of the passive zone.

The blasted ore in the passive zone can be partially recovered by extraction on the lower sublevel. Therefore, the total extraction height h_T is the distance between the floor of the lower sublevel drift and the apex A, formed by the remaining blasted ore in the passive zone (see Fig. 20.2.35). Because it is difficult to define the shape and dimensions of this passive zone, one must usually estimate the total extraction height h_T using experience from sublevel caving operations with similar conditions. As a rule of thumb, the lowest location of the apex A can be very approximately defined as the intersection of 1.05 rad (60°) inclined planes from points 1 and 2, as shown in Fig. 20.2.35. The minimum inclination of these planes is 60°. The highest location of this apex is defined by the floor of the upper sublevel drifts. The height depends greatly on the drilling design.

20.2.3.3 Horizontal Spacing of Sublevel Drifts

An approximate horizontal spacing S_D of the sublevel drift axes can be determined knowing H_T and W_T. This determination is based upon utilizing the idealized relations between the ellipsoids of extraction and loosening described in Fig. 20.2.15. Assuming similarity of eccentricity of ellipsoids and that the loosening ellipsoid is 2.5 times bigger than the ellipsoid of extraction, the width of the extraction ellipsoid is 40% of the width of the appropriate ellipsoid of loosening. (For these idealized conditions, one can calculate the width of the ellipsoid of loosening on any horizontal section of extraction ellipsoid.)

In sublevel caving, one needs to determine the width W_L of the loosening ellipsoid on a horizontal section just at the level where the extraction ellipsoid has its maximum width W_T (Fig. 20.2.36). The width of the loosening ellipsoid on this level indicates the approximate horizontal spacing S_P of the sublevel drifts.

Assuming that the relations and principles of idealized gravity flow (depicted in Fig. 20.2.15) can be applied to sublevel caving, the total width W_T of the extraction ellipsoid in sublevel caving is about 60 to 65% of the width of the loosening ellipsoid on the level where the extraction ellipsoid has its maximum width W_T. The width is about 60% for heights up to about h_T = 60 ft (18 m). Above that the width W_T is about 65%. Hence the approximate horizontal spacing S_D of sublevel drifts is

(1) For extraction heights $h_S \leq 60$ ft (18 m):

$$S_D < \frac{W_T}{0.6} \tag{20.2.10a}$$

(2) For extraction heights $h_S > 60$ ft (18 m):

$$S_D < \frac{W_T}{0.65} \tag{20.2.10b}$$

The basic geometric unit of sublevel caving is in principle defined by the relation between the horizontal spacing S_D of sublevel drifts and vertical height of the sublevel h_S. In conventional sublevel caving (see Fig. 20.2.36), this relation is

$$S_D \leq h_S \tag{20.2.11}$$

which means that the basic geometric unit has a square shape, or deviates only slightly from square.

Improved precision of longhole drilling in recent years has resulted in a tendency to increase the height of the sublevels with the goal of decreasing the volume of development. Ore draw can continue over heights greater than the extraction height h_T in the ore, but the dilution then rapidly increases. Large-dimension sublevel caving is discussed later in this chapter.

20.2.3.4 Thickness of the Blasted Slice

An approximate guide for the thickness of the blasted slice b (burden spacing) on the front of the sublevel is usually

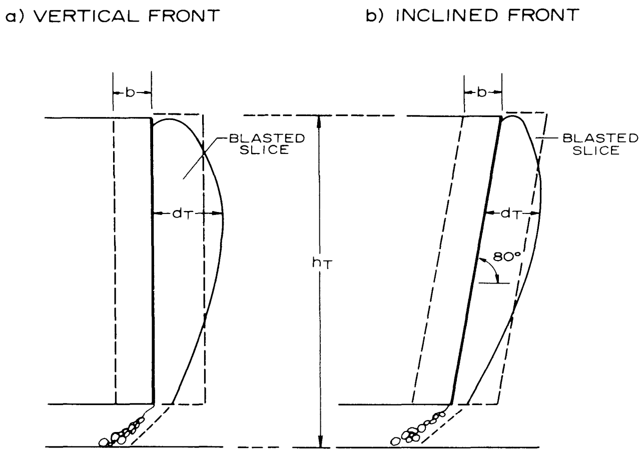

Fig. 20.2.37. Vertical (a) and inclined (b) front of sublevel caving. Inclined front decreases dilution.

$$b \leq \frac{d_T}{2} \qquad (20.2.12)$$

where d_T is found from Eq. 20.28. This relationship can be used as an estimate for traditional sublevel caving. For large-dimension sublevel caving, made possible by new drilling equipment having the capability to drill very long drillholes (13 to 20 ft, or 40 to 60 m) with satisfactory precision, different relationships are valid, because the eccentricity of the flow ellipsoid rapidly increases with its height.

20.2.3.5 Inclination of the Front

The sublevel front is usually vertical or inclined at 1.4 rad (80°). This inclination is favorable not only for drilling and charging of drillholes but also for minimizing the dilution. The effect of the sublevel front inclination is evident as shown in Fig. 20.2.37, where Fig. 20.2.37a has a vertical front and Fig. 20.2.37b has an inclined front. The extraction height h_T is the same in both cases.

With a vertical front (Fig. 20.2.37a), the extraction ellipsoid intrudes deeper into the caved waste rock. This is particularly true when the sublevel is high.

The inclination 1.4 rad (80°) of the front shown in Fig. 20.2.37b will cause a change of the gravity flow shape, and the extraction ellipsoid will be much slimmer than with a vertical front. It can be simply said that in this case, the extraction ellipsoid has a tendency to be inscribed within the blasted ore slice. The intrusion of the extraction ellipsoid into the caved waste is smaller, and therefore, the dilution, with ore extraction over the same extraction height h_T, will be smaller than with a vertical front.

20.2.3.6 Extraction and Dilution

The dilution process in sublevel caving can take many different anomalous and unexpected forms; for example, the outflow of a certain quantity of caved waste-rock almost at the beginning of ore extraction that is then followed by normal ore flow; or the random occurrence of waste rock pockets at different stages of the ore extraction, etc. The cause of such phenomena is still not defined satisfactorily, although big blocks are surely of great importance. In ore which tends to arch over the extraction opening, one can expect a higher probability of anomalous dilution.

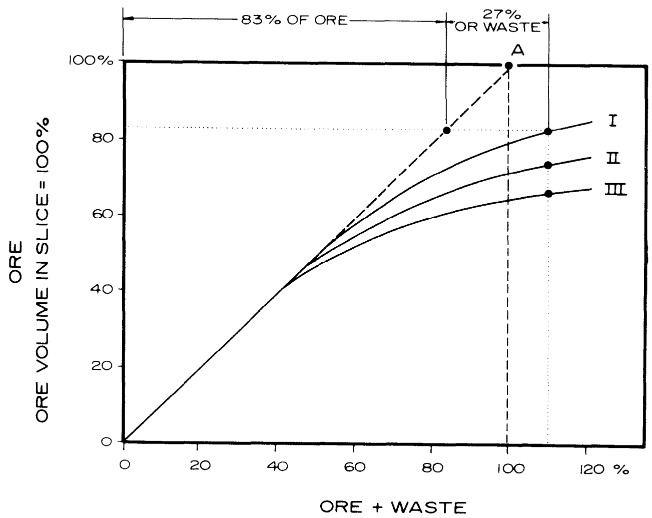

Fig. 20.2.38. Simplified process of dilution development in sublevel caving. Curve I represents good extraction, and Curve III represents bad extraction with high dilution.

The ideal process of normal dilution development in sublevel caving is illustrated in Fig. 20.2.38 as a function of the volume of the ore and the volume of extracted material (consisting both of ore and caved waste). It is expressed as a percentage of the volume of ore slice where the total volume of the ore slice is 100%. Theoretically, the best extraction will be defined by the line OA. This means one will extract 100% of the ore without any waste. Such a situation is not possible in practice.

Fig. 20.2.38 shows three different cases of ore extraction with different dilutions, that is, with different volumes of waste extracted together with the ore. The optimum total extraction (ore and waste) can have different values depending upon the value of the ore and the overall economics. In Fig. 20.2.38, it is assumed that extraction will stop after drawing 110% of the slice volume.

Curve I in Fig. 20.2.38 represents good extraction because ore recovery is 83% with only 27% waste.

Curve II demonstrates a case in which the distribution is about 75% ore and 37% waste for a total extraction of 110%. Of course, this is not as good as the previous example, but in certain conditions, primarily in cases when a simple and cheap process of waste separation is applicable, it can be acceptable.

Curve III characterizes a bad extraction. In this case, for a draw of 110% total material, only 65% of ore with waste as high as 45% is extracted.

As a guide for evaluating the extraction (assuming 110% of material [ore and waste] extraction), the following classification can be used:

1. Class I: Extraction is good as the ore is at least 80% and the waste is less than 30%.
2. Class II: Extraction is acceptable in certain cases as the ore is at least 75% and the waste is no more than about 35%.
3. Class III: Extraction is poor as the ore constitutes about 65% and the waste 45% or more.

Total dilution development is already defined at the beginning of the dilution process. Approximate relations are evident from the chart in Fig. 20.2.39. For example, if after a material extraction of 71%, the ratio is 60% ore and 11% waste (see point A in Fig. 20.2.33), one can expect that with 120% material extraction (see point B in Fig. 20.2.39), the ratio will be about

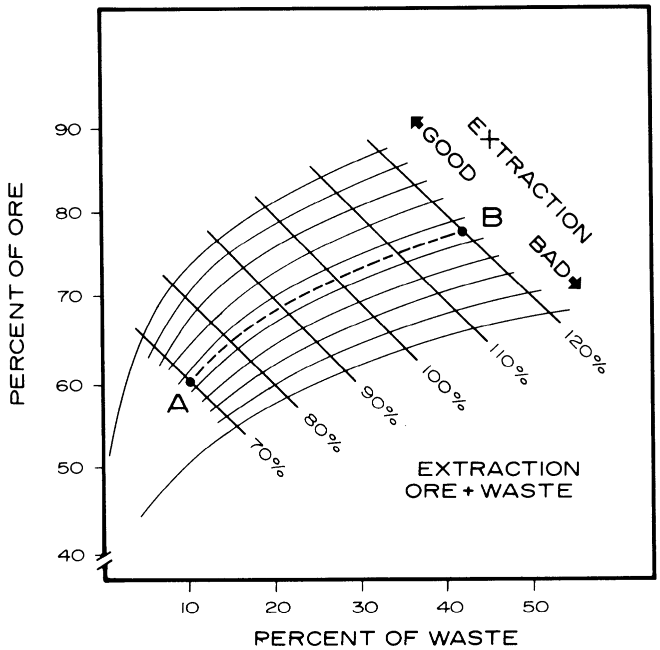

Fig. 20.2.39. Simplified chart of dilution development.

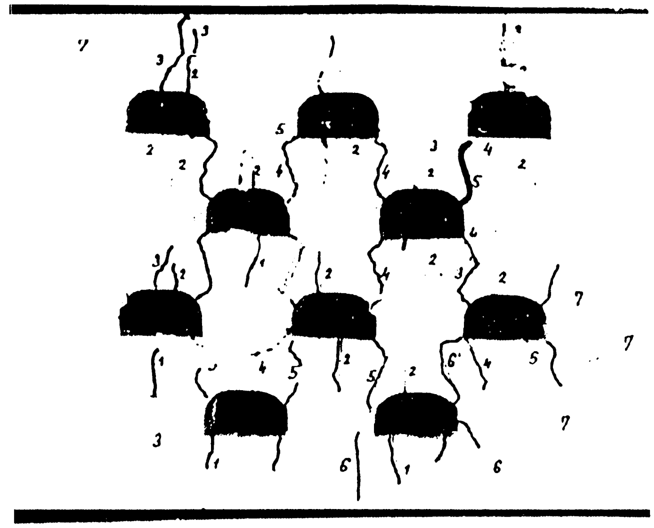

Fig. 20.2.40. Plaster model showing characteristic failure of a "checkered" sublevel drift structure. Failure is caused by shear cracks between the closest corners of overlying sublevel drifts.

77% ore and at least 43% waste. The limit between good and bad extraction can be different in different ores with different ore grades. The approximate boundary between good and bad extraction is indicated in Fig. 20.2.39 by the line A-B.

20.2.3.7 Opening Stability

The stability of the opening of the sublevel caving structure is usually very good, and so this method can be safely utilized even in relatively soft ores and rock masses. Nevertheless, it is necessary to understand that the checkered pattern of sublevel drifts generates a special stress distribution that determines the stability limits of the structure. This checkered pattern of drifts results in high shear stress concentrations in the rock lying between the closest corners of drifts on the upper and lower levels.

Under the same loading conditions, the stress concentration increases with decreasing sublevel height h_S and drift spacing S_P and with increasing sublevel drift width W_P and height h_P (see Fig. 20.2.36). Under critical loading, the failure of the sublevel structure is characterized by shear cracks between the closest corners of sublevel drifts on upper and lower levels. This characteristic pattern of cracks is shown on a plaster model in Fig. 20.2.40. Detailed stress analysis can be provided using finite element or other methods of analysis.

The location and direction of failure shear cracks follow the zones of maximum shear stress concentration. This is shown in Fig. 20.2.41 in which the dotted lines represent the isochromatic lines obtained on a photoelastic model with the same geometry and with the same kind of loading as the plaster model in Fig. 20.2.40. The higher the order of the isochromatic line (marked in Fig. 20.2.41 with numbers 1, 2, 3, etc.), the higher is the maximum shear stress. Therefore, from Fig. 20.2.41, it is evident that the shear cracks correspond to the maximum shear stress concentration. It is necessary to emphasize that the introduction of sublevel caving at depth requires a careful stress and stability analysis for definition of a safe geometry.

With regard to the checkered location and relatively short distance between sublevel drifts, damage to the rock mass re-

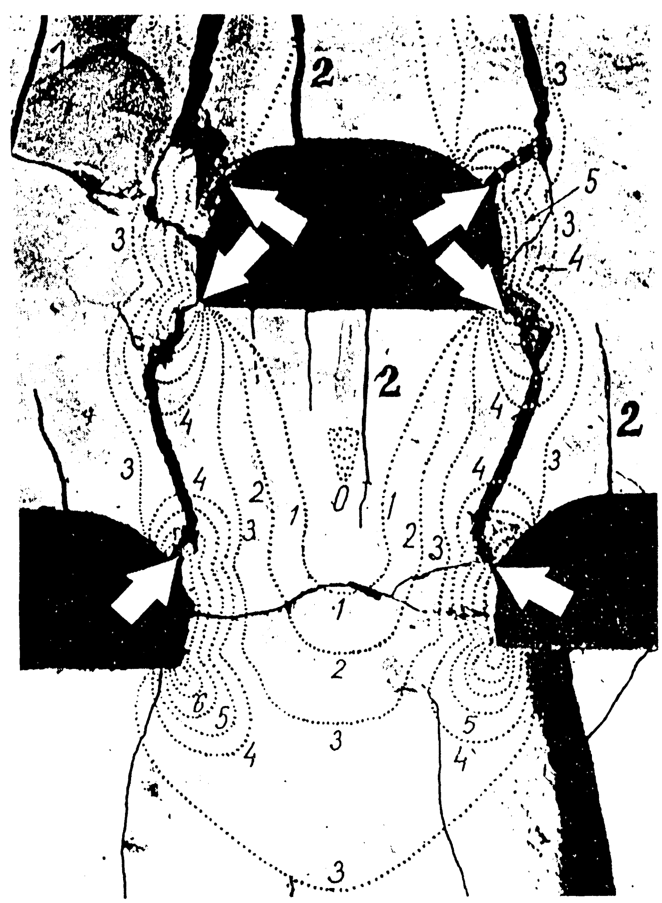

Fig. 20.2.41. Detail of plaster model shown in Fig. 20.2.40, completed by dotted isochromatic lines obtained on a photoelastic model. Main shear cracks on the plaster model correspond with the zones of maximum shear concentration visualized by the highest density (and order) of isochromatic lines.

sulting from poor drifting practice (drilling and blasting precision of drift geometry and location) is very undesirable for good stability. Therefore, correct geometry and relative location of drifts and good smooth blasting is very necessary, particularly in soft ores and rocks and at greater mining depths. The precision of the correct geometry, smooth contour of drifts and exact location of drifts is extremely important when using large-dimension sublevel caving.

Example 20.2.1. This involves the determination of an approximate geometry for traditional sublevel caving, given the following information:

Width of the sublevel drift W_D = 5 m (17 ft)
Height of the sublevel drift h_D = 3.5 m (12 ft)
Blasted ore has a high density
Shape of the drifts: flat roof

Solution. To determine the parameters and geometry: Total extraction height is estimated to be h_T = 21 m. Shape of the sublevel drift with flat roof is favorable for ore extraction, and the effective extraction width of the sublevel drift (according to Fig. 20.2.25) is about 70% of W_D, i.e., $a = W_D \times 0.7 = 5 \times 0.7 = 3.5$ m.

With respect to the location of sublevel drifts in the vertical direction (see Figs. 20.2.35 and 20.2.42), the approximate sublevel height is h_s = 12.5 m. In accordance with Fig. 20.2.35, the sublevel drift is located at about ⅔ of h_T, i.e., $\frac{2}{3} \times 21 = 14$ m. Because the drift section is not exactly rectangular and its height is 3.5 m, its floor is about 1.5 m below the $\frac{2}{3}h_T$. Therefore, h_S *is* about $14 - 1.5 = 12.5$ m.

Theoretical width W' of the extraction ellipsoid for a total extraction height h_T = 21 m is determined from Fig. 20.2.33 to be about W = 6.8 m.

Approximate total width and depth of the extraction ellipsoid, using formulas 5 and 6, is

$$W \leq 6.8 + 3.5 - 1.8 = 8.5 \text{ m}$$

$$d_T \leq \frac{8.5}{2};\ d_T \leq 4.25 \text{ m}$$

Spacing of the burden b after Eq. 20.2.8 is

$$b \leq \frac{4.25}{2};\ b \leq 2.12 \text{ m}$$

Because the total extraction height h_T = 21 m > 18 m, the horizontal spacing of sublevel drifts is calculated using Eq. 20.2.7:

$$S_D < \frac{W_T}{0.65} = \frac{8.5}{0.65} = 13.07 \text{ m};\ S_D \leq 13.07 \text{ m}$$

For good ore extraction, it is favorable when the adjacent zones of loosening at the elevation of the maximum width W_T of the extraction ellipsoid intersect. Therefore, the spacing becomes:

$$S_D = 12 \text{ m} < 13.07 \text{ m}.$$

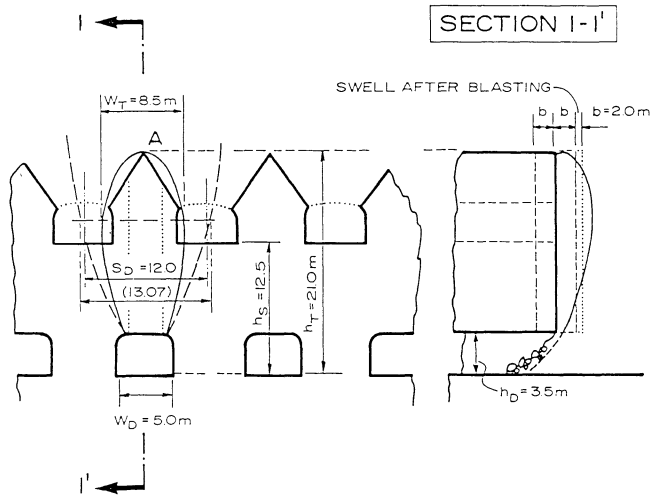

Fig. 20.2.42. Sublevel geometry in 20.2.1. Conversion factor: 1 ft = 0.3048 m

As shown in Fig. 20.2.42, the complete geometry of the sublevel caving layout in the example is:

Sublevel height h_S = 12.5 m (42 ft).
Horizontal spacing of sublevel drifts S_P = 12 m (40 ft).
Total width of the extraction ellipsoid W_T, assuming a total extraction height h_T = 21 m (70 ft), is W_T = 8.5 m (28 ft).
Spacing of the burden b = 2 m < 2.12 m (7 ft < 7.1 ft).

It must be emphasized that the system for the determination of sublevel caving geometry presented in this section is over simplified and should serve as a guide only. It can be used for developing a preliminary concept of the mining method geometry, especially in cases where no other data are available.

20.2.3.8 Large-dimension Sublevel Caving

The development of more modern, powerful, and efficient mining equipment, and especially the development of high-performance drilling machines permitting long, precision holes, has led to the introduction of new drilling designs in sublevel caving.

The tendency to replace classic fan drilling with a design using a majority of parallel drillholes began, more or less, simultaneously in different places of the world. At a sublevel caving symposium held in Stockholm, a new drilling design used at LKAB-Kiruna and as shown in Fig. 20.2.43 was presented (Anon., 1972). On the left side is the traditional pattern; on the right is the new "silo" pattern. Very similar geometry was presented at the same symposium using the layout utilized at Pea Ridge Mine, Sullivan, MO, by Meramec Mining Company. A design similar to the "silo" method was also developed by Kirov's Apatite Combine (apatite/nepheline ore on the Kola Peninsula, USSR), including the interesting combination of design and blasting sequence shown in Fig. 20.2.44:

Fig. 20.2.44a Mining in layers, consisting of 11-ft (3.2-m) slices (before blasting).
Fig. 20.2.44b Mining in layers with an advance arrangement (*t* is thickness of blasted ore and *l* is length of the sill pillar).

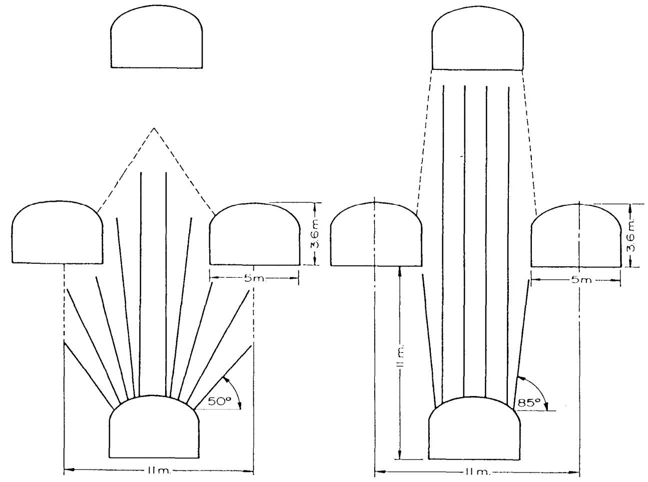

Fig. 20.2.43. Drilling design at LKAB-Kiruna: old design on the left; new "silo" design on the right, (after LKAB information brochure, 1989). Conversion factor: 1 ft = 0.3048 m.

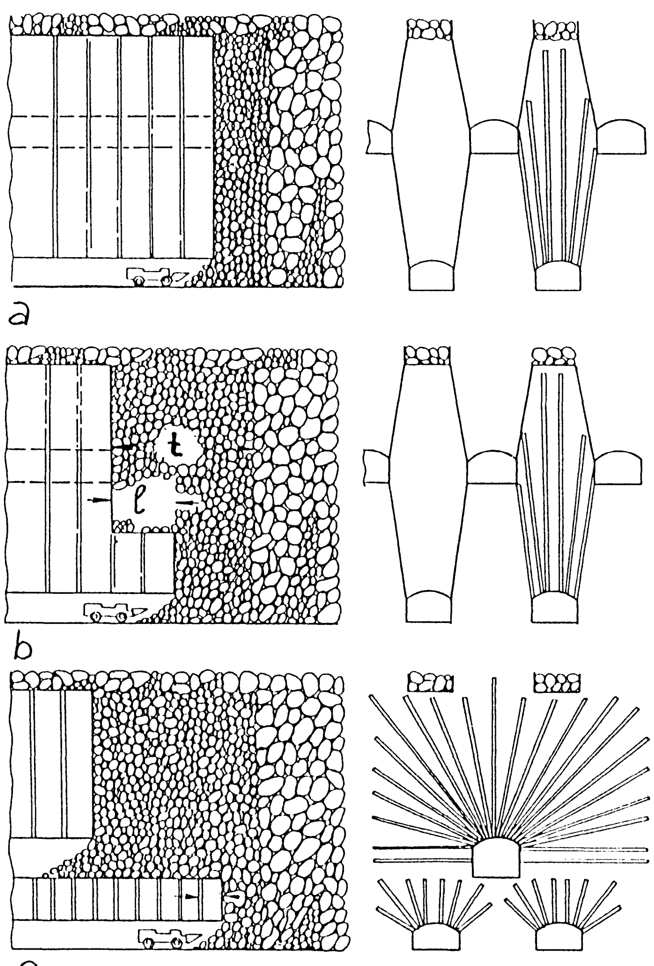

Fig. 20.2.44. Different alternatives of sublevel design as developed at Apatite, USSR, 1975.

Fig. 20.2.44c Mass advance blasting above the sill pillar using 4.2-in. (105-mm) blastholes and extraction of ore swell in the first phase; and in the second phase, blasting and extraction of sill pillars and overlying fragmented ore.

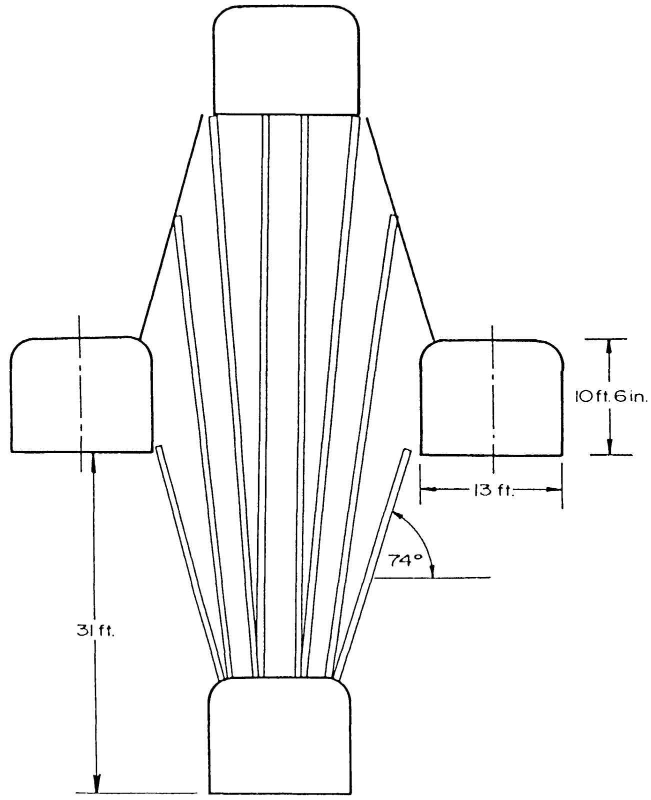

Fig. 20.2.45. Drilling pattern at Craigmont Mines. Conversion factor: 1 ft = 0.3048 m.

Fig. 20.2.45 shows another example of the same principle of drilling pattern as used at Craigmont Mines in Canada. The pattern discussed in Figs. 20.2.43 through 20.2.45 introduces longer drillholes than normally used in traditional sublevel caving. More or less parallel holes enable better and more uniform fragmentation. Steeper drillholes on both sides of the fan are better for gravity flow, not causing any changes in the principles of gravity Fig. 20.2.46. flow. As it is evident from Fig. 20.2.46, even a bin bottom inclination of 60° does not influence the parameters of the gravity flow area. Gravity flow in sublevel caving can be favorably influenced when the outside drillholes of the fan are very steep, that is, having an angle from horizontal of at least 65, 70, or 75° depending on the coarseness of the material. With such a steep angle, a much bigger part of the blasted ore will flow to the extraction opening (drift) in a mass flow, thus decreasing dilution. Such steep angle generates the gravity flow shape, by which the intermediate sublevel drifts are located in the middle of the extraction ellipsoid height.

In order to minimize expensive development works (sublevel drifts and haulage drifts), the LKAB-Malmberget Mine introduced large-dimension sublevel caving, the geometry of which is shown in Fig. 20.2.46. Sublevel height is 67 ft (20 m), horizontal distance between the axes of sublevel drifts is 75 ft (22.5 m), and sublevel drift width is 20 ft (6.0 m). Outside drillholes of the fan have an angle from horizontal of about 70°; drillhole diameter is 4 in. (100 mm).

Even this large-dimension geometry of the extraction ellipsoid (see Fig. 20.2.31) corresponds to the principle of gravity

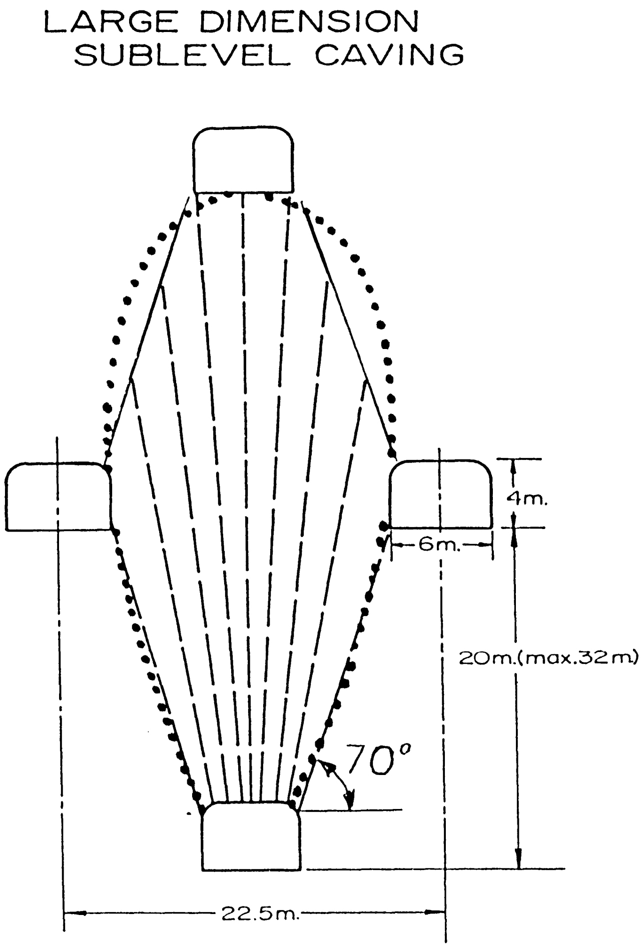

Fig. 20.2.46. Drilling pattern of large-dimension sublevel caving at LKAB-Malmberget. Conversion factor: 1 ft = 0.3048 m.

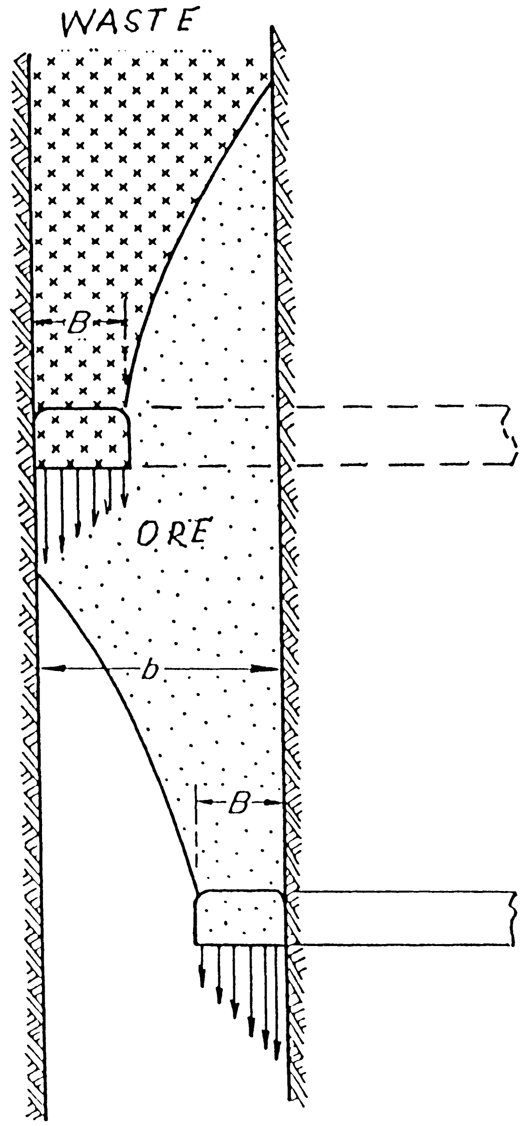

Fig. 20.2.47. Most favorable location of sublevel drifts in a narrow and more or less vertical ore body exploited by longitudinal sublevel caving.

flow as specified above. For better visualization, the typical form of an extraction ellipsoid is circumscribed (see dotted line) on the drilling fan. The same is valid for even larger dimensions that are introduced in parentheses in Fig. 20.2.46.

In order to satisfy the parameters of gravity flow, such large-dimension sublevel caving can give good results, especially when two (or more) burdens (slices) are blasted to ensure sufficient thickness of the fragmented ore. Naturally, sublevel caving with large-dimension geometry is especially favorable for application in magnetite ore where the increased dilution is easily balanced by very simple and inexpensive magnetic separation.

20.2.3.9 Longitudinal Sublevel Caving

In certain cases, sublevel caving can be successfully utilized for exploitation of narrow ore bodies. In such conditions, a transverse location of sublevel drifts is expensive, and therefore, sublevel drifts should be located in a longitudinal direction.

When the ore body is more or less vertical and wider than the sublevel drifts, the sublevel drifts should be located in the ore body as shown in Fig. 20.2.47. Such arrangement enables minimizing ore losses and dilution. A vertical or very steeply dipping ore body is the most favorable for longitudinal sublevel caving.

When the ore body layer is inclined, then we must be aware that gravity flow is influenced mainly by inclination of the ore body (hanging wall and footwall), but also by the width of the ore body layer. Probably the clearest and most simple explanation of the influence on the inclination and width of an ore body on the gravity flow is evident from the models a and b in Fig. 20.2.48. In longitudinal sublevel caving, gravity flow characteristics, a function of the ore body inclination and width, are defined in the figures by the different size and shape of the black passive zones. Fig. 20.2.48a shows the occurrence of passive (black) zones in ore layers of three different thickness with little inclination. This figure also shows three ore bodies (layers) as in case a, but in case b, the ore body inclination is steep. A direct comparison of three different thicknesses using the same inclination as well as different inclinations, defines the characteristics of the gravity flow by its passive zones.

This question is additionally explained in Fig. 20.2.49a, b, c, and d. These four cases show also the occurrence of passive zones P as a function of the inclination of the ore body and the location of the drift.

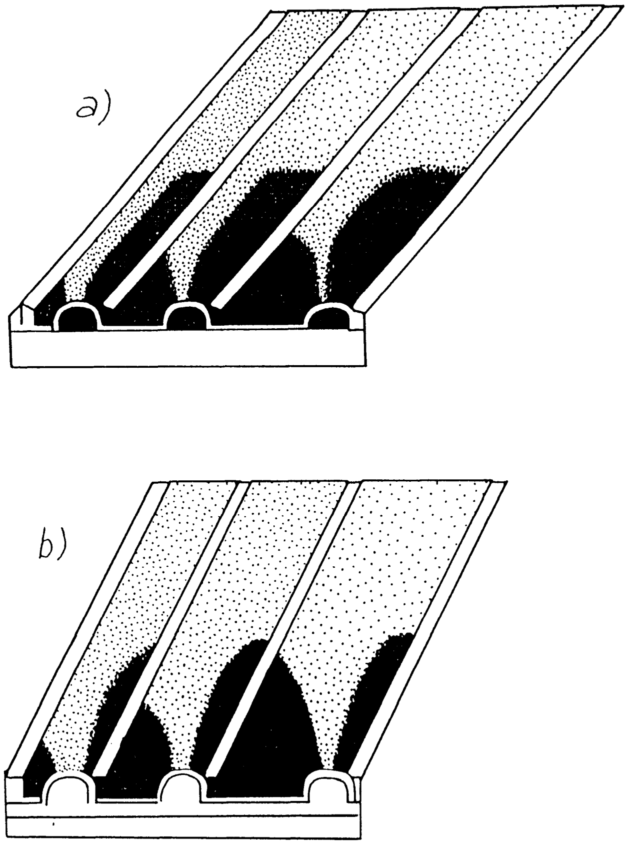

Fig. 20.2.48. Models of longitudinal sublevel caving.

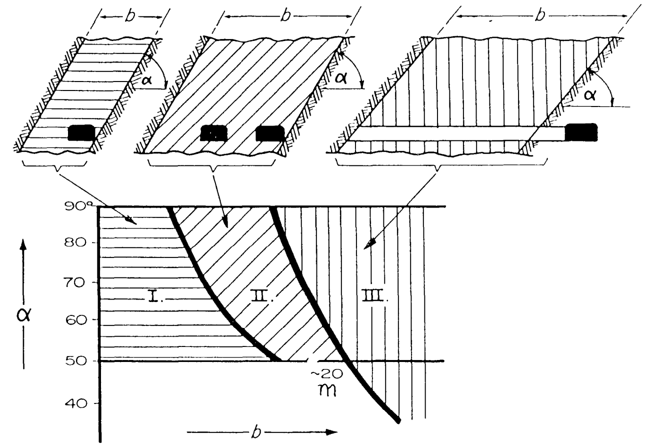

Fig. 20.2.50. Longitudinal (or transverse) sublevel caving as a function of ore body inclination and width.

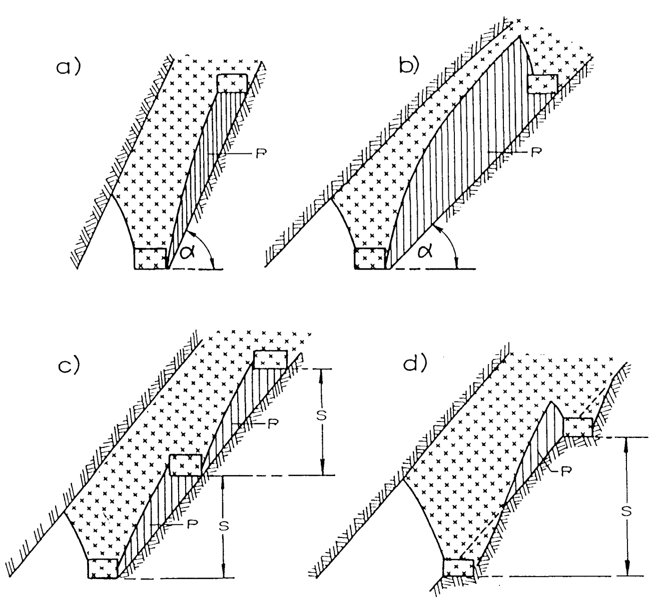

Fig. 20.2.49. Passive zone P as function of ore body inclination, ore body thickness, and location of the sublevel drifts.

Longitudinal sublevel caving is practical mainly for steep ore layers and for relatively narrow ore bodies. Maximum width of the ore layer is around 70 ft (20 m). For this thickness, it is then useful (in order to increase recovery) to utilize two sublevel drifts on each sublevel. A very approximate relation between ore body inclination, its thickness, and arrangement of sublevel drifts (longitudinal or transverse sublevel caving) is shown in Fig. 20.2.50.

20.2.4 SUBLEVEL CAVING IN COMBINATION WITH OTHER MINING METHODS

Sublevel caving can be used in many cases as a method for second-phase exploitation especially when the ore value is sufficiently high. Sublevel caving also can be used in all cases of room (stope) and pillar mining in flat ore deposits under economically favorable conditions. The necessary conditions are that the rooms are backfilled and that the rib pillars are accessible and remain sufficiently stable when perforated by the sublevel drift. The sublevel drift is located on the longitudinal axis of the pillar and follows the floor (or is below the floor) of adjacent rooms.

This method can be used to recover the remaining pillars (and sill pillars) of stopes that were backfilled. In this case, it is advantageous to have an ore body that is steep or very steep, after having been exploited primarily in backfilled stopes.

Sublevel caving was also successfully used to recover the pillars (about 33 to 37 ft, or 10 to 11 m thick) between the backfilled stopes at Kamoto Mine, operated by GECAMINES/ EXPLOITATION in Zaire in two parallel ore layers 50 ft (15 m) thick and only inclined 30 to 35°. Ore layers 50 ft (15 m) thick were separated by 50 ft (15 m) of waste. The combination of backfilled stopes (with overhand mining direction) and sublevel caving (with underhand mining direction) enabled the mining out of about 72% of the ore without dilution, attaining a total recovery of 82 to 85%.

It is possible to introduce a number of cases where sublevel caving can be successfully used to recover ore reserves remaining in situ after the main phase of exploitation.

Sublevel caving can be very successful in the recovery of remnant sill pillars and the passive zone of fragmented ore above the sill pillars, overlying the haulage drifts in a continuous caving area (Chapter 20.3). When extraction of the ore by continuous

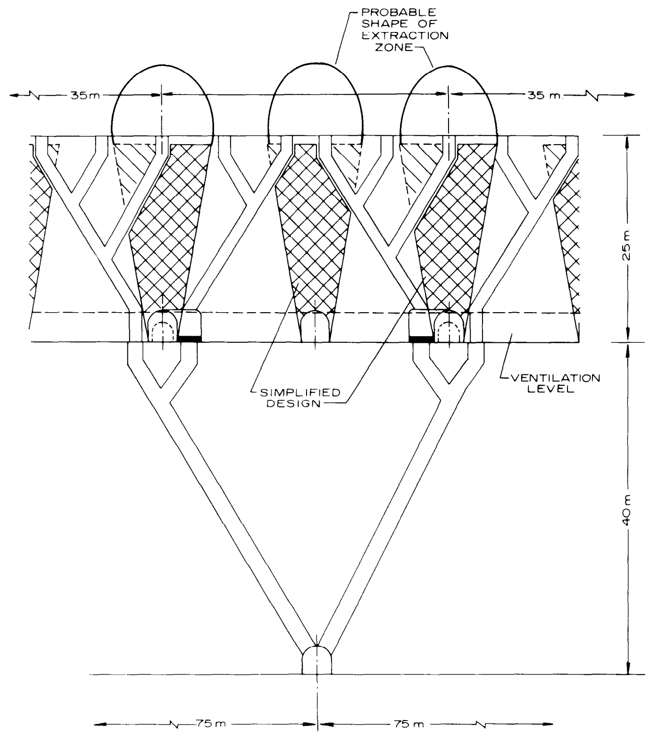

Fig. 20.2.51. Original recovery of remaining ore in sublevel caving after traditional (gravity) block caving at El Teniente mine. Conversion factor: 1 ft = 0.3048 m.

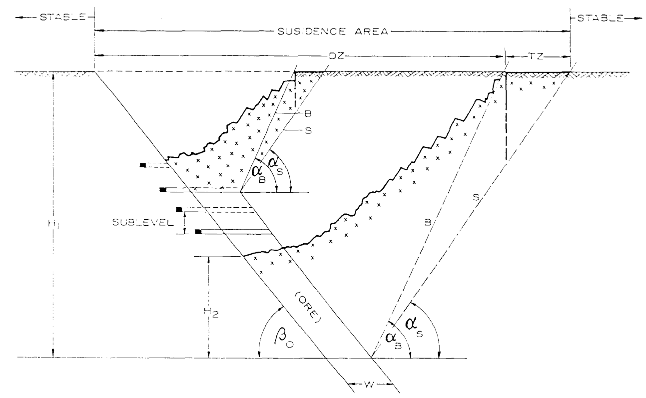

Fig. 20.2.52. Progressive caving and subsidence of hanging wall overburden, induced by sublevel caving.

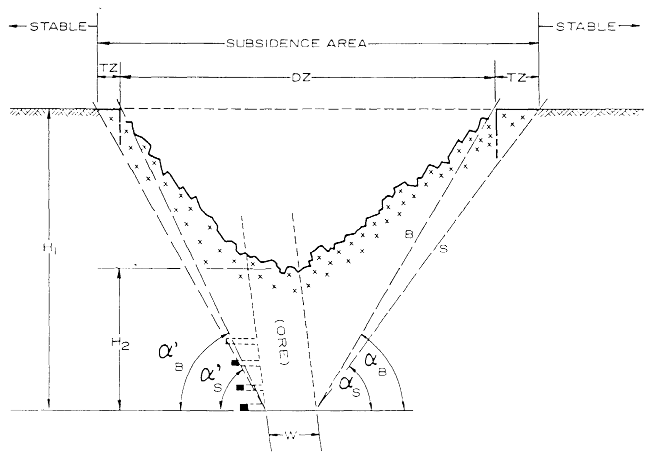

Fig. 20.2.53. Progressive caving and subsidence caused by sublevel caving, both in hanging wall and footwall.

caving is completed, then waste rock remains in the drawpoint drifts. The assumption that the relatively high passive zone of fragmented ore and the sill pillar above the production level will be recovered by the following, lower extraction level is very questionable, especially with regard to the very large height of modern block caving. The conditions for recovery are especially good when the production (extraction) level of continuous (block) caving has been constructed for load-haul-dumps. This enables very easy and inexpensive recovery of a big part of the remaining ore, because the most expensive part (large dimension drift) is there and was already paid for by block caving exploitation.

An original solution was developed by the engineering team in the mine planning and engineering department of El Teniente Mine, operated by CODELCO in Chile, for ore recovery after traditional gravity block caving. In this case, the underlying level 83 ft (25 m) below the new extraction level (see Fig. 20.2.51) was slightly modified and used as a sublevel drift level. Even this arrangement, although it does not recover all remaining ore reserves, was very efficient and inexpensive, because it required only fan drilling and mucking. All other necessary infrastructure for sublevel caving was there, because it was realized by the development of the block.

20.2.5 SURFACE EFFECTS FROM SUBLEVEL CAVING

Concerning environmental constraints, the basic condition for utilizing sublevel caving is that an inevitable deformation of the surface (subsidence) is permissible. Sublevel caving will induce progressive failure, caving, and subsidence of the overlying waste rock masses, undercut by mining. In the following section, the extent of the damage zone will be considered.

20.2.5.1 Shape, Size, and Extent of Disturbance

The shape, size, and course of the progressive failure and caving depend on many factors. Probably the most important are the inclination of the ore body β_0, ore body width W, depth H_1 of the bottom level of future underground mining, height H_2 of the caved waste rock zone which generates the arching forces that resist failure, shear strength of the rock masses involved in progressive failure, defined by effective cohesive strength c and friction angle ϕ, and density of the waste rock masses γ.

When the ore body inclination creates a stable footwall slope, the progressive failure and caving will occur only in the hanging wall as shown in Fig. 20.2.52. Mining of a vertical or very steeply dipping ore body will induce progressive failure both in the hanging wall and the footwall (Fig. 20.2.53). A simplified mechanism of progressive failure is characterized by the break plane B at angle α_B and the sliding plane S with angle α_S. The magnitude of these angles is a function of certain configurations of H_1, W, H_2, C, ϕ, γ, and α.

As is evident from Figs. 20.2.52 and 20.2.53, the deformation zone DZ is limited by the intersection of the break plane B with

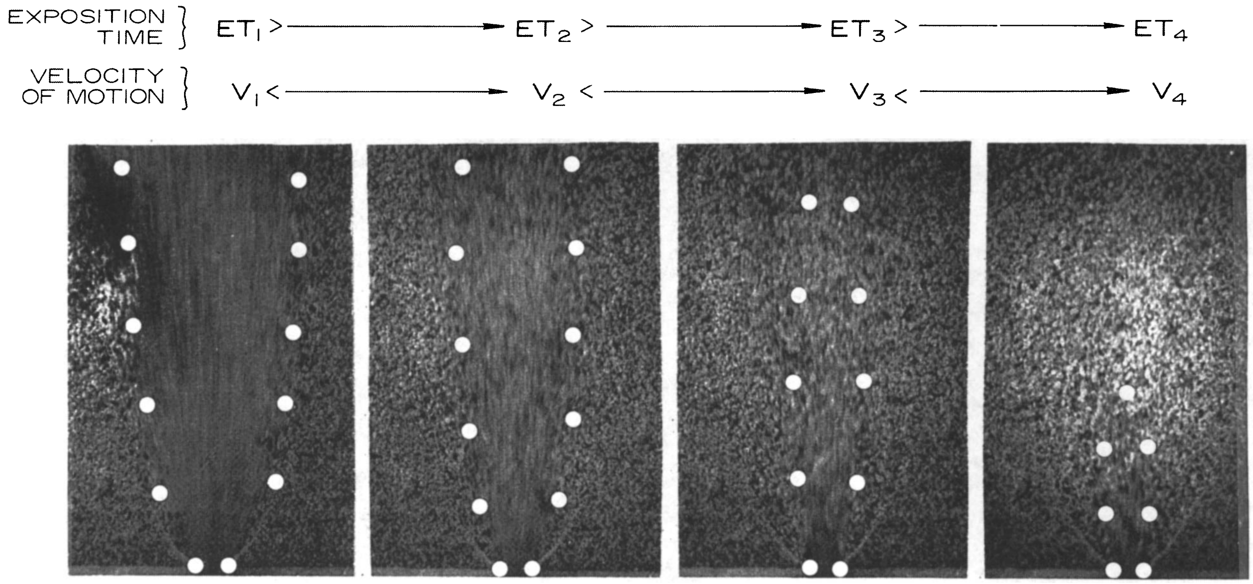

Fig. 20.2.54. Various velocities of motion zones $V_1 < V_2 < V_3 < V_4$ for granular material above the bin outlet. Material flows out continuously through the outlet and the bin is continuously refilled. Velocity of motion zones is visible on photographs, using various exposure times $ET_1 > ET_2 > ET_3 > ET_4$.

surface. Between the intersection of break plane *B* and sliding plane *S* with the surface is a so-called transition zone *TZ*. Even this zone is not sufficiently stable for the location of any permanent construction. The intersection of the sliding plane *S* with the surface represents the boundary of subsidence area. Naturally, the stable surface is outside the subsidence area.

For the safe location of permanent structures on the surface (shafts, concentrator, buildings, tailings, dams, etc.) and underground (crushing plant, pumping station, main orepasses and waste-rock passes, permanent tunnels and drifts, etc.), it is necessary to define the consequence of sublevel caving; that is, it is necessary to determine the unstable zones of progressive failure (limited by sliding planes) and the subsidence area. All important construction must be located outside of these zones.

20.2.6 OTHER CONSIDERATIONS

20.2.6.1 Distribution of Motion in Gravity Flow

Figs. 20.2.11 through 20.2.14 illustrate the principles of gravity flow for granular and coarse materials flowing freely from an extraction opening. Figs. 20.2.11 and 20.2.12 demonstrate the principles of velocity distribution of a particle's motion in the gravity flow. The velocity of this motion is greater at shorter distances from the extraction opening and from the vertical axis of the gravity flow. Inversely, the movement velocity of a particle decreases at greater distances from the extraction opening and from the centerline (vertical axis) of the gravity flow. The particles of a granular material are motionless outside of the contour of the ellipsoid of loosening or outside of an active zone.

For completeness and a better visualization and understanding of the distribution of particle motion in the gravity flow, the approximate elliptical shape of the flow zones at various velocities can be demonstrated in a single model (Fig. 20.2.54). On this model, which has a constant gravity flow (outlet is open and the model is continuously refilled from the top), the zones with various velocities of material movement can be identified using photographs with different exposure times. The relation between the velocity of particle movement and exposure time makes the moving particles in the model appear on the film (and photograph) as lines. An increased time of exposure allows smaller velocities of particle movement to be traced and recorded on the film. By shortening exposure time, all particles having a velocity of movement below that which corresponds to the ratio between exposure time and movement velocity, are “shut out”, appearing on the photograph as being motionless. Fig. 20.2.54 shows four subsequent cases of decreasing exposure time ET. The zones of particle movement velocity $V_1 < V_2 < V_3 < V_4$ were obtained at exposure times $ET_1 > ET_2 > ET_3 > ET_4$. The borders of these velocity zones are marked with white dots on each of the photographs.

20.2.6.2 Sublevel Caving with Large Vertical Intervals Between the Sublevels

As was explained in the discussion of Figs. 20.2.5 and 20.2.6, the slope of a bin bottom inclined 45 or 60° (from horizontal) does not influence gravity flow parameters or the occurrence of passive zones. Very steep bin bottoms are necessary in order to eliminate or minimize passive zones in bins of granular materials. For example, in order to obtain a satisfactory flow (without passive zones) of iron pellets (spherical-shaped) in a shaft furnace, it was necessary to increase the slopes of the furnace bottom to at least 77° (Kvapil, 1967). The same principle is valid for a basic geometry concept of sublevel caving with large vertical intervals between the sublevels.

The main unit of sublevel caving is determined by the geometry of a sublevel front on its vertical, transverse section. Fig. 20.2.55 shows that the geometry is determined by points 0, 1, 2, 3, 4 and (4), (3), (2), (l), (0), by inclination α of the peripheral drillholes of the fan (forming the slopes on intact ore), and by width *w* and height *h* of the sublevel drift. Naturally, the most

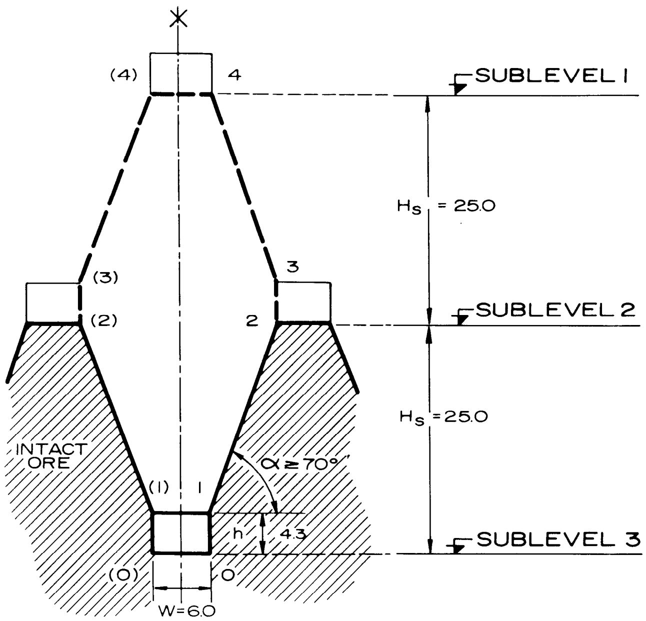

Fig. 20.2.55. Principal unit of sublevel caving geometry having large vertical distances between sublevels. Conversion factor: 1 ft = 0.3048 m.

important factor is the angle α. With an increased content of small fragments and fine particles in coarse material, the angle α should be increased to about 75°. With respect to the different types of coarse material (see Fig. 20.2.2), for material types I and II, an angle of 70° or even less could be used: and for material IV, the angle used could be about α = 75°. Such determination of sublevel caving geometry is meant only as a first approximation. Any higher stages of design require further study, using the principles specified in the foregoing text.

20.2.6.3 Importance of Precise Sublevel Caving Geometry

The importance of precise sublevel caving geometry on mining efficiency needs to be emphasized. Any deviation from the regular pattern of sublevel caving geometry (the relative position of sublevel drifts, as well as the precision, regularity, and axisymmetric shape of the sublevel drift sections) can cause disturbances in fragmentation, gravity flow, and stability. This deviation could result in decreased ore recovery, increased dilution, and increased costs for drift reinforcement and maintenance.

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Chapter 20.3
BLOCK CAVING

DOUGLAS E. JULIN

20.3.1 GENERAL DESCRIPTION

Block caving is a general term that refers to a mass mining system where the extraction of the ore depends largely on the action of gravity. By removing a thin horizontal layer at the mining level of the ore column, using standard mining methods, the vertical support of the ore column above is removed and the ore then caves by gravity. As broken ore is removed from the mining level of the ore column, the ore above continues to break and cave by gravity. The term block caving probably originated in the porphyry copper mines where the area to be mined was divided into rectangular blocks that were mined in a checkerboard sequence with all the ore in a block being removed before an adjacent block was mined. This sequence of mining is no longer widely used. Today most mines use a panel system, mining the panels sequentially, or by establishing a large production area and gradually moving it forward as the first area caved becomes exhausted. The term block caving is used for all types of gravity caving methods.

There are three major systems of block caving and they are differentiated by the type of production equipment used. (1) The first system based on the original block cave system is the *grizzly* or *gravity system* and is a full gravity system wherein the ore from the drawpoints flows directly to the transfer raises after sizing at the grizzly and then is gravity loaded into ore cars. (2) The second system is the *slusher system* which uses slusher scrapers for the main production unit. (3) The last system is the *rubber-tired system* which uses load-haul-dump (LHD) units for the main production unit. Each of these systems is described in some detail later.

Block caving is the lowest cost of all mine exploitation systems with the exception of open pit mining or in situ recovery.

20.3.2 SUITABLE ORE BODIES

20.3.2.1 Ore Body Characteristics

A typical ore body suitable for block caving is a porphyry-type deposit with well-disseminated mineralization and of fairly large lateral and vertical extent such as a copper porphyry deposit. It has also been used in molybdenum porphyries, hematite, asbestos, and diamondiferous deposits. It can be applied to steeply dipping vein deposits of sufficient width and in thick, flat-lying deposits. The rock strength can be fairly weak or fairly strong, but the total mass must have sufficient fractures in different orientations to allow the rock mass to break up under gravity into pieces small enough to pass through the drawholes into the production drifts. The lateral extent of the ore body must be large enough to insure that a cave can be established. The horizontal area required to establish a cave will depend on the strength of the rock mass, but generally the minimum horizontal dimension of the mining area should be about 300 ft (90 m). The height of the ore column should be sufficient to allow a reasonable productive life to the individual drawpoints and also to insure a reasonable rate of return on the development and production costs. The rate of return will also depend on the value of the mineral in the ore.

20.3.2.2 Determining Cavability

After determining that the size of the ore body is large enough to justify consideration for block caving, then the *cavability* must be determined. The frequency and orientation of the various fracture sets must be evaluated. The first look at the ore is usually through diamond drillholes. Some of the fracture sets can be identified by looking at the drill core; however, trying to determine the orientation of the fractures can be difficult unless the drill cores have been oriented. Determining the *rock quality designation* (RQD) of the drill core has become rather standard practice when block caving is being considered. RQD is determined by measuring the length of drill core pieces over 4 in. (100 mm) in length and dividing the accumulated length by the actual length of the drillhole section. Usually, the increments used are 10 ft (3 m). RQD can certainly be a first indicator for determining cavability, but it does not give the fracture orientation, it does not adequately identify the fracture intensity, nor does it give a complete picture of all the fracture sets. Access to the ore body by exploratory drifting is almost essential before a final determination of cavability can be made.

At least three sets of major fractures are necessary to justify the use of block caving. Two vertical sets at approximately 90° orientation to each other and a third set lying approximately horizontal will form rectangular blocks when the fractures separate. The spacing of the fractures is also important since the spacing will determine the size of blocks that will appear at the drawpoints. The fracture and joint fill material must also be identified since the composition of that material will help determine how easily the fractures will separate or if the fracture fill material is actually stronger than the host rock. The same analysis must be made of the waste capping to be sure that it will also cave as the ore is removed. If it appears that the capping will break more coarsely than the ore, then dilution of the ore will be minimized. If the capping will break more finely than the ore, then dilution will be a greater problem with a greater amount infiltrating into the ore. Major faults and dikes must also be identified since they may have an effect on the caving action and on the mine layout. Heslop and Laubscher (1981), White (1979b), and others have initiated systems for determining rock mass classifications. These classifications incorporate such information as joint spacing, joint conditions, intact rock strength, joint filling, and other factors. Based on the rock mass classification, the ease or difficulty of caving is determined.

As can be seen from the above discussion, a thorough geologic and rock mechanics study should be made before a decision to use block caving is made. There are numerous computer programs available to help in the evaluation.

20.3.2.3 Productive Capacity

Block caving is best suited for high daily production. The system has been used at mines with production rates ranging from 2000 tpd (1800 t/d) to more than 80,000 tpd (72,000 t/d). The productive capacity of a mine will be determined by the rate of vertical draw that can be made and the horizontal area that can be undercut at any given time without producing unusual ground control problems. Draw rates at various mines vary from

6 in. (150 mm) to 2 ft (0.6 m) per day. The daily rate of draw will be determined by the rate at which the ore will cave. Strong, less-fractured rock may cave very slowly whereas weak, highly fractured rock may cave very rapidly. There is no method to predetermine how rapidly a given ore body will cave. Experience at other mines with similar type rock will be the best guideline, and after that, experience at the new mine will be the final criterion.

20.3.3 SELECTING THE MINING SYSTEM

Once the cavability of the ore body has been determined and the approximate size of material that will arrive at the drawpoint evaluated, a mining system must be selected. The size of broken material will probably dictate which system is suitable. Mine site location, availability of labor, and economics will also be determining factors.

Drawpoint spacing is dependent on material size. If the ore breaks very fine, then close drawpoint spacing will be required. If the ore breaks fairly coarse, then wider drawpoint spacing can be used. It has been demonstrated at operating mines and by "sand-box" studies that the area of influence by one drawpoint is determined by the size of material in the cave. For fine ore, the area of influence is rather small and for coarser material, the area of influence increases. In summary: (1) For fine material, the full gravity system is most suitable. (2) For somewhat coarser material, the slusher system should be considered. (3) For quite coarse material, the LHD system may be best. Of course, other factors must be considered such as the sophistication of the work force, the cost of labor, the availability and capital cost of equipment, and the technical requirement for maintaining the equipment plus any other factors that may be unique to the particular mine being evaluated.

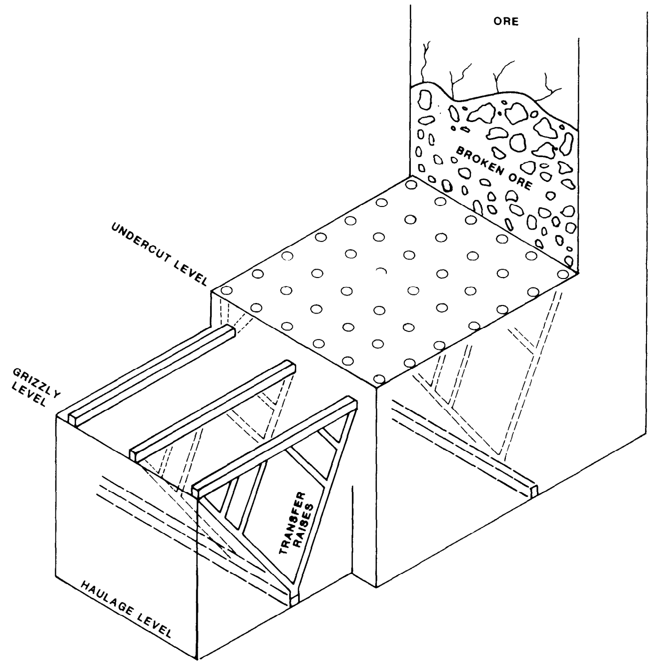

Fig. 20.3.1. Grizzly system—typical layout.

20.3.3.1 Choice of Various Forms of Block Caving

Full Gravity (Grizzly) System. The grizzly system consists of the haulage level, transfer raises, grizzly level, finger raises, and undercut level (see Fig. 20.3.1 for a typical grizzly layout). The haulage and grizzly levels are driven across the ore block to be mined. This work can be done simultaneously since they are on different levels. Finger raises are driven from the grizzly level to the undercut elevation, and then short horizontal connections are driven between the tops of the finger raises to form pillars that will later be longholed and blasted to initiate the caving action. Transfer raises are driven between the haulage and the grizzly level with a loading chute at the haulage level and grizzly rails for sizing at the grizzly level. After undercut blasting, broken ore flows down the finger raises and is sized by the grizzly rail spacing, passes into the transfer chutes, and thence is loaded into rail cars. Sledge hammers are used to break up oversized pieces at the grizzly, although some mines have tried using mechanical breakers. Some large pieces may hang up in the finger raises, and these are usually broken up by secondary blasting using packaged dynamite placed strategically against the oversized rocks.

The San Manuel copper mine in Arizona and the Andina copper mine in Chile are excellent examples of the grizzly system.

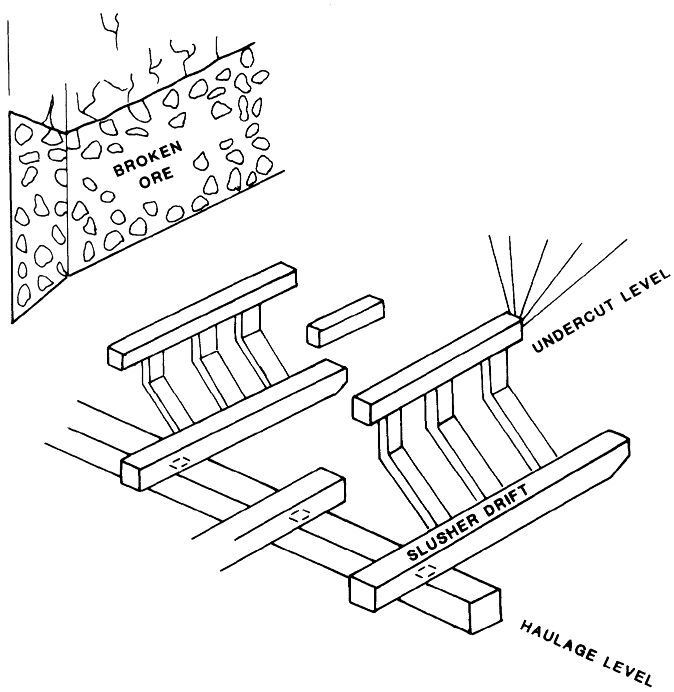

Fig. 20.3.2. Slusher system—typical layout.

SLUSHER DRIFT SYSTEM. The slusher system is used for rock that will break into moderate-sized pieces. Fig. 20.3.2 shows a typical slusher system. The system consists of the haulage level, slusher drifts immediately above the haulage, finger raises, and the undercut level. Haulage drifts are driven on even centers across the block to be mined. Ventilation drifts are then driven at the haulage elevation half-way between the haulage drifts. The slusher drifts are driven at right angles to the haulage at even intervals, usually with every other slusher drift driven at 180° to the previous drift. After placing concrete support in the slusher drift, finger raises are driven to the undercut elevation and concreted. The undercut drifts can be driven from the finger raises or can be developed from a separate access simultaneously

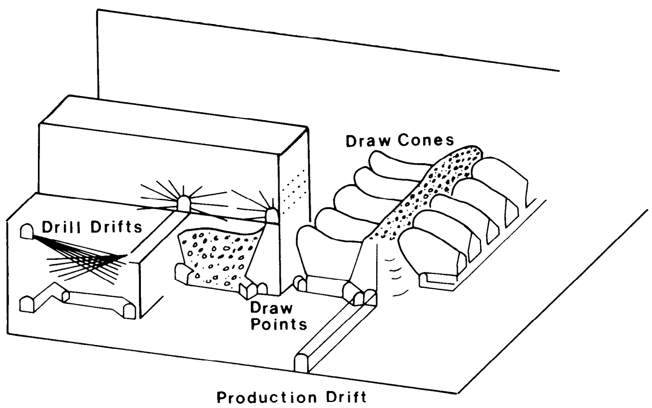

Fig. 20.3.3. LHD system—typical layout.

with the lower level development. Longhole drilling is done from the undercut drifts and then blasted to complete the undercut and initiate caving.

The Climax molybdenum mine in Colorado is a good example of the slusher system.

LHD (RUBBER-TIRED) SYSTEM. If geologic studies show that ore will break into relatively large pieces, rubber-tired equipment should be considered. Fig. 20.3.3 shows a typical LHD system. LHD units are much better suited to handling the larger pieces. LHD equipment usually requires that the drawpoints be spaced at greater intervals to allow room for the equipment to operate. The wider spacing is suitable for the coarser rock. This system consists of the haulage level, ore transfer raises, the production level, drawpoint entries, draw cones to the undercut level, and undercut drifts. Longholing and blasting is used to form the undercut that promotes caving. The large draw cone will allow larger pieces of ore to move down near the drawpoint where small drills and explosives can be used to break up the larger pieces. Experience at operating mines has shown that high hangups practically never occur with this system. With this system, using separate accesses, the haulage, production, and undercut levels can be developed simultaneously. The haulage and production levels should be separated by a substantial distance to provide adequate storage in the transfer raises so that the loading of the trains is not delayed waiting for the LHDs. When the haulage and production drifts are complete, then the transfer raises and drawpoint entries can be driven. Drilling of the draw cones can be done from the drawpoint entries or from the undercut level. It is much safer to do the blasting of the draw cones from the undercut level; therefore, the drilling should also be done from the undercut level. Longholing from the undercut is the final step before the undercut blasting is done. The Henderson molybdenum mine in Colorado, the Ertsberg copper mine in Indonesia, and the El Teniente copper mine in Chile are examples of mines that use the LHD system.

20.3.4 DEVELOPMENT

The sequence and type of development opening will vary somewhat depending on which of the three systems is used and perhaps on the configuration of the ore body. Two of the main components, the haulage and the undercut levels, are fairly standard. The third main component, the production level, will be substantially different.

20.3.4.1 Haulage Level

The haulage level is located under the production level, the distance depending to some extent on which system is used. With the grizzly system, short transfer raises are usually driven between the production (grizzly) level and haulage level to allow for some storage of production rock for loading the haulage units. There is no reason why the length of raises cannot be extended to allow for greater storage if it seems desirable. With the slusher system, the haulage level is usually directly beneath the slusher loading hole so the ore can be loaded directly into the haulage cars although long transfer raises have been used at some mines. Using transfer raises with a slusher system may require an unusual amount of raising since each slusher drift will require an orepass or at least an orepass branch to transfer the ore to a gathering level. With the LHD system, the haulage and production levels should be separated by a substantial distance to provide adequate storage of production rock and to be able to reduce the number of raises required to meet production goals. Raises should be inclined at + 60 to 70° from the horizontal. This will insure good flow of the rock and provide additional breaking of the larger rock fragments as they drop down the orepass.

The Ertsberg mine in Indonesia has eliminated the need for large separation of production and haulage levels through the use of central storage bins and a distance of 33 to 49 ft (10 to 15 m) between production and transfer levels. The production ore is fed through grizzlies into central bins that discharge onto a 5-ft (1.5-m) belt that in turn discharges to a central crusher-conveyor-orepass system.

20.3.4.2 Production Level

In the grizzly system, the grizzly level becomes the production level. These drifts are usually driven at a fairly small cross section and are used for access to the finger raises and grizzlies for sizing the ore before it is sent down the transfer raises. The drifts may or may not be concrete lined although concrete lining is recommended. With the slusher system, the slusher drifts may be considered the production level. The drifts must be driven at a cross section adequate to fit the slusher scraper size. Scrapers used are usually 5 to 6 ft (1.5 to 1.8 m) wide. The drifts are generally concrete lined. The production level for the LHD system is the LHD drifts that provide access to the drawpoints.

20.3.4.3 Undercut Level

The quality of work on the undercut level is very important. The undercut drifts are driven a suitable distance over the tops of the drawpoints. They are to be used for the longhole drilling that will effect the final undercut blast to produce caving. The undercut drifts must be driven with great accuracy to be sure they are in proper alignment with the workings below. Crosscuts between the undercut drifts may be driven from time to time to facilitate access between the drifts and to delimit areas to be blasted during some specified time period.

20.3.4.4 Longhole Drilling

Longhole drilling is the last step before undercut blasting. Its purpose is to remove the pillars between the undercut drifts and to form the horizontal slot that will allow the ore above to commence caving. The longholes may be drilled horizontally through the pillars or at some angle above the undercut level in order to form an apex between the drawholes (the flattest angle is usually about 45°). The drill pattern must be spaced evenly

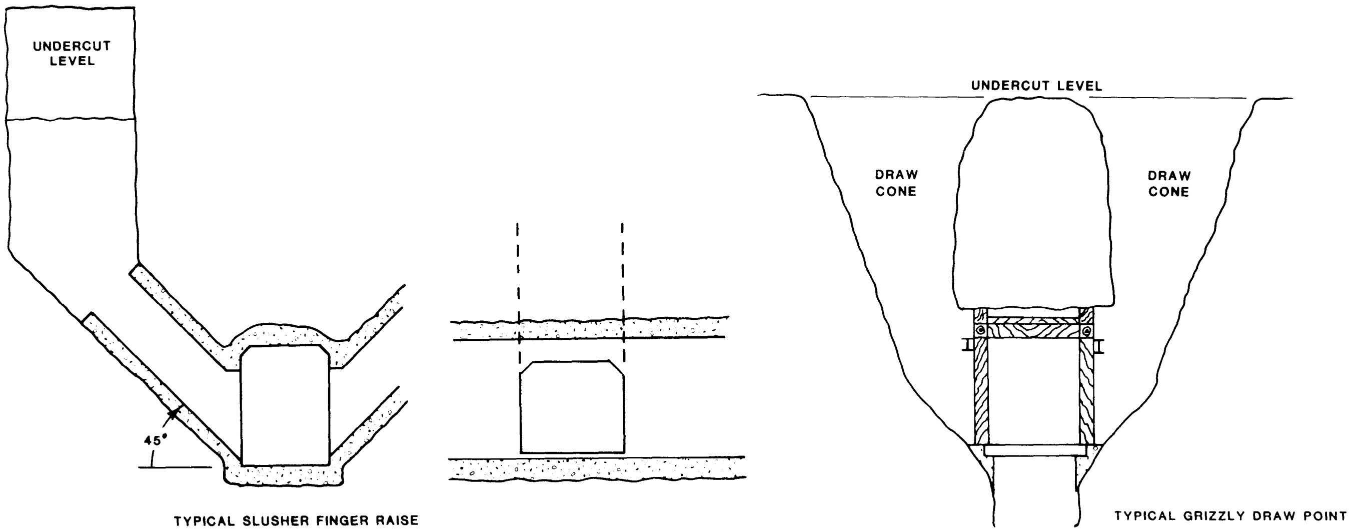

Fig. 20.3.4. Drawpoint design–slusher and grizzly.

along the undercut drifts with the spacing between drill patterns determined by the hole size and powder load to be used. The height or thickness of the undercut formed varies substantially between various mines. The height of the blasted zone can vary from as little as 8 ft (2.4 m) to 50 ft (15 m). There is some question as to how much benefit is derived from a very high undercut. If the ore is very blocky and caving may be difficult to initiate, then the higher undercut may be justified.

20.3.4.5 Drawpoints

The drawpoints for the grizzly and slusher systems are driven at right angles to the grizzly or slusher drift. They are also inclined to the horizontal. The angle varies from nearly vertical down to 45°. The cross section should be large enough to allow most of the rock fragments to pass through the opening without requiring secondary breakage. After driving the inclined section of the drawpoint out to the drawpoint center, a vertical section may be driven upwards to the elevation of the undercut level. This will allow the undercut level to be raised further above the production level to insure an adequate pillar between the cave area and the production level. See Fig. 20.3.4 for a typical drawpoint design. Drawpoints may or may not be lined with concrete. A concrete lining will allow a better flow of the broken rock and will retain its size better than unlined rock. If repair is required because of excessive erosion, then repair can be more easily accomplished if the drawpoint was concreted originally. It is important to maintain the size of the drawpoints to prevent flooding of the production drift with broken rock.

With the LHD system, the drawpoint entry must be nearly horizontal to allow entry of the production unit. The connection between the production level and the undercut level is usually a fairly large draw cone formed by drilling and blasting. The draw cone is sized to allow larger pieces of rock to move down near the brow of the drawpoint where they can be drilled and blasted if they are too large for the LHD bucket. The brow of the drawpoint should be high enough to allow the LHD bucket to lift a load, but not so high as to allow broken rock to flood out into the production drift passageway where it will hinder free travel of the LHD.

20.3.4.6 Boundary Weakening

Boundary weakening has two purposes. One is to minimize side flow of the waste material into the broken ore column and the second and more important is to facilitate the caving. This is especially important during the initial caving period. All rock has a tendency to arch, and this arching will often hinder attainment of a fully active cave. Boundary weakening should be placed on two adjacent sides of the initial caving area. The extent of the weakening will depend on the strength of the ore. For a very weak, highly fractured ore body, some corner raises and drilling along the two boundaries may be sufficient. For a stronger, less-fractured ore body, a completely fractured zone about 8 to 10 ft (2.4 to 3 m) may be required. This type of boundary weakening is formed by driving sublevel drifts along the boundaries at vertical intervals about 60 ft (18 m). The number of sublevels required will be determined by the height of the ore column and the strength of the rock. Two or three lines of vertical drillholes are then drilled between the sublevels. When these drillholes are loaded and blasted, they will produce a completely fractured zone that will weaken two legs of a potential arch and greatly facilitate the initial caving action. Fig. 20.3.5 shows this type of boundary weakening. Once an active cave has been established, pre-splitting may be all that is required, or it may be possible to eliminate all boundary weakening.

20.3.4.7 Undercut Blasting

Undercut blasting is the last step to initiate the caving. This should be done with great care and the cave area constantly checked for pillars of unblasted ore. It is advisable to blast only a few longhole patterns at a time so that the task of checking for pillars is not too difficult. All pillars should be removed before the next blasting sequence. If pillars are left in the cave area, they will hinder proper caving action and can transmit damaging weight into the workings below. The end result is either excessive expenditures for repair or, even worse, the loss of ore.

20.3.4.8 Support

Support of the rock is very important in block caving. As the cave area is enlarged, the weight that was supported by the

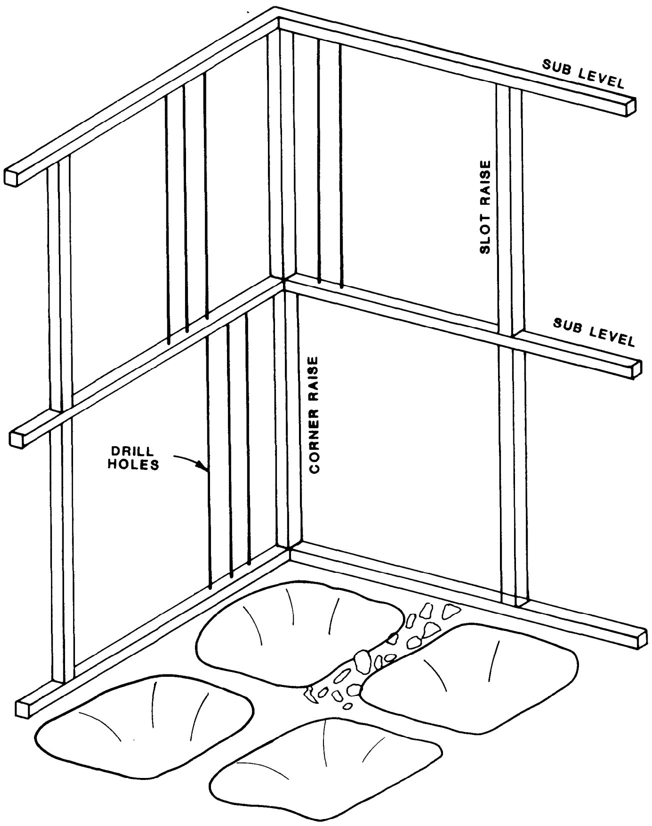

Fig. 20.3.5. Boundary weakening.

undercut area will be transferred to the area around the undercut area. This will place added stress on the rock and workings immediately adjacent to the cave area. It is important to maintain as much of the initial integrity of the rock as possible. Rock bolts and concrete are the most common method of support used in block cave mines. Rock bolts should be installed as soon after the openings are made as possible. It is preferable to have rock bolting as an integral part of the normal drill-blast-muck cycle. Timber and steel sets have been used for support of the main haulage ways although concrete is being used more often. The most common support in grizzly drifts, slusher drifts, and LHD drawpoints is concrete. Reinforcing steel is not recommended where blasting concussion will be present. It has been found that the concussion will often separate the concrete from the reinforcing steel, exposing it and causing substantial problems with loose rebars hanging out into the working area. The concrete should have high strength, no segregation, and minimal cold joints.

20.3.4.9 Ventilation

A good ventilation system and frequent monitoring of the air quality is essential with a block caving system. The production phase for all three systems will generate a substantial amount of dust, especially if the ore is very dry. Contaminated air should be removed from the fresh airstream soon as possible. Intake and exhaust laterals that are separated from the main mine workings are the most desirable. The intake lateral should be so positioned that controlled amounts of air can be introduced into the main haulage or production drifts and then to the individual working places. Contaminated air should be captured as close to the working place as practical and sent to exhaust vent drifts and thence to a main exhaust lateral where it is discharged from the mine. The amount of air required will depend on the number of working places being utilized at any given time and on the type of equipment being used. Air requirements for diesel equipment is substantially greater than for compressed-air or electrically operated equipment.

Table 20.3.1. Drawpoint Spacing and Block Height

Mine	Median Fragment Size ft (m)	Drawpoint Spacing ft^2 (m^2)
Ertsberg	2.3 (0.7)	2540 (236)
El Teniente	2.3 (0.7)	2410 (224)
Grace	2.6 (0.8)	1776 (165)
Henderson	1.6 (0.5)	1600 (148)
Creighton	1.6 (0.5)	1184 (110)
Climax	3.0 (0.9)	1141 (106)
Urad	2.0 (0.6)	900 (80)
Thetford	1.6 (0.5)	624 (58)
Mather	0.6 (0.2)	344 (32)
San Manuel	1.3 (0.4)	258 (24)
Lakeshore	0.6 (0.2)	280 (26)

20.3.5 PRINCIPLES AND PRACTICES

20.3.5.1 Drawpoint Spacing

Drawpoint spacing is important to insure good recovery of the ore. If the drawpoint spacing is too far apart, then good ore will be lost or dilution may become so great that the ore becomes uneconomical. If the drawpoint spacing is too close together, then the mining costs are greater than they should be. There are no set formulas to accurately determine what drawpoint spacing should be. Comparing the ore to other block caving mines is the best guideline. Each drawpoint has a certain area of influence on the broken ore above. The drawpoint spacing should be so spaced that these areas of influence will overlap slightly to insure that the total ore column is moving downward as the draw progresses. Table 20.3.1 shows drawpoint spacing at several mines.

White (1979a) has constructed a regression curve based on the information from Table 20.3.1 that can be used to approximate the draw area based on median fragment size. For example, if the medium fragment size is 0.8 m (2.6 ft), then the draw area would equal 131 m^2 ($y = 152 \times 0.8 + 9.8 = 131.4$ m^2, or 1414 ft^2).

A fragment size estimate based on a fracture frequency study does not account for attrition of the fragments as they are drawn down through the ore column. A conservative approach in selecting drawpoint spacing based on the foregoing formula should be used.

20.3.5.2 Height of Blocks

The height of ore column mined can vary a great deal from mine to mine or even in the same mine. The critical height of ore is such that the mineral content will pay for the development, production, milling, and overhead costs, and still provide some profit. The grade of the ore and selling price of the product will have a substantial effect on critical height. The productive life of each working place should also be considered. If the tonnage is exhausted too rapidly, it may be difficult to develop new working places rapidly enough to maintain the desired daily production. It is obvious that the higher the ore column, the cheaper the development cost per ton will be; however, there are other considerations that will affect the final decision. The configuration of the ore body may determine where the mining level or levels should be established. It is usually not very practical to mine waste material to recover the ore above so that a horizontal horizon in the ore body may set a mining level. The number of tons that can be drawn through a drawpoint before repairs are required may also have some effect on the height of ore drawn per drawpoint. If several repairs will be required before all the ore is exhausted, it may be advisable to chose a lower height of ore. The height of ore drawn will also have an effect on the amount of dilution drawn with the ore. The higher the column of draw, the more potential there is for greater dilution. If the thickness of the dilution zone is small compared to the ore column, or if the dilution will break coarser than the ore, then higher columns of ore can be drawn. If the dilution zone is very thick in comparison to the ore column, or if the dilution material will break much finer than the ore, then shorter ore columns are more desirable. Ore columns varying from 100 to 820 ft (30 to 250 m) have been drawn successfully.

20.3.5.3 Dilution Control

It is not possible to predetermine the exact dilution factor, but some estimate can be made based on experience at other similar mines. The dilution factor typically can vary from 10 to 25% of the total ore drawn. The amount of dilution that can be accepted often is a function of ore grades, grade of the dilution material, costs, and metal prices. Regardless of these factors, the more dilution tonnage that is drawn, the higher the cost will be per pound of mineral recovered.

One of the greatest dangers of block caving is the potential for excessive dilution that could significantly reduce the grade of the ore or result in a significant loss of ore. The only method to insure good ore recovery with minimal dilution is through carefully monitored draw control. The draw must be distributed evenly over the entire caved area, being sure that the rate of draw does not vary substantially between drawpoints. The ore adjacent to an uncaved block should not be completely drawn down before the adjacent ore has been caved. Dilution can easily migrate from the sides into the newly caved area, cutting off the ore above and preventing it from caving. The purpose of good draw control is to maintain a minimum area of contact between the ore and the dilution (waste) zone above. An uneven draw line will increase the area of contact between the ore and dilution, creating more opportunity for the ore and dilution to mix. Concentrating production in a small area of the caved ore will create a funnel for the dilution to flow down into the ore zone to become intimately mixed with the surrounding ore. The more dilution that is mixed with the ore, the lower the grade of ore drawn and the more tons will have to be drawn to recover the mineral, or a greater amount of ore may be lost. There are various methods of draw control. One that has been used successfully at the Climax and Henderson mines is to limit the draw in the drawpoints against uncaved ore to a small percentage of the ore column. The draw is increased by equal percentage increments in each line of drawpoints as they progress away from the cave line. The principle is to maintain an ore-to-waste contact of 45 to 50° moving away from the uncaved ore. Fig. 20.3.6 demonstrates this method. Other methods are used at the different mining properties, but the main parameter is to maintain an even draw among the drawpoints.

20.3.5.4 Mining Limits

Mining limits are usually determined by economic limits in porphyry-type deposits. In this type deposit, there may be no definite separation between mineralized and waste rock; rather the grade of ore gradually diminishes until it becomes too low to recover economically. In this case, the mining limit is set by the minimum height of economic-grade ore that will still contribute to the profitability of the mine. Where there is a sharp differentiation between the ore and the waste rock, the setting of mining limits is somewhat simpler. If the lateral extent of the ore body is very large, it may not be practical to mine the full width of the ore body as one block. In that case, it will be necessary to divide the mining level into several blocks and mine them sequentially. If there are to be more than two mining blocks, then the new mining block should be immediately adjacent to the first mining block. If mining is done on two sides of an unmined block, the potential for excessive rock pressures in the unmined block becomes much greater.

20.3.5.5 Observation of Cave Action

When new caving is initiated, it is natural to wonder how the caving action is progressing. Unfortunately, it is not possible to see into the cave area at least until the first surface subsidence occurs. Various methods have been tried to monitor the caving action with varying degrees of success. The method used depends on how much money and effort the owner is willing to invest. One method is to drive an observation drift along one or two sides of the initial cave area at some distance above the undercut level and outside the cave line. Crosscuts are driven over to the cave limits. If an open arch is forming over the cave, the arch can be observed from the crosscuts. It is also possible to hear the caving action if the cave is active. If the arch should become stable and caving action has stopped, the observation drifts can be used to drill and blast longholes out over the arch to help restart the caving action. Obviously, this method is rather expensive. Once the cave is fully active, the observation points are covered with broken rock and are no longer useful.

Another method is to use drillholes that are over the area of the initial cave to survey the top of the arch. These drillholes can be used to install co-axial cables that are cemented in the holes. As the caving progresses upward, the cables will bend or break. Using a galvanometer, the resistance of the cable can be measured to determine the location of the break. Kinks or breaks may occur anywhere in the cable where there is a displacement of the rock. The method does not locate the top of the broken ore, so the method is not the most accurate. Wooden or metal plugs attached to a cable can also be dropped down the drillholes to measure the top of the arch. When the plug reaches an obstacle (theoretically the broken ore) then the plug is pulled back up the hole until it reaches the arch back. By measuring the length of cable, the top of the arch can be measured fairly accurately, and the top of the broken ore is approximately located. Obviously, the more drillholes that are used, the more complete the survey. Fig. 20.3.7 shows how this plug is used.

20.3.5.6 Weight Problems

Weight problems are inherent to block caving. As rock is removed in one area, the support of the rock above that area must be transferred to the surrounding in-place rock. It follows that the pressures will increase due to the extra support require-

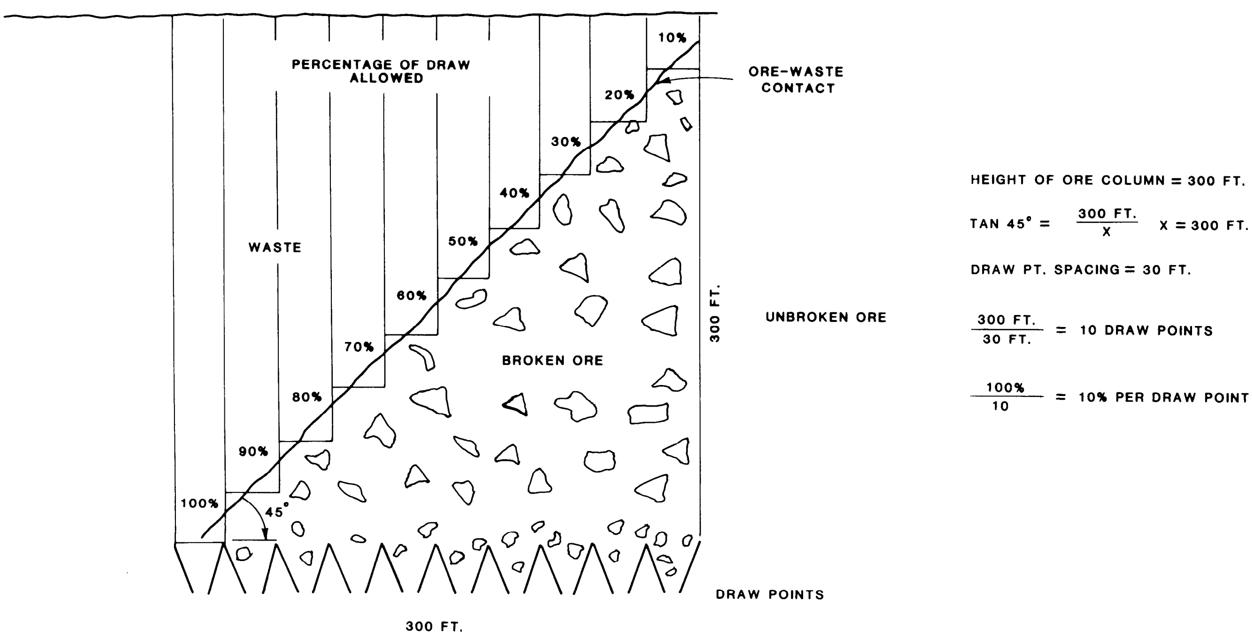

Fig. 20.3.6. Draw control section. Conversion factor: 1 ft = 0.3048 m.

ments. As the mine excavation for development progresses, the in-place rock will also take on additional weight (or pressure). The more rock that is removed for development, the greater the pressure on the remaining rock. This pressure is transferred to the workings in the production area. When the undercut is blasted, there is a momentary release of that pressure that can cause cracking of the concrete support due to relief of the compression. If the tension cracking that results from this relief is too severe, it may be necessary to place rock bolts in the concrete to prevent slabbing.

The advancing cave line is usually the area of greatest pressures that will cause damage during the development stage. The mine workings directly under and in front of the advancing undercut are most subject to weight damage. By advancing the undercut blasting as rapidly as possible, the length of time that the workings are exposed to this excess weight can be minimized. If the undercut cannot be advanced rapidly, then the final connections between the production level and undercut level should be delayed until just before the undercut is blasted. The general idea is to retain as much rock in place as possible until just before the undercut blast so that support of the rock can be retained.

Another source of weight damage is from unblasted pillars or stubs left in the undercut area. These result from incomplete blasts or poor mining practice. Pillars of ore in the cave area can interfere with the proper caving action and transfer unusual pressures to the workings below causing severe damage. It is important that the undercut area be checked after every undercut blast to be sure that no unblasted pillars remain. Any remaining pillars must be removed before the undercut blasting continues. Some weight will be transferred to the areas around the cave limits. If possible, no critical workings should be constructed immediately adjacent to the cave limits. Where it is necessary to have workings in the critical area, they should be well supported. Once caving reaches the surface, weight problems will become somewhat reduced, but the mining personnel should continue to be alert for possible areas of isolated weight problems. To minimize the effect of weight at the advancing undercut line, the line between caved and uncaved ore should be advanced at some angle to the main workings. The cave line should never be stopped for an extended period of time immediately over or adjacent to a main working. The longer the workings are exposed to the excess weight the greater the danger of major damage.

There are several methods of monitoring weight during the production period. The simplest is to install extensometer stations at various locations in the various working areas to measure the divergence between the back and floor. The rate or amount of divergence can signal where weight problems may be occurring.

20.3.5.7 Promoting an Active Cave

In caving, all rock has a natural tendency to form an arch. The dimensions required to produce an active cave will depend

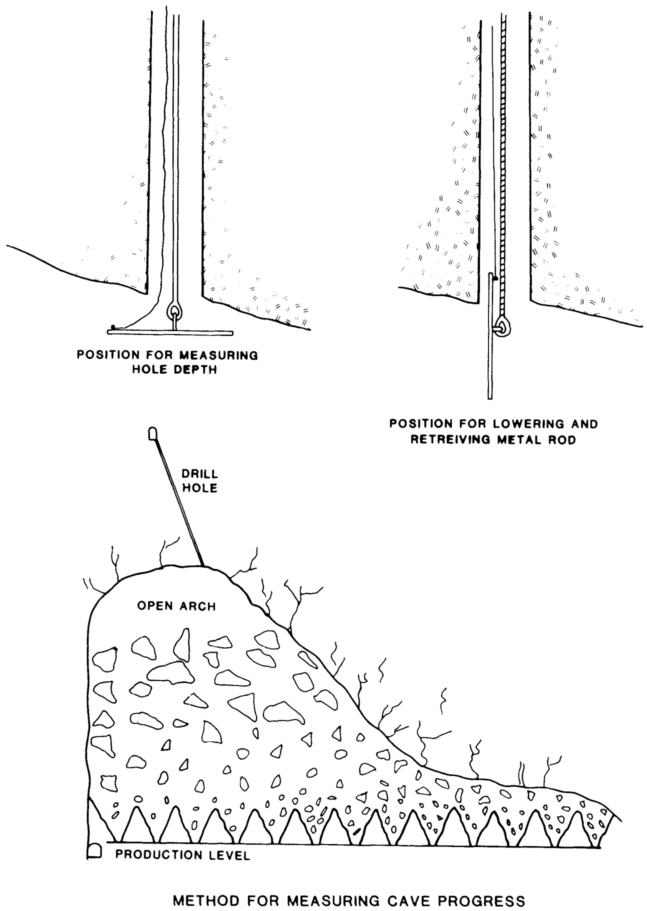

Fig. 20.3.7. Cable and plug for finding top of arch.

on the strength of the rock mass. If the rock mass is highly fractured and weak, a fully active cave will be started with a fairly small undercut area. The stronger the rock mass the greater the total undercut area that will be required to establish an active cave. The mine layout should provide for expanding the undercut area in two adjacent directions. The minimum dimensions of the cave area will determine when an active cave will start. That is, if an arch is formed over the short axis of the undercut area, then expanding the cave along the long axis will probably not produce a fully active cave. In order to break a stable arch over the undercut area, another line of undercut should be blasted, preferably along the short axis. This will cause additional caving action until another stable arch is formed. Then another line of undercut should be blasted. Eventually, the shear strength of the rock mass will be overcome and caving will be continuous. Boundary weakening can be a great aid in establishing an active cave especially during the initial caving sequence. Boundary weakening is discussed in 20.3.4.6.

A number of people have developed methods to determine rock competency classifications for rock support and cavability. Laubscher (1977) has used his classification to determine the undercut area required to establish an active cave. This method assigns a hydraulic radius required for caving based on the various rock classifications. The method can be used for planning the initial undercut area, but some care should be used in advancing the undercut blasting since the size of area required may vary for different types of ore bodies.

20.3.5.8 Induced Caving

Usually increasing the size of the undercut area will eventually produce a cave, but there may be occasions where the undercut area is becoming so large that it does not seem advisable to continue the undercutting until an active cave is established. There also may be occasions where the mining limit has been reached in one dimension and no further expansion is possible. It then becomes necessary to try some type of induced caving. If the new mine is in an area of previous mining, then some of the old drifts above the caving elevation can be used for access. If the original mine plan included some observation drifting, then these drifts can be used for access above the cave area. If neither of these options is available, then some auxiliary raising and drifting may be necessary. The usual methods for trying to promote caving are through longhole drilling or powder drifts. Longholes can be drilled around the perimeter of the undercut area and blasted as a presplit to try to induce further caving action. Longholes drilled over the top of the stable arch and blasted will usually promote some additional caving action. In the case of drilling out over the top of the arch, efforts should first be made to locate the top of the arch so that the longholes will be properly located and the powder load can be calculated with some confidence. Drifts over the top of the arch can be loaded with explosives and well stemmed to prevent a possible "gunbarreling" effect. This method has been effective in finally inducing a fully active cave. In this case, the top of the arch should be located so that the proper powder load can be calculated.

20.3.5.9 Development Rate

The rate of development must be balanced with the production rate. Generally, if the number of new tons caved during a year is equal to the number of tons drawn, then the productive capacity should remain constant. If the tonnage above the individual drawpoints varies significantly, then some adjustments must be made so that the number of available drawpoints is adequate to meet the production requirements. The amount of lead time required to prepare the drawpoints must be carefully calculated. The lead time required will depend on the available manpower and the efficiency of the manpower and equipment being used. Generally, about three years will be required to prepare a new production drift from the time the initial access is started until the undercut longholes are blasted. Undercut drifts should be completed about six months to one year before undercut blasting to allow adequate time to complete the longholing. Longholing should precede blasting by about six months.

20.3.5.10 Number of Active Drawpoints for Production

The number of drawpoints required for a given level of production will depend on the rate that the ore will cave, the amount of tonnage that will flow through the drawpoint per day, and the productive capacity of the equipment. The productive capacity of the equipment can be calculated from the tables supplied by the various manufacturers. These tables should be used with some discretion since there are always some delays not accounted for in the tables. The rate of draw will depend on how rapidly the ore will cave. Very weak ores may cave very rapidly while stronger ores may cave at a very slow rate. The rate of draw should be fast enough so that there is room for the in-place ore above to cave but not so rapidly that a large void is created between the in-place ore and the broken ore. Draw rates can vary anywhere from 6 in. to 2 ft (150 to 610 mm) per day. The amount of tonnage that can be expected from a given drawpoint also will depend on how many hangups occur during a given time period. Once the number of drawpoints required has been determined, a small percentage should be added to allow for hangups and for repair. Availability will probably range between 80 to 90% for new drawpoints. As the drawpoints become older and require repair, the availability may decrease. When calculating the capacity of the production equipment, be sure not to forget the haulage equipment and its accessibility to the loading points. If all the loading points are concentrated in one or two loading drifts, it may not be possible to reach the desired tonnage even though theoretically there are adequate drawpoints.

20.3.5.11 Secondary Blasting

The caving action will always produce some oversize boulders that will not pass through the drawpoints. This is especially true during the initial caving period when attrition has not had an opportunity to reduce the size of the rock fragments. The common method of breaking oversize rock is by use of concussion blasts. Five- to 8-lb (2.2- to 3.6-kg) bags of explosives ("bombs") are placed in strategic spots between rock fragments. In order to place the bombs, the bags are tied to sticks 10 to 20 ft (3 to 6 m) long. Primacord is laced through the bag for detonation, and using the stick, the bomb can be placed against the hangup. If hangups in several drawpoints are to be blasted simultaneously, then the primacord can all be tied together and ignited with one detonator. If hangups consist of clusters of smaller rocks, a high-pressure water hose can be used to loosen the hangup. With the LHD system, large rock fragments are usually close enough to the drawpoint brow that they can be drilled with a carrier-mounted jackleg or small single-boom jumbo and loaded with stick dynamite.

Shooting hangups, especially with the slusher system, is hazardous work, and employees require special training.

20.3.5.12 Subsidence

Surface subsidence will always occur with block caving. The time from initial production to the first surface disturbance will depend on several factors: depth of mining level below surface, strength of ore, and rate of draw are the major factors. The first sign of subsidence may be a circular hole or funnel appearing somewhere within the boundaries of the undercut area. Subsidence can also be a general settling of the surface within the boundaries of the undercut area. Eventually, the total area of caving will settle, and escarpments will form. Escarpments may be nearly vertical to as flat as 45° or less, depending on rock strength and topography. Tension cracks will form around the escarpments. If a 45° line is drawn from the undercut level to the surface, this is the area in which tension cracking may occur. No permanent structures should be placed inside this line since there is a high probability that they will be disturbed by the tension cracks.

20.3.6 MINE DESIGN

20.3.6.1 Grizzly Systems

The grizzly system is best suited for ores that break very finely and require closely spaced drawpoints. It is also favored where labor is plentiful and fairly cheap. The system is labor intensive and can be used where sophisticated mining equipment is not available. The system requires considerably more development than the other systems, and therefore development costs are comparatively high.

HAULAGE LEVEL. The haulage or gathering level is driven some distance below the grizzly level to allow room for storage for loading the trains. This distance should be about 60 ft (18 m) but may be much greater if orepasses are used to transfer the ore to a central gathering level. If long orepasses are used, there should be a short section just below the grizzly level so that the tons of ore drawn per drawpoint can be measured for draw control purposes. The haulage level may be subjected to extreme weight conditions, so often the haulage drifts are supported with concrete.

The cross-sectional size will be determined by the haulage equipment dimensions and safety requirements for manway clearances.

RAISES. Transfer raises are driven from the haulage level to the grizzly. Usually, each transfer raise will serve multiple grizzlies. The number of grizzlies that can be served by one transfer raise will depend on the distance between the two levels and the grizzly chamber spacing. The transfer raises are usually lined with timber or concrete. Since this system is used in weaker ores, lining the transfer raise is necessary to insure that the raise will last the life of the production from above. Chutes are usually installed at the bottom of the transfer raises although other loading mechanisms could be used. Grizzly rails are installed at the top of the transfer raise in order to size the material passing down the raise. The cross-sectional size will be determined by the amount of ore storage required within the constraints of the ground conditions.

GRIZZLY DRIFTS. Grizzly drifts are driven to connect the top of the transfer raises that lie in a given line. The drifts serve as manway access to the drawpoints and for distribution of fresh air to the grizzlies. The grizzly drifts are usually lined with concrete or timber for support against the weight that may occur during undercutting and production. The pillar above the grizzly drift may be supported with rock bolts. The grizzly drifts can be relatively small in size since no production equipment will be required to pass through. They are connected around the fringes of the ore block for ease of access between the drifts and for ventilation purposes.

DRAW (FINGER) RAISES. Draw raises are driven from the grizzly to the undercut level. Two raises are driven on opposite sides and at right angles to the grizzly drift and located at the top of the transfer raise. These raises are driven nearly vertical or inclined at an angle not less than 45° above the horizontal. The distance between the grizzly and undercut level depends on how much pillar is required between the two levels. A 15-ft (4.5-m) pillar is common.

The raises may or may not be lined with concrete. The brow at the intersection of the draw raise and the grizzly drift is supported with posts and beams to help maintain its size. During production, if the brow becomes too worn, the broken muck will flood the grizzly drift, making access to the grizzly difficult.

UNDERCUT DRIFTS. Undercut drifts are driven to connect the tops of the draw raises. The driving of the undercut drifts forms pillars that are later drilled and blasted to initiate the caving action. These drifts are driven large enough to accommodate the drilling equipment that will be used to drill the pillars and to provide expansion room for the blasted pillar rock. When the undercut drifts have been completed, the pillars are then drilled and blasted to initiate the caving. Great care must be taken to see that the blasted pillars have been completely removed before any additional pillars are blasted.

VENTILATION. The ventilation of a grizzly system is fairly simple. Fresh air is passed through the haulage drifts and then up to the grizzly levels. Fresh air is circulated through the drifts passing from one fringe drift to the other and then to a main exhaust vent drift and to the surface. With such a system, the working areas may not always be getting the best of air, especially where dry and dusty ore is encountered.

20.3.6.2 Slusher System

The slusher system can be used for fine-breaking ores but is better suited where the ore breaks into medium-coarse pieces. Since the system uses electric slushers for production, somewhat more mechanical expertise will be required, although slushers are not usually a high-maintenance item. Where a substantial amount of the production rock will be coarse, large slushers are most appropriate (125 to 150 hp, or 93 to 112 kw). The scrapers will be 5 to 6 ft (1.5 to 1.8 m) in width.

HAULAGE DRIFTS. Haulage drifts are driven on even centers across the block to be mined. The distance between haulages is determined by the slusher drift length and the drawpoint spacing. The size of drift is a function of the size of haulage equipment. The drifts can be supported by timber or steel, but concrete is the support most often used.

SLUSHER DRIFTS. Slusher drifts are usually driven directly above and at 90° to the haulage drifts. The slusher drift consists of a loading cutout directly above the haulage, a slusher cutout at the end opposite from the slusher drift, and the slusher drift. The size of opening depends on the size of scraper being used. The width of the finished slusher drift should be only 1 ft (0.3 m) greater than the width of the scraper. This will help prevent side spill as the loaded scraper is pulled to the drawhole and produce greater payloads for each scraper trip. The slusher cutout, loading cutout, and slusher drift are usually lined with concrete, although other support materials have been used. The floor of the slusher drift should have rails installed as this provides a better sliding surface for the scraper and prevents gouging of the floor concrete. The drift should also have an upward grade to 5%. This will allow for water drainage toward the haulage and also will favor the loaded scraper for somewhat easier loading. The slusher drifts are usually staggered along the haulage. That is, every other slusher drift is driven at 180° from the previous slusher drift.

VENTILATION DRIFTS. Ventilation drifts are driven one-half the distance between each pair of haulage drifts directly under the tail ends of the slusher drifts from two adjacent haulages. Small ventilation raises to the slusher drift above serve for exhausting contaminated air from the slusher drift. The ventilation drift is sized according to the amount of exhaust air that it will be required to handle. Ventilation drifts are not usually concreted but may require roof bolts and shotcrete.

FINGER (DRAW) RAISES. Finger raises are driven at right angles to the slusher drifts at an angle of approximately 45° above the horizontal. The raises are advanced to the centerline of the drawpoint spacing and then driven vertically to the undercut level. Finger raises are driven from both sides of the slusher drift, usually in opposite sets although they may be staggered from one side of the drift to the other. The size of the finger raise is determined by the size of material that the designer wishes to pass with ease through the finger raise. If the finger is too small, then the frequency of hangups is greater, increasing the need for secondary breakage. If the finger is too large, then the material reaching the slusher drift may be too large for the scraper to handle. Lining of the finger raise is desirable for two reasons. The height of the brow of the drawpoint must be controlled. If it becomes too high, then the broken rock can flood the slusher drifts, making it impossible to pull the scraper behind the muck pile for efficient loading. It has been found that rock rubbing against rock will wear faster than rock against concrete. When

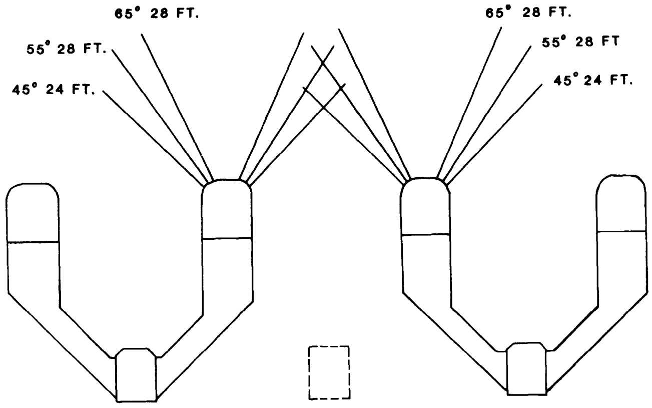

Fig. 20.3.8. Typical drill pattern for undercut blasting—slusher drift. Conversion factor: 1 ft = 0.3048 m.

repair of the drawpoint becomes necessary, it is much easier to repair concrete than to try to form new concrete against worn rock.

UNDERCUT DRIFTS. Undercut drifts are driven over the tops of the finger raises. The size of undercut drift is usually determined by the size of equipment used for the longhole drilling. Great care should be taken so that the drifts stay on the centerline of the drawholes to insure that the longhole drilling is properly spaced for an effective blast of the final undercut. The elevation of the undercut level is determined by the amount of pillar desired between the slusher drifts and the cave bottom. The pillar should be adequate to provide for wear during the production period and for protection of the slusher drift if weight should occur.

LONGHOLE DRILLING. Longhole drilling is usually done in the shape of a fan with the bottom holes being drilled at a minimum angle of 45°. The size of the longholes and the powder load required are dependent on the rock strength, and these two will determine the spacing between the fans. The drilling and blasting of the drill fans in two adjacent undercut drifts form an apex over the slusher drift that protects the slusher drift and also help the ore over the apex move to the finger raises. Care must be taken when blasting the drill rings to be sure that no unblasted pillars of ore are left. These pillars will interfere with good caving and could cause weight to be transferred to the workings below. No more than two or three drill fans in an undercut drift should be blasted at one time. Fig. 20.3.8 shows a typical drill pattern for undercut blasting.

VENTILATION. Ventilation with a slusher system is somewhat more positive than with the grizzly system. An intake ventilation lateral can be driven around the fringes of the mining area or underneath the haulage level. Fresh air flows through the loading drifts and into the slusher drift. The air then passes through the slusher drift to the vent connection and into the exhaust ventilation drifts where it flows to a main vent lateral and then is discharged. The volume of air feeding each slusher drift can also be controlled by placing a vent box at each vent connection. By changing the size of vent box opening, the amount of air in each slusher drift is controlled. This system does require constant monitoring to be sure that various controls are in proper position, especially in areas of high secondary blasting.

20.3.6.3 LHD (Rubber-tired) System

The LHD system has been gaining in popularity in recent years because of the high productivity and efficiency that can be obtained. The system requires less development per ton of ore and has a high production-output capability. The savings in mining manpower can quickly be negated, however, because of the greater mechanical manpower required. If the mechanical maintenance program is properly organized, executed, and monitored, then real savings can be realized. It should be emphasized that a well-trained mechanical crew is required. If skilled manpower is not available in the local labor market, then a good training program at the mine site will be required.

The system is best suited to ore bodies that are not intensely fractured and will cave in larger pieces that would not be practical to handle with the other two systems. Diesel equipment requires substantial room to maneuver and therefore will require wider drawpoint spacing. This is especially true of 5-yd^3 (3.8 m^3) equipment and larger.

HAULAGE LEVEL. The haulage level should be located well below the production level so that adequate storage is available in the centralized orepasses. This will provide sufficient loading capacity so that haulage trains are not directly dependent on LHDs for loading and larger-sized ore cars can be used. The size of the haulage drifts is dependent on the size of haulage equipment. Since the haulage level is located well below the caving level, the drifts are not usually subjected to unusual weight, and therefore standard support methods are adequate.

PRODUCTION LEVEL. Production drifts are driven on even centers across the ore body or production block. The spacing between production drifts is determined by drawpoint layout and spacing. The size of the drifts depends on the size of production units being used. The larger the production unit, the greater the cross section required. The length of the unit will also have some effect on the width of the drift since adequate room to maneuver the unit into the drawpoint is required. The production drifts can be subjected to future heavy weight; therefore, good support is required. These drifts should be roof bolted and concreted or at least shotcreted.

DRAWPOINTS. Drawpoints are driven horizontally from the production drifts. They are driven at some angle to the production drift, usually about 45° to facilitate entry of the LHD for production. The distance from the brow of the drawpoint to the opposite rib of the production drift must be sufficient for the LHD to be at nearly zero degrees articulation when entering the muck pile. The height of the brow of the drawpoint must be high enough to allow the bucket of the LHD to raise its load but not so high that the broken rock will flood out into the production drift. The drawpoint is usually lined with concrete in order to maintain the proper size and also for support against the weight from undercutting and blasting. Steel lintels in the brow have also proven beneficial.

Drawpoints from two adjacent production drifts may be connected by a crosscut. The crosscut is used later for forming the draw cone. One draw cone will feed broken ore to two drawpoints (Fig. 20.3.9).

UNDERCUT LEVEL. The undercut level is driven at some distance above the production level, usually in the range of 50 ft (15 m). This system has the advantage that more pillar can be left between the production and caving level, offering greater protection to the production level. The undercut drifts are usually driven directly above the production drifts. The size of the undercut drifts is determined by the size of equipment used. The drilling equipment may require more room than the LHDs used for driving the drift because of room required for drill rod changes. It is good practice to use the same type of LHD for development as is used for production. This helps reduce the number and types of equipment required. These drifts are not usually supported unless rock conditions require some temporary support.

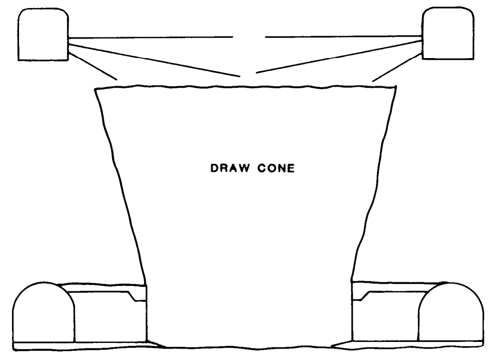

Fig. 20.3.9. LHD drawpoint and cone.

LONGHOLE DRILLING. Drilling of the undercut longholes and for the draw cones is done from the undercut level. Drilling the draw cones from the undercut level does require more drilling than if done from the production level, but the blasting sequence for the draw cones is much safer from this level. Blasting for the draw cones is usually done one or two rings at a time. When drilling is done from the bottom, once the first rings have been blasted, access to the remaining rings is difficult and hazardous. A typical drilling pattern for the undercut and draw cone is shown in Fig. 20.3.10.

Blasting of the draw cones immediately precedes blasting of the undercut. The cones should only precede the undercut blasting by one or two drawpoints. This way, the minimum amount of ground ahead of the cave line is opened and more support is offered to the weight that precedes the cave line. Blasting of the longholes is done two or three fans at a time. With this system, it is fairly easy to check for unblasted pillars. As with the other systems, the cave line should be advanced at some angle to the major workings to minimize the amount of weight transferred to the production level.

OREPASSES (STORAGE BINS). Orepasses are driven between the production and haulage levels for transferring production rock to the trains. The orepasses should be spaced intermittently along the production drifts so they can be serviced by several drawpoints. The distance between orepasses is determined by the most efficient haul distance for the LHDs and the amount of tonnage per orepass required per shift or per day to service the haulage trains below. By use of branch orepasses, several production drifts can be serviced by one loading point on the haulage level. Loading chutes are installed at the bottom of each orepass and a dump pocket installed at the production level. Orepasses are not lined because of their length, so care must be taken to place them in good-quality rock. If the rock is good, one orepass should be able to pass approximately 2 million tons (1.8 Mt) before it becomes too worn for further use. Raise boring machines can prove to be economical if the orepasses are fairly long. The use of these machines can greatly simplify the driving of long orepasses.

VENTILATION. Ventilation requirements for the LHD system are substantially higher than for the other systems because of the high ventilation requirements for diesel equipment. Mine Safety and Health Administration standards for diesel equipment are published (Chapter 11.5). A system of distribution drifts completely separate from the production level is almost obligatory in order to bring the proper amount of clean air to the various working areas.

Main ventilation laterals can be driven underneath the production level. Fresh air is introduced to the production level at

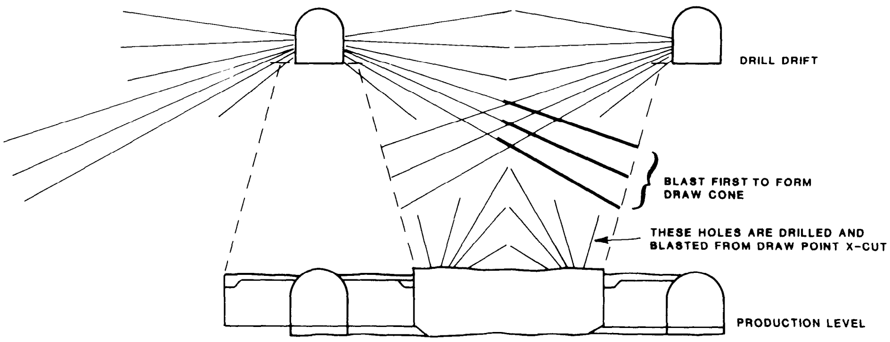

Fig. 20.3.10. Drilling pattern for drawcone and undercut.

strategic points throughout the mine by raises from the intake lateral to the production level. Small ventilation raises adjacent to each orepass dump point pick up the contaminated air and take it to small exhaust vent drifts below the production level, thence to the main exhaust lateral, and finally discharge it to the surface. The volume of air discharged at each exhaust raise has to be replaced with clean air immediately downwind from the exhaust raise in order to maintain the proper quantity of airflow in the mine.

20.3.7 ECONOMICS

Block caving is probably the least expensive underground mining system since the amount of drilling and blasting per ton of ore produced is much less than with any other system. The actual cost per ton for development is difficult to quantify since it depends so much on the height of ore column being developed on any given mining level, the type of block cave system being used, and the drawpoint spacing. The recent severe depression of the mining industry has caused most mining companies to aggressively seek cost-cutting measures and increased efficiencies. Costs that were applicable in the early 1980s are no longer applicable. The best approach for the engineer who is trying to develop a cost projection for a new mining property is to generate the costs for each item of work required to develop a mine, based on the use of modern equipment and high manpower efficiencies, and then to apply the total to the complete block of ore to be mined. Great care must be taken in using historical costs from other mines since the methods of stating costs per ton vary significantly from one mine to the next. Some mines carry development costs as an operating expense whereas other mines may capitalize development costs and amortize them over the life of the ore block or complete ore body. In the case where a mine treats development costs as an operating cost, if development is progressing at a rate higher than production, then costs per ton produced may appear high. If the development rate is slowed down, the costs per ton produced will appear low. Some mines use a mixture of operating cost and capital costs. Main entries, ventilation laterals, and equipment purchases are capitalized; and haulage drifts, drawpoints, and undercutting may be charged as operating expense. When using historical costs from another property, be sure that the method of costing is thoroughly understood.

A recent publication by the US Bureau of Mines (Anon., 1987) can be used to get a "ball-park" cost estimate of US mining costs.

20.3.8 CASE STUDIES

20.3.8.1 El Teniente Mine, CODELCO, Rancagua, Chile*

The El Teniente copper mine, a division of CODELCO Chile, is located 31 miles (50 km) northeast of Rancagua City. The first mining level is situated at an elevation of 6506 ft (1983 m).

Main access to the mine is through a 5-mile (8-km) tunnel on Teniente 8 level. Access to the mining levels above is through two main shafts. Shaft B services 5 to 1 levels. Shaft C services levels 8 to 5. Both shafts are equipped with friction-type hoists with double-deck cages having a capacity of 350 employees. Total hoisting capacity of shaft B is 37.5 tons (34.0 t) and of shaft C is 32 tons (29.0 t). Other shafts are available for various services.

Present daily production is 99,500 tons (90,273 t), coming from four separate mining areas:

North Mine (Teniente 4)	32,273 tons (29,272 t)
South Mine (Teniente 1)	13,219 tons (11,990 t)
Central Mine (Teniente 4)	43,922 tons (39,837 t)
Fortuna Mine (Teniente 4)	8,073 tons (7,322 t)

A total of 40,653 tons (36,873 t) is sent to the Sewell Concentrator and the balance to the concentrator at Colon. The mine operates three 8-hour shifts per day for 360 days/year. Total work force is 3198.

GEOLOGY. The El Teniente mine is one of the largest copper ore bodies in the world today. The ore body has a generally triangular shape in plan view, elongated to the north, where it is centered in the Teniente Porphyry and extends toward the southwest into the Sewell Diorite. The central area consists of a low-grade zone related with the Braden Pipe. The vertical development includes an oxidized-leached zone that averages 328 ft (100 m) thick, a supergene enrichment zone that reaches a depth of about 1640 ft (500 m) in the northern part of the ore body, and a hypogene or primary mineralization zone, proven to a depth of 5249 ft (1600 m).

Eighty percent of the copper and molybdenum mineralization occurs in andesites of the lower member of the Farellones formation. This formation (Miocene) was subdivided in the area of the ore body into three members separated by angular discon-

* Based on information from Gabriel Gutierrez C., El Teniente Division, CODELCO.

tinuities: the lower member consisting of massive andesitic flows, the middle member with epidotized andesitic flows with reddish lacustrine sedimentary intercalations, and the upper member, with basaltic intercalations.

The intrusions in the deposit are diorite, dacite porphyry, latite porphyry, and andesite and lamprophyre dikes. In the central part of the ore body, there is a complex megastructure designated as the Braden Breccia.

The main hydrothermal stage is superimposed on the late magmatic stage and it is characterized by quartz-sericite-chlorite and anhydrite bearing veinlets and alteration halos. It develops to the northeast and subordinately northwest in veinlets, which commonly do not affect the whole rock. The main sulfides associated with this stage are chalcopyrite and pyrite, which are disseminated in halos and as veinlet fillings.

METHOD OF EXPLOITATION. There are two basic types of ore at El Teniente, primary and secondary, that are characterized as hard (primary) ore and soft (secondary) ore. The primary ore that breaks into coarser fragments is mined by an LHD system. The secondary ore that breaks into smaller fragments is mined with the grizzly (full gravity) system.

Grizzly (Full Gravity) System—This system has been used in the secondary ore for many years at Teniente. It is currently being used in the South Mine (Teniente 1) and the North Mine (Teniente 4). Typical mining block dimensions are

width	197 to 262 ft (60 to 80 m)
length	295 to 394 ft (90 to 120 m)
height	394 to 590 ft (120 to 180 m)

Fig. 20.3.11 shows the arrangement of the full gravity system.

Production drifts are driven at 49-ft (15-m) centers across the block to be mined. Drift size is 6 by 7.5 ft (1.8 by 2.3 m). Grizzlies are spaced at 24.6 ft (7.5 m) along the production drift. Rail spacing for the grizzlies is 13.5 in. (343 mm). Vertical coned-out raises from both sides of the grizzly are driven to the undercut level. These cones have an opening or drawhole at the grizzly elevation of 5 by 6.6 ft (1.5 by 2.0 m) and widen at the top to a cone 11.5 ft (3.5 m) in diameter at the undercut level.

The undercut drifts are at an elevation 26 ft (8 m) above the production drifts and are driven parallel to the production drifts. A pattern of 2.5-in. (63-mm) diameter longholes are drilled at 5-ft (1.5-m) centers along the undercut drift. The total height of the undercut after blasting is designed for 26 ft (8 m). Blasting is done on a daily basis ranging from one to eight blasts per day.

The broken ore from the drawpoints passes through grizzlies for sizing and into orepasses of 5 by 5 ft (1.5 by 1.5 m) cross section to the transfer level. The transfer level is used to control the flow of ore to the main orepasses. From the transfer level, the ore is transferred to orepasses that have a cross section of 6.5 by 6.5 ft (2 by 2 m) to storage bins on the haulage level. The ore is then loaded to rail cars and transported to the concentrators.

The primary rock in Teniente 4 South and Teniente 4 Fortuna area has a much lower fracture frequency than the secondary ore in Teniente 1 South and Teniente 4 North. The full gravity system is not suitable for this harder rock and larger fragment size; therefore, a new system of mining was necessary.

Mechanized LHD System—This system requires four main levels (Fig. 20.3.12): undercutting level, production level, rock-breaker level, and haulage level. In Teniente 4 South, the distances between levels are

undercut to production	59 ft (18 m)
production to rock-breaker level	105 ft (32 m)
production to haulage	213 ft (65 m)

The effective draw area for each drawpoint is 2798 ft^2 (260 m^2). In the Fortuna mine, the distances are less:

undercut to production	52 ft (16 m)
production to rock-breaker level	98 ft (30 m)
production to haulage	141 ft (43 m)

The effective draw area per drawpoint is 2100 ft^2 (195 m^2).

Fig. 20.3.13 shows a typical production level layout for the LHD system. The drawpoint crosscuts are driven at an angle of 60° to the centerline of the production drifts. The distance between the production drifts is 98 ft (30 m), and the distance between drawpoint entries is 56.8 ft (17.3 m). The drawpoints are well supported with steel and concrete (Fig. 20.3.14).

Blasting of the undercut to initiate caving consists of three steps:

1. An initial separation slot is formed by drilling 3-in. (76-mm) diameter vertical holes 42.6 ft (13 m) long. This double row of holes, spaced 6.5 ft (2 m) apart, when blasted forms a line of weakness to assist caving action and as an expansion chamber when the undercut holes are blasted.

2. Drilling and blasting the drawbells through which the caved ore flows to the drawpoints. These 2.5-in. (63-mm) diameter holes are drilled in a series of fan-shaped patterns from the drawpoint crosscut on the production level. When blasted sequentially, a drawbell is formed approximately 39 by 49 ft (12 by 15 m) at its top and 49 ft (12 m) high. This insures that large rock fragments can move down near the drawpoints for easy access for secondary breakage.

3. From the undercut drifts, a fan-shaped pattern of drillholes 2.5-in. (63-mm) in diameter are drilled. In Teniente 4 South, these fans may have 25 holes, whereas in the Fortuna area, only 17 holes are drilled. The total number of holes is adjusted according to the rock mechanics of the area. Fig. 20.3.15 shows the drill pattern and explosives load for one area of Teniente 4. The blasting of these drillholes is the final step in initiating the cave. Fig. 20.3.16 shows the total LHD system.

Production units are 5- and 6-yd^3 (3.8- and 4.6-m^3) LHDs. The LHDs tram the ore to orepasses that are spaced at 197- to 328-ft (60- to 100-m) intervals along the production drift. The ore drops to the rock-breaker level where the fragments are reduced to – 30 in. (– 762 mm). Reduction of the oversize material is with mechanical rock breakers. From the rock-breaker level, the ore moves through orepasses to the haulage level where it is stored in bins for loading into haulage trains.

Although the primary ore is much more competent than the secondary ore, efficiency with the mechanized system has proven to be much better than the full gravity system. A comparison of mine efficiencies is shown in Table 20.3.2.

HAULAGE AND CRUSHING SYSTEM. There are five different train systems at Teniente:

Teniente 1 — 30-in. (762-mm) gage track,
15-car trains, 10-ton car capacity
Teniente 5 North — 30-in. (762-mm) gage track,
15-car trains, 20-ton car capacity
Teniente 5 Fortuna — 30-in. (762-mm) gage track,
15-car trains, 20-ton car capacity
Teniente 5 Central — 56.5-in. (1435-mm) gage track,
12-car trains, 28-ton car capacity
Teniente 8 — 56.6-in. (1435-mm) gage track,
18-car trains, 100-ton car capacity

The crusher for treating oversize ore, mainly from the LHD areas, is a 54- by 74-in. (1.37- by 1.88-m) gyratory crusher. Its capacity is 2866 tph (2600 t/h).

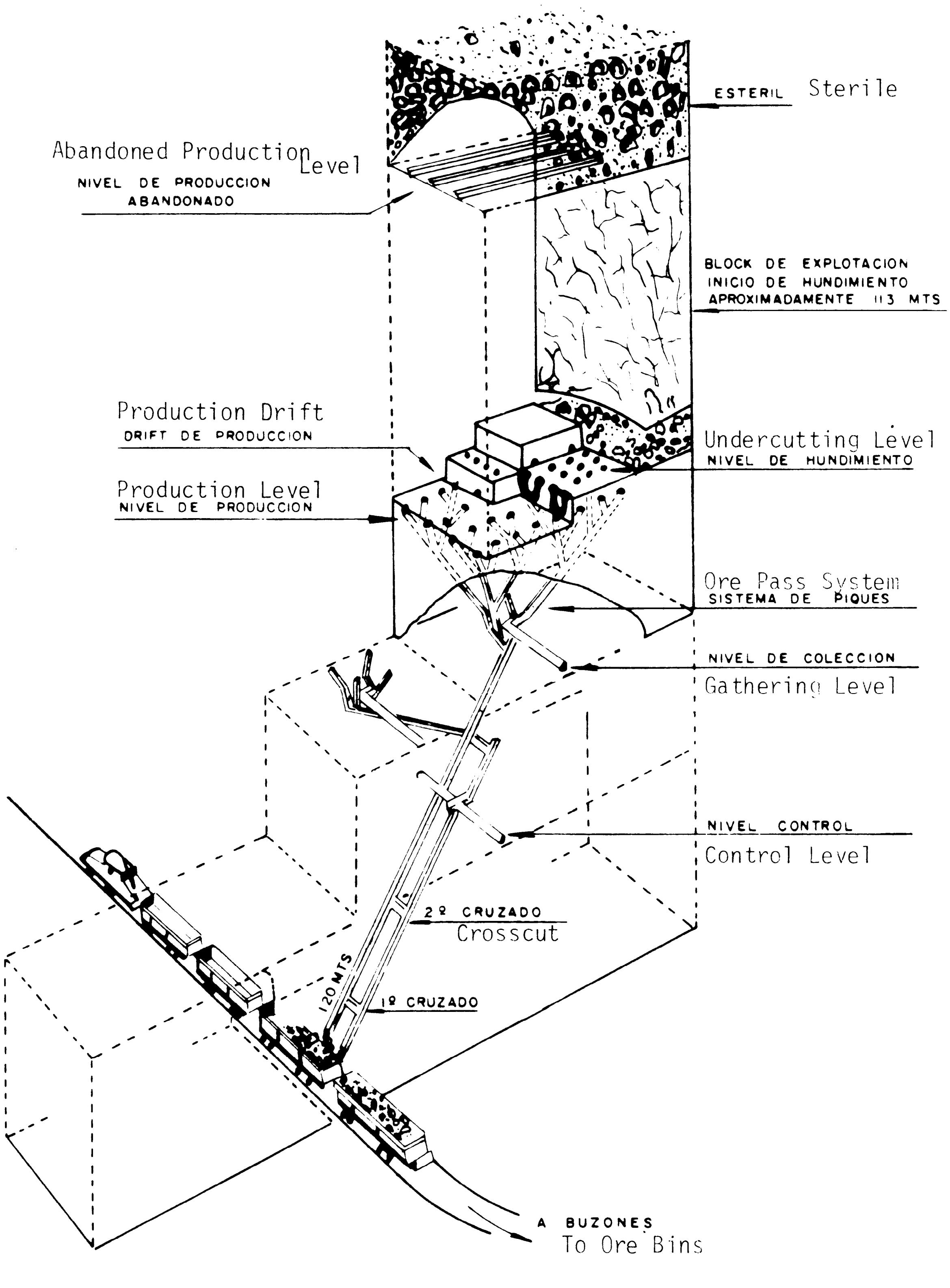

Fig. 20.3.11. Isometric of a manual caving block — El Teniente.
Conversion factor: 1 ft = 0.3048 m.

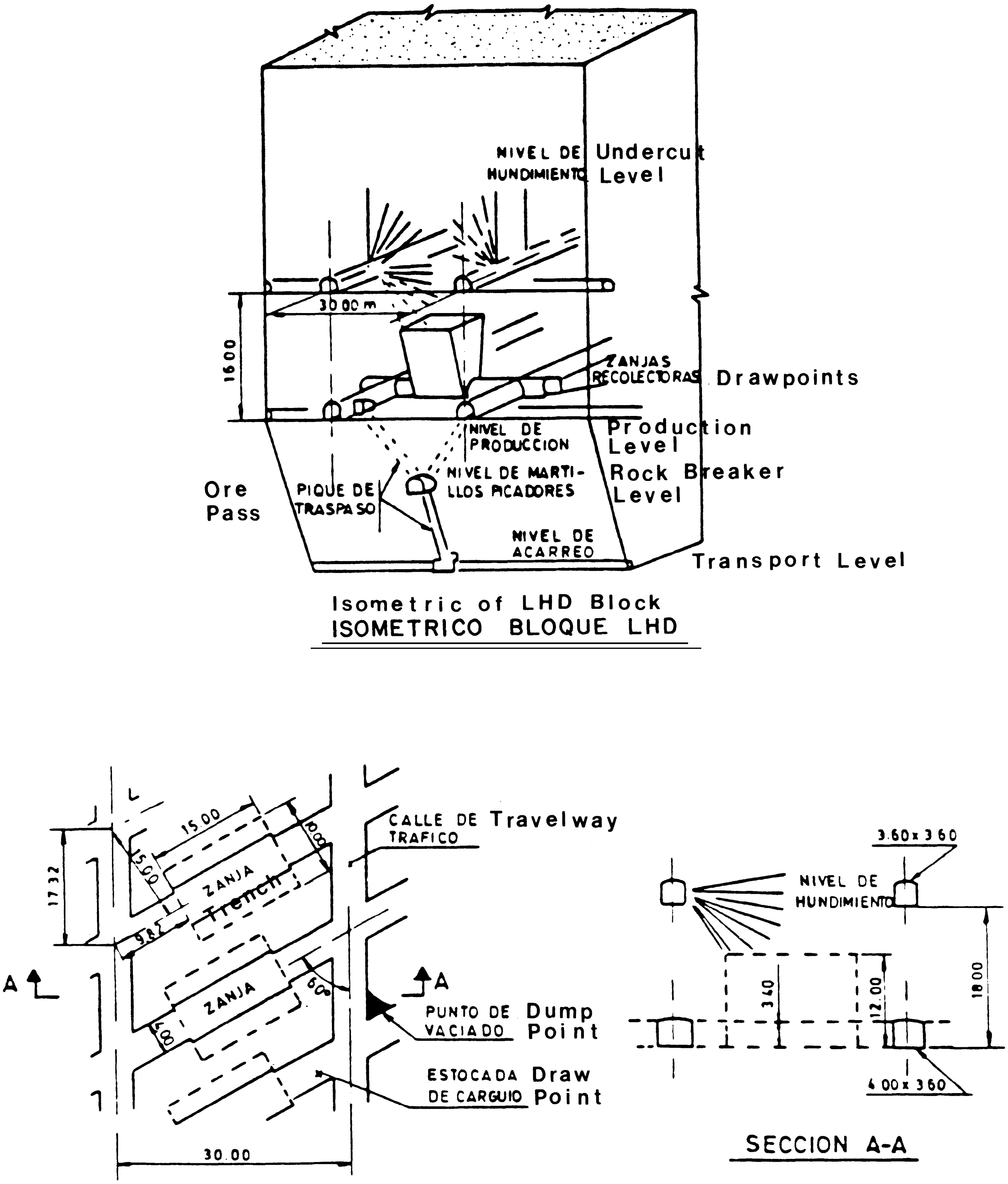

Fig. 20.3.12. Isometric of LHD block, plan of production block. Conversion factor: 1 ft = 0.3048 m.

VENTILATION. The main ventilation system of El Teniente mine is composed of several independent systems that supply the production and haulage areas. Distribution and air return is carried out through sublevels and overlevels developed exclusively for ventilation. There are 14 fans operating currently, six of which are used as intake fans and eight for exhaust air. Together, they move approximately 2,460,000 cfm (1160 m^3/s).

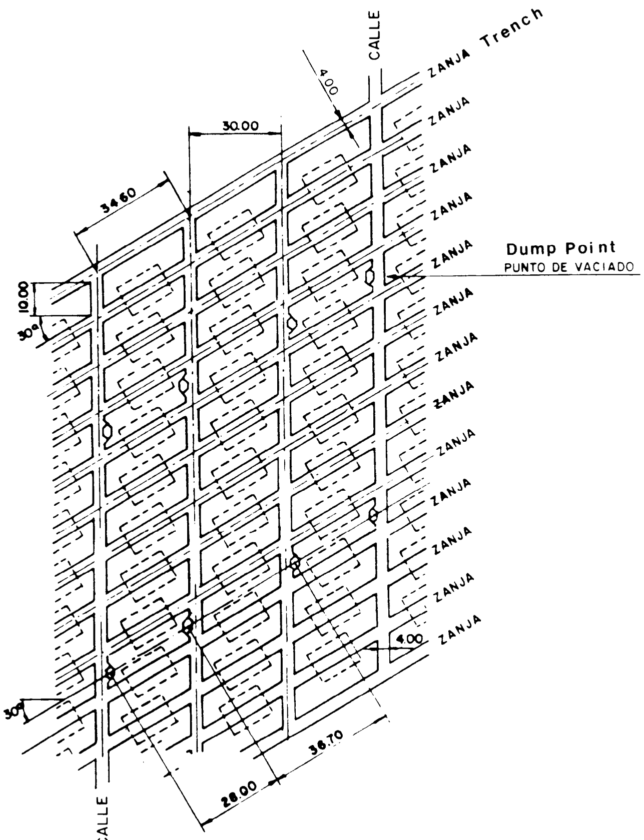

Fig. 20.3.13. Production level layout — El Teniente. Conversion factor: 1 ft = 0.3048 m.

20.3.8.2. Ertsberg East Mine, Freeport Indonesia, Inc., Irian Jaya, Indonesia*

INTRODUCTION AND ORE BODY. The Ertsberg East underground mine is located in a remote area of the province of Irian Jaya (formerly Dutch New Guinea) in the Republic of Indonesia, approximately 75 miles (120 km) inland from the Arafura Sea. The mine is in rugged mountainous terrain, with the highest surrounding peaks exceeding 16,400 ft (5000 m) above sea level. The mine offices, shop buildings and mine portal are situated at an elevation of 11,800 ft (3600 m).

The Ertsberg East (GBT) ore body is a deposit of copper-rich garnet/monticellite skarns that occur along a major east west regional structure known locally as the hanging wall fault zone (HWFZ). This structure, which is the controlling feature of the ore body, is a high-angle (60 to 80°) normal fault interrupting a 3.2 m.y. quartz diorite intrusive. The quartz diorite intrusive served as the heat pump/source rock for the skarning event and is the southern boundary of the ore zone. The upper reaches of the deposit are exposed on the side of a high-relief, south-facing slope on an upthrown block of the Jayawijay mountain range.

The general geology of the deposit can best be described through a north-to-south cross section (Fig. 20.3.17). The HWFZ waste units on the north side of the fault consist of a sedimentary sequence of limestones interbedded with quartz-rich siltstone layers. These sedimentary units were metamorphosed by the skarning event to a series of interbedded marbles and quartz hornfels. The metamorphosed units dip steeply to the north. The ore-bearing portion of the HWFZ consists of two units: the first is a calcite-silica cemented breccia, and the second is a discontinuous highly altered clay.

The ore zone is divided into a skarn section on the footwall side of the HWFZ and a breccia within the hanging wall fault zone itself. In roughly descending order of overall abundance, the metallic minerals are bornite, digenite, chalcopyrite, chalcocite, idaite, covellite, marcasite, valleriite, and pyrite. The skarns additionally consist of a variable mix of garnet, monticellite, diopside, forsterite, calcite, and, in places, magnetite, while the HWFZ breccias are calcitic adjacent to the skarns and clay altered near the hanging wall (Fig. 20.3.18).

HISTORY

Original Discovery and Selection of Mining Method—An exploration program started in 1975 first located the Ertsberg East deposit. In 1977, an exploration adit was collared. Driving of drifts during the exploration phase revealed that most of the ore body would not stand unsupported over a broad span. By the end of 1978, data assembled indicated the ore body might be mined using caving techniques. Mineralization definition drilling indicated that minable widths could range from 197 to 394 ft (60 to 120 m) and more, depending on cutoff grade. The ore body is bounded by a sharp fault cutoff to the north. The southern boundary is an economic cutoff. Two methods of extraction were considered: block caving and sublevel caving. An analysis of the two methods was undertaken and minable reserves calculated for each, along with manpower and operating costs. Analysis of the two alternatives showed block caving to be the preferred method.

Review of the rock mechanics data supported the conclusion that the rock was cavable and further revealed that the broken ore would be composed of better than 50% fines. In the northern section of the ore body, in the breccia zone, the rock was projected to break easily producing the majority of the fines. The southern zone was expected to be more difficult to break but was predicted to follow the breccia. These conclusions led to the decision to use a conventional slusher scraper system.

Mine Development—Analysis of the geologic samples obtained (core samples) indicated the ore would break into very fine particles. This dictated closely spaced drawpoints with an area of influence of 17.1 by 17.1 ft (5.2 by 5.2 m) to insure maximum ore recovery. Drawpoint width was set at 4.9 ft (1.5 m). Slusher lines were spaced at 32.8-ft (10-m) intervals. Six lines made up one block (123 by 196.4 ft) (37.5 by 60 m) and four blocks made up the initial 246 by 393.7 ft (75 by 120 m) area of undercut. Primary development was with rail-mounted equipment. The slusher drifts, haulage drifts, and other areas needing ground support were supported by up to 1.6 feet (0.5 m) of monolithic concrete.

The undercut level 32.8 ft (10 m) above the slusher level was developed by driving a series of crosscuts in the block and then extracting pillars using jackleg drills. Ore was drawn from the cave area via finger raises and moved by slushers to grizzlies equipped with 14-in. (356-mm) spacings (Fig. 20.3.19). Transfer raises beneath the grizzlies held about 40 tons (36 t) each (Fig. 20.3.20). The ore was loaded into 8-yd^3 (6-m^3) side-dump cars through center-loading chutes spaced at 32.8-ft (10-m) intervals. Initially ore was hauled to the surface by trolley locomotives and later by diesel locomotives.

OPERATING EXPERIENCE

Start-up—The initial mining area of 246 by 394 ft (75 by 120 m) was subdivided into four 123- by 197-ft (37.5- by

* Contributed by Terry Owen, Freeport Indonesia, Inc.

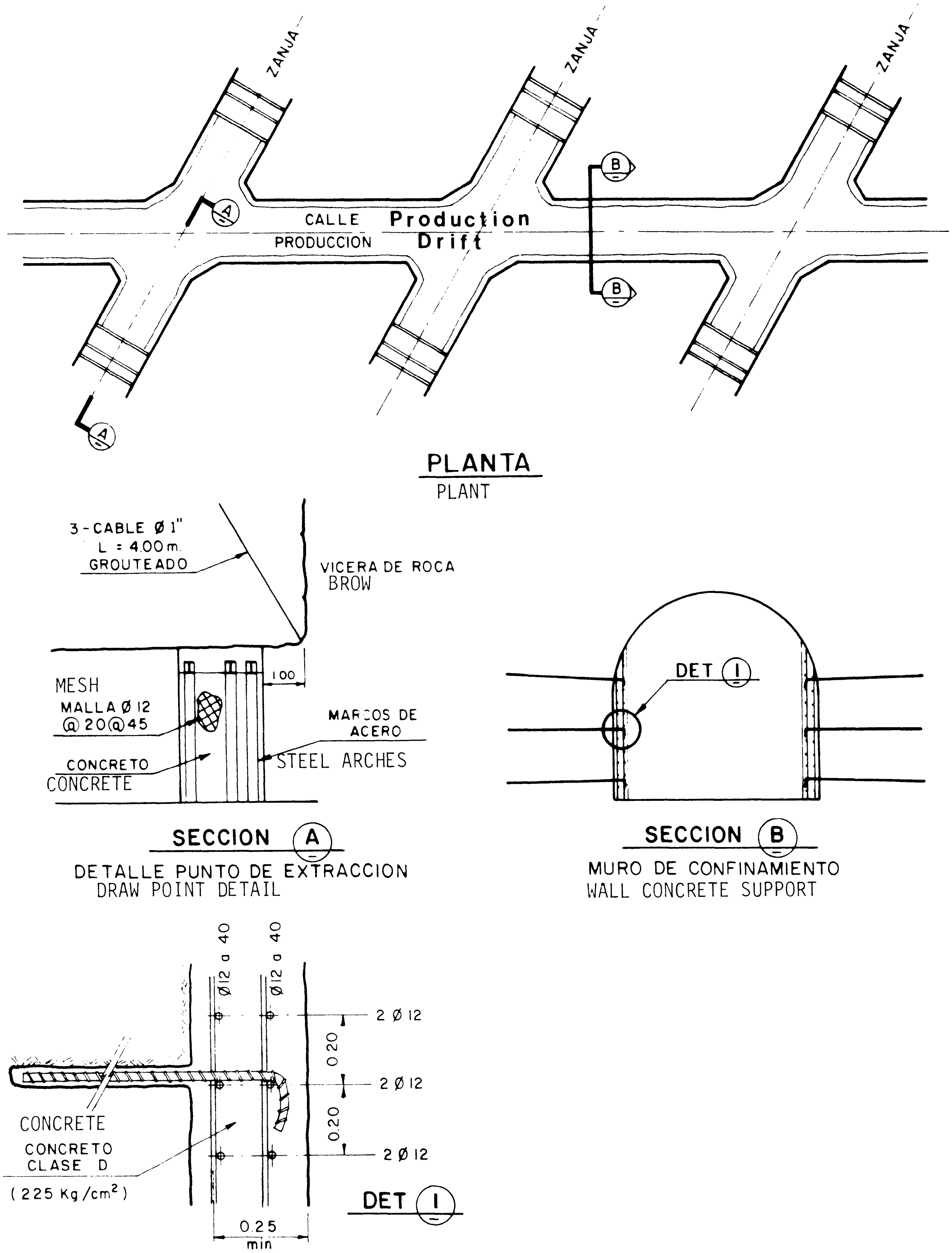

Fig. 20.3.14. Production level support.
Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 psi = 6.895 kPa.

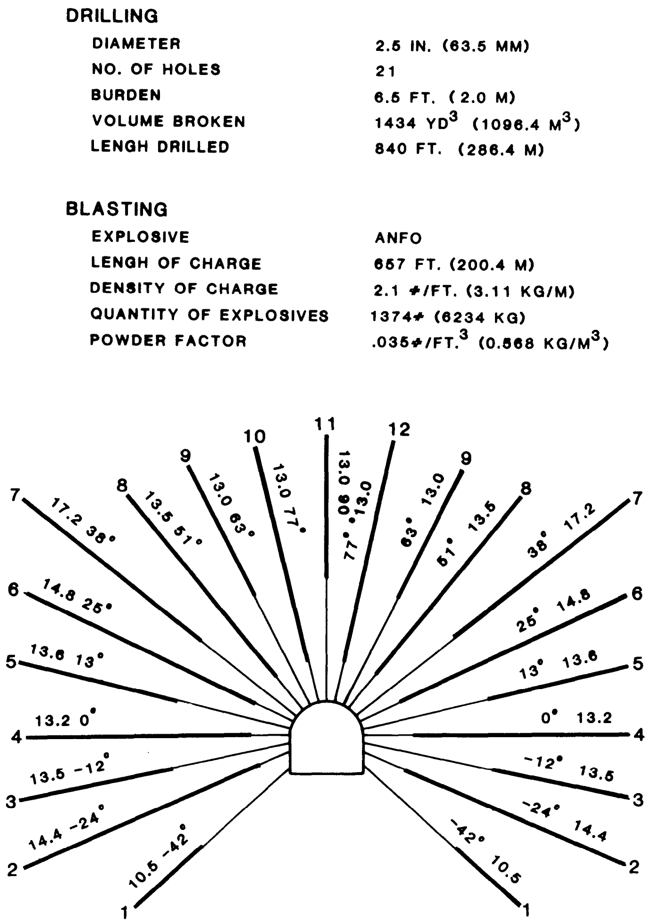

Fig. 20.3.15. Longhole drilling pattern.

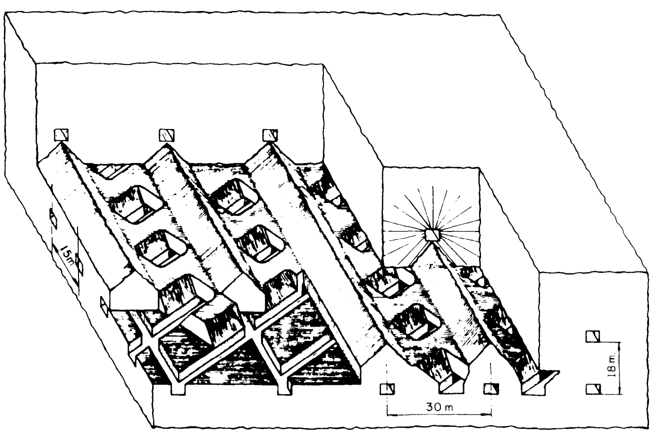

Fig. 20.3.16. Isometric of LHD system — El Teniente. Conversion factor: 1 ft = 0.3048 m.

60-m) blocks. This was determined to be the minimum block size necessary to induce caving action and facilitate the desired production draw of 4960 tpd (4494 t/d).

Boundary weakening was conducted on two sides of the block. Actual undercutting commenced in November 1980. By

Table 20.3.2. Mine Efficiency Indicators

Full Gravity Method	
Productivity (employee)	200
Production (day)	33,000
Available open area (m²)	56,000
Extraction velocity (m²/day)	0.49
Exploitation rock	secondary andesite
RQD (rock quality designation)	Below 12%
LHD Mechanized Method	
Productivity (employee)	850
Production (day)	54,000
Available open area (m²)	67,000
Extraction velocity (m²/day)	0.50
Exploitation rock	primary andesite
RQD (rock quality designation)	Over 60%

Conversion factors: 1 ft² = 0.0929 m², 1 ton = 0.9072 t.

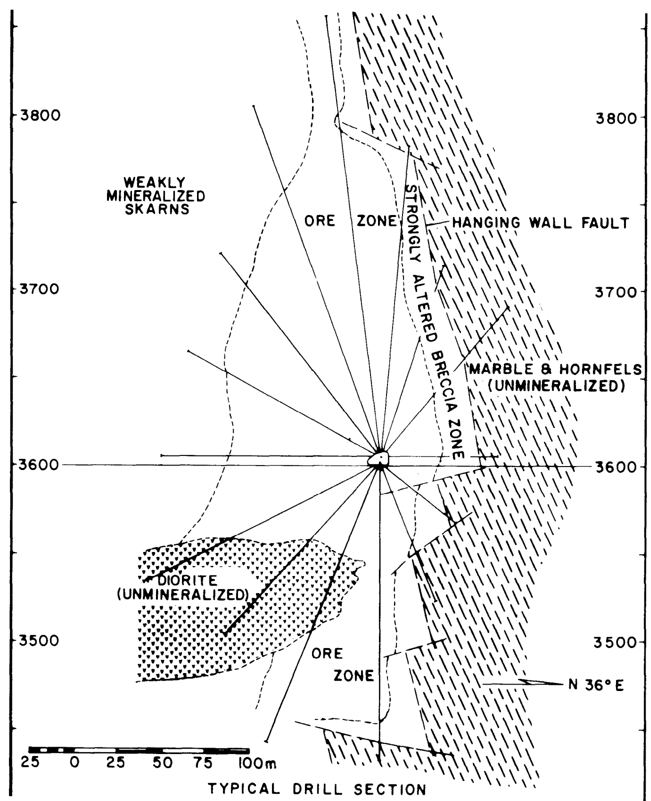

Fig. 20.3.17. Typical drill section.

April 1981, the first four sub-blocks were undercut (Fig. 20.3.21).

Problems—Almost from the beginning, the ore did not cave as planned. By the end of the first six months of production, it became obvious the system had serious problems. Although the production goals were being met, the cost was high. During the caving startup, numerous boulders were encountered, especially in the southern drawpoints, resulting in excessive secondary blasting.

The north side had its own set of problems:

1. The northern drawpoints were encountering boulder hangup problems.

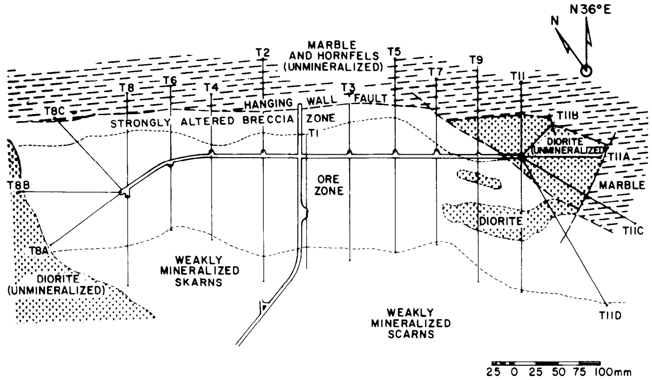

Fig. 20.2.18. Generalized 3600 level plan.

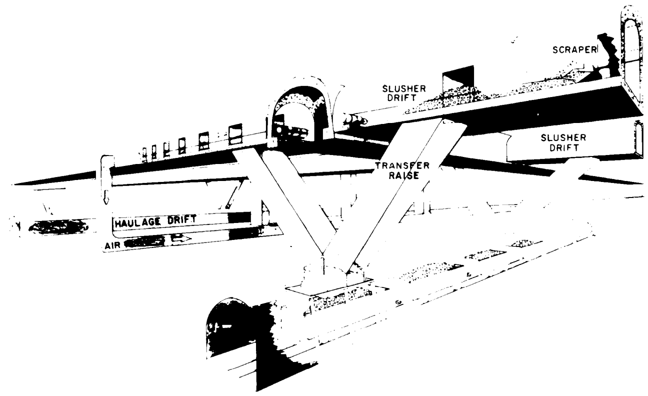

Fig. 20.3.20. Ore transfer raise system.

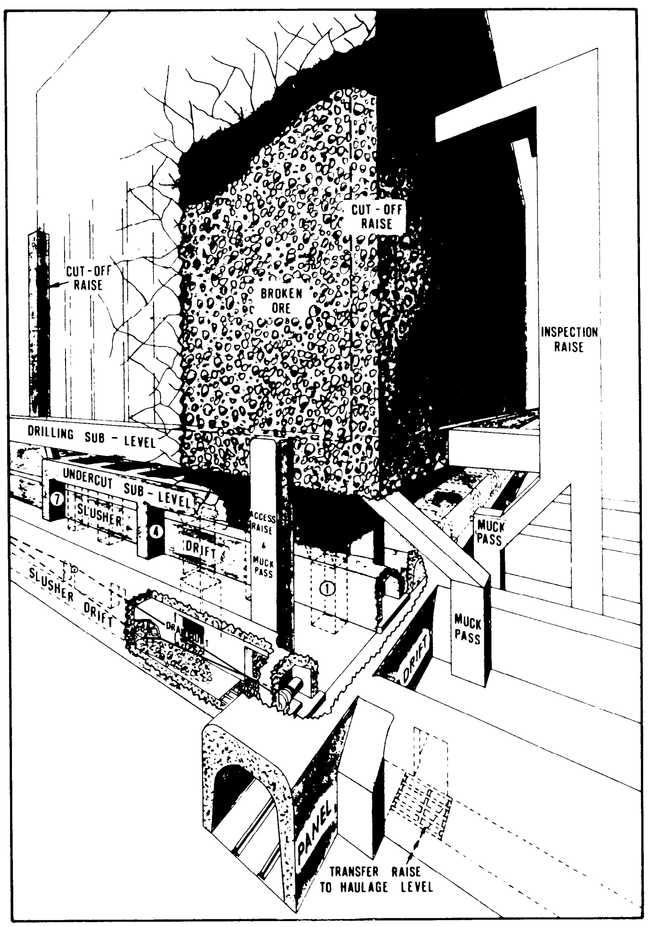

Fig. 20.3.19. Copper mining block caving.

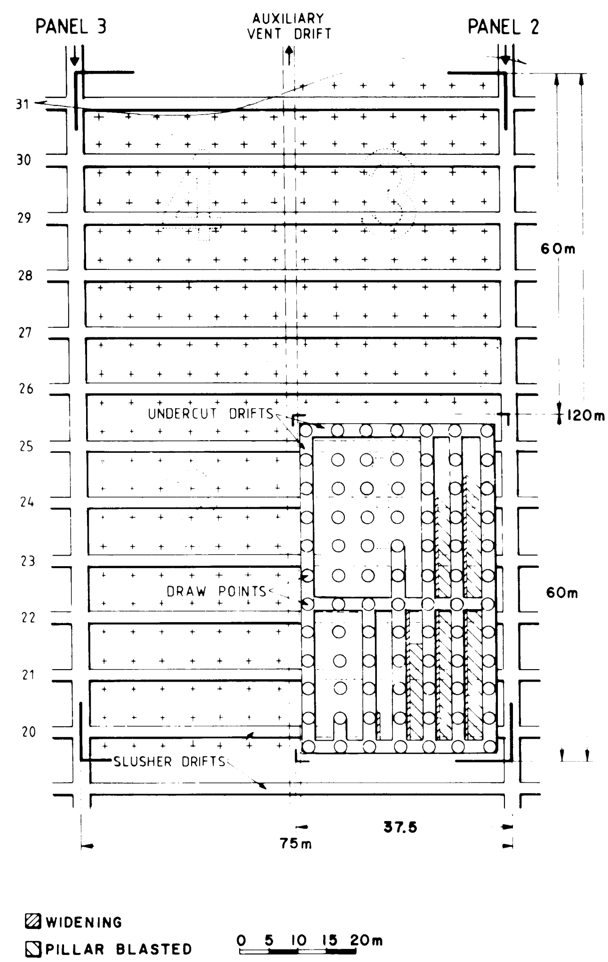

Fig. 20.3.21. Proposed stope plan at undercut level. Conversion factor: 1 ft = 0.3048 m.

2. The breccia was slow to cave, and if the drawpoints were not drawn continuously, the ore packed. The packing caused weight to be shifted on to the drawpoints. Even though the slusher drifts had up to 18 in. (460 mm) of reinforced concrete, weight on the drawpoints became so excessive that one slusher drift was lost for six months.

In November 1981, it was discovered the cave was not propagating as predicted and a void had developed above the cave area. Investigatory diamond core holes were drilled around the perimeter of the cave. Evaluation of the gathered data indicated:

1. The cave had arched, reaching an almost stable condition.
2. The cave back was highest on the south side.
3. The skarn area of the deposit was fragmenting into extremely large boulders, and without proper caving action to

assist breakage, the drawpoints were experiencing severe weight problems from these boulders.

Several problems had to be addressed at once:

1. Drawpoints were too small. Instead of encountering the fine muck predicted, they were being asked to handle numerous large boulders.

2. Close spacing of the drawpoints coupled with the minimum height between levels was allowing the uneven weight distribution from the massive boulders to cause ground failures around the drawpoints.

3. Repair costs due to weight problems and secondary blasting was excessive.

4. Excessive secondary blasting and the lack of cave propagation was severely limiting production.

5. The haulage system was experiencing problems of its own due to the type and number of chutes being in use below the slusher extraction levels.

6. The overall operating costs were running 50% above plan and were rising.

DESIGN CHANGES

Slusher Blocks—Several changes were implemented to solve these problems:

1. The next slusher block developed (block 5) incorporated a drawpoint that was 25% larger in an attempt to ease the boulder problem. The distance between drawpoints was enlarged to increase pillar size.

2. The undercut level was raised 7 ft (2.1 m) to a total of 40 ft (12.1 m) to protect the drawpoints during undercutting.

3. The lower 16 ft (4.9 m) of the drawpoints was concreted to better protect them during the undercutting and secondary blasting phases.

4. A cave inducement program was initiated. Initial efforts concentrated on the few areas that could be reached from the perimeter of the cave on the existing level.

5. A new adit on the 3780 level was collared to permit better access over the top of the cave for additional inducement.

LHD Block 6—Even as these steps were being implemented, it was recognized that major changes must be made to get a better fit between the ore characteristics and the extraction methods. The operating philosophy of Freeport Indonesia (FI) played an important role in the events that followed:

1. Key people within the organization came from a variety of mining backgrounds, including both caving and non-caving operations.

2. FI management was open to suggestion and changes. FI management demonstrated a continuing confidence in the operating staffs ability to overcome the caving problems and garner productivity improvements.

These two factors encouraged the staff to explore nontraditional ideas, often quite unfamiliar to many of the operators.

Time was a critical element at this point. If this project was going to survive, it was imperative that new methods of extraction be elected and implemented quickly. Only a minimum of time was available to perform evaluations.

In order to handle the large boulders with any efficiency, the drawpoint openings had to be at least doubled in size. Additional pillar support around each drawpoint and between the extraction and undercut levels was also needed. Drawpoints needed to be isolated individually for blasting or repair without shutting down an entire area. Transferring of the ore from the drawpoint to the haulage trains had to be simplified.

From these discussions, a concept evolved to develop a test block (block 6) using LHD equipment. A design was laid out that made best use of existing technology. Drawpoint dimensions were widened to 41- by 41-ft (12.5- by 12.5-m) spacing with 12-ft (3.66-m) width drawpoints. The undercut was raised 10 ft more (3 m) to a total of 49 ft (15 m). This permitted drawbells with 60° slopes (the minimum muck run angle), yet still provided sufficient pillar support between drawpoints. In addition,

1. 3-yd^3 (2.2-m^3) development loaders already on site were "borrowed," and two additional LHD units were ordered to complete the test.

2. Drawbells were formed using existing bar and column mounted drills.

3. The undercut pillar extraction was accomplished using jackleg drills.

4. Block 6 was located near to an existing rail haulage orepass to keep the loaders isolated from the slusher drifts.

RESULTS. Second generation results yielded the following:

Slusher Blocks—Block 5 slusher changes did little to improve productivity. Drift support was improved, but boulders were still a severe problem requiring substantial secondary blasting.

LHD Block 6—In contrast, completion of Block 6 (LHD test block) proved to be a success.

Virtually all the boulders passed through the drawpoints. Those that did not were easily accessible for drilling and blasting. Because of this accessibility and the more robust drawpoint size, blasting damage was reduced significantly.

Due to increased pillar size, boulder-weight damage to the drifts and drawpoints was minimal.

POST LHD INTRODUCTION. During the development of LHD block 6, it became apparent that this system had the flexibility necessary to handle the problems encountered to that point. Thus additional LHD development blocks were laid out. All caving blocks since the successful completion of block 6 have been developed as LHD blocks. Table 20.3.3 shows improvements to the underground operating efficiencies in the years since the change. Each succeeding block caving process has been additionally modified, incorporating new techniques that have been identified as increasing the productivity of the ore extraction process or providing easier development of the block (Fig.

Table 20.3.3. Productivity Data—Underground Mine Ertsberg East Mine

Tons Mixed per Full-time Underground Employee

Department	Month of September	Underground Employees	Tons Mined	Tons/ Employee	% Increase Over 1982
Operations	1982	504	143,961	286	—
	1983	528	266,419	505	77
	1984	619	365,195	590	106
Maintenance	1982	184	143,961	782	—
	1983	225	266,419	1,184	51
	1984	229	365,195	1,595	104
Other	1982	15	143,961	9,597	—
	1983	9	266,419	29,602	208
	1984	17	365,195	21,482	124
Total Underground	1982	703	143,961	205	—
	1983	762	266,419	350	71
	1984	865	365,195	422	106

Pounds of Copper in Concentrate per Underground Employee

September	Pounds	Underground Employees	Pounds/ Employee	% Increase Over 1982
1982	8,412,138	703	11,966	—
1983	12,755,538	762	16,740	40
1984	14,318,187	865	16,553	38

Conversion factors: 1 lb = 0.4536 kg, 1 ton = 0.9072 t.

20.3.22). Ultimately, these changes have resulted in lower costs per ton produced. From 1982 to 1983, underground production costs per ton mined dropped 77%. Through 1984, the cost was lowered by 146% compared to 1982. The table shows the relationship to productivity on a per-ton basis during this period.

LHD SYSTEM. Present operations include two operating block caving areas within the Ertsberg East mine and a new block caving development called the DOM mine. While development and production systems in these three areas vary according to geologic conditions and rock mechanics requirements, they all have some common elements.

Undercutting—The original room and pillar undercutting system was slow and arduous and generally resulted in severe weight transfer that damaged the extraction drifts during the undercutting phase. A new method called the trench undercutting system has been developed to permit the undercutting to be accomplished using large down-hole hammer drills and permit development of undercutting and production drifts simultaneously.

In this system, trench drifts are driven on the extraction level between the planned extraction drifts. Simultaneously, 60 ft (18 m) above the extraction level, on the undercut level, drill drifts are driven directly over the extraction drift locations. Then using down-hole hammer rigs drilling 5- to 6-in. (130 to 150 mm) holes, a fan pattern is drilled that, when blasted, will open up a V slot. The trench drift forms the bottom of the V, and the drill drifts form the top edges of the V. Blasting starts at one end of the drill drift, the end where the V slot has been created. The swell muck from the blast is mucked out of the slot from the trench drift. All drilling and blasting take place on the undercut level leaving the trench drifts clear for mucking. Currently, face advances of 32.8 to 42.6 ft (10 to 13 m)/week per drill machine are obtained drilling 394 ft (120 m) per fan with 8.2-ft (2.5-m) spacing. Approximately 1000 tpd (900 t/d) of muck is produced from each fan during undercutting. Once the undercutting has been advanced there is a destressed inverted V pillar left over the extraction drift.

Draw—During the evolution of the LHD system, different draw configurations were tested. The initial draw pattern was a 41- by 41-ft (12.5 by 12.5-m) pattern with the drawpoints directly opposite each other. Staggered drawpoint alignment along with enlarged drawpoint areas of influence have been tested. It was found that behavior of the caving rock dictated the drawpoint area of influence. Enlargement of the drawpoints area of influence to 45.9- by 52.5 ft (14- by 16-m) allowed larger drifts and drawpoints. This in turn allowed larger loaders (5 yd^3 or 3.8 m^3 capacity), thus increasing productivity. Minimum drawpoint width is now 11.8 ft (3.6 m), with 13.1 ft (4 m) being used in the areas where 5 yd^3 (3.8 m^3) loaders are working.

Ground Support—Monolithic concrete is the primary ground support. Tests have been conducted using shotcrete but, due to the varying ground conditions, shotcrete has not proven practical. Future efforts focus on increasing the pillar size within the production area as well as between production drifts. Cable bolting of the production drifts is also being tested.

SUMMARY. Productivity improvements and cost reductions continue to play an important role in the operation at Freeport Indonesia. The Ertsberg East mine could not have achieved its success without the technological improvements that have been incorporated to achieve the productivity and cost reduction. The project now ranks as one of the lowest-cost operations in the world.

A critical factor in the success of this program was FI management support. Senior management's commitment to the operators involved, in giving them full management support and as the operators implemented the major revisions to the initial mining plans, helped ensure the final success of the changes.

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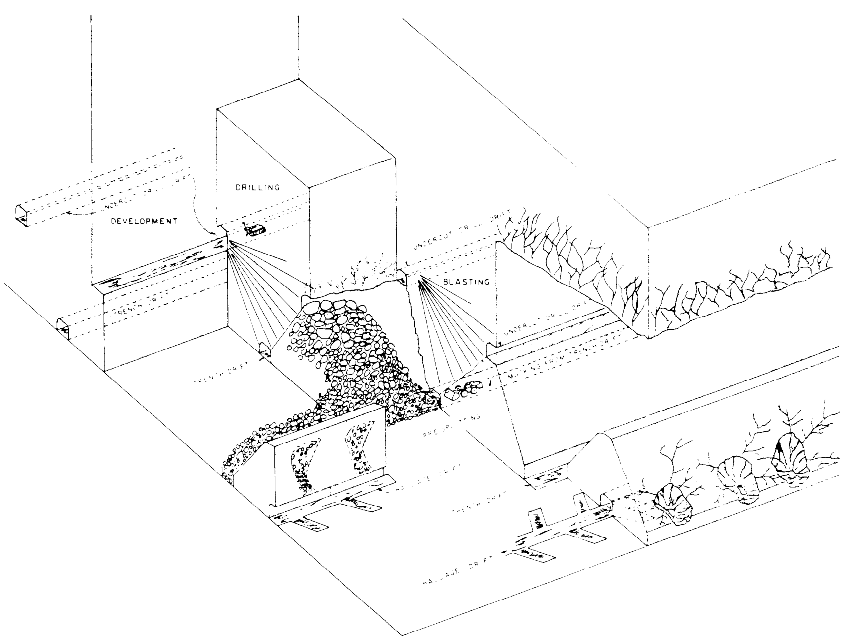

Fig. 20.3.22. Trench undercut system.

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Section 21 Underground Mining: Comparison of Methods

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Chapter 21.0
INTRODUCTION

STEPHEN A. ORR

Underground mining is a dynamic art and science with a diverse set of design, production, and economic criteria that must be considered before selecting an optimal method to extract the ore. While all methods can be technically classified as either self-supported, supported, or caving, the multitude of subsystems and existing variations bear witness to the fact that each ore body is unique and, as such, justifies an individual approach to optimize mineral extraction. In this sense, underground mining is an art. The ability to evaluate the unique characteristics of each ore body and utilize sound engineering principles to design the optimal system constitutes effective method selection.

The task by no means ends at this juncture, however. Subsequent implementation requires an ability for modification. Very rarely are all the facts available to make a concrete decision prior to development of the resource. As a result, the risk of a wrong decision can never be eliminated. This requires a significant degree of flexibility with each method. Each unit of exploration, development, or exploitation work contributed to a deposit enhances knowledge of the ore body. The best mining methods are those that correctly account for the known parameters but provide an allowance for change at minimal cost and adapt to unanticipated conditions.

Section 21 provides a comparative discussion of underground mining methods and the basic tools for mining-method selection, evaluation, and implementation. The description of selection criteria along with the features of each commonly used method provide a basis for evaluation and design of the best means to mine a resource underground. Steps have also been addressed to define frequently encountered problems with various mining methods.

Due to the distinctive nature of coal mining, a unique group of problems not applicable to underground hard-rock mining can be anticipated. For this reason, Section 21 has been divided into separate hard-rock and coal segments, with the first three chapters (21.1, 21.2, and 21.3) devoted to method selection criteria, features, and a summary for hard-rock mining, and the final chapter (21.4) covering the similar topics for coal mining.

Chapter 21.1
HARD-ROCK MINING: METHOD SELECTION CRITERIA

STEPHEN A. ORR

The selection of a suitable underground mining method for an ore deposit involves consideration of a diverse set of criteria. A multitude of dynamic parameters exist, and their importance cannot be overstated. Mining is somewhat unique as it requires heavy initial capital expenditure in preproduction development prior to generating a positive cash flow. In this respect, there is a significant degree of risk, because the ability to change methods once mining has initiated is difficult and could augment capital expenditures to the point where the project is no longer viable.

All mining methods can be classified as either self-supported, supported, or caving, and the determination of which method is most applicable rests upon evaluation of parameters that can be grouped into seven basic categories (Table 21.1.1):

1. Geotechnical considerations.
2. Mineral occurrence.
3. Ore body configuration.
4. Safety/regulatory factors.
5. Environmental factors.
6. Economic considerations.
7. Labor/political considerations.

Each of these criteria can become the principal determining factor in method selection, but the obvious predominance of one consideration should not preclude careful evaluation of all parameters. Method selection is the fundamental decision made in a mine project, and a proper choice is critical as it impacts virtually all other major decisions. As has often been said, “mines are not discovered; they are made.” The degree of consideration given to complete evaluation of an ore deposit can make the difference between a geologic inventory and a mine.

Table 21.1.1. Underground Hard-Rock Mining Method Selection Criteria

Evaluation Parameter	Considerations
Geotechnical	- Lithology
	- Groundwater
	- Geophysics
	- Ore genesis
Mineral occurrence	- Continuity of ore zones within mineralized strata
	- Occurrence of mineral within ore zone (geologic grade)
	- Economic mineral occurrence within ore zone (mining grade)
Ore body configuration	- Dip
	- Plunge
	- Size
	- Shape
Safety/regulatory	- Labor intensity of method
	- Degree of mechanization
	- Ventilation requirements
	- Refrigeration requirements
	- Ground support requirements
	- Dust controls
	- Noise controls
	- Gas controls
Environmental	- Subsidence potential
	- Groundwater contamination
	- Noise controls
	- Air quality controls
Economic	- Minable ore tons
	- Ore body grade
	- Mineral value
	- Capital costs
	- Operating costs
Labor/political	- Costs, influences

21.1.1 GEOTECHNICAL EVALUATION

Geotechnical considerations in method selection include an evaluation of lithology, groundwater, geophysics, and ore genesis of the deposit. This analysis should occur concurrent with the exploration drilling phase and must include an evaluation of the ore zone hanging wall and footwall host formations and general surface topography. As a result, a percentage of the drilling must provide core. For a detailed discussion of geomechanics, see Section 10.

21.1.1.1 Lithology

Drill core structural mapping provides critical information on faulting, jointing, foliation, shears, and oxidation that could affect the integrity of both ore and host formation zones. As a three-dimensional model of the ore body is built, so must all structural features be included. These become the parameters by which future method selection and subsequent design is influenced. For example, a highly sheared hanging wall would certainly preclude open stoping as a viable option and, depending upon geophysical testing and ore body configuration, could favor sublevel caving as the optimal method. The degree of foliation and oxidation within an ore zone can have a significant impact on the amount of ground support required or whether the ore zone is cavable. Additionally, detailed evaluation of the surface topography from aerial photographs and magnetic surveys will reveal geologic structures, such as major faults or magnetic anomalies. These allow a broad overview of the area and can subsequently be tied to detailed lithologic core mapping.

21.1.1.2 Groundwater

Initial exploratory drilling will usually define the existence of groundwater. It is important to determine water levels within the formation, permeability of the formation, initial flows, and sustained flows. Initial quantification during drilling can be made of the water-bearing strata interval and a selected number of holes then designated for piezometer installation. This will monitor water pressure within the borehole and become an indicator of whether the system is recharging itself. Flows can be measured from drillholes with packers and flowmeters, and evaluation of flows over a period of time will provide data on the system’s recharge rate. Finally, formation permeability can most simply be analyzed by pumping through drillholes into the formation. Flows are monitored along with pumping pressures and rate of pressure discharge once pumping is terminated. Selected formations through which the drillhole has penetrated can be selectively isolated by setting packers in the hole at specified points above and below the strata. These hydrologic studies are imperative to proper design and can certainly influence the choice of mining method. Intersection of consistent flows may dictate a more selective method such as open cut and fill stoping to allow water influx control.

21.1.1.3 Geophysics

Once the drill core has been lithologically evaluated, samples from the ore zone and host formations should be subjected to geophysical testing. Basic data for evaluation includes tensile strength, compressive strength, modulus of elasticity, Poisson's ratio, angle of internal friction, and specific gravity. Additionally, a determination of in situ stress can be made by utilizing the borehole overcoring technique.

The Brazilian disc test is commonly used to determine tensile strength for drill core with the axial-load compression test providing compressive strength. Triaxial testing places a confining load perpendicular to the axial load and allows delineation of the modulus of elasticity and the angle of internal friction. Poisson's ratio can then be calculated from tangential and axial strain.

In situ stress measurements provide a quantification of existing stress within the rock. As a result, holes must be drilled in the formation specifically for this purpose. The data generated becomes a foundation for determining premining stress and modeling subsequent behavior during the mining cycle.

A series of measurements in multiple orientations should be made on laboratory and in situ tests to (1) account for planes of weakness such as jointing or foliation and (2) measure in situ stresses in all dimensions (see Chapter 10.2 for a detailed discussion of rock mechanics testing).

All generated data should be computer modeled using element analysis under a variety of mining scenarios to project areas of high stress concentration and possible rock displacement magnitude during the mining sequence. This will allow optimization of a mining method to the unique geophysical characteristics of the deposit. For example, a method which creates large voids such as open stoping (stope and pillar) or vertical crater retreat (VCR) mining would probably be a poor candidate if high stress concentrations exist or would be generated in the hanging wall. Subsequent failure would severely dilute ore and possibly threaten surrounding mine activity. Alternatively, specific design parameters can be customized to minimize projected stress. Using the preceding example, both open stoping or VCR mining might be feasible methods if the maximum vertical stope height were reduced, stope length along the strike shortened, or cablebolts added to the hanging wall (Figs. 21.1.1 and 21.1.2). Once valid base data are obtained, application of computer modeling allows generation of a multitude of mining scenarios.

Unfortunately, any initial model must make a significant number of assumptions that greatly increase the risk of error. As a result, an active rock mechanics monitoring program during development and subsequent mining is absolutely critical to provide an early warning of impending ground control problems and consistently refine the geomechanics model. Each new bit of absolute data increases the projection accuracy of future mine planning. Strategic installation of extensometers and borehole load cells in the ore body and host formations will provide constant monitoring to quickly delineate stress concentrations, avert potential failure, and collect data for subsequent modeling.

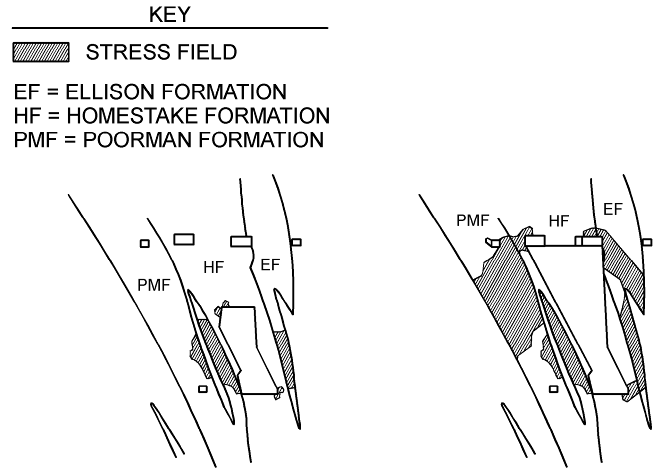

Fig. 21.1.1 Cross section effect of VCR stope height on stress distribution.

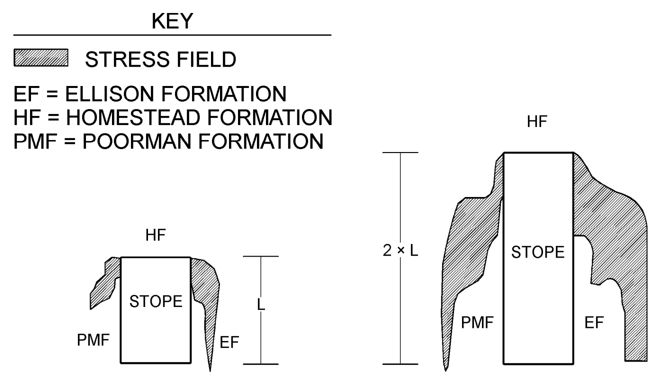

Fig. 21.1.2 Plan view. Effect of VCR stope length on stress distribution.

21.1.1.4 Ore Genesis

Ore genesis evaluation is predominantly applied to deposit exploration for generation of a new ore body or addition of reserves (Sections 4 and 5). In limited respects, however, genesis can become a preliminary mining method evaluation parameter. For example, an epigenetic vein deposit typically contains high-grade ore shoots. optimization of mining would require a selective method which minimizes dilution. Telethermal deposits, formed by hydrothermal fluid migration, usually occur in flat beds with uniform ore occurrence within a well-defined stratigraphic interval. These deposits, typified by the Missouri lead-zinc belt, are particularly amenable to room and pillar or stope and pillar mining. Sedimentary syngenetic deposits, unless they have undergone regional metamorphism, are usually structurally incompetent. This would require a method that minimizes active mine openings and utilizes permanent pillars within the ore zone. Additionally, genesis is an excellent preliminary means to project the mineral occurrence within a deposit.

21.1.2 MINERAL OCCURRENCE

The spatial distribution of a mineral within the deposit can significantly impact the choice of mining method. There is actually a two-fold consideration which constitutes continuity of ore zones within the mineralized strata and the occurrence of mineral within the ore zone. A large mesothermal copper porphyry deposit, in which the ore has permeated rocks as disseminated veinlets and spots, typically has higher-grade cores with a massive mineralized halo zone. Depending upon cutoff grade, this type of deposit is particularly amenable to non-selective bulk mining methods. The geophysical evaluation would determine if block caving, sublevel caving, VCR mining, room and pillar mining, or open stoping should be utilized.

Ore bodies that occur as concentrated chutes or loads, such as those within the Golden Mile of the Kalgoorlie District in Western Australia, require a very selective mining method. This provides significant grade control throughout the mining cycle to ensure that dilution is minimized. In this case, a method such as shrinkage or cut and fill stoping would be appropriate. While these methods are more labor intensive than their nonselective counterparts, they are amenable to mechanization. The hazard that must be avoided is improperly sized equipment for the ore body. Oversizing equipment for a small deposit does nothing more than burden mining with excessive dilution and diminish profits (see Chapter 21.3 for further discussion on common pitfalls in method selection).

The grade of a deposit is a reflection of mineral units entrained within the delineated tonnage. This will normally be termed as both a geologic grade and mining grade. Very rarely will the two correspond since mining grade is profit dependent and consequently always higher. This represents the concentration of mineral units within a given tonnage that can be mined at the desired profit. It will often influence the mining method because deposits with a high-grade core can often be mined at greatest profit by a selective method that ignores lower-grade reserves even though they are economic. Optionally, the method may be designed to allow initial mining of high-grade reserves for quick recovery of project capital then mine lower-grade ore for the duration of mining life (see 21.1.6, Economic Considerations).

21.1.3 ORE BODY CONFIGURATION

The physical parameter of an ore body will often preclude use of many mining methods. Orientation considerations such as dip, plunge, and strike along with the size and shape of the ore body are key evaluation factors. Many methods, such as shrinkage stoping, open stoping, VCR mining, and sublevel caving, depend upon gravity ore flow to extraction points. However, deposits with footwall dips less than the broken ore rill angle will not allow complete extraction with these methods. Generally, ore bodies with dips less than 55° are not amenable to gravity flow. Additionally, hanging-wall dip must be considered in any method with structurally incompetent host rock. The hanging wall will often begin caving during extraction of a gravity system, and as shown in Fig. 21.1.3, waste is soon pulled through the extraction cone, progressively diluting and finally displacing ore. Those deposits with flat footwall dips are best mined with cut and fill or panel longwall methods. Where applicable, room and pillar is certainly preferred for its high mechanization and productivity amenability.

Plunge is the vertical angle component along strike and it affects the number of mineral units per vertical and horizontal linear distance. In this respect, plunge has a significant impact on the amount of development required for each ton of ore. A flat-plunging ore body will ultimately require much more development than its steeper counterpart. Methods such as sublevel caving or undercut and fill stoping may be chosen in this

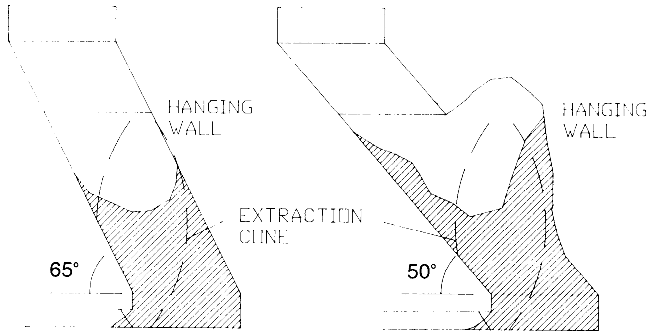

Fig. 21.1.3 Cross section showing dilution incurred in a VCR stope with flat hanging wall.

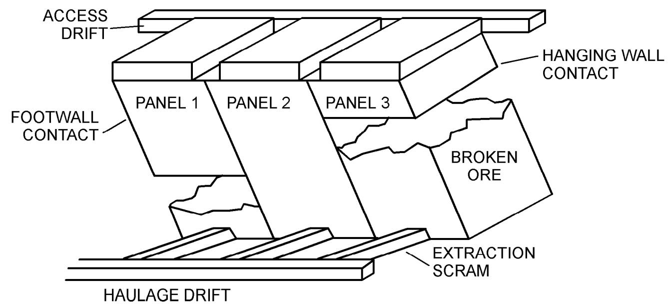

Fig. 21.1.4 VCR transverse panel mining along strike.

instance because mining can progress down-plunge and minimize preproduction development.

The strike or azimuth of an ore body in the long dimension can affect a mining method by its length. Open voids have a critical geophysical length-to-width parameter beyond which they are no longer stable. Open stoping, shrinkage stoping, or VCR mining that creates large openings are often designed in a series of smaller panels to avoid hanging wall failure in deposits with a long strike dimension (Fig. 21.1.4). These panels are then mined and backfilled in an alternating pattern along strike or complete extraction sacrificed and permanent ore-bearing pillars left as support.

21.1.4 SAFETY/REGULATORY FACTORS

The health and safety of personnel is of paramount importance in the selection of any mining method (Chapter 11.1). Irrespective of which commodity is extracted, a mine's most valuable asset is its people. Consequently, a number of safety considerations need to be included in any mining method evaluation. Primarily, those methods that are the least labor intensive offer an inherent safety advantage by minimizing the exposure of personnel to hazards. Replacement of shrinkage stoping with more recently developed VCR mining has been due, in part, to safety benefits derived by elimination of mining work on a broken ore pile in the active stope.

Maximizing mechanization not only increases efficiency and reduces labor requirements but also reduces the physical work effort and resultant occupational injuries. Many previously termed "labor-intensive" mining methods have been adapted to mechanization. Typical cut and fill stoping that utilized jacklegs and slushers has now incorporated rubber-tired jumbos and load-haul-dump (LHD) units. This has reduced instances of back injury, carpel tunnel syndrome, and a multitude of accidents.

Mechanized mining systems do contain unique inherent safety considerations. Increased use of diesel equipment requires special environmental controls such as adequate airflow to effectively sweep exhaust contaminants from the working face and evaluation of heat generation and effect on working-area temperature. Proper airflow is a function of diesel horsepower, and when the requirement cannot be achieved, consideration should be given to electric or pneumatically powered equipment. Additionally, a 100-hp (75-kW) diesel machine adds about 800,000 Btu (844,000 kJ)/operating hour while a comparably powered electric unit contributes only 250,000 Btu (264,000 kJ)/operating hour. Thus mines with a high geothermal gradient must weigh the cost of additional refrigeration against use of electric equipment.

In mines with radon gas emissions, adequate airflow is the best means to dilute gas concentrations to an acceptable level. Radon daughters attach themselves to diesel emission particles or dust. Water is the best and most common means for dust suppression. Exhaust scrubbers or most recently, ceramic filters, are effective controls for diesel emissions. In all cases, respirators are absolutely essential for personnel operating in these conditions.

Virtually all underground methods utilize noisy equipment. Manufacturers have only achieved limited success in reducing equipment decibel levels through engineering controls. To date, personal ear protection still provides the best means to prevent hearing loss from noise exposure.

The problems of elevated dust, gas and noise levels are compounded by the enclosed underground environment. If not properly mitigated, overexposure risks will be incurred with any underground method.

An improper selection of mining method can create serious ground-control-related safety hazards. For example, utilization of block caving in an ore body with poor natural cave characteristics will create bridging problems. Attempts to dislodge hangups expose personnel and equipment to unstable ground, and eventual failure can result in an air blast. Use of open stoping in a deposit with structurally weak host rock can promote massive hanging wall failure, which risks surrounding mine structures and personnel safety. Therefore, all possible safety risks which could be created as a result of a method's application must be anticipated in advance.

21.1.5 ENVIRONMENTAL FACTORS

Most underground mining, except the caving methods, have the distinct advantage of creating a minimal disturbance to the environment. Many underground operations exist below communities, and through proper choice of method and sound operating procedures, surface impact from mining is virtually nonexistent.

Where consideration must be given to minimizing impact on the surface environment, any mining method that could result in subsidence should be avoided (Chapter 10.6). Both block caving and sublevel caving are systems employed in ground where either the ore and/or the host rock will naturally cave once extraction is initiated. These methods usually result in significant surface subsidence. In cases where block or sublevel caving is applicable, geotechnical considerations may preclude use of any other unsupported method. Supported methods must then be evaluated, and depending upon ground competency, either cut and fill or timbered cut and fill stoping utilized. The resultant productivity of the operation will be reduced and operating cost increased. If economically viable however, these methods will allow mining of the resource with minimal environmental impact.

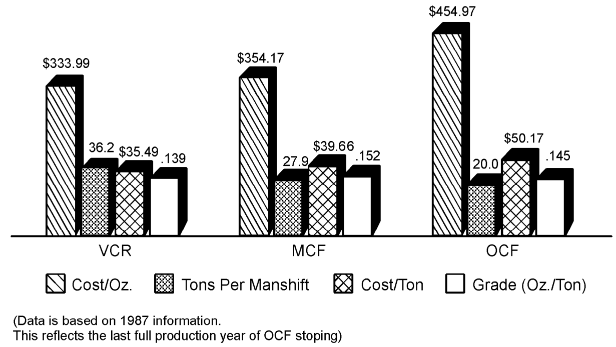

Fig. 21.1.5 Comparative unit cost, productivity, and grade of VCR, MCF, and OCF mining at the Homestake gold mine. Conversion factors: 1 ton = 0.9072 t, 1 oz/ton = 31.25 g/t.

Unlike surface mining, dust or gas emissions to surface air are limited to ventilation exhaust discharge locations (Chapter 11.7.2). Placement of discharge should be downwind from populated areas.

21.1.6 ECONOMIC CONSIDERATIONS

The economic feasibility of an ore deposit is dependent upon the following basic parameters:

1. Minable tons.
2. Ore body grade.
3. Mineral value.
4. Capital costs.
5. Operating costs.

Method selection plays an integral role in these considerations since it impacts all factors except mineral value. As a result, proper extraction method design dictates a project's profit margin, and in this sense, mineral value influences the mining method (Section 6).

An ore body's minable inventory is a reflection of the tons and grade that can be mined at a desired profit. The mining method will significantly influence this inventory by affecting selectivity. For example, open cut and fill (OCF) stoping offers a high degree of extraction control and will optimize the mineral content of every ton mined. Unfortunately, selective methods generate higher operating costs because they are more labor intensive and consequently less productive than bulk methods. This increased cost will often diminish the benefits of optimizing mined ore grade. Fig. 21.1.5 shows a comparison between VCR, mechanized cut and fill (MCF), and OCF mining at the Homestake gold mine. While OCF allowed a 4% increase in extracted grade over VCR, the 55% lower productivity translated to 36% higher operating costs and resulted in an uneconomic method. MCF was specifically introduced to replace OCF as a relatively selective method that employs higher mechanization to reduce operating costs and boost productivity. Table 21.1.2 shows a comparison of relative direct costs of various mining methods.

Normally, a method is chosen that generates a minable inventory to sustain consistent profitable cash flow for the longest period of time (Chapter 2.5). This allows for full project capital recovery and provides cash flow to use for exploration and development of additional reserves. A factor such as market price, however, can adversely affect this philosophy. Commodities such

Table 21.1.2. Hard-Rock Underground Mining Methods Relative Direct Cost Comparison

Method	Relative Cost
Block Caving	1.0
Room and Pillar Mining	1.2
Sublevel Stoping	1.3
Sublevel Caving	1.5
Vertical Crater Retreat Mining	4.3
Mechanized Cut & Fill (w/LHD)	4.5
Shrinkage Stoping	6.7
Conventional Cut & Fill (w/slusher & jackleg)	9.7

as strategic metals and precious metals that are subject to drastic price fluctuations, often dictate initial "high grading" of an ore body to reduce capital recovery time. This is a means to mitigate the risk of market volatility, and if the remaining mineral inventory can still be mined profitably, it may be the most profitable economic scenario. This is true particularly if loans are involved and discounted cash-flow considerations include service of a debt. This practice usually has the inherent disadvantages of sacrificing lower-grade reserves that could otherwise have been blended into a consistent economic grade resulting in the ultimate extraction of more mineral units. Essentially, the goal is to generate the optimal mix between quickest return on investment and highest return on investment.

Choice of a mining method has a significant impact on capital requirements and revenue-generation lead time (Chapter 23.3). Some mining methods, such as block caving, are particularly development intensive and, as a result, require more preproduction capital expenditure and a longer lead time prior to revenue generation. Once in the production phase, however, operating costs are lower since most ore body development is completed. Oftentimes, ore body grade does not allow much flexibility since low-grade deposits require high extraction rates to maintain low unit costs for profitability. This dictates a bulk method such as block caving, sublevel caving, open stoping, or VCR mining. So while shrinkage or OCF stoping requires significantly less lead time prior to production, either method generates lower productivity and higher unit costs since concurrent development is constantly required to maintain ore flow. Accordingly, the choice of OCF or shrinkage stoping to reduce preproduction leadtime could significantly diminish the operation's profitability.

Cutoff grade is a dynamic number affected by commodity value and cost to produce the product. In this respect, each mining method will generate a unique cutoff grade. Calculation of this cutoff when choosing a mining method should be based on all operating costs incurred to produce the commodity. This corresponds to the active cutoff grade utilized in a producing mine. Including capital costs in the calculation generates an unrealistically high number that is not a true reflection of the grade required to produce a sustained desired cash flow for given market conditions. Consideration of capital and operating costs could preclude use of a less-selective caving method and eliminate economic tons from the mining inventory.

21.1.7 LABOR AND POLITICAL CONSIDERATIONS

The choice of any mining method is influenced by the availability and cost of labor (Chapter 8.5). While the preceding discussion has assumed a relatively high labor cost, the economic situation in many countries results in an abundant work force, which is often unskilled. Depressed labor costs make mechanization a less economically attractive alternative in this case. Additionally, governments in these situations may be less profit motivated and more interested in maintaining maximum employment. Economic considerations take a diminished role in favor of providing consistent production. A mechanized method requiring trained personnel may not be a practical alternative since introduction of machinery will probably not reduce the work force.

The political climate in many countries can result in relatively unstable governments. Significant risk associated with these conditions favors a mining method that minimizes capital outlay and provides the quickest return on investment. For instance, sublevel caving may be rejected in favor of cut and fill stoping because of the latter method's ability to generate revenue in significantly less time with less capital investment. In these cases, mitigation of risk on investment becomes a predominant factor in the choice of a mining method.

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Chapter 21.2

HARD-ROCK MINING: METHOD ADVANTAGES AND DISADVANTAGES

THOMAS G. WHITE

21.2.1 INTRODUCTION

Once an underground mineral deposit has been located, conventional wisdom dictates that certain criteria must apply as to whether the deposit is an ore body or just an interesting prospect. Ever since Georgius Agricola used the word *philosophy* to apply to underground mining in 1556, society and miners have had tugs-of-war, further complicating the relevant issues.

The purpose of this chapter is to describe why a certain underground hard-rock mining method may have distinct advantages or disadvantages as compared to an alternative method. Specific case studies are summarized, which is helpful when attempting to relate textbook mining methods to real-world scenarios. The continued rapid improvements and innovations in equipment for drilling, blasting, loading, hauling, crushing, and hoisting have resulted in the acceptance or rejection of specific mining methods and techniques. These equipment advances are discussed as well, with emphasis on how mining methods changed to better utilize the benefits of the newer equipment.

Today there are pressures on underground miners from a vast array of fronts to confuse the technical issues and complicate the selection of mining techniques and project ore body definition. The mining company that does not pay serious attention to environmental concerns, citizenry complaints and interests, land status, and a host of other criteria when making a mining method selection in the last decade of the 20th century is in for a rude awakening.

In 1988, there were an estimated 3000+ underground mines in the world each producing less than 150,000 tpy (135,000 t/a), and some 600 underground mines exceeding that rate. Of the larger tonnage mines, only 25% are in North America with another 25% in Africa. Obviously, method selection for the two continents would employ some non-geologic criteria. Considering as well, Asia, Europe, and Australia with their respective non-technical constraints, final method selection will most assuredly be more than a simple geologic and rock mechanics exercise.

To better understand the features, or advantages and disadvantages, of various mining methods from an overall socio-economic and technical basis, a basic understanding of the predominately used underground mining methods is required. When considering the basic principles employed, there are relatively few mining methods used today: caving methods, naturally supported stopes, and artificially supported stopes.

Since each ore body is unique, and with different environmental and location considerations, the variations of each mining method are limitless. The mining method employed must above all be flexible to deal with the various surprises Mother Nature and the elements spring on the unsuspecting.

Table 21.2.1 summarizes the principal features of the major hard-rock mining methods.

21.2.2 CAVING METHODS

21.2.2.1 Sublevel Caving

Sublevel caving is considered by some to be a variation of top slicing. In top slicing, the ore is removed by driving a series of timbered drifts and crosscuts at the top of the ore body, and when the timbers are removed or blasted, the caved ore is mucked out. Top slicing requires ground so weak that when the supports are removed, the ore readily will collapse. Sublevel caving has been adapted to ores in stronger ground that require drilling and blasting. Since the method does rely upon caving of the walls, the technique is termed a caving technique. Some of the more common advantages and disadvantages of sublevel caving are summarized in the two operational case studies cited below.

KIRUNA. Starting in the early 1950s, LKAB's very large Kiruna iron-ore mine in the arctic north of Sweden has been mined by sublevel caving. In Kiruna's case, there are several advantages to the method:

1. Because the ore body is huge and tabular, large-scale, standardized mechanization is possible.
2. Sublevel caving is well suited to selective mining, which is important because of the varying phosphorus content throughout the mine.
3. The method permits many drifts to remain open, which aids in overall flexibility.

A disadvantage is that planning is complicated, with three major work steps occurring: development, fan drilling and loading, and hauling.

Prior to the advent of sublevel caving at Kiruna, shrinkage stoping methods had been used, but greater tonnages and increased selectivity resulted in sublevel caving techniques being adopted. Rock mechanics techniques leaned toward sublevel caving since the Kiruna deposit is wide and steep with relatively weak hanging walls.

At Kiruna, some other disadvantages of sublevel caving include the high investment cost, relatively extensive development work, and the potential for rock mechanics problems at depth.

Kiruna is one of the outstanding examples of sublevel caving techniques in the world and is an interesting compromise of high-speed haulage, large work force, heavy development requirements, and imaginative engineering practices.

CRAIGMONT. The Craigmont mine in south-central British Columbia, Canada, although now depleted, is another good example of sublevel caving. In 1961, after life as an open pit mine, underground mining methods of blasthole stoping and cut and fill stoping were tried but with limited success. It was decided to adopt the sublevel caving method once the open pit ceased operations for the following reasons:

1. Knowledge of ground conditions was obtained from previous mining.
2. Ground adjacent to the ore was reasonably incompetent.
3. It was more cost effective than block or panel caving because of the irregular nature of the ore body.
4. Sublevel caving did not tie up ore in pillars.
5. A high degree of mechanization could be employed.

In most sublevel caving operations, dilution is the main drawback. Kiruna's large tabular ore body minimizes that problem, while the irregular ore zones at Craigmont accentuate the problem. In an ore body with a reasonably flat dip, sublevel caving yields poor recovery as compared to high recovery from an ore body with a vertical dip.

Table 21.2.1. Summary of Features of Major Mining Methods

Methods	Advantages	Disadvantages
Sublevel caving	• High degree of mechanization is possible • Good selectivity can aid in grade control if ore at near vertical dip • High tonnage and productivity are possible	• High initial investment • Potential for dilution if ore body thin or near horizontal
Block caving	• Can be very cost effective • High production levels can be achieved • Grade control through drawpoint monitoring an asset	• Surface subsidence • Big development effort • High capital costs
Room and pillar mining	• High degree of mechanization possible with excellent productivities • Flexible and safe • Good grade control	• Ground movements can become onerous • Capital intensive • Ore left in pillars
Sublevel stoping, blasthole stoping, and VCR mining	• Easily mechanized • High productivities • Large equipment can be utilized	• High capital investment • Grade control can be a problem • Strong engineering and technical support required
Shrinkage stoping	• Small stopes can be mined • Minimal development costs • Simple drilling and mucking equipment	• Broken ore required as fill material • Grade control can be difficult • Not a high-tonnage or productive method
Cut and fill stoping	• Ground movement minimized • Dilution can be controlled more easily • Compatible with nonfilling methods • Undercut and fill techniques can be utilized	• Labor intensive unless efficient support system possible • Ventilization can be difficult • Can be costly so requires a higher grade ore body • Mill required for tailings fill
Timbered stoping	• Excellent ground control • Maximum ore extraction possible • Applicable for a wide variety of mining problems	• Costly and requires special skills • Not suitable for deeper mines • Low productivities • Nearby source of timber required

21.2.2.2 Block Caving

Block caving methods require large massive ore bodies with a proper fracture pattern. The surface capping should be fairly competent to avoid plugging the cave. Once the ore is undercut, the settlement causes crushing and fracturing of the ore. As the broken ore is drawn off, additional settlement and crushing occur. Blocks can be 300 to 600 ft (91 to 183 m) high and may extend the full width of the ore body, although less than 200 ft (61 m) in width is generally preferable. Panel caving is a variation of block caving and geologic structure, and rock competency can determine whether or not blocks or panels should be mined.

The advantages of block caving are

1. It is a reasonably inexpensive underground method since drilling, blasting, ground support, and labor costs are reduced.
2. Ventilation is generally simplified.
3. Production levels can approach those of large open pits.
4. Low-grade ore bodies can be more economically mined.
5. Grade control through drawpoints aids mine planning.

Block caving has several disadvantages:

1. It is accompanied by major surface subsidence.
2. Development time and money can be excessive.
3. Drift and drawpoint maintenance is expensive.
4. Ground conditions due to weight can damage or shut down production drifts.
5. Draw control is a constant and rigorous endeavor.

Two outstanding examples from practice can again be cited. AMAX's Climax, CO, mine was one of the pioneer block caving operations. The nearby Henderson operation was able to draw upon the Climax experience and make several improvements, notably the LHD (load-haul-dump) vs. scram slusher drifts.

HENDERSON. The Henderson mine is a large-scale, rubber-tired, continuous panel caving system. The porphyry molybdenum deposit has horizontal dimensions of 2200 by 3000 ft (670 by 914 m) and is 600 ft (183 m) thick. The production levels are 55 ft (17 m) above the undercut levels. Drawpoint crosscuts are on 40-ft (12-m) centers. Longhole drills are used to bell out the ore in front of the advancing cave line. Ventilation drifts are required for dust control and diesel exhaust.

The LHD drawpoints are large enough to pass most oversize, which is a decided advantage over the Climax mine with its smaller drawpoints. Certainly Henderson, because of the combinations of geology, good engineering, and company experience should be viewed as one of the best examples of modern caving technology.

EL TENIENTE. The large El Teniente copper mine in Chile is a classic example of block caving technology, with continuous modernization since 1905. Recent earthquakes and ore of lower metallurgical recovery have resulted in setbacks, but at nearly 90,000 tpd (80,000 t/d) and reserves in excess of one billion tons, the El Teniente deposit will be a block caving operation to study for years.

Some of the El Teniente ores are wet and clayey in nature, and older orepass height design had to be restricted to 66 ft (20 m) with drifts set 98 ft (30 m) apart and an orepass spacing of 49 ft (15 m). Only a high-grade ore (+ 1% cu) such as El Teniente could absorb associated costs. Another major advantage is that no ore hoisting is required. Newer blocks in the mine are harder and of lower grade and could not be economically mined with the older methods. Increased mechanization through the use of larger LHDs, more productive drilling and blasting equipment, and newer underground crushing and haulage systems were required.

The disadvantages of the older El Teniente systems were obvious to the operators, and modernization of the equipment and caving techniques resulted in a highly competitive operation being developed. No underground mining methods can compare to the combination of high productivity and accompanying large initial capital investment of block caving operations.

21.2.3 OPEN STOPING METHODS

21.2.3.1 Room and Pillar Mining

Room and pillar mining is the most classic of unsupported or open stoping techniques and, in one form or another, is practiced in most mining methods, particularly in mine development. Room and pillar or stope and pillar mining is, at its classic best, utilized in mining Missouri lead, Wyoming trona, New Mexico potash and uranium, limestone and salt worldwide, Canadian lead-zinc and uranium, and a host of other commodities and locations. In these cases, the ore zones are reasonably flat lying and generally continuous so either a random or regular pillar-support pattern can be established.

By necessity, the method involves hauling the ore and mineral from the face to a central hauling, hoisting, or crushing station. Continuous mining techniques in place of drilling and blasting can be used with the less abrasive and softer ores. It is not uncommon to have underground productivity rival block caving operations. The deposit's size, shape, thickness, and rock structure are the primary criteria for evaluating the suitability of an ore body for room and pillar mining.

Large drills, loaders, trucks, and support equipment cause many room and pillar operations to rival an open pit in planning, maintenance, and operating cycle. Not recognizing the potential efficiencies of modern equipment can hamper room and pillar economics.

Room and pillar mining methods have several decided advantages:

1. The nature of the applicable ore bodies lends itself to a high degree of mechanization and labor efficiencies.
2. Extreme flexibility exists, and rapid changes can be made when required.
3. Grade control can be maximized by adjusting drift dimensions and leaving support pillars in low grade or waste.
4. Productivity can approach or exceed block caving systems if the ore horizons are wide and high.
5. Method can be made very safe and training due to repetition is facilitated.

Disadvantages of the method include:

1. Ground swelling and movement as large areas are opened creates difficulties.
2. Roof and back maintenance can become excessive if larger open areas are required.
3. To provide necessary roof support, high-grade pillars must at times be left.
4. Capital intensive due to mechanized equipment requirements.

BUICK. The New Lead Belt near Viburnum, MO, is made up of eight mines, including the Buick, utilizing room and pillar techniques in lead, zinc, and copper ore bodies. The host rock is dolomite and easily mined by the largest underground equipment available. Ground control is primarily roof bolting and proper pillar spacing, although overlying shale can be a problem if mining gets too close to the undulating shale layers. Rooms are generally 32 ft (10 m) wide and can exceed 100 ft (30 m) in height as successive levels are mined out.

As with all mines in the area, high-grade ore, up to 30% lead, must at times be left in the pillars. High-grade pillars are weaker, and overmining of high grade must be avoided to avoid major groundfalls. In the higher-grade areas, filling techniques may be possible for pillar removal, such as Cominco is doing at their Magmont mine.

The central shops, ventilation system, and high-speed haulage facilitate mining of the 6-mi (9.7-km)-long Buick ore body, which is part of a trend approaching 40 mi (64.4 km) in length. The wet nature of the area requires massive pumping systems, and mine development is hampered when wet ground is encountered. Ore grade at Buick in the earlier years of operation exceeded 13% lead and 4% zinc, with current grades about one-half of these earlier levels.

FMC. Trona (soda ash) mining in Wyoming continues to be a growing and quickly changing industry, with FMC being the pioneer, commencing operations shortly after World War II. Since then, a continuous series of expansions, including solution mining, has taken FMC to nearly 4 million tpy (3.6 Mt/a). Continuous boring (mining) machines are used for driving the development entries to the room and pillar areas. Various face-breaking techniques are used, including kerf cutters and drills with loaders and shuttle cars used for ore removal.

Because of the extent of the trona horizons, FMC designs some of their mine layouts for up to 50 years of safe access. Due to the unique nature of trona mining, FMC has had to design much of its specialized equipment. High wear rates on the continuous miners require more frequent rebuilds than a coal application requires. Setting the stopes up for high production panels during retreat mining is of utmost importance. Generally, the commercial beds are 8 to 10 ft (2.4 to 3.7 m) thick and of large areal extent. Low-height mining equipment, therefore, is required with continuous redesign occurring. Trona mines in Wyoming are considered gassy, so permissible equipment is generally used with elaborate mine ventilation systems required.

In FMC's earlier days, some caving systems were tried and then halted when rock mechanics studies and effects showed weight was not relieved but only transferred. FMC's mining plans, although centering on room and pillar, do embody solution mining. Trona mining in Wyoming requires significant labor training and safety education, which is an advantage when the work force is eventually trained, but is a tremendous disadvantage when turnover is high.

21.1.3.2 Sublevel Stoping and Vertical Crater Retreat Mining

The other predominant type of open stope mining is sublevel stoping, which can be employed when the ore zones or veins are reasonably wide, steeply dipping, and the wall rock is strong and competent. Crosscuts are driven in the ore body, with sublevel drifts used for drilling the ore, which then is blasted in successive vertical lifts. Recent developments in drilling and blasting technology have resulted in sublevel stoping becoming more versatile and widely accepted. Other terms for, or variations of, sublevel stoping are longhole and blasthole stoping and vertical crater retreat (VCR) mining, usually considered as a separate method.

Recently, VCR mining methods have become more successful and widely used, partially due to improved drilling, blasting, and rock mechanics techniques. Different terms are used to define some of the new blasting techniques, such as mass blasting and spherical charge.

Drawpoint design and slot placement are critical for sublevel stoping systems to ensure that expansion space is available for the blasted ore. Crew safety, oversize considerations, and equipment size also dictates that the sublevels be properly designed.

EL SOLDADO. Sitting on the western slopes of the Chilean Andes Coastal Range is Exxon's recently modernized and expanded El Soldado mining operation. The region has been a continuous copper producer for 150 years, with successful variations of sublevel open stoping being practiced from the district's earlier production era. It was recognized by the operators in the late 1970s that the excellent ground conditions and minimal support requirements, combined with ore body access through mountainside adits, could quickly lead to mine expansion and modernization.

By 1983, open stoping, using sublevel blasthole techniques and including large-diameter blastholes and fully mechanized equipment, provided virtually all of the mine production. The basic technique involves undercut fan drilling of a stope 328 ft (100 m) long by 115 ft (35 m) wide with 295 ft (90 m) between sublevels. Sublevels are developed for LHD drawpoints at the base of the stope. The 3-ft (75-mm) fan drillholes to form the slot and undercut the block are up to 82 ft (25 m) in length. The 6.5-in. (165-mm) diameter vertical blastholes on a 11.5- by 15-ft (3.5- by 4.5-m) pattern are 230 ft (70 m) in length and charged with ANFO and prills in a deck blasting arrangement.

The blasting success at El Soldado is partially attributable to the advancements that ring-drill, up-hole, and down-hole drilling equipment have made. Combined with LHDs up to 13 yd^3 (9.9 m^3) capacity and 60-ton (54.4-t) underground haul trucks, the 12,700 tpd (11,500 t/d)mining rate is very effectively and efficiently achieved.

The advantages of the sublevel stoping method as practiced at El Soldado are many:

1. Easily mechanized.
2. Large equipment utilized.
3. Productivity high.
4. Repetitive techniques aid training and safety.

Disadvantages of the system as compared to other methods are few. Capital equipment outlays and development costs were high but justifiable due to the grade and tonnage of reserves. If development schedules are rushed and blocks mined prematurely, the sublevel open stope technique can suffer. Planning and scheduling correctly are of paramount importance.

LEAD. Homestake's 100 + -year-old gold operation in South Dakota has undergone continual mining method upgrades. By the late 1970s, it was recognized that, due to increasing costs and a shortage of skilled miners, cut and fill, shrinkage, and square set methods were becoming outmoded. Highly mechanized bulk systems were studied and VCR techniques were adopted.

The VCR techniques as used at Lead have gone through many different stages of testing, evaluation, and equipment modification. As a deep and older operation, many compromises were required, which finally resulted in a cost-effective and productive mining system.

Like all sublevel stoping techniques, uniform and steeply dipping ore bodies are an asset if not a requirement. With VCR mining, as practiced at Lead, zones of erratic mineralization that lead to dilution are economic due to the high productivity. Topsill widths are a minimum of 24 ft (7.3 m) with the 6.5-in. (165-mm) diameter downholes drilled with a Mission DTH hammer on an 8 by 8 ft (2.4 by 2.4 m) pattern. Maximum stope height is 150 ft (45.7 m).

Blasting techniques when slabbing down the back into the open stope are critical. The broken ore must be mucked to provide void space for the next blast but not overmucked since support is provided by the broken ore. LHDs of 2-yd^3 (1.5-m^3) capacity are used to muck in the lower-level scram drifts.

As practiced at Lead, VCR mining requires strong technical support to insure that optimum hole spacing and alignment and proper blasting procedures are followed. It should also be noted that backfilling of many of the mined-out VCR stopes is essential. One could argue that VCR then is a supported system, but while mining activities are occurring, no additional support is required, so the VCR method is referred to as "unsupported."

Advantages of VCR mining as practiced at Lead are:

1. The method is easily mechanized and cost effective.
2. Development is simple.
3. Large-diameter downhole drilling is possible.
4. Good fragmentation.
5. Broken ore gives wall support.
6. Good safety can result.

Disadvantages of VCR techniques are

1. Quality engineering and technical expertise are a must.
2. Capital investment for equipment is high.
3. Drilling and blasting techniques must be precise.
4. Powder factor can be high.
5. Grade control can be a problem.

21.2.4 STOPING METHODS USING ADDITIONAL SUPPORT

When the predominant mining method requires artificial support techniques, there are numerous support choices available. Depth, rock type, ore continuity, geologic setting, economics, environmental concerns, and a host of other criteria must be understood when selecting the supporting methods. In the past, many near-surface operations have used timber alone for support. Stulls (posts) and caps are the simplest timber supports, with complicated square set timbering systems being used when depths increase.

Gravity alone contributes approximately 1 psi/vertical foot (22.6 kPa/m), which could translate to 72 tons/ft^2 at 1000 ft (690 t/m^2 at 300 m) if no load were transferred to the adjoining abutments. Even hydraulic supporting equipment cannot carry a heavy surface load at depth for a prolonged period, which leads to backfilling techniques that can support 100% of the overburden weight once subsidence occurs and compacts the fill material.

Since World War II, great advancements have been made in equipment and techniques to lessen the significance of timber supports and increase the use of cut and fill mining. Some of the reasons are

1. Rock and cable bolting techniques have become more widespread.
2. Unavailability of cost-effective timber and high labor costs associated with timber-set installation.
3. Improved filling technology and equipment when using mill tailings or appropriate fill material.
4. Advancements in underground drilling and hauling equipment.

The more common types of artificially supported mining methods include:

1. Shrinkage stoping.
2. Cut and fill stoping (overhead).
3. Undercut and fill mining.
4. Timber and prop support systems.
5. Top slicing and longwall and shortwall mining.

Obviously, various methods can be combined within a given mine as economics, geology, continuity, depth, and other factors change. Many cut and fill operations have embraced some form of sublevel stoping techniques such as VCR or blasthole stoping.

21.2.4.1 Shrinkage Stoping

Shrinkage stopes are a combination of open stoping and supported stoping (see Chapter 18.3). Either the ore zone must be very thick or exist as a steeply dipping vein where both the wall and ore can stand with minimal support. The ore swells in place as it is broken and in turn provides ground support and a working area for the miners. Once the ore is drawn off, the remaining open stope can remain empty or backfilled with sand-fill or waste to prevent caving. With increasing depth or adverse ground conditions, lateral stresses can cause the walls in a shrinkage stope to squeeze and compress the ore so it cannot easily be pulled from the stope. The basic criteria for shrinkage stoping to be effective are

1. When broken, the ore must not pack due to clay, fines, water, or chemical reaction.
2. Ore zones should not be too deep such that gravity-caused ground problems result.
3. The wall rock and ore must be strong enough to stand with minimal support.
4. Limit the size of stopes, maintain a rapid mining cycle, and promptly draw stope empty when mining is completed.

The advantages and disadvantages of shrinkage stoping become obvious when viewing the above criteria. Because sublevel open stoping or cut and fill stoping are viable alternatives, shrinkage stoping may be viewed as an adjunct technique in some cases. The disadvantages of shrinkage stoping also include:

1. Ore, at least two-thirds of the total, is tied up as fill material until the stope can be drawn empty.
2. Safety problems exist with loose rib and back material.
3. It is less easily mechanized.
4. Grade control can be troublesome if irregularities exist in wall rock.

21.2.4.2 Cut and Fill Stoping

For many years, the open cut and fill stoping method has been commonly used throughout the world. Many of the longer-lived deposits have become progressively deeper, making cut and fill techniques even more widely practiced. The method is best suited for vein or bedded deposits that dip fairly steeply and have surrounding rock, mainly the hanging wall, that will not stand for a long period without support.

Normally, a slice of ore 8 to 10 ft (2.4 to 3 m) thick is drilled, blasted, and partially mucked, and the back scaled and bolted as required. The remaining broken ore is removed through ore-passes, the orepasses extended to the next level, the stope back-filled, and another cycle begins. The individual mining steps are reasonably well defined, but what can become the biggest logistics problem of all in a cut and fill mine is the production, transportation, and placement of the fill material.

MOUNT ISA MINES. The Mount Isa operations in northwestern Queensland have utilized many different mining methods over the past 20 years, including sublevel open stoping with both slusher and LHDs for mucking, sublevel caving, mass firing of large pillar blocks, and cut and fill techniques. The older cut and fill methods employed at Mount Isa relied on vertical access, whereas more recent cut and fill techniques utilize footwall openings permitting easy access to the ore bodies by mobile equipment. Although both up-holes and horizontal drillholes are used, the flat-backing provides better ground conditions. The individual stopes are grouped into 13,000- to 20,000-ton (11,800- to 18,100-t) blocks, which provides an optimum mining cycle and minimizes orepass placement. Fill is added to the open stope to provide enough height for equipment movement and still provide easy access to the back for roof bolting, scaling, and other ground control methods.

Fill towers and drain pipes must be constructed prior to fill placement so that excess water can be removed as soon as possible. The normal fill is deslimed mill tailings with cemented fill used when additional strength or a better working area is required. As with most cut and fill operations, the availability of fill material is a constant juggling act. Hydraulic fill from milling is generally the cheapest material and fairly easy to distribute but is not available in sufficient quantity to meet all needs. Furnace slag has been used to reduce cemented fill costs. Fill material from the surface siltstone deposit is used in large quantities for backfilling as well.

Mass blasting adjacent to the cemented backfill areas has been done with great success, which speaks highly of the success achieved in fill technology by Mount Isa personnel.

CANNON MINE. Near Wenatchee, WA, the Cannon gold and silver mine is located. The operation is worth describing for several reasons. Open pit methods would have been possible, but environmental concerns and surface disturbance precluded this option. Initial high-grade mining was 200 to 400 ft (61 to 122 m) below the surface. More recent mining has stopes 50 to 100 ft (16 to 33 m) wide by 300 ft (98 m) long with stopes separated by 25-ft (8-m) pillars.

An underground, automated fill-mixing station permits combining river pebbles, sand, and cement. The concrete is dumped directly into the open stopes from haulage trucks. Coarse mill tailings is also used as fill material. The pillars between the backfilled stopes are recovered and backfilled with 3% concrete or dry fill.

The combination of large drilling and hauling equipment combined with the fill techniques makes Cannon one of the more mechanized and modern cut and fill operations.

ADVANTAGES AND DISADVANTAGES. Because the variety of ore bodies suitable for cut and fill methods is numerous, the advantages of the method are equally varied.

For instance, if pillar recovery only is the reason for filling, increased ore extraction would be a major advantage. Generally, however, the classic advantages can be cited as

1. Sampling of the ore block as it is being mined is facilitated.
2. Large-scale ground movement is kept to a minimum.
3. Steeply plunging ore bodies can safely and cost effectively be mined.
4. Dilution can be more easily controlled.
5. There can be compatibility with non-filling mining methods as geology or rock type changes.

Disadvantages of cut and fill mining also can be varied but generally are

1. Dependent upon degree of mechanization, it can become quite labor intensive and require more skills than alternate techniques.
2. Safety problems can be more rigorous, especially if open ground is not adequately scaled and bolted.
3. Ventilation is more difficult.
4. The method is more expensive per ton, therefore not readily applicable to lower-grade ore bodies.

21.2.4.3 Undercut and Fill Stoping

Some operators have chosen an undercut and fill stoping method. The method is generally chosen when compared to overhand cut and fill techniques when bad ground conditions exist, particularly in the back. The advent of more sophisticated roof bolting and ground control techniques has somewhat lessened the need for undercut and fill systems. In an undercut and fill operation, the ore is mined from the top down while carrying the fill line in the back. The overhand cut and fill method, or conventional cut and fill as some operators refer to it, carries the fill in the floor with the ore in the back being removed.

Successful underhand systems rely upon the theory of a voussoir arch forming within the cemented sandfill. The weight not supported by the arch is carried by timber posts and mats.

Undercut and fill stoping methods are used by Inco and Falconbridge in Canada, with great success. Fill technology by both operators has been developed to a high degree. Inco developed the method in the 1950s to facilitate transverse pillar mining. Many mining companies have used the method with the following advantages:

1. When fill in the back is properly consolidated, it is far safer than an ore back.
2. Pillar and rib mining is easier.
3. Grade control is precise.
4. Allows mining to proceed safely when ground conditions would preclude other methods.

Disadvantages of the undercut and fill method would include:

1. Quality of the fill material has to be very good, which adds to the expense.
2. Supporting the fill with timbers and mats adds to the cost.
3. Poor fill in the back can lead to major safety problems.
4. Blasting and drilling techniques must be very precise to maintain the integrity of the overhead sill.

21.2.4.4 Timber and Prop Support Systems

It is all too easy for the modern miner to categorize timber and prop-supported mining systems as archaic. Yes, they are in one fashion or another, a throwback. The earliest underground miners used some sort of wood support, and during the latter part of the 19th century, timbered mines were becoming sophisticated and commonplace. Prior to 1860, open stoping using stulls was the common underground mining method, and with sometimes disastrous consequences, throughout the world.

The Ophir mine in Virginia City, NV, was very high grade, and in 1860, the width of the ore body became too great for the current roof and rib support systems. Philip Deidesheimer, a young mining engineer, was assigned the task of designing an improved timbering system. What he devised at the Ophir was probably the most unique and elaborate timbering system ever used.

In the western United States at that time, many high-grade ore bodies were developed with similar exotic timbering systems. It became quickly apparent that some method of rapid backfilling in conjunction with timbered systems was required to further reduce ground movement. In the last 100 years, backfilling has reduced the complexity of timbering systems so that posts, caps, and girts (ties) comprised the only supports.

Timbering systems, in one fashion or another, are still commonly used. Most undercut and fill stoping methods rely upon timber supports to hold the overhead fill, particularly during placement. When driving development headings or haulageways, timber systems are still used to support bad ground where bolts and mats would otherwise not work. In very high-grade ore bodies, some form of timbering can yield greater ore recovery than more common filling systems. Some mine operators are far removed from an adequate supply of sandfill, which may lead to timbering systems.

The major advantages of timber supported systems are

1. Excellent ground control method.
2. Maximum ore extraction possible.
3. Trained crews can easily install the system.
4. Applicable for a wide variety of mining problems.
5. Does not require hydraulic fill, so mine can be separate from mill.

The disadvantages are

1. Requires adequate timber supply.
2. Training and education of crews different than with more common methods.
3. Costly as compared to bolting or other support systems.
4. Not as applicable to large tonnage operations.
5. Timbers alone cannot support heavy ground in deeper mines.

21.2.5 OTHER MINING METHODS

There are several mining methods that will not be discussed in detail but are worth mentioning because of their past or possible future application in noncoal underground mines.

21.2.5.1 Top Slicing

In the early part of the 20th century, top slicing techniques were somewhat common, particularly underground in the Lake Superior iron ore district of the Mesabi Range. Basically, top slicing consists of driving a series of heavily timbered drifts and crosscuts and blasting out the supporting timbers. The unsupported ore will then cave or subside onto the floor of the opening and permit slicing downward in successive lifts, using the broken timber mat to prevent further subsidence until the lower-level supports are again blasted.

In modern day mining, top slicing is considered expensive and very seldom used. For it to be a successful method, timber must be cheap and plentiful. Generally, high ore extraction with little dilution is possible. Subsidence will occur, and ventilation can prove difficult to control. A significant fire hazard exists, especially since the broken timbers are not accessible for normal inspection, and old timbers continue to break up and form kindling.

21.2.5.2 Longwall Mining

Although considered a coal mining technique, longwall mining has seen application in Wyoming trona, South African gold and potash, iron ore in Europe, and others. Many underground sedimentary deposits, including salt, uranium, and sulfide minerals, have potential future applications. Longwalling is considered a supported stoping method because of the massive props used to support the roof over the face, although controlled caving of the roof is also involved (Chapter 20.1).

The basic requirements for longwall mining to be considered as a method include deposit size and thickness, back and overburden strata, dip, face characteristics, abrasiveness and hardness, and gas conditions. The method has commonly been used in Europe, in fact, since the late 17th century.

Longwall mining today is characterized by some sophisticated equipment including cutters and shearers, hydraulic roof supports, and automated loading and conveying systems.

21.2.5.3 Shortwall Mining

The equipment and concepts used in shortwall mining closely match those in longwall, with but one important difference: the shortwall face is generally less than 200 ft (61 m), whereas in longwall mining the face can easily exceed 1000 ft (305 m).

21.2.6 MINING METHODS—ROCK MECHANICS AND PLANNING

21.2.6.1 Rock Mechanics

All too often, the initial information available on a new mining venture is inadequate to define all of the necessary ingredients required for making a successful operation right from the onset. Most competent engineers are aware that even the best information does not guarantee design tonnage and budgeted costs. Successful mines are more often than not a series of compromises, built around a broad yet definitive plan that must use flexibility as one of the prime design criteria.

Rock mechanics has been practiced for centuries in one form or another and today is an integral and significant part of all mining engineering curriculums. The science of rock mechanics has developed in a slow but steady fashion as mine successes and failures were analyzed, determining reasons for departures from theory. Sensor and monitoring devices such as load calls, earth pressure cells, extensometers, and sloughmeters are all part of the mining engineer's rock mechanics arsenal. Combine that with mathematical models, computer simulations, and case studies, and concepts quickly become applicable for prediction of mining method selection and parameters.

Rock mechanics can and has assisted caving operations in selecting equipment and drawpoint configurations. Being able to accurately predict the broken ore size in a caving operation is essential for designing haulage systems. Ideally, the best caving system has (1) the ore horizons in highly fractured ground with discontinuities and flows that form planes of weakness, and (2) the sublevels, haulageways, and drawpoints in competent and easily supported rock under the ore zone. Thorough rock mechanics studies and geologic interpretation are essential to establish a successful block caving operation. The ore-zone rock characteristics and nonore zone beneath the ore body rock and geologic criteria are important as well.

If studies indicate that block caving is not practical, then sublevel caving, sublevel open stope, or even mechanized cut and fill could be considered. In each case, rock mechanics studies along with geologic studies and ore body configuration are crucial.

The advent of large and productive mechanized equipment has caused some mine operators to design their mine around the equipment rather than the equipment around the mine. Both approaches are compatible but not exclusive. Most room and pillar operations have sufficient flexibility to design haulageways and pillar patterns to satisfy a broad range of equipment and productivity criteria. Several room and pillar operators, lacking an understanding of rock mechanics, have pushed too hard for productivity and mineral recovery and suffered disastrous consequences.

A more thorough understanding of rock mechanics allows planning engineers to schedule more proficiently drilling, blasting, mucking, supporting, and backfilling cycles. Success in minimizing rock bursts has primarily been a result of proper mining method selection and scheduling. There are several older mining operations that have used rock mechanics studies as part of the criteria for altering or fine tuning mining methods. The Mount Isa operation in Australia, Lead in South Dakota, Kiruna in Sweden, and most of the larger South African gold producers have dramatically changed their mining methods through improved knowledge of rock mechanics.

Rock mechanics and related topics are detailed in Section 10.

21.2.6.2 Subsidence

Surface subsidence is an issue that is gaining significant attention around the world, particularly in North America. Many hard-rock mining engineers would like to think that subsidence is a coal issue, but that is not the case.

The conflicts between miners and nonminers over land use will fan the discussion in future years. This does not bode well for the mining companies that are near populated areas and are employing caving techniques or near surface operations robbing pillars. Mining method selection should out of necessity use subsidence possibilities as one of the criteria for mine evaluation and design.

Planned subsidence questions should also involve near-surface aquifer loss or contamination. Unplanned subsidence is worse in that overlying tailings impoundments or surface facilities can be damaged with corresponding human tragedy.

Again rock mechanics and mine planning are significant tools in making a mining method selection. Add to that environmental concerns over groundwater contamination, ventilation releases, and subsidence problems and the modern day miner has a host of parameters to consider when making a mining method selection.

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Chapter 21.3
HARD-ROCK MINING: METHOD SELECTION SUMMARY

DENNIS KRANTZ AND TIM SCOTT

21.3.1 IMPORTANCE OF PROPER METHOD SELECTION

21.3.1.1 Flexibility of Method

One of the most important outcomes of proper method selection is a solution that allows the maximum degree of flexibility. The ability to adapt to unexpected underground conditions or external considerations can go a long way in determining the immediate and long-term success of a mining operation.

There is no one mining method that allows the mine operator to be able to accommodate all the known and unexpected conditions, but the mining method chosen should be the one that best accommodates the known and expected parameters at the lowest possible cost per ton mined, and still allows the flexibility to handle the unexpected. During the initial phases of method selection, there are conditions that are clear cut and have little possibility of being different than expected. There are areas, however, where the information necessary to make an accurate determination of conditions is not available. It is in these areas that flexibility in the chosen mining method is most important.

Proper method selection incorporates the flexibility to respond to changes in both internal and external conditions. Internal conditions are those that are dictated by the deposit itself, whereas external conditions are determined by outside considerations such as business or market requirements.

The first main consideration is ore deposition. In some cases, the choice of mining methods is relatively clear cut. A flat-lying or vertically dipping deposit automatically eliminates certain mining methods from consideration. It is those deposits that occupy the middle ground that make the selection process more difficult. A deposit that lies near the angle of repose dictates that the method chosen has the flexibility to handle stopes where the broken ore moves freely to a drawpoint as well as those where it does not.

The other consideration of ore deposition is whether the mineralization is uniform throughout the deposit or is erratic. The mining method chosen should allow the flexibility to mine the higher-grade portions of the ore body where and as they occur. A mining method that does not allow pillaring of the low-grade sections or requires that they are broken and treated as ore greatly increases the overall costs.

The next major consideration where flexibility is important is the ground conditions of the deposit and surrounding host rock. Proper method selection allows for mining areas with varying ground conditions without major changes in the mining method. An example of this is vertical crater retreat mining, where the broken ore can be left in the stope for support much like a shrinkage stope or drawn empty like a sublevel stope.

A third major internal consideration should be the working environment. In this area, the mining method chosen should be able to accommodate changes in ventilation requirements, water inflows, etc. If the mining method is chosen assuming that only minor ventilation is necessary, there can be major problems if it becomes necessary to control airflows throughout a mine that has too many dead headings or interconnected workings.

External conditions involve changes that require a different but no less important type of flexibility from the chosen mining method. These considerations are usually of a more long-term nature. A mining method that allows for adjusting the mined grade to accommodate changes in the market will possibly allow the mine to continue to operate through low periods in a market cycle. The mining method should also allow for increases in the tonnage produced to take advantage of upward swings in the market cycle. A mining method that dictates that each stope be operated in a set sequence will more than likely not allow for changes in the tonnage and ore grade produced and therefore will not be readily adapted to swings in the market place.

In the final analysis, the mining method chosen should be the one that best suits the deposit being mined and the conditions most likely to occur. Consideration should be given when choosing the method to how well it can handle changes in conditions that are considered to be possible.

21.3.1.2 Capital Costs and Cash Flow

The selection of a mining method should consider the capital cost associated with bringing a mine into production and its continued operation. At the same time, it is necessary to consider the cash-flow requirements of the method chosen.

Ideally, the method chosen should not require that all necessary capital costs be incurred before any production. The full cost of mining a deposit can be greatly reduced if the capital costs are incurred as the mine is being brought up to full production. This has the result of producing a positive cash flow sooner, which allows the operator to reduce the amount of capital required to bring a mine into production.

Equipment and initial development costs are normally the two major components involved in bringing a mine into production. The mining method chosen should not require that the bulk of the initial development work be completed before any production commences. If instead the development coincides with some degree of production, the final costs are decreased to some extent. The earlier production can then be used for a gradual startup and break-in of the milling facilities. This ideally would allow both the mining and milling to reach full production in the same time frame that will improve the cash flow of the operation.

21.3.1.3 Matching Politics and Economics of the Area

Major consideration should be given in the initial phases of the mining method selection to the political and economic conditions in the area. Both can have a major impact on the success of the method chosen. Whether the mine is located in the US or abroad plays a major role in this.

Political concerns are probably not a major concern in an industrialized area, but if the mine is located in an area lacking any major employers, there may be restraints placed on the operation. These may require hiring a major portion of the work force from the native (local) population, or simply the desire to have as many people employed as possible.

Economic concerns normally are related directly to the skill level and cost to employ the people who will make up the mine work force and must be considered when choosing the mining method. Choosing a highly mechanized mining method for a deposit in an area that lacks a skilled labor pool increases costs by requiring a long lead time to train a crew or the necessity to bring in outside people. The same increase in costs holds true for choosing a labor-intensive method in an area of high employment or a small labor pool that would require higher labor rates to draw the necessary people.

The same basic considerations apply for the equipment required by the mining method. An area where parts and service are readily available would favor a highly mechanized method. Conversely, an area where parts and service are not readily available requires the maintenance of a large parts inventory and the ability of the operations to perform a much larger variety of specialized maintenance.

The final method selection should take into consideration the skill and size of the available labor pool and their effects on equipment selection and ultimately the overall mining costs.

21.3.1.4 Environmental and Safety Considerations

Both environmental restraints and safety have major impacts on mining costs and ultimately on method selection. While some of them are easy to quantify, others are harder to identify but still can have a major impact.

There are environmental restraints, such as mine waste disposal, water discharge, etc., that are readily apparent and the effects considered in the method selections. There are others that are just as important to the overall success that are easily overlooked or not given full consideration. One of these is the final disposition of the property after the operation has finished. This phase can result in major costs required for final closure that could be reduced if they are considered as an integral part of the method selection. For these reasons, the effects of long-term subsidence, drainage, etc., should be considered.

It is hard to place a final dollar figure on safety and the working environment, but they do have an effect on the success of the method chosen and must be considered. A method that requires a major effort to maintain a safe and comfortable working environment increases the mining costs. If the conditions are too severe, they will affect work performance, absenteeism, labor turnover, insurance premiums, etc., all of which have a direct effect on the final costs.

In summary, there are many things that are directly or indirectly influenced by mining method selection. All should be considered, and their impact on costs considered when making a selection. Once a method has been selected and the operation is underway, it is difficult and expensive to make changes.

21.3.2 COMMON PITFALLS IN METHOD SELECTION

The most common pitfalls in the selection of a mining method are the same basic considerations that, when properly considered, result in a successful operation. These areas may be considered, but due to lack of experience, poor or disregarded information, or simply the belief that they can be solved when the problem arises, lead to problems that could have been avoided.

21.3.2.1 Method vs. Tonnage and Grade Requirements

Probably the most common pitfall is choosing a method that is not suited to the tonnage and grade requirements that are necessary to meet milling and marketing requirements. Most mining methods can be scaled up or down to some extent, but have a general tonnage and grade flexibility range that makes them best suited for a particular size and type of deposit.

Block caving might be suitable for a deposit and material that have a large and relatively stable market. In most cases, it would not meet the needs of an operation that must adjust to relatively large swings in market demand or pricing.

Recognizing the final market for the material and its relative stability helps to choose the mining method that best allows the operation to respond in a cost effective manner.

Another common area for problems is the adaptability of the chosen methods. These are generally associated with the ability of the mining method to adapt to unexpected conditions within the deposit, or to changes dictated by market conditions involving the mining sequence and production scheduling.

It is a common failure when choosing a mining method to assume that ground conditions, grade distributions, etc., are reasonably constant throughout the deposit. Rarely is this the case, and a method chosen without consideration of how it will handle these variations will not always be able to accommodate them.

At the same time, the method chosen should have the ability to accommodate the swings in production scheduling and resulting changes in mining sequence that would normally be expected for the material being mined. If the method chosen requires mining in a rigid sequence, the operation may be required to shut down during low points in the market cycle. If instead it can accommodate ceasing operations in low-grade/high-cost areas, it may be possible for the operation to run during low points in the cycle, then resume mining in those areas at a later point when it is economically feasible to do so.

21.3.2.2 Environmental/Health/Safety Concerns

It is easy for those choosing the mining method to fail to realize or adequately consider the effect the method has on the environmental and health and safety phases of the operation. This failure can later be readily apparent in the case of external environmental constraints, but may not be as obvious in the areas of health and safety. The effects on both areas should be considered through the entire life of the operation and its final closure. Neither is the determining factor in the final method selection, but they can greatly affect mining costs.

21.3.2.3 Complexity of the Method

A common pitfall in the choosing of a mining method is to choose one that is not suitable to the availability and skill level of the labor pool, or whose equipment requirements are not easily met and serviced in the area. This is most commonly done when choosing a mining method for a locale where the engineer is not familiar with the cultural and economic conditions.

A mining method that requires a relatively high degree of literacy and mechanical ability on the part of the hourly operator may be the best choice for on operation in the United States, but is probably not the best choice for a third-world country. Conversely, choosing a labor-intensive method for an area with high labor costs may not be cost effective.

In the final analysis, the mining method chosen should be the best balance between (1) labor and equipment costs and (2) availability and skill levels of the local labor pool.

There are numerous other areas that, if not considered during the selection process, will most likely result in problems later in the operation. It is advisable to consider every aspect of the methods operation and how well it will adapt to any reasonably foreseeable problem. Even recognizing that the method does not easily handle a problem is of value in making the final selection and in implementing the chosen method.

21.3.3 STEPS FOR PRACTICAL APPLICATION

21.3.3.1 Research of Similar Deposits

Anticipation of problems before they occur is a key consideration in formulating a mine plan. Time spent on research and consideration of all possibilities during the planning stage is more than repaid later in the project life. Avoidance of common pitfalls in the selection of a mining method or methods begins with an examination of other deposits with similar features that have been successfully mined by other operators. Research of written material, direct interviews with personnel at other mining operations, and onsite visits will identify common aspects of the projects and the successful resolution of problems associated with these aspects. This background study of how others have handled problems similar to those the development engineer may reasonably expect to encounter is one of the best foundations for mine planning.

21.3.3.2 Establishment of Desired Goals

The background information gathered form the study of other similar deposits is the basis for establishing an aggressive set of goals that are achievable within constraints imposed by the nature of the deposit. These constraints include the tonnage and grade of the minable reserve base, the confidence level of the reserves, and the potential for expansion or contraction of the reserve base through changes in market conditions and ongoing exploration.

Project goals are also impacted by the magnitude of necessary development and the time delays between the beginning of development, the beginning of production, and the beginning of a positive cash flow. Consideration of production and marketing sequence requirements in light of qualitative estimates of cash flow and changes in market conditions will also put limits on reasonable goals. It is important to maintain a proper balance between a positive approach that will promote continuation of the project and a realism of the goals that the deposit will support.

21.3.3.3 Method Selection: Advantages vs. Disadvantages of Each

The selection of an appropriate mining method benefits from a method-by-method tabulation of advantages and disadvantages. This tabulation will examine the various features of the deposit such as tonnage, grade, homogeneity, geometry, and accessibility and how these features have been handled by other successful mining operations. The planner should always consider the possibility of using more than one method for different portions of a deposit or of modifying a method as a result of unique deposit characteristics or the experience of other operators.

21.3.3.4 Mine Life Plan

A basic plan for the life of a mine looks at the schedule and timing for the entire project from development through production to final reclamation. It applies the chosen mining method or methods to each portion of the deposit in appropriate sequence and evaluates the subsidiary needs of all phases of the project. During this process, it is valuable to reconsider the choice of mining method, how that choice was made, and whether modifications of the method are needed.

21.3.3.5 Scheduling Development/Production

A viable mine plan ultimately includes a detailed scheduling of all phases of the project. These include manpower requirements, both numbers and necessary skill levels, and equipment requirements including type, size, quantity, and cost.

A very important though sometimes overlooked aspect of this phase of the planning process is that of materials handling. This includes the obvious components of transport of supplies to the active areas and ore to the process facilities as well as other components such as waste-to-ore ratios, the handling of waste without sacrificing production, and the potential benefit of interaction within a plan. For example, shift scheduling can help simplify the movement of waste and supplies and development excavation can frequently be used as a source of fill for cut and fill mining operations.

21.3.3.6 Continuing Modification of the Plan

The steps in the planning process, although outlined in a logical sequence, are greatly dependent upon each other. This interdependence requires a continuing examination and possible modification of previously established components of the plan based on other evolving features as it develops from the beginning general framework. A periodic revisitation of earlier decisions results in detailed documentation that will ultimately be useful to evaluate and control the project. This continuing modification of the plan is a process that may continue through development and into the established production phase of the mine.

21.3.4 RECOGNITION OF PROBLEMS: IMPORTANCE OF MODIFICATIONS AND FLEXIBILITY

The ultimate level of profitability of a mining project is enhanced by flexibility in the mine plan and choice of mining methods.

21.3.4.1 Establishing a Mine Plan: Expectations vs. Actual

The procedure used to establish the parameters under which the project is to operate does not end when the project is being implemented but rather should be ongoing throughout the project. Just as the various elements of a mine plan and the information gained from the research of similar deposits are used in an interrelated manner to refine the details of mine planning and method selection, so the experience gained as the project proceeds and the comparison of the actual to what was expected or predicted by the mine plan is the best information with which to refine the mine plan. The extent to which the initial plan has the flexibility to allow the introduction of these data into the process may determine the level of success of the project.

21.3.4.2 Internal (Short-Term) Modifications

Internal conditions dictated by the deposit are again a major consideration and the first to be examined. The evaluation of ore deposition from exploration data only may be inaccurate due to unexpected changes in orientation or geometry between drill intercepts or points where information has been gathered. A deposit that may have seemed to mandate a particular mining method may in fact be of a marginal nature as revealed by the more extensive excavation that occurs during development. In such a situation, the mine plan must have sufficient flexibility to allow the mining method to be changed and still meet the other goals of the project as defined by production scheduling, economic analysis, and manpower and equipment availability.

In a similar manner, unexpected variations in ore grade encountered by mine development may require a change in the mining method to maximize the profitability of the ore block or may require redefinition of the boundaries of the mining blocks to allow low-grade uneconomic material to be left as pillars.

Another important impact of grade variation is on the scheduling of mining activity. The sequence of production from the various stope blocks may require some change to meet steady-state grade requirements of the processing plant. These changes affect the entire balance of material handling within the mine and must be accommodated by changes in material flow.

The final elements of an internal or short-term nature to be considered are the ground and working environment conditions. A mine plan must have the flexibility to modify mining methods in response to unexpected ground conditions. In addition, it must have the flexibility to carry changes in mining method through the other aspects of the plan, including production scheduling, ventilation, and material handling.

21.3.4.3 External (Long-Term) Modifications

External or long-term conditions provide a more fundamental basis for the mine plan and are usually more difficult to modify without major impact on the other aspects of the plan. In addition, external conditions interface with elements beyond the control of the mine operator such as market conditions and the cost and availability of capital, equipment, and manpower to support the mining activity.

Impacts on the capital and cash costs of a project may have internal sources from change in scope, method, or scheduling of the project that have already been discussed. Modification of cash-flow and capital requirements of a project from external sources are to a great extent beyond the control of the mine operator. They are a particular problem for projects with a long lead time between commitment to the project and actual production and positive cash flow. The mine plan must have enough flexibility built in at the start to accommodate a reasonable range of expected change.

On occasion, advanced metallurgical test work or actual mill operation in its early stages may show an optimum blend of ore that is somewhat different from what was planned. In addition, problems with achieving the projected grade control with the chosen mining method may result in an unfavorable grade of ore being delivered to the mill. In either case, modification will be necessary to bring mine output in line with the requirements of the processing facility. These modifications usually involve mining methods and production scheduling, which in turn affects the other aspects of a mine plan.

A third area of concern for the impact of external conditions is that of fluctuations in market conditions or requirements. As a result of these fluctuations, the quality of the mill product may be unacceptable in the market place. The fluctuations may be due to changes in the supply from other sources, the demand for the product, or in the technology of secondary processing operations.

Price fluctuations may also require adjustment of the quantity of product and perhaps the life of the project in order to meet overall economic goals. All other phases of a mining operation must have the flexibility to accommodate these changes if the mine is to be profitable.

21.3.5 SUMMARY

Underground mining offers a wide variety of methods that can be adapted to suit virtually any size and type of deposit. These methods allow for the exploitation of deep deposits that would otherwise be uneconomic, and in some cases, they are competitive with surface mining methods (Section 16).

Unlike surface mining, underground mining offers a wide variety of methods, several of which might be adaptable to a given deposit. It is deciding which methods are applicable, and then choosing from those the one that offers the most cost-effective method of extraction, that is the key to a successful operation.

The success of the mining method is to a great extent dependent on the thoroughness of the final selection. The main criteria for selection will always be generally the deposition and ground conditions of the ore body itself, but it is the determination of the likelihood and effect on mining of a host of other interrelated conditions that make the critical difference in the selection process. The degree to which these conditions are foreseen, their effects taken into consideration, and adaptions made yields benefits that far outweigh the efforts involved.

Chapter 21.4
COAL MINING: METHOD SELECTION

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The focus of this chapter is on generally accepted current industry practices, together with an emphasis on the methodology used to generate accurate decisions on the economic viability of a given mining method used on a given set of reserves.

It is worth noting that most of the subjects mentioned in this chapter— as in the previous three —are covered in greater detail elsewhere in this *Handbook,* and the reader is encouraged to supplement the information contained in this chapter with those additional resources, especially in Section 17 and Chapters 18.1 and 20.1.

21.4.1 GEOTECHNICAL CONSIDERATIONS

A thorough review of the geotechnical features associated with a given set of reserves is critical in selecting the optimal mining method for those reserves. The characteristics of the seam itself, the roof, and the floor must be thoroughly examined to assure the viability of the mining methods selected.

21.4.1.1 Seam Characteristics

THICKNESS. The thickness of the seam mined has a major impact on the mining method selected and the equipment required for that method.

For continuous miner units, the current effective range is between 36 and 120 in. (1 and 3 m). Machines for this height range are readily available from all current manufacturers. Heights in excess of 120 in. (3 m) call for specialized equipment, while thicknesses of less than 36 in. (1 m) are currently mined only in the Appalachian coal fields and with auger-type continuous miners.

Conventional drill and blast equipment is useful from 36 in. (1 m) to heights in excess of 120 in. (3 m), although current economics have substantially reduced its application in US coalfields. Some areas in East Kentucky are still shooting on the solid, without mechanical undercutting, in seam heights down to 28 in. (0.7 m), although the economics are extremely marginal at those heights.

Longwall mining is similar to room and pillaring with continuous miners, in that there are four distinct seam thickness ranges, each of which calls for different sets of equipment choices.

There are six basic conditions that any longwall system has to meet:

1. The armored face conveyor that (AFC) must have sufficient width and height to provide adequate capacity (typically a minimum of 1200 tph or 1100 t/h).

2. If a shearer is used, there must be adequate height between the top of the AFC and the underside of the shearer to allow the efficient passage of coal cut on the inby side of the shearer (8 in. or 200 mm).

3. The shearer itself must be thick enough to provide motors of reasonable power (300 hp or 220 kW plus).

4. There must be some clearance between the top of the cutting machine and the underside of the support canopies (typically, a minimum of 8 in. or 200 mm.).

5. The machine length must be minimized to enhance the ability of the system to negotiate seam undulations while staying within the seam.

6. The cutting machine must be able to effectively vary the cutting height to accommodate changes in seam thickness.

These six conditions effectively limit seam heights of 30 to 42 in. (0.8 to 1.1 m) to in-web shearers, where the bulk of the machine sits off the AFC and in between the cutting drums, or plows, which have no motors. (The plows also require hard bottom, soft-to-medium coal hardness, and uniform seam thickness).

The range between 42 and 54 in. (1.1 and 1.4 m) effectively opens the options to in-web double-ended ranging drum shearers (DERDS), single-ended ranging drum shearers (SERDS), or plows for the same six reasons. At the time of this writing, SERDS have been the system of choice in the United States for this range. The Europeans have had a substantial amount of success with the in-web DERDS at these heights, and it is anticipated that this technology will be adapted in US coal mining in the near future.

In the range of 54 to 120 in. (1.4 to 3.0 m), which covers the vast majority of current longwall installations, the DERDS is the machine of choice. Plows and SERDS can also be used at these heights, but current technology provides the maximum available power and flexibility in the DERDS.

At ranges above 120 in. (3.0 m), DERDS are also the machine of choice, but the power, size, and stability required at these heights dictates somewhat specialized machines.

Seam thicknesses above 16 ft (4.9 m) are mined by sublevel caving behind the shields, multiple-lift longwalling, hydraulic mining, or other very specialized mining techniques. As of this date, only one face with multiple-lift equipment has been tried in the United States, and the vast bulk of experience at these heights has been in Europe and Asia. Detailed information on these systems is available from European manufacturers.

RESERVES. The size and shape of the available reserves are critical from two standpoints:

1. A sufficient amount of reserves must be present to justify the capital cost of the mining system selected.

2. The shape of the reserves can place limits on the type of mining system employed.

Consider the two principal mining methods: (1) room and pillar with drill and blast or continuous miners and (2) longwall. In the case of continuous miners or drill and blast methods, reserves of as little as 100,000 recoverable tons (91,000 t) have been exploited, although 500,000 recoverable tons (450,000 t) is a much more realistic figure in market conditions prevalent at the time of writing.

The shape of the reserves can be highly irregular, provided the economically minable reserves are contiguous.

If the case of longwalls, reserves of 7 to 8 million recoverable tons (6 to 7 Mt) are typically the minimum, although use of previously purchased equipment and/or the existence of other reserves that are amenable to the same equipment can shift this number downwards. In any instance, a detailed economic evaluation is required to accurately match capital costs against available reserves.

The reserve shape for longwall mining is much more critical, because effective longwall operations require large rectangular blocks of coal, preferably situated side-by-side to minimize development and moving costs. Current United States longwall practice favors panel widths of 500 to 1000 ft (150 to 300 m), and lengths of 4000 to 10,000 ft (1200 to 3000 m) plus, with the trends favoring longer and wider. In general, panel lengths of 2000 ft (600 m) or less are not economical at the time of writing, although site-specific conditions may alter that minimum somewhat.

QUALITY PARAMETERS. Quality parameters of the salable product can be substantially impacted by the mining method selected. More importantly, the very marketability of a given reserve block may be a function of the mining method. In general, a high degree of variability in quality characteristics favors the smaller unit production systems (continuous miner, drill and blast) that provide flexible mining to produce a consistent product. One alternative to this is blending with other coals to produce a consistent product. The only other is to mine the reserve as efficiently as possible, and trust the sales group to find an acceptable market at an acceptable price.

The basic quality parameters that must be addressed in the review of any coal reserve are sulfur, moisture, fixed carbon, volatile matter, calorific value, and ash. Increasingly, markets are becoming more and more specialized, and additional quality parameters that are associated with the targeted market must be closely analyzed to assure reserve viability. Some other quality indicators are sodium ash fusion and rare earths.

In addition to the inherent qualities of the coal, some of these parameters will be changed by processing. Notable among these are sulfur, ash, moisture, calorific value, and size consist.

A very realistic view should be taken of the in-seam and out-of-seam extraneous material that will be taken or, more accurately, given with a particular mining method. In general, longwalls will produce higher reject levels than room and pillar with drill and blast or continuous miner methods, due to their lesser ability to deal with seam variations.

This may not be true in seams with competent immediate roof and floor consistent seam sections and limited and gentle grade changes, but such conditions are not always found.

In any event, considerations of reject material generated by the mining method chosen cannot be reviewed too carefully.

Higher-than-anticipated reject levels increase mining costs, increase coal preparation and attendant refuse disposal costs, and may imperil the actual sales contract itself. It has meant the demise of many a mine. In current conditions, non-longwall reserves that have associated reject levels higher than 30 to 40% are generally considered to be marginal to impossible, depending upon mine productivity, preparation costs, haulage costs, and contractual sales obligations.

STRUCTURAL CHARACTERISTICS. The structural characteristics of a coal seam can have a substantial impact on the mining method selected and the success of that method. It should be noted that a thorough exploration program of a property is the best initial investment that can be made, and one that will prevent many unwelcome surprises and/or expensive mistakes in the course of mining.

The structural characteristics to be discussed are those of regional and local dip, strike, cleats, and faults.

Regional dip and strike will control drainage, and may limit the options in mining methods, equipment, and haulage equipment. Severity of the grade can also affect weight transfer from gob areas. Localized changes in dip and strike (rolls) must be considered because of their adverse impact on track and belt installations, drainage, and the requirement to take additional rock from floor and/or roof in these areas. Additionally, these rolls will impede productivity. Finally, localized rolls will require higher equipment clearances than normal.

Current longwall equipment is limited to grades of 20% without special modification. Rubber-tired equipment can handle grades up to 15% in dry conditions, but equipment power must be adequate, and wet muddy conditions reduce that maximum rapidly. Rail equipment is limited to grades of 5% without special traction devices and/or hoist assist. Crawler-mounted equipment is limited to approximately 25%.

In general, grades below 10% can be mined without specialized equipment. Grades of 10 to 15% necessitate equipment review and a careful review of potential productivity. Grades of 15% or greater require specialized equipment and mine designs.

Cleats in the seam must be reviewed for stability and cutting/blasting directional preferences. The more prominent cleat, the face cleat, and the less prominent, the butt cleat, typically are positioned at 90° to one another. In general, a mining orientation of 45° to each is preferred, and is increasingly critical in thicker seams to avoid heavy rib sloughage.

Faulting is a critical problem in any block of reserves, and particularly those with displacements that exceed seam thickness. Again, if the presence of faulting is likely, the exploration program cannot be too thorough. In conjunction with drilling, new techniques such as radio imaging and cross-hole seismic surveys are useful in detecting faults (Chapter 10.3). In general, the presence of substantial faulting favors the use of room and pillar mining over longwall mining due to its increased flexibility and economic advantage in mining smaller blocks. In any event, the presence of faulting adversely impacts the economics of any property.

PHYSICAL PROPERTIES. Physical properties that affect the selection of mining methods are hardness, strength, cleat spacing, and chemical characteristics.

Hardness is usually measured by the standard ASTM Hargrove Grindability Index; it is one measure that predicts the difficulty and cost of cutting the coal, in conjunction with cleat orientation and spacing. It is a particularly important factor if longwall plows are considered, as they function best in soft cutting conditions. Coal strength is the determining factor in the design of effective pillars and barriers. In addition, some coal seams have separate layers of coal with unequal strength characteristics, which must be considered for design calculations. Cleat spacings have a major impact on cuttability and actual strength determinations used for pillar design. In general, closer cleat spacing favors cuttability and decreases strength. Finally, a review of chemical characteristics of the coal should be made to assess the likelihood of spontaneous combustion. Specialized mine planning, ventilation, and material handling schemes are required to deal effectively with this problem.

GEOLOGIC ANOMALIES. Included in this category are linear deposition, seam washouts, partings, and intrusions. Linear deposition occurs when the coal is laid down in a channel, and this can be very misleading without a thorough exploration program. Similarly, washouts of the seam by roof or floor can be very localized, sporadic, and difficult to predict without a good knowledge of the general geologic trends of the seam and its depositional environment. The presence of substantial seam thinning or washout is critical in longwalls and can force abandonment of panels. Radio-imaging and in-seam seismic techniques have proven most beneficial in detecting geologic as well as structural anomalies. Seam partings increase run-of-mine reject and can pose cutting or blasting difficulties when composed of harder material. They can also generate difficulties in geotechnical design and force changes in mine plans if thick and persistent. Another difficulty is posed by seam intrusions, typically found in coals subjected to volcanic and/or tectonic activity. They produce both obstacles to reserve continuity and chemical changes (metamorphosis) in the coal immediately adjacent to the intrusion.

21.4.1.2 Roof Characteristics

ROOF LITHOLOGY. Roof characteristics are a critical factor in the selection of coal mining methods and associated extraction rates. In general, the study of roof characteristics can be limited to 60 times seam thickness, since that is the maximum height at which disturbances generated by mining have occurred. In fact, the first 5 to 15 seam thicknesses of roof material are even more critical in high extraction mining methods that cave the roof, since all of the cave and most of the major fracturing will occur within this horizon.

In high-extraction mining methods (longwalls and retreat-mined room-and-pillar operations), cavable roof is a must. This requires either shales or weak sandstones within the first 5 to 15 seam thicknesses. Strong massive sandstones or limestones should be very carefully noted, since they both limit caving and transfer stress. The presence of aquifers within 60 seam thicknesses should be noted, since the fractures generated by high-extraction mining may reach that far, although 30 seam thicknesses are the more usual maximum. Pillar design, mining layouts, and extraction rates for retreat mining should consider the possibility of strong strata bridging across the panel and withholding true vertical stress from the pillars until a sufficiently large stress is generated to force failure of those strong strata, resulting in catastrophic pillar failure. During longwall mining, the same situation results in a complete lack of cave until additional mining and increased stress exceed support design limits, force massive caving and resultant airblast, or some unhappy combination of the above effects, with accompanying support difficulties in gateroads as an additional unpleasantry.

Based on this data, panel designs, extraction ratios, and support designs for longwalls can be developed.

GEOLOGIC ANOMALIES. All of the preceding data should be reviewed for the existence of anomalies, particularly within the defined caving zone for high extraction mining or alternatively, the support zone above mine openings. Bed thicknesses should be mapped and isopachs prepared. The existence of sandstone channels should be noted. Any data that indicate a change in immediate roof lithology should be noted and examined, especially the existence of thin (less than 5 ft or 1.5 m) and weak shale roof below more competent strata; this condition almost guarantees problems. In deltaic depositional environments, thick sandstones can disappear in very short lateral distances, and these changes can substantially modify stress fields generated by mining.

21.4.1.3 Floor Characteristics

FLOOR LITHOLOGY. Floor characteristics are usually not reviewed with the care that roof and seam characteristics receive, which is unfortunate. The first 10 ft (3 m) of floor should be reviewed as carefully as the roof and seam, because it can have an equally large impact on equipment selection, pillar design, and productivity. Soft and/or weak floor limits pillar sizes, forces compromise in powered roof supports for longwalls, and impedes movement and productivity of tracked and wheeled equipment. It will also increase run-of-mine reject.

Floor strata should be carefully studied for the following data within 10 ft (3 m) of the seam: lithology, fracture density and orientation, strength, brittleness, and the presence of clays. Additionally, tests should be run to determine the behavior of the strata when exposed to water. In general, the presence of thin, brittle, and/or highly fractured material immediately below the seam is suspect, particularly if coal leader seams are present. In the case of longwall supports, base lift should be fitted if there is even a possibility of soft floor.

In general, soft floor conditions are more easily dealt with by continuous miner/drill and blast techniques than by longwalls, at the expense of increased reject levels.

GEOLOGIC ANOMALIES. Geologic anomalies that affect floor conditions are the same that affect the roof. Murphy's law dictates that floor lithologies will be different where drilling was not done. The value of a thorough geologic exploration that established overall trends, possible anomalies, and an understanding of the depositional environment cannot be overstated in its relation to the optimal selection of equipment and mine design.

21.4.1.4 Stress Field Considerations

The stress field in which a mine is developed is one of the most important items to be considered in the selection of a mining scheme. It affects the stability and design of every hole in that stress field and, even more importantly, the stability, shape, and size of the coal and rock surrounding those holes. Those two items taken in aggregate define a mine. The stress field consists of the initial isotropic stress field generated by the overburden, and subsequent modifications made by tectonics, geologic anomalies, and other mining.

OVERBURDEN. Overburden or depth of cover in general defines both the vertical and horizontal stresses encountered in mining. In general, depending upon overburden lithologies and densities, sedimentary strata generate approximately 1 psi of vertical compressive stress per foot (22.6 kPa/m) of overburden. These resultant vertical stresses, coupled with typical sedimentary rock strengths, limit coal mining to overburden depths of less than 4000 ft (1200 m), with the vast majority of mines in the United States at depths of 2000 ft (600 m) or less. As regards mining methods, high-extraction (greater than 70%) mining with continuous miner/drill-and-blast methods has typically been limited to depths of 2000 ft (600 m) or less. Practice in the Eastern coalfields indicates that under current economic conditions, high-extraction mining with continuous miner/drill-and-blast methods is limited to 1200 to 1500 ft (370 to 460 m) in conjunction with mining conditions and specific economic factors. Conversely, longwall mining is routinely being utilized at depths of 2000 to 2500 ft (600 to 760 m). Mining at depths greater than 2500 ft (760 m) is a rarity, and is possible only where geologic and market conditions are exceptionally favorable.

GEOLOGIC ANOMALIES. As discussed previously, the original stress field is generated by overburden, and is theoretically isotropic. In practice, the horizontal components of this stress are always modified by geologic structures and anomalies. In principle, the maximum horizontal stress will parallel the major jointing pattern and/or major faulting in the area. It may also exceed the vertical stress by a substantial margin. An estimate of both the direction and magnitude of the horizontal stresses should be made, because high horizontal stresses will generate roof and/or floor failures that are substantially dependent on mine orientation and layout. Given higher-than-normal horizontal stress magnitudes, longwalls will cope with these conditions better than continuous miner/drill and blast techniques, due to the lesser amount of roof that must be maintained and the economics of production that usually allow more expensive roof control measures to be taken.

MINE WORKINGS. Mine workings above or below will modify the stress field associated with a given seam, sometimes quite substantially. The effects are dependent on the depth of overburden, the interseam thickness, the amount of extraction in the mine workings, the size and shape of the remaining coal pillars, the order of mining, the length of elapsed time since mining, and the strength of the interseam strata. Influences from existing mine workings have been reported at interseam thicknesses of 800 ft (240 m), although that appears to be a practical maximum, with the vast majority of cases at 500 ft (150 m) or less. In general, any pillars/barriers in the existing mine workings focus the vertical stress field, and the effects of the increased stress can be seen both above and below the old workings. Large barriers or pillars increase the probability of increased stresses. Then again, the presence of high vertical stress concentrations favors the use of longwalls, due to their enhanced ability to deal with difficult roof and the enhanced economics which allow for more expensive roof control measures.

21.4.2 SAFETY/ENVIRONMENTAL/REGULATORY CONSIDERATIONS

Coal mining is an inherently dangerous business. It also has a direct impact on the environment wherever it is practiced. As a result of these factors, it is also arguably the most regulated industry in the United States. As a result of this, safety and environmental issues must enter into the selection of any mining system.

21.4.2.1 Mine Environment

The mine environment is critical to the safety and productivity of the mines. Although regulatory agencies set standards for all critical mine environment issues, the operator should review his conditions against those regulatory requirements. In many instances, and particularly in the case of longwall mining, the minimum regulatory requirements may not be adequate.

VENTILATION. Mine ventilation is a subject of substantial weight and is more thoroughly addressed elsewhere in this *Handbook* (Chapters 11.6 and 11.7.2). However, it may be useful to review some general rules of thumb for underground coal mining.

Room and Pillar Mining—In general, 25,000 cfm (11.8 m^3/s) delivered to the last intake stopping of the section in mining heights of 4 to 8 ft (1.2 to 2.4 m) will prove adequate. Higher mining heights typically require more air to generate the minimum required velocity of 60 fpm (0.3 m/s) over the miner. Lower heights will conversely dictate a lower air quantity. A special precaution should be noted if diesel-powered face equipment is used. Required air quantities are also a function of the number and type of diesel engines used in that equipment. Double-split section ventilation effectively doubles the air quantities required. Double-split section ventilation is almost mandatory with diesel-powered face equipment. Auxiliary face fans are quite common in seam heights of 8 ft (2.4 m) and higher and are quite effective, but special attention should be paid to localized air recirculation that is typically generated by these systems. In general, ventilation systems that are initially overdesigned will subsequently prove to be most helpful as time-related deterioration of airways increases mine resistance. Any mine larger than two sections with a life of more than 10 years will benefit highly from life-of-mine computer simulation for selection of main fans, design of mains, and other critical decisions. With the advent of personal computers, it can be done rapidly and conveniently at the minesite at minimal cost. Blowing ventilation systems should be considered in instances where isolation from the atmosphere and/or old workings cannot be maintained.

Longwall Mining—The ventilation of longwalls is absolutely critical due to the high production rates and associated generation of dust and methane. In general, actual quantities delivered across the face range from 40,000 cfm (18.9 m^3/s) in seam heights of 4 ft (1.2 m) with minimal methane liberation, to 150,000 cfm (70.8 m^3/s) or greater in seam heights of 7 ft (2.1 m) or more with substantial methane liberation. It has been proven that face velocities above 500 fpm (2.5 m/s) can entrain increased quantities of dust, causing problems in compliance with the federal respirable dust standard of 2 mg/m^3. It should also be noted that the amount of quartz detected in respirable dust samples lowers the effective allowable respirable dust concentrations. Strong roof that caves in large pieces will adversely affect the ability to maintain adequate face velocities due to high leakage rates through the gob, and may be a particular problem during the first 500 to 1000 ft (150 to 300 m) of mining in a panel. Daily wash-down of face supports may be required to minimize entrainment of dust generated by caving of the roof. In some instances, water sprays have been installed on roof supports to minimize this dust, although maintenance can be a problem. Maintenance of bleeder systems around the perimeter is always a problem due to the high stress fields encountered, and high safety factors for pillar designs as well as supplemental roof support is required in these areas. Adequate ventilation of gob areas may require surface boreholes with high-head fans to assure adequate dilution and removal of methane. Seams with high methane quantities typically require degasification boreholes from the surface or in the longwall panel in advance of mining. Infusion of high-pressure water into the seam in advance of the longwall face has proven beneficial in reducing respirable dust generation in some coal seams, notably the Pocahontas and Pittsburgh seams.

GROUND CONTROL. Ground control is a complicated subject more thoroughly covered elsewhere in this *Handbook* (Chapter 10.5), but it may be useful to briefly discuss its implications on mining method selection. In principle, roof control difficulties increase as a function of increasing cover and percentage of extraction. Roof bolts are the principal means of roof control at present, ranging in order of cost from mechanical shell anchor bolts to fully grouted resin bolts to larger-diameter point-anchor bolts. Timber and concrete posts and cribs are also used quite commonly. More substantial support is available with roof trusses, steel arches or square sets, and polyurethane foam injection. In practice, ground conditions that require more than roof bolting, cribs, and the very occasional use of more substantial support will not be economically viable with continuous miner or drill and blast techniques. Because of their enhanced productivity, generation of high stress, and use in the deeper seams, longwalls both require and can afford more expensive and elaborate ground-control measures.

NOISE. Noise is increasingly a problem underground as machine power has increased (Chapter 11.8). Also the long chain conveyors associated with continuous haulage systems and longwalls can increase noise levels substantially while running empty. Although not typical in most coal mines, air-powered drills have always delivered noise levels well above allowable levels.

Substantial efforts have been made over the years by the manufacturers, users, and the US Bureau of Mines to reduce the noise levels produced by the machinery. However, in many instances, the operator's position is still exposed to noise levels in excess of the 8-hr weighted exposure requirement of 90 dB maximum. It is worth mentioning that noise levels in excess of 90 dB do permanent damage, and the damage is cumulative. The most effective preventative is personal hearing protection. The types of hearing protection available and approximate levels of noise reduction are listed below in order of effectiveness:

Soft foam ear plugs	30–35 dB
Rigid ear muffs	24–29 dB

The effectiveness of any improvised ear plugs, such as cotton, cigarette butts, or other expedients, is largely illusory and cannot be recommended.

Many companies are now requiring ear protection at all times, with required use in high-noise-level areas, and this effort is highly recommended; noise damage is insidious and cumulative.

STATE AND FEDERAL REGULATIONS. As noted earlier, the mining industry, and more particularly coal mining, is heavily regulated (Chapters 3.3 and 11.1). In that environment, state and federal regulations have an impact on mining method selection. Because of the wide variations in state regulations, these must be reviewed very carefully. Of possible interest is the total ban on blasting on the solid in many states.

There is little purpose here again in discussing specific issues in most states, or even at the federal level, since they are subject to revision. It is worth noting that longwall mining, which has heretofore been the focus of a limited amount of state and federal regulations, is being increasingly regulated by both.

21.4.2.2 Outside Environment

SUBSIDENCE. Subsidence is an increasing concern with all underground coal mining, and particularly with longwalls, although any mining method that provides high extraction ratios will induce subsidence (Chapter 10.6). Many states are now routinely requiring subsidence predictions as a part of the mine permitting process, and this trend will continue.

Technical issues in predicting subsidence accurately have effectively been solved on a regional basis during the last five years. Regional studies in the West and in the Appalachians have produced data that support accurate regional prediction models for longwall and high-extraction room and pillar mining.

Areas that require surface support are typically limited to 50% extraction, which means that room and pillar mining with continuous miners or drill-and-blast methods are the only choices. Extraction rates of 50 to 85% typically generate partial subsidences, and the actual extent and degree of subsidence associated with this level of extraction are difficult to predict.

Compensation for damage to surface structures is a given, and any plan that utilizes high-extraction mining should be reviewed for potential costs associated with surface structures, as well as areas of the surface under which high-extraction mining is prohibited.

HYDROLOGIC IMPACTS. The fracturing of roof strata associated with high-extraction mining has an impact on the subsurface hydrologic regime. Also the change in surface topography generated by subsidence can affect surface drainage, particularly in the case of flat land used for agriculture. The changes to the surface are obviously permanent, but the subsurface hydrologic regime is more variable. Typically, the increased fracturing increases permeability, with the usual affect of diminishing or effectively eliminating sources of water for wells, natural springs, and other sources of domestic water. In many instances, as the fractures fill with clayey material, these water sources have returned. In others, they have not. Many companies have adopted a policy of replacing these water sources for the surface owner if his/hers are damaged or destroyed. Therefore, the use of high-extraction mining systems should include a review of surface ownership and potential hydrologic damage consequences as a function of mine planning.

STATE/FEDERAL REGULATIONS. Increasingly, the mandate for the regulation and reclamation of surface mining and the surface activities of underground mines given to the federal Office of Surface Mining or its equivalent on a state level is being extended to underground operations and their associated surface effects. Regulations and interpretations thereof are quite dynamic, so the operator must familiarize himself or herself with the applicable regulatory agencies and regulations in his/her area. In general, operators of high-extraction underground mining methods can look for increasing constraints upon their operations.

21.4.3 OPERATIONS PLAN DEVELOPMENT

21.4.3.1 Introduction

It is necessary to develop a relatively detailed mine plan when comparing alternative mining systems or when considering the economics feasibility of a particular mining method in a given reserve area. Both underground and surface requirements should be addressed. An underground mine plan should be developed that integrates mine plan projections, production forecasts, mine ventilation system, mine water supply/dewater system, haulage system(s), power system, and mining equipment. These plans are all interrelated and therefore, they cannot be considered separately. A surface plan should be developed that considers (1) the surface construction and support facilities that will be required to facilitate mining, (2) the transportation system that will be used to transport run-of-mine coal to cleaning and/or load-out facilities, and (3) coal preparation requirements. Finally, the staffing requirements to support both the underground and surface operations should be developed.

21.4.3.2 Underground Mine Plan

MINE PLAN PROJECTIONS/PRODUCTION FORECAST. The overall mine design and layout (the location and orientation of all mains, submains, and panels) is often dependent on several of the geotechnical factors already discussed in this chapter, that is, faults, overlying mine workings, etc. These factors should be considered when laying out all mine projections. In general, mains and submains are costly to develop because of the high cost of purchasing, installing, and maintaining long-term sustaining items such as belt, track, pipe/pipe couplings, ventilation controls, outby power centers, and power cable. For this reason, development of mains and submains should be minimized to the extent possible. Similarly, mining costs associated with the development of longwall headgate and tailgate panels are relatively high. For this reason, the current trend in longwall mining is toward longer, wider faces that reduce both mains development and headgate/tailgate development while maximizing longwall tons.

The number of entries developed on each respective main, submain, or panel depends on the following factors:

1. Production considerations. For any given set of continuous miner or drill-and-blast equipment, there is an optimal number of entries and pillar sizes that will maximize productivity. “Super-sections” and drill-and-blast sections typically require 7 to 10 entries, while single continuous miner units require 4 to 7 entries.

2. Ventilation considerations. Sufficient cross-sectional areas must be provided to insure the availability of required air quantities. If CO monitoring systems are contemplated, the belt entry could be used as an additional intake.

3. Haulage considerations. If belt and track/roadways are placed in separate entries, an additional entry will be required.
4. Ground-control considerations. The number of entries and the pillar sizes selected must provide stable roof and floor conditions, particulary in mains or submains with long lives.

Typically, main headings are developed utilizing a 6- to 10-entry system, submains are developed utilizing a 4- to 8-entry system, continuous miner panels are developed utilizing a system of 5 entries or more, and longwall panels are developed utilizing a 2- to 5-entry system.

The production expected from a continuous miner or longwall unit can be estimated after carefully analyzing the following information: (1) prior mining experiences in the same seam, especially those in the same general locality, (2) mining experiences in seams with conditions very similar to those expected for the newly proposed mine, and (3) actual production rates achieved with the mining equipment selected. In order to develop a production estimate, the number of operating days per year and the number of production shifts per day will have to be factored in. The current trend is to operate two production shifts and one maintenance shift per day for approximately 240 days/yr. The production estimate can then be used to determine the linear advance rate for each unit (feet or meters of advance per day). The linear advance rate for each main, submain, or panel can be used to determine the length of time required to drive each set of entries. Mine development can easily be mapped showing monthly, quarterly, or yearly mine development.

In a longwall mining application, the number of continuous miner units that will be required to keep mains development and longwall panel development ahead of longwall mining should be determined. The total number of miner units used throughout the life of the mine should be minimized to the extent possible in order to keep equipment costs and manpower requirements down. The longwall start-up rooms should be developed a minimum of three months ahead of the longwall and, if possible, the longwall start-up rooms should be developed six to nine months ahead of the longwall to allow for unforeseen slowdowns in production. However, longwall panel development should not exceed the nine-month time frame by very much because conditions are likely to worsen in a panel after an extended period of time. Also the requirements for belt, track, and other sustaining items will increase if several headgate panels are developed far in advance of the longwall, only to be idled.

Mainline development in a new reserve block should coincide with the orderly retreat of units from a reserve block where mining is nearing completion. Here again, avoid developing mains or submains too early. Mains, submains, shafts, slopes, etc. should be developed as necessary to match mine development with production and ventilation requirements.

The annual tonnages from each active production unit should be tabulated to provide a production forecast for the projected mine. Downtime resulting from equipment moves (from panel to panel) should be factored in the production forecast, whether it involves longwall equipment or continuous miner equipment. If poor mining conditions are expected in localized areas due to geologic conditions such as severe dips or washouts, then the production rate should be reduced to reflect this information.

VENTILATION. It is essential to determine the ventilation requirements for a proposed reserve area and incorporate these requirements into the mine plan projections. The ventilation plan for a projected underground mine should include the following information: (1) the air quantities needed during all phases of mining; (2) the size, shape, and number of airways needed to achieve the desired air quantities; (3) the general location and timing for all shafts, slopes, drifts, mine fans, etc.; and (4) the capital costs associated with the ventilation system.

General guidelines for required quantities of air were discussed in 21.4.2.1. Design engineers are advised to use generous design quantities and conservative friction factors in planning the ventilation network.

The mine ventilation plan must be reviewed and approved by state and federal agencies. These agencies may have established certain requirements and/or guidelines that often have a direct bearing on the ventilation.

The mining equipment selected can also impact face ventilation requirements. For example, diesel mining equipment requires a minimum air quantity for each piece of diesel equipment utilized in the face (see Chapter 11.5).

It will be necessary to determine the size, shape, and number of airways needed in order to meet ventilation requirements throughout the mine. In most instances, the mining height is simply the seam height plus the thickness of any extraneous material that is expected to be mined with the seam, for example, soft fire-clay bottom. However, in thicker coal seams (more than 12 ft or 3.7 m), the entire seam may not be mined due to ground-control problems resulting from the increased height. In lower seam heights, it may be necessary to mine out-of-seam material in order to provide clearance along track haulageways. Here again, the selection of mining equipment can have a direct impact on the minimum entry height; that is, the height of the stage-loader on a longwall face will affect the minimum entry height that can be taken during development of each headgate entry in the mine. The width of the airway is determined by ground-control analysis. Maximum entry widths are usually preestablished by state and/or federal regulations.

The size, location, and number of openings (shafts, slopes, or drifts); the size, location, and number of mine fans; and the number of airways for each main, submain, or panel is a function of the size and shape of the reserve area, the size and shape of the airways (entries), the number of units, and the overall mine design/layout. Due to the complexity of mine ventilation networks, computer analysis and simulation of the mine ventilation system is used to develop and optimize the overall ventilation plan. Additional shafts, slopes, or drifts (with or without fan capabilities) are variables that can be used in lieu of developing a large number of airways, and these alternatives should be considered. In large reserve areas, it is usually not necessary to have all projected mine openings and/or mine fans established as part of the initial mine development. Instead, the need for additional intake/return capability usually becomes necessary due to mine expansion: as the mine development progresses into new reserve blocks and the size of the mine increases, the requirements of the mine ventilation system also increase.

Once the ventilation requirements for the reserve area have been detailed, the costs associated with the ventilation system can be estimated. (Particular attention should be paid to the date that each phase of the ventilation system needs to be in operation.) The capital cost estimate should include the costs to develop ventilation shafts, slopes, and/or drifts and also the cost to purchase, construct, and install each fan and fan housing. The yearly operating costs for the mine ventilation system should also be estimated (e.g. fan maintenance costs, power costs, airway maintenance costs, etc.).

EQUIPMENT SELECTION. It is not necessary to detail equipment specifications and select manufacturers at this early stage in mine planning. What is necessary is to determine what parameters the equipment will be required to work in and then determine the mining equipment that is best suited for this type of application.

When selecting equipment for either continuous miner applications or for longwall applications, the geotechnical considerations discussed earlier in this chapter define the working environment for the equipment. What is the height range that the equipment will be required to work in? What grades will the equipment routinely have to negotiate? What roof and floor conditions can be expected throughout the mine? Will out-of-seam material have to be taken? These are all important factors that can be answered by analyzing the roof, floor, and seam characteristics. Mining at lower heights requires low coal equipment that also translates to lower production rates.

Once the working parameters for the equipment are determined, the size, type, and amount of equipment best suited for the mine can be determined. For example, if out-of-seam floor and/or roof rock will periodically be mined, then the cutting equipment should be designed with the motor horsepower, frame/drum size, bit lacing, etc., necessary to cut the rock. Note that the equipment being considered must also meet safety, environmental, and regulatory considerations.

Once the general specifications for the mining equipment are known, relatively accurate cost estimates can be obtained from several equipment manufacturers.

HAULAGE. The haulage system(s) necessary to transport run-of-mine coal to the surface and to transport personnel and materials underground should be determined. Track systems, belt conveyor systems, and combination belt/track systems are all being used to meet haulage requirements in large underground applications. The method of choice in the United States in recent years has been to utilize belts to transport coal and utilize track or rubber-tired equipment to haul personnel and supplies. In shaft mines, an underground belt system can be utilized in conjunction with an underground storage bunker that provides surge capacity until the coal is hoisted to the surface.

The location of all mainline belts, submain belts, and panel belts should be determined. Since belt transfer points represent a high maintenance area for the life of the mine, they should be minimized to the extent possible. The current trend is toward longer belts of higher strength with bigger drives and bigger motors. A booster drive installation can also be considered for long belt applications; the main belt drive works in conjunction with a series of booster drives. In either case, transfer points are reduced.

The peak tonnage requirements for each belt line should be determined. The maximum number of units dumping on a particular belt line at any given time can be taken directly from the mine plan projections. The tons per hour from each unit depend on the equipment being utilized. Typically, a feeder is used in a continuous miner unit to load coal uniformly onto the belt; generally, the peak discharge rate from a feeder ranges from 5 to 10 tph (4.5 to 9 t/h) depending on equipment size, type, and specifications. The peak discharge rate from a longwall unit ranges from 10 to 30 tpm (9 to 27 t/min) depending on the size, type, and specifications of the equipment used on the longwall face.

The belt length, the difference in elevation, the peak loading rate, and the run-of-mine coal density are used to calculate the belt speed, belt width, belt loading (in pounds per inch, or grams per millimeter of width), and belt tensions. This information can then be used to aid in the development of specifications for drives, take-ups, and belt structure.

The mainline track system used to transport men and supplies is typically comprised of 60-to-100 lb (27-to-45 kg) track on wooden ties, ballasted, with sufficient vertical and horizontal clearance to permit mining equipment to be hauled in and out of the mine on a low-boy. If rubber-tired equipment is used, haulage roads should be graveled, graded, and maintained with a grader to minimize equipment downtime and travel time to the active areas.

MINE WATER SUPPLY/DEWATER SYSTEM. The mine water system necessary to supply water underground should be determined (Chapter 12.1). First of all, mine water usage should be estimated (designed for full production). To estimate the peak water usage for the mine (in gallons or liters per minute) and to establish the minimum water pressure required to operate effectively, the following information should be compiled: (1) the water volume/pressure required to operate all face equipment, including ancillary equipment such as water sprays, scrubbers, etc.; (2) the water volume/pressure necessary to operate water sprays and water deluge systems at belt drive installations; and (3) the water consumption expected from other miscellaneous activities such as equipment maintenance and washdown. Typically, continuous miner units require from 10 to 40 gpm (75 to 150 L/min) at a working pressure from 200 to 300 psi (1.38 to 2.07 MPa), and longwall units require from 60 to 120 gpm (227 to 454 L/min) at a working pressure from 200 to 300 psi (1.38 to 2.07 MPa). The actual water consumption at belt drives is relatively small, usually ranging from 5 to 10 gpm (19 to 38 L/min). However, the mine water supply system must be capable of supporting fire fighting activities at each drive and along each belt line. Booster pumps should be considered to maintain higher pressures when dust suppression is required.

Second, the best potential source of water for the projected mine and the supporting facilities should be determined. Several alternative water sources may be available and all of these options should be analyzed in order to determine the most cost-effective alternative. For example, there may be an existing water source from a nearby mine or other facility that can be utilized. Water may be available from overlying, underlying, or adjacent mine workings. There may be groundwater inflows expected into the proposed mine that can be used for the mine water supply. It may be relatively easy and inexpensive to develop a groundwater well to supply water for the mine and support facilities.

Once the water quantity and pressure requirements have been estimated and the water source has been determined, the mine water supply system can be developed to meet these requirements. It will be necessary to determine the pipe size, type, and pressure rating required for the water supply pipe to be used on each main, submain, and panel. An elevated water tank near the mine portal may be needed to achieve the working pressures required for mining and to provide surge capacity (it is desirable to be able to store a two-to-three-day supply of water in case of interruptions in the water supply). The need for elevated water tanks, pumps, and/or booster pumps depends on the location of the water source, the structural characteristics of the seam (strike, dip, and elevation changes along mains, submains, and panels), the size and shape of the reserve area, and the number of units. Here again, computer-aided analysis and simulation can be used to help develop and optimize the plan for the mine water supply system.

The water required to operate the mine bathhouse and other mine support facilities should be added to the underground water requirements in order to calculate the total water demand for the facility.

Similarly, a mine dewatering plan should be developed that addresses how water accumulations will be handled and removed from the mine. There are many potential sources for unwanted water inflow and accumulation in a mine. For example, groundwater flow into mine workings can be a problem, particularly when mining nears the seam outcrop. Also water from adjacent or overlying mine working can create a chronic dewatering problem as water continually seeps through fracture planes. This seepage may become much more prevalent during full-seam mining methods. Water accumulation can also result when the underground water-supply pipe breaks as the result of a roof fall or other reason. The volume of water that will have to be continually removed from the mine should be estimated after carefully analyzing the potential water sources and after reviewing prior mining experiences with respect to the seam being mined. Again, the system of pipes, sumps, pumps, etc., that will be needed to effectively dewater the entire mine workings should be designed.

21.4.3.3 Surface Plan

SURFACE FACILITIES. When analyzing a given reserve area, there may be several alternative sites that can be used as the main portal for the mine. It will be necessary to determine which location is best suited for the mine. Surface ownership and access/availability of water, electricity, roads, and railways must be considered. A primary consideration is the overall surface topography, particularly when the mining is in steep terrain. There may be a strip bench left from prior mining or there may be another relatively flat area available that can cut initial development costs considerably. Another primary consideration is to have the mine portal located as close to the preparation plant as possible, particularly when an existing preparation facility is going to be used. Also, as discussed earlier in this chapter, there are several underground considerations that may influence the location of the mine portal. For example, there may be an area in the reserves where faults, washouts, or lower seam heights are prevalent and as a result, this area would not be suitable for mainline development. Once all these factors have been considered, the most cost-effective location for the mine portal can be selected. In large underground mines, it may be cost effective to utilize several different portals in order to reduce the travel time required to transport men to and from the active workings.

The total surface area needed to support the proposed mine will not be fully known until the plans for the surface facilities, the materials handling system, and the coal preparation facilities are clearly defined. Mining in areas with steep terrain will require extensive excavation to establish bench area to work from with correspondingly high development cost. For this reason, it may be cost effective to locate some support facilities such as preparation or bathhouse facilities away from the mine portal. Similarly, it may be cost effective to locate some supporting facilities such as shops or manstation facilities underground. It may be more effective to locate shops underground in shaft and slope mines due solely to equipment haulage constraints. These factors should be considered when developing the mining plan.

The surface facilities necessary to support the mine should be determined. These plans should address the preliminary design, location, timing, and costs for the following requirements: (1) access and/or haulroads; (2) site preparation; (3) office facilities, warehouse facilities, shop facilities, bathhouse facilities, personnel facilities, parking facilities, etc.; (4) substation(s); (5) supply yard(s); (6) ventilation structures such as fans, fan housing, shafts, etc.; (7) environmental structures such as sediment ponds, diversion ditching, sumps, etc.; and (8) disposal areas for trash, breaker rock, etc.

Further discussion of the surface mine plant is contained in Chapters 7.2, 17.3, and 17.5.

MATERIALS HANDLING SYSTEM. The materials handling system located on the surface is used to handle, store, and transport the run-of-mine coal. It may also be necessary for this system to handle and segregate underground waste rock that is occasionally generated during mine development for overcasts, drive areas, belt trenches, etc.

There should be adequate storage capacity at the minesite to insure that underground coal production is not affected when unforeseen problems occur that stop the flow of coal from the minesite, for example, a major breakdown at the preparation plant. As a rule of thumb, there should be enough storage capacity at the minesite to store one week of mine production. Of course, the available surface area and the surface topography significantly impact the storage capabilities at the minesite and, as a result, it may be too cost intensive to develop a large storage area.

The trend today for large-scale operations is toward live storage where the run-of-mine coal stored at the minesite is automatically retrieved by the materials handling system without the need for men and machines to rehandle the material. Typically, either a reclaim tunnel system is used with open storage, or silo(s) are used, or a combination of both.

A rotary breaker is often located at the minesite for the purpose of removing large rock from the run-of-mine coal in the early stages of the material handling system. Transportation and handling costs can be significantly reduced if rock is removed before the run-of-mine coal is transported to the preparation plant.

If the preparation facilities are not located at the minesite, then the transportation method to get the run-of-mine coal to these facilities should be selected. The coal is routinely transported by either trucks, railroads, or overland conveyors but other methods are available such as slurry systems, aerial trams, or pneumatic systems that may be desirable in certain applications. Small-scale operations typically stockpile their run-of-mine coal on the ground and then periodically load and haul the coal in trucks to nearby cleaning facilities. Trucking offers a low initial cost; however, the operating costs to rehandle and truck the coal are generally high. Large-scale operations typically use either a loadout facility dumping into railcars or an overland conveyor to transport run-of-mine coal to nearby preparation plants. Although the initial capital costs for these systems are high, the operating costs are generally low in comparison with trucking and other systems.

COAL PREPARATION. In almost every instance, run-of-mine coal requires cleaning before it can be sold, and this constitutes a significant part of the overall mining cost. However, certain areas in a given reserve base or certain mining methods such as longwall mining may produce a run-of-mine product that is suitable to meet coal quality parameters or that may be blended raw with clean, processed coal to form a marketable product. If these conditions are likely to exist, a tremendous cost savings will be realized if the raw blend product can be segregated. Material handling systems are being utilized that have a nuclear analyzer to detect a raw blend product, which in turn triggers a flop gate and separates the raw blend coal.

In many instances, a company will have active coal preparation facilities located near the projected reserve area. It will be necessary to determine if these cleaning facilities can handle the run-of-mine feed. The quality parameters (percentage ash, sulfur, moisture), the percentage of fines in the run-of-mine coal, etc. will also be considerations when determining if the existing plant can continue to operate effectively and meet quality constraints. Plant modifications may be possible that will allow the facility to handle the increased tonnage and/or help the facility to better meet the coal quality parameters.

If existing preparation facilities are not available for the proposed reserve area, then the location and the detailed design/layout for the preparation plant will have to be developed so that it can process the run-of-mine coal output and meet all coal quality parameters. The capital cost to construct a coal preparation facility can be staggering, and careful planning is essential to develop accurate cost figures. The total capital cost required to construct the preparation plant should be determined. In addition, the costs required to operate the preparation facility should be determined (the operating cost is typically figured as a cost per ton of clean, or processed, coal).

21.4.3.4 Staffing

Labor costs represent a significant portion of the total mining cost and, for this reason, an accurate estimate of the manpower requirements should be determined for the proposed mine. The manpower needs for any given year of mining largely depend on the following factors: (1) the number and type of production units in operation, (2) the overall extent of the active underground workings, (3) the type of equipment, (4) the general mining conditions, and (5) the availability of regional support groups. All of these factors should be carefully considered when developing the staffing requirements for the proposed mine.

The number and type of production units utilized during each year are developed in the mine plan projections and production forecast. Typically, a continuous miner section employs from 7 to 12 persons while a longwall section employs from 5 to 10. The exact number of employees utilized on each production unit depends on the face production system and the type of equipment utilized. For example, in longwall mining, fewer shield operators are required to routinely advance shields with electrohydraulic controls as compared to a longwall face with manually operated hydraulics shield control.

The need for outby support personnel (i.e., motormen, beltmen, trackmen, etc.) increases as the number of active production units increases. The overall size of the mine also has a bearing on how many outby workers will be needed; additional personnel will be needed to maintain airways and maintain belt and/or track haulageways in large mines. Certain mining conditions can also dictate the need for additional mine personnel. For example, if wet mining conditions are expected throughout the life of the mine, then pumpers will be needed to maintain the mine dewatering system. Also, if poor roof conditions are expected throughout the mine, then additional personnel will be needed to construct cribs and timbers, spot bolt, and rehabilitate mine haulageways and airways.

In some instances, companies employ regional support staffs to perform functions such as accounting, engineering, payroll, etc., for several company mines and support facilities. If this situation does not apply, the mine will have to hire technical and administrative personnel to perform these services at the minesite or the work will have to be contracted out.

21.4.4 ECONOMIC CONSIDERATIONS

21.4.4.1 Capital and Operating Costs

In the real world, the selection of mining methods is inextricably tied to the costs associated with those methods. These costs are grouped into two areas: capital costs, and operating costs.

Capital costs are these associated with the purchase of equipment or facilities, and these costs have a substantial impact on both the initial financing of a mine and its subsequent operating performance. Capital costs to open a single-unit drift mine range from $200,000 to $500,000, and costs for a multiple-unit mine requiring shaft/slope access can easily total $30 to $50 million, excluding in-mine infrastructure and mining equipment. A complete longwall system currently costs $12 to $16 million, without regard to additional costs for mine infrastructure purchase or improvement costs. Similarly, a complete set of equipment for a continuous miner section costs $1.5 to $2 million.

In the case of continuous miner units, rebuilt/used equipment can usually be obtained at substantially lower prices, with the trade off of less than state-of-the-art performance and a probable increase in maintenance costs. Longwalls are always designed as a system for a specific set of mining conditions, and the purchase of any used longwall equipment should be very carefully reviewed for compatibility with existing mining conditions prior to purchase.

The cost of capital and the various methods for financing capital requirements are beyond the scope of this chapter (see Section 6), but limitations in the ability to obtain capital can by itself dictate the mining method selected.

Operating costs include all those variable expenses associated with mine operation. The principal items are labor, supplies, power, haulage, preparation, and administration. Cost calculations are discussed in the following.

21.4.4.2 Mine Budget Development

It is necessary to develop accurate mining costs when comparing alternative mining systems or when considering the economic feasibility of a particular mining method in a given reserve area. A mine budget spreadsheet appears in Table 21.4.1 for a hypothetical underground coal mine for a 15-year operating life.

Before accurate mining costs can be developed, it is first necessary to develop a relatively detailed mine plan when the operations plan is completed, a cost sheet or mine budget summarizing all of the costs required for mining should be developed that includes (1) the cost for labor and related benefits, (2) the supply costs necessary to support daily mining activity, (3) power costs, (4) contracted services/equipment rental, (5) coal transportation costs, (6) preparation and tipple costs, (7) the cost of utilities, travel, and civil penalties, (8) the cost for administrative expenses, (9) the costs to pay coal royalty, taxes, and insurances, and (10) the capital costs expensed over the life of mine. Most of the detailed cost information needed for the mine budget will be generated during the development of the mine operations plan.

Once the mine costs have been established for a proposed mining plan in a given reserve area, the economic viability of the plan can be determined by calculating the present worth of the investment. Alternative mining systems can be compared by reviewing the respective cost sheets and comparing the present worth figures. Other factors that influence the economic viability of a proposed mine plan are (1) the availability of markets to sell the coal and (2) the availability of capital to invest in the project.

LABOR COSTS. Staffing required to operate the mine will be determined during the development of the mine operations plan. This information should be used to determine the corresponding labor costs. The mining industry is labor intensive, and the cost of labor represents a significant portion of the total mining cost.

The salary structure and benefit earnings for the general workforce and the mine supervisory/technical personnel are often quite different, particularly when the general workforce is a signatory member of the United Mine Workers of America (UMWA) or other union. For this reason, labor costs are usually calculated separately for the general workforce (non-exempt) and the supervisory or salaried (exempt) personnel. These figures are then combined to give the total labor cost.

If unionized, wage rates for union employees are specified in the contract, and these rates can be used to compute wages. Development of an average daily wage rate can simplify the computation process. The overtime cost is usually given as a percentage of the regular labor costs. Typically, overtime costs range from 5 to 15% of the regular wage costs. An overtime rate of 10% is used in the example shown in Table 21.4.1. The costs to provide benefits such as paid vacation days, paid holidays, additional pay received for unused vacation days, annual retrain-

Table 21.4.1. Mine Budget Spreadsheet

	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004
Tons Production—Raw	486,569	485,000	446,791	447,854	1,525,301	1,718,697	1,532,044	1,575,134	1,545,845	1,528,309	1,411,259	1,450,067	1,301,837	1,271,407	610,020
—Clean	280,400	288,480	228,359	246,164	959,800	1,099,600	1,057,500	1,128,700	1,098,900	1,024,400	984,900	985,300	890,300	842,100	414,800
—Reject	42.37%	40.52%	48.89%	45.03%	37.07%	36.02%	30.97%	28.34%	28.91%	32.97%	30.21%	32.05%	31.61%	33.77%	32.00%
Work Days	242	242	242	242	242	242	242	242	242	242	242	242	242	242	171
Employees—UMWA	82	82	82	82	129	129	129	129	129	129	107	107	82	82	60
Employees—Salary	17	19	19	20	24	24	24	24	24	23	22	22	22	20	15
Unit Shifts	954	948	948	834	948	1,404	1,402	1,412	1,412	1,314	1,175	1,170	1,076	668	322
Employee Shifts	23,934	26,886	27,419	29,282	34,872	40,729	40,729	40,729	40,729	37,534	34,340	34,340	32,564	28,396	14,108
Merchantable Tons—Day	1,163	1,062	838	773	3,966	4,544	4,370	4,664	4,541	4,233	4,070	4,071	3,679	3,480	2,426
—U. Shift	293.9	271.2	213.9	224.3	1,012.4	783.2	754.3	799.4	778.3	779.9	838.2	842.1	827.6	1,260.6	1,288.2
—Employee Shift	11.7	9.6	7.4	6.4	27.5	27.0	26.0	27.7	27.0	27.3	28.7	28.7	27.3	29.7	29.4
Labor UMWA—Regular	8.93	8.68	10.97	10.17	3.48	3.04	3.16	2.96	3.04	3.26	3.39	3.39	2.81	2.97	3.19
—Overtime	0.89	0.87	1.10	1.02	0.35	0.30	0.32	0.30	0.30	0.33	0.34	0.34	0.28	0.30	0.32
—Benefit Earnings	1.60	1.55	1.96	1.82	0.35	0.31	0.32	0.30	0.31	0.33	0.34	0.34	0.50	0.53	0.32
Taxes & Insurance	2.99	2.90	3.67	3.41	1.53	1.34	1.39	1.30	1.34	1.44	1.49	1.49	0.94	1.00	1.41
Health & Welfare	1.50	1.46	1.85	1.71	0.69	0.60	0.63	0.59	0.60	0.65	0.67	0.67	0.47	0.50	0.63
Total Labor UMWA	15.28	14.85	18.76	17.41	6.14	5.36	5.57	5.22	5.36	5.75	5.98	5.98	4.81	5.09	5.67
Labor Salary—Regular	2.96	3.27	4.13	4.08	1.24	1.08	1.13	1.05	1.08	1.11	1.11	1.11	1.20	1.18	1.27
—Overtime	0.15	0.16	0.21	0.20	0.06	0.05	0.06	0.05	0.05	0.06	0.06	0.06	0.06	0.06	0.06
—Related	0.71	0.78	0.99	0.98	0.30	0.26	0.27	0.25	0.26	0.27	0.27	0.27	0.29	0.28	0.30
Total Labor Salary	3.82	4.21	5.32	5.26	1.60	1.40	1.45	1.36	1.40	1.44	1.43	1.43	1.55	1.52	1.63
TOTAL LABOR & RELATED	19.10	19.91	25.64	25.38	7.74	7.85	8.16	7.65	7.85	7.78	7.41	7.41	7.79	7.20	7.30
Supplies—Operating	4.41	3.54	3.71	3.19	0.98	1.39	1.26	1.10	1.06	1.08	1.01	1.04	0.99	0.85	0.67
—Maintenance & Repair	5.77	5.19	5.58	4.73	3.56	3.66	3.40	3.18	3.20	3.37	3.23	3.30	3.33	3.34	3.18
TOTAL SUPPLIES	10.18	8.73	9.29	7.92	4.54	5.05	4.67	4.29	4.26	4.46	4.24	4.34	4.32	4.19	3.86
Power	1.00	1.16	1.45	1.25	0.98	0.97	0.95	0.96	1.01	1.08	1.06	1.08	1.12	1.17	1.17
Contracted Services/Equipment Rental	0.26	0.23	0.29	0.27	0.07	0.06	0.06	0.06	0.06	0.06	0.07	0.07	0.07	0.08	0.11
Transportation Coal—Rail	1.60	1.37	1.64	1.28	1.44	1.41	1.29	1.23	1.24	1.33	1.27	1.31	1.30	1.35	1.31
—Truck	2.26	1.80	2.16	1.69	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Coal Preparation & Tipple Allocations	4.00	3.56	3.55	3.04	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00
Utility, Travel & Civil Penalties	0.25	0.24	0.30	0.28	0.08	0.07	0.07	0.07	0.07	0.07	0.08	0.08	0.08	0.09	0.12
Administrative Expenses	2.00	1.78	1.78	1.52	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
Coal Royalty, Taxes, Insurance	0.44	0.39	0.39	0.33	0.44	0.44	0.44	0.44	0.44	0.44	0.44	0.44	0.44	0.44	0.44
Depreciation & Depletion	2.45	2.42	3.07	2.84	1.76	1.55	1.61	1.50	1.56	1.66	1.42	1.26	1.29	1.33	2.36
TOTAL COST	43.53	41.61	49.56	45.81	21.55	21.89	21.75	20.69	20.99	21.38	20.49	20.49	20.91	20.35	21.17

Assumptions: A typical mine budget spreadsheet is shown for a proposed union-affiliated (United Mine Workers of America) mine using continuous miner units and a longwall. Initial mine development is projected in 1990 with 2 continuous miner units. A longwall is added in 1994. See text for explanation (21.4.4.2). Costs in US dollars/ton (constant). Conversion factor: \$1/ton = \$1.10/t.

ing, and clothing allowances should be computed. The insurance costs to provide health insurance coverage and other company-paid insurances as well as the cost to pay taxes such as state and federal unemployment taxes, social security taxes, workers compensation, black-lung trust fund payments, and any other applicable taxes should be computed. The payments required to be made into the union health and welfare funds should be computed, if applicable. The total non-exempt labor cost is computed by combining the costs for wages, benefit earnings, taxes, insurance, health, and welfare.

The salaries projected for the supervisory/technical personnel can be combined to compute the labor costs for exempt personnel. The overtime percentage for salary personnel is usually smaller than it is for the non-exempt labor because not all supervisory personnel are eligible for overtime pay. An overtime rate of 5% is used in the example shown in Table 21.4.1. The additional labor expenses for taxes, insurance, pensions, etc. should be computed. In a well-established company, the related labor expenses are well known, and they are therefore often computed as a percentage of the salaries. In the example shown in Table 21.4.1, the related labor expenses are estimated to be 24% of the wages. The total labor cost for the supervisory/technical personnel is computed combining the costs for wages, overtime, and related expenses.

The total labor cost is easily computed by combining the non-exempt and the exempt labor costs.

SUPPLY COSTS. An accurate cost estimate of the supplies necessary to support routine mining activity is essential. Supply costs are usually separated into two main categories, operating supplies and maintenance and repair costs.

Operating supply costs include the costs of mine supplies used for roof control (roof bolts, straps, plates, timbers, cribs, etc.), ventilation (concrete block, motor mix, curtain, etc.), cutting (miner bits, roof drill bits, drill steel, etc.), and other miscellaneous items (rock dust, sand, etc.). Researching the operating supply costs at other mines in the same seam, and researching the operating supply cost at other mines in similar conditions, can help to develop an accurate operating supply cost.

Typically, the operating supply costs for continuous miner units ranges from $2.00 to $7.00/ton ($2.20 to $7.70/t). The average supply cost will depend largely on advance rates, seam height, and overall mining conditions. Operating supply costs tend to be more expensive when mainline headings are being developed as opposed to room and pillar panels. This is a result of the need to provide long-term roof control and to establish clearances in mainline belt and track entries, all of which work to increase costs. Operating supply costs required in areas outby the working face increase as the mine advances and the extent of the active workings increases.

The operating supply cost is much lower for longwall mining due to the increased tonnage rates coupled with the reduced demand for supplies (no roof bolts, straps, plates, curtains, etc.). Typically, the operating supply cost for longwall units ranges from $0.50 to $2.00/ton ($0.55 to $2.20/t).

Maintenance and repair supply costs include all parts and tools used to maintain, service, and repair all mine equipment located at the face and outby, including power centers, belt installations, belt drives, pumps, etc. The costs associated with equipment rebuilds or overhauls is generally capitalized and it therefore, is not included in the maintenance and repair supply costs.

Typically, the maintenance and repair supply costs for continuous miner units will range from $2.50 to $7.50/ton ($2.75 to $8.25/t). The average maintenance and repair supply cost will depend on the overall mining conditions and the type of mining equipment selected. The establishment of a good preventive maintenance program at the mine will help to reduce maintenance and repair supply costs as well as increase face availability.

Again, maintenance and repair supply costs are much lower for longwall mining. Typically, the maintenance and repair supply cost for a longwall unit ranges from $0.50 to $2.00/ton ($0.55 to $2.20/t). In some cases, it is desirable to collect or accrue maintenance and repair charges during mining of a longwall panel that will be used for repairs scheduled at the end of the panel. In this case, a fixed cost per ton is charged during longwall mining, and the money is set aside in a longwall accrual account. When the longwall panel is completed, the money is spent for pre-established maintenance and repair purposes such as refurnishing the longwall shearer, etc.

POWER COSTS. The annual power costs required to operate the mine will be determined during the development of the mine operations plan.

In the example shown in Table 21.4.1, the annual power cost estimate required to operate the proposed mine has been tabulated.

CONTRACTED SERVICES/EQUIPMENT RENTAL. Several mine services such as road maintenance, building maintenance, construction work, etc., may not be done within the company. Instead, these jobs may be contracted out to other companies when these services are required. Similarly, it may be advantageous to rent or lease certain equipment outright, particularly when the equipment will only be needed for short periods of time.

In the example shown in Table 21.4.1, road maintenance work, cleaning of sediment ponds, and other construction work needed periodically at the mining operation will be contracted out. Off-road haulers will be periodically used to haul wasterock away from the mine site and dozers will be periodically used to grade both the coal stockpile area and the wasterock disposal area. The cost estimates to provide these contracted services and to rent the equipment needed are combined.

TRANSPORTATION COSTS. The method of transporting coal to the preparation facilities and the corresponding costs are determined during the development of the mine operations plan. If the preparation facilities are located at the minesite, then there are no coal transportation charges.

In the example shown in Table 21.4.1, the run-of-mine coal initially is hauled by trucks to a nearby coal tipple where the coal is loaded into railroad cars and transported to the coal preparation facilities. Construction of the permanent materials handling system and loadout facility will be completed by the second year and, at this time, run-of-mine coal will be loaded directly into railroad cars, reducing the transportation costs.

COAL PREPARATION AND TIPPLE ALLOCATIONS. Coal preparation costs are determined during the development of the mine operations plan.

In the example shown in Table 21.4.1, the run-of-mine coal is being transported to a centralized coal preparation plant that processes run-of-mine coal from several mines located in the same general vicinity. The percentage of the total preparation costs attributed to each mine is charged back to the respective mines as well as any other applicable coal processing and/or tipple charges.

UTILITY, TRAVEL, CIVIL PENALTIES. The cost to provide utilities other than power, such as water, sewage, telephone, etc., should be estimated and included in the mine budget. Business travel for the purpose of continuing education, examining new equipment and mining methods, etc., should also be included. The cost of civil penalties resulting from state and/or federal citations should also be included in the budget. While no company wants to plan for civil penalties, they are a fact of life in mining, and they increase costs slightly.

If any other miscellaneous expenses are expected, they should also be included in the mine budget.

In the example shown in Table 21.4.1, the mine expenses pertaining to utilities, travel, and civil penalties are combined.

ADMINISTRATIVE EXPENSES. Several mine support services such as engineering, accounting, payroll, human relations, coal sales/marketing, etc., may be done at a local and/or a regional level for several different mining operations owned by a company, rather than at each individual minesite. In this case, a percentage of the total cost to provide these services is allocated to each mining operation.

In the example shown in Table 21.4.1, all of the services discussed above are being provided by a centralized or divisional support group that supports several mines located in the same general vicinity. The percentage of the total cost for this support group is charged to each respective mine.

COAL ROYALTY, TAXES, INSURANCE. It will be necessary to pay coal royalty charges (also in dollars per ton) if the mineral rights are not owned outright by the mine operator. Also the tax charges for coal severance tax, federal reclamation tax, property tax, state and federal income taxes, applicable state/county mineral depletion taxes, etc., should be estimated and included in the mine budget. The costs to provide coverage for fire insurance, accident insurance, property insurance, company vehicle insurance, etc., should also be estimated and included.

In the example shown in Table 21.4.1, the mine costs to pay coal royalties, taxes, and insurance are combined.

DEPRECIATION AND DEPLETION. Capital expenditures that are required over the life of the mine are determined during the development of the mine operations plan. Generally, the capital expenditures are broken down in four major categories.

1. Equipment purchases. New mining equipment that is purchased either to start the mining operation or new equipment that is purchased to replace old, outdated equipment.

2. Equipment rebuilds. Mining machinery is frequently overhauled or rebuilt on a regular schedule (2–3 years), and then put back into production. Equipment rebuilds costs typically average about 60% of the cost of a new piece of equipment, and this can represent a significant cost savings to the mine.

3. Sustaining items. The cost for items such as belt structure, belt, track, high voltage cable, and pipe is normally capitalized. The mine plan projections reflect the total feet of mine advancement during each year of mine development. This information should be used to determine the sustaining items required during each year of mine development.

4. Special projects. Capital expenditures to establish items such as mine offices/bathhouse facilities, site preparation, fan installations, substation installation, etc.

All capital costs should be tabulated by the year in which the expenditure is to be charged. This provides a summary of the annual capital costs required over the life of the mine. These expenditures will have to be financed in some fashion (either cash, loan, lease agreement, etc.) in the year in which they occur. However, if the large capital expenditures were all charged to the mine at one time, the mine cost sheet would reflect these large increases. For that reason, it is desirable to charge the cost of equipment back to the facility in a way that spreads the cost of the equipment over the life of the equipment (amortization). A convenient way to do this is to calculate the depreciation of the capital expenditures and then set the amortization schedule equal to the depreciation.

Depreciation is a tax deduction for the exhaustion, wear and tear, and obsolescence of property used in a business. Capital expenditures must be depreciated rather than expensed. The first step in computing depreciation is to determine the estimated useful life of the asset or its allowable depreciable life (refer to applicable tax codes). Once the depreciation has been calculated for all of the anticipated capital expenditures, the annual depreciation figures should be determined for each year of mine development. The annual depreciation figures are used in income tax computations in addition to producing the amortization schedule for the capital costs.

Depletion refers to the recovery of an owner's economic interest in mineral reserves through federal tax deductions related to the removal of the mineral over the economic life of the property. In the example shown in Table 21.4.1, there is no depletion credit because the reserves are not owned by the mine operator. In this example, a royalty is being paid to the owner of the mineral rights and the owner is entitled to the depletion credit.

Also, as shown, depreciation charges represent the cost of capital that is being charged to the mine.

21.4.4.3 Economic Viability

MARKET FORECASTING. Market forecasting is a subject in and of itself and is outside the scope of this chapter (see Chapter 2.3). However, it is important to note that the success of any mining scheme is subject to the market for the coal produced. Given the inherently cyclical nature of the coal market, the rapid and large impact of environmental regulation on its customers, the increasing globalization of the market, and its competition with other energy sources both domestically and abroad, market prices can change dramatically. That fact, when coupled with the large capital investments required and the long life of large reserve blocks, dictate that any economic decisions be based on the best market forecast available, and should include both best and worst case scenarios.

CASH FLOW. In order to compare alternative mining systems in a given reserve area or when considering the economic feasibility of a particular mining plan, the present worth of the investment is calculated (Section 6). Potential markets available must first be analyzed to determine the revenues expected from the sale of the coal. Annual expenditures (determined in the mine budget development) and annual coal revenues are computed over the life of the mine. The use of an after-tax discounted cash flow technique can then compare the present worth of the various alternatives. It is important to look at the analysis after taxes because of the impact taxes will have on a potential investment.

21.4.5 HUMAN RESOURCES

Human resources available can have a substantial impact on the viability of any mining method selected. As was stated earlier, mining is a labor-intensive business and will continue to be, despite the technological advances that constantly are being made. Furthermore, the best of technology is limited in effectiveness without the support of a motivated, well-trained, and dedicated work force.

Of the mining methods discussed, drill-and-blast mining is the most difficult to use in relation to the work force. This method has rarely been successful without the availability of a labor pool with substantial experience in these techniques, and that labor pool is shrinking rapidly or has effectively disappeared in most mining areas. For this reason, this technique is rapidly vanishing from the coal mining industry.

Continuous mining units are the industry standard, and all coal mining areas have a labor pool that is familiar with the operation and maintenance of this equipment.

Longwalls are relatively new to the United States coal industry, but most mining areas have at least one or two companies using this technology. Furthermore, there are many examples in the industry of companies with no prior longwall experience successfully implementing longwalls. This has been done by carefully selecting the personnel involved and mounting a strong training program prior to the arrival of the equipment, during installation, and continuing through startup activities.

Chapter 22.1
RAPID EXCAVATION

C.D. BREEDS AND J.J. CONWAY

22.1.1 INTRODUCTION

For the purposes of this section on innovative mining methods, *rapid excavation* is defined as underground excavation by means faster than conventional methods. However, many of the techniques and mining methods described below are well proven in civil construction and in a small number of US mines. A major innovation would be broader acceptance of these technologies by the mining industry.

Two major organizations in the United States promote the use of rapid excavation techniques for civil and mining applications. The Executive Board for Rapid Excavation and Tunneling Conferences (RETC) was established in 1971 to disseminate technical information in this rapidly advancing field of underground construction. The RETC and its proceedings provide a wealth of case study information related to site investigation, groundwater control, design and analysis, equipment, instrumentation, materials handling, and support for rapid excavation projects in soft ground and hard rock. The more recently established Institute of Shaft Drilling Technology (ISDT) provides a forum for discussing and reporting advances in shaft drilling. Short courses in mining techniques, shaft sinking, and boring techniques are provided through the ISDT and are highly recommended for engineers and owners planning major shaft projects.

This chapter draws extensively from publications of these organizations, field experience in rock cutting and excavation engineering, and input from equipment manufacturers and contractors. Each segment has been written to provide the reader with a description of the equipment used and an overall appreciation of selection methodology. Emphasis is placed on methods and equipment used for mine access construction and mine development. Rapid excavation methods associated with development and production mining (e.g., longwall mining, continuous mining, and stoping methods) are discussed elsewhere in the *Handbook* (see Chapters 17.4, 17.5, 18.1, 18.2, 19.1, and 20.1).

22.1.1.1 Rapid Excavation System Performance

A short section on system performance evaluation is provided for each rapid excavation method described. Simple empirical techniques, which utilize existing case study data and qualitative information, are used to estimate the probable range of system performance. This approach is considered to be applicable at a conceptual level of project planning. More detailed analyses, rock cutting experimentation, and equipment/system performance predictions are available from equipment manufacturers, but, due to space constraints cannot be adequately dealt with here.

22.1.1.2 Cost Estimating

Since the inception of mechanized mining, many papers have been published which enumerate the absolute cost advantage of mechanical vs. conventional construction. However, technical advancement in equipment design, owner experience, and increasing competition among contractors decreases the utility of absolute cost estimates especially when presented in a medium with an anticipated useful life of a decade or more. The approach in this chapter is, therefore, to present a list of the main components important to estimating project costs and to direct the reader to potential unit cost providers.

22.1.2 MECHANICAL ROCK CUTTING TECHNIQUES AND THEIR APPLICATION TO MECHANICAL MINING EQUIPMENT

The mechanics of mechanical rock breakage, and the parameters important to determining cuttability and production rates are presented in Chapter 9.2 of this *Handbook.* The objective of this chapter is to describe five basic cutting methods and their application to mechanical mining equipment.

These basic cutting methods, defined in terms of tool type, are illustrated in Fig. 22.1.1 and include:

1. Drag bit cutting.
2. Point-attack bit cutting.
3. Disk cutting.
4. Button cutting.
5. Roller cutting.

22.1.2.1 Drag Bit and Point-attack Bit Cutting

The application of both drag bits and point-attack bits is similar. The tools are inserted in tool holders (or boxes), which are integral parts of the cutting head, and may be held in place by a circlip or spring. Point-attack bits are commonly free to rotate in their holders. It has been claimed that this feature promotes more even tool wear (self sharpening) and better overall tool life, although research by Hurt and Evans (1981) disputes this. During cutting, the bits are pushed into the rock, developing cutting forces parallel to the direction of head rotation and normal forces parallel to the direction of head thrust. As these forces build up to critical values, a macroscopic failure surface develops ahead of the bit, and a piece of rock spalls away. The pick then moves ahead into the space left by the spalled chip until a new rock buttress is encountered, and tool forces again build up. The cutting process is thus a cyclical one with rapid fluctuations in tool forces. Adjacent bits produce parallel grooves, and interaction between these has an important influence on cutting efficiency.

Roadheaders use drag and point-attack bits almost exclusively. These tools also find application on tunnel boring machine (TBM) cutterheads, but in this role they are generally limited to machines operating in weaker formations.

22.1.2.2 Disk Cutting

Disk cutters (Fig. 22.1.1c) generally consist of solid steel alloy discs with a tapered cutting edge. The disk is mounted in a bearing and is free to roll in response to applied forces acting parallel to the rock surface. These rolling forces are analogous to the cutting forces applied in drag bit cutting.

Thrust and drag forces are applied to the disk through the bearing and act normal and parallel respectively to the rock surface. Disks used in practice may be of the simple type illus-

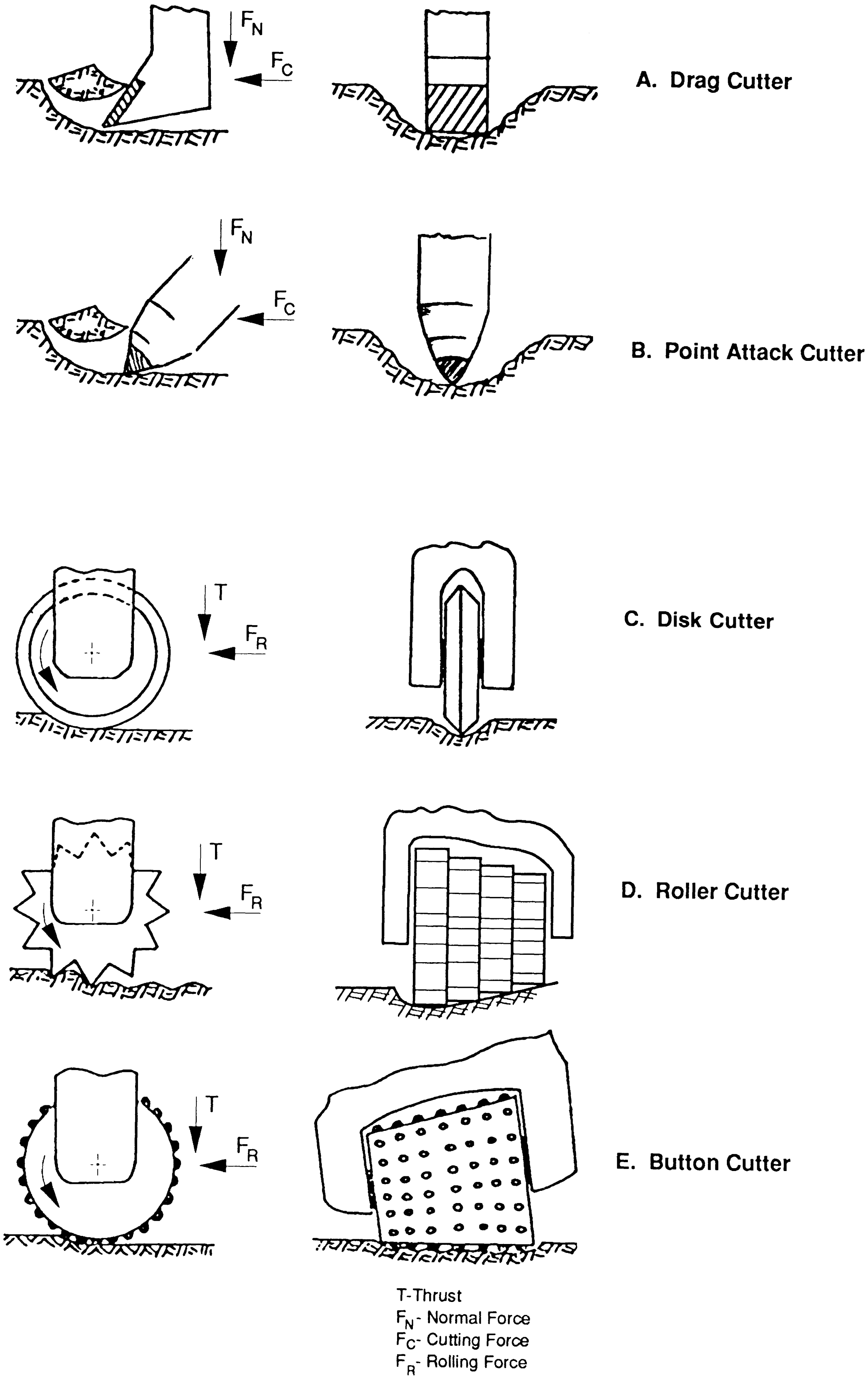


Fig. 22.1.1. Rock cutting techniques (after Roxborough and Rispin, 1973).

trated, or may consist of multi-edge varieties, including types with successively smaller disk diameters giving a tapered or conical arrangement. Frequently these multi-row disks employ carbide inserts with chisel points imbedded nearly flush with the circumference.

Simple disk cutters are used primarily on full face TBMs, and multi-row disks on raise boring machines (RBMs). Thrust forces acting on the cutting head push the cutter into the rock building up stresses which cause local rock failure. Because of the translatory motion of the cutting head, the disk rolls forward cutting a groove in the rock. As in the case of drag cutters, interaction between adjacent grooves has been shown to have an important influence on cutting efficiency.

22.1.2.3 Roller or Mill Tooth Cutting

Roller or mill-tooth cutting is similar to disk cutting except that instead of a tapered disc edge, the tool is equipped with circumferential teeth (Fig. 22.1.1d). As the cutter moves in response to rolling forces, each tooth in turn is pushed into the rock, acting like a wedge, and causing local failure.

22.1.2.4 Button Cutting

Button cutters consist of cylindrical or conical tool bodies inset with tungsten carbide buttons (Fig. 22.1.1e). The tool is mounted in a bearing in the same way as disk cutters or roller cutters and is free to roll in response to applied forces acting parallel to the rock surface. Thrust forces cause high stress concentrations beneath each button as they roll across the rock surface, resulting in local failure and pulverization of the rock. The area of influence of each button is small and results in a fine-grained product. Because the product size is small, specific energy requirements are high and button cutting is the least efficient of the rock cutting methods discussed. Button cutting is used in applications in which high rock strength and abrasivity preclude the use of other methods. These cutters also find application as reaming cutters used for final profiling on RBMs and TBMs.

22.1.3 BASIC METHODS OF PREDICTING INSTANTANEOUS CUTTING RATES

22.1.3.1 Introduction

When considering the feasibility or cost effectiveness of employing a mechanical excavation system, the central questions are (1) Can this machine cut this rock? (2) If so, how fast? and (3) What is the cost of maintaining this performance? Clearly there is a need for a reliable method of performance prediction. Two aspects of machine performance need to be assessed to answer the above questions. First, machine performance in terms of cutting rates or penetration rates must be assessed. Second, the overall system performance and reliability, with particular reference to those aspects that impact machine utilization, must be assessed. In the following discussion, methods of predicting or estimating cutting rates or penetration rates will be described, while methods of overall system assessment will be addressed in subsequent segments dealing with specific mechanical excavation methods.

Prediction of cutting rates requires information on rock material properties, rock mass properties, and machine characteristics. The link between these three groups of data is provided by what may be termed rock-tool or rock-machine interaction models, and the result of applying such a model is an estimate or prediction of performance. In the following discussion, prediction methods are placed into two broad categories depending on whether the interaction model is theoretical or empirical.

Before discussing performance prediction, the following terms must be defined:

Cutting rate (used in conjunction with roadheaders and boom-type tunneling machines) is the rate at which rock is excavated during cutting (volume excavated/cutting time), usually expressed in units of ft^3/hr (m^3/h). Care must always be taken to determine whether quoted "cutting rates" refer to what may be termed the *instantaneous cutting rate* (ICR) or the *operational cutting rate* (OCR). Cutting rates determined under highly controlled conditions, such as a research field test, in which cutting time is recorded as the actual time spent in cutting (determined from instrument measurements of power consumption against time) are instantaneous cutting rates. Under typical operational conditions, cutting time is generally taken as synonymous with utilization. Minor delays resulting, for example, from adjusting the boom position at the end of each cutting traverse, or reduced rates of production during final profiling, are neglected. Cutting rates determined using utilization as the cutting time are termed operational cutting rates. Clearly, performance predictions based on instantaneous cutting rates, without an appropriate cutting time correction, will be overly optimistic. Back analyses suggest that operational cutting rates commonly have values in the range of 0.45 to 0.60 times the instantaneous cutting rate. For final profiling, this figure may drop to 0.3, while during bulk production, an experienced operator may achieve a ratio as high as 0.85.

Specific energy is a commonly used measure of cuttability that is defined as the work done to excavate a unit volume of rock. In the context of rock cutting, specific energy should not be thought of as a fundamental property of the rock. Rather, it is a function of rock properties, cutting tool design, and cutting tool interaction, in the same way as compressive strength is a function of specimen size, shape, and test conditions. Measured specific energies are many times greater than theoretically determined values, the difference being accounted for in energy lost to frictional heating, vibration, and so on.

Penetration rate (used in conjunction with full-face shaft or tunnel boring machines) is the rate of advance measured during the cutting cycle, normally expressed in inches or feet (meters)/revolution or feet (meters)/hour. For practical purposes, instantaneous and operational penetration rates are considered equal.

Utilization is the time remaining for excavation when planned and unplanned machine stoppages have been accounted for. Stoppages are required for a variety of reasons including support installation, survey work, pick replacement, routine and non-routine maintenance, muck haulage delays, shift changes, and so on.

Advance rate is the rate of tunnel or drift advance, usually expressed in units of feet (meters)/day, feet (meters)/shift, etc., and is equal to

$$\text{OCR/face area} \times \text{utilization} \quad \text{or} \quad \text{penetration rate} \times \text{utilization} \tag{22.1.1}$$

22.1.3.2 Theoretical Models of Rock Cutting

Theoretical models have been proposed that attempt to analyze peak forces required, or work done, to excavate a unit volume of rock, and to relate these to fundamental rock properties such as shear and tensile strengths and internal friction angles. All these models have certain weaknesses that limit their usefulness for solving practical problems in machine design and performance. These weaknesses relate to a poor understanding of both the state of stress developed in the rock as a result of the applied forces and the mechanics of crack initiation and propagation. In addition, materials are generally considered to be homogeneous, and the important influence of pre-existing fractures is ignored.

Even with the simple case of a single cutting tool, a complex three-dimensional state of stress must exist in the rock around the tool tip. It is generally acknowledged that in the immediate contact area of the tool, intense crushing of the rock must occur, and that the properties of this crushed material differ markedly from those of the "intact" material. Theoretical approaches generally assume a simplified two-dimensional stress distribution, such as a point or line load, and neglect the properties of the crushed zone and the important role of this zone in transmitting stresses from the tool to the intact rock. Further, in practical cutting applications, multiple tools are arranged in a manner that promotes interaction between adjacent cuts, which has been shown to improve the overall efficiency of the system. This introduces a further level of complexity to the three-dimensional stress distribution that tends to be neglected in theoretical models.

Both brittle and plastic failure modes have been considered in theoretical rock cutting models, the appropriateness of each depending on the initial properties of the rock, and changes induced during cutting. Even in brittle rock, plastic deformation may occur in the intensely stressed zone adjacent to the tool tip. Failure criteria based on both tensile and shear stresses has been applied to rock cutting, although in practice, failure may be initiated in one mode and change to the other as the stress distribution changes during crack propagation. Thus a rigorous theoretical description of rock cutting must incorporate a sophisticated failure model, which accounts for both localized differences in material behavior and transient responses to a changing stress distribution. However, there can be little justification for developing or applying such a failure model until equally sophisticated three-dimensional stress distribution models are available.

APPLICATION OF THEORETICAL CUTTING MODELS TO ROADHEADERS. In the case of roadheaders, the limitations of theoretical models are compounded by the relatively large number of pick geometries available, the mode of roadheader operation (which involves continually varying normal forces), depths of cut and mode of cutting [i.e., sumping, traversing, etc. (Fowell and McFeat-Smith, 1976)], and a generally less-controlled cutting environment. Cutting theories applicable to roadheaders are not considered sufficiently developed at this time to be useful as prediction tools and are not discussed further here.

APPLICATION OF THEORETICAL CUTTING MODELS TO TUNNEL AND SHAFT BORING SYSTEMS. In the case of full-face excavation systems, theoretical modeling problems are less acute. Here, variations in cutter geometries are limited to variations in disk diameter and blade width. In addition, the cutting process is more controlled, involving relatively constant penetration rate and depth of cut, and only a single cutting mode. Because of this, some progress has been achieved in the application of theoretical cutting models, albeit oversimplified, to prediction of the performance of full-face TBMs. The better theoretical models of TBM performance are widely used as prediction tools, however, occasionally a significant deviation occurs. Whether the problem is in the model or in the ability of the sample or geotechnical data to represent the rock mass is not clear.

To be useful, such models must be able to predict thrust forces and rolling forces corresponding to specific depths of penetration in relieved cutting. Conversely, the models may predict achievable penetration given machine constraints governing available thrust and rolling forces. A model of this type will predict machine advance per revolution for a given machine power and tool spacing; a separate calculation of yield per revolution is not required.

Roxborough and Phillips (1975) have presented expressions for thrust force F_t and rolling force F_r, acting on a disk during unrelieved cutting:

$$F_t = 4\sigma_c \times \tan \theta/2 \times (Dp^3 - p^4)^{0.5} \tag{22.1.2}$$

$$F_r = 4\sigma_c \times p^2 \times \tan \theta/2 \tag{22.1.3}$$

where s_c, is unconfined compressive strength (UCS), q is disk edge angle, D is disk diameter, and p is depth of penetration. Based on breakage patterns observed during actual cutting tests, they concluded that the failure process is controlled by shear stresses acting on the plane connecting the apices of adjacent grooves.

Comparison of experimentally determined forces (for Bunter sandstone) with calculated values presented by these workers indicated good correlation. Farmer and Glossop (1980) have presented these equations (slightly modified in the case of F_t), and claim that expressions of this general form are in reasonable agreement with experimentally determined results.

Roxborough and Phillips (1975) also suggest that the optimum spacing/penetration ratio is given by

$$S/p = \sigma_c/\tau \tag{22.1.4}$$

where t is shear strength of the rock. Again, good correlation was demonstrated between calculated and observed *S/p* ratio for Bunter sandstone.

Eqs. 22.1.2 to 22.1.4, however, provide only a partial solution for prediction or head design. Using Eq. 22.1.4, the optimum spacing for a given penetration p can be calculated. Using this value of p, it should then be possible to calculate F_r and F_t for individual tools, using Eqs. 22.1.2 and 22.1.3. The total number of tools can be determined from the optimum spacing and head diameter, and hence the total torque and thrust requirements can be determined. However, these will be overestimated because Eqs. 22.1.2 and 22.1.3 apply to unrelieved cutting, whereas the actual spacing is selected to minimize tool forces.

Because of the current limitation of theoretical models, practical design approaches use empirical methods, as described in 22.1.3.3.

22.1.3.3 Empirical Methods of Predicting Instantaneous Cutting Rates for Roadheader and Boom-type Tunneling Machines

Because of the theoretical difficulties of modeling roadheader cutting performance, approaches to this problem are essentially empirical. It can be claimed that theoretical considerations have shed some light on which material and machine parameters have an important influence on performance, but while these parameters appear in many empirical performance equations, they are always associated with dimensionless constants derived from actual cutting trials or performance data.

The simplest empirical prediction methods are based on the extrapolation of performance records of specific roadheader models under specific geotechnical conditions that match those of the proposed site. While this approach has the advantage of simplicity, it also has a number of weaknesses. It is very difficult to collect high-quality roadheader performance data under other than the highly controlled conditions of a research project. Performance data collected under typical operational conditions,

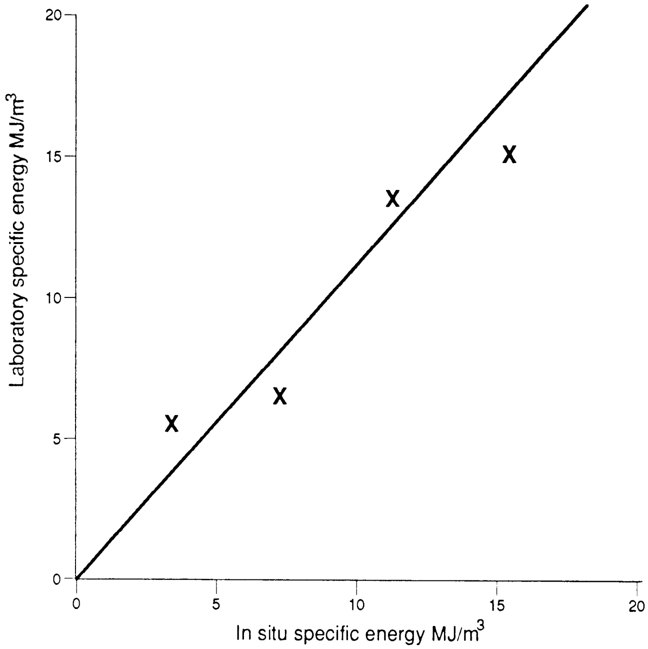


Fig. 22.1.2. Correlation of laboratory specific energy and in situ specific energy (after McFeat-Smith and Fowell, 1977). Conversion factor: 1 Btu/ft^3 = 0.0373 MJ/m^3.

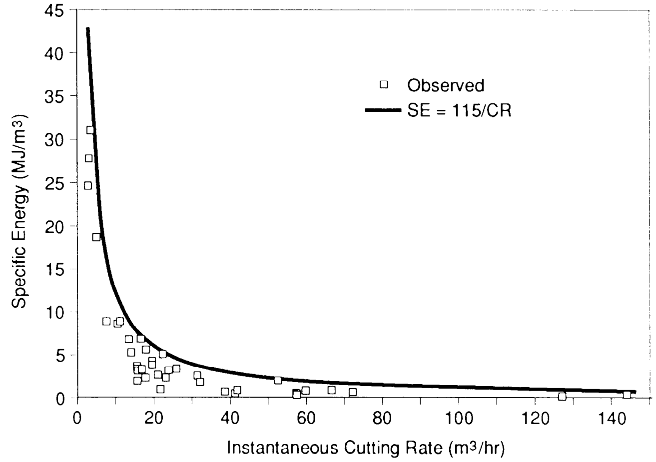


Fig. 22.1.3. Comparison of instantaneous cutting rates and specific energy requirements for a DOSCO MKIIA (after McFeat-Smith and Fowell, 1977). Conversion factors: 1 Btu/ft^3 = 0.0373 MJ/m^3, 1 ft^3/hr = 0.0283 m^3/h.

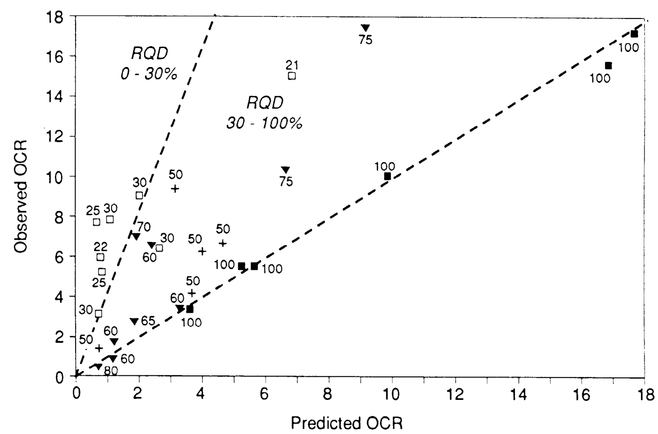


Fig. 22.1.4. Comparison of observed operational cutting rates and operational cutting rate predicted using data from McFeat-Smith and Fowell.

which constitutes the bulk of the data base, must be treated with caution. Furthermore, a good match between geotechnical conditions at the proposed site and a past site may not exist. In this case, the process becomes rather subjective, and there is no clear means of deciding what weight to attach to particular parameters.

In an effort to remove some of the subjectivity and identify important performance predictors, McFeat-Smith and Fowell (1977) investigated the relationships between rock index properties, laboratory specific energies (determined from small-scale cutting tests), in situ specific energies (determined from field-scale cutting tests), and instantaneous cutting rates for a variety of British Coal Measure rocks. Application of multivariate statistical methods to the results of laboratory tests enabled these workers to derive prediction equations that use a small number of index properties to predict specific energy requirements for rock cutting. These predictions were shown to correlate well with field specific energy measured during actual cutting trials (Fig. 22.1.2). Field specific energy was shown to be related to cutting rate using a very simple rock/machine interaction model. Fig. 22.1.3 shows a plot of measured in situ specific energy against cutting rates for Coal Measure strata reported by McFeat-Smith and Fowell (1977). Included on this plot is a theoretical curve developed from the rock/machine interaction model:

$$SE = HP/ICR \qquad (22.1.5)$$

where SE is specific energy, HP is head power, and ICR is instantaneous cutting rate. This curve provides a very good upper bound fit to the measured data, and in most cases shows that actual specific energy was less than predicted to achieve a given cutting rate. This may reflect the tendency of rock mass structural features to reduce specific energy requirements.

Fig. 22.1.4 shows a plot of observed OCRs from various sources, vs. predicted cutting rates using McFeat-Smith and Fowell's predictive equations and the simple rock/machine interaction model given by Eq. 22.1.5. Once again, in nearly all cases, actual performance was better than predicted. When *rock quality designation* (RQD) is considered, the data are seen to fall into two broad fields, although considerable scatter is still present. However, those data points for 100% RQD fall close to the lower bound (i.e., predicted = observed). Many of the points included in Fig. 22.1.4 are for roadheaders with up to twice the cutting-head power of the machine utilized in McFeat-Smith and Fowell's work, and cutting rates for this machine were predicted simply by inserting an appropriate value of HP in Eq. 22.1.5 It would appear, therefore, that these predictive equations may be applicable to a range of machines, provided that appropriate cutting time factor corrections are made. Also, direct determinations of specific energy using core grooving tests could be used in conjunction with Eq. 22.1.5 to predict instantaneous cutting rates.

The predictive equation approach has also been used by Aleman (1983) who has demonstrated good correlations be-

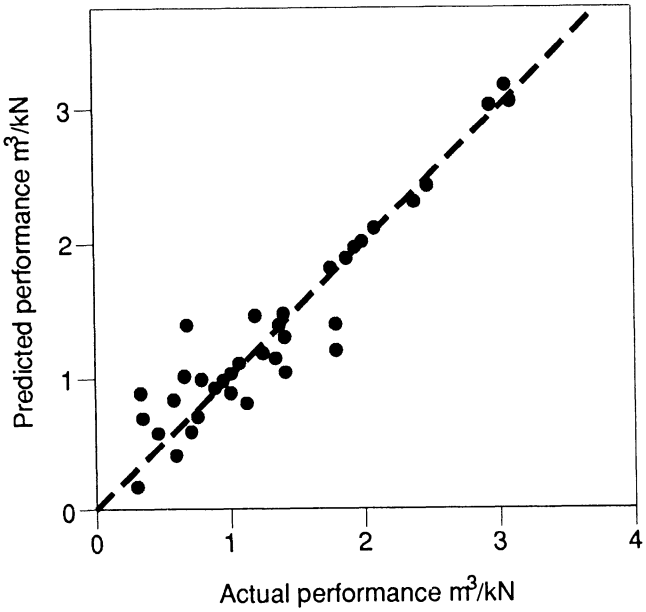


Fig. 22.1.5. Comparison of observed and predicted roadheader performance using Aleman's method (after Aleman, 1983). Conversion factor: 1 ft^3/lbf = 6.6367 m^3/kN.

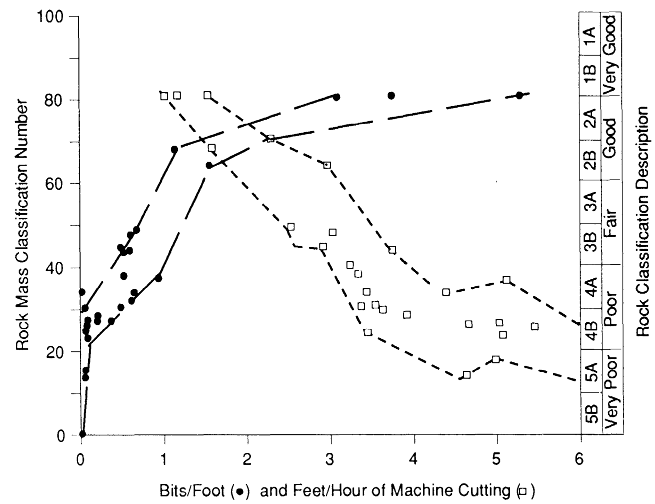


Fig. 22.1.6. Roadheader performance vs. rock class, P.21A Test, 2375 Level (after Sandbak, 1985). Conversion factor: 1 ft = 0.3048 m.

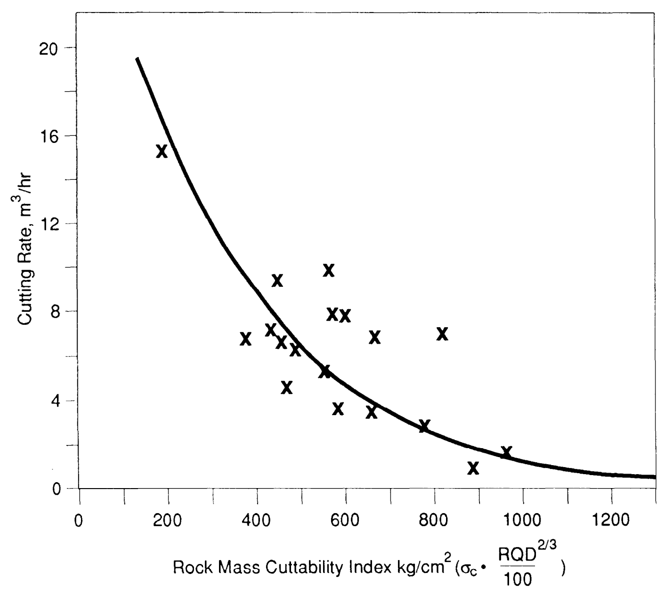


Fig. 22.1.7. Relationship between machine advance rate and rock mass cuttability index (after Bilgin et al., 1988). Conversion factors: 1 ft^3/hr = 0.02832 m^3/h, 1 psi = 0.0703 kg/cm^2.

tween predicted and observed performance for a variety of roadheaders (Fig. 22.1.5). Important aspects of Aleman's approach are the inclusion of RQD and an assessment of microfracturing in the predictive equations so that a secondary assessment of the influence of rock mass condition is not required. The predictive equations are used in conjunction with a more sophisticated machine model than that of Fowell and McFeat-Smith. This approach takes into account the limitations of available arcing force, and head rotation speeds, to derive instantaneous cutting rates.

It is worth noting at this point that many of the indices or parameters that appear in predictive equations, or are referred to in the literature as being significant predictors of roadheader performance, are often strongly correlated. Several parameters show strong correlation with unconfined compressive strength. Therefore it is not surprising that this parameter is shown to be significant in most of the studies undertaken. Where such correlations can be demonstrated, some predictive equations can be simplified to give expressions primarily in terms of unconfined compressive strength.

With good data collection, cutting rates can generally be correlated with rock mass and rock material properties at specific sites. The results of this type of study provide a useful means of predicting performance of a specific machine type under a variety of geological conditions. But since the results are not presented in terms of specific energy (McFeat-Smith and Fowell, 1977) or machine characteristics (Aleman, 1983), the results cannot, strictly, be extrapolated to machine types other than those for which they were derived. Two good examples of this type of study have been reported by Sandbak (1985) and Bilgin et al. (1988).

Sandbak demonstrated correlations between performance (operational cutting rate and bit consumption) and Bieniawski's *rock mass rating* (RMR) for a Dosco SL-120. Although the scatter in the results is rather large (Fig. 22.1.6), the overall trends are clear. Cutting rates are lowest in strong rock with few fractures, corresponding to high RMR values, and highest in low-strength, heavily fractured rocks corresponding to low RMR values.

Bilgin et al. (1988) collected detailed data on machine performance (for a Herrenknecht SM1), rock mass properties and rock material properties. Statistical analyses showed significant correlations between operational cutting rate and the product of UCS × $RQD^{2/3}/100$ (or *rock-mass cuttability index* RMCI, Fig. 22.1.7). These workers also investigated the applicability of the RMCI to prediction of cutting rates for a Dosco Mk 2A and a Pk 2r at other sites and found a reasonable correlation. These results suggest that the RMCI may be applicable to a variety of machines provided that an adequate allowance can be made for variations in head power. It should also be noted that the

Table 22.1.1. Summary of Empirical Roadheader Performance Prediction Methods

Reference	Rock Material Properties	Rock Mass Properties	Machine Characteristics	Rock-Machine Interaction Model	Comments of Applicability
Fowell and McFeat-Smith 1976, McFeat-Smith and Fowell, 1977	(1) Cone indentor hardness, shore hardness, Cementation coefficient, UCS (2) Cone Indenter hardness, 'plastic hardness.'	Not considered	Head power	Empirical—predicts specific energy and instantaneous cutting rate.	(i) Prediction equations can be expressed largely in terms of UCS. (ii) Needs CTF correction for OCR. (iii) Developed for DOSCO MK IIA, may be applicable to other light-medium-duty roadheaders.
Bilgin et al., 1988	Unconfined compressive strength (UCS)	Rock quality designation (RQD)	Not considered	Empirical and machine specific. Predicts OCR for lightweight-mediumweight roadheaders.	Developed for shield-mounted Herrenkneckt SMI. Needs correction for other machine powers and non-shielded or non-stelled machines.
Sandbak, 1985	UCS	Rock mass rating (RMR)	Not considered	Empirical and machine specific. Predicts OCR for lightweight-mediumweight roadheaders.	Developed for DOSCO SL 120. Would need correction for other machine powers/head configurations.
Aleman, 1983	UCS Cerchar abrasivity	RQD microfracturing	Torque, arcing force, head RPM	Empirical—uses machine specific constants linked to geotechnical properties in prediction equations for ICR.	Derivations of empirical constants not presented in cited references.
Farmer and Garritty, 1987	UCS	Deformation modulus	Head power, Energy transfer ratio.	Empirical—predicts cutting rate for selected energy transfer ratio.	
Hurt et al., 1981	Not considered	Not considered	Number & orientation of tools, radial distance, torque, and arcing forces.	Predicts ICR if tool forces are known.	Requires detailed knowledge of head design, machine characteristics.

Herrenknecht machine was shield mounted, a condition which generally results in a more rigid system and hence higher cutting rates than for a non-shielded machine.

The methods discussed above tend to be machine specific and/or utilize empirical constants that are machine specific. Other approaches to performance prediction, which are based on fundamental material properties, have been proposed. Farmer and Garritty (1987) make use of a strain energy approach to predict roadheader performance. Input data for this model includes UCS and deformation modulus of the rock mass. The machine model consists of head power coupled with an energy transfer ratio, which accounts for energy loss as a result of frictional heating, vibration and so on. Results indicate that as little as 1 to 2% of the available energy is actually used in rock breakage.

Table 22.1.1 summarizes the main features of the prediction methods discussed above, including limitations on applicability.

Currently, the best procedure for roadheader performance prediction may be the use of one or more of the empirical methods discussed above, provided that the limitations of these methods are understood. All of the approaches outlined above are deterministic in nature; however, actual case history data indicate considerable scatter even within a single rock type, which may approximate to normal or lognormal distributions. This is because of the inherent variability in rock material and rock mass properties and in operational conditions (including machine and pick condition, operator skill, etc.). A better approach to both the analysis of case history data and the prediction of machine performance is the use of probabilistic methods. Using this approach, geotechnical and operational variables are input to the analysis in the form of probability distributions, the output also being in the form of a probability distribution.

Empirical Methods of Predicting Instantaneous Cutting Rates for TBM and Shaft Boring Systems. Reliable empirical approaches to estimating penetration rates for full-face boring machines are generally far less complex than those previously described for roadheaders. Howarth (1986, 1987) reviewed seven published methods for the prediction of TBM advance rates and concluded that one simplified method (Farmer and Glossop, 1980), based on thrust per cutter and tensile rock strength, provided good correlation with actual penetration rates from 20 case histories. A complex model suggested by Lislerud et al. (1983) was considered to have the potential for more accurate penetration prediction but was discounted on the basis of cost. A brief review of these two methods is provided below.

METHOD DESCRIBED BY FARMER AND GLOSSOP (1980). Farmer and Glossop derived a relationship between the average cutter force F_L, the penetration per revolution P, and rock tensile strength s_{tf} equating the energy input per unit length of cut to the energy required to satisfy fracture surfaces in the rock:

$$P = K\, F_L/\sigma_{tf} \tag{22.1.6}$$

The value of K was obtained by least squares regression of eight cases where P, F_L, and s_{tf} were measured, so that in SI units:

$$P = 624\, F_L/\sigma_{tf} \tag{22.1.7a}$$

with P measured in mm/rev, F_L in kN, and s_{tf} in kPa. The equivalent equation using English units can be written:

$$\frac{P = 0.0158 \times F_L}{\sigma_{tf}} \tag{22.1.7b}$$

with P measured in in/rev, F_L in lbf and s_{tf} in psi.

A similar equation has been suggested by Graham (1976) based on use of the Robbins TBM in hard rock with unconfined compressive strengths ranging from 20,000 to 29,000 psi (140 to 200 MPa).

$$P = \frac{3940\, F_L}{\sigma_{cf}} \tag{22.1.8a}$$

where, s_{cf} is uniaxial compressive strength in kPa.
The equivalent equation written in English units becomes,

$$P = \frac{0.1\, F_L}{\sigma_{cf}} \tag{22.1.8b}$$

As simple ratios are typically used to relate tensile and uniaxial compressive strength (e.g., ranging from 1 : 10 to 1 : 20) either equation, possibly supplemented by additional performance data, may be used.

METHOD DESCRIBED BY LISLERUD (1983, 1988). Lislerud has developed a TBM performance prediction method based on rock mass factors (rock mass jointing, intact rock strength, brittleness, and abrasivity) and machine factors (thrust per cutter, cutter edge bluntness, cutter spacing, cutter diameter, torque capacity and RPM, and cutterhead curvature and diameter). Lislerud's equation for net penetration is written (in SI units):

$$P = i_b \times K_s \times K_d \text{ (mm/rev)} \tag{22.1.9}$$

where i_b is the basic penetration rate in mm/rev and is a function of the thrust per disk and the *drilling rate index* (DRI) as shown in Fig. 22.1.8 (DRI is based on testing described by the Norwegian Institute of Technology), K_d is a correction factor for cutter diameter; and K_s is a correction factor for joint rating and frequency (see Fig. 22.1.9).

As noted by Howarth (1986), this method requires a considerable amount of geotechnical and laboratory test data and is probably only suited to foliated, high-grade metamorphic rocks such as those found in Scandinavia. In less anisotopic rocks, use of the simpler relationships suggested by Farmer and Glossop (1980) and Graham (1976) is warranted.

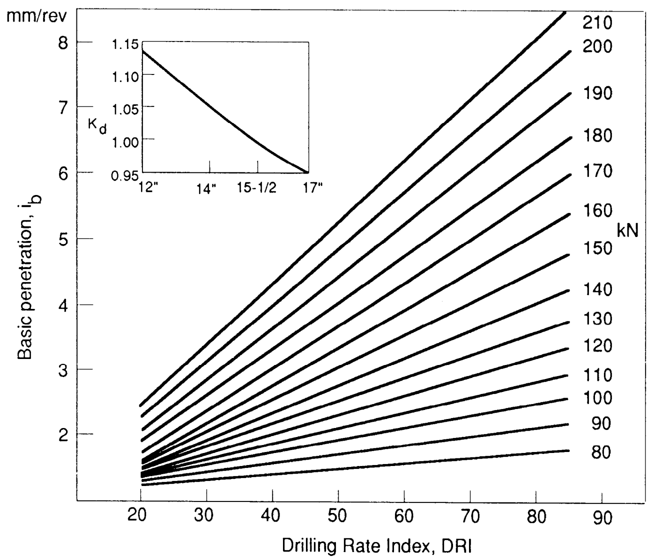


Fig. 22.1.8. Basic penetration as a function of drilling rate index (DRI), average thrust per disk, and cutter diameter (after Lislerud, 1988). Conversion factors: 1 in. = 25.4 mm, 1 lbf = 4.4482 N.

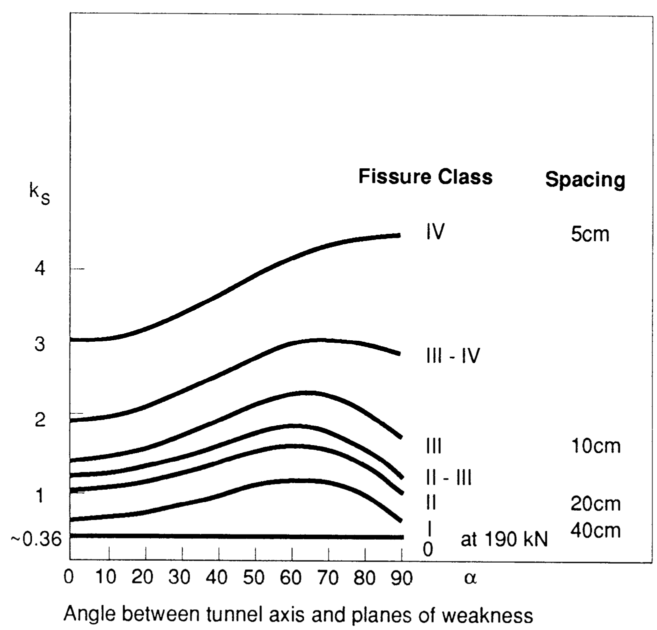


Fig. 22.1.9. Correction factor K_S as a function of fissure class and angle between tunnel axis and planes of weakness (after Lislerud, 1988). Conversion factors: 1 in. = 2.54 cm, 1 lbf = 4.4482 N.

22.1.3.4 Summary

Empirical methods currently provide the best means of estimating machine performance. Such estimates can be made directly based on previous experience in similar ground conditions or can utilize one of the predictive equations. In considering the results, the user should be aware of the limitations of each method. Input should also be sought from machine suppliers or

Table 22.1.2. Case Study Cycle Times for Blind Shaft Drilling

Cased Study Reference Source		D. Runge and J.T. Zeni (1987)	H.E. Hunter (1982)	H.E. Hunter (1982)	H.E. Hunter (1982)	Crookston et al. (1983)
Mobilization	(days)	5	12	7	4	
Drilled depth	(ft)	1050	2243	2188	2188	2371
Drilled diameter	(in.)	87 & 66	120	72	72	120
Drilling time	(days)	53	129	64	66	238
Casing length	(ft)	394	2194	2132	2131	2371
Casing diameter	(in.)	67	85	36	36	96
Prepare for casing	(days)	(inc)	5	2	1	—
Run casing	(days)	6	21	8	7	—
Grout casing	(days)	(inc)	10	5	7	—
Pump casing	(days)		6	4	2	—
Maintenance	(days)	7	(inc)	(inc)	(inc)	—
Total cycle	(days)	71	183	90	90	—
—Drilling	(ft/day)	20	18	34	32	22
—Casing and grout	(ft/day)	66	61	142	142	—
Formation		Weak shale	Sandstone and shales	Sandstone and shales	Sandstone and shales	Shale
Circulating		Reverse circulation	Reverse circulation	Reverse circulation	Reverse circulation	Reverse circulation

Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

specialist consultants before decisions on a machine's suitability for a particular application are made.

22.1.4 SHAFT CONSTRUCTION SYSTEMS AND EQUIPMENT

Three major rapid shaft excavation systems are described in this subsection, namely, blind shaft drilling, vertical tunneling mole or V-mole, and raise drilling. Data are included to provide the reader with a means of comparing the relative merits of each system and to assist with method selection.

22.1.4.1 Rotary Blind Drilling Systems

Large-diameter shaft drilling systems are an extension of conventional rotary drilling techniques used extensively for oil well boring. Extensive development work was pioneered by the Atomic Energy Commission (AEC) during the 1960s as part of the US nuclear testing program at the Nevada Test Site. Blind drilled shafts have since provided rapid access for underground mining projects throughout the world and are proven under a wide range of operational and site conditions (see Table 22.1.2 for summarized case study data). System components and operational considerations are described below. A generalized blind shaft drilling equipment set-up is shown in Fig. 22.1.10.

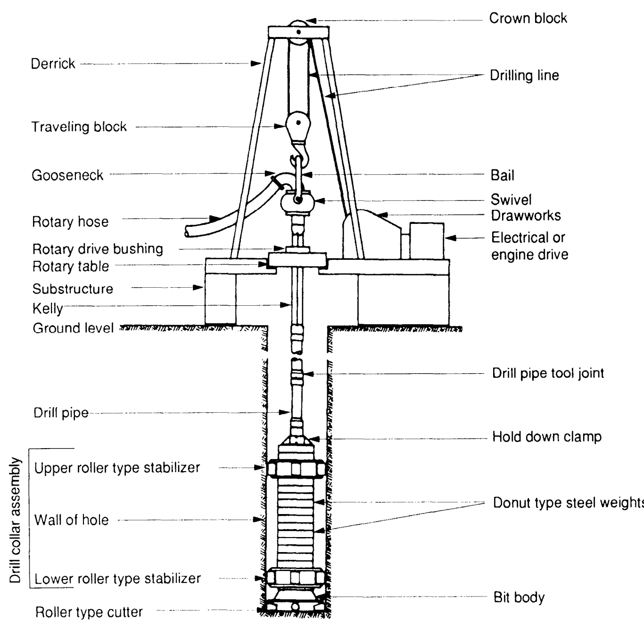


Fig. 22.1.10. Schematic blind shaft drilling equipment setup.

Shaft Collar and Foundation. A shaft collar is typically excavated using either an auger rig or conventional drill-and-blast mining during mobilization of the blind shaft drilling equipment. Collar depth depends on the overall length of the bottom-hole drilling assembly and is designed so that the assembly can be positioned below the drilling rig's rotary table. The collar may be lined with steel, shotcrete, or concrete, depending on ground conditions. A cast-in-place, reinforced concrete foundation will be designed to support the drill rig.

Drilling Rig. Major components of the drilling rig include a mast and substructure, drawworks and tugger hoists, rotary table, crown and traveling blocks, hook, swivel and kelly. The drill pipe and down-hole drill tools are supported by the mast through a conventional crown and traveling block/hook assembly. Static hook load capacities, for large-diameter blind-shaft drilling, may range from several hundred thousand pounds (kilograms) to more than a million pounds (half million kilogram) requiring more substantial masts than typically used for conventional rotary drilling. The rotary drilling motion is transferred from the rotary table to the drill pipe using a square section kelly bar. Mud is pumped to the down-hole system via a swivel located above the kelly.

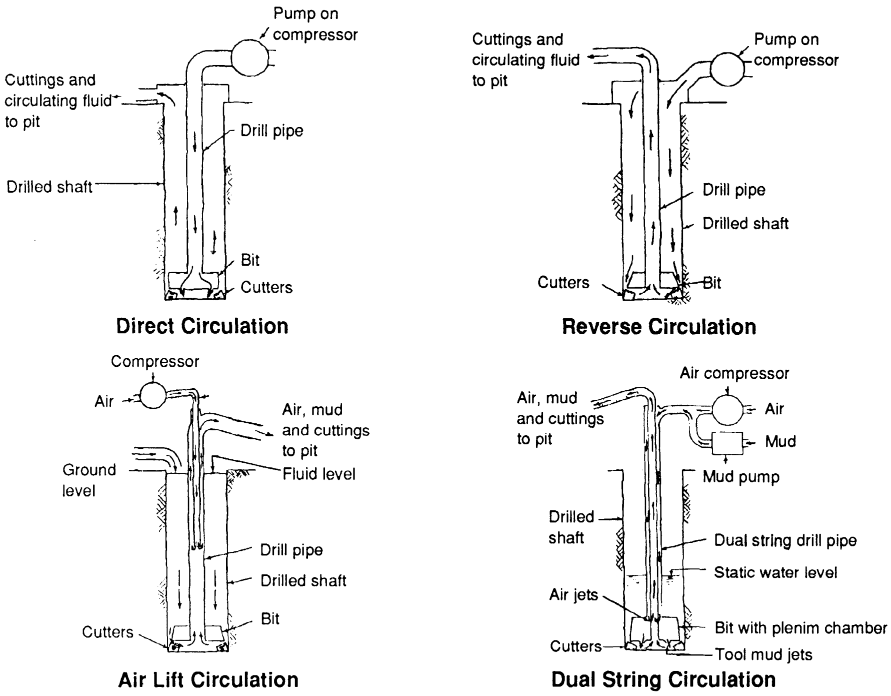


Fig. 22.1.11. Mud circulation systems for blind shaft drilling.

DOWNHOLE DRILLING TOOLS. These include the drill pipe and bottom-hole drilling assembly. Drill pipe is selected based on maximum tensile and torsional loading conditions and consideration of mud circulation requirements; data sheets are available from the pipe manufacturers. A common US drill pipe, used for large-diameter blind shaft drilling, has an outside diameter of 13 3/8 in. (340 mm), weighs 90 lb/ft (134 kg/m), and requires in excess of 100,000 ft-lb (136 kN-m) of makeup torque.

The *bottom hole drilling assembly* includes a drill bit, mandrel, stabilizers, and donut weights. Cutters are mounted in cutter mounts or saddles and bolted to the underside of the flat-bottomed drillbit. The drill bit is in turn bolted to the mandrel that serves as a base for locating donut weights. Donut weights are added to provide the required normal force at each cutter (typically from 10,000 to 20,000 lb (44.5 to 89 kN) per cutter), which is a function of the relative hardness of the formation to be drilled. Donut weights are secured to the drill pipe by a hold-down clamp which forms the top of the bottom-hole assembly. Stabilizers may be added directly above the mandrel and toward the top of the bottom-hole assembly to assist directional control.

OPERATIONS. An important element of blind shaft drilling is to maintain a straight, vertical alignment. The key to effective directional control involves minimizing the fraction of total effective bottom-hole assembly weight transferred to the bit while maintaining acceptable penetration rates. This maximizes the pendulum effect experienced by the bottom-hole assembly and, in conjunction with stabilizers, provides a straighter shaft.

Most large-diameter drilled shafts use air-assisted reverse circulation. Drilling mud is added to the hole at ground level and circulated through the cutters and up the inside of the drill pipe. Air is added inside the drill pipe causing a density imbalance that induces flow rates sufficient to remove the drill cuttings. Air-assisted reverse circulation and alternative mud circulation systems have been described by Lackey (1982) for blind-shaft drilling at the Nevada Test Site (Fig. 22.1.11). The "dual-string airlift reverse circulation" method incorporates a plenum chamber in the drillbit and dual string drill pipe. Mud and compressed air are pumped down the outer annulus of the drill pipe to the plenum chamber where the air separates from the mud. Mud flows through the plenum chamber and is forced through fluid jets located in the bit in order to clean the hole bottom. The air is routed through the top of the plenum chamber into the inside of the inner string and induces upward flow of mud and drill cuttings. Mud flows from the top of the drill pipe into the first mud pit where up to 90% of the cuttings drop out. Overflow from the first mud pit goes to the second and third where the remaining cuttings settle out.

GROUND SUPPORT. A cake of mud is deposited on the shaft walls during drilling. The thickness and strength of this cake may be optimized based on cuttings removal and ground support requirements to prevent mud loss during drilling. Control of mud density and head (i.e., height of mud column) acting against this impermeable surface permits shaft excavation in poor ground.

SHAFT LINING. The final lining for a blind drilled shaft typically consists of a ring-stiffened steel liner. This is equipped with external guides to facilitate grout line deployment and is outfitted internally.

Liner sections are fabricated offsite in lengths compatible with transportation and handling requirements; sections may be up to 60 ft (18 m) in length. The liner is lowered into the mud-filled hole using either casing jacks or the drill rig. Each liner section is aligned and welded to the one below to provide a completely water-tight membrane. Loads on the casing jacks may be limited, in the case of deeper shafts, by capping the bottom liner section so that the liner can be "floated" into place. Water is pumped into the casing to control buoyancy as the capped liner is lowered into the shaft. When liner installation is complete, the annulus between the steel liner and the shaft wall is filled with grout.

Other lining systems, compatible with the concept of rapid, remotely controlled placement, have been developed but are not

Table 22.1.3. Breakdown of Blind-drilling Project Costs

Component	Percentage of Total Project Cost
Management	2
Engineering and administration	8
Drilling	25
Casing	24
Welding casing	10
Cementing casing	7
Site construction	3
Drilling mud	7
Drillbit cutters and stabilizers	6
Drillhole surveys	1
Other (inc., radiographic inspection, crane services, tools)	7

in common use. These include slip forming (both bottom-up and top-down), jump forming, precast concrete cylinders, and remotely placed shotcrete. Finally, if ground conditions permit, conventional lining placement techniques (e.g., involving slip forming, jump forming, or shotcrete, placed using a galloway) may be used.

BLIND-DRILLING SYSTEM PERFORMANCE PREDICTION. The performance of a blind shaft drilling system is simply defined as a function of the operational penetration rate and the system utilization. Penetration rate is, in turn, a function of the geology (rock strength, fracture frequency, hardness, abrasivity); drill assembly (cutter type, size, and spacing; cutter load; and available torque); and cuttings removal system. Moss et al. (1987) developed a *drillability index* to predict relative penetration rates in rock rated from exceptionally poor (Q = 0.001 [Barton et al., 1974]) to fair (Q = 10) with intact strengths ranging from 3000 to 43,000 psi (21 to 297 MPa). Variations in average penetration rates were smaller than expected and correlation with the index was poor. Several important observations were made as a result of this case study:

1. Lower than predicted rates of penetration in clay were thought to be due to plugging of the bit. Associated problems included reduced mud circulation rates and poor control of shaft verticality.

2. An increase in the rolling resistance when drilling in rock of lower rock mass quality was thought to result from fragments that were larger than those normally resulting from the cutting action.

The results of this study serve to illustrate the potential shortcomings in generic performance prediction systems. The operational penetration rate for a blind-shaft drilling project can be estimated using available equipment specifications and the simple relationships suggested by Farmer and Glossop (1980) and Graham (1976). Adjustments are required for available thrust, calculated as 30% of the sum of the weights of down-hole components corrected for buoyancy from the drilling mud and imperfect hole cleaning (Maurer, 1962).

High utilization factors are possible for well-planned operations; for example, Hunter (1982) reports utilization factors of 74, 80, and 79% for the three Crown Point Project shafts.

BLIND-DRILLING COSTS. Table 22.1.3 provides a breakdown of blind-drilling cost components and their relative contribution to total project costs. It can readily be seen that the project costs are dominated by the acquisition and installation cost of the steel casing used for final lining.

22.1.4.2 Other Blind Boring Systems

A manned blind-shaft boring (BSB) system, with operators located underground, was developed and demonstrated by the Robbins company in the late 1970s. A 24.5-ft (7.5-m) diameter shaft was sunk to a depth of 587 ft (179 m), proving the application of horizontal tunnel boring methods to vertical shaft boring. The BSB used a full-face rotary cutterhead equipped with 56, 13-in. (330-mm) disk cutters with a conveyor bucket elevator mucking system (Fig. 22.1.12). A second generation shaft boring machine (SBM) (Fig. 22.1.13) has subsequently been developed by a Redpath/Robbins team to mechanically excavate a 20- to 24-ft (6- to 7.3-m) diameter shaft. The second generation machine incorporates a 10-ft (3-m) diameter cutter wheel fitted with 28, 15.5-in. (394-mm) cutters for a 20-ft (6-m) shaft diameter and a boom-mounted clam-type mucking unit.

Hendricks (1985) presents a detailed prediction of the performance of the SBM in hard rock (18,000 to 30,000 psi or 124 to 207 MPa) shaft construction. Fig. 22.1.14 has been reproduced from this text to simply illustrate the predicted cost and schedule advantages of the SBM over conventional shaft mining methods.

22.1.4.3 Vertical or V-mole System

The V-mole is a horizontal tunnel boring machine modified for vertical deployment by the German firm Wirth. First introduced to construct large diameter (16 to 21.5 ft, or 4.88 to 6.55 m) shafts in Europe in the early 1970s, it has since been used to construct four 23-ft (7-m) diameter shafts for an Alabama coal mine. Summary data for these case studies are presented in Table 22.1.4. The equipment, shown in Fig. 22.1.15, consists of the cutterhead, drive assembly, thrust and directional control cylinders, kelly, gripper assembly, and work platforms.

The gripper assembly, consisting of 8 to 12 grippers, provides resistance to the thrust and torque required for rock boring. Rotary motion is transmitted from the gripper assembly to the cutterhead through a kelly and up to 6 thrust cylinders are controlled by the operator to provide the required penetration rate. Muck is removed into a pilot hole by scrapers located on the cutterhead. The shaft lining is placed from work platforms located above the gripper assembly providing a continuous excavation/lining cycle. Services and support equipment are deployed using techniques traditionally associated with conventional shaft sinking.

In addition to the obvious differences in down-hole tools, the V-mole requires a pilot hole for muck removal. However, offset reaming can be controlled by the operator, allowing some deviation from pilot hole direction to be accommodated during sinking.

22.1.4.4 Raise Boring Systems

Raise boring has been used to drill shafts ranging in inclination from horizontal to vertical with a majority of applications involving large-diameter holes steeper than 45° (see Table 22.1.5 for summarized case study data). System components and operational considerations are described below. A generalized raise boring equipment setup illustrating the available range of deployment methods is shown in Fig. 22.1.16.

CONVENTIONAL RAISE BORER.

Setup and Equipment—A *raise collar* is sometimes used to support the raise drill and provide sufficient vertical clearance for the reaming head during holing through. Conventional shaft collar excavation and lining techniques typically are used to construct the raise collar. The raise drill is positioned on a steel substructure anchored to the collar lining after completion of the pilot hole.

The *pilot hole* can be drilled during mobilization of the major plant using rotary drilling methods, however, more pilot holes are drilled with the raise drill after it is positioned. Since the

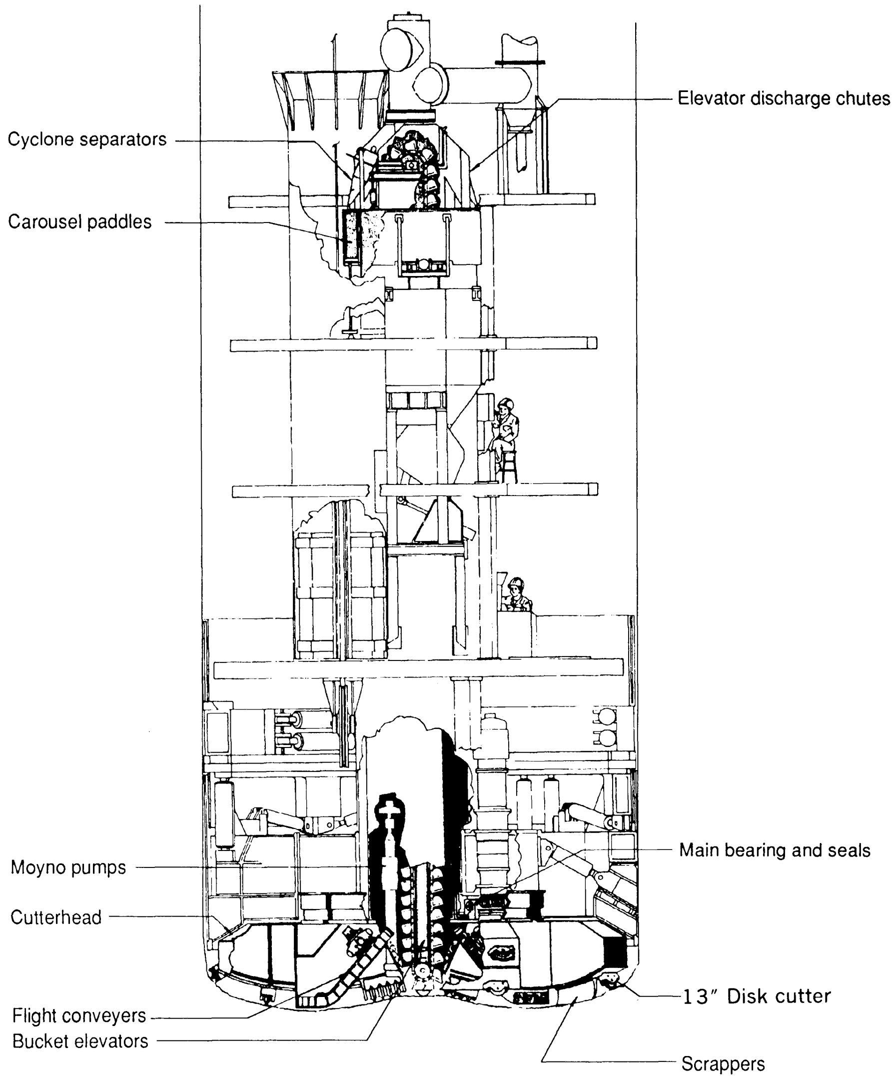


Fig. 22.1.12. Blind-shaft-borer down-hole equipment (*courtesy:* Robbins Co., Seattle, WA, Model 2418B-189).

accuracy of this borehole directly influences shaft verticality, great care is taken to ensure that the pilot hole is drilled within owner-specified tolerances. Directional surveys are routinely performed every 25 or 50 ft (7.5 to 15 m) using conventional oil well survey techniques.

A skid-mounted raise drill typically consists of a crosshead, positioned between two cylindrical guideposts, and hydraulic rams that lift the head and apply thrust, via the drill rods, to the reaming head. Rotary motion is provided through a ring gear and motor/reducer installed on the crosshead.

The raise borer reaming head is transported underground during drilling of the pilot hole. Underground set-up involves assembly and attachment of the reaming head to the raise drill rods and preparation of the underground mucking system.

Operations—Initial raise drilling is conducted at low thrust and RPM until all cutters on the reaming head are in contact with the rock. Thrust and rotation are varied by the raise operator to provide optimum penetration for each stratum encountered. Cuttings fall through the reaming head and are removed at the shaft bottom using the mine's mucking equipment; common practice involves maintaining the cutting pile flush with the mine roof to reduce airborne dust levels.

The reaming head is immobilized and suspended from steel fixtures cast into the shaft collar concrete following break-through. It can be removed from the shaft, after the raise drill and steel foundation are demobilized, using a small crane.

A comprehensive paper by Worden (1985) provides a detailed, pragmatic description of activities involved in the reaming cycle.

Ground Support—Raises are commonly unlined since raise boring is typically used in relatively competent formations. However, if ground conditions or use dictate the installation of a final lining, there are a few rapid lining systems to choose from.

DOWN-REAMING RAISE BORING. A small proportion of raise-bored shafts have been excavated using an upward-drilled pilot hole with downward reaming to full shaft diameter. The advantage of reduced pulling capacity, associated with down-reaming, is more than offset by problems associated with muck handling (muck must travel down the pilot hole alongside the drill pipe) and cutter replacement.

DOUBLE-PASS RAISE BORING. Shaft size limitations in conventional raise boring are primarily associated with exponentially increasing torque requirements and the cost/feasibility of machine and tooling upgrades. As noted earlier in this chapter, the required torque is a function of the sum of the individual cutter rolling resistances multiplied by the mount radius and the torque required to overcome friction between the drill pipe and pilot borehole. As both the number of cutter kerfs and the aver-

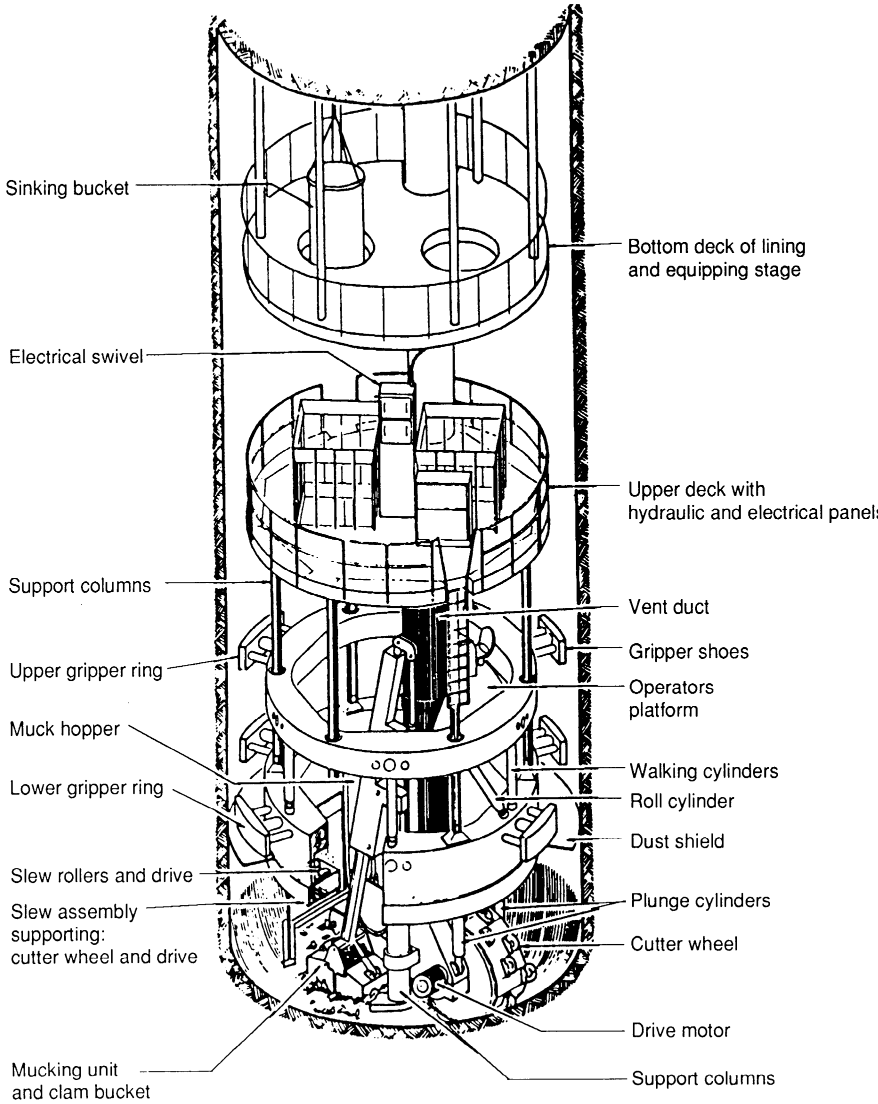


Fig. 22.1.13. Shaft boring machine, 20 to 24 ft (6 to 7.3 m) (*courtesy:* Robbins Co., Seattle, WA).

age mount radius are proportional to the shaft diameter, the required cutterhead torque is proportional to the square of the shaft diameter. Excavation of shaft diameters beyond the single-pass capability (machine and drill-pipe capacity) of onsite equipment can be accomplished by reaming a smaller shaft with second-pass reaming to full size. Reaming heads should be selected to optimize the torque distribution and drilling load. Stabilizers are essential when second-pass reaming in longholes to prevent drill string whip. Alternatively, the raise may be sequentially reamed in short sections.

TWO-STAGE SEQUENTIAL REAMING-HEAD RAISE BORER. The two-stage sequential reaming head was first introduced in South Africa in 1985 as an alternative method of reaming larger-diameter, deeper shafts in hardrock. In operation, the smaller head is sumped in and advanced about 3 ft (1 m) (Fig. 22.1.17). This head is then retracted, and the remaining shaft area is reamed by the larger head; this cycle is repeated until the raise is completed. The first sequential-head raise borer, using an 8-ft (2.44-m) primary and 12-ft (3.66-m) secondary reamer, was used to bore three, 300-ft (91-m) deep ventilation raises at the Western Areas Goldmine. Wirth, in conjunction with Rocbor Raise-boring and Mining Contractors, subsequently developed the HG330 raise borer 14-ft (4.3-m) primary and 20-ft (6-m) secondary reaming head) that has been used during construction of raises up to 3200-ft (975-m) deep (Schmidt and Fletcher, 1987). Many of the early problems, typically associated with a protypical method, have been resolved according to Schmidt and Fletcher (1987). Outstanding issues, traditionally associated with large-diameter shaft construction, include excavation in poor quality and blocky ground, presence of large groundwater influx, and the impact of pilot hole accuracy on final shaft verticality. This latter constraint currently restricts most large-diameter raise developments to shafts that will not be outfitted.

Blind Raise-boring—Blind raise boring, or boxhole drilling has been used in the South African goldfields to construct small-diameter (5 to 6 ft [1.52 to 1.83 m]) raises up to 500 ft (152 m) in length. Raise boring can be conducted with a predrilled pilot hole or blind (without pilot hole) at advance rates between 4 and 6 ft (1.22 to 1.83 m)/hr (Friant et al., 1985).

Raise-boring System Performance Estimation—System performance, at a conceptual level, can be estimated using the case study data in Table 22.1.5 or by using Eq. 22.1.7 or 22.1.10. For example, a mine requires construction of 400-ft (122-m) long, 10-ft (3-m) diameter ventilation shafts in granitic rock with uniaxial

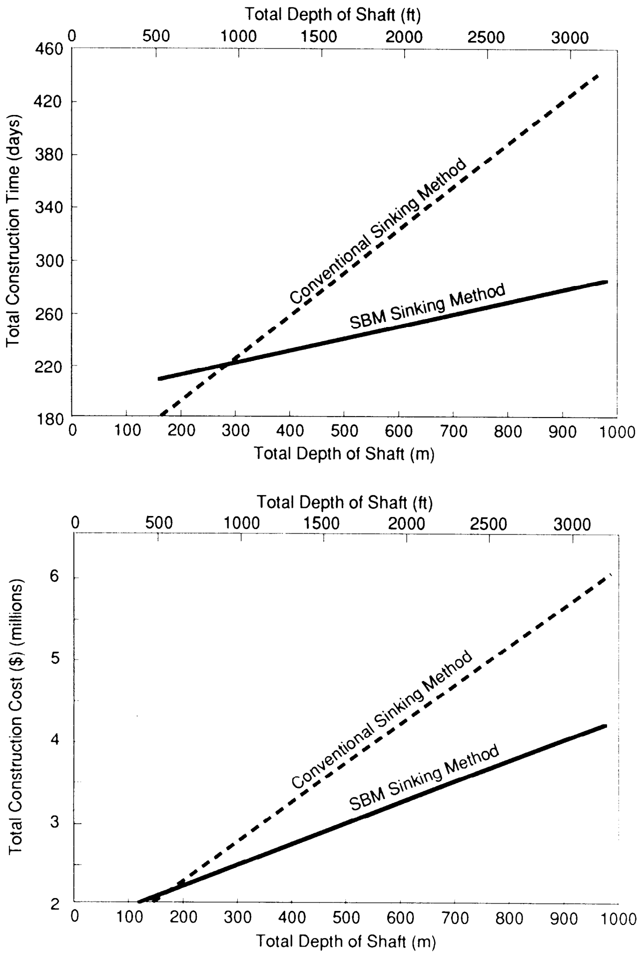


Fig. 22.1.14. Predicted performance of shaft boring machine (after Hendricks, 1985).

compressive strength ranging from 24,000 to 30,000 psi (165 to 207 MPa). Excavation is in good quality rock, and stress or structurally controlled failures of the rock mass are not considered to be a problem for this shaft diameter. From the case study data, pilot-hole drilling rates can be expected to range from 6 to 9 fph (1.83 to 2.74 m/h), with reaming rates from 2 to 3.7 fph (0.61 to 1.13 m/h). The project appears to be well within the range of a number of raise borers, including the RBM-7, and will require a nominal 12.25- or 13.75-in. (311- or 349-mm) pilot hole; pipe size and tooling requirements should be selected by the contractor.

Example 22.1.1. Estimate the penetration rate for a raise drill; using the following machine data available from the manufacturer,

Maximum available thrust	2,000,000 ft-lb	8,900 kN-m
Maximum torque	2,150,000 ft-lb	8,474 kN-m
Maximum recommended thrust/cutter	25,000 lbf	111 kN

Solution. Assume s_{tf}/s_{cf} for granite = 1 : 14.
From Eq. 22.1.8,

$$P = \frac{0.1 \times 25{,}000}{27{,}000} \qquad P = \frac{3940 \times 111}{186 \times 10^3}$$

$$= 0.093 \text{ in./rev.} \qquad = 2.35 \text{ mm/rev}$$

$$P_{HR} = \frac{0.093 \times 8 \times 60}{12} \qquad P_{HR} = \frac{2.35 \times 8 \times 60}{12}$$

$$= 3.7 \text{ fph, or} \qquad = 1.13 \text{ m/h.}$$

which must be derated for utilization

Raise-bore Costs—Economic factors of mechanical raise boring have been discussed by Norman and Dye (1978), and a detailed breakdown of a raise-bore contractor's bid is presented in Nash (1982). Table 22.1.6 details the essential elements of a raise boring project for which costs should be estimated, and

Table 22.1.4. Case Study for V-mole Shaft Construction

Shaft Location (reference source)	Drilled Diameter (in.)	Drilled Diameter (m)	Depth (ft)	Depth (m)	Excavation Duration (days)	Advanced Rate Average (ft/day)	Advanced Rate Average (m/day)	Advanced Rate Maximum (ft/day)	Advanced Rate Maximum (m/day)	Type Formation
Raine, 1934	192	4.88	758	231	119	6.4	1.95	NA	NA	Sandy shale and sandstone
(Table 1 in	192	4.88	797	243	42	18.9	5.76	NA	NA	(relative composition in
reference)	192	4.88	745	227	25	29.6	9.02	NA	NA	Bruemmer, and Wollers,
	192	4.88	643	196	15	44.4	13.53	NA	NA	1976, Table 1.)
	192	4.88	748	228	28	26.3	8.02	NA	NA	
	192	4.88	991	302	31	31.6	9.63	NA	NA	
	192	4.88	748	228	37	20.1	6.13	NA	NA	
	192	4.88	735	224	21	35.4	10.79	NA	NA	
	192	4.88	748	228	29	25.9	7.89	NA	NA	
(Table 2 in ref-	258	6.55	1532	467	44	32.1	9.78	NA	NA	Sandy shale and sandstone
erence)	258	6.55	1358	414	30	45.3	13.81	NA	NA	
	258	6.55	358	109	15	23.2	7.07	NA	NA	
(Table 4 in ref-	276	7.01	1548	472	67	23.1	7.04	105.8	32.25	Sandy shales, sandstones with
erence)	276	7.01	1978	603	69	28.7	8.75	95.3	29.05	65% quartz content, shales
	276	7.01	1877	572	32	59.0	17.98	122.9	37.45	and coal seams.
	276	7.01	1929	588	32	62.2	18.96	107.0	32.61	(Maximum UCS = 27,000 psi, or 186 MPa)

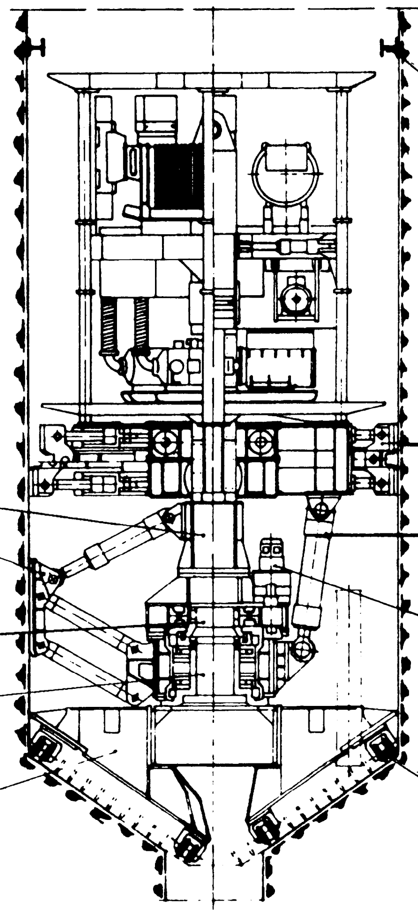


Fig. 22.1.15. Wirth V-mole—vertical mole (after Raine, 1984).

Fig. 22.1.18 provides a rough guide to project costs as a function of raise diameter and required torque.

22.1.4.5 Selection of Shaft Construction Method

Selection of the appropriate shaft construction technique for a given site involves an in-depth analysis of site geomechanical and geohydrological conditions, design criteria (e.g., diameter and depth, shape, use, life, etc), and availability and location of equipment, plus a determination of their relative impact on project cost and schedule. Cost is nominally the overriding consideration; however, timing may be critical, and higher shaft construction costs may be off-set by rapid access to the ore body.

SITE-SPECIFIC DATA REQUIREMENTS. The geotechnical data set required for shaft design and construction bid package preparation is essentially the same for all shaft construction methods. The site investigation program should incorporate a fully logged core hole located within one shaft diameter of the proposed shaft centerline. (Note: a borehole on shaft centerline is preferred; however, the verticality of this borehole may impact the construction method if a pilot hole is required.) This borehole should be geotechnically logged (e.g., recovery, RQD, discontinuity description, lithology, rock description) and core samples selected for testing (e.g., uniaxial compression, slake durability, swelling, hardness, petrographic analysis). Samples may also be required by cutter manufacturers for proprietary drillability testing. Hydrogeologic data should be obtained by profiling the borehole using a down-hole (in line with drilling) or straddle packer test tool. Data analysis will provide the location and magnitude of groundwater inflows, shaft wall stability and rock support requirements, shaft liner design loads, rock mass data for blast design, and estimates of drill penetration rates, bit wear, cuttability, and overall suitability for mechanical excavation. These services are available from mining geotechnical companies and have been described extensively in the literature (e.g., see Hoek and Brown, 1980; Goodman, 1980).

METHOD SELECTION GUIDELINES. As previously noted, the overall selection of an appropriate shaft construction method must be made on the basis of cost and schedule. This analysis may be supported initially by conceptual designs as described in this section. However, a final decision should be made in consultation with personnel experienced in the field application of each technique.

Factors influencing the selection of each shaft construction method have been assembled and are presented in Table 22.1.7. At a broad conceptual level, the blind-shaft drilling method is preferred for conditions where

1. Freezing would be required for groundwater control during conventional shaft excavation.
2. Shaft lining requirements dictate the use of a fully hydrostatic steel/concrete lining.
3. Rapid access outweighs added cost.
4. Disturbance of the surrounding rock is a prime criterion.
5. Access is not available for subsurface muck removal.

Use of the blind shaft borer (BSB) or V-mole may be preferable where

1. Adverse impacts of groundwater inflow can be economically mitigated prior to construction (e.g., through grouting) and/or are relatively easy to handle during excavation.

Table 22.1.5. Case Study Data for Raise-drilled Shaft Construction

Raise Borer Reference, Model Number	Inclination (deg)	Rock Type	UC Strength (ave) (psi)	Tensile Strength (ave) (psi)	Raise Depth (ft)	Machine Diameter (in.)	Pilot Hole Diam (in.)	Drilling Time		Pilot Rate (fph)	Reaming Rate (fph)
								Pilot Hole (hr)	Ream (hr)		
Harrison, 1972	90		43,000	3,071	167	60			44		3.8
Dynatec, 1989, RBM-7	90	Limestone	20,000	1,429	318	60	11.00	30	41	10.60	7.76
Harrison, 1972	90		41,000	2,929	319	60			84		3.8
Harrison, 1972	90		35,500	2,536	452	60			136		3.3
Dynatec, 1989, RBM-7	90	NA	17,500	1,250	631	60	11.00	40	47	15.78	13.43
Dyantec, 1989, RBM-7	90	Quartz Diorite	26,000	1,857	1,040	60	13.75		204		5.10
Dynatec, 1989, SBM-1000	90	Dolomitic Limestone	30,000	2,143	1,122	60	12.25	148	14	7.58	8.14
Woodward, E.M., 1983	90	Granites	20,250	1,446	234	72	12.00	58	83	4.03	2.8
Dynatec, 1989, RBM-7	90	Quartz, Biotite	23,000	1,643	358	72	11.00	26	40	13.77	8.95
Folwel, 1972	21.5	Arkose Granite	22,500	1,607	405	72	12.25	55	111	7.36	3.6
Dynatec, 1989, RBM-7	90	NA	23,000	1,643	1,350	72	11.00	132	165	10.23	8.18
Dynatec, 1989, RBM-7	90	NA	26,000	1,857	1,366	72	13.75		279		4.90
Dynatec, 1989, RBM-7	90	NA	17,500	1,250	239	84	11.00	19	26	12.58	9.19
J.S. Redpath, 1989	81	Norite, Gabbro	40,000	2,857	365	84	9.00	320.9	109.4	1.14	3.3
J.S. Redpath, 1989	65		26,500	1,893	390	84	11.00	44.7	166.7	8.72	2.3
Dynatec, 1989, RBM-7	90	Quartz Diorite	26,000	1,857	1,075	84	13.75		210		5.12
J.S. Redpath, 1989	90	Fine-grained Calcite	14,500	1,036	1,932	84	13.75	703.5	1,206	2.75	1.6
J.S. Redpath, 1989	90	Fine-grained Calcite	14,500	1,036	1,981	84	13.75	478.2	1,048	4.14	1.9
Woodward, E.M., 1983	90	Granites	20,250	1,446	50	96	12.00	9.3	29	5.38	1.7
Woodward, E.M., 1983	90	Graintes	20,250	1,446	182	96	12.00	67.5	82.7	2.70	2.2
Dynatec, 1989, SBM-1000	90	Dolomitic Limestone	22,000	1,571	211	96	12.25	18	24	11.72	8.79
J.S. Redpath, 1989	90	Quartz Diorite, Porphyry	34,000	2,429	373	96	11.00	87.9	262.2	4.24	1.4
J.S. Redpath, 1989	90		40,000	2,857	410	96	12.25	45.5	101.5	9.01	4.0
Dynatec, 1989, SBM-1000	90	NA	26,000	1,857	505	96	12.25	67	56	7.54	9.02
Dynatec, 1989, RFB-7	90	Schist, Argillite	17,500	1,250	620	96	11.00	40	57	15.50	10.88
J.S. Redpath, 1989	90		20,000	1,429	1,025	96	12.25	103.1	321.5	9.94	3.2
J.S. Redpath, 1989	67	Silicified Limestone	44,300	3,164	675	108	13.75	131.1	606.1	5.15	1.1
Dynatec, 1989, RBM-7	90	Anorthositic Gabbro	23,000	1,643	125	120	11.00	9	34	13.89	3.68
Dynatec, 1989, RBM-7	90	Anorthositic Gabbro	23,000	1,643	128	120	11.00	16	43	8.00	2.98
Dynatec, 1989, RBM-7	90	Quartz, Biotite	23,000	1,643	220	120	11.00	24	50	9.17	4.40
J.S. Redpath, 1989	86.5	Schistose, Siliceous Sil	13,000	929	335	120	12.25	42	209.5	7.98	1.6
J.S. Redpath, 1989	70	Silicified Limestone	14,300	1,021	805	120	12.00	252.5	486.1	3.19	1.7
J.S. Redpath, 1989	65	Quartz Diorite	47,000	3,357	866	120	13.75	146	508	5.93	1.7
	90		27,500	1,964	325	144	13.75		173		1.9
	90	Massive Limestone	33,500	2,393	350	144	13.75		175		2.0
	90	Salty Limestone	18,000	1,286	625	144	13.75		219		2.9
	90	Limestone, Shale	15,000	1,071	1,000	144	13.75		250		4.0
ROB-81F (EMJ, 1981)	90		27,500	1,964	2,300	144	13.75	660	817	3.48	2.8
J.S. Redpath, 1989	90	Fine-grained Calcite	14,500	1,036	936	150	13.75	97.7	920	9.58	1.0
J.S. Redpath, 1989	90	Fine-grained Calcite	14,500	1,036	940	150	13.75	132.3	739	7.11	1.3
Dynatec, 1989, SBM-1000	90	NA	26,000	1,857	1,017	168	12.25	167	1,017	6.09	1.00
Dynatec, 1989, RBM-7	90	NA	23,000	1,643	372	192	13.75	60	620	6.20	0.60
IR RBM-211 (Nash, 1982)	90	Limestones, Sandstones	12,500	893	210	243	13.75		126		1.7
IR RBM-211 (Nash, 1982)	90	Sedimentary Shales	5,000	357	210	243	13.75		97		2.2

Conversion factors: 1 in. = 25.4 mm, 1 psi = 6.895 kPa, 1 ft = 0.3048 m.

2. Rock quality permits stand-up times compatible with the V-mole's mining cycle.

3. Rapid access outweighs added cost.

4. Minimum disturbance of the surrounding rock is a prime criterion.

5. Access is available for setup and underground muck removal.

6. Geologic structure permits pilot-hole drilling to the tolerances required by the shaft designer (V-mole only).

7. Immediate access to drilled strata for geologic logging, instrument installation, and testing is required.

8. There are no existing mine openings (BSB only).

Raise drilling may be preferred where

1. Site conditions (e.g., rock quality and absence of large groundwater inflows) provide for stable excavation conditions.

2. Access is available for setup and underground muck removal.

3. Geologic structure permits pilot-hole drilling to the tolerances required by the shaft designer.

4. Design requirements (e.g., diameter, depth) are compatible with available equipment capabilities.

Under these conditions, raise drilling may offer a less costly alternative to all other methods of shaft construction.

22.1.5 RAPID EXCAVATION SYSTEMS FOR HORIZONTAL AND SUBHORIZONTAL MINE DEVELOPMENT

Three rapid excavation systems are described in this segment, namely, full-face tunnel boring, mobile miners, and roadheaders or boom-type tunneling systems. Data are included to provide the reader with a means of comparing the relative merits of each system and to assist with method selection.

22.1.5.1 Full-face Tunnel Boring Systems

Full-face boring systems or TBMs have been in common use in civil tunneling for many years but are used less frequently in mining projects. Nevertheless, TBMs in European coal mines and the TBM at the Stillwater Mining Company's platinum/palladium mine near Nye, MT (Tilley, 1989) are proving the

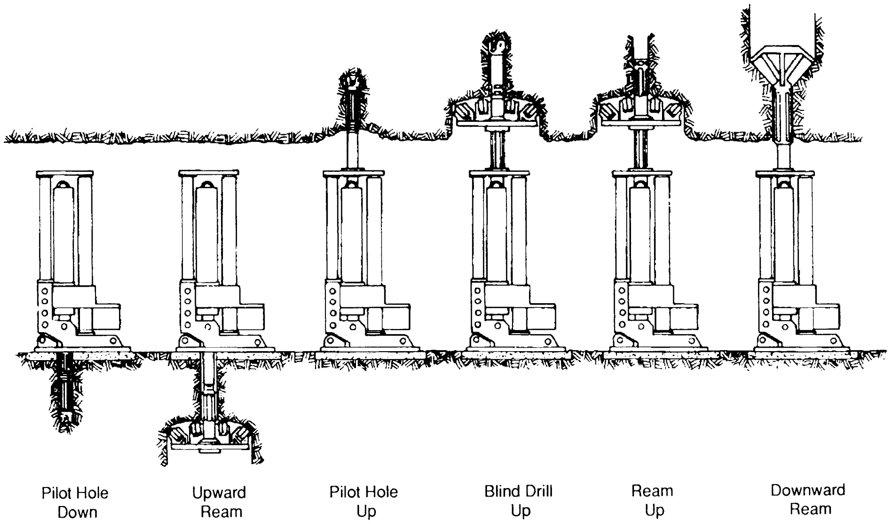


Fig. 22.1.16. Alternative raise boring methods (after Friant et al., 1985).

viabilty of TBMs in mine development. These experiences may spur more widespread acceptance and utilization of this method of mechanical drivage within the industry. Constant development and improved tooling have resulted in machines that are capable of advancing large-diameter openings in strong igneous and metamorphic formations at rates that compete favorably, and in many cases exceed, conventional drill and blast methods. Tilley has reported advance rates up to 165 fpd (50 m/day) and 700 ft/wk (213 m/wk) for operations at the Stillwater mine.

In addition to high advance rates, TBMs leave a smooth profile and minimize ground support requirements. A disadvantage of TBMs is their wide turning circle, although a range of mini-fullface machines are available that have smaller turning radii. The high initial cost of these machines is balanced by low running costs compared to drill and blast excavation systems. Fig. 22.1.19 shows the cutterhead, shields and operator console of a hard-rock double-shield machine designed for use in claystone, mudstone, sandstone and shale excavation. Fig. 22.1.20 shows a large-diameter hard-rock TBM equipped with a slotted shield to facilitate support installation. Breakthrough of a 20-ft (6-m) diameter drivage in limestone is also shown. System components and operational considerations are described below. A generalized TBM equipment setup is shown in Fig. 22.1.21.

SETUP AND EQUIPMENT. Full-face boring machines consist of a rotating cutting head fitted with disk cutters, drag bits, button bits, or various combinations of these. Advanced machines are available on which the tool type can be changed and tool spacing varied. These developments have arisen from the need for machines that can cope with a variety of poor ground conditions. The cutting head may be an open structure with spoke-like cutting arms, or it may completely conceal the face except for muck-removal openings and access ways for tool maintenance. The open-type head gives better access to the face and tools, and can be used with a forepoling arrangement.

Cutting forces are provided by the head rotation, while normal forces are provided by the thrust of the machine against the tunnel face. Reaction to this thrust is provided by grippers mounted on the TBM body, which in turn react against the tunnel sidewalls. Mucking is performed by buckets mounted on the periphery of the cutter head, and muck is removed via a central conveyor system.

OPERATIONS. Preparations for TBM excavation typically involve portal construction, placement of a concrete pad on which the TBM will be assembled, and installation of support services and equipment. Careful excavation, including the use of controlled blasting techniques, is usually required to mine the setup area to the tight tolerances required. Placement of a thin, 1- to 2-ft (0.3- to 0.6-m) concrete layer against the start-up face is recommended to reduce out-of-balance loads. TBM excavation is a continuous process, with cutting, mucking, and support installation proceeding concurrently. As the cutting head rotates, it moves forward, reacting against the grippers. The grippers are repositioned periodically when they reach the limit of their travel. On Wirth and Jarva TBMs, the grippers are also used to steer the machine. Robbins and Domag TBMs steer while boring using a floating main beam.

Mucking in the immediate vicinity of the face is done by buckets located on the head periphery, and a central conveyor system that moves the muck through the body of the machine to a bridge conveyor. The bridge conveyor allows access for track laying and service installation without disrupting the mucking operation. A variety of mucking systems can be used to haul muck to the surface, but typically conveyors or shuttle trains are used. Adequate muck removal rates are critical to optimum face advance rates.

The orientation of the TBM is controlled by the grippers, in conjunction with a laser beam and microprocessor-controlled guidance system. These allow precise positioning of the machine, but problems may still be encountered in weak or soft ground in which the effectiveness of the grippers is greatly reduced. In these situations, 3-dimensional orientation control is facilitated by TBMs, that steer while boring.

SUPPORT SERVICES AND EQUIPMENT. The trailing gear following the TBM provides an interface with the support utilities and equipment installed in the completed tunnel up to 650 ft (200 m) behind the machine. Components, and their configuration, are primarily a function of tunnel size and may include:

1. Bridge conveyor required to transport muck from the TBM to the muck cars.

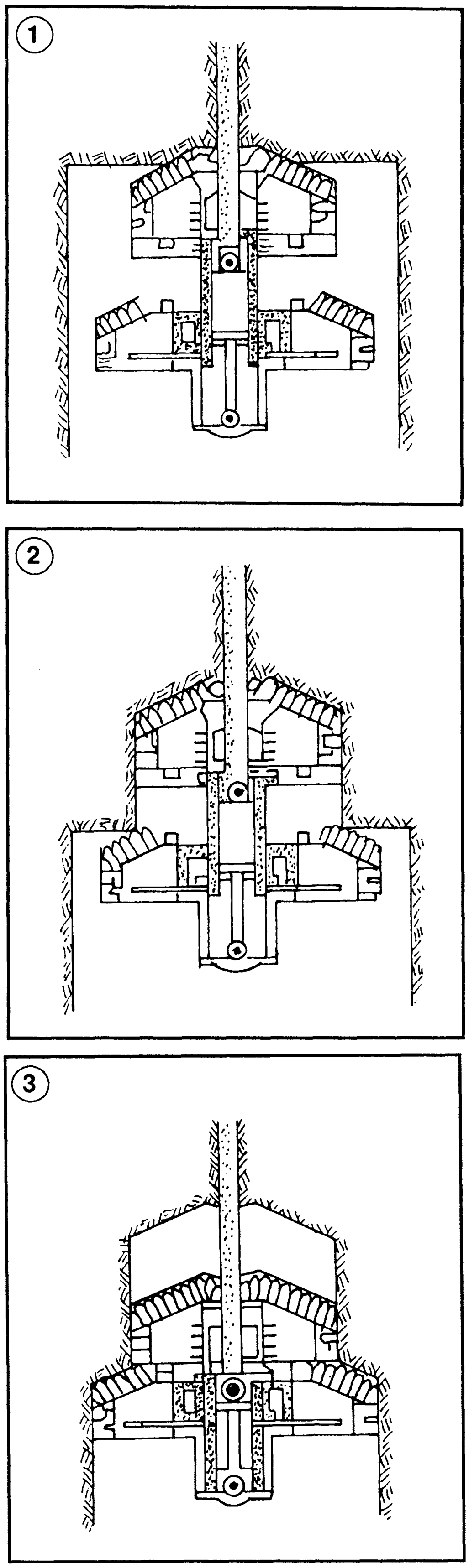


Fig. 22.1.17. Sequential reaming sequence (after Schmidt and Fletcher, 1987).

2. Dual-track rail system with remote muck car loading and handling (for larger diameter tunnels).

3. Telescopic ventilation line, auxiliary fan(s), and scrubber system.

4. Hydraulic power unit, transformer(s), and trailing cable.

5. Supply hoist for unloading and moving supplies.

6. Rock drill (for bolt installation), rock drill power unit, and rock bolt supplies storage.

7. Mechanical shop and cutter and supply storage area.

System components are briefly described below; more detailed descriptions and specifications are contained in the references cited at the end of this chapter.

Haulage System—Rail systems are commonly used to transport muck during TBM excavation. Various types of muck cars and trailing gear are available that facilitate continuous loading. Additional details, regarding conventional muck haulage systems, are presented in Chapter 9.3. Alternate muck removal systems, incorporating pneumatic or hydraulic transport of crushed muck, have been used in a small number of civil tunneling projects.

Ventilation System—The ventilation system typically consists of 24- to 36-in. (610- to 914-mm) ventilation pipe with in-line booster fans located several thousands of feet (meters) apart along the tunnel. The system is designed and deployed to maximize the cross-sectional area available for mucking equipment and is normally configured to exhaust air to the portal (Chapter 11.6).

Electrical System—The electrical system typically consists of a high-voltage feeder cable with stepdown transformers mounted on the trailing gear and at strategic locations along the tunnel to service ventilation fans, lighting, pumps, etc. (Chapter 12.4). Total installed power requirements can be roughly approximated at twice the predicted TBM consumption.

GROUND SUPPORT. Rock support requirements (Chapter 10.5) for hard-rock TBM drivages are generally minimal and estimates have suggested that the savings in rock support costs (compared to drill and blast) can offset the cost of the machine in as little as 4 miles (7 km) of tunnel drivage (McFeat-Smith, 1982). Hard-rock TBMs are commonly equipped with a partial or slotted shield, and when support is required, conventional rock support methods are used. Both soft-rock and hard-rock TBMs can be equipped with a full shield and segmental linings installed. This equipment enables hard-rock TBMs to cope with localized occurrences of soft ground.

TBM PERFORMANCE PREDICTION—SIMPLIFIED APPROACH. As noted in 22.1.3, simplified approaches to TBM performance prediction, such as those presented by Farmer and Glossop (1980) and Graham (1976), are preferred over methods that rely on detailed site data (e.g., as suggested by Lislerud, 1983) due to cost considerations. These simplified formulas are consistent with the conceptual level planning approach incorporated herein. However, the owner is well advised to maximize geotechnical data collection and interaction with the TBM manufacturer so that uncertainties in performance prediction are minimal.

Table 22.1.8 compares data available from one TBM manufacturer with cutting rates predicted using Eqs. 22.1.7b and 22.1.8b.

System Utilization—TBM utilization is defined as the ratio of the productive TBM operating time to the total time available for tunnel drivage. An estimate of the machine cutting rate, derated for downtime using the system utilization factor, can therefore be used to provide an estimate of the project schedule.

TBM system downtime has been analyzed extensively by Nelson et al. (1985); Tables 22.1.9 and 22.1.10 have been reproduced from this source to illustrate potential downtime sources,

Table 22.1.6. Raise-bore Project Cost Elements

Item	Estimated Quantity	Unit	Unit Price	Total Price	Approximate Percentage of Total
Mobilization	____	L.S.	____	____	2–10
Site preparation	____	L.S.	____	____	5–20
Underground setup	____	L.S.	____	____	2–8
Production					
• Pilot bit cost	____	ft	____	____	2–8
• Reamer cutter cost	____	ft	____	____	20–70
• Operating labor	____	ft	____	____	5–10
• Supplies/maint.	____	ft	____	____	5–10

Note: High cutter cost is a function of rock properties (e.g., strength, hardness, abrasivity, bedding, etc.). Risks can be offset by requiring manufacturers to bid cutter cost on a per foot (meter) basis.

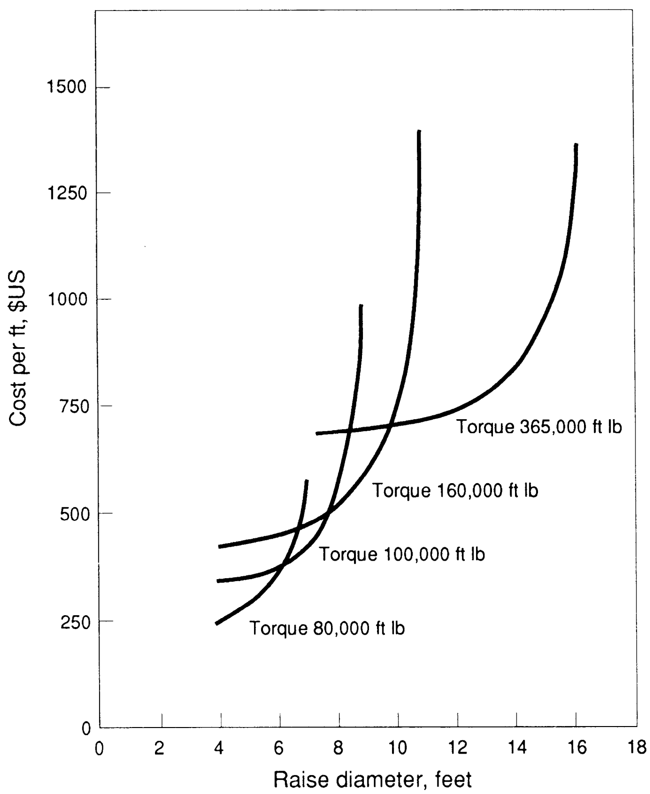


Fig. 22.1.18. Raise costs as a function of diameter and torque (modified from *Engineering and Mining Journal,* Anon., 1981). Conversion factors: 1 ft = 0.3048 m, 1000 ft-lb = 1.356 N-m.

average TBM utilization factors, and the relative impact of each major downtime component on utilization.

One other major cause of downtime, excluded from the utilization calculation in Table 22.1.10, involves remedial work required when mining through major faults and shear zones. Major collapses and large inrushes of mud and water, heavily squeezing ground, and gouge that clogs the mucking system are a few of the reasons for significant delays when mining through tectonically altered zones. Case studies involving TBM excavation under adverse ground conditions have been reported by McFeat-Smith (1987) and are reported in summary form in Table 22.1.11. The importance of a carefully executed geotechnical site investigation designed to evaluate potential impacts of these features and to provide data for remediation, and selection of an appropriately equipped TBM cannot be overstressed.

TBM PERFORMANCE PREDICTION.

Example 22.1.2. The following data apply to the construction of a 24-ft (7.32-m) diameter tunnel (Kerckhoff No. 2) in granitic rocks (s_{cf} = 20,250 psi or 140 MPa; s_{tf} = 1450 psi or 10 MPa), 25% quartz content.

TBM Data (Robbins 243-217)

Diameter	24 ft 1 in.	7.34 m
No. Cutters/Diameter	57 × 15.5 in.	57 × 394 mm
Thrust	2,280,000 lbf	10.14 MN
Torque	1,980,000 ft-lbf	2.68 MN-m
Cutterhead Horse-power	2200 hp	1640 kW
Cutterhead Speed	5.8 rpm	5.8 rpm

Determine the overall rate in advance.

Solution. Assuming operating thrust/cutter = 70% of machine capability,

$$F_c = \frac{2{,}280{,}000}{57} \times 0.7 = 28{,}000 \text{ lbf/cutter}$$
$$(124.5 \text{ kN/cutter})$$

1. Farmer and Glossup

$$P = \frac{F_c \times 0.0158}{1450} = 0.306 \text{ in./rev}$$
$$(7.77 \text{ mm/rev})$$
$$P_{HR} = \frac{0.306 \times 5.8 \times 60}{12} = 8.9 \text{ fph}$$
$$(2.71 \text{ m/h})$$

2. Graham method

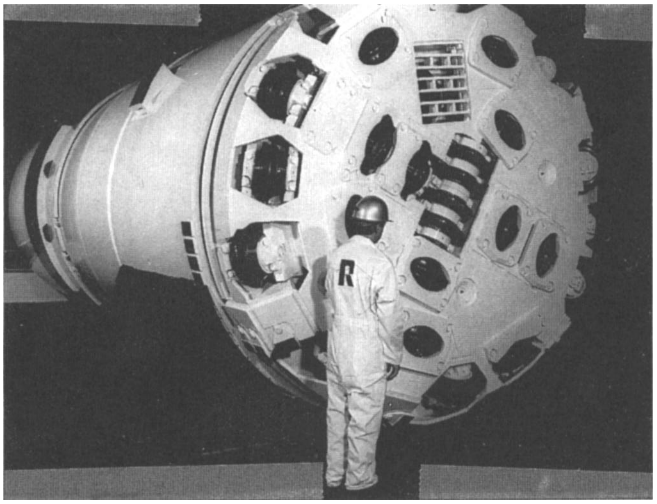
$$P = \frac{F_c \times 0.1}{20{,}250} = 0.138 \text{ in./rev}$$
$$(3.5 \text{ mm/rev})$$
$$P_{HR} = \frac{0.138 \times 5.8 \times 60}{12} = 4.0 \text{ fph}$$
$$(1.22 \text{ m/h})$$

Assume utilization = 40%.

P_{sys} = 1.6 to 3.6 fph (0.5 to 1.1 m/h),
or 38 to 86 ft/24hr-day (12 to 26.4 m/day)

Table 22.1.7. Comparison of Factors Influencing the Selection of Blind Drilling, Raise Drilling, and Conventional Shaft Construction Methods

Factor Influencing Selection	Blind Shaft Borer (BSB)	Blind Drilling	Raise Drilling	Vertical V-mole	Conventional Shaft Sinking
Design considerations					
Safety	Utilizes underground operators working under cover and behind shields. No operation of large moving equipment required.	Does not require miners to work underground.	Only requires underground labor set-up and mucking. No labor required in the shaft prior to shaft lining.	System utilizes underground operators working in a controlled environment. Considered safer than conventional shaft sinking as operators are located remote from the working face.	Requires equipment operation in confined environment. Considered to be the most dangerous of the five shaft construction methods.
Shaft size	Depth is limited by skip-hoist rope capacity.	Limited by required depth, available equipment, and cost. Drilled shaft diameters range from well size to 20+ ft.	Nominally limited by available machine torque. Short (300 ft) shafts have been raised at 20 ft diameter.	Limited by available equipment from 16 to 23 ft in diameter.	Required to be larger than 10–12 ft for most applications. Upper limit not controlled by method.
Shaft depth	Depth is limited by skip-hoist rope capacity.	Limited by required diameter, available equipment and cost. See Table 22.1.2 for case study data.	Raise depths up to 3200 ft have been reported.		
Shaft verticality	Verticality can be controlled within extremely tight tolerances. Equal to or better than conventional.	Deviation can be estimated from geological data. Difficult to maintain an absolutely vertical shaft in sub-vertical structure. Suggested tolerance for design purposes = 0.25–0.5°.	Shaft verticality is controlled by pilot borehole. Pilot hole accuracy controlled by directional survey and careful drilling practice. Design tolerance should be specified based on use requirements.	Shaft verticality is controlled by pilot hole. However, offset reaming can be controlled by the operator providing additional control when following a deviated pilot borehole.	Verticality can be controlled within extremely tight tolerances. The most accurate shaft construction method with regard to verticality.
Ground disturbance	Minimal mechanical disturbance of shaft wallrock.	Provides minimal mechanical disturbance of shaft wallrock.	Provides minimal mechanical disturbance of shaft wallrock.	Provides minimal mechanical disturbance of shaft wallrock.	Conventional (drill-and-blast) excavation can result in deep seated blast damage.
Timing/schedule	Setup time is 1 month or more. Advance rate should be between 20 and 40 ft/day. In general muck haulage and lining limit advance.	Typically much faster than conventional shaft construction methods. See Table 22.1.2 for case study excavation rates.	Reaming range from 15 fph (5 ft dia.), 3 fph (10 ft dia.), to 1 fph (15 ft dia.)	Reported rates between 30 fpd (16 ft dia.) and 60 fpd (23 ft dia.)	Generally restricted to one round per shift (e.g., 1 fph)
Operational considerations					
Groundwater	Same techniques as conventional sinking.	Method provides superior control of groundwater during excavation.	Groundwater controlled by pretreatment where necessary.	Large projected inflows require pretreatment (e.g., using grouting or freezing)	Large projected inflows require pretreatment (e.g., using grouting or freezing)
Support during excavation	Temporary or final support can be installed a short distance behind advancing face.	Support provided by hydraulic pressure and impermeable polymer skin permitting excavation in very poor ground conditions.	Not possible.	Temporary/final support can be installed a short distance behind advancing face.	Temporary/final support can be installed a short distance behind advancing face.
Final lining	Installed during excavation.	Steel/concrete composite lining typically used in weak ground. May incorporate bitumen layer for groundwater control and be designed for full hydrostatic, bituastatic or lithostatic loading conditions.	None usually required. Many rapid lining systems available—see text.	Final lining may be installed during excavation.	Final lining may be installed during excavation.
Miscellaneous			Requires existing underground access.	Requires existing underground access.	
Other considerations					
Shaft outfitting		If a final steel lining is installed, considerable time can be saved through surface installation of guides, brackets and pipes. All furnishings can be aligned prior to welding to the downhole liner assembly.	Pilot hole deviation usually prevents raises from being used for man or materials winding.	Outfitting in-line with excavation and final lining if required.	Outfitting in-line with excavation and final lining if required.
Costs					
—Initial		High	Medium	High	Low
—Operating		Medium	Medium	Medium	High



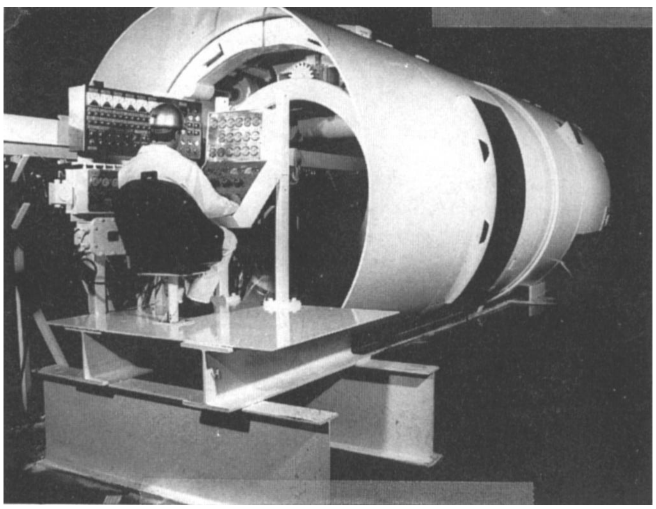


Fig. 22.1.19. Model IIII-234 Robbins hard-rock double shield (*courtesy:* Robbins Co., Seattle, WA).

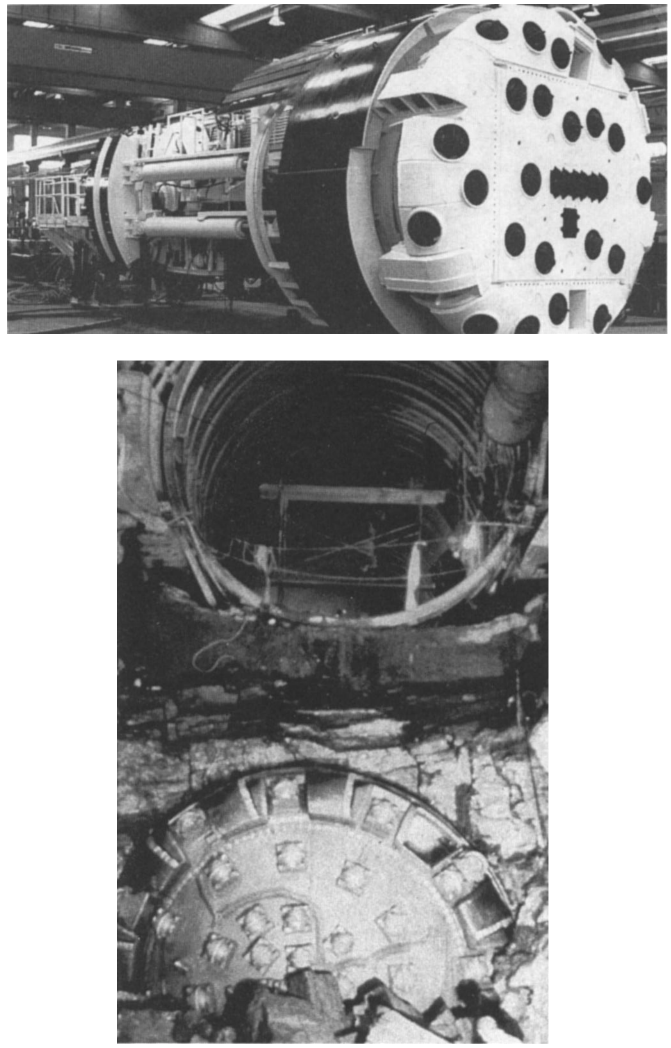


Fig. 22.1.20. Large-diameter TBM and breakthrough of a 20-ft (6-m) diameter TBM drivage in limestone/shale (*courtesy:* Robbins Co., Seattle, WA).

This compares to a forecast rate of 55 fpd (16.8 m/day) based on case study data analysis and an actual production rate of 60.5 fpd (18.4 m/day) (Woodward, 1983).

22.1.5.2 Mobile Miner

A prototype of the Robbins mobile miner (Fig. 22.1.22) was introduced in 1984 for development of a 3773-ft (1150-m) decline at Mt. Isa mine, Australia. Advances of up to 12 ft/shift (3.66 m/shift) were made while mining a 12-ft (3.66-m) high, 21-ft (6.4-m) wide section in high-strength quartzite (16,000 to 39,000 psi or 110 to 269 MPa) (Boyd, 1987). Upgrades in dust control and sealing systems improved the initial utilization rates from 17 to 34%, with the best single eight-hour shift resulting in 12 ft (3.66 m) of drivage for 5.3 hours cutting time. Redesign of the cutter wheel, to avoid high imbalanced loads and other upgrades resulting from the Mt. Isa experience are anticipated to provide 50% utilization. This compares favorably with the 36% utilization required for break-even with drill and blast (Robbins, 1986).

22.1.5.3 Roadheader Systems

Roadheaders have been in use in mining and tunneling for many years, and are known under a variety of names including boomheaders, boom-type tunneling machines and selective tunneling machines. Fig. 22.1.23 shows two of the many varieties of roadheader, and Fig. 22.1.24 illustrates commonly used terminology for the various machine components.

Roadheaders were originally developed as a means of advancing roadways in underground coal mines and early machines were limited to cutting relatively low-strength strata. Continuous development of these machines has greatly extended the range of applications, and they now are used in a wide variety of mining and civil tunneling work, including mine production (Sparks, 1980). Improvements in cutterhead design (Hurt et al., 1982) and the increasing use of water-jet-assisted cutting (Barkam and Buchanan, 1987; Timko et al., 1987; Hood, 1985) will result in a further extension of the range of roadheader applications in coming years.

Roadheaders offer a number of advantages over full-face tunneling machines, chiefly related to flexibility. Roadheaders can cut a variety of cross sections, limited only by the basic dimensions of the machine, and are able to cut tight curves or corners. They are thus usable, for example, in room and pillar operations. Roadheaders can selectively cut narrow bands or

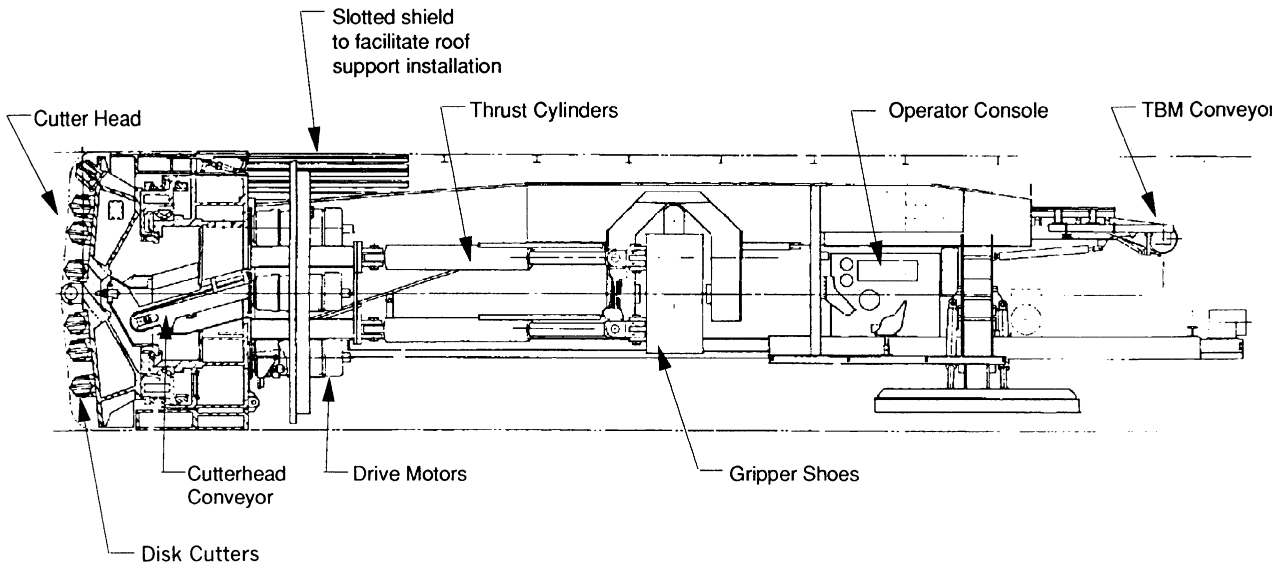


Fig. 22.1.21. Sectional view of tunnel boring machine.

beds and are thus suitable production mining tools if a careful sequence of mining and mucking is followed. Also they have lower initial costs than full-face machines.

Roadheaders also offer advantages over conventional drill and blast methods. One of the most important advantages of roadheaders is the avoidance of blast damage to the rock and the consequent savings in ground support costs. In addition, because mechanical excavation is a continuous process, shift time is more effectively utilized. Navin et al. (1985) estimated that support requirements in openings excavated by a roadheader were reduced by 40% compared to drill and blast excavations, and advance rates by roadheader were 1.5 times as rapid as drill and blast. McFeat-Smith (1982) reported average utilization of about 50% in roadheader excavation systems compared to 33% for drill and blast.

EQUIPMENT. The main components of roadheader excavation systems are discussed in some detail in this segment. However, because roadheader equipment is in a constant state of development, the reader is advised to consult manufacturers with regard to specific machines, detailed specifications, and applications.

Roadheader cutting assemblies consist of a cutting head on a movable, hydraulically powered boom, mounted on a rotatable turret attached to a track-driven chassis. In addition to the cutting head, the machine also incorporates either a gathering arm or chain conveyor mucking system to remove broken rock from the face. The machine may be controlled by an operator seated on the machine or located some distance away, perhaps inside a shield, or beneath a supported roof section. To improve machine stability during cutting, many machines are equipped with hydraulically powered stelling rams which are used to brace the machine off the excavation sidewall.

Pearse (1988) has studied the products of 13 roadheader manufacturers and tabulated basic specifications for about 60 machines, with total installed powers ranging from 70 to 600 kW, and weights ranging from 17 to 130 tons (15 to 120 t) (Table 22.1.12).

Two types of cutting heads are available from several manufactures and are interchangeable on specific machines. These heads are termed transverse and in-line heads (Pearse, 1988), or ripping and milling heads (Kogelmann and Schenck, 1982). In-line heads rotate co-axially with the boom, and arcing forces developed during transverse cutting may need to be resisted by stelling rams, particularly if the machine is of light weight. According to Pearse (1988), these heads are best suited to cutting rock with an unconfined compressive strength of 12,000 psi (80 MPa) or less. In-line heads require less thrust when sumping, and the head shape allows greater selectivity in cutting specific beds or bands, but stelling rams can be rendered ineffective if weak rock bands are present in the drift sidewalls. Transverse heads rotate at right angles to the boom axis, so arcing forces are resisted by the weight of the machine, and stelling is not usually required. An important advantage of this is that machines can be lighter than similarly powered machines with an in-line head.

Advances in cutter booms have resulted in machines equipped with telescopic booms, or booms with extended length for cutting high backs or crowns. Telescopic booms are useful for cutting on steep gradients or on weak floor materials where the thrust from the machine's travel system would not be adequate. In such cases, the machine can be stelled, and the thrust provided by the booms' telescoping hydraulics reacting against the machine. Kogelmann and Schenck (1982) have described soft-rock cutter booms (SRBs) and hard-rock cutter booms (HRBs), and report that the trend is toward the development of the latter. HRBs are reportedly more stable in cutting, producing less vibration and, consequently, less damage to bearings and picks. In low-strength formations, however, SRBs perform satisfactorily, and the increased cost of an HRB-equipped machine is not justified.

Better cutter head design and analysis methods (Hurt and MacAndrew, 1981; Hurt et al., 1982) can lead to improved pick utilization and pick life and reduced torque variation during cutting. These lead to reduced vibration and bearing damage. Cutterheads supplied with machines should be designed using sound engineering principles, and field modifications and repairs (e.g., replacing pick boxes) should be subject to the same controls and standards. Water-jet-assisted cutting heads are now available as standard or optional features on many roadheaders. Research in the application of low-, medium-, and high-pressure water jets

Table 22.1.8. Summary Case Study for Tunnel Boring Machines

Rock Type	Average Uniaxial Compressive Strength (psi)	Tensile Rock Strength (calc) (psi)	Machine Diameter (ft)	Cutterhead hp (hp)	Thrust (lb)	Torque (lb-ft)	Cutting Disk No.	Cutting Disk Dia. (in.)	Max. Thrust/ Cutter (lb/disk)	Estimated Penetration: Farmer & Glossop (in./rev)	Estimated Penetration: Graham (in./rev)	Cutter-head rpm (rpm)	Estd. Cutting Rate (Graham) (ft/hr)	Estd. Cutting Rate (f & g) (ft/hr)	Act. Average Rate (ft/hr)	Act. Best Rate (ft/hr)
Phyllite, gneiss, amphibolite	9000	643	10.50	1000	1,320,000	400,000	24	19	55000	0.95	0.43	12.13	25.95	57.37		30.2
Quartzite, gneiss, dolomite	24500	1750	11.58	1000	1,250,000	420,000	25	17	50000	0.32	0.14	11.00	7.86	17.37	12.1	
Claystone, mudstone, shale	11700	836	11.67	900	1,397,700	411,900	25	17	55908	0.74	0.33	10.91	18.26			
Limestone, mudstone, sandstone	15650	1118	17.38	1200	1,596,100	1,000,000	36	17	44336	0.44	0.20	7.33	7.27	16.07	12.2	
Basalts	17700	1264	11.00	900	1,174,000	381,050	27	17	43481	0.38	0.17	11.57	9.96	22.01	23	
Mica schist	20000	1429	18.00	1750	2,000,000	1,056,000	40	17	50000	0.39	0.18	7.07	6.19	13.69	8.9	
Mica schist	21300	1521	18.17	1200	1,755,000	2,160,000	39	17	45000	0.33	0.15	7.01	5.18	11.46		
	14200	1014	27.88	2420	2,915,000	2,188,706	61	17	47787	0.52	0.24	4.57	5.38	11.90		
Diorites and granites	20000	1429	11.00	800	1,062,000	321,184	26	17	40846	0.32	0.14	11.57	8.28	18.30	11.5	
Granitic gneiss	20000	1429	11.58	800	1,080,000	384,000	27	15.5	40000	0.31	0.14	11.00	7.70	17.02		
Granitic gneiss	22000	1571	10.33	600	867,000	260,145	26	14	33346	0.23	0.11	12.33	6.54	14.46		
Granitic gneiss	20500	1464	9.92	400	724,000	238,000	24	14	30167	0.23	0.10	12.84	6.61	14.62	7.57	
Granodiorite, aplite, tonalite	20250	1446	24.08	2200	2,280,000	1,980,000	57	15.5	40000	0.31	0.14	5.80	4.01	8.87	5.22	9
Mica schist, quartz	14200	1014	20.42	1400	1,760,000	1,075,000	44	15.5	40000	0.44	0.20	6.24	6.15	13.60	6.53	
Dolomite, siliceous limestone	19000	1357	20.42	1200	1,760,000	1,075,000	44	15.5	40000	0.33	0.15	6.24	4.60	10.16	8.2	9.1
Dolomite, siliceous limestone	19000	1357	20.42	1200	1,760,000	1,152,173	44	15.5	40000	0.33	0.15	6.24	4.60	10.16	7.7	
Sandstone, mudstone, coal	14000	1000	19.00	900	1,274,000	943,000	41	14	31073	0.34	0.16	6.70	5.21	11.52	10.8	
	14200	1014	15.58	1200	1,440,000	769,230	36	15.5	40000	0.44	0.20	8.17	8.06	17.82	9.84	
Phyllite, mica schist	21250	1518	14.75	1050	1,400,000	689,000	35	15.5	40000	0.29	0.13	8.63	5.69	12.58	7.61	
Phillite, mica schist	21250	1518	14.75	1050	1,400,000	689,000	35	15.5	40000	0.29	0.13	8.63	5.69	12.58	7.78	
Phyllite, mica schist	21250	1518	14.75	1050	1,400,000	689,000	35	15.5	40000	0.29	0.13	8.63	5.69	12.58	7.64	
Phyllite, mica schist	21250	1518	14.75	1050	1,400,000	689,000	35	15.5	40000	0.29	0.13	8.63	5.69	12.58	9	
Granite, granitic gneiss	42250	3018	25.25	2000	2,565,000	2,000,000	57	15.5	45000	0.16	0.07	5.04	1.88	4.16		
Diorite, greywacke, phyllite	16500	1179	14.17	900	1,216,000	631,000	34	15.5	35765	0.34	0.15	8.99	6.82	15.08	11	
Dolomitic limestone	22000	1571	14.17	900	1,216,000	631,000	31	15.5	39226	0.28	0.12	8.99	5.61	12.40		
Shale, sandstone, tillite	21800	1557	9.90	400	740,000	238,000	24	14	30833	0.22	0.10	12.86	6.37	14.08	7.87	
Schist, gneiss	20000	1429	9.90	400	740,000	238,000	24	14	30833	0.24	0.11	12.86	6.94	15.35	11.1	
Granite, gneiss	20100	1436	18.00	1200	1,950,000	1,135,000	42	15.5	46429	0.36	0.16	7.07	5.72	12.65	4.72	5.3
Dolomite, layered shale	24600	1757	18.50	1200	1,680,000	1,086,200	42	15.5	40000	0.25	0.11	6.88	3.92	8.66	9.04	18.9
Dolomite, layered shale	24600	1757	18.50	1200	1,680,000	1,086,200	38	15.5	44211	0.28	0.13	6.88	4.33	9.58		
Granite, schist	19250	1375	22.00	1200	1,960,000	1,176,000	46	15.5	42609	0.34	0.16	5.79	4.49	9.92	4.8	
Dolomite, phyllite	17000	1214	13.75	600	1,010,000	537,000	34	14	29706	0.27	0.12	9.26	5.67	12.53	5.87	
Limestone, shale	21500	1536	9.50	400	720,000	253,000	24	14	30000	0.22	0.10	13.40	6.55	14.48	7.47	
Shale, sandstone	12500	893	21.33	960	1,400,000	1,262,000	46	14	30435	0.38	0.17	5.97	5.09	11.25	6.94	
Shale, sandstone	12500	893	20.00	720	1,400,000	950,000	43	14	32558	0.40	0.18	6.37	5.81	12.84	7.55	
Dolomitic limestone, shale	22000	1571	32.33	2400	2,570,000	3,304,000	64	15.5	40156	0.28	0.13	4.43	2.83	6.26	5.9	10
Dolomitic limestone, shale	22000	1571	32.33	2400	2,570,000	3,304,000	64	15.5	40156	0.28	0.13	3.94	2.52	5.57	5.7	12.5

Table 22.1.8. Summary Case Study for Tunnel Boring Machines (cont.)

Rock Type	Average Uniaxial Compressive Strength (psi)	Tensile Rock Strength (calc) (psi)	Machine Diameter (ft)	Cutterhead hp (hp)	Thrust (lb)	Torque (lb-ft)	Cutting Disk No.	Dia. (in.)	Max. Thrust/ Cutter (lb/disk)	Estimated Penetration: Farmer & Glossop (in./rev)	Estimated Penetration: Graham (in./rev)	Cutter-head rpm (rpm)	Estd. Cutting Rate (Graham) (ft/hr)	Estd. Cutting Rate (f & g) (ft/hr)	Act. Average Rate (ft/hr)	Act. Best Rate (ft/hr)
Shale, limestone, siltstone	17500	1250	10.67	500	852,000	350,000	27	14	31556	0.28	0.13	11.93	7.53	16.66	6.75	
Shale, limestone, siltstone	25000	1786	10.67	500	852,000	350,000	27	14	31556	0.20	0.09	11.93	5.27	11.66	6.75	
Dolomitic limestone, shale	18500	1321	35.33	2400	2,760,000	3,510,000	69	15.5	40000	0.33	0.15	3.60	2.73	6.03	5.2	10
Dolomitic limestone, shale	18500	1321	35.33	2400	2,760,000	3,5p,000	69	15.5	40000	0.33	0.15	3.60	2.73	6.03	5.1	7
Dolomitic limestone, shale	21000	1500	35.25	2400	2,760,000	3,510,000	69	15.5	40000	0.29	0.13	3.61	2.41	5.33	6	
Limestone, sandstone, shale	18000	1286	14.96	800	2,702,900	766,700	34	14	79497	0.68						
	32500	2321	10.67	500	700,000	310,000	28	14	25000	0.12	0.05	11.93	3.21	7.11		
Limestone, sandstone	29050	2075	19.00	900	1,850,000	1,050,000	40	15.5	46250	0.25	0.11	6.70	3.74	8.26	7	10.4

Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 psi = 6.895 kPa, 1 lb = 0.4536 kg, 1 hp = 0.7457 kW.

Table 22.1.9. Downtime Categories for TBM Operations

Major Category	Individual Downtime Sources
TBM maintenance and repair	Cutterhead check and routine maintenance
	Lube oil system
	Hydraulic system
	Cutterhead motors
	Electrical system
	TMB conveyor
Backup system	Tunnel power supply
	Utility lines (air, water, fanline)
	Laser guidance and surveying
	Trailing floor conveyor
	Tripper or car pass
	Train delay—muck bound at heading
	Shaft or portal equipment
	Installing rail and switches
	Derailments
Ground conditions	Water inflow rock support installation— bolts and straps, steel sets
	Gripper/sidewall support
	Equipment clearance
	Scaling loose material
	Muck jams in conveyor and hoppers
Cutter changes	
Other	Scheduled downtime
	Probe hole drilling
	Contractual downtime
	Moving TBM to new location
	Shift changes

Source: Nelson et al. (1985).

to assist cutting has indicated substantial improvements in pick life and dust suppression, and has reduced potential for frictional ignition of methane. In general, machines equipped with water jets can cut higher-strength formations than equivalent machines without water jets.

Roadheader face mucking systems generally consist of an apron with either a scraper chain or gathering arm system (Fig. 22.1.25). Scraper chains generally extend around the perimeter of the machine, while gathering arms load muck onto a short conveyor that passes through the body of the machine. In either case, the muck is generally discharged onto a bridge conveyor. From this point, the muck may be removed by a wide variety of methods including shuttle trains, conveyors, LHDs, trucks, and so on. This allows roadheaders to be used in conjunction with an existing mine haulage system, provided that adequate capacity exists. Without adequate capacity, the face will become muck bound and excavation will be delayed. In low-strength formations in which high cutting rates are possible, the mucking system may become the limiting factor in controlling advance rates.

Roadheader cutting booms are generally mounted on a custom chassis with track propulsion. Track dimensions control ground pressures and should be carefully assessed in conditions in which the invert rocks are weak or prone to slurrying. Increasingly, booms are being mounted on a variety of other machines, such as hydraulic breakers, trucks, traveling gantries, and inside shields.

Conventional manual/hydraulic control systems are being increasingly superceded by electronic/hydraulic control systems linked to microprocessor-based guidance and profile control systems. Guidance systems consist of fixed laser sources mounted some distance behind the roadheader and photoelectric targets mounted on the roadheader. Deviations of the roadheader from its desired position and orientation are detected by the targets, and corrections are automatically made. In addition to guidance control, automatic profile control is also available. These systems consist of a microprocessor programmed with the required excavation profile, and transducers that continually monitor the position of the cutting boom. The boom hydraulics are controlled electronically to ensure that the correct profile is cut. These developments enable very accurate alignment and profile control, which eliminates overbreak and ensures that the cutting sequence is optimum; work stoppages to allow time for survey work are also reduced. Electronic systems are also being used on roadheaders to monitor the condition of mechanical and hydraulic components, enabling preventative maintenance to be scheduled, reducing unplanned downtime.

OPERATIONS. The excavation cycle commences with sumping or forming a cavity in the rock face to the operating depth of the cutterhead. During this operation, the cutter boom is kept stationary while the cutting head rotates and the whole machine is gradually moved forward on the tracks. When the sump has been formed, it is enlarged by tracking the cutting head across the face to the outer perimeter of the excavation. At the end of this arcing cut, the head is moved up or down, and another arcing cut is made across the face in the opposite direction to the first cut. Proceeding in this way, the opening is gradually enlarged to the full dimensions of the face. When the full cross

Table 22.1.10. TBM Excavation System Utilization and Downtime Percentages[a]

Project	Tunnel Section	Utilization[b]	Downtime: TBM Maintenance and Repair	Downtime: Backup System	Downtime: Ground Conditions	Downtime: Cutter Changes	Downtime: Other[c]	Downtime: Excluded Special Causes	Downtime Excluding Special Causes
1C0011	Outbound	39.2	18.3	19.2	16.7	2.8	3.3	Water inflow, stops at shafts and conrail	60.8
	Inbound	45.1	8.8	27.4	10.5	2.0	6.2	Water inflow, stop at shaft	54.9
1C0031	Outbound	35.4	18.8	18.7	19.3	2.2	5.6	Conrail	64.6
	Inbound	35.1	26.0	17.2	14.7	2.7	4.3	Stop at shaft, relocate crane mucking	64.9
Culver Goodman	Densmore and Goodman Legs	41.1	12.0	14.5	13.2	17.1	2.1	Probe hole	58.9
TARP	East Heading	44.4	23.3	21.3	1.3	6.3	3.4	Water inflow, mining past shafts, relocate TBM	55.6
	Average for All Projects	40.0	17.9	19.8	12.6	5.5	4.2		60.0

[a] Time percentages calculated excluding time before trailing floor assembly completed and excluding shift time required for special causes.
[b] TBM operating time, including time required for thrust cylinder reset.
[c] Other includes downtime without explanation, shift changes, probe hole drilling, etc.
Source: Nelson et al. (1985).

Table 22.1.11. Case Histories of Tunnel Boring in Adverse Ground

Machine Type Tunnel Dia.	Geology	Geological Feature	Av. Rate Advance ft/week	Delay to Progress	Tunneling Problems in Adverse Zones	Design	Comments
TBM (medium weight) 11.5 ft	Moderately strong sandstones (Class 4)	75 ft length of intensely jointed mudstones and sandstone	465	76%	Generally minor—Arch supports used	Reliable mucking systems and easy access for support installation	TBM well designed for conditions
TBM (heavy weight) 11.5 ft	Dolerite sill intrusion into sedimentary sequence	1200 ft of competent, (Classes 2–3) 50,000 psi dolerite sill	295	60%	Slow cutting. Average progress reduced to 115 ft/week. Cutter costs very high.	Triple button disks used although single disks may have been better	Operating considered to be a success for such hard rock.
TBM (heavy weight) 11.5 ft	Pure mudstone at roof level (Class 3) overlying limestone (Class 1)	60 ft of soil infill zone in roof (Class 5) caused by dissolution of limestone	295	342 hours	Delays mainly for support and mucking due to collapse of roof.	Long roof shield prevented installation of heavy temporary support, conveyors chocked	Inadequate design features enhanced delays
TBM (medium weight) 11.5 ft	Sandstone, mudstones	660 ft throw fault giving 50 ft clay gouge zone with boulders (Class 5) and 33 ft shattered zone (Class 4)	390	67 hours	Generally minor—Arch supports used—timber packing required for gripper pads	Suitable access for installation of arches close behind face. Single gripper pads most appropriate	TBM very well designed for this severe condition

Conversion factor: 1 ft = 0.3048 m.

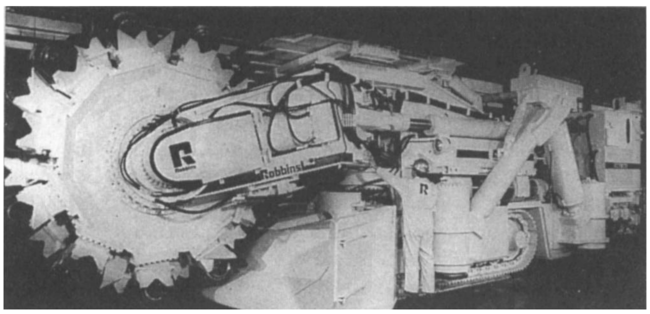
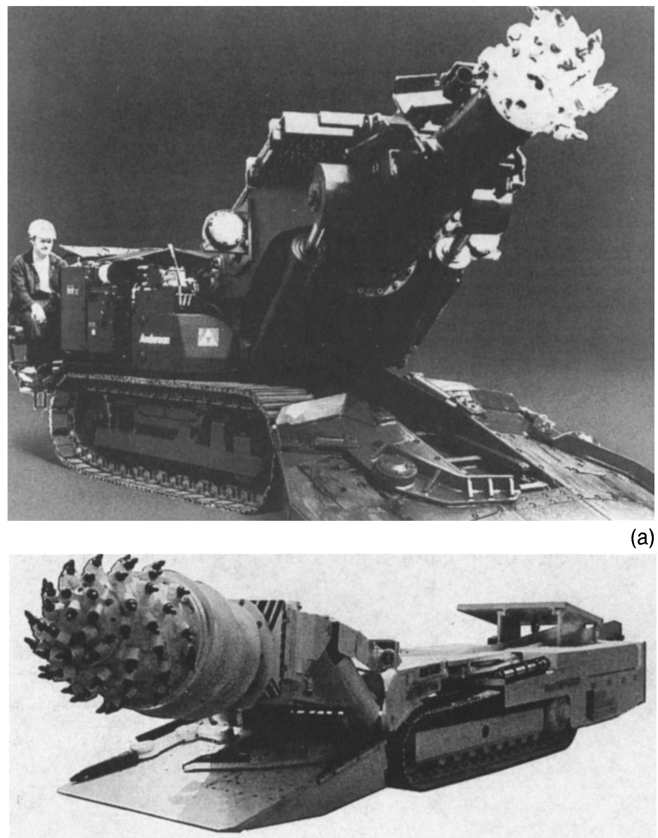


Fig. 22.1.22. Robbins mobile miner (*courtesy:* Robbins Co., Seattle, WA).

section has been excavated and the muck removed, a new sump is formed, and the entire process is repeated. The pattern of sumping and arcing cuts, relative to the direction of cutterhead rotation and geologic structure, influences the efficiency of roadheader excavation.

SUPPORT SERVICES. In the vicinity of working roadheaders, high concentrations of airborne dust, generated during both cutting and transport of the muck, and high ambient temperatures commonly occur. In addition to exceeding statutory respirable dust limits, excessive dust may completely obscure the face, resulting in inefficient excavation and increased overbreak. High temperatures and humidity result in labor inefficiency and overheating of electrical motors. The use of waterjet-assisted cutting leads to a reduction in dust levels, but does not entirely eliminate the problem, and may actually increase humidity at the face. Meyeroltmanns (1982) has described practical methods of using ventilation to control airborne dust in the vicinity of roadheader faces, methods that also assist in controlling heat and humidity.

GROUND SUPPORT. One of the primary advantages of selecting a roadheader excavation system over drill-and-blast methods is the elimination of blast damage and the consequent savings in rock support costs. McFeat-Smith (1982) estimates that in excavations requiring temporary support, the cost saving may be on the order of 10 to 15%, and in suitable ground, the need



(a)

(b)

Fig. 22.1.23. Roadheader-type tunneling machines. (a) Model RH 25 (*courtesy:* Anderson, Strathclyde) (b) Model ABM 330-1 (*courtesy:* Alpine Equipment Corp.).

for support may be eliminated entirely. All types of rock support can be adopted for use in conjunction with roadheaders. However, because it can be relatively difficult to reverse a roadheader away from the face, the machine must be covered prior to shotcrete application. When ground conditions require, roadheaders

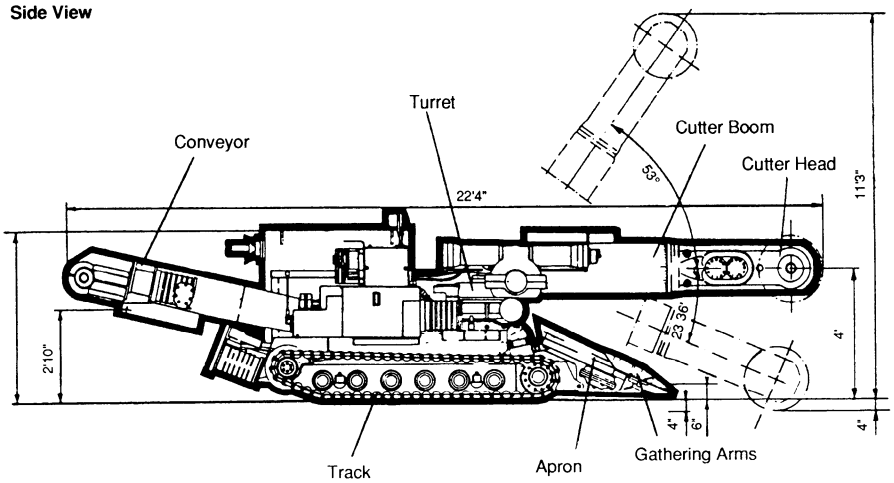


Fig. 22.1.24. Typical roadheader showing main components. Conversion factor: 1 ft = 0.3048 m.

can be mounted inside shields (Fig. 22.1.26) or advance within self-advancing powered supports (Fig. 22.1.27). The roadheader is supported on a slide mechanism and can move independently of the shield. Excavation, mucking, and erection of a segmental lining can proceed concurrently. The use of short shields allows tight turning circles to be maintained, although the system is less maneuverable than a non-shielded roadheader. Several cutter booms can be mounted in shields when large-diameter openings are required.

ROADHEADER SYSTEM PERFORMANCE PREDICTION. Methods of predicting instantaneous and operational cutting rates for roadheaders were presented in 22.1.3. Overall system performance, assessed in terms of advance rate, is a function of OCR, face area, and utilization:

$$\text{Advance rate} = \text{OCR/Face Area} \times \text{Utilization} \qquad (22.1.10)$$

Utilization is defined as the time available for advancing the face when all planned and unplanned machine stoppages have been accounted for. Machine stoppages generally fall into one of the following categories:

1. Planned maintenance of roadheader and backup equipment.
2. Unplanned maintenance of roadheader and backup equipment.
3. Mucking delays.
4. Ground control.
   a. Rock support installation.
   b. Control of water inflows.
5. Survey work.
6. Meal time/shift change time.

Guidance for typical roadheader utilization has been suggested by Kogelmann (1988) based on the type of ground support to be installed:

Support Type	% of Cutting Time per Available Face Time
None	60 – 80
Rock bolts	40 – 50
Shotcrete	40 – 50
Shotcrete and rock bolts	30 – 35
Steel sets	30– 35
Steel sets with full lagging	20 – 25

Examples of the application of cited performance prediction methods are presented below to further illustrate the process.

Example 22.1.3. This example is based on results reported by Sandbak (1985) for mine drift development in quartz monzonite and dacite porphyry. The roadheader is a Dosco SL-l-20 with an 82-kW cutter motor.

Table 22.1.13 summarizes key geotechnical properties for sections of drift over which cutting rates were recorded. Two prediction methods were used, and the results of these can be compared with observed OCR data given in the table.

Solution.

1. Prediction Based on Method of Bilgin et al. (1988)

The first step in the application of this method (see 22.1.3) is calculation of the rock mass cuttability index, RMCI (see Fig. 22.1.7). This is given as:

$$\text{RMCI} = \text{UCS (kN/cm}^2) \times \frac{\text{RQD}^{2/3}}{100} \qquad (22.1.11)$$

which is then used in the prediction equation to determine OCR:

$$\text{OCR} = 28.06 \times 0.997^{\text{RMCI}} \qquad (22.1.12)$$

Because the prediction equation given by Bilgin et al., is for a somewhat more powerful machine than the Dosco SL-120, a slight linear correction is applied to Eq. 22.1.12 as follows:

$$\text{OCR} = 28.06 \times 0.997^{\text{RCMI}} \times \text{HP}/95 \qquad (22.1.13)$$

where HP is the head power of the Dosco SL-120.

These results are in remarkably good agreement with observed OCR data.

2. Prediction Based on Method of Fowell and McFeat-Smith (1976, 1977)

The first step in this prediction is the calculation of specific energy SE required for cutting, given by:

$$\begin{aligned}\text{SE} = &-4.38 + 0.14(\text{CI})^2 + 3.3(\text{UCS})^{1/3} \\ &+ 0.000018(\text{SN})^3 + 0.0057(\text{CC})^3 \qquad (22.1.14)\end{aligned}$$

where CI is cone indenter hardness, UCS is unconfined compressive strength, SH is shore hardness, and CC is cementation coefficient.

Table 22.1.12. Typical Roadheader Specifications

Type	Wt	Head drive	Total power	Head type	Cut ht. max.	Cut width (one position)	Ground pressure	Travel speed	Loading system
	t	kW	kW	I/T	m	m	bar	m/min	(footnote)
Alpine Equipment Corp., PO Box 132, State College, PA 16804, US.									
ABM-40	15	40	90	I/T	3.8	4.6	0.6	0.16	D/GA/SW
ABM-110	24	76/110	190	I/T	4.0	5.1	1.0	0.16	D/GA/SW
ABM-132	34	132	200	I/T	3.9	5.2	1.0	0.16	D/GA/SW
ABM-160	45	160	370	I/T	4.1	5.6	1.0	0.16	D/GA/SW
ABM-200	47	200	410	I/T	4.1	5.6	1.1	0.16	D/GA/SW
ABM-300	70	300	500	I/T	6.5	9.1	1.3	0.16	D/GA/SW
ABM-400	90	400	600	I/T	8.0	11.0	1.3	0.16	D/GA/Sw
Anderson Strathclyde plc, 47 Broad Street, Glasgow G40 7QW, Scotland.									
RH25	25.4	82	157	I	4.25	6.0	1.2	6.8/13.6	GA
RH25L	26	82	164	I	3.8	4.5	1.2	6.8/13.6	GA
RH22	35	112	187	I	5/5.3	5.4/6	1.7	2.6/2.8	GA
RH1/4	66	112	224	I	6.0	6.4	1.45	0.10	GA
RH90	90		300	I	5.0	6.0	new machine		GA
Atlas Copco-Eickhoff GmbH, Hunscheidtstr 154, D-4369 Bochum 1, FRG.									
ET-110	25–30	110–132	185	I/T	4.0	5.3	1.4	0.5	D/GA
ET-200	40–45	160–200	340	I/T	4.7	6.85	1.4	0.15	D/GA
ET-300	80–90	200–300	460	I/T	5.3	7.9	1.7	0–5.6	D/GA/FC
ET-400	100–110	300–400	560	I/T	6.3	2.0	0–8.4	0–5.6	D/GA/FC
Special components for mounting (see text)									
ETS-110	45*	110	*	I/T	11.0*	8.3	*	*	none
ETS-200	75*	160	*	I/T	11.0*	8.3	*	*	none
ETB-110	22	110	†	I/T	to cut TBM tunnel ledges				none
ETS-200	35	160	†	I/T	to cut TBM tunnel ledges				none
* Depending on type of excavator. † Powered from other source.									
Dosco Overseas Eng. Ltd., Ollerton Rd., Tuxford, Notts. NG22 OPQ, England.									
MK IIA	27.7	48.5	123.5	I/T	4.1	3.0–5.8	1.5	4.7	EC
MD 1000	28.5	50	135	I/T	4.2	2.7–5.7	1.2–1.6	7.2	SW/GA
MD 1100	31.5	82	157	I/T	4.2	2.7–5.7	1.4–1.7	7.2	SW/GA
SL 120	33	82	165	I/T	4.1	2.0–4.3	1.5	13.8	GA
MK IIB	44	82	194	I/T	6.0	3.0–7.4	1.2	8.4	SW/GA
LH 1300	44	142	254	I	4.1	3.2–5.6	1.5	10.1	SW/GA
LH1300(H)	45.7	142	285	I	4.1	3.5–6.0	1.5	9.2	SW/GA
MK III	83	142	254	I	6.0	4.0–7.1	1.4	5.4	SW/GA
TB2000	76	119×2	424	I	3.3	4.0–7.7	1.9	9.6	SW/GA
TB3000	123	250×2	686	I	6.0	4.5–8.9	2.2	12.8	SW/GA
TM1800	n/a	48.5	104	I	5.2 dia.		n/a	n/a	various
SB 400	n/a	142	198	I	4.7 dia.		n/a	n/a	various
SB 600	n/a	142	198	I	5.8 dia.		n/a	n/a	various
Eimco (GB) Ltd., Team Valley, Gateshead, NE11 OSB, UK.									
TM Series: Approx. weight 100 t, 2 × 150 kW motors									GA
Herrenknecht GmbH, D-7635 Schwanan-Allmannsweier, FRG.									
SM2	n/a	80	95	I	1.5–2.2.dia.		n/a	n/a	SC/BH
SM1	n/a	95	132	I	2.0–3.0 dia.		n/a	n/a	SC/BH
Mannesmann-Demag, Buscherhofstr 10, D-4000 Dusseldorf 13, FRG									
H55	68	160	180 + 160	I	10	11–12.5	1.0	2.2	nil/FC
*Hydraulic excavator.									
Mitsui Miike Co. Ltd., 1-1, 2-chome, Nihonbashi Muromachi, Chuo-ku, Tokyo, Japan.									
S50, S90, S100, S125									

Table 22.1.12. Typical Roadheader Specifications (cont.)

Type	Wt	Head drive	Total power	Head type	Cut ht. max.	Cut width (one position)	Ground pressure	Travel speed	Loading system
	t	kW	kW	I/T	m	m	bar	m/min	(footnote)
Paurat GmbH, PF 1220, D-4223 Voerde 2 (Friedrichsfeld), FRG. In U.K.: Dowty Mining Equipment Ltd., Tewkesbury, Glos. GL20 8HR.									
E169	44	140	225	I	2.3	3.4	1.45	9.3/18.6	FC
E195	43	170	263	I	4.2	5.2	1.5	9.3/18.6	GA
E134	70	230	353	I	3.05	4.1–6.6	1.7	5.4	FC
E200	110	350	512	I/T	6.0	7.6	1.8	4.5	FC
E242B	120	300	480	I	7.5	8.9	1.5/1.8	17.6	GA
Salzgitter Maschinenbau GmbH, PF 511640, D-3320 Salzgitter 51, FRG.									
STM 100	28	100	200	T	4.0	5.2	1.5	10	GA
STM 160	45	160	282	T	4.2–5.0	6.2–6.9	1.3–1.5	10	GA
STM 200	75	200	330	T	5.3	7.5	1.6	8	GA
STM 300	120	315	509	T	6.1	7.5	1.8	8	GA
TYAZHMASH, 26, B. Serpukhovskaya Ul., 113093 Moscow, USSR.									
4PP-5	75	200	—	I	14–35m² area		—	—	GA
Voest-Alpine AG, PF 2, A-4010 Linz, Austria.									
F-6A	12	30–41	60–82	T	3.4–4.0	4.5	1.4	5.0	GA
AM50	24	110	170	T	2.0–4.8	4.8	1.3	6.0	GA
AM65	32–36	132–175	214–305	T	4.3–4.9	6.9	1.2–1.35	5/13/20	GA
AM75	45–52	160–200	290–330	T	4.7–5.1	6.8–7.0	1.2–3.8	4–15	GA
AM100	84–96	250–400	450–700	T	5.5–6.4	7.3–7.7	1.8–2.1	3–21	GA
Westfalia Lunen, D-4670 Lunen, FRG.									
Fuchs WF-40	9	37	70	T	3.8	4.1	1.0	10	FC
also WF-50 with 50 kW cutter motor									
Dachs 53	13	79	101	T	4.3	5.2	0–95	26.7	FC
Luchs B-110	25	110	200	T	4.1	6.0	1.0	10–31.5	FC
also N-110 and H-110									
WAV 130	32	130	250	T	4.2	5.3	1.5	10–20	GA
WAV 170	45	200	300	T	5.4	6.3	1.7	7.5–26.7	GA
WAV 178	73	200	360	T	7.1	8.3	1.6	5.0	GA
WAV 178/300	73	300	437	T	7.7	8.9	1.6	5.0	GA
WAV 300	90	300	470	T	5.4	7.9	1.6	5.0	FC

BH = Backhoe, D = Disc, FC = Flight chain, GA = Gathering arm, SC = Scroll, SW = Star wheel.

Source: Pearse, 1988.
Conversion factors: 1 ft = 0.3048 m, 1 hp = 0.7457 kW, 1 ton = 0.9072 t.

Cone Indenter hardness and Shore hardness have been shown to be linear functions of unconfined compressive strength (Atkinson et al., 1986), so that Eq. 22.1.14 can be rewritten in terms of USC and CC:

$$\mathrm{SE} = -4.38 + 0.14\,(0.0377\ \mathrm{UCS} + 0.254)^2 + 3.30\ \mathrm{UCS}^{1/3} + 0.000018\,(0.441\ \mathrm{UCS} - 8.73)^3 + 0.0057\mathrm{CC}^3 \qquad (22.1.15)$$

Cementation coefficient is based on petrographic descriptions of the rock (McFeat-Smith, 1977).

When SE has been calculated, instantaneous cutting rate ICR can then be calculated from Eq. 22.1.5:

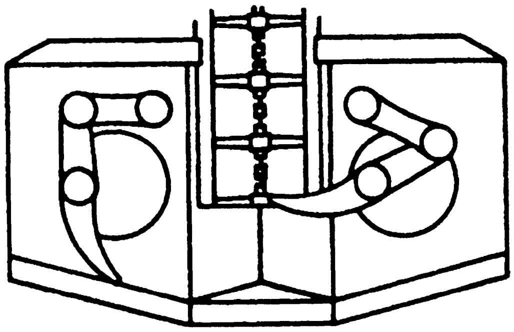
$$\mathrm{ICR} = \mathrm{HP/SE}$$

Because the prediction equations use SE based on actual cutting time, a *cutting time factor* (CTF) correction must be made to Eq. 22.1.5. CTFs for bulk excavations with an experienced operator are estimated to have values in the range 0.65 to 0.85, while for final trimming, it may drop to 0.3. Overall values for bulk excavation and final profiling of a face may be in the range of 0.45 to 0.65. In this example, 0.45 was assumed and applied as follows:

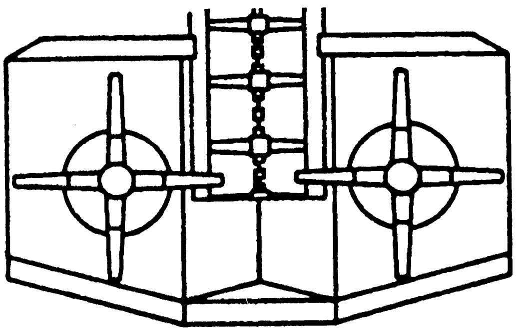
$$\mathrm{OCR} = \mathrm{HP/SE} \times \mathrm{CTF} \text{ or} \qquad (22.1.16)$$
$$= \mathrm{ICR} \times \mathrm{CTF}$$

In this example the predicted values are low compared to the observed cutting rates, and this may be attributable to rock mass factors.

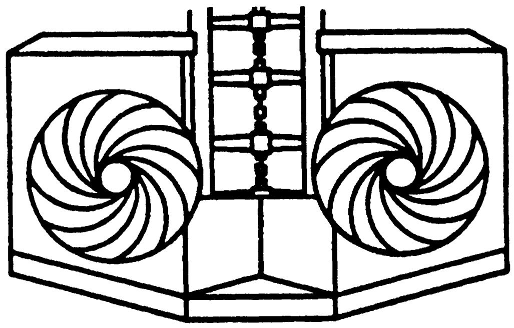
Example 22.1.4. This example is based on results reported by Bilgin et al. (1988) for drivage of sewer tunnels in Turkey using a Herrenknecktt SM1, and coal mine development drives using a Dosco MKIIA. Results are presented in Table 22.1.14.



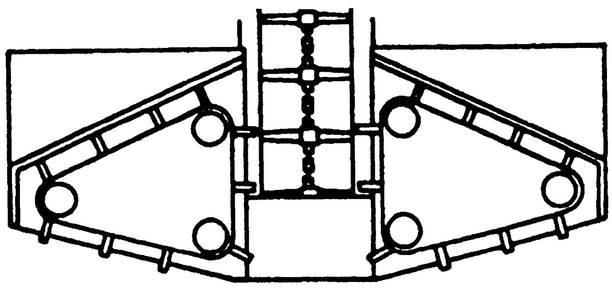
Gathering - Arm Loader
For blocky, interlocked, wet and sticky materials. Effective loading on steep slopes.



Star - Wheel Loader
For dry, non-interlocked and non-sticky materials.
High loading rates at continuous flow.
Low maintenance.



Spinner - Disk Loader
Same as Star-Wheel Loader



Scraper - Conveyer Loader
For non-blocky, non-abrasive materials

Fig. 22.1.25. Roadheader loading (gathering) systems (after Kogelmann, 1988).

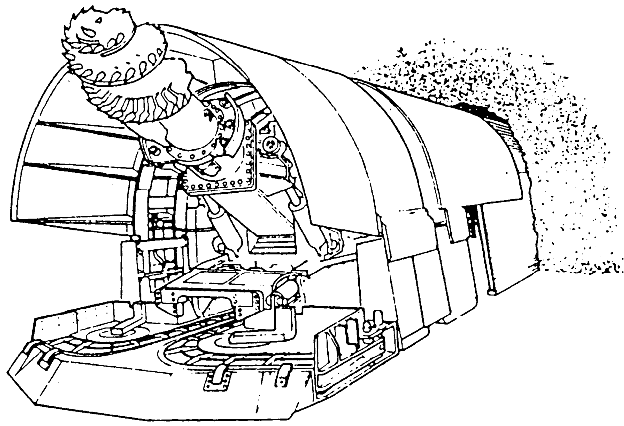


Fig. 22.1.26. Roadheader in horseshoe-shaped shield support system (after Kogelmann, 1988).

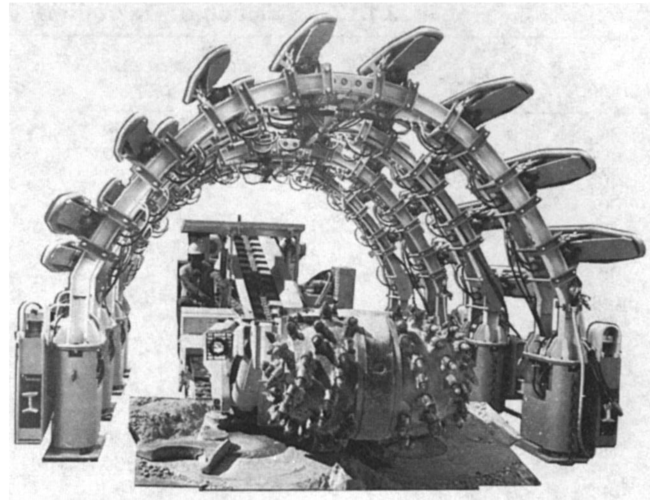


Fig. 22.1.27. Model ABM-T roadheader with waling, hydraulic roof support system (*courtesy:* Alpine Equipment Corp.).

Results for both of the methods used are in good agreement with the observed results. Again, a CTF of 0.45 was assumed.

Example 22.1.5. This example is based on cutting trials in British Coal Measures reported by Fowell and McFeat-Smith (1976) using a Dosco MKIIA. Although individual results (Table 22.1.15) show appreciable scatter, mean predicted values are in reasonably good agreement with the observed noncoal values. Again, a CTF of 0.45 was assumed. The correlation could be improved if suitable rock mass property corrections were made.

ROADHEADER PROJECT COSTS. Advance rates can be predicted based on the foregoing estimates of operational penetration rate and equipment utilization. Roadheader project costs are calculated based on these advance rates with loading of the following costs and resources:

Component	Costs Available in	Costs Calculated at
Capital/lease cost (1)	($/wk)	($/ft, $/m^3)
Labor (operating and support)	($/hr)	($/ft)
Cutters and maintenance (1)	($/yd^3, $/m^3)	($/ft)
Mucking - equipment (1)	($/mo)	($/ft)
- operators	($/hr)	($/ft)
Ventilation - equipment (1)		($/ft)
- installation and power		($/ft)
Support - shotcrete	($/yd^3, $/m^3)	($/ft)
- concrete	($/yd^3, $/m^3)	($/ft)
- rock bolts (1)	($ each)	($/ft)
- steel sets (1)	($ each)	($/ft)

Notes: (1) Available from manufacturer.

22.1.5.4 Selection of Tunnel Construction Method

Selection of the appropriate tunnel/drift construction technique will involve an in-depth analysis of site geomechanical and geohydrologic conditions, design criteria (e.g., diameter, length, shape, use, and life, etc.), availability and location of equipment and qualified labor, project schedule requirements and cost. The approach in this *Handbook* is considered to be applicable at a conceptual level of project planning. More detailed investigations and analysis should be undertaken using expert services available from geotechnical and design engineers, construction contractors, and equipment manufacturers prior to final method selection.

SITE-SPECIFIC DATA REQUIREMENTS. Field data required for design and construction bidding are essentially the same for all tunnel construction methods. The site investigation program should include geologic mapping and fully logged corehole along the tunnel alignment; in certain circumstances, horizontal boreholes are the most cost effective method of evaluating in situ conditions. Site investigation coreholes should be geologically and geotechnically logged (e.g., core recovery, RQD, discontinuity description, lithology, rock description, etc.) and core samples selected for testing (e.g., uniaxial and tensile strength, slake durability, swelling, hardness, etc.); samples will also be required by TBM manufacturers for proprietary drillability testing. Hydrogeologic data can be collected in-line with core drilling or after drilling has been completed.

Data analysis will provide estimates of groundwater inflows and parameter values that input to the design of grouting programs; rock quality, strength testing, and in situ stress will be used to evaluate stability, design rock support or lining systems, and for blast design; laboratory strength and index test data will be used to estimate drill penetration rates, bit wear, cuttability, and suitability for mechanical excavation.

METHOD SELECTION GUIDELINES. The choice of a tunneling method should be made using the approach outlined in 22.1.5 and the selection factors presented in Table 22.1.16. Mechanical mining systems are now available that can rapidly mine a broad range of rock types at gradients up to 25%, while negotiating relatively tight and variably curved alignments. Further innovations and developments in cutting technology, especially with regard to water and particle-assisted rock cutting; mucking systems (e.g., with regard to continuous mechanical conveying and hydraulic and pneumatic transport); and machine features (e.g., access to cutterhead for maintenance, guidance systems, gripper and steering systems, etc.) are underway. However, motivation for these developments must come from a receptive and innovative mining industry.

Table 22.1.13. Prediction of Operational Cutting Rates Using Data From Sandbak (1985)

Lithology	RQD	UCS (MPa)	UCS (psi)	RMCI	Observed OCR (m^3/h)	Observed OCR (ft^3/hr)	Predicted OCRs (3) (1)	Predicted OCRs (3) (2)
qm	72	200	29,000	353	5.9	208	142	53
qm	70	170	24,650	294	11.9	420	84	34
qm	70	170	25,650	294	5.1	180	196	78
qm	35	120	17,400	131	13.6	480	120	44
qm	58	116	16,820	177	17.8	629	80	35
dp	38	172	24,940	198	17.8	629	75	22
dp	18	100	14,500	70	21.7	766	90	33
qm	22	95	13,780	76	11.9	420	162	62
dp	39	90	13,000	106	13.2	466	133	58
qm	39	92	13,300	108	17.0	600	103	45
qm	55	85	12,300	125	20.8	735	80	39
dp	18	90	13,000	63	14.9	526	134	52
qm	56	115	16,700	172	13.2	466	110	47
qm	52	130	18,850	185	17.4	614	80	32
qm	52	130	18,850	185	11.9	420	117	46
qm	63	157	22,750	253	13.2	466	86	33
qm	63	157	22,750	253	9.3	328	122	47
qm	52	130	18,850	185	12.3	434	113	45
qm	50	145	21,000	201	12.3	434	108	40
qm	50	145	21,000	201	12.7	449	105	39
qm	7	70	10,150	26	12.7	449	176	72
qm	47	115	16,700	153	14.4	508	106	43
qm	47	115	16,700	153	12.7	449	120	49
qm	44	120	17,400	153	14.0	494	109	43
			Mean OCR		13.7	484	109	43

Notes: (1) Using Bilgin et al., 1988.
(2) Using Fowell and McFeat-Smith, 1976.
(3) Predicted OCR, as % of observed
qm = quartz monzonite
dp = dacite porphyry
CC = 8

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Table 22.1.14. Prediction of Operational Cutting Rates Using Data From Bilgin et al. (1988). Example 22.1.4.

Location	RQD	UCS (MPa)	UCS (psi)	CC	RMCI	Observed OCR (m^3/h)	Observed OCR (ft^3/hr)	Predicted OCRs (3) (1)	Predicted OCRs (3) (2)
1	70	104	15,100	8	180	6.9	244	290	110
2	22	150	21,700	9	120	5.9	208	99	90
3	50	83	12,000	8	115	9.4	332	160	100
4	50	70	10,150	7	96	6.7	237	89	170
5	50	73	10,600	9	101	6.3	222	89	150
6	50	76	11,000	9	105	4.1	145	63	230
7	80	156	22,600	9	295	0.5	18	92	980
8	50	157	22,750	9	217	1.5	53	104	330
9	60	82	11,900	7	128	3.3	117	69	300
10	60	94	13,650	8	147	6.5	230	174	130
11	60	126	18,300	9	197	1.6	57	87	390
12	60	128	18,600	10	200	0.8	29	45	730
13	25	159	23,000	9	139	7.7	272	183	60
14	25	146	21,200	9	128	5.2	184	107	100
15	65	106	15,400	7	175	2.6	92	105	310
16	30	103	15,000	7	101	9.0	318	128	95
17	30	89	12,900	9	88	6.4	226	76	130
18	30	154	22,350	9	152	3.1	104	89	160
19	30	133	19,300	9	131	7.8	275	166	80
20	21	55	8,000	8	43	15.0	530	94	80
21	75	30	4,350	9	55	10.2	360	77	75
22	75	23	3,350	8	42	17.4	614	109	60
23	75	43	6,300	8	79	10.4	369	109	70
24	75	32	4,650	8	58	11.4	403	90	70
25	75	16	2,300	7	28	26.5	936	139	50
					Mean OCR	7.5	265	113	105

Notes:
(1) Using Bilgin et al., 1988.
(2) Using Fowell and McFeat-Smith, 1976
(3) Predicted OCR, as % of observed

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Table 22.1.15. Prediction of Operational Cutting Rates Using Data from Fowell and McFeat-Smith (1976). Example 22.1.5.

Lithology	UCS		Observed OCR		Predicted OCRs (1)		
	(MPa)	(psi)	(m^3/h)	(ft^3/hr)	(m^3/h)	(ft^3/hr)	(2)
Coal	24	3,500	10.3	364	9.4	332	90
Coal	24	3,500	9.4	332	9.4	332	100
Coal	24	3,500	8.6	304	9.4	332	110
Coal	24	3,500	14.0	494	9.4	332	70
Coal	24	3,500	9.8	347	9.4	332	100
Coal	24	3,500	57.2	2020	9.4	332	16
Coal	24	3,500	17.2	607	9.4	332	55
Coal	24	3,500	25.7	900	9.4	332	37
Coal	24	3,500	32.4	1140	9.4	332	30
Coal	24	3,500	64.8	2,300	9.4	332	15
Coal	24	3,500	25.7	910	9.4	332	37
L. Mudstone	34	4,900	10.0	353	8.0	293	80
L. Mudstone	34	4,900	7.2	254	8.0	293	111
L. Mudstone	34	4,900	7.8	275	8.0	293	103
L. Mudstone	34	4,900	11.6	410	8.0	293	70
M. Mudstone	25	3,600	6.8	240	9.2	325	135
M. Mudstone	25	3,600	1.2	42	9.2	325	720
M. Mudstone	25	3,600	6.8	240	9.2	325	135
M. Mudstone	25	3,600	5.9	208	9.2	325	156
M. Mudstone	25	3,600	6.2	220	9.2	325	148
M. Mudstone	25	3,600	7.8	275	9.2	325	118
M. Mudstone	25	3,600	6.9	244	9.2	325	133
M. Mudstone	25	3,600	14.3	505	9.2	325	64
M. Mudstone	25	3,600	18.4	650	9.2	325	50
M. Mudstone	25	3,600	26.7	943	9.2	325	34
M. Mudstone	25	3,600	18.6	657	9.2	325	49
M. Mudstone	25	3,600	29.9	1060	9.2	325	31
U. Mudstone	40	5,800	4.9	173	7.4	261	151
U. Mudstone	40	5,800	1.6	57	7.4	261	463
U. Mudstone	40	5,800	7.4	261	7.4	261	100
U. Mudstone	40	5,800	4.6	162	7.4	261	161
U. Mudstone	40	5,800	3.3	117	7.4	261	224
U. Mudstone	40	5,800	6.9	244	7.4	261	107
U. Mudstone	40	5,800	10.6	374	7.4	261	70
U. Mudstone	40	5,800	8.6	304	7.4	261	86
U. Mudstone	40	5,800	23.5	830	7.4	261	31
U. Mudstone	40	5,800	2.1	74	7.4	261	352
U. Mudstone	40	5,800	1.2	42	7.4	261	617
Mean OCR for Coal			25.0	880	9.4	332	40
Mean OCR for Lower Mudstone			9.2	324	8.0	283	90
Mean OCR for Middle Mudstone			12.5	441	9.2	283	64
Mean OCR for Upper Mudstone			6.8	240	7.4	261	110

Notes: (1) Using Fowell and McFeat-Smith, 1976
(2) Predicted OCR, as % of observed
CC = 8

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Table 22.1.16. Comparison of Factors Influencing the Selection of Horizontal Tunneling Method

Factor Influencing Selection	Tunnel Boring Machine	Roadheader	Mobile Miner	Conventional Mining
DESIGN CONSIDERATIONS				
• Safety	Miners work under supported ground. Special safety precautions may be required where access to the face of the machine is required for cutter changes.	Shields and hydraulic supports available for poor ground conditions. Equal safety potential to TBM, much safer than conventional mining.	Main application in hard, high strength rock requiring little-to-no support. Safety rated equivalent to TBM with improvement, to dust suppression system.	Inherently the least safe of all development mining alternatives.
• Tunnel line and shape	Method restricted to circular shape. Tunnels up to 35-ft diameter have been excavated in hard rock.	Multiple heading levels permit a wide range of opening sizes and shapes to be mined. Single heading range is from 6-20+ ft.	Flat-back, rectangular, minimum height controlled by size of cutter wheel (> 10 ft).	Can produce full range of sizes/shapes typically required.
• Tunnel length	Unsuited, due to capital equipment and mobilization costs to short drivages. TBM applications seldom economical for single drivages less than 7000 ft.	Non-circular tunnels up to about 2,000 ft (with sections larger than 200 sq. ft.) in softer sedimentary formation can be driven more economically using a roadheader.	No restriction.	No major physical restriction.
• Alignment	Clockwise rotation causes TBM to drift to left. Deviation is easily controlled, and alignment ensured through use of laser guidance system. Laser deviation may present problem in extremely long drivages.	Accurate alignment and profile control available using laser guidance systems and microprocessor cutting boom control.	Minimum operating radius is about 40 ft. Alignment controlled by continuous survey or laser.	Easily controlled, no major restrictions.
• Minimum radius	About 600 ft. Curves as tight as 300 ft are possible with modified equipment.	25 ft.	65 ft (horizontal) 260 ft (vertical)	No major restrictions.
• Normal maximum gradient	About 25% limitation associated with mucking and gripping systems. Larger gradients are possible but require equipment modifications (e.g., hydraulic and lubrication systems).	About 25% controlled by equipment stability and mucking.	About 25% controlled by equipment stability and mucking.	About 50% controlled by equipment ingress/egress and mucking.
• Ground disturbance	In good rock, TBM produces a smooth, hydraulically efficient bore. Overbreak and rockwall damage are practically eliminated.	Minimal rockwall damage due to mining. Special precautions and careful operation required to avoid overcutting profile.	Minimal ground disturbance associated with mechanical cutting (cutterwheel fitted with disc-cutters).	Care required to minimize rockwall damage and overbreak. Recent advances in high-speed digital seismography allow accurate analysis of blast related rockwall damage and subsequent re-design to minimize effects.
• Timing/schedule	Advance rates of 165 ft/day and 700 ft/week have been reported for mine-sized development drivages with moderate utilization (<35%).	Cutting rates in low strength rocks up to twice as fast as drill-and-blast (see 22.1.3.5). Typically restricted to cutting rock <15,000 psi.	Initial poor utilization (17%) improved to 31%, 50% utilization rates are achievable with advance rates (10′ × 21′ heading) on the order of 12 ft/shift.	Typically the slowest of the four methods compared here. Advance rates typically equal to the smallest tunnel dimension per 8-hr shift for mine development headings.

Table 22.1.16. Comparison of Factors Influencing the Selection of Horizontal Tunneling Method (cont.)

Factor Influencing Selection	Tunnel Boring Machine	Roadheader	Mobile Miner	Conventional Mining
OPERATIONAL CONSIDERATIONS				
• Groundwater	Large predicted inflows may require ground pretreatment (e.g., grouting). Groundwater control during excavation can result in significant reduction in utilization.	No major restrictions to groundwater handling as part of excavation cycle. Performance may be improved by pretreatment.	No restrictions to groundwater handling as part of excavation cycle.	No restrictions to groundwater handling as part of excavation cycle. However, inflows typically slow progress and result in difficult mucking conditions. Wet conditions may require more expensive explosive types.
• Ground support	Smooth, undamaged profile less likely to require ground support. Most conventional ground support systems can be installed in line with excavation.	Mechanical cutting provides smooth profile minimizing the need for ground support. Required ground support can be installed at the face.	Used in hard rock applications where ground support requirements are minimal. Ground support can be installed at face.	Uneven profile and overbreak typically requires more rock support than used in mechanical excavation. Project costs may increase by up to 15% unless careful blasting practices employed.
• Rock homogeneity	Cutting rate and cutter consumption are primarily a function of rock type and properties. Best performance is in good quality, homogeneous rock. Poor quality and mixed face conditions can require careful planning and execution, and proper machine selection.	Care must be taken to ensure that variations in rock properties along proposed alignment are within the capabilities of the equipment.	No major restrictions. Current use in high strength quartzite.	Conventional drill-and-blast can be used in practically all rock conditions. Preferred over TBM in mixed face conditions, where large variations in rock mass strength will be encountered, or when tunneling through rock masses containing faults and shear zones.
• Rock temperature	High rock temperatures coupled with machine heat may require special cooling systems (e.g., air and water spray).	No restrictions.	No major restrictions.	No major restrictions.
• Muck removal	Muck removal typically by rail, which usually limits advance rate. Continuous conveyors and pneumatic/hydraulic systems may ensure a continuous mucking cycle, at required mucking rates, in the future.	Muck discharge via conveyor at rear of machine. Rail and trackless haulage available. Moderately more versatile than TBM.	Same range of options as for roadheader.	Muck loaded from tunnel floor to truck or rail based haulage system, as part of excavation cycle. Scoop trams may be utilized for short haulage distances.
• Auxiliary services	Auxiliary services housed on trailing gear (power, compressed air, water, ventilation, etc.). Power consumption high relative to other methods.	Similar to those required by TBM. Although installed power typically considerably less than for TBM, efficiency of converting power to rock breakage is less favorable.	Similar to those required for TBM. Improvements in dust suppression required based on initial field trials.	Power requirements considerably less than for mechanical systems. Drilling requires compressed air or water cooling.

Table 22.1.16. Comparison of Factors Influencing the Selection of Horizontal Tunneling Method (cont.)

Factor Influencing Selection	Tunnel Boring Machine	Roadheader	Mobile Miner	Conventional Mining
• Auxiliary equipment	Muck cars and train, roof support (if required).	Auxiliary equipment typically suited to other mine duties.	Auxiliary equipment typically suited to other mine duties.	Auxiliary equipment includes drill jumbo, loader, scoop-tram, truck or shuttle car or rail based haulage. Auxiliary equipment typically suited to other mine duties.
OTHER CONSIDERATIONS				
• Costs	High	Low-Medium	Medium	Low
—Capital	Medium	Medium	Medium	High
—Operating	Low	Low	Low	Potentially High
—Support				
• Utilization	30–50%	30–80%	17–50%	N/A
• General		Capable of working in multi-face operations requiring cyclic excavation/support activities.	Improved utilization coupled with a relatively small turning radius suggest that boring machines of this type will gain favor for development in hard rock mining.	Versatile equipment can be easily allocated to other mine construction/ore mining operations.

Conversion factors: 1 ft = 0.3048 m, 1 psi = 6.895 kPa.

Chapter 22.2
AUTOMATION AND ROBOTICS

ROBERT H. KING

Mining automation and robotics are controversial topics. Some miners see automation as a threat to their jobs or as a change that will remove the art and romance from mining. Many believe robotics will never progress to the point of cost-effective application in the rough mining environment. Nevertheless, some predict it is the inevitable future of mining because of the possible health, safety, and productivity gains. Many of these different opinions stem from different perceptions of a robot. When thinking of a mining robot, people visualize a wide range of forms, from a human mimic like Asimov's R. Daneel Olivaw to the Star Wars' R2D2 (a metallic but somewhat humanoid form). However, machines that look like present mining equipment but are computer controlled represent the most likely short-term breakthroughs.

Other industries successfully have adopted automation to improve productivity, but early experiments in mining did not show similar results. The reason is that mining tasks are not a series of cyclic motions readily accomplished by factory-floor mechanisms. Mining takes place in the geologic environment where conditions are highly variable and unpredictable. As a result, mining systems must have substantial cognitive abilities to recognize and deal with these unpredictable variations. They should be "intelligent mining systems" rather than automated systems (King, 1988; also see Chapter 8.3).

The path to intelligent mining systems follows the progression of mechanization, remote control, teleoperation, and robotics. The mining industry has adopted *mechanized* equipment to a large extent. *Remote-controlled* equipment, where an operator is stationed some distance from the machine but within line-of-sight, is found on some continuous mining machines in underground coal mines and some load-haul-dumps (LHDs) in underground metal mines. *Teleoperated* equipment, where operators can be stationed further away, beyond line-of-sight, is currently in the experimental stage. Research projects currently focus on robotics, the last stage of the progression. *Robotics* has been described as the intelligent connection between perception (the ability to understand the world around the robot) and action. It may provide the greatest benefits in productivity and safety since it incorporates artificial-intelligence-based algorithms that show promise for adapting to the dynamic mining environment.

Mechanization in mining has reached the stage where, in an attempt to improve productivity, machines have become increasingly expensive and complex and pose excessive demands on already highly stressed operators. Often the potential capacity of a machine outstrips the manipulative skills of the person operating it. Also the efficient use of large complex machines calls for levels of precision that many times are beyond the capability of even highly trained miners. In addition, these expensive machines should not sit idle while miners do other tasks.

These problems occur in surface and underground mines, but underground mining is receiving more attention since it usually is less productive and more hazardous. In addition, maintaining an artificial environment in an underground mine that is conducive to optimal work performance is expensive. By its nature, underground mining can be hot or humid; in addition, the equipment produces dust, noise, fumes, and other hazards. Explosive or toxic gases and the constant danger of rock falls add to the increasing list of health and safety concerns that have initiated volumes of federal and state regulations for maintaining safe work places for humans in underground mines. These regulations cause tremendous capital and operating expenditures that prevent many mineral deposits from being mined profitably. For example, in both underground coal and metal mines, miners excavate many more entries or drifts than necessary to produce and remove the ore or coal in order to provide a safe environment for humans. These difficulties will increase as mines deplete near-surface reserves that have the best working conditions.

To solve these problems, mining engineers and researchers are applying some robotics technologies without waiting for researchers to produce completely autonomous (self-controlling) equipment. For example, they are providing aids for machine operators, information for managers, and mechanical failure prediction. Mines can gain great advantage by developing equipment and associated mining systems that ease the burden of the operators by removing them, as far as possible, from hazardous and stressful environments and providing the opportunity for greater productivity.

22.2.1 STATE OF THE ART IN OUTDOOR AUTONOMOUS VEHICLES

Most robotics texts primarily cover manipulator robots intended for manufacturing and have limited use for mining application. Fu, Gonzalez, and Lee (1987) is an exception because it is a recent text that covers control, sensing, and intelligence in a fashion that can be applied to mobile robots as well as factory-floor manipulators.

Another excellent reference is the survey of collision avoidance and ranging sensors by Everett (1987). It is especially useful, since it describes the sensors necessary for perception of complex dynamic environments.

Because there are few general reference texts on intelligent mobile robot applications in complex natural environments, individual reports of specific projects describe the current state of the art. Most of this work is for the military. Even though the military and mines share problems in dealing with complex natural environments, mining requires additional development. The following reviews the work done for military applications.

Hennessy and King (1989) describe development of the Autonomous Land Vehicle (ALV), sponsored by the Defense Advanced Research Projects Agency (DARPA) (part of the Strategic Computing Program) and contracted through the Army Engineer Topographic Laboratories. Martin Marietta Information and Communications Systems in Denver built the ALV and conducted experiments with it to advance autonomous navigation technologies. The ALV drove autonomously on a black-top surfaced road and cross country. Fig. 22.2.1 shows that the vehicle is an eight-wheeled, skid-steered vehicle designed for on and off road. It has all-wheel hydrostatic drive powered by a Caterpillar 3208 turbo-charged diesel engine. A foam-filled fiberglass enclosure covers the computers and 100,000-Btu/hr (29.3 kW) air conditioning system. The ALV weighs 18,000 lb (8165

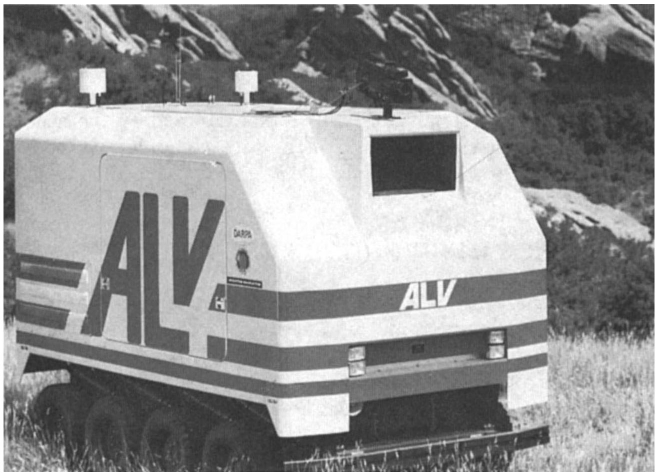

Fig. 22.2.1. Autonomous land vehicle, ALV (after Hennessy and King, 1989).

kg) and requires a 30-kW air-cooled generator for electrical power. Concerns about control system stability limit the vehicle's on-road top speed to 15 mph (25 km/h) and its weight and high center of gravity keep top speed to 6 mph (10 km/h) off road. Computers control the vehicle during autonomous experiments, and operators ferry it to and from the test track under radio remote control. It has a stereo vision teleoperation capability (including a work station in a remote laboratory).

The ALV perceives the environment and locates itself with three sensors: color video, a scanning laser ranger, and a gyrocompass. Data from the color, CCD (charge-coupled device), 512 × 480 pixel (picture element), 57.6° horizontal × 72.5° vertical field-of-view camera produce images for road-edge definition. The Environmental Research Institute of Michigan 256 × 64 pixel scanning laser ranger with 64-ft (20-m) ambiguity interval and 80° horizontal × 30° vertical field of view locates obstacles within the video field of view and characterizes off-road features. The Bendix land navigation system determines vehicle pointing, pitch, roll, and position with a north-seeking directional gyrocompass and two odometers. Researchers have investigated several additional sensors such as forward-looking infrared radar, low-light video, passive multi-spectral, millimeter wave radar, doppler ground speed radar, accelerometers, and ultrasonic rangers. The processing and communications architecture is shown in Fig. 22.2.2.

McTamaney (1987) described the FMC mobile robotic test bed developed for DARPA's advanced ground vehicle technology (AGVT) program. The FMC test bed transmitted data over a five-channel line-of-sight microwave using automatic tracking antennas and a 4.5-MHz band width carrier between the vehicle and a mobile command center to allow off-board processing over a 200-ft to 2.5-mi (60-m to 4-km) range. The command center receives data from robotic sensors, processes the information, and transmits control commands in real time to the vehicle (a modified M113 armored personnel carrier). On-board Intel microprocessors in a multi-processor configuration communicate with the control center, control the mobility actuators and monitor mobility sensors, and generate navigation reports that describe local attitude, position, and speed. Vehicle sensors include a 9600-Hz sonic transmitter and 16 microphones that produce a 16 × 24 range image every 200 ms for obstacle detection at 50 to 110 ft (15 to 33 m). Because of problems with engine and track noise interference and a blind spot, FMC also developed a 30-kHz sensor with 2- to 70-ft (0.6- to 21.3-m) range and 180° field of view.

The software perceives, plans, and controls navigation. The planning system uses a terrain map data base and a planning knowledge base to build a sequence of tasks and routes that will reach the goal. The perception system processes sensor data taken along the route to guide the vehicle along the planned routes by building a local map and fusing it with the planner's world map. The result is added to the vehicle mobility knowledge in the navigation system to create mobility commands.

Harmon (1987) describes the ground surveillance robot (GSR), a modified M114 armored personnel carrier. The program objective is to develop a second-generation autonomous vehicle test bed that can traverse unknown natural terrain between two given geographic locations. The GSR perceives the environment with 33-ft (10-m) acoustic rangers, a high-resolution gray-scale camera, a low-resolution color camera, and a laser range finder. Dead reckoning sensors (track speed, gyrocompass, and a doppler) define relative vehicle position and attitude. Since none of these provide accurate information in all circumstances, their data is fused to obtain the best relative position estimate. A satellite navigation system provides absolute position within 330 ft (100 m).

A world model is constructed with objects represented by triangles using a class tree with inheritance. The world map is used for long-range path and route planning. Route planning software relies on a minimum cost algorithm that considers energy requirements, estimated unknown hazards, estimated velocity, and distance to the goal. Intermediate goals are developed (trading off energy consumption, opportunity to gain valuable new information, and progress to the goal) in unknown terrain if the final goal is not within the sensor's field of view.

Carnegie-Mellon University's Navigation Laboratory (NAVLAB), based on earlier work with the Terregator, has a

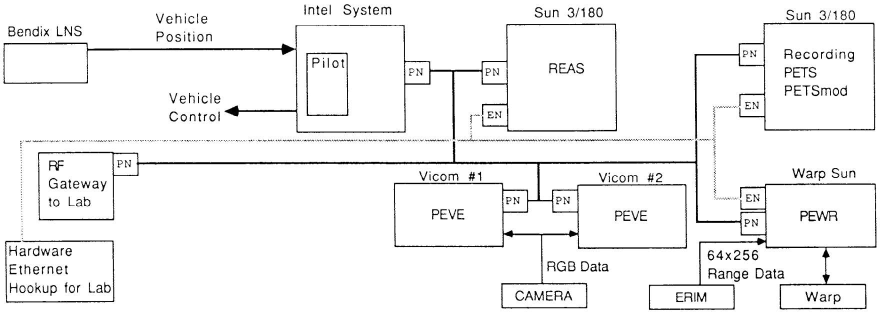

Fig. 22.2.2. Autonomous land vehicle processing and communications architecture (after Hennessy and King, 1989).

sensor suite similar to that of the ALV, but it does not have cross-country mobility capabilities (Goto and Stenz, 1987). It has successfully navigated sidewalks and a curved road using the CODGER (communications database with geometric reasoning) software system.

The robotic command center (RCC) is being designed to house workstations and communications to control multiple unmanned military machines (McTamaney et al., 1988). The current system assigns two vehicles to each of two drivers in the command center, which also houses a commander. The commander will plan wide-area operations and monitor them while the other workstations drive the vehicles with computer-aided remote driving or teleoperation. The system architecture is based on FMC's AGVT and extended to control multiple vehicles. Microwave and RF communications will link the vehicles to the RCC.

22.2.2 AUTONOMOUS VEHICLE ACCOMPLISHMENTS VS. MINING NEEDS

These autonomous vehicle projects demonstrated and developed considerable technology that can be borrowed and adapted to mining:

1. Closed-loop control systems utilizing sensor data to detect errors from planned or specified actions.
2. Pattern recognition algorithms.
3. Computing systems that permit real-time multi-tasking operations.
4. World modeling algorithms.
5. Path planning algorithms.
6. Work stations and intelligent user interfaces for supervising multiple semi-autonomous machines.
7. Ultrasonic ranging.
8. Laser ranging.
9. Land navigation systems.
10. Teleoperation.
11. Sensor fusion algorithms.

Mining engineers and researchers must adapt these advances to the mining environment and mining needs and develop additional technology for cost-effective application for the following reasons:

1. Some mining equipment (like mobile haulage vehicles) can navigate from a map. They do not need to explore.
2. Mining engineers can modify the layout to reduce the navigation and obstacle detection problems.
3. The mining environment is harsher in some respects than any that the autonomous vehicle research programs have encountered.
4. Mining equipment must operate faster and more precisely than most of the present autonomous vehicles.
5. Mining equipment must operate reliably over long periods of time and not just show success with a few demonstrations.
6. Mining machines must have better on-board machine health monitoring and diagnostics.
7. Excavation and drilling equipment require geo-sensing.
8. Some of the autonomous vehicle program technologies are not cost-effective for mining. For example, the software development and compute power for robotic image processing is not cost-effective in the dusty mine environment.

The comparison of autonomous vehicle development and mining needs shows that mining engineers and researchers must (1) develop computer representations of mining-specific knowledge in areas like layout and planning, machine capabilities, mine geometry, machine interactions, and machine conditions; (2) develop cost-effective sensors that will perceive and withstand the special mining environment; (3) analyze (e.g., using pattern recognition algorithms) and reason about mining specific sensor data to intelligently control machines; and (4) discover completely new mining methods or new approaches to traditional methods through removal of constraints imposed by the necessity of human operators. Some of this work is underway and is described under the next topic.

22.2.3 MINING ROBOTICS PROJECTS

Evans and Mayercheck (1988) and Frantz and King (1977) thoroughly reviewed mine automation, and Chapter 12.6 on mine monitoring and communications describes typical automated control strategies for stationary systems like conveyors. Therefore, this segment provides a reference and design source focus on research efforts that anticipate future intelligent mining systems. Cost-effective application requires real-time sensing and reasoning at the same level as a highly experienced human operator. This is a very difficult and expensive research and development task, considering the highly variable and dangerous mining environment. Therefore, mining researchers have not begun to develop equipment for all applications. They are focusing their efforts on the most promising. To achieve short-term results at the least cost, these projects borrow technology from the autonomous vehicle programs described in 22.2.1, allowing them to focus their limited resources on the specific mining problems listed in 22.2.2.

22.2.3.1 Autonomous Continuous Mining

Schnakenberg (1990) described the US Bureau of Mines efforts to develop enabling technology for computer-assisted control of mining equipment to improve worker safety and productivity. The work covers basic machine electrical, mechanical, and hydraulic components; sensors, algorithms, and languages for intelligent closed-loop control; obstacle detection and guidance sensors and algorithms; geosensing sensors and algorithms; machine health sensors and knowledge-based systems; and path planning algorithms. It is applicable to a wide range of mining equipment, but Bureau personnel are carrying out and demonstrating results on a modified underground coal continuous-mining machine fitted with electrically actuated controls in the Pittsburgh Research Center Mining Equipment Test Facility (METF).

Schiffbauer (1990) developed the Bureau of Mines Network (BOMNET) based on an Intel iSBC 286/12 central processing unit connected for multiprocessor networking through a Bitbus to several peripherals, sensor conditioner modules, communications devices, and a Solidrive magnetic bubble memory cartridge on the miner. The system hosts the iRMX86 operating system and PLM 86 software. Fig. 22.2.3 shows the Bitbus configuration that connects a single board computer for on-board attitude and location sensors, a Sun 3/60 workstation for the laser-based angular positioning system, a Sun 4/110 supervisory control workstation, and a Symbolics 3670 unit for machine health status.

Sammarco (1988a) developed closed-loop control for all machine functions except tramming using output from rotary variable differential transformers and hydraulic-pressure, temperature, and flow transducers (Fig. 22.2.4). He achieved closed-loop control for shearing, stab jack, and conveyor swing, based on the position of each component when operating in free space and cutting coalcrete. He controlled tram speed, direction, and pivot in an open loop (time clock limited) mode with the control safety relay, pump, gathering head/conveyor, and cutter motors

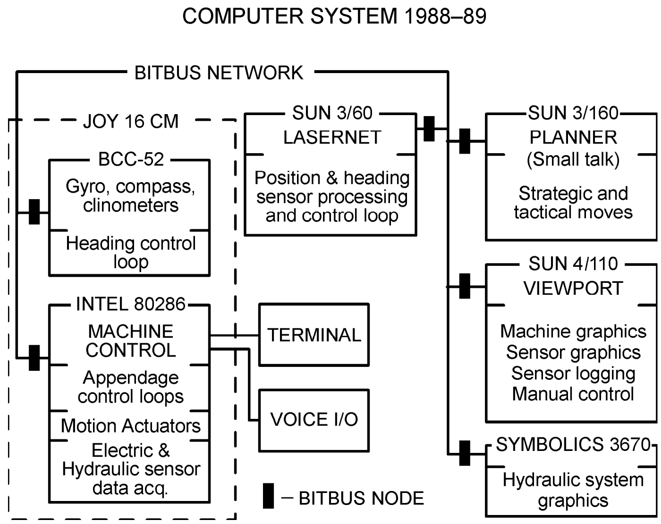

Fig. 22.2.3. US Bureau of Mines autonomous continuous miner computer network (after Schiffbauer, 1990).

latched. Sammarco established set points in degrees of movement reported by the resolvers and evaluated errors caused by lag times in both the hydraulic and electrical systems. The system operated accurately in free space and when subjected to the vibration and stress of cutting coalcrete. Sammarco described initial efforts to fail/safe the control with checks on microprocessor power and software execution, and recommended additional verification of sensors and machine during startup.

In an effort to extend closed-loop control to tramming, Sammarco (1988b) experimented with on-board machine attitude and location sensors including a gyroscope for entry centerline control (yaw), flux gate compasses for redundant heading data referenced to magnetic north, and clinometers for gravity referenced pitch and roll. The objective of the on-board system is to provide attitude and location within a cut. The on-board data will be fused with the laser-based angular positioning system for long-range guidance. Sammarco found that the gyro

1. Drifts 9.73 degrees/hr in Pittsburgh.
2. Has cumulative errors.
3. Has moving parts that will require maintenance.
4. Is very expensive.
5. Needs a 5- to 10-minute warm-up period.
6. Requires special power sources, signal conversion, and control.
7. Provides ±0.02° heading changes within milliseconds.

For comparison, Sammarco determined that local magnetic fields affect the flux gate compass which was not able to keep up with a machine pivoting at 3.3 °/sec, but it does not drift, its errors do not accumulate, it has no moving parts, it does not require warm-up, and it is inexpensive. The present flux gate requires the machine to be stopped for 2 sec to accommodate the present 2-sec averaging window; however, he believes modifications might be possible.

Anderson (1989) developed the laser-based angular positioning system shown in Fig. 22.2.5 for precise, longer-range navigation guidance of the continuous miner in the face area. The positioning system reference is a mobile control structure, patterned after a mobile roof support that will follow the continuous miner along an entry. Either surveyors or a long-range automatic total station will determine the precise coordinates of the mobile control structure after it is set at the face. During the excavation of each cut, the continuous miner will locate itself relative to the mobile control structure with a laser scanner mounted on the mobile control structure seeking retroreflective targets on the miner. Multiple scanners that sweep 90° each and multiple targets reduce obstruction problems and provide redundancy. Anderson's algorithms use two geometries (Fig. 22.2.5) to determine position and attitude, one where two or more laser photodetectors sense the angular position of two or more targets on the machine, and the other where one photodetector senses three or more targets. She is developing a fusion algorithm that assigns confidence (weights) to each angle measured based on the knowledge that certain orientations affect measurement reliability.

Mining machine control will ultimately become intelligent. That is, the miner will be able to alter its cutting plan if sensors report deviations from the conditions and events used to generate initial plans. Bureau of Mines researchers are developing three software modules for this objective. The navigational goal scheduler will plan the sequence of machine actions and positions necessary to complete a cut, and represent them as a series of goals. An action planner will plan the actions necessary to progress from one goal to another. The contingency goal scheduler will allow operations to continue when the miner can not reach a planned goal. It will attempt to discover the problem and plan an alternative goal. To speed the application of intelligent control to mining, the Bureau contracted with the National Bureau of Standards to transfer their standard architecture for intelligent control of manufacturing robotic work cells and the space station to the autonomous continuous miner program (Albus et al., 1987).

The control developed by Sammarco uses only position sensors for controlling cutting horizon; as a result, frequent contingencies may arise because seam height varies dramatically in some mines. Mowrey (1990a) studied waveforms generated by accelerometers mounted on the machine and on the coal to develop algorithms that will automatically discriminate between vibrations resulting from cutting coal and rock. Natural gamma radiation had also been used successfully for cutting horizon definition (Mowrey and Maksimovic, 1990; Nelson and Bessinger, 1990). Infrared imaging is another technique under investigation because it discriminates between coal cutting and rock cutting by producing images of the higher-temperature areas that result when cutting rock (Mowrey, 1990b). However, this technique requires compute-intensive video image processing.

The autonomous continuous miner will cost much more than present machines. Therefore, it must be very reliable to be cost effective. High reliability also reduces maintenance personnel exposure to face hazards. Consequently, Mitchell (1990) is developing an expert system prototype to interrogate the various hydraulic sensors and report on machine health. The system will diagnose hydraulic problems based on sensor and user input. It will explain how it reached conclusions, and provide recommendations for repair. Mitchell used a hierarchy of rules to represent the knowledge of hydraulic diagnostic experts and backward chaining to search through the knowledge base to diagnose the condition of the hydraulic system. She implemented the system with an expert system shell called Goldworks. Berzonsky (1988) covers the electrical system, and eventually, machine health systems will cover all continuous miner components. He developed a prototype expert system for interactive troubleshooting and training of personnel for electrical system fault diagnosis with the Level 5 expert system shell.

Carnegie-Mellon University (CMU) developed the locomotion emulator for the autonomous continuous miner research program at the US Bureau of Mines. It has general locomotion and can emulate the characteristics of different vehicles like a

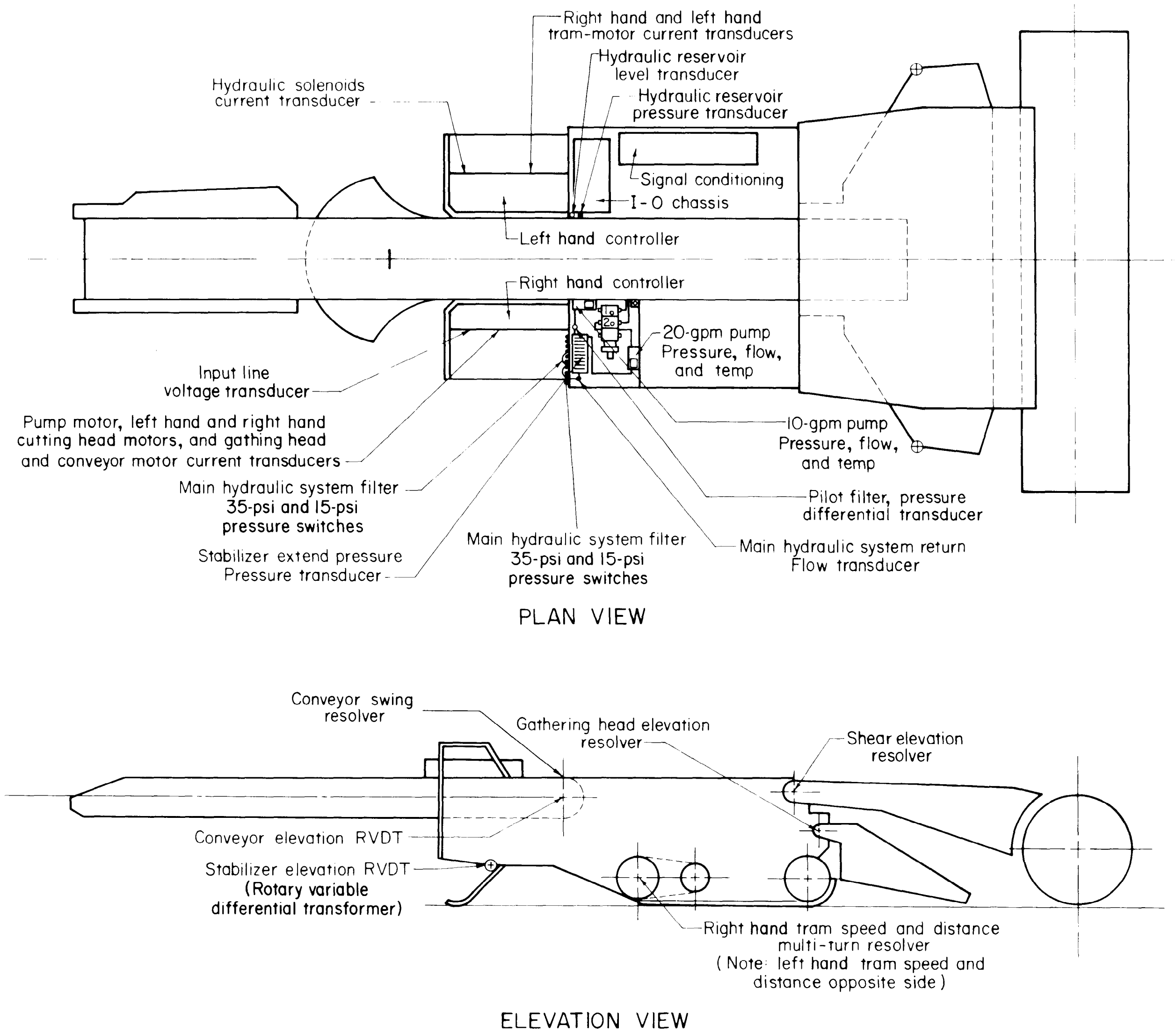

Fig. 22.2.4. US Bureau of Mines autonomous continuous miner sensors (after Sammarco, 1988a and b). Conversion factors: 1 psi = 6.895 kPa, 1 gpm = 3.785 L/min.

unicycle, Ackerman steered, and skid steered units (Schnakenberg, 1990).

22.2.3.2 Teleoperation for a Highwall Mining System

Kwitkowski et al. (1988) teleoperated a low-profile continuous miner and haulage system to extract coal from strip mine highwalls at the METF. The system will mine without exposing personnel, since the operator will remain in a supervisor's station on the highwall bench while the miner and haulage system (Fig. 22.2.6) extend up to 3300 ft (1000 m) into the untapped highwall reserves. The teleoperated miner will use a laser beam as a guidance reference. A tether carries operator commands, audio/video signals, and sensory data between two microcomputers (housed in the supervisory station) and the machine. Bureau researchers modified a low-profile machine (Fig. 22.2.7) that the manufacturer equipped with tethered remote control. They added two-color video systems, microcomputer and communications hardware, explosion-proof housings for the new electronics, and a suite of sensors to measure machine health, attitude, and position. The miner can load coal into a variety of haulage systems, but the researchers chose the multiple-unit continuous haulage system previously developed by a Bureau contractor (Evans and Mayercheck, 1988).

22.2.3.3 Computer-assisted Longwall Equipment

Fisher and Palowitch (1978) and Hartley (1982) described the extensive programs on longwall automation in the 1970s at the US Bureau of Mines, US Department of Energy, and the UK National Coal Board Laboratories. Recent work has focused on coal interface detection, automatic shield advancement (shearer initiation), and system monitoring. Nelson and Bessinger (1990) recently evaluated and applied results of the interface detection research to measure boundary, coal thickness.

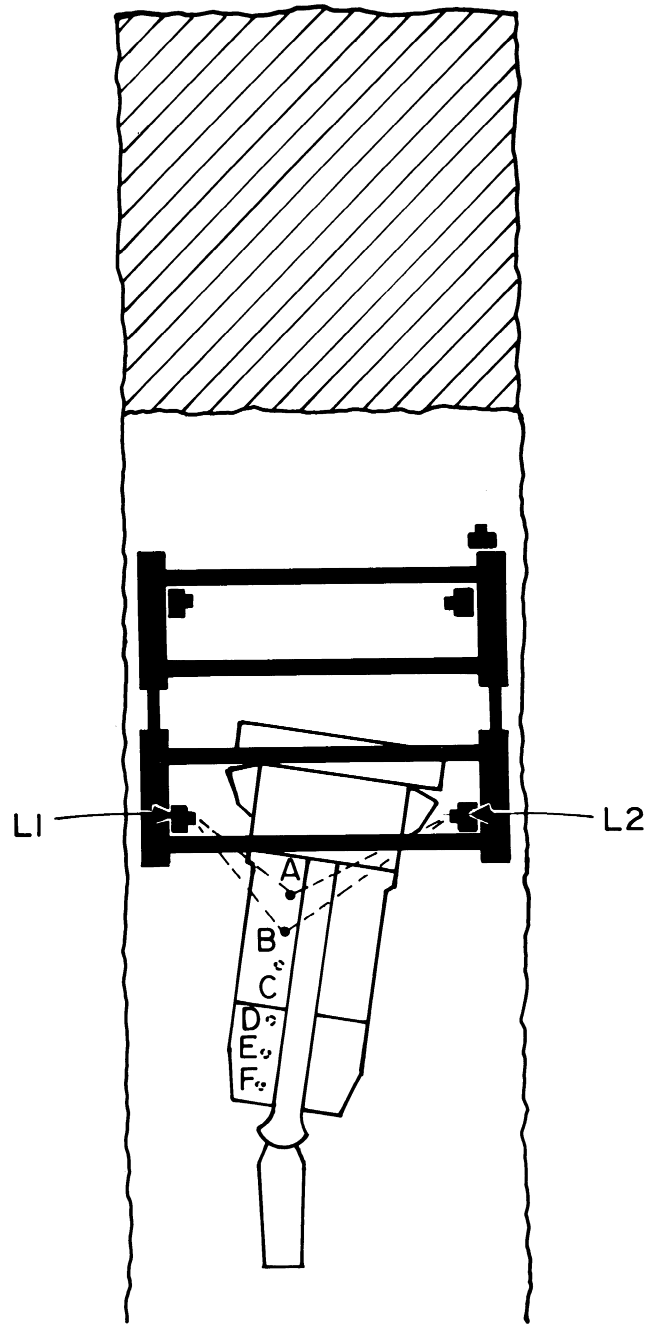

Fig. 22.2.5. US Bureau of Mines autonomous continuous miner laser and target positioning system (after Anderson, 1989).

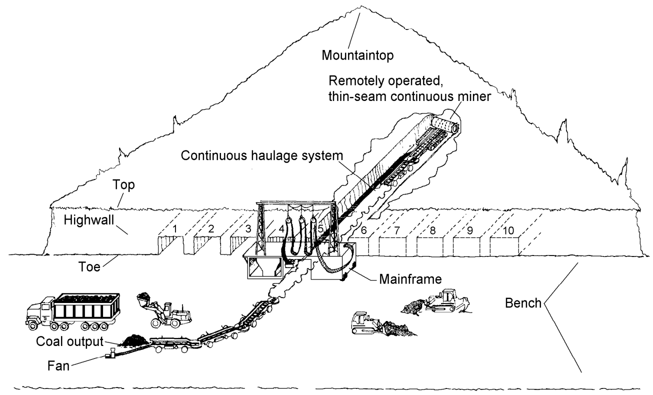

Fig. 22.2.6. Artists conception of the highwall mining system (after Kwitkowski et al., 1988).

Their evaluation revealed that a natural gamma background sensor was most appropriate, and they installed one with a single scintillation crystal, a photomultiplier tube, and a control panel with counts-to-thickness conversion, selectable sampling time, and digital thickness display on shearing machines in two mines. These instruments aided shearer operators to consistently leave 6 to 8 in. (150 to 200 mm) of roof coal in one mine. They successfully modified the sensor for more rugged mounting in shield toes for floor, coal thickness measurement.

Owen (1988) reviewed the development of the National Coal Board monitoring and information processing and reporting systems for longwall faces. The National Coal Board developed standard systems that collect data from in-mine transducers and transmit them to a surface computer for analysis and reporting. The software is divided into modules: a basic operating system, data transmission and interface with applications software, data files, applications software, and tables of colliery characteristics. The number of sensors for machine condition monitoring is growing rapidly and covers haulage and loading equipment and shearers and plows. The surface computer console operator gathers data about delay causes by calling the face whenever the display screen shows production interruptions. The system reports the resulting data at the end of shift and in longer-term management reports. Owen stresses that the information available made several improvements but causes considerable upheaval in traditional mine management structure and practice. Lever et al. (1990) are building similar information systems for continuous mining.

22.2.3.4 Semi-autonomous Load-Haul-Dump Machines

In underground mines, the lowest productivity and greatest hazards occur on mobile mining equipment operating in stopes and at the face. Mobile fragmentation machines are difficult to control autonomously because the level of sensing and intelligence necessary to handle exceptions in the highly variable geologic environment is sophisticated and underdeveloped. Controlling the direction of cutting or drilling and determining the proper levels of force to use in rock masses with variable properties are difficult. However, there are significant economic and safety gains associated with autonomous excavation equipment.

Primary transportation vehicles, which also may include loading, have some of these problems in distinguishing and handling oversize material, but not nearly to the extent of extraction equipment. In addition, they can apply the navigation perception, reasoning, planning, and control successes from the previously described autonomous vehicle programs. LHD vehicles are like many intermediate haulage vehicles such as shuttle cars and trucks. They follow a cyclic operating pattern over various well-defined paths and thus become good candidates for autonomous vehicle technology application. For example, at the AMAX Henderson mine, LHDs load molybdenum ore from drawpoints, tram to an orepass, dump, and return or switch to another drawpoint. King (1988) estimated a favorable return on investment from semi-autonomous operation of the Henderson LHD fleet. An autonomous LHD at Henderson must

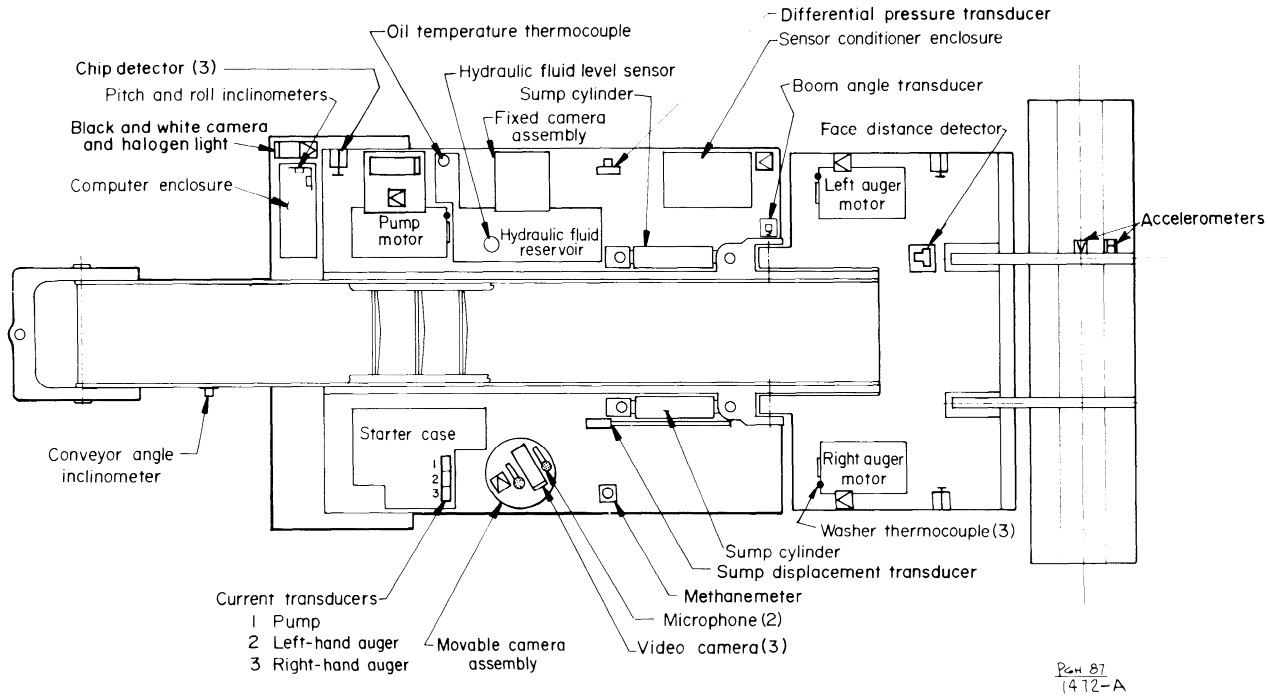

Fig. 22.2.7. Teleoperated highwall miner sensor locations (after Kwitkowski et al., 1988).

1. Sense vehicle position along the route.
2. Relate sensor data to stored map information to determine location.
3. Follow drift centerlines.
4. Plan paths between dump points and orepasses.
5. Compare sensed position with planned position and initiate appropriate control commands.
6. Sense vehicle operating status and vehicle health.
7. Key on features or targets for special tasks like high-speed turns.
8. Perform end-of-travel tasks (loading and dumping).
9. Detect and avoid obstacles.

These goals are similar to those for shuttle cars, trucks, and front-end loaders, and therefore much of the technology is transferable. Thus several researchers are focusing on intelligent control for LHDs. Supervised autonomy can reduce the number of operating units by increasing operating time per shift since computer-controlled machines can operate during lunch and between shifts and reduce operator errors. Inexpensive, robust sensors requiring minimal processing are a key to cost-effective implementation, so ultrasonic ranging is a good candidate. It is appropriate for underground mines because the equipment travels in well-known areas, and the ultrasonic ranger can locate the machine relative to features in an on-board map. For example, range readings to the ribs can help maintain a straight path along the entry centerline, or the LHD can cue on rib features for turning (Strickland and King, 1990).

The problem with ultrasonic ranging is that conventional sensors report the range to the first object in the sensor field of view in a multi-object environment, but the first object may not be the one of interest. Attempts to solve the problem with an array of sensors, fusing with other sensor types, horns to reduce the field of view, and building maps from multiple range readings from different robot locations are not appropriate for mining. For example, world modeling and reasoning software can build a model of an object when mobile robots take many readings from different view points. However, this solution requires the robot to roam around an area, plan paths to reveal occlusions, etc., reason about what it finds, and replan the discovery path. This technique is not appropriate for a vehicle trying to accomplish an industrial task where it has to reach an objective quickly. Therefore, current work is developing algorithms that recognize patterns in ultrasound return signals that distinguish between objects (Gordon and King, 1990).

Baiden (1988) is developing a monitoring system and expert system to report on LHD machine health. It will reduce downtime, lower consumption of components, and increase machine life. Baiden selected a radio link to transfer data from the 73 on-board sensors to a remote microcomputer. Over three months, the system collected 50 megabytes of data describing the conditions of seven machine subsystems: engine, brakes, hydraulics, drive train, electrics, fuel, and exhaust. Baiden is developing data reduction and pattern recognition algorithms and interfacing the monitoring system to a maintenance assistant expert system. The maintenance assistant uses rules to represent the knowledge of LHD maintenance experts in the M.l expert system shell. Because of the difficulty of representing such complex knowledge with rules, he has focused on the braking subsystem to date.

Granholm et al. (1988) studied and tested LHD guidance systems. In one of their tests, they teleoperated an LHD with a TV camera. In another test, the operator drove the LHD using a TV and monitor placed inside a cab with the windows covered. They are also experimenting with painted-line-following techniques similar to those used by automated guided vehicles in factories and warehouses. The researchers painted a 4-in. (100-mm) wide line on the back of a haulage drift and used CCD cameras to locate the line. They found the system could detect 0.4-in (10-mm) deviations from the path.

In a similar teleoperation effort, Nantel et al. (1988) installed a video camera on a LHD and transmitted the image 75 ft (23 m) to a remote operator in the Noranda Lyon Lake mine. The purpose was to help the operator load the bucket fully when operating by remote control in a production drawpoint below a bulk mining stope. This was an important study because for many years, the phases of robotic development in mining will require the use of supervision. Supervisors might need video images to properly control robotic equipment in contingency situations. Video transmission using radio waves (without tethers and leaky feeders) is difficult, as these researchers report. They could practically transmit only 330 ft (100 m). They transmitted digital video data from a camera with 510 $\times$ 492 pixel images with a transmitter providing 1W of video-modulated radio frequency at 439.25 Mhz powered by 12-V dc from the LHD circuit. The LHD operator used the 4-in. (100-mm) monitor provided with the system to successfully load the LHD bucket.

Kallio (1988) reported on similar teleoperation work with cameras mounted on the machine and on the rib. LHD operators expressed that teleoperation was easier than remote control. Kallio's paper also described a computerized equipment control and monitoring system (CECAM), that is a microcomputer-based production and condition data acquisiton and storage system. Kallio further reported that a wire guidance system, similar to that used on factory automated guided vehicles, was used to control an LHD tram path.

22.2.3.5 Computerized Drills

Recent advances have made mine drilling equipment partially automatic. If this trend continues, drills will become more autonomous, which will improve their output, reduce labor cost, but increase their purchase price. So far, the increased capital investment has not shown adequate savings to US mine operators, who use few automatic drills. However, robotic technology is appealing since drilling is an inherent element in most mines' production cycle. Underground and surface metal, nonmetal, and coal mines all employ drills to penetrate rock for exploration, blasting, and ground support. Although the machines' size, shape, and function vary, enough similarities exist for advanced robotic technology duplication. Furthermore, many other industries like tunneling, underground nuclear waste cavity excavation, underground missile site development, and civil construction use mobile heavy drilling equipment.

Bristow (1985), Kelly and England (1986), King (1987), Montan (1984), and Ulvelin and Puhakka (1988) describe the advance of drilling technology from hand drilling in the early 1900s to computer-controlled drilling today. These developments are driven by the fact that human operators are not capable of obtaining maximum equipment productivity. For example, they cannot reposition drills fast enough to keep up with the fast penetration-rate capability of hydraulic drills, and they cannot accurately position drills as quickly as computer control. In addition, when human operators leave for shift change, for example, expensive drills must be shut down unless they are computer controlled.

Bristow (1985) describes a basic computerized jumbo that contains a microprocessor; a control console; angle transducers at boom joints; linear transducers for extension, feed, and crowd; hydraulic sensors; and electro-hydraulic valves. The drill operates in either automatic, manipulator, or manual mode. In automatic mode, a drill pattern in the computer defines the position, direction, and depth of holes in relation to the tunnel axis. The operator can modify and store drill patterns with a portable computer to avoid collaring in holes left from the previous round. The sequence of events begins by inputting a reference into the computer such as the relation between the tunnel axis and a fixed laser beam. Then the drill is manually trammed to the face, and a drill rod is aligned with the laser. The operator presses the navigate button, and the jumbo drills the round if it is physically able to reach all the hole positions. A video display shows the entire pattern and shows which holes have been drilled. The video display graphs penetration rates for each drill to help the operator determine when to change bits and whether geological conditions have changed. The display also shows boom positions relative to the reference axis. In manipulator mode, the computer actuates the boom and drill movements that are manually commanded. The movements are therefore more efficient and accurate in response to "joy sticks" than in the manual mode. Display features are also active in manipulator mode.

Peck, Scoble, and Edwards (1988) described work on algorithms that will make computerized drills more intelligent. Their sensors monitored rotary speed, torque, thrust, air-flushing pressure, and instantaneous penetration rate with a microprocessor on a surface blasthole rig. Then they developed relationships between the changes in these data and rock mass properties. Their work will allow drills to characterize the rock mass. As a result, the control system can recognize and respond to geologic condition changes automatically and optimize penetration rates while minimizing wear and maintenance on drilling equipment.

Hay and Howie (1988) instrumented a roof bolting machine to study similar relationships, but for a different reason. They described work to determine the optimal type and installation of coal mine roof bolts. Furthermore, their research will ultimately provide intelligence for autonomous roof bolting machines. Their system will replace the feel and expertise of the expert roof bolter operator to adjust standard bolting patterns to achieve the best support when rock mass properties change. In cooperation with a private contractor, they built a real-time measurement and display system that calculates instantaneous specific energy at the current bit position in 1-in. (25-mm) intervals and logs each hole drilled on an experimental machine at the US Bureau of Mines Spokane Research Center. Sensors on this machine measure torque, thrust, penetration rate, and rotation rate. The system aids the roof bolter operator to locate voids, inclusions, and changes in strata. The sensors report data to an on-board computer in a permissible enclosure. Some data (specific energy vs. bit and void position) can be downloaded to a removable semiconductor memory device for additional processing on an off-board "fresh air" microcomputer that is also used for software development. Hay and Howie plan to extend the system to closed-loop automatic control by adding servo valves to the hydraulic control circuit. They will add the full-control bolting system that reports anchorage strength and installation torque of each bolt and matches the bolt installation procedure to the existing geologic conditions. Using this combination, the system will identify roof properties during drilling and select and carry out the appropriate roof bolt type and installation procedure to provide optimal support. Hay and Howie plan to add vibration sensors and processing algorithms with an expert system to assess machine health. They can also constantly update geological data for an intelligent mine-wide monitoring system.

22.2.4 SUMMARY

Most of the present work in mining robotics focuses on individual machines. Some researchers integrate them into systems and develop new systems. Some notable examples are Wolfenden and Shaw (1988), Kwitkowski et al. (1988) Schnakenberg (1990), and Frantz and King (1977). Integrated approaches are necessary to develop intelligent mining systems.

A compilation of the individual robot design studies and the synthesis works show that the major components for mining robots are:

1. Mobility.

2. Intelligence.
3. Robust components.
4. Supervised autonomy.
5. Navigation and guidance.
6. Geologic perception.
7. Environmental perception.
8. Location perception.
9. Machine health perception.
10. Obstacle detection and avoidance.
11. Path planning.
12. Closed-loop control.
13. Real-time multiple tasking computer architecture.
14. Intelligent user interfaces.
15. Teleoperation.

A summary of the tasks necessary to perfect a mining robot include the following:

1. Acquire and test candidate sensors
2. Specify and purchase sensors.
3. Mount sensors.
4. Specify and purchase computer processing, memory, data acquisition, control, and interfacing hardware.
5. Mount computer processing, memory, data acquisition, control, and interfacing hardware.
6. Interface sensors.
7. Establish teleoperation capability with communications link and workstation.
8. Develop, encode, test, and modify (a) data reduction algorithms, (b) fusion algorithms, (c) object recognition algorithms, (d) control algorithms, and (e) software architecture.
9. Test and modify computer processing, memory, data acquisition, control, and interfacing hardware.
10. Develop, encode, test, and modify object recognition algorithms.
11. Develop, encode, test, and modify control algorithms.
12. Develop, encode, test, and modify software architecture.
13. Test and modify computer processing, memory, data acquisition, control, and interfacing hardware.

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Chapter 22.3
HYDRAULIC MINING: JET-ASSISTED CUTTING

DAVID A. SUMMERS

22.3.1 INTRODUCTION

Hydraulic mining, the use of water to break and remove rock, is not new. But within the last twenty years, as it has become commercially possible to generate fluid flows of higher pressures, the technology has found new applications. These new ways provide advantages for this method of excavation, which is therefore becoming more widespread. This chapter describes the development of the technology, some of the lessons learned, and some of the applications.

22.3.1.1 Terminology

Although high-pressure water jets were first investigated as a new mining tool, they have become more extensively used for cutting metal, glass, and other industrial materials. The terminology used by this industry is relatively common; only a few terms may need clarification.

22.3.1.2 Definitions

Gooseneck: The sequential combination of two articulated elbows to allow monitor flexibility.

Miners inch: The volume flow rate of water flowing through a l-in. (25mm) square hole under a head of 6 in. (150 mm). Forty miner inches are equal to 1 cfs. (The term is historic; a miners inch roughly translates into a flow of 0.03 m^3/s under a pressure of 1.75 kPa.)

Monitor: The accelerating nozzle, with its support and steering mechanism used in the large-volume excavation of alluvial deposits.

Nozzle: The shaped section at the end of the delivery pipe through which the fluid is accelerated to its final velocity.

Orifice: The smallest cross section within the nozzle area, normally at the downstream end, which controls (for a given volume flow) the velocity of the fluid.

Roadheader: A mechanical excavation machine fitted with a rotating head mounting picks that are dragged over the rock surface. The machine excavates the tunnel face a segment at a time.

Specific energy: The amount of input energy required to remove a unit volume of material.

Tunnel boring machine (TBM): A full-face, continuous mining machine, generally fitted with disk cutters that are compressed against the face of the tunnel as they rotate. The tunnel is excavated across its full cross section at one time.

22.3.2 HISTORICAL BACKGROUND

22.3.2.1 Hydraulic Mining Development

Water naturally erodes soil and rock, but at a speed that is normally too slow to be of interest. However, water can penetrate into the cracks and natural grain boundaries of material weakening it and helping remove softer deposits. Such erosion of soil occurs after a heavy rain. Only a small increase in pressure is required to obtain a water flow that will remove poorly consolidated material.

In the early days of the California gold rush, once the gold-bearing rock had been found, large mine workings were common. Working under a steep slope of relatively soft material was a slow and dangerous process when the material was removed manually. In February 1852, Chabot and Matteson first used a 40-ft (10-m) long, 4-in.(100-mm) diameter rawhide hose to direct a jet of water at the ore and remove the soft material (Kelley, 1954). The water was pressurized by the drop from the top of the bank, some 30 ft (9 m). Since the water also carried the ore to the sluices and cradles, where the gold could be removed, hydraulicking proved to have an overwhelming advantage and rapidly spread throughout the gold fields (Fig. 22.3.1)(Anon., 1853). A reliable supply of water was a perennial problem, and typically cost between $0.75 and $1.00/miners inch. However, this volume increased a miner's production by more than three-and-a-half times over that using a pan, with less physical strain (Wilson, 1912).

The hose was changed to canvas, and in 1855 the goose-neck joint was added (Egenhoff, 1931). This was followed by the addition of a deflector that improved the maneuverability of the monitors that had grown more than 10 ft (3m) in length (Kallenberger, 1970). By 1886, more than 2375 million yd^3 (1868 Mm^3 of dirt had been removed, creating an environmental problem (Egenhoff, 1931). Chapter 15.1 of the *Handbook* discusses hydraulicking as a current placer mining method.

Water jets remove material on a grain-by-grain basis. But the sandy rock containing the gold held a significant amount of fine material that was also liberated. These fines did not settle as the water passed down through the sluices, because of its relatively high speed. The particles only settled out when they reached the relatively slow-moving rivers that flowed from the mountains to the sea. This rapidly filled the rivers with silt, and when the weather produced rain, the rivers overflowed their banks, flooding the adjacent farms. In order to stop this silting of the rivers, the farmers took the mining companies to court, and in 1883 were successful in obtaining injunctions from Judge Sawyer (Kelley, 1959), which effectively closed many mines. This "anti-dumping" legislation marked the end of the boom period in the California hydraulic mines that had produced more than $150 million in gold. A contemporary comment stated, "At one blow, the Sawyer decision reduced the visible assets of the State of California by 100 million dollars" (Kallenberger, 1970).

The technology had spread through most of the west and abroad. In 1867 a device invented by Chausov was used in the area around Lake Baikal in Russia for gold mining (Yufin, 1965). By the turn of the century, water jets were used to mine peat in Prussia, and in 1935, Muchnik (1952) first used water jets underground for the mining of coal. Hydraulicking proved particularly useful in the thin, steeply dipping coal seams of the Don and Ural basins, but it was not until 1952 that a complete hydraulic operation was started at the Tyrganskii Uklony Mine (Muchnik, 1956).

Hydraulic mining expanded steadily in Russia because of the relatively low cost of the technology, the low levels of manpower

Fig. 22.3.1. Early water-jet monitor being used in Idaho (Idaho Historical Society).

required, and the relatively high volume of coal that could be produced from difficult-to-mine deposits. Trials were also performed in other countries. Because of the high volumes of water required, and the difficulty in adapting this technology to relatively flat seams, experiments in Germany (Gottwald, 1968) and the United Kingdom (Jenkins, 1961) were not successful. In Japan (Sato, 1988), China (Tian, Sun, and Zou, 1985), and, most effectively, in Canada (Grimley, 1972), successful mines were developed.

At the Sparwood mine in British Columbia, a monitor used 1200 gpm (4.7 m^3/min) of water at pressures up to 1800 psi (12.5 MPa), to successfully mine coal from a 40-ft (12-m) thick seam dipping at some 45°. The operation had an average productivity, over 10 years, of 3000 tons (2700 t)/shift from a machine that required only two men to operate. As a further advantage, the water used to mine the coal also carried the coal from the mine. Economic conditions and other problems, however, have recently caused both this mine and the one in Japan to close, so that at present the only extensive hydraulic mining operations are being practiced in China and Russia.

22.3.2.2 Water-jet Cutting of Rock

EUROPEAN BEGINNINGS. Hydraulic mining of coal and soft rock was sufficiently successful that trials were begun on cutting stronger rock. Because such rock was believed to be harder to break, Russian investigators built a single-shot water cannon to drive slugs of water at pressures of up to 100,000 psi (700 MPa) (Chermensky, 1976). Studies in the United Kingdom also used a water cannon to examine the ability of water jets to cut rock (Leach and Walker, 1966).

COOLEY THEORY. In the United States, Cooley (1975) extended the European work using a cannon capable of reaching 600,000 psi (4200 MPa). On the basis that the best system was the most efficient, Cooley analyzed his results (Fig. 22.3.2) to conclude that a jet pressure of 20 times the uniaxial compressive strength of the rock should be used. Thus 600,000 psi would be needed to effectively cut 30,000 psi (210-MPa) granite.

The same premise formed the basis of work carried out at Bendix Laboratories and at IIT Research Institute (IITRI) under Singh (Labus, 1976), who similarly sought a more effective method of cutting rock by building very high-pressure water-jet systems.

COOLEY THEORY REVISED. The experimental data that Cooley analyzed came from single-shot devices, with rock and nozzle fixed. Other work, in Russia and Leeds (Summers, 1968) had, however, shown that a water jet could cut granite at 9000 psi (63 MPa). This occurred when a continuous jet was traversed over the rock. Penetration of small fluid wedges into the cracks between the granite grains was followed by pressurization of the water, crack extension, and ultimate material removal. Although

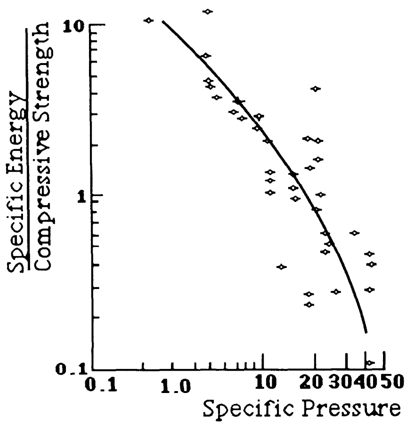


Fig. 22.3.2. Relationship between specific pressure and specific energy, proposed by Cooley.

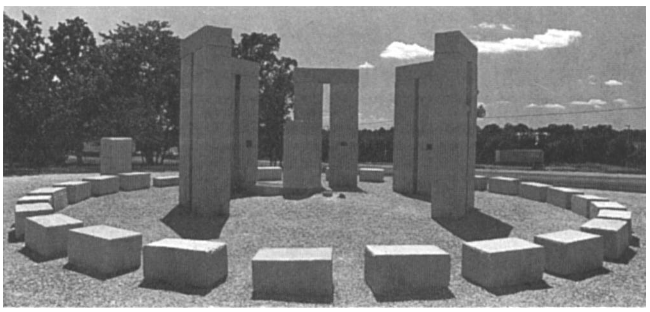


Fig. 22.3.3. Stonehenge, a UMR monument carved from 30,000-psi (207-MPa) granite at 15,000-psi (103-MPa) jet pressure.

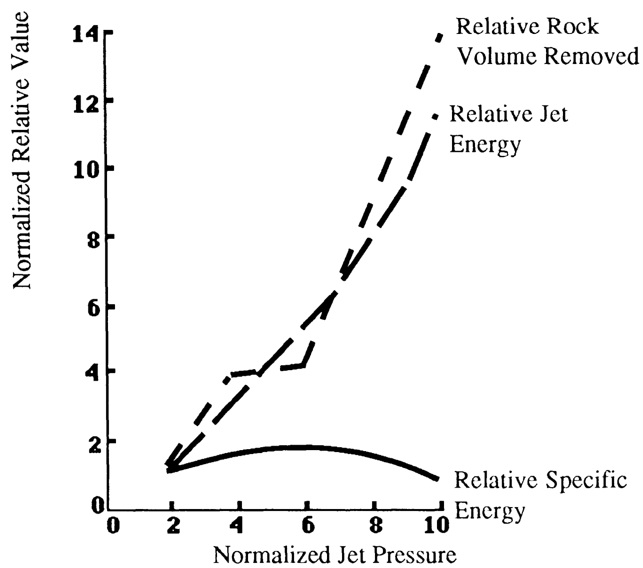


Fig. 22.3.4. Specific energy change with change in relative jet pressure.

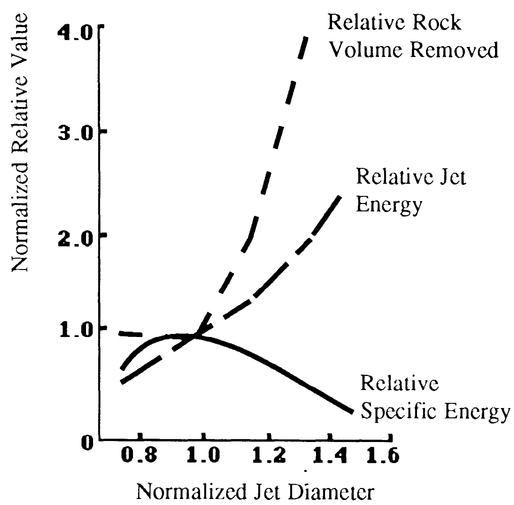


Fig. 22.3.5. Specific energy change with change in relative flow rate (nozzle diameter).

slow, such a jet drilled a hole 9 in. (230 mm) deep in a granite block. The jet pressure was less than 1.5% of that suggested in the earlier relationship.

Subsequent development has produced equipment for cutting soil and rock at commercially available pressures, as shown by the cutting of the UMR (University of Missouri—Rolla) Stonehenge (Fig. 22.3.3). Research has shown that the mechanism of crack growth can be more easily exploited by increasing the fluid flow at a given pressure rather than by increasing the pressure at a constant flow. This can be illustrated by data from a factorial experiment in which both flow rate and pressure were varied in cutting coal. As the pressure increased (Fig. 22.3.4), so the volume of coal removed increased, but the specific energy of cutting only decreased slightly. In contrast, where the power was increased by enlarging the jet (Fig. 22.3.5), the volume removed increased more than equivalently, with a larger drop in specific energy.

This mechanism can be usefully exploited in concrete cutting. Normal concrete has small microcracks around the aggregate boundaries. As the concrete weathers, these cracks extend and the concrete strength reduces. Deteriorated concrete is thus identified as material with significant crack lengths. The pressure of a water jet can be adjusted so that it will easily extend the larger cracks in weathered material, but has insufficient pressure to enlarge the much smaller cracks in "good" concrete. Thus one can develop a tool that can evaluate and remove only damaged material.

22.3.2.3 Limitations of Water Jets Alone

Because water jets penetrate material through crack extension, it is difficult for a water jet to easily penetrate fine-grained materials and those with a high intergranular strength. However, if the crack density under the attacking water jet could be increased, much more material could be removed. Three different

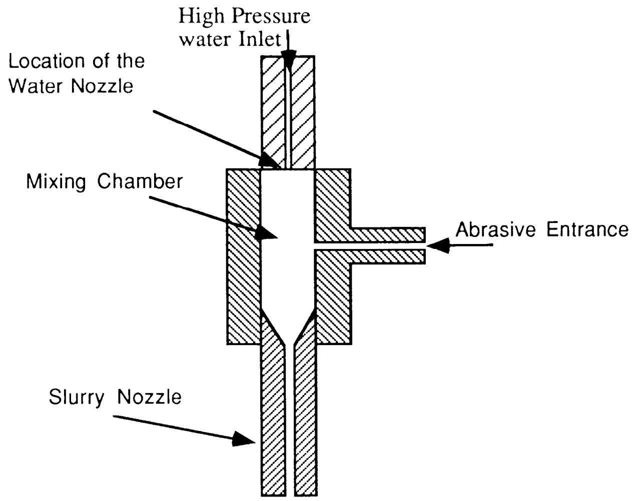


Fig. 22.3.6. Conventional abrasive injection into water.

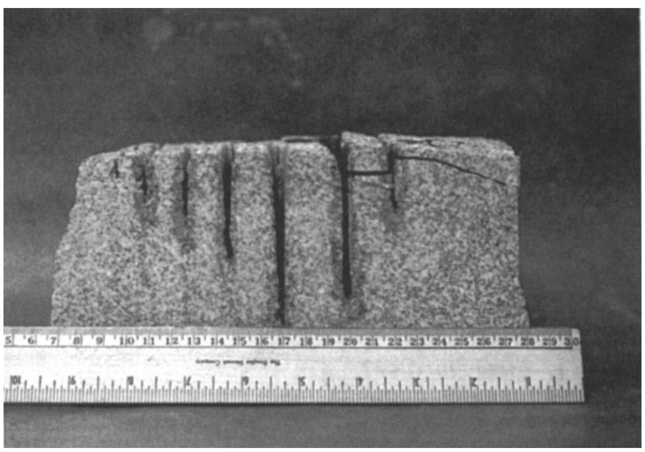


Fig. 22.3.7. Granite block cut by a 30,000-psi (207-MPa) water jet with increasing concentrations of abrasive. Cuts start at the left with no abrasive in the water.

basic mechanisms have been proposed to create these additional cracks.

The most widely used method in mining is to add a water jet to a mechanical cutting device. This method has proven to be the most successful to date. Two other approaches have been developed. The first is to add an abrasive to the water-jet stream (Fig. 22.3.6). By accelerating the abrasive to the fluid velocity, the two combine to cut slots up to ten times deeper than water alone (Fig. 22.3.7). First developed for industrial cleaning, it was modified for use at much higher jet pressures (Hashish, 1982). By adding 1 lb/min (0.4 kg/min) of a garnet abrasive to a water-jet flow of 1 gpm (4 L/min) at 35,000 psi (250 MPa), a virtually universal cutting tool was developed.

Although abrasive-laden water jets are making significant inroads into industry, particularly in the cutting of metals and ceramics (Posner and Kim, 1987), the high quality required for the water supply, and the relative delicacy of the equipment has limited its use in mining. Two solutions are currently being developed. At the US Bureau of Mines (USBM) in the Twin Cities, Savanick and Krawza (1987) have shown that the mixing of abrasive into a larger volume flow at 10,000 psi (70 MPa) (Fig. 22.3.8) will create a jet that can drill through quartzite at 6 in./min (150 mm/min). More recently, researchers at UMR have used equipment developed in England (Fairhurst, Heron, and Saunders, 1986) to drill quartzite at equivalent penetration rates but with a jet pressure of only 5000 psi (35 MPa). The jet is formed by mixing abrasive into the jet before the water is finally accelerated (Fig. 22.3.9). Although both tools have a potential in mining, neither has reached widespread use.

The third alternative is to induce cavitation bubbles into the water-jet stream (Johnson et al., 1972). The destructive nature of cavitation has been known for many years. The power available is illustrated by the damage to the Tarbela High Dam Project (Kenn, 1983). Due to an unfortunate geometry, water released into the main generator tunnel during the start-up process cavitated as it left the control gates. Within 24 hrs, the induced cavitation had demolished an 8-ft (2.4-m) concrete wall and chewed a hole 11 ft (3.4 m) deep into the tunnel wall. However, cavitating jets have limits in rock cutting, not the least of which is ensuring that cavitation does not destroy the cutting tool as fast as it erodes the rock. Cavitating jets, while promising, have been sidelined as an area of research recently, and their application in the near future is unlikely.

22.3.3 WATER-JET-ASSISTED CUTTING

22.3.3.1 Early American Research

In the early 1970s, there was considerable concern in the USBM over the dust and ignition hazards created when machines excavate hard rock. One way to suppress these hazards was to add water to the cutting head. Low-pressure jets were kept from the cutter tips by the cuttings, so higher pressures were proposed. Such a jet would reach the crushing zone around the tool, dampening the dust and cooling the bit. The jet would also drive water into weak planes in the coal, reducing its resistance to cutting.

Field trials were undertaken at the USBM Experimental Mine at Bruceton, PA (McNary et al., 1976). High-pressure water was fed to a small roadheading machine, where it was formed into a jet directed to cut into the coal between the drums of the roadheader. Lower cutting forces were measured, and dust was reduced by 70%. Although this program was discontinued, it provided an incentive for three subsequent programs.

22.3.3.2 Jet-assisted Tunnel Boring Machines

FIRST FIELD TRIAL. The slow pace of tunnel construction in the early 1970s led the National Science Foundation (NSF) to seek better methods of cutting rock. In a courageous and forthright approach, the agency made initial awards to a wide range of sources (Brown et al., 1976) to develop ideas of possible merit and assess their viability. At that time, tunnel boring machines were becoming popular, but they suffered from rapid cutter wear and the limited horsepower that could be transmitted through the gears and bearings to the rock surface. High-pressure water jets that transmit high levels of horsepower through very small erosion-resistant pipes and nozzles were thus a competitive candidate in the NSF program.

A water-jet-assisted TBM was built by a consortium consisting of the Robbins Co., Flow Industries, and Colorado School of Mines (CSM). Preliminary tests at the CSM Earth Mechanics Institute showed that water jets would improve the performance

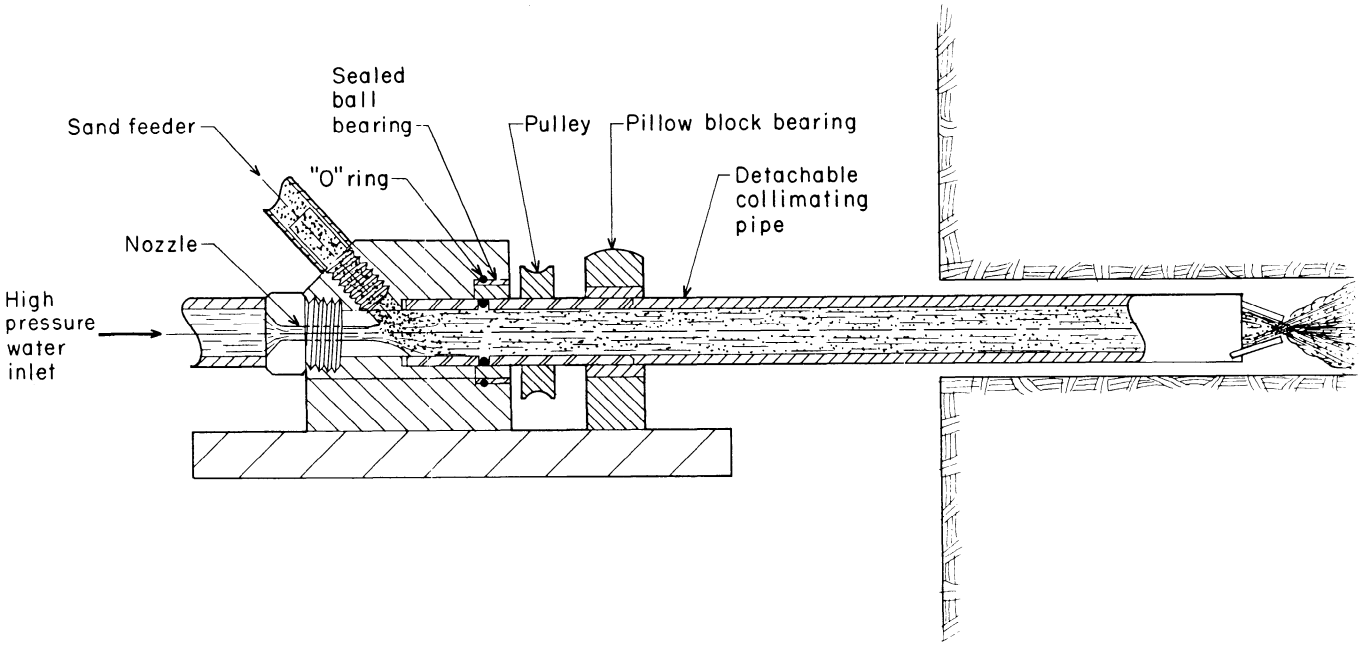


Fig. 22.3.8. An abrasive water-jet drill.

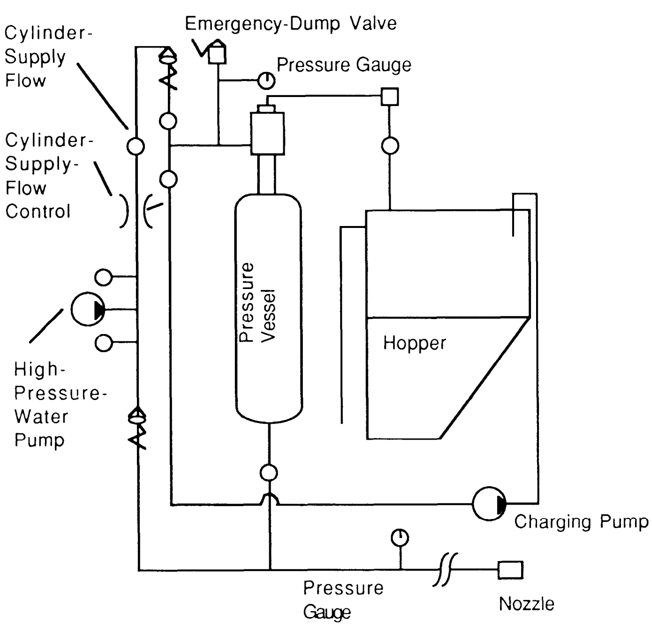


Fig. 22.3.9. Early schematic of DIAJET method of abrasive injection.

of the disk cutters of a TBM in two ways (Miller and Wang, 1976). Improvements were found both if the jets were directed to work in the same groove as the disk cutter and also if they cut an intermediate path between two adjacent cutters.

A field trial, funded by NSF and the USBM, was performed, in late 1976, at a granite quarry in Skykomish, WA. An 8-ft (2.4-m) diameter TBM was fitted with a number of nozzles fed from

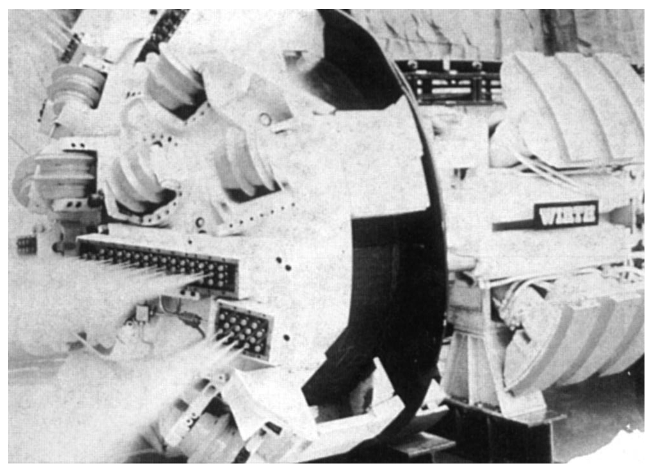


Fig. 22.3.10. Water-jet-assisted tunnel boring machine.

two 500-hp (350-kW) pumps (Fig. 22.3.10). After advancing the TBM without the jets, to generate a database and locate the machine in a consistent rock, the full power of the water jets was applied. The machine immediately began to advance at a faster rate. Unfortunately, the initial nozzle body assembly was located too close to the face of the machine and the unexpectedly rapid advance left large ribs of rock between the disk cutters. These ribs protruded into the path of the nozzles and caused significant damage to them and their supports.

Under normal circumstances, this relatively minor accident would not be worthy of comment. However, its impact on machine design philosophy had a great and often unappreciated effect on machine performance. To avoid problems the nozzles were set back from the face, out of the path of the rock between the cutters. This rock would then be chipped away by the cutters,

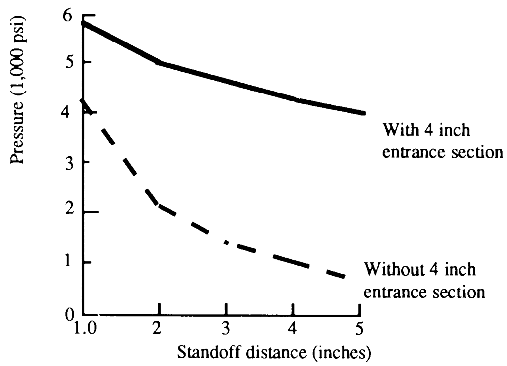


Fig. 22.3.11. Improvement in pressure where a 4-in. (100-mm) tube is used to lead into a water-jet nozzle.

before it reached the nozzles. To understand the unforeseen consequence, one must consider the fluid-flow path to the nozzle.

Water was supplied to the TBM through a fixed line at a pressure of 55,000 psi (380 MPa). In order to feed water to the nozzles in the rotating cutting head, it had to pass through a rotary coupling. Given the state of the art, a single, centrally located high-pressure swivel was mounted on the TBM. The high-pressure water was fed from this swivel into a rotating manifold on the face of the machine, where it divided to enter the nozzles.

Because of the low-volume flow in each jet, it was assumed that a simple orifice inserted into the manifold would provide an adequate jet. In retrospect, this has been found to be incorrect. Recent work at the USBM (Fig. 22.3.11) has shown the need for a short entrance section between the manifold and the nozzle (Kovscek, Taylor, and Thimons, 1988). This pipe stabilizes the jet flow before it leaves the orifice. Without the stabilizing section, the jets issuing from the nozzles were not of high quality.

The definition of *high quality* relates to the internal structure of the jet, which, in turn, controls jet effectiveness. The simplest illustration involves the range of the jet. As the jet leaves the nozzle, it is decelerated and disrupted by the air through which it passes. Thus each jet has an effective range. In an effective jet, this range is on the order of 200 nozzle diameters (Selberg and Barker, 1977). To achieve high pressures, at the low flow volumes used with a TBM, small orifice diameters about 0.01 in. (0.25 mm) are normally used. The effective range for such a jet would be 200 times 0.01 in, or 2 in. (50 mm). However, in order to protect the nozzles, they are frequently set back 4 in. (100 mm) from the rock. Thus the majority of the jet energy has been lost before it reaches the rock, severely restricting system performance. Nevertheless, the Skykomish trials showed that adding water jets to a TBM could improve performance by roughly 37% (Miller and Wang, 1976).

INITIAL PROBLEMS. The large horsepower apparently required for the water jet system suggested that these improvements would only be obtained at high cost. The move toward stronger disks which cut deeper into the rock, when taken with the high lateral traversing speeds of the cutters on the outer edges of the machine face, placed increasing demands upon the performance of the water jets. NSF commissioned Hustrulid (1976) to compare these power requirements with the potential gains from adding water jets to future TBMs. It was his conclusion, being unaware of the power loss problem, that, the gains would not be sufficient to justify the additional effort. As a result of that review, further research in the United States was halted.

EUROPEAN STUDIES. The potential gain from water-jet assistance still appeared attractive. Under an agreement with Bergbau-Forschung, a full-scale TBM was fitted with a modified array of jets and tested in Germany. At a fixed advance rate, force reductions above 50% were found when the jets were used (Knickmeyer and Baumann, 1987). This is an advantage since TBMs require considerable force to penetrate rocks. To achieve this force, large, heavy and expensive equipment is required. Adding water jets allows the TBM to drive at the same speed, but, with less force being needed, a smaller, lighter, and cheaper machine can be used. This advance has, however, been overtaken by another development, which has supplanted the interest in water-jet-assisted full-face tunneling machines.

22.3.3.3 Alternative Approaches

MOST EFFECTIVE PRESSURE. The use of 50,000-psi (350-MPa) pressure on the TBM arose, in part, from the Cooley study, and the assumption that high pressures were needed to cut granite. Although this is not the case, a similar discussion, on the most effective pressure for cutting coal, was taking place at the same time.

The benefits of hydraulic coal mining were clear by this time. However, most of the coal in the United States lies in horizontal seams where disposal of large volumes of water is difficult and where the underlying shale deteriorates when wet. Further, tests by Palowitch had not provided the optimistic levels of production suggested by work elsewhere (Frank, Fogelson, and Chester, 1972). Accordingly, the USBM sought better alternatives for using water jets to mine coal.

The most effective pressure to mine coal first had to be established. The Bureau funded three programs. The first used water jets at a pressure of 10,000 psi (70 MPa) (Summers, Barker, and Mazurkiewicz, 1977); the second suggested a pressure of 60,000 psi (420 MPa) (Singh, Labus, and Finlayson, 1974); while a third program designed a machine to work at 100,000 psi (700 MPa) (Read et al., 1974). After first-year funding, the programs at the higher pressures were terminated and the low-pressure program (at UMR) was continued.

COMPARISON OF RESULTS. It is important to explain this choice. It may be difficult to understand why, when a jet at a pressure of 60,000 psi (420 MPa) is not effective, that a 10,000-psi (70-MPa) system can work.

The reason for this apparent *non sequitur* lies partly in the volume of water in the cutting jet, and partly in the way in which jets penetrate coal at different pressures. Under normal circumstances, for equivalent power, the higher the pressure at which a jet is operated, the lower the volume which is available. This can be seen from the basic hydraulic power equation (in English units):

$$\text{horsepower} = \frac{\text{flow (gpm)} \times \text{pressure (psi)}}{1714} \tag{22.3.1}$$

However, the jet velocity increases as the pressure increases, initially in a form approximated by:

$$\text{velocity (fps)} = 12.5\sqrt{\text{pressure (psi)}} \tag{22.3.2}$$

The reason for the approximation is that the relationship holds only directly at the orifice, is controlled by the nozzle design, and neglects compressibility, which becomes a factor at pressures above 15,000 psi (100 MPa). As the water pressure increases, the volume available not only reduces, but must also flow faster. This means that the diameter of the jet must be significantly reduced in order to conserve volume. This causes two problems.

Initially, experiments show that, to a first approximation, the depth to which a water jet will cut is given by:

$$\text{depth of cut} = k\,\frac{\text{pressure}^{1.0}\ \text{diameter}^{1.5}}{\text{traverse velocity}^{0.33}} \qquad (22.3.3)$$

where k is a constant that varies with the material being cut and the units being used.

Therefore, the first problem is that under constant flow, increasing the pressure has less effect than decreasing the diameter, and performance may not increase as much as anticipated as pressure goes up (see Figs. 22.3.4 and 22.3.5). Under a constant horsepower restriction, an actual decrease in performance may result.

The second problem relates to the way a jet cuts. Lower-pressure water jets penetrate into and exploit pre-existing cracks in the coal. In this way a water jet at 10,000 psi (70 MPa) can cut a single slot in coal that may be 25 times the jet width. However, at a higher pressure, as the diameter is reduced, not only does the jet hit fewer cracks, but it is so powerful that it cuts right past the crack rather than taking advantage of it. The result is a very narrow slit cut into the material.

This narrow cut raises yet another problem in rock removal. Rock underground is under considerable vertical and horizontal stress. If a very narrow slot is cut into that material, then the volume of rock stress-relieved by the cut is often sufficient to swell and close the cut, reestablishing pressure on the cut material, and holding it in place. If a wider slot is cut, then this will allow total stress relief of the rock/coal column, which becomes easier to move. This behavior gave a major advantage to the development of the UMR machine.

Hydrominer and Its Development. Before introduction of the longwall shearer, one of the most successful mining machines in England was the Meco-Moore cutter loader. This used two horizontal cutting booms to slot a longwall face, while a third vertical cutter ran at the back of the cut, freeing the web from the solid. The resulting block of coal fell under its own weight and was transported to the face conveyor by a small cross conveyor. This machine produced large coal with relatively little dust.

In 1974, the USBM funded development of the Hydrominer, in which a similar concept was adapted to improve the performance of a coal plow (Summers, Barker, and Mazurkiewicz, 1977). Four oscillating water-jet pairs were placed around the perimeter of a plow blade so that the coal ahead of the blade would be undercut, cut at the back, and at the top of the machine (Fig. 22.3.12). Thus, as with the Meco-Moore machine, the coal would be de-stressed and easy to move.

Surface trials showed that the machine produced large coal, important when it costs \$1.50/ton (\$1.70/t) to clean 4-in. (100-mm) coal and \$6.00/ton (\$6.70/t) to clean –0.5in. (–1.25-mm) coal. The coal was mined without dust or the risk of spark ignition. Wide slots were ensured by dividing the water at each nozzle into two adjacent streams, thus cutting a slot 2 in. (50 mm) wide.

While two generations of cutting heads were built in the United States, the technology was not tested underground. The German manufacturer GHH worked with Bergbau-Forshung to develop and field test a different model based upon the same principle. Although the machine mined coal at 65 fpm (20 m/min) on a longwall face with a 1.5-ft (0.5-m) web in 5-ft (1.4-m) high coal (three times the production of an equivalent shearer), some difficulties were observed (Schwarting et al., 1981). These related in part to the use of single jets instead of twin jets. The single, narrow jet slot under the coal was closed by stress relief, and the design did not provide a continuous mechanical edge along the face of the plow to assist in the breakout of the coal.

The water jets only cut the perimeter of the web, while the central mass of coal was removed by the mechanical wedging action of the plow blade. The machine showed considerable promise despite the early problems with its use. However, the potential market size for this machine was not considered sufficient to justify the cost of development. The concept behind the machine has, however, been incorporated into the design of a new version being constructed under US Department of Energy funding.

22.3.4 FUNDAMENTALS OF JET-ASSISTED CUTTING

Although there are many situations in which high pressure water jets and water jets working with mechanical assistance have shown potential, as yet they have made relatively little impact on the industry. One area in which water jets have, however, already found an application is directly assisting in cutting by mechanical drag bits. Before discussing the development of the two major efforts in this area, it is pertinent to describe the underlying premise upon which this technology is based and to provide an insight into several of the critical points that are involved in the successful application of this new technology. The most cogent summary of this technology has been given by Geier, Hood, and Thimons (1987) and it is appropriate to review that here (also see Chapter 9.1 in this *Handbook*).

22.3.4.1 Drag-bit Cutting Fundamentals

When a cutting tool is dragged across a rock surface, it will crush the rock ahead of it and successively chip pieces of material from the surface until a single large chip breaks out ahead of the tool. This frees the tool from the high forces and confinement, allowing it to reinitiate the cycle (Fig. 22.3.13). The movement of the tool over the surface forces some of the crushed rock under the tool at relatively high pressure. As the tool drags over this material there is considerable friction that can develop very high temperatures and is a source for face ignitions of gas. Concurrently, the crushed material dissipates the force exerted on the uncut material since the force is evenly distributed around the envelope of this zone. The crushed material is thus a significant hindrance to the performance of the tool. The high pressures and temperatures that are exerted on the tool lead to premature tool failure.

22.3.4.2 Principles of Water-jet Assistance

The most obvious use of a water jet is to assist the cutting tool by removing the crushed rock from the vicinity of the tool. Thus the material causing premature tool failure is eliminated. At the same time the force applied by the pick is no longer distributed around the crushed zone, but is concentrated at those prominent points in the rock resisting pick advance. The force is higher, and the rock fails more rapidly and at lower overall levels of applied force. The removal of the crushed material in a stream of water naturally reduces the amount of dust that would otherwise be liberated into the air.

The water jets must, however, be applied in the right location and at the right flow rate and pressure. To act on the crushed zone, it has been found most effective if the jets are directed to within 0.07 in. (1.8 mm) of the leading edge of the cutting tool

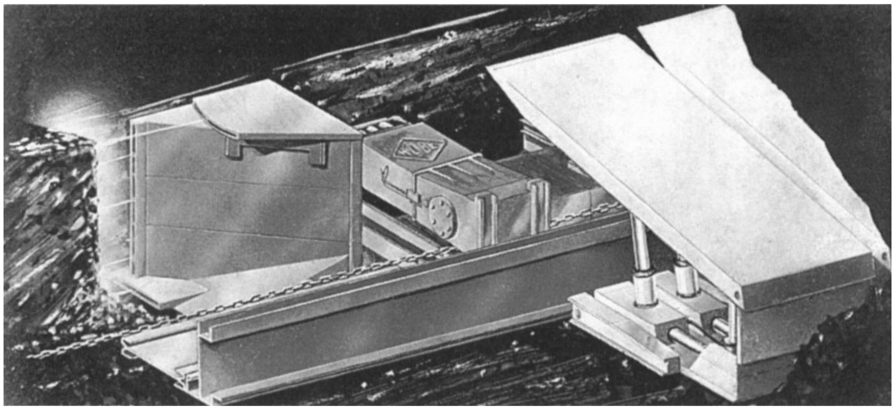


Fig. 22.3.12. Artist's sketch of Hydrominer.

(Fig. 22.3.14). This requires careful positioning of the jet nozzles, which must concurrently be placed as close to the leading edge of the tool as can safely be accomplished in order to get the most energy transfer to the material to be removed. It cannot be emphasized too strongly that, if the jet is not directed at the proper place within the zone of crushed rock right at the base of the tool, then its effectiveness is much reduced. In fact, poor positioning can result in no improvement in cutting conditions. Many tests that have reported poor results have suffered from this failure to accurately locate the point of jet application.

The gain in performance is much more obvious when water jets are added to radial picks rather than point-attack picks. In the latter case, as Hurt and Evans (1980) have pointed out, the removal of material at the tip of the tool merely shifts the force application to the shoulders of the tool, often with a reduction in tool effectiveness. Thus a different strategy must be sought for more effective application in this latter case.

22.3.4.3 Water-jet Cutting Parameters

In order for the jets to do the most good in removing the fine material around the pick, it is necessary that the jet reach that material with sufficient energy. Apart from ensuring that the jet is directed toward the correct location, it must be remembered that the nozzle supplying the jet is set back a considerable distance from the surface. Much discussion has been given to the question of raising the pressure of the jet in order to improve performance of water-jet-assisted machines. Preliminary experimentation has shown that increased pressure has not had any significant effect. This is really not surprising if the actual role of the water jets is as described previously. The need for pressure is to ensure that the jets reach that crushed material, through any fragments of uncut material that may pass between that surface and the nozzle. However, as was discussed earlier, jet effectiveness is better achieved in many documented cases where water flow rather than jet pressure is increased. Given the success that jets have had at lower flow rates for dust suppression on working faces, there is some room to increase the flow rate to the nozzles (by increasing the diameters) and to operate the jets at lower pressure. This would further reduce the cost of the additional water-jet components in the hybrid systems and may produce even greater gains than those catalogued to date.

22.3.5 JET-ASSISTED DRAG BITS

22.3.5.1 Preliminary Studies

In 1974, Hood began a program in South Africa to improve the performance of mechanical bits in cutting the quartzites of the gold reefs. The major problem with the bits was that they melted under the forces necessary to cut the rock (Hood, 1977). Hood calculated that a high jet pressure would be needed to get sufficient water into the cutting area to cool the bit and keep it intact. Experiments showed not only improved bit life, but also a greater depth of cut with lower cutting forces (Fig. 22.3.15). Where a drag bit without water would reach maximum available force with a cutting depth of 0.2 in. (5 mm), with two 10,000-psi (70-MPa) jets on the corners of the bit, it could cut 0.35 in. (9 mm) without stalling the machine.

This significant increase in performance was due to more than keeping the tool cool, and reflected a synergistic interaction between the water jet and the cutting bit in the zone of rock reaction. Underground trials of the equipment then showed that it was possible to achieve even greater relative performance in the field.

22.3.5.2 American Investigations

Report of Hood's work, when combined with the research at CSM, led to an investigation of water-jet-assisted pick and drag-bit cutting (Ropchan, Wang, and Wolgamott, 1980) in coal measure rocks. The benefits found were rock specific, but it was projected that the addition of water jets to a machine would reduce costs per foot (meter) drilled by up to 40%. However, the position of the water jet relative to the tip of the cutting bit was shown to be critical.

22.3.5.3 Cooperative Program

Following this work, the US Department of Energy collaborated with the National Coal Board (NCB), to develop a water-jet-assisted roadheader. Early in 1980, a Dosco MK2A machine was fitted with an intensifier delivering water at 10,000 psi (70 MPa) to 15 nozzles mounted on the cutterhead (Tomlin, 1981).

Rock at the test site, however, was too strong (average range: 17,500 to 20,000 psi, or 120 to 140 MPa) for the lightweight MK2A. A sandstone face was therefore constructed across the end of the tunnel. Few benefits, in terms of cutting performance, were obtained in cutting the sandstone, which the machine could cut adequately under its own power. These benefits were the elimination of frictional sparking and the virtual elimination of dust.

A more dramatic result was, however, achieved when the cutting head was turned onto the native rock. Without water jets, the pick head could not be kept in the cut, due to boom instability, and the head continuously stalled under overload. At the same time it was noted that the picks were rapidly worn and chipped during the cutting. When point-attack picks with water-

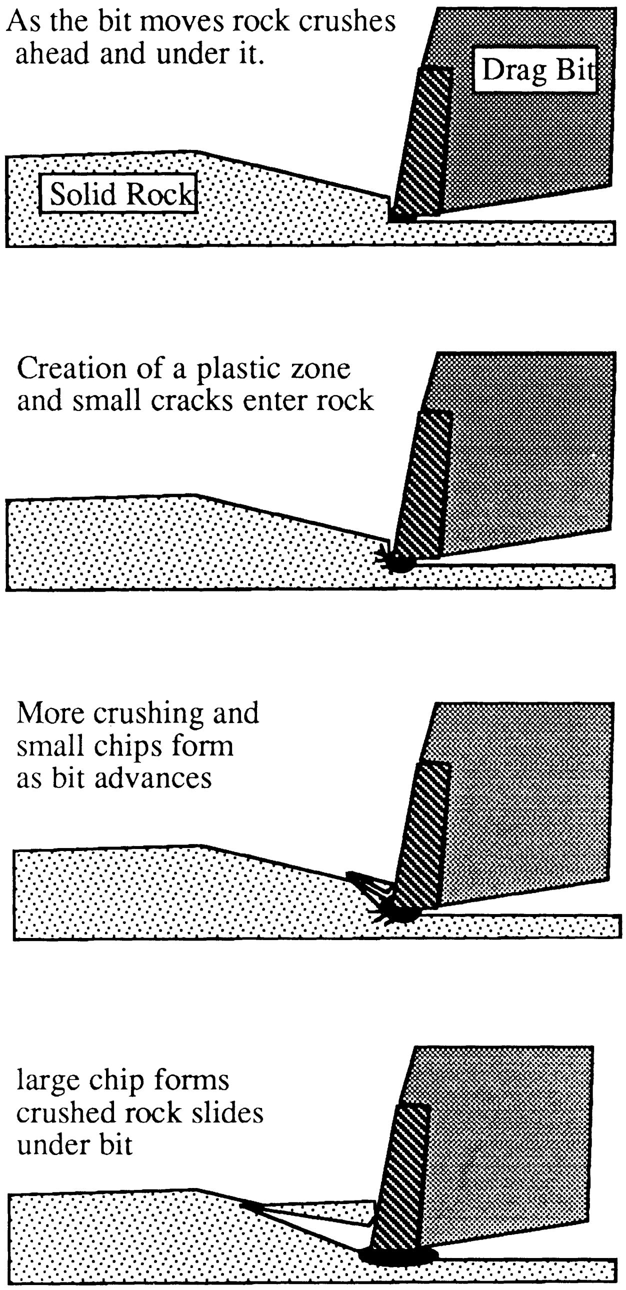
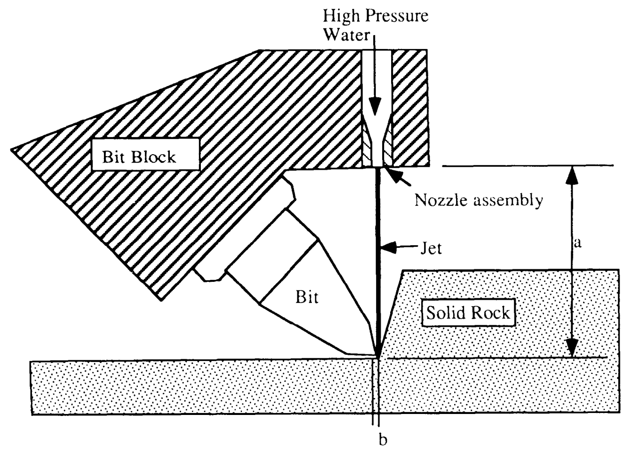


Fig. 22.3.13. Action of a drag bit moving over rock.



Where possible the standoff distance a should be less than 200 jet diameters
The distance b that the jet leads the bit tip should never be more than 1.5 mm.

Fig. 22.3.14. Geometry for maximum effect from a water jet assisting a drag bit. Action of a drag bit moving over rock.

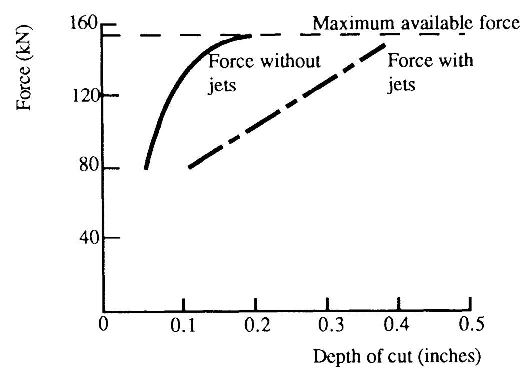


Fig. 22.3.15. Improvement achieved by Hood in adding water jets to a drag bit in cutting norite. Conversion factors: 1 in. = 25.4 mm, 1 lbf = 4.448 N.

jets were tried, the boom stabilized significantly and reduced pick wear was observed, but excessive vibration in the boom was still noted.

When water jets were tried with radial picks, the cutting performance of the machine was vastly improved. With a flow of 1 gpm (4 L/min) per nozzle, it was found that cutting rates were increased 2.0 to 2.5 times. The results were sufficiently promising that a short tunnel was driven with this combination of cutters and jets. Optimum cutting was found to occur where the leading nine picks of the head were assisted by a jet at 10,000 psi (70 MPa).

Under normal conditions, the Dosco machine would be restricted to operating in rocks with a compressive strength below 10,000 psi (70 MPa). A much heavier machine, costing about $650,000, would be needed to mechanically cut the test rock. Use of a lighter machine would bring the equipment cost down to around $250,000 for the roadheader and $150,000 for the water-jet modification. In addition, the removal of the spark and dust hazards and the smaller envelope of the machine make its use more universal.

22.3.5.4 Commercialization of Technology

Both British roadheader manufacturers, Anderson Strathclyde and Dosco, worked with the NCB to develop the new technology as improvements on their basic product lines (Barham and Buchanan, 1987). Machines were developed which showed significant increases in performance over existing equip-

Fig. 22.3.16. Water-jet-assisted roadheader (*courtesy:* Dosco).

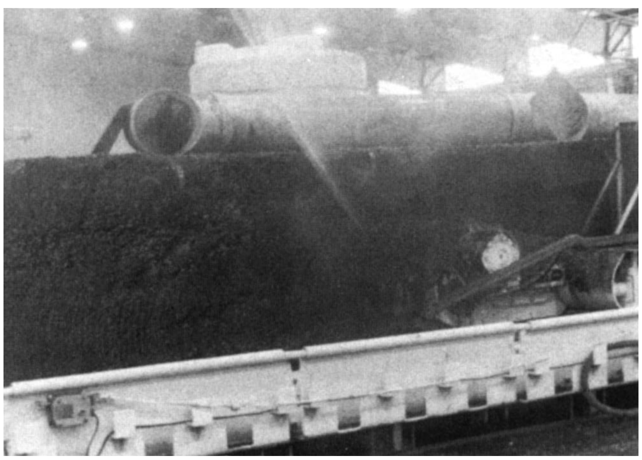


Fig. 22.3.17. Water-jet-assisted shearer.

ment (Fig. 22.3.16). At Sutton Manor Colliery water jets at 10,000 psi (70 MPa) helped drive a 1150-ft (350-m) entry in rock that had strengths ranging to 24,000 psi (165 MPa). The tunnel-driving rate was increased by 50% while dust generation was reduced by 70%. Bit wear was reduced to roughly 25% of that without water jets.

Over 70 water-jet machines had been sold by these two companies by mid-1987. The average cost of a retrofit package for Dosco machines had fallen to $75,000, and the technology has now been incorporated into equipment available from manufacturers in Germany, Austria, and Japan.

22.3.5.5 Development of a Water-jet-assisted Shearer

With the advantages of water-jet-assisted rock cutting established, a similar approach is being taken to coal excavators. Soviet scientists (Ekber and Kuzmich, 1980), were the first to apply water jets to assist plows and trepanners. Recent emphasis has been to add water jets to shearer picks (Fig. 22.3.17).

The USBM has contracted with Eickhoff, to develop a water-jet-assisted shearer (Neinhaus et al., 1986). In the United Kingdom, Minnovation has placed a small pump inside the shearer drum to overcome the problems associated with phasing the water jets on the picks (Tuke, 1988). This is required, since it would be unproductive and hazardous to have the jets operating when the picks are not in contact with the coal. A six-month trial of this system at Bolsover colliery, using water jets at a pressure of 22,800 psi (157 MPa), indicated the following advantages to the system:

1. 6.4% improvement in shearer efficiency.
2. 16% reduction in haulage force.
3. 18% improvement in mechanical efficiency and an improvement of more than 200% in operational lifetime of the cutting arm.
4. Pick life was extended by a factor of ten as a minimum, and significantly less wear was found in the transmission components between the shearer motor and the cutting drum.
5. 40% reduction in water consumption on the shearer for dust suppression.
6. Significantly smoother cutting characteristics.

An alternative system of water delivery was developed by British Coal using an Anderson Strathclyde machine and tested at Golborne colliery (Barham and Buchanan, 1987). When 9500-psi (65-MPa) water jets were added, the power required for a given performance was reduced by 40%. The use of the water-jet-assisted picks reduced the number of picks needed on the drum, lowering fines by some 20%. Dust was reduced by 33%, using one-third less water than conventionally used for dust suppression.

The joint USBM/Eickhoff machine underwent trials at the Auguste Victoria Colliery in West Germany. The shearer was supplied with a flow of 30 gpm (120 L/min) of water at 10,000 psi (70 MPa). The major difference from the British trials is that in the German machine, point-attack picks were used, in contrast to the radial picks used on the British machines.

22.3.6 Practical Application of Jet Assistance

There has been a significant amount of research on the use of water jets to assist mechanical cutters. There have also been enough demonstration projects that have provided favorable results to show that the technology can work. Unfortunately, as with many things, it is not quite as easy to make the system work as might appear. The technology suffers from the perception that it is simple to apply. In fact, there are certain small but critical factors that must be addressed if the application is to provide an advantage. Although these have been discussed in some detail above, it is perhaps pertinent to review them from the point of view of an applications engineer. These include the stages that must be gone through, potential costs involved, and some idea of the productivity that may be anticipated as a result.

The fundamental object of the mining tool is to break rock as cheaply and expeditiously as possible. The major tool for doing this is a series of drag bits fitted on a drum, which, when rotated cut a series of grooves in the rock. By the proper design of the pattern, these grooves overlap and allow extraction of the entire rock surface. The addition of water jets to the process will not, at this stage, change this basic approach to rock removal. What then are the potential gains that an engineer should look for?

When the water jets are properly applied to the cutting zones around the individual rock bits, they enhance the cutting performance of each bit. The improvement comes in three ways:

1. The bit can cut through harder rock and with the same amount of power.
2. The bit can cut to a greater depth in rock of the same strength with the same amount of power.
3. The bit will last longer.

Potential cost savings will arise only if the machine is used to take advantage of these possible gains. For example, rather than purchasing a larger machine to cope with a harder rock, the modification to a smaller machine would provide a significant cost savings. Because it is early in the development of equipment, there is still a significant amount of overhead cost associated with the addition of water jets to cutting drums. It should be noted, however, that pumps are available that deliver 75 gpm (20 L/min) at 5000 psi (35 MPa) for under $5000. Experimental evidence would appear to indicate that this pressure is sufficient to enhance the performance of cutting picks in most rock types. The relative cost benefit of using a smaller machine can be as high as $250,000, as discussed in 22.3.5.3 above. In most mines, however, the volume of rock encountered that the normal machine will have difficulty in cutting may be significantly less than 20%. In this circumstance, the volume of rock to be removed may not justify the equipment modification by itself.

However, if existing equipment is already cutting the rock adequately, then it may be difficult to see much direct gain from the use of water jets. This is because the jets enhance the ability of each individual pick to cut, but the pattern of the picks will probably need to be modified to take advantage of a deeper cutting distance. This has not been common practice. The increased life is not likely by itself to provide sufficient cost savings to justify the modification to the cutting head.

Simply adding a water-jet stream to the vicinity of the path of the rock bit will not provide much improvement in performance. The jet must carry its energy to within 0.06 in. (1.5 mm) of the leading edge of the cutting tool and must be directed along the plane of the tool edge. To get the jet to this point with most of its energy retained, is not always as simple as it sounds. For example, a 60-psi (400-MPa) water jet issuing from a 0.12-in. (0.3-mm) orifice will likely be totally dissipated within 4 in. (100 mm) of the nozzle. Unfortunately, because of the cutting head designs now in use, it may be difficult to place the jet nozzle much closer than this distance from the rock.

Jet performance can be improved in four ways:

1. The pressure can be lowered.
2. The diameter of the jet can be increased.
3. The flow conditions into the nozzle can be improved.
4. Additives might be added to the water.

Of these options, the first is unlikely to be desirable. A minimum pressure is required to help cut the rock, or improvement rapidly disappears. Increasing the nozzle diameter provides a number of benefits in terms of jet cohesion and enhanced cutting ability, but carries with it significant problems in terms of the resulting working condition (jet cohesion disappears rapidly when the machine has to work underwater). Phasing the water-jet system so that the water flows only when the pick is in contact with the rock can improve this situation significantly. Valves have now been developed so that this is a viable option. However, the pumps normally used with the water-jet assistance have not allowed much increase in flow volume. Working conditions in some mines also mitigate against the use of larger volumes of water.

Improving the flow path into the jet nozzle may require a redesign of the cutting head. Such a redesign may well be necessary to take full advantage of the changed cutting conditions for each bit. There should be a lead-in section of straight pipe, at least 3 in. (75 mm) long and preferably 4 in. (100 mm) long. Where this is not possible, then flow straighteners should be placed behind the nozzle to enhance jet cohesion. Manifold designs to distribute the water should be arranged to minimize turbulence and to ensure that the water is divided before it is accelerated. The flow path should be one of constantly decreasing cross sections, since significant energy losses can occur any time the flow path expands.

Maintenance costs for this system are likely to be a little higher than with conventional cutting heads. If individual picks wear too long, for example, then the jet would not cut close enough to the tip to provide any effective support for the cutting process. Nozzles can also wear, and visual evaluation of jet structure is usually not an adequate measure of performance. Running a wooden 2×4-in. plank across the pick head should show some penetration of the wood, as an example of a simple test to check on nozzle quality. Nozzles are relatively inexpensive (less than $30 each) and, with normal clean water, should have a long life. But, it should be remembered that, as with the picks themselves, nozzles are wear items and will need to be replaced. The cost may, however, be more than offset from the increased life of the bits. Further, the increased depth of cut achievable will allow an increase in the separation of the picks so that less will be required on the head. This change, however, needs to be made with some caution. Although the use of water jets will reduce the vibrations caused by individual picks, the increased spacing and change in pattern might, if improperly done, induce other waveforms into the drive train that will lead to premature equipment failure from fatigue.

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Chapter 22.4
HYDRAULIC MINING: BOREHOLE SLURRYING

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Borehole mining, also known as slurry mining, is a process in which a tool incorporating a water-jet cutting system and a down-hole slurry pumping system are used to mine minerals through a single borehole drilled from the surface to the buried mineralized rock. Water jets from the mining tool erode the ore to form a slurry. The slurry flows into the inlet of a pump at the base of the tool. The material is lifted to the surface in a form suitable for pipeline transfer to a mill or processing plant.

As defined by this paper, borehole mining appears to be a likely prospect for the near future. It offers a number of important advantages over conventional open pit and underground mining methods, and it can access mineral deposits that presently are not mined because of technical or economic difficulties. This method can achieve essentially immediate production because there is no need to drive openings to and in a proved ore body to prepare it for mining; in contrast, conventional mining methods require from three to five years before production and return on investment can be expected. The fragmentation and transportation systems are incorporated into a single machine that is remotely operated from the surface by a two- or three-person crew, thus eliminating health and safety problems inherent to underground mining. The equipment is amenable to automated operations.

The environmental disturbance is minimal and short term; no overburden is removed, and subsidence can be avoided.

22.4.1 BOREHOLE MINING TOOLS

The earliest patent for a tool that used a water jet to fragment rock adjacent to a borehole and a down-hole slurry pump to lift the broken ore to the surface was issued to Clayton in 1932.

Patents on similar borehole-mining tools were issued to Aston (1950), Quick (1955), Fly (1964), Pfefferle (1969), Wennenborg (1973), Archibald (1974), and Brunelle (1977). Fly's apparatus was built and used to excavate sandstones, limestones, and shales to a maximum depth of 350 ft (100 m). Mining rates of 1 yd^3/min (0.8 m^3/min) were achieved, and cavities were excavated to a lateral distance of 30 ft (9.1 m) from the borehole. The apparatus had two sidewall nozzles operated at 800 psi (5.52 MPa) and 200 gpm (12.6 L/s) each to form the water jets. The slurry was caused to flow into the intake of a down-hole jet pump, which hoisted it to the surface. The jet pump was operated at about 800 psi (5.52 MPa) and 500 gpm (31.6 L/s). Jets were also formed by forcing water through the water courses of a tricone rock bit attached to the base of the tool. These jets kept the slurry in suspension so that it could be taken into the down-hole slurry pump. This tool used a single, pressurized water supply to operate the sidewall jets, the pump, and the tricone jets.

The apparatus described in the Wennenborg patent was built by the FMC Corp. and tested in phosphate ore in eastern North Carolina. This device uses a high-volume, low-pressure water jet to slurrify the ore and an eductor to lift the slurry to the surface. Its most novel aspect is that it provides a method for drilling into, as well as mining, a deposit of granular ore. All previous mining tools required a predrilled, cased borehole. The Wennenborg device is designed for mining unconsolidated, easily drilled sediments, such as North Carolina phosphates.

The apparatus described in the Archibald patent was built by Marconaflo, Inc., and used to mine uraniferous sandstones and tar sands on an experimental basis. The jet-cutting unit consists of a single nozzle and high-pressure piping that rides on a vertical rail attached to the main body of the device. This rail allows the nozzle to move independently of the slurry pump; the nozzle could be slid up and down as well as rotated 180° about a vertical axis. The vertical motion allows cutting to occur at various horizons without lifting or dropping the entire device, and it lets the intake of the slurry pump to be cleared of blockages by the cutting jet. The cutting jet is operated at 400 to 500 psi (2.76 to 3.44 MPa) and 150 to 170 gpm (9.4 to 10.7 L/s).

The slurry pumping system contains a pump mechanically driven from the surface and 20-ft (6-m) long sections containing a drive shaft and slurry conduits. Both a Moyno pump and a centrifugal pump have been used to pump slurry in the Marconaflo apparatus. The device operated in a 30-in. (762-mm) diameter borehole and produced 30 to 45% solids in the slurry. It was tested successfully by mining a uraniferous sandstone from a roll-front deposit in the Gas Hills of Wyoming from a depth of 180 ft (54.9 m) and by mining tar sands from a depth of 350 ft (107 m) in the McKittrick oilfield near Bakersfield, CA.

In 1974, the US Bureau of Mines (USBM) contracted with Flow Industries, Inc., to design, fabricate, and test a borehole mining tool. The USBM equipment has an eductor for a down-hole slurry pump, whereas mechanically driven slurry pumps were used in the Marconaflo equipment. It contained separate conduits for the eductor driven water and the cutting-jet water, contrasted with the FMC and Fly systems that used a single conduit.

The Bureau's system, shown schematically in Fig. 22.4.1, is composed of the mining tool suspended from a crane in a 16-in. (406-mm) diameter cased borehole. The mining tool generates a high-velocity water jet that erodes and slurrifles ore. The slurry is drawn into the inlet of an eductor and lifted to the surface where it is metered and deposited into a slurry discharge tank (Fig. 22.4.2). The ore settles in the tank while the water overflows into a pond, which is the source of water for pumps that supply the cutting jet and the eductor.

The mining tool, which is operated while suspended from a crane (Fig. 22.4.3), is in the form of a 12-in. (305-mm) diameter cylinder capped with a three-passage swivel. The cylinder is composed of a kelly section, a series of standard sections, and a mining section.

The cutaway view of the three-passage swivel is shown in Fig. 22.4.4. The outer part is stationary and is supported by a crane. The swivel core rotates relative to the exterior while simultaneously passing the three pressurized streams: the water supply to the cutting nozzle, the drive water to the eductor pump, and the slurry output. The swivel is connected to a kelly section by eight bolts. The kelly section is a cylinder 22 ft (6.71 m) long and 12 in. (305 mm) in diameter with two 0.75-in. (19-mm) webs welded along its length. The webs key into a rotary turntable, thereby transmitting torque to the mining tool.

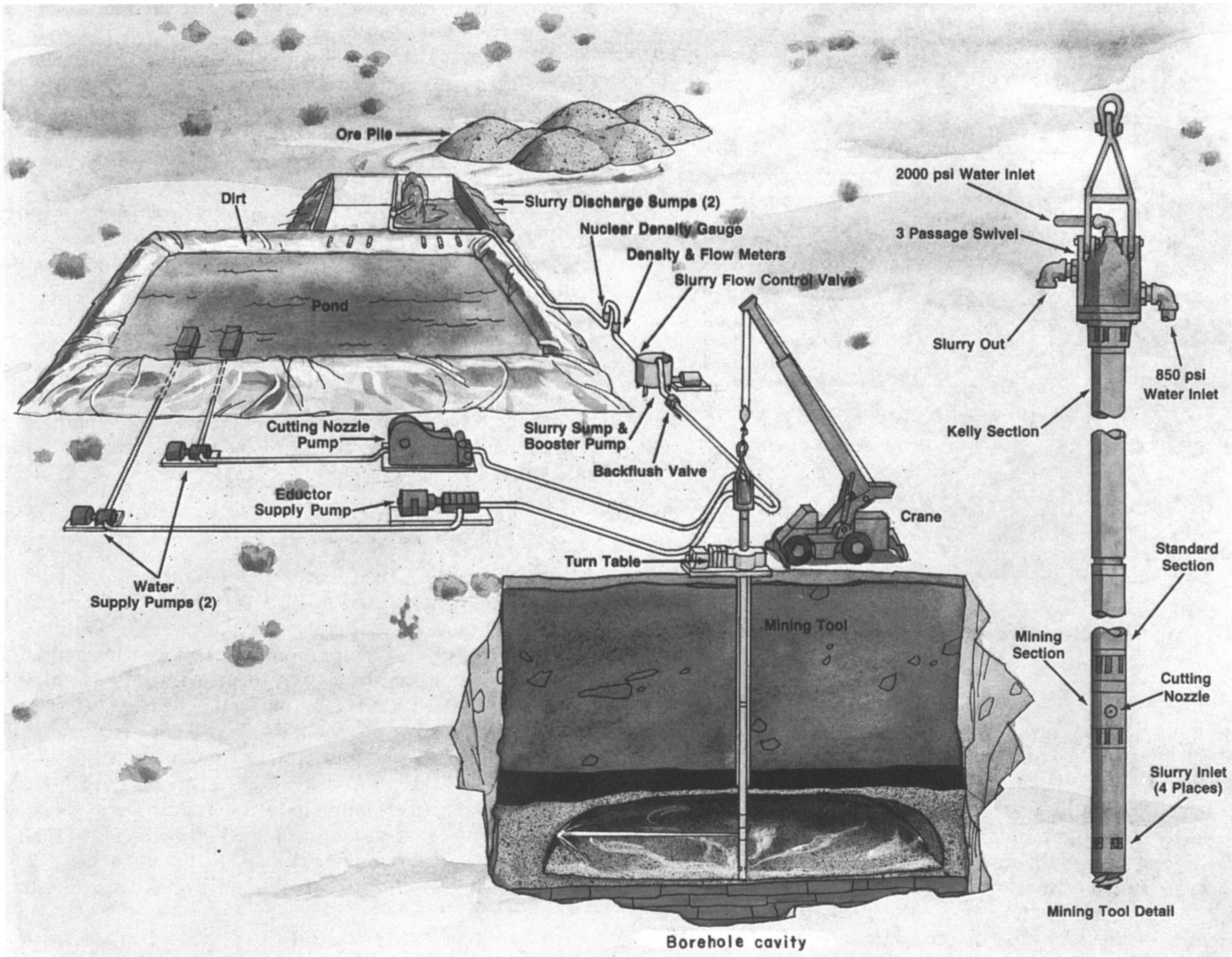

Fig. 22.4.1. US Bureau of Mines borehole-mining system. Conversion factor: 1 psi = 6.895 kPa.

Fig. 22.4.2. Slurry discharge.

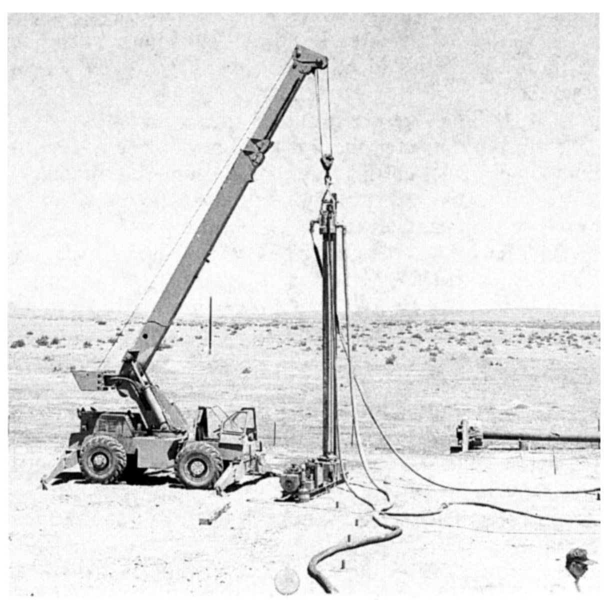

Fig. 22.4.3. Borehole-mining tool suspended from crane.

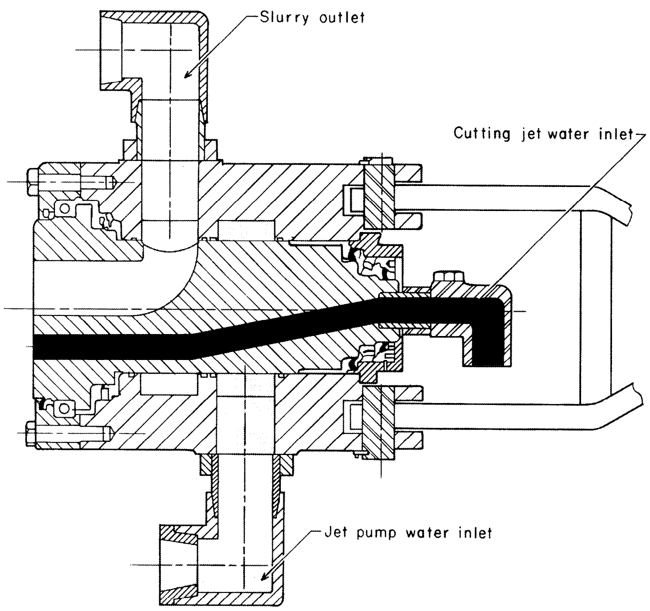

Fig. 22.4.4. Cutaway view of three-passage swivel.

This turntable is driven by an hydraulic motor and governed by hydraulic controls and limit switches, which allows for rotary speeds of 0 to 20 rpm and for automatic oscillation for any interval from 0 to 360°.

The internal configuration of the kelly section is shown in Fig. 22.4.5. The 12-in. (305mm) diameter cylinder houses a 4-in. (102-mm) diameter slurry discharge pipe and a 2-in. (51-mm) diameter supply pipe for the cutting jet. The space inside the cylinder not occupied by pipes is the conduit for the jet-pump drive water. The end of the kelly section is covered by a flange that provides for the interconnection of the conduits in adjacent sections: two circular spaces for the jet-cutting and slurry-output pipes, the two kidney-shaped spaces for the eductor drive water. The kelly section is connected to a string of standard sections, each 20 ft (6 m) long and 12 in. (305 mm) in diameter, that provides the length to reach the ore. The internal configuration of these sections is identical to that of the kelly section just described.

The mining tool is terminated with a mining section 12 in. (305 mm) in diameter and 6 ft (1.83 m) long (Fig. 22.4.6). It is composed of jet-cutting and slurry-pumping modules (Fig. 22.4.7). The jet-cutting module contains a flow-turn and nozzle device that maximizes the effective cutting length of the water jet. This device was designed by TRW Defense and Space System Group under a USBM contract. The nozzle profile consists of a smooth, transitory curve from nozzle entrance to the outlet orifice. Upstream of the nozzle is a short-turn elbow with flow-splitting plates to guide the flow around the elbow with reduced flow disturbances.

The slurry-pumping module contains the eductor (jet pump) and a conical tool. The jet pump has a nozzle that generates the high-velocity water jet. The venturi effect caused by the discharge of the jet draws slurry into the pump through screened intake ports. The slurry mixes with the drive water in the throat of the jet pump and enters a diffuser where it acquires the pressure to lift it to the surface. The intake ports are screened to prevent oversize material from blocking the pump. Should the material block the inlet, a fast-acting valve ("back-flush" valve) is closed in the slurry discharge line at the surface, forcing the jet-pump drive water to flow out of the pump intake and clear away the blockage. The conical tool, which is bolted to the base of the mining section, facilitates entry into cuttings that fill the void caused when the tool is raised. A 50-gpm (3-L/s) water jet issues downward from the center of the auger and agitates the cuttings below, thereby helping the mining section enter the muck pile.

Flow Industries, Inc., independently produced a borehole mining tool based on the USBM design, but with some notable design changes. The Bureau of Mines and Flow Industries, Inc. products are similar in that both are composed of 20-ft (6.1-m) lengths of cylinders connected by flanges, and the slurry pumps have the same design. They differ in that the cutting nozzle of the Flow product is controlled independently from the remainder of the tool, similar to the Marconaflo tool. This permits water-jet cutting to be performed anywhere along the length of the borehole while the pump is low in the sump, where the slurry density is highest.

22.4.2 COAL MINING

Flow Industries, Inc., under a USBM contract, conducted borehole mining operations in 1975-76 at a site 3 miles (4.8 km) south of Wilkeson, WA. This site contained a seam of bituminous coal 17.8 ft (5.4 m) thick, dipping at 42°. Three vertical boreholes (two shallow and one deep) were drilled through the dipping coal seam and cased to the hanging wall. The two shallow boreholes (25 and 35 ft, or 7.6 and 10.7 m) were used to conduct preliminary tests designed to optimize mining procedures to be followed during a 4-hour production test in the deep (88-ft, or 26.8-m) borehole. The preliminary test results were as follows:

1. The cutting jet was more efficient at cutting coal than the slurry system was at removing the coal from the borehole. Thus, the maximum mining rate was limited by the slurry-pumping rate.
2. A cutting radius of 10 ft (3 m) was attainable with the 4500-psi (31-MPa), 100-gpm (6-L/s) jets.
3. Shale tended to clog the jet pump because it breaks into acicular particles that lodge between the nozzle and the sidewall of the jet pump.

A production rate test was conducted in the deep borehole, which was lined with 16-in. (406-mm) steel casing. The parameters of the cutting jet were similar to those used in the preliminary test except that a single, high-discharge jet (4500 psi or 31 MPa, and 200 gpm or 12.6 L/s) was used to increase the effective cutting range to 15 ft (4.5 m).

This test yielded a production rate of 8 tph (7t/h). This rate, along with the fact that no mechanical failures of the mining tool occurred during the field program, indicates that it is technically feasible to mine coal remotely from the surface through a borehole. It was concluded, however, that the test production rate was too low for commercial feasibility.

Flow Industries, Inc., conducted a further study for the US Department of Energy on borehole mining of pitching coal seams. This study proposed an optimum borehole mining plan and system for mining coal seams less than 10 ft (3 m) thick that pitch more than 25°. This plan utilized a double-drill system. This method uses a larger vertical hole for the slurry pump and a smaller slant hole for the cutting tool. This slanted cutting hole is drilled down-dip on the footwall of the seam to intersect the vertical hole at maximum depth. The cuttings drain down the cutting hole to its intersection with the pumping hole, from which they are pumped to the surface.

According to Boyce, the borehole mining method is not economically competitive with other surface mining methods for

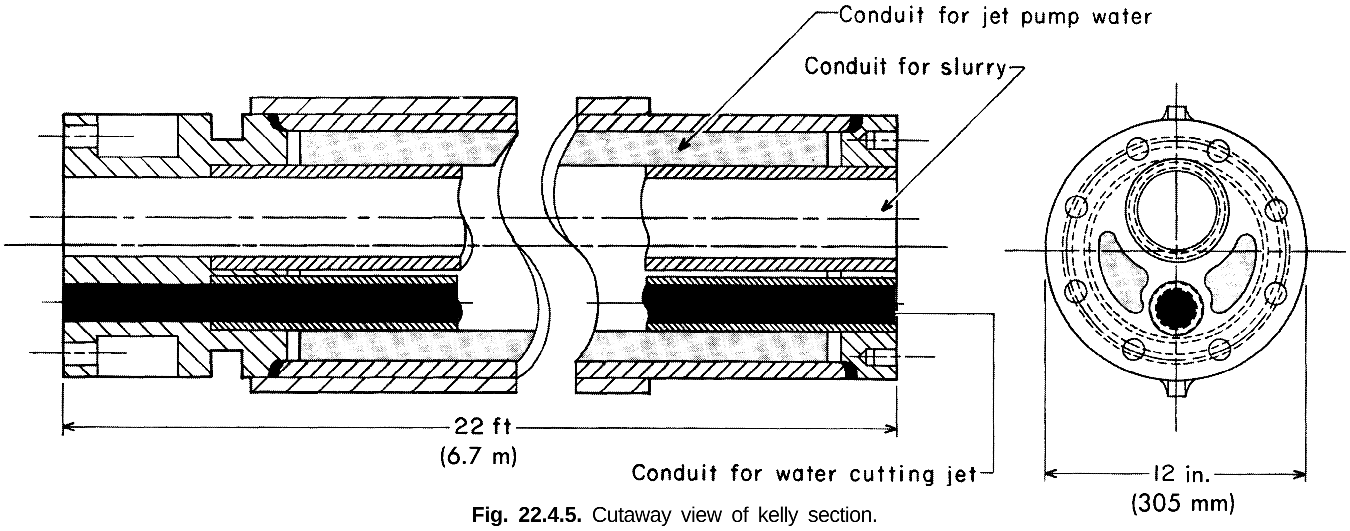

Fig. 22.4.5. Cutaway view of kelly section.

Fig. 22.4.6. Mining section.

horizontal coal seams with the same ratio of overburden to coal. It is generally agreed that borehole coal mining methods are applicable mainly to steeply dipping coal beds.

22.4.3 URANIUM MINING

Uranium sands are considered to be a likely prospect for borehole mining because (1) the ore has a high unit value, (2) the sandstones can be cut by low-pressure (1000- to 3000-psi or 6.90- to 20.69-MPa) water jets, and (3) many deposits are shallow, small, irregularly shaped, and isolated; these deposits cannot be mined by conventional methods, but are amenable to the selective capabilities of the borehole system.

Marconaflo, Inc. and Rocky Mountain Energy (RME) conducted tests from 1973 to 1975 in the Powder River Basin and the Gas Hills of Wyoming using the equipment described above. A total of about 700 tons (630 t) was mined, but problems were encountered with the sand plugging and severely wearing the down-hole Moyno slurry pump used in these tests.

The USBM cooperated with RME at its Nine-Mile Lake site, Natrona County, WY, in a borehole-mining test in 1977 and 1978. RME prepared the site, drilled a water supply well, constructed a pond and lined it with polyethylene, and drilled three 16-in.-ID (406-mm) cased boreholes to a depth of 100 ft (30.5 m) into the Teapot sandstone ore body. Flow Industries, Inc., under contract to the Bureau of Mines, modified the tool used for coal at the Wilkeson, WA, site and conducted the sandstone mining operations. A shallow deposit at Nine-Mile Lake was chosen for the test because the slurry pump is limited to differential lifts of 200 ft (61 m). The modifications included fitting of the mining tool with a turning-vane nozzle ensemble designed to pass 300 gpm (18.9 L/s) at 2000 psi (13.8 MPa), the flow conditions chosen for efficient erosion of the Teapot sandstone.

During mining operations, approximately 940 tons (850 t) of ore were mined from depths of 75 to 100 ft (22.9 to 30.5 m) at an average rate of 8 tph (7 t/h) from standoff distances as great as 25 ft (7.6 m). Slurry densities ranged from 0 to 46% by weight with an average of 700 determinations being 7.2%. The tests also showed the following:

1. The average jet-cutting rate was about 16 tph (14.5 t/h) at 520 hp (388 kW). The slurry pump normally works at a lower rate because the tool moves vertically as one piece, thereby lifting the pump out of the slurry sump during part of the mining cycle. The pumping rate could be made equal to the jet-cutting rate if the cutting jet could be moved independently from the slurry pump.
2. The optimum jet-cutting traverse rate across the sandstone was between 40 and 80 in./sec (1016 mm/s and 2032 mm/s).
3. The jet-cutting rate was proportional to the horsepower of the cutting jet.

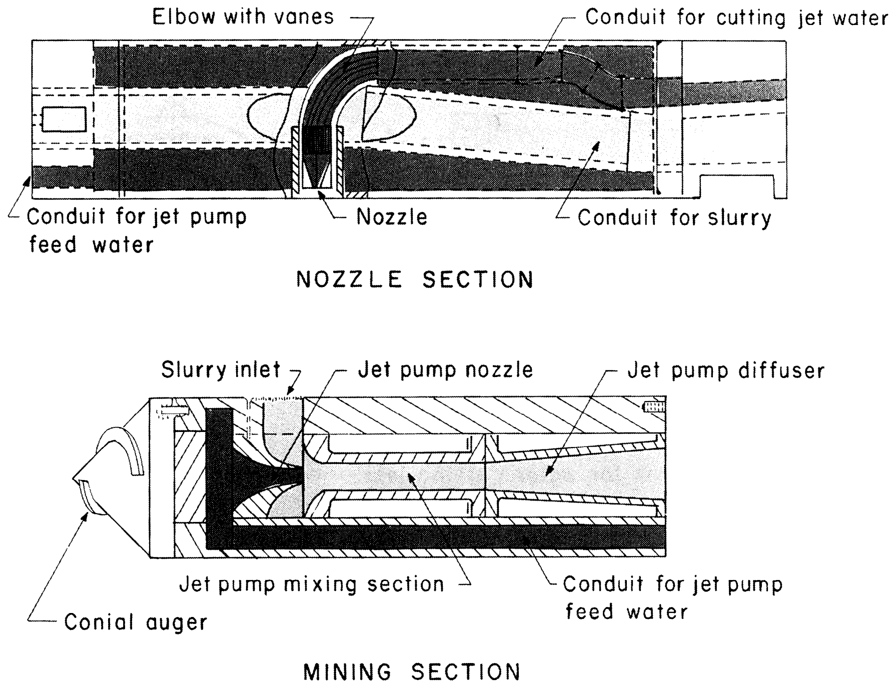

Fig. 22.4.7. Internal configuration of mining section.

Fig. 22.4.8. Cavity produced during borehole mining.

A photographic survey of the borehole cavities created in the Teapot sandstone ore body was taken using equipment developed for the purpose by the Bureau of Mines. Fig. 22.4.8 shows the cavity created in one of the boreholes at Nine-Mile Lake. The white, 2-in. (50.8-mm) diameter PVC pipes in the foreground are placed in monitor holes drilled 10 and 20 ft (3.05 and 6.10 m) from the center of the borehole. A 1-in. (25.4-mm) diameter steel pipe 26 ft (7.93 m) from the borehole is shown in the background. This photographic survey showed that roof failure was confined to a 7-ft (2.1-m) radius from the center of the borehole. Presumably, this indicates that rock within this radius was damaged during drilling.

22.4.4 OIL-SAND MINING

The possibility of mining oil in the 383 known shallow US oilfields has become a matter of interest because of the energy crisis of the 1970s. Oil could be produced from these shallow fields by surface mining methods. However, open pit mining of oil sands could be expected to meet with environmental objections including the following: (1) disruption of the surface, (2) increased air pollution from volatile hydrocarbons uncovered in the open pit, (3) accumulation of waste rock piles, (4) accumulation of tailings, and (5) surface water pollution. Clearly, added costs for restoration and reclamation would be incurred.

Borehole mining offers an alternative method of mining the oil sands with minimal disturbance to the environment because no overburden is removed, no waste rock piles are generated, tailings can be backfilled into the borehole cavity, no surface streams are polluted because a closed-loop water system is employed, and surface subsidence can be avoided by backfilling the mined cavities. Thus borehole mining appears to be more acceptable on an environmental basis than does surface mining. The greatest concern would be possible degradation of groundwater quality.

In 1975, Marconaflo, Inc. conducted borehole mining tests in 300- to 380-ft (91.4- to 115.8-m) deep oil sands at McKittrick, CA. The equipment was that used in uranium mining with the exception that a down-hole centrifugal slurry pump replaced the Moyno pump. During this program, approximately 2000 dry tons (1810 t) of oil sands were mined.

In 1979, the USBM and Flow Industries, Inc., performed borehole mining tests on a site in Kern County, near Taft, CA, in the Midway Sunset oil field. This test demonstrated that the borehole mining technique was an environmentally compatible method for mining oil sands. The experiment measured production rate along with the environmental impacts.

Mining was conducted from July 25 to Aug. 24, 1979. During the operation, 994 tons (902 t) of oil sands were extracted from two holes. The mining rate ranged from 0 to 45 tph (0 to 41 t/h). Typical operating parameters are listed in Table 22.4.1. A range of values is given when the parameter varied significantly.

One major complication was encountered during the operation. The borehole became filled with rocks that accumulated at the base of the mining tool because they were prevented from

Table 22.4.1. Operating Parameters for Oil-Sand Mining, Kern County, CA

		Typical Value	Range
Cutting jet:			
Pressure	psi	400 (2.75 MPa)	100–2500 (0.69–17.2 MPa)
Flow rate	gpm	300 (18.9 L/s)	100–500 (6.3–31.6 L/s)
Hydraulic power	hp	50 (37.3 kW)	10–700 (7.46–522.2 kW)
Nozzle diameter	in.	0.60 (15.2 mm)	0.62–0.75 (15.7–19.1 mm)
Line diameter	in.	1.70 (43.2 mm)	NA
Rotation rate	rpm	10	4–15 (4–15 rpm)
Traverse rate	in./sec	60 (1.52 m/s)	1–120 (0.025–3.05 m/s)
Vertical cutting increment	in.	2 (50.8 mm)	NA
Angle of cutting arc	deg	180 (180°)	0–360 (0°–306°)
Depth	ft	130 (39.6 m)	110–150 (33.5–45.7 m)
Jet pump:			
Pressure	psi	1000 (6.89 MPa)	450–1500 (3.10–10.3 MPa)
Flow rate	gpm	500 (31.6 L/s)	350–650 (22.1–40.9 L/s)
Agitation jet flow rate	gpm	90 (5.7 L/s)	60–110 (3.8–6.95 L/s)
Hydraulic power	hp	300 (224 kW)	100–600 (74.6–448 kW)
Nozzle diameter	in.	0.70 (17.8 mm)	NA
Agitation jet diameter	in.	0.188 (4.78 mm)	NA
Throat diameter	in.	2.5 (63.5 mm)	2.5–2.9 (63.5–73.7 mm)
Nozzle line diameter, effective	in.	2.5 (63.5 mm)	NA
Secondary flow:			
Rate	gpm	400 (25.3 L/s)	300–600 (19.0–38.0 L/s)
Solids	wt %	15	0–35
Specific gravity		1.1	1.0–1.3
Slurry flow:			
Rate	gpm	800 (50.6 L/s)	600–1100 (37.9–69.5 L/s)
Line diameter	in.	3.75 (95.25 mm)	NA
Solids	wt %	7	0–18
Specific gravity		1.05	1.0–1.15
Mining rate	tph	15 (13.6 t/h)	0–45 (0–40.8 t/h)

entering the slurry pump by a screen over the inlet. The tool was unable to penetrate this pile of rocks. Mining had to be terminated in each borehole when the pile got so high that the jet could not be lowered below the casing (110 ft or 33.5 m). The addition of a crusher to the mining tool would prevent the rocks from obstructing the mining tool.

Ground subsidence is possible in borehole mining operations. To evaluate it, a series of surveys collected information on changes in ground elevation at the site. Surveys were performed to obtain baseline elevations before mining, weekly during the project, and 30 days after the mining stopped. No significant short-term subsidence was detected.

22.4.5 PHOSPHATE MINING

Magnet Cove Barium Co. used a waterjet borehole mining tool to extract 8000 tons (7260 t) of ore from four boreholes near the Pungo River in North Carolina in 1964. FMC Corp. conducted borehole phosphate mining tests in Beauford County, NC, in the mid-1970s. The FMC borehole mining tool, incorporating a drilling tool and a mining tool in a single unit, was used in four test holes. The borehole mining equipment operated from a barge.

The USBM and Flow Industries, Inc., in cooperation with Agrico Mining Co., conducted borehole phosphate mining tests in St. Johns County, near St. Augustine, FL, in 1980. The purpose of the test was to determine if phosphate ore could be mined economically in an environmentally compatible manner with the Bureau's borehole mining system.

Between Apr. and Aug. 1980, 1700 tons (1540 t) of phosphate ore were produced (Fig. 22.4.9) from three boreholes that ranged from 232 to 253 ft (70.7 to 77.1 m) deep. Mining in the first hole was conducted to determine the feasibility of mining with the borehole filled with water. This wave experiment yielded 860 tons (780 t) at an average rate of 36 tph (33 t/h) while cutting with a submerged jet (Fig. 22.4.10) in a 360° arc. The specifications for water-jet mining in borehole 1 are shown in Table 22.4.2.

When the water was pumped from the cavity, the immediate roof failed, indicating that the water pressure had supported the roof. Although the rock in the ceiling of the cavity failed, no surface subsidence was noted. However, no further mining could be performed in this hole. This experiment did indicate that borehole phosphate mining in a submerged mode is technically feasible.

Attempts to mine in an air-filled cavity were made in borehole 2 (Fig. 22.4.11) where mining was confined to a 30° arc and a 330° pillar supported the roof. However, a roof failure occurred after 300 tons (272 t) of ore had been produced. From this test, it was concluded that (1) the roof rock did not have sufficient strength to permit mining in an air environment, and (2) any future mining would require that the cavity be filled with water.

A third borehole tested an "air-shielding concept" designed to combine the need to have flooded conditions and the advantages of mining in air. Under this concept, the water jet was in a shroud of compressed air; this allowed cutting at longer standoff distances while retaining the roof support and increased pumping capability gained by working under a hydrostatic head of water. The air-shielded water jet issued from the usual borehole cutting

Fig. 22.4.9. Phosphate ore deposited at outlet of mining tool.

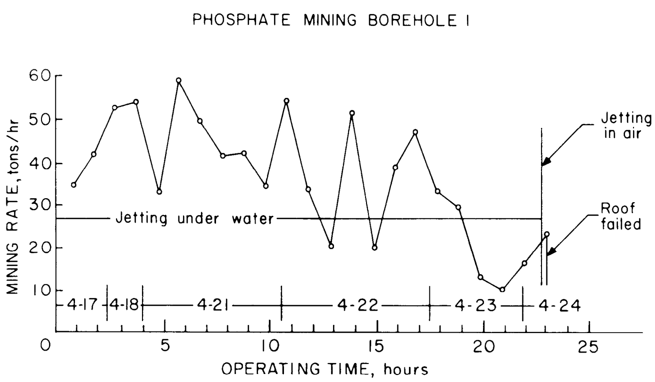

Fig. 22.4.10. Phosphate production in borehole 1. Conversion factor: 1 ton = 0.9072 t.

nozzle with an annulus for compressed air around it. Compressed air was supplied through conduits in the borehole mining tool connected to an air compressor at the surface. The water-jet specifications for mining in borehole 3 are shown in Table 22.4.3.

A total of 430 tons (390 t) was mined in this borehole without actuating the air shield in order to establish the baseline solids production (Fig. 22.4.12). After 23 hours of mining, the solids content of the slurry began to decrease, indicating that the submerged jet had reached its maximum effective radius. At this point, the air-shield was activated, and an additional 137 tons (124 t) was mined at the rate of 25 tph (23 t/h). The cavity radius was about 18 ft (5.5 m), and no roof failure had occurred when the mining stopped. This experiment indicated that phosphate can be mined effectively in a flooded cavity and that

Table 22.4.2. Parameters and Specifications for Mining in Wave Experiment Borehole 1

Parameter		Specifications
Cutting jet:		
Pressure	psi	500–2000 (3.45–13.79 MPa)
Flow rate	gpm	500–750 (31.6–47.3 L/s)
Diameter	in.	0.475 and 0.966 (12.1–24.5 mm)
Jet pump:		
Pressure	psi	700–1500 (4.83–10.3 MPa)
Flow rate	gpm	400–700 (25.3–44.2 L/s)
Nozzle diameter	in.	0.68 and 0.80 (17.3–20.3 mm)
Throat diameter	in.	2.00 and 2.25 (50.8–57.2 mm)
Turntable speed	rpm	2–15 (2–15 rpm)
Mining arc	deg	360 (360°)
Mining depth	ft	232–253 (70.7–77.1 m)
Vertical increment	in.	2–6 (50.8–152.4 mm)

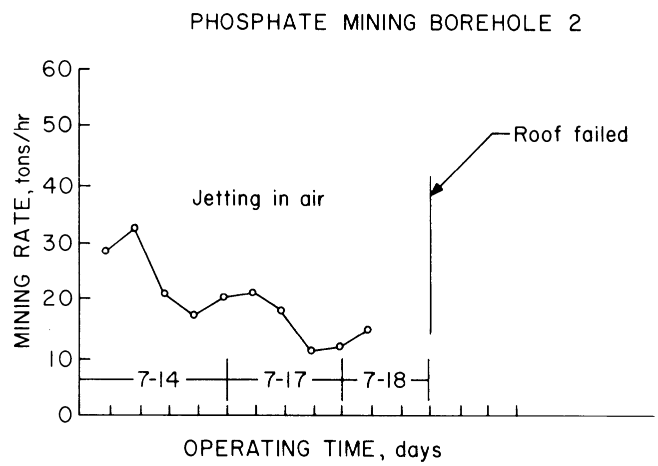

Fig. 22.4.11. Phosphate production in borehole 2. Conversion factor: 1 ton = 0.9072 t.

Table 22.4.3. Parameters and Specifications for Mining in Borehole 3

Parameter		Specifications
Cutting jet:		
Pressure	psi	1000–1900 (6.89–13.1 MPa)
Flow rate	gpm	423–499 (26.7–31.5 L/s)
Diameter	in.	1.00 (25.4 mm)
Air-shield:		
Pressure	psi	250 (1.72 MPa)
Flow rate	cfm	150 std (70.8 L/s)
Nozzle opening	in.	0.030 (0.76 mm)
Jet pump:		
Pressure	psi	490–1000 (3.38–6.89 MPa)
Flow rate	gpm	432–491 (27.3–31.0 L/s)
Nozzle diameter	in.	0.70 (17.8 mm)
Throat diameter	in.	2.00 (50.8 mm)
Turntable speed	rpm	1.8 (1.8 rpm)
Mining arc	deg	360 (360°)
Mining depth	ft	235–249 (71.6–75.9 m)
Vertical increment	in.	2–6 (50.8–152.4 mm)

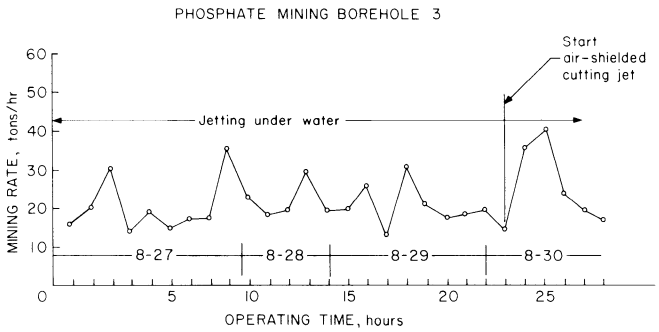

Fig. 22.4.12. Phosphate production in borehole 3. Conversion factor: 1 ton = 0.9072 t.

air shielding substantially increases water-jet effectiveness while operating underwater.

Further borehole mining in St. Johns County was performed in 1984 and 1985 by Agrico Chemical Co. and International Minerals and Chemical Co. The borehole mining tool used was built by Flow Industries based on the USBM design, but with the addition of a cutting nozzle that can be moved independently of the slurry pump. This test confirmed the findings of the Bureau of Mines tests and in addition demonstrated that well casings could be removed and reused.

Large deposits of phosphate ore occur on the Continental Shelf off the coasts of North Carolina, Georgia, and Florida. Serious consideration has been given to methods of mining these deposits. It appears that borehole mining may have potential as a method for offshore phosphate mining.

22.4.6 BACKFILLING OF BOREHOLE-MINED CAVITIES

Surface subsidence and the presence of tailings piles are the major potential adverse environmental impacts of borehole mining operations. Methods of mitigating these impacts have been investigated by the Bureau of Mines. Three methods were tested for backfilling cavities at the Nine-Mile Lake site with the sand produced during previous borehole uranium mining operations. The project consisted of intervals of backfilling followed by photographic surveys to determine the distribution of backfill in the hole. The backfilling methods investigated included bulk dumping down the borehole, slurry jetting in air, and slurry jetting under water. Slurry jetting underwater was found to be the most effective method. More than 90% of the sand removed from the cavity was returned by that technique.

A 1% by weight cement-sand mixture was introduced into a 4-in. (102-mm) ID pipe through a hopper, upstream of the centrifugal slurry pump (Fig. 22.4.13). The outlet pipe from the pump was connected via a loose victaulic coupling (which acted as a swivel) to a similar pipe terminated by a 4-in. (102-mm) ID elbow in the borehole. Slurry was injected at the rate of 350 gpm (22.1 L/s) through a 4-in. (102-mm) pipe rotating under water in the cavity. Sand was backfilled at the rate of 16 tph (14.5 t/h).

Analysis of cores taken from the backfilled cavity after 6 months indicated that adding 1% by weight of cement to the backfill did not increase the stability of the backfill material. It is estimated that a 5% mix would be required.

Backfilling experiments were conducted along with borehole phosphate mining experiments conducted by the USBM in cooperation with Agrico, Inc., in St. Johns County, FL. These tests, and follow-on tests, conducted by Agrico and International Minerals and Chemical Co. and described by Woolsey and Dibble (1987), demonstrated that waste clay, a major environmental and disposal storage problem in the phosphate mining industry, could be successfully thickened and injected into the cavities created by mining.

22.4.7 PRESENT STATUS

The technical feasibility of the remote extraction of coal, oil sands, uranium ore, and phosphates as a slurry through a borehole has been demonstrated. It has also shown that borehole mining can be performed so that the associated environmental impact is minimal.

No commercial borehole mines are in existence, but borehole mining has been the subject of several field trials. Borehole mining of phosphates was the most successful of the field trials. The productivity was higher than that of the other commodities because of the lack of induration of the phosphate ore, and because of the high-positive suction head of the slurry pump owing to the fact that mining took place with the borehole filled with water.

Borehole mining fulfills the need for a method to mine “incremental” uranium ore. Incremental ore refers to those small, irregular, high-grade uranium ore bodies that, although adjacent to working open pits, cannot be mined from these pits because of engineering limitations. The small size and the irregularity of these deposits make them ideal candidates for borehole mining because of the high areal selectivity of the borehole mining method.

The borehole mining field tests of oil sands and coal demonstrated the technical feasibility of the remote extraction of these commodities through boreholes, but the rates at which these fuels were produced under the conditions tested were too low for commercial viability. The tests demonstrated the need for developing borehole mining equipment that will allow higher productivity.

Backfilling of borehole-mined cavities by horizontal, underwater jetting of slurry into the cavities was proven to be feasible. Backfilling is likely to be an attractive method to prevent subsidence in those cases where a suitable supply of granular fill is available. Disruption to the environment would be minimal unless fill would have to be obtained from a borrow pit.

Environmental monitoring for groundwater pollution and subsidence conducted during these mining tests indicated the virtual absence of both phenomena. This indicates that borehole mining may be an attractive candidate for mining environmentally sensitive areas.

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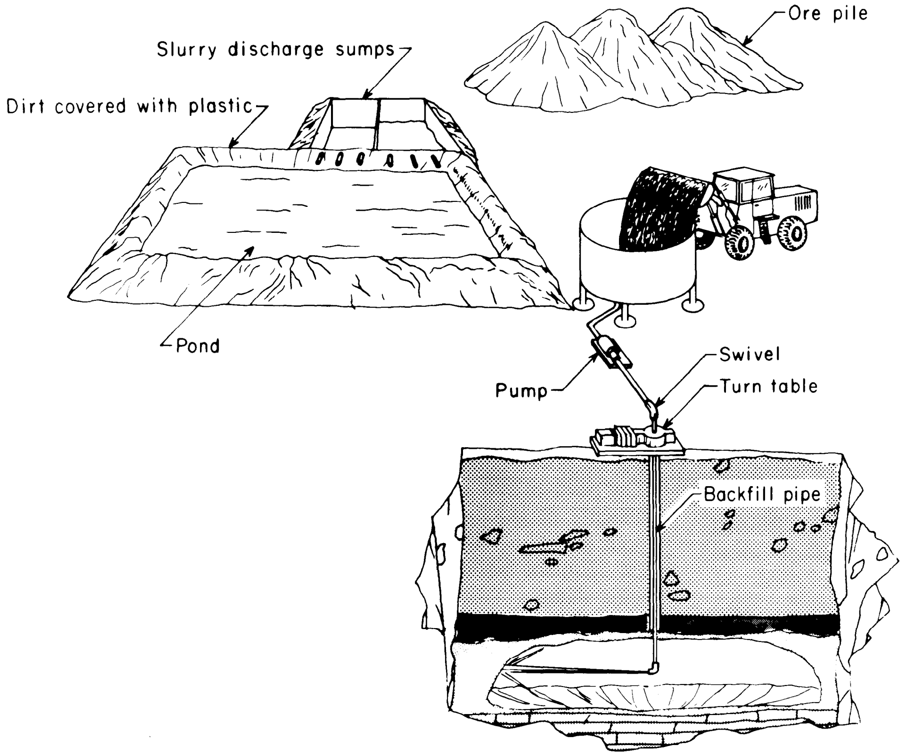

Fig. 22.4.13. Schematic of backfilling apparatus.

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Chapter 22.5
METHANE DRAINAGE

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22.5.1 INTRODUCTION

Methane is a gas that is slightly lighter than air and is colorless, odorless, and nonpoisonous. It occurs in the air of most coal mines, because when coal was formed from ancient peat swamps, methane also was formed. Air containing 5 to 15% methane and at least 12.1% oxygen concentration will explode if ignited.

All coalbeds contain methane, the amount of which can vary from 0.01 ft^3/ton (3.1 x 10^{-2} m^3/t) to more than 600 ft^3/ton (18.7 m^3/t) (Skow, Kim, and Deul, 1980). Usually, the amount of methane in coal increases with higher rank and greater depth. The gas is adsorbed on the micropore structure of the coal matrix and compressed in the fracture system of the coalbed. Wherever coal is uncovered by erosion or mining, the equilibrium that exists in the coalbed under confining pressure is disturbed, and methane is emitted.

The rate of methane emission underground is dependent upon the pressure gradient between the coalbed and the mine and also upon the permeability of the coalbed. Not all the gas is emitted at the working face where the coal is extracted; as much as 70 to 80% of the methane in the mine atmosphere enters from the exposed ribs and adjacent strata. But the emission rate per unit area of coal exposed is many times higher at the face than from the ribs. Thus, emission in the face area is frequently a major determinant of the quantity and delivery rate of fresh air required to ventilate the mine.

In the early years of coal mining, inadequate ventilation, failure to test for methane, use of open lights, smoking, failure to remove or control dust accumulations, and the improper use of black powder for blasting were the most frequently cited factors in explosions (Deul and Kim, 1988). Although both the number of explosions and the number of fatalities per million tons of coal produced have declined, approximately 10% of all underground coal mine fatalities are still due to explosions.

Ventilation is the only technique used universally in underground mines for the control of methane. To maintain operations within federally mandated requirements, the general approach has been to increase the quantity of ventilation air as methane emissions increase. However, there are times that even large quantities of air are not sufficient to dilute and remove the methane safely. Faces with high coal production rates, newly developing mines, and mines where development headings are being driven in virgin coal are all potential candidates for excess methane emissions.

The only alternative to decreasing the rate of coal production in order to stay within the statutory limits is to decrease the amount of methane emitted from the coalbed. To do this requires either removing the gas from the coal before it is mined or controlling the emission rate during mining. Toward this goal, a number of researchers have developed a basic understanding of the storage and migration characteristics of methane in coal, techniques to remove gas from coal before it is mined, and methodologies to control emissions into the working place during mining.

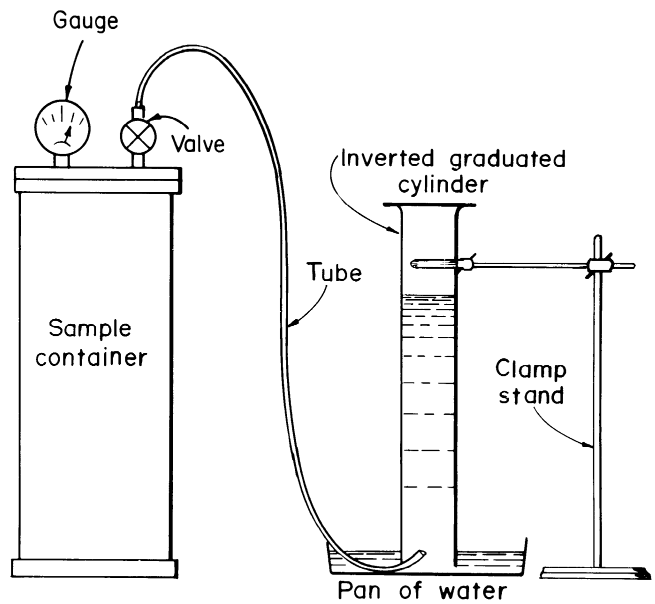

Fig. 22.5.1. Equipment for measuring gas content of coal cores.

22.5.2 DEFINING THE METHANE PROBLEM

Coal for gas-content testing is usually obtained from core samples recovered from vertical exploratory drillholes. Generally, it is possible to obtain only a limited amount of sample for gas content testing because of the coal-quality testing needs of the coal company. Therefore, it is important in all cases to obtain the cleanest section of coal, that is, coal without obvious extraneous shale, pyrite, or other noncoal inclusions. Collection of multiple samples or even testing of the entire coalbed is the preferable sampling procedure.

The equipment (Fig. 22.5.1) required to measure the actual volume of gas desorbing from the coal sample consists of a sample container, an inverted graduated cylinder sitting in a pan filled with water, and a ring stand and clamps to hold the graduated cylinder in place. The desorbed gas that collects in the sample container is periodically bled into the graduated cylinder and measured as the volume of water displaced. This procedure is performed at the drill site and subsequently in the laboratory.

22.5.2.1 Calculation of Gas Content

The gas content of a sample is composed of lost, desorbed, and residual gas, each of which is determined by slightly different techniques (Kissell, McCulloch, and Elder, 1973). Generally, coal begins to desorb gas when the pressure equilibrium in the coalbed is disturbed. Therefore, gas desorption takes place as the coal sample is recovered. The volume of gas desorbed is known as *lost gas,* primarily because it is gas that is lost from the sample during the recovery and collection process. The amount of lost gas depends on the drilling medium and the time required to retrieve, measure, and describe the core and seal the sample in the can. The shorter the time required to collect the sample and seal it in the can, the greater the confidence in the lost-gas calculation. In general, because of its speed, wireline retrieval of the core is preferable to conventional coring. If air or mist is used in drilling, it is assumed that desorption begins immediately upon penetration by the bit. With water, desorption is assumed to begin when the core is halfway out of the hole, that is, when the gas pressure in the sample is assumed to exceed that of hydrostatic head.

The volume of lost gas can be calculated by a graphical method based on the following relationship. For the first few hours of emission, the volume of gas desorbed is proportional to the square root of the desorption time. A plot of the cumulative volume desorbed after each measurement against the square root of the time that the sample has been desorbing ideally should produce a straight line.

Desorbed gas is simply the total volume of gas drained from the sample and measured in the graduated cylinder. Generally, a sample is allowed to desorb gas until a very low emission rate is obtained, usually an average of less than 4×10^{-4} ft^3/day (10 cm^3/day) for a one-week period. The time required to reach this low rate of emission varies depending on the size of the sample, the physical characteristics of the coal, and the amount of gas contained in the sample.

When the measurement of desorbed gas is discontinued, the coal sample will usually still contain gas. To complete the gas content determination, the amount of residual gas must be measured. Residual gas is released by crushing the coal sample to a fine powder in a sealed container. When crushing is complete, the container is allowed to cool to room temperature, and the volume of gas released is measured by the same water displacement method used for determining desorbed gas. The crushed powder and any uncrushed lumps are weighted separately. The volume of gas released is attributed only to the crushed powder. The total gas content of a particular sample is the volume of lost gas and desorbed gas divided by the total sample weight plus the residual gas content.

Theoretically, it is possible to crush a coal sample at any point after collection and to obtain the total gas content (excluding lost gas) of the sample. This procedure generally is not considered appropriate if maximum information from the sample is desired. By crushing the sample before the desorption process is complete, it is impossible to obtain the relative amounts of desorbed and residual gas. This distinction is important because the actual residual gas, which will not desorb from the sample while sealed in the canister, probably represents gas that will not flow to a methane drainage borehole, and possibly represents gas that will not be emitted into a mine atmosphere. During the process of mining coal, the coal is broken up into a variety of sized pieces; however, the majority of these pieces usually will not duplicate the very fine powder that crushing produces in the residual gas content procedure. Table 22.5.1 lists the highest measured gas contents in the United States (Diamond, LaScola, and Hyman, 1986).

22.5.2.2 Geologic Influences

Previous horizontal borehole methane drainage studies that utilized holes drilled in advance of mining found that geologic features (e.g., clay veins, sandstone channels, and the coal's natural fracture system) have a significant effect on borehole gas productivity (Prosser, Finfinger, and Cervik, 1981). Gas flow rates are generally found to be higher when drainage holes are drilled perpendicular to the direction of the face cleat (Fields, Cervik, and Goodman, 1976). However, coalbed discontinuities such as clay veins and sand channels tend to isolate blocks of coal and restrict gas flow.

CLEAT ORIENTATION IN COALBEDS. The natural vertical fracture system in bituminous coalbeds, *cleat,* is usually composed of two components (sets) separated by at least 30° (McCulloch, Deul, and Jeran, 1974). The more dominant fracture plane is the *face cleat;* the minor fracture plane is the *butt cleat.* The face cleat is more continuous, crosses bedding planes in the coal, and may extend for longer distances. The butt cleat is commonly short, may be curved, and is a discontinuous feature that frequently terminates against the face cleat.

The orientation of coal cleats is important to efficient gas drainage. The face and butt cleats provide a directional permeability; that is, the flow of gas is greater in the direction parallel to the face cleat. Experiments conducted underground in coal mines have shown that horizontal holes drilled perpendicular to, and therefore intersecting, the largest number of face cleats will yield larger amounts of gas in a given amount of time than holes drilled perpendicular to the butt cleats.

The cleat system of a coalbed also affects vertical gas drainage holes drilled from the surface into a coalbed. A vertical hole completed in a coalbed with a well-developed face and butt cleat will collect gas at a higher rate from fractures oriented in the face cleat directions than from fractures trending parallel to the butt cleat direction. Therefore, an elliptical drainage pattern (Fig. 22.5.2) will be developed, with the long axis parallel to the face cleat. Differences in face and butt cleat permeability are most pronounced in blocky coals. In friable coals, where the face and butt cleats are more nearly equal in prominence, coalbed permeability tends to be anisotropic.

COALBED DISCONTINUITIES. When rocks are subjected to stress, they usually respond by bending and breaking or fracturing. When measurable movement occurs after breaking, a fault is produced. Fracture zones are areas that contain a higher number of fractures than adjacent areas. Fractures and faults when open and extensive are highly permeable to methane and water migration.

Movement along a fault zone may cause localized compression of the coal, which results in decreased permeability. Faults and fractures are related to geologic forces. Study of local geology and particularly structure can indicate the existence of these features and the extent to which they may affect methane drainage.

CLAY VEINS. Clay veins are wedges of indurated clays and silts that penetrate the coalbed from either above or below (Fig. 22.5.3). They can be vertical (parallel to the cleat direction) or can form an angle of about 45° with the vertical (parallel to shear direction). Clay veins encountered in mines are usually crooked, are frequently angular, and interfinger with the coal rather than forming smooth contact surfaces. They may be hard enough to damage mining equipment. Thickness normally ranges from 1 in. to several feet (25 mm to a few meters), but may be as much as 15 ft (4.6 m). A total length of more than 1000 ft (300 m) has been observed underground (Hadden and Sainato, 1969).

Clay veins often form cells of coal that isolate large volumes of gas under high pressure. In one study, a test hole drilled into a cell measured an in situ gas pressure of 263 psi (1813 kPa) (Hadden and Sainato, 1969). When mining encounters such a cell, gas is released rapidly into the mine environment, creating a hazard. Furthermore, the chance of an explosion is increased by sparks that may be generated when mining into clay veins with both continuous miners and longwall systems.

Methane drainage rates from horizontal boreholes in the Pittsburgh coalbed generally averaged 250 ft^3/day (7.1 m^3/day) per ft (m) of hole. In an area of the Pittsburgh coalbed containing

Table 22.5.1. Highest Measured Gas Contents of US Coalbeds

Coalbed or Formation	Location	Depth, ft	(m)	Gas content m^3/t^a	ft^3/t^b	Coal Rank
Peach Mountain	Schuylkill, PA	685	(208)	21.6	691	Anthracite
Pocahontas No. 3	Buchanan, VA	1,864	(568)	21.5	688	Low-volatile bituminous
Mary Lee	Tuscaloosa, AL	1,504	(458)	18.7	598	High-volatile A bituminous
Tunnel	Schuylkill, PA	608	(185)	18.3	586	Anthracite
New Castle	Tuscaloosa, AL	2,132	(649)	17.5	560	Medium-volatile bituminous
Hartshorne	Le Flore, OK	1,439	(438)	17.1	547	ND
Mesaverde Group	Sublette, WY	3,496	(1065)	17.0	544	High-volatile bituminous
Vermejo Formation	Las Animas, CO	1,158	(353)	17.0	544	Low-volatile bituminous
Beckley	Raleigh, WV	830	(253)	15.3	490	Medium-volatile bituminous
Pratt	Tuscaloosa, AL	1,365	(416)	15.1	483	ND

ND, Not determined. [a] Laboratory derived. [b] Estimated in-place.

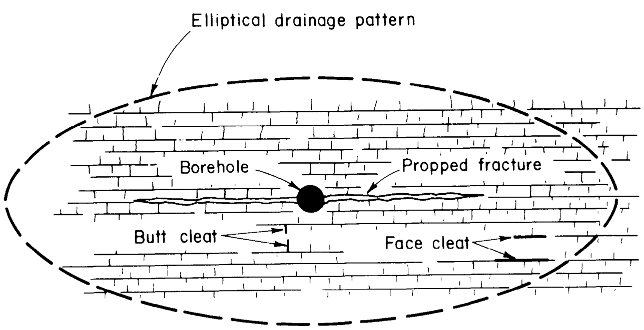

Fig. 22.5.2. Elliptical gas drainage pattern resulting from differences in cleat permeability.

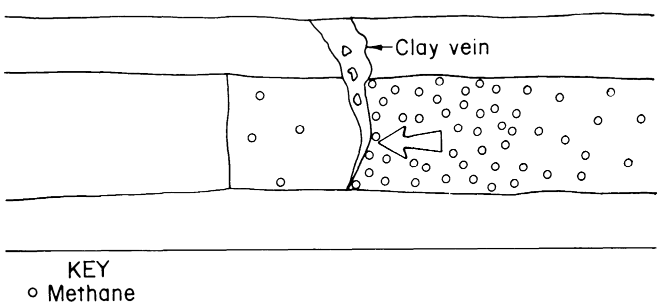

Fig. 22.5.3. Clay vein blocking methane migration.

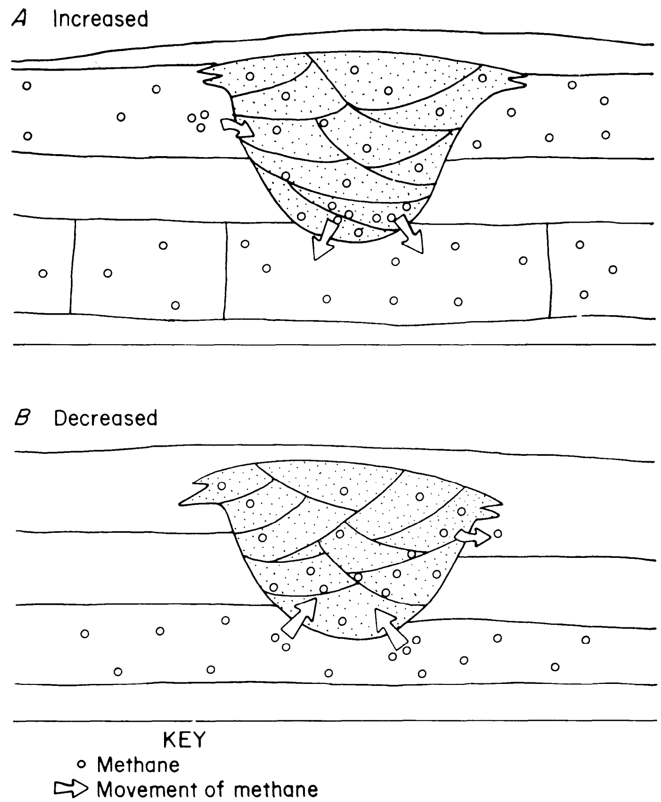

Fig. 22.5.4. Sandstone channels impacting methane migration.

clay veins, the methane emission rate was less than half of the average (Prosser, Finfinger, and Cervik, 1981). Since methane drainage is usually conducted in virgin areas, the geologic data to locate small discontinuities such as clay veins may not be available. In such cases, the degasified area may be limited to blocks of coal delineated by the clay veins and/or other discontinuities.

SAND CHANNELS. Sand channels are lenticular, sometimes sinuous bodies of sandstone extending into and sometimes completely through a coalbed (Fig. 22.5.4) (McCulloch et al., 1975). They are erosional depositional features formed by the cutting action of paleostreams that were eventually filled with sediment. As deposition of the sediment filled a particular channel, new channels formed. Thus a deltaic environment may contain a succession of sand-filled channels. These channels often impact coalbed permeability, thereby reducing the flow of gas through the cleat system and into methane drainage boreholes. Identifying the channels prior to initiating a methane drainage study can allow for a more effective drainage design. This is accomplished by locating drainage boreholes in areas where they are least impacted by the reduced permeability zones.

Many geologic factors affect methane emission rates in underground coal mines. The orientation of the cleat system with respect to methane drainage boreholes may have an effect on the competency of the hole and the rate at which methane gas is produced. Fault and fracture zones, clay veins, and sandstone channels can aggravate methane emission problems and complicate attempts to drain gas in advance of mining. Predicting cleat orientation and indicating the probable location and extent of other geologic features through geologic reconnaissance, computer-assisted mapping, and other techniques can improve the application of drilling technology and drainage systems to the control gas in underground bituminous coal mines.

22.5.2.3 Impact of Mine Design

ROOM-AND-PILLAR/LONGWALL DEVELOPMENT. Mine operators in the United States predominantly use continuous mining machines in room and pillar mines. The longwall (retreat) method of mining also requires the use of a continuous miner to develop the panel. These longwall development sections are essentially identical to room and pillar mining sections. Therefore, all the 200 mines identified as being significantly gassy (daily emissions of methane of at least 100,000 ft^3/day or 2830 m^3/day) utilize room and pillar mining techniques (Grau and LaScola, 1984). Methane problems on room and pillar or longwall development sections normally occur in one of two areas: the working place or returns.

Methane control standards established by the Mine Safety and Health Administration are defined within Title 30 of the *Code of Federal Regulations,* CFR, Part 75—Mandatory Safety Standards—Underground Coal Mines. The *working place* is defined in Paragraph 75.2 (g) (1) as the area of a coal mine inby (towards the face) of the last open crosscut. In this area, all electrical equipment must be permissible. The *returns* are the entries on the section through which the ventilating air is coursed after it has swept the face area and is contaminated with gas. Any electrical equipment used in the returns must also be permissible. The *fresh air entries* are those splits that course the air to the working face for ventilation purposes. During normal operations, the distribution of methane gas in the air at the face and in the returns must be kept below 1.0%. Methane gas concentration in the fresh-air entries is typically very low (< 0.2%).

RETREAT LONGWALLS. The retreating longwall is the predominant longwall mining method used in US underground coal mines. In fact, of the 92 operating faces in the United States in 1989, only one was an advancing face (Sprouls, 1989).

It is significant to note that 16 of these operating retreat longwalls are in the 10 mines identified as having the highest methane emissions in the United States. Methane problems on retreat longwall sections normally occur in one of five areas: working face, returns, fresh air entry, bleeder system, or gob. The definitions and allowable methane concentrations (working place, returns, fresh air entry) described for room and pillar sections are also applicable for retreat longwalls.

A gob area is associated with full-extraction mining (e.g., longwall) and is the area of the mine roof that collapses after the coal is fully removed. When the immediate roof collapses and falls into the space created by the removal of the coal, fractures propagate up into the overlying strata. If there are methane-bearing strata above the longwall panel, then the gas can migrate from the overlying areas into the collapsed material.

A *bleeder system* is a set of entries that carries methane-laden air from the mine after the air has been directed through abandoned areas that have been wholly or partially extracted. The concentration in the bleeder system cannot by law exceed 2%.

22.5.2.4 Regional Impacts: Geographic Variations

During 1985, 180 coal mines in the United States had methane emissions in excess of 100,000 ft^3/day (2831 m^3/day) (Grau, 1987). The total amount of methane gas produced by these mines equaled 304 million ft^3/day (8.6 Mm^3/day) and was emitted from mines located in 12 states.

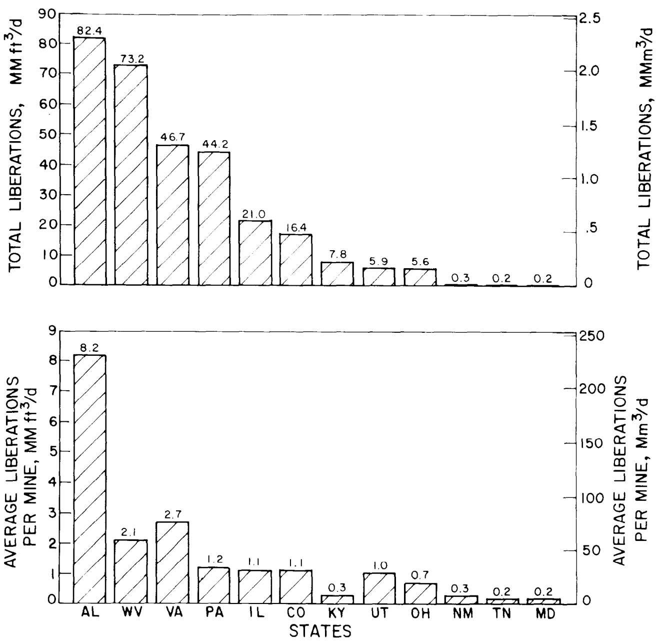

Fig. 22.5.5. Coal-mine methane emissions.

The relationship between coalbeds and methane emissions was analyzed, and the data are shown in Fig. 22.5.5. Most emissions came from the Pittsburgh Coalbed, which produced 80 million ft^3/day (2.3 Mm^3/day), or 26.5% of the total methane emitted. Mines operating in the Mary Lee and Pocahontas No. 3 Coalbeds emitted slightly less than mines in the Pittsburgh Coalbed. The highest average emission rate per mine came from the operations in the Mary Lee Coalbed at 8 million ft^3/day (227,000 m^3/day).

22.5.3 METHANE DRAINAGE TECHNOLOGY DURING MINING

Typically, as methane emissions increase on a section, more air is directed to the section to dilute the gas. However, as coalbeds with higher gas contents are exploited in conjunction with the demand for ever-increasing production rates, the mine ventilation engineer may find that it is not possible to direct a sufficient amount of air onto the section to ensure the safe extraction of the coal. To enable the mine to maintain production, various techniques were developed to remove gas from the coal. These techniques include drainage from horizontal, vertical, and directionally drilled boreholes.

22.5.3.1 Horizontal Borehole Drainage

The method of drilling a horizontal borehole into a coalbed to remove methane has been known since the early 1960s; however, much of the technology to effectively drill these holes was perfected in the 1980s (Spindler and Poundstone, 1961; Cervik, Fields, and Aul, 1975; Prosser, Finfinger, and Cervik, 1981). Fig. 22.5.6 illustrates the principle of methane drainage through horizontal boreholes. Basically, the horizontal borehole extends the low-pressure atmosphere within the mine to some distance within the coalbed.

There is a concomitant twofold effect on the coalbed gas reservoir that occurs as gas is removed. First, gas content is

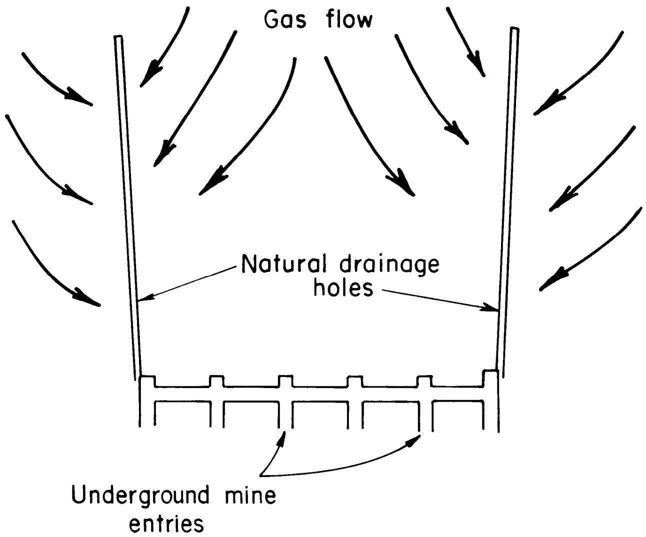

Fig. 22.5.6. Concept of methane drainage by horizontal boreholes.

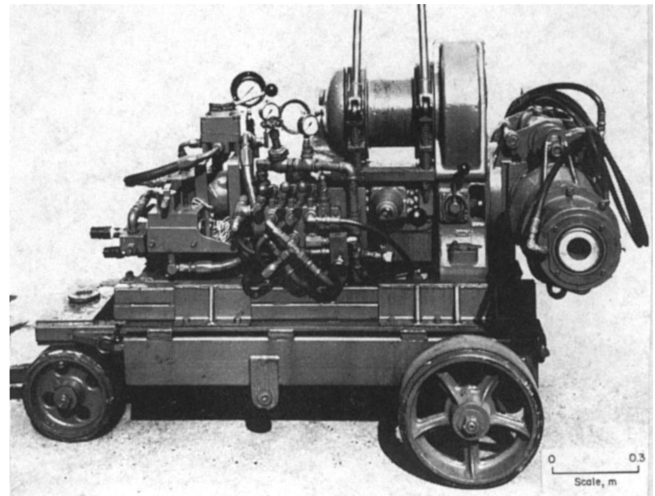

Fig. 22.5.7. Horizontal methane drainage drill.

lowered. This was shown in one study to be as great as 50% after four years of drainage. Second, the gas pressure is lowered.

For the mine ventilation engineer, the practical effects of horizontal methane drainage are a reduction in methane emissions during mining, an increase in mine safety, a potential for increased productivity, and a potential for selling the captured gas.

The use of horizontal boreholes has been shown to consistently reduce methane emissions by more than 50% thus improving the safety of the mining environment. Also, on a longwall development section in an Alabama mine, a 50% increase in productivity was achieved in addition to a 54% decrease in methane emissions. Finally, an economic analysis has shown that a 25% rate of return on investment can be realized from the sale of 3 billion ft^3 (85 Mm^3) of methane at a mine site price of \$2.25/1000 ft^3 (\$79.00/1000 m^3) (Baker, Grau, and Finfinger, 1986).

Horizontal methane drainage is a many-faceted technique for methane control. It can be used to drain large blocks of coal, prior to mining and to control both excess face emissions and methane problems in the returns. The earliest reported studies of this technology were conducted to determine the proper procedures for drilling short horizontal boreholes (Hadden and Sainato, 1969). During these studies, the electrohydraulic drills used were either hand-held or post-mounted. These drills were easy to transport underground but were not capable of drilling holes of lengths up to 1000 ft (300 m). Later studies which investigated factors that influenced bit trajectory led to the design and development of equipment for drilling horizontal boreholes in excess of 1000 ft (300 m) (Cervik, Fields, and Aul, 1975).

22.5.3.2 Horizontal Drilling Operations

Methane drainage from virgin coal can be incorporated into the mining cycle. In this process, long horizontal holes are drilled in outside entries, ahead of the advancing section. Methane gas that would normally flow toward the face areas of an advancing section is diverted away through the horizontal drainage holes. This process lowers gas pressure immediately ahead of the face.

The rate at which methane gas can be removed from the coalbed is proportional to the length of the drainage hole. Drilling long horizontal holes in a coalbed requires maintaining the drillbit on the proper trajectory within a relatively small target zone. There are two basic techniques that have been developed for guiding the path of a drillbit during horizontal drilling in coalbeds: rotary drilling and down-hole drilling (Cervik, Fields, and Aul, 1975; Kravits, Sainato, and Finfinger, 1985). Both techniques require the use of an underground drill unit that has been constructed to allow drilling in a horizontal orientation. There are a variety of units that can be used for this application (Fig. 22.5.7) (Finfinger and Cervik, 1980). The drills are normally either pneumatically or electrically powered. The primary function of the drill is to provide both rotation and thrust to the drillbit. Bit thrust and rotational speed are the two drilling parameters that affect the path of the bit in a vertical plane during rotary drilling. The applied thrust determines not only the penetration rate but also the path of the bit. Table 22.5.2 summarizes the effect of thrust and bit rotational speed on bit trajectory (Cervik, Fields, and Aul, 1975). Levels of thrust and rotational speed shown in Table 22.5.2 are meant as guidelines and may vary from one coalbed to another.

Table 22.5.2. Effect of Thrust and Bit Rotational Speed on Hole Trajectory

Thrust, lb	(kg)	Bit, rev/min	Effect on bit trajectory
800	(362)	700–900	Downward or dropping
1200	(544)	400–600	Holds angle
2000 or greater	(907)	200–300	Upward or lifting

The procedures for dropping the bit trajectory are to reduce thrust to about 800 lb (362 kg) and increase bit rotational speed to about 800 rpm. At the end of each 10-ft (3-m) segment of drilling, several slow reaming passes are made to wear the bottom of the hole. This procedure is repeated for each 10 ft (3 m) of penetration for distances of 50 to 100 ft (15 to 30 m).

Procedures for lifting the bit trajectory are easier and generally more effective. Thrust is increased to 2000 to 3000 lb (900 to 1360 kg), and bit rotational speed is about 200 rpm. A hole deviating by 1° is brought back on course within 30 ft (9 m) of horizontal drilling.

In order to determine the path of the drillbit, it is necessary to periodically survey the angle of the borehole (Cervik, Fields,

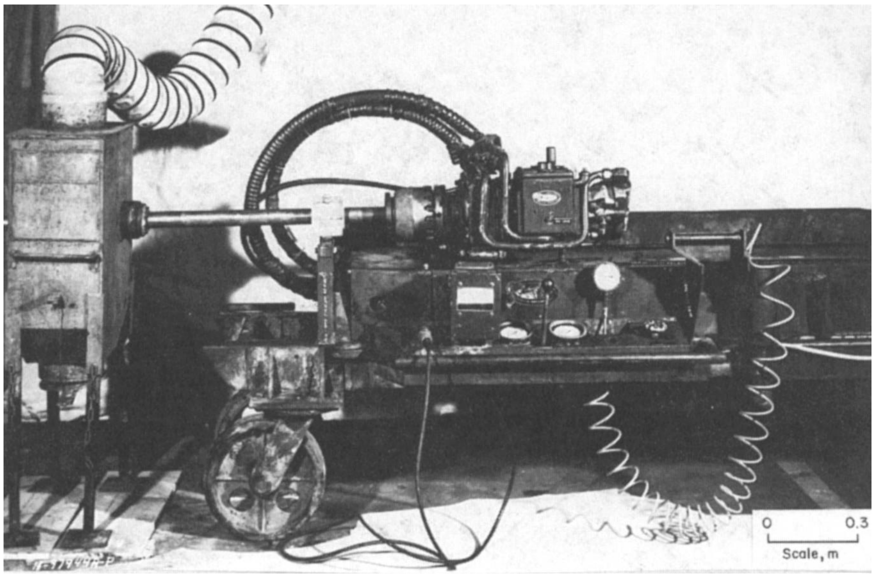

Fig. 22.5.8. Stuffing box attached to front of horizontal drill.

and Aul, 1975; Rommel and Rives, 1973). Surveying is conducted with instruments designed specifically for this purpose, such as photographic or electronic devices. Both instruments provide similar data on borehole inclination and compass heading. From successive surveys, it is possible to compute the path of the borehole trajectory and implement changes to the drilling operation to match the trajectory of the borehole to the dip of the coalbed.

A down-hole motor can be used for rotational power and borehole trajectory control in conjunction with the underground drill unit used to provide bit thrust. The down-hole motor hydraulically rotates the drillbit without rotating the drill rods connecting the drill unit to the motor. With an in-hole motor assembly, borehole trajectory control can be achieved by placing a slightly bent (2° or less) drill rod directly behind the motor and aiming the bend in the desired direction. The drillbit will directionally react based on the orientation of the bent drill rod. This type of drilling provides a more positive directional control system than rotary drilling (Kravits, Sainato, and Finfinger, 1985).

To control methane emissions during drilling, a pilot hole is drilled into the coalbed for a distance of 10 to 20 ft (3 to 6 m). The hole is then cased with 4-in. (102-mm) diameter steel pipe. The pipe is then grouted in place and connected to a stuffing box (Fields, Cervik, and Goodman, 1976). The stuffing box (Fig. 22.5.8) has a spring-loaded door on the bottom and a flexible hose at the top. Drill cuttings and water drop to the bottom of the stuffing box, and methane gas is drawn out through the top. The gas produced can be discharged in the returns, but this requires large volumes of air to quickly dilute the gas below the explosive limit. An alternative approach is to pipe the undiluted methane gas to the surface. The gas can then be captured in almost pure form and transported in pipelines as a form of natural gas. Methane from coal has properties similar to those of natural gas and can be used in many cases as a direct substitute. With the increased cost of natural gas, methane from coal can be a valuable asset.

Horizontal drainage holes are effective in any mine when the coalbed has relatively good permeability. In-mine drainage studies have been conducted in a variety of coalbeds, and the studies in the Pittsburgh and Pocahontas No. 3 Coalbeds are summarized as follows

CASE HISTORY 1: PITTSBURGH COALBED. In an initial study, three horizontal holes were drilled in a 500-ft (152-m) wide section of the Pittsburgh Coalbed, the hole in the center of the section being used to continuously monitor gas pressure within the coalbed (Deul and Kim, 1988). The holes on the right and left sides of the section were drilled about 5° off the projected development of the outside entries.

Initially, gas pressure within the coalbed was 33 psi (227 kPa). When the second hole was drilled, the gas pressure declined. When the third hole was completed, the measured gas pressure within the coalbed was reduced to 9 psi (62 kPa). The decline in gas pressure occurs when methane that would normally flow toward the face is drained from the coal through the horizontal holes.

In another study in the Pittsburgh Coalbed, two holes were drilled in Bethlehem Mines Corporation Marianna Mine (Prosser, Finfinger, and Cervik, 1981). One hole was 1066 ft (325 m) long, the other 2500 ft (762 m) long. The methane production rate was 180,000 ft^3/day (5097 m^3/day) from the first hole and 330,000 ft^3/day (9345 m^3/day) from the second hole. The gas drainage rate averaged 143 ft^3/day/ft (12 m^3/day/m) of drainage hole. Over a one-year period, methane emissions at the face were reduced by 90%.

CASE HISTORY 2: POCAHONTAS NO. 3 COALBED. A study conducted in the Pocahontas No. 3 Coalbed in an area known to have low permeability indicated that horizontal drainage holes were not effective at reducing face emission (Deul and Kim, 1988). The mine section was about 360 ft (110 m) wide. A gas-pressure monitoring hole was drilled near the left side of the section to a depth of 108 ft (33 m) and filled with packers to about 98 ft (30 m). Gas pressure in the 10 ft (3 m) of open hole at the inby end was monitored continuously. Hole 2 (left side) and hole 3 (right side) were drilled to 255 and 259 ft (78 and 79 m), respectively. Gas pressure in hole 1 stabilized at 650 psi (4481 kPa) and remained constant during drilling. After 14 days, pressure declined to 636 psi (4385 kPa). During this period, the total quantity of methane emitted through the face areas of the section, approximately 4 million ft^3 (113,000 m^3), showed little or no decrease. The small decline in gas pressure (2%) can be attributed to normal bleed-off through the face areas of the section during an idle period rather than to drainage through the horizontal holes.

The low permeability in this area of the Pocahontas No. 3 Coalbed was related to a large fault traversing the mine property. Subsequent tests have indicated that coalbed permeability improves as the distance from the fault increases. Therefore, horizontal drainage holes may be a feasible method of controlling methane in other mines in the Pocahontas No. 3 Coalbed.

22.5.3.3 Cross-measure Boreholes

Methane gas produced by longwall operations in the United States is controlled by dilution with ventilation air and by surface gob boreholes. Surface gob boreholes cannot always be drilled, however, because of a lack of available surface sites. Furthermore, topography may be too severe to develop adequate surface locations, or access to private property may be denied. Consequently, an alternative method of controlling gob gas is needed that is independent of the mine ventilation system and surface right-of-way problems.

The single-entry longwall is the predominant mining system in Europe where both retreating and advancing longwalls are used (Cervik, 1981). However, the proportion of faces utilizing advancing techniques dominates (82% in the United Kingdom and 75% in West Germany). Post-World War II mechanization and mining of deeper and gassier coalbeds forced the application of gob drainage systems on European longwalls. The most commonly used method of controlling gob gas during mining in Europe is the cross-measure borehole technique.

CROSS-MEASURE BOREHOLE SYSTEM DESIGN. The design of cross-measure boreholes in the United States is based on European experience (Schatzel, Finfinger, and Cervik, 1982; Campoli, Cervik, and Schatzel, 1983). Endpoints of the cross-measure boreholes are normally spaced 200 to 300 ft (61 to 391 m) apart. All boreholes are drilled over a support pillar to protect the borehole when the longwall face passes the collar of the hole. Borehole length, inclination, and horizontal direction should be chosen to accomplish the following:

1. Maintain the height of the borehole over the far end of the support pillar at least eight times the coalbed thickness. Estimates of the height of the rubblized zone in the gob range from four to eight times coalbed thickness.

2. Terminate the borehole about 100 ft (30 m) into the gob. Researchers have shown higher methane concentrations in the gob on the return side than the intake side because of the pressure differential in the mine's ventilation system.

3. Intercept all coalbeds within 100 ft (30 m) above the mined coalbed.

4. Angle the borehole at least 45° (0.79 rad) to the axis of the longwall.

Generally, gas production begins when the face is 75 to 100 ft (22 to 30 m) beyond the end of the borehole but before it passes the collar of the hole.

DRILLING EQUIPMENT AND PROCEDURES. An electrohydraulic or pneumatic drill is used to drill the cross-measure boreholes (Fig. 22.5.9). The first 25 to 30 ft (7 to 9 m) of each borehole are drilled with a 4-in. (101-mm) diameter drill, while the remainder of the hole is drilled with a 2-in. (50-mm) diameter bit. Plastic standpipes are cemented into each cross-measure borehole after the completion of the drilling phase. The cementing phase is important because, if not completed properly, mine air could short-circuit into the cross-measure borehole, reducing the borehole's ability to draw gas from the gob. Access to the surface is provided by a surface borehole. The borehole is lined with steel pipe and cemented the entire length. An underground pipeline is used to connect the surface borehole to the cross-measure boreholes.

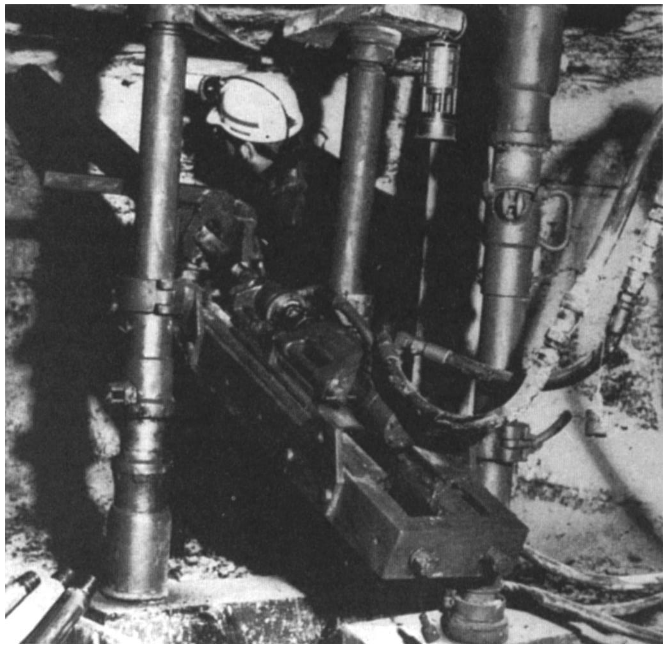

Fig. 22.5.9. Drill unit for cross-measure boreholes.

GOB GAS PRODUCTION. A cross-measure borehole study was conducted on a retreating longwall in Bethlehem Mines' Cambria 33 mine (Campoli, Cervik, and Schatzel, 1983). An enlarged view of the test panel is shown in Fig. 22.5.10. The underground pipeline was located in the center entry of the three-entry return-air gateroad. All cross-measure boreholes were drilled from the center entry, which was supported by a line of cribs. The center entry remained open and passable after the panels on both sides were removed.

Gob gas was extracted from 11 of the 12 cross-measure boreholes. Hole 1, the only borehole that failed to produce gob gas, was drilled with its endpoint only 27 ft (8 m) from the starting point of the panel. Apparently, the strata in this area did not fracture; thus no flow paths were created to permit gob gas movement toward the hole.

No free flow of methane gas occurred from any cross-measure borehole. Even with an applied partial vacuum, flow did not occur until the face passed 75 to 100 ft (30 m) beyond the end of the hole. Flow did occur however, before the face passed the collar of the hole.

Figs. 22.5.11 and 22.5.12 show methane and gob gas (methane plus air) flow rates, respectively, for each cross-measure borehole. Generally, the flow-rate curves for each cross-measure borehole follow the same pattern. The gob gas production life of the cross-measure boreholes ranged from 14 to 149 days. Borehole inclination and penetration into the gob were two important parameters that appear to affect gas production life of boreholes. Borehole life seemed to be greater for inclinations of about 25°. Boreholes with higher inclination angles also tended to produce higher average methane flow rates than lower inclination boreholes. Borehole penetration into the gob ranged from 90 to 136 ft (27 to 41 m), which is only 16 to 25%, respectively, of the width of the longwall panel. Optimum borehole inclination and penetration into the gob appeared to be at least 30° and 100 ft (30 m), respectively.

EFFECTS OF THE CROSS-MEASURE SYSTEM. Methane gas enters mine workings during longwall mining from fractured

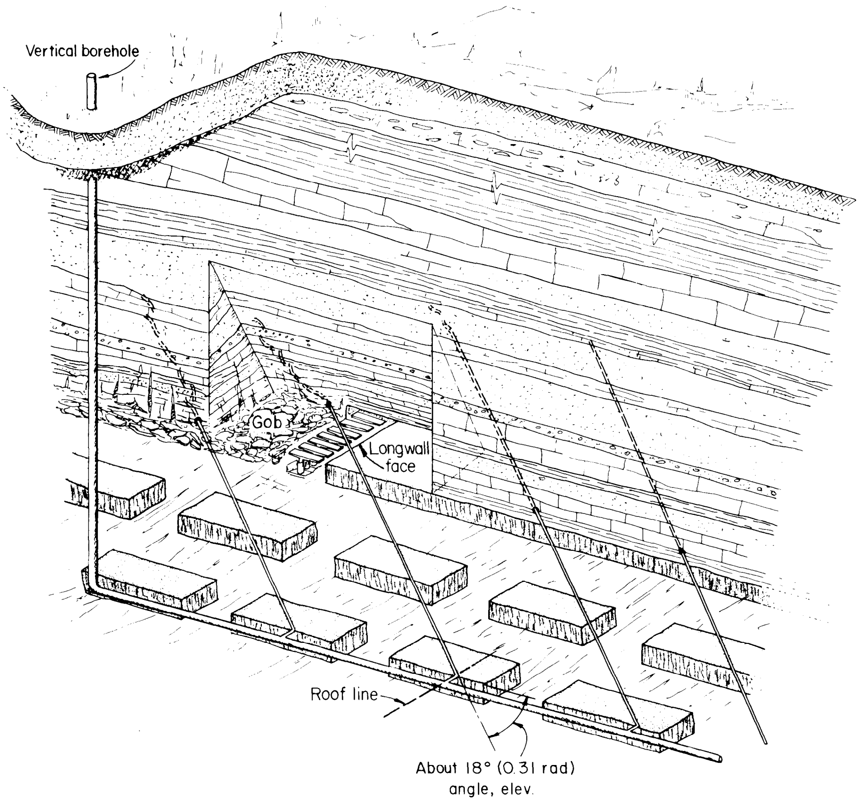

Fig. 22.5.10. Schematic of cross-measure borehole system for gob gas control.

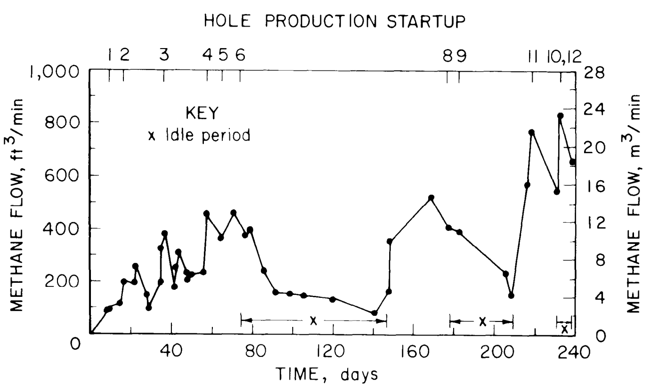

Fig. 22.5.11. Methane production history from cross-measure borehole system.

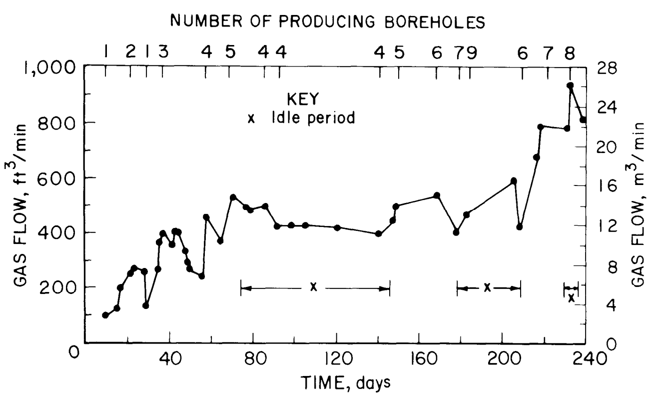

Fig. 22.5.12. Gob gas production history from cross-measure borehole system.

roof and floor strata and from the mined coalbed. Ventilation and methane surveys conducted at the tailgate of the longwall showed that the flow of methane gas from the mined coalbed during mining was about 20 ft^3/min (0.6 m^3/min). During nonmining periods, however, the flow of gas was negligible. The contribution of floor strata to methane emitted in the gob is unknown. Also coalbeds in the roof strata are known methane source beds.

The cross-measure borehole system captured about 50% of the methane produced by the mining operation. The other half was removed by the mine ventilation system. Methane flows in both systems declined during the life of the panel.

The effect of the cross-measure borehole system on methane in the mine ventilation system was dramatic. The methane flow in the return air from the longwall increased by a factor of > 2 when the cross-measure borehole system was not in operation. These data clearly demonstrate the effectiveness of the boreholes in controlling methane in the gob.

22.5.4 METHANE DRAINAGE TECHNOLOGY IN ADVANCE OF MINING OR WITHOUT MINING

22.5.4.1 Vertical Boreholes

The amount of gas contained within any given area of a coalbed depends on the coal's inherent storage capacity and the physical conditions of the coalbed. The storage capacity of coal is primarily a function of rank and, as such, cannot be affected by the presence of a methane drainage borehole. Certain physical characteristics of the coalbed can be changed which will affect the release and migration of gas in the coalbed. By removing accumulated water from a vertical borehole, two important changes occur in the coalbed: pressure is reduced, and water saturation level is lowered. Reducing pressure releases additional gas from the coal itself as the coalbed adjusts to the change. Increasingly larger amounts of gas are released for each incremental decrease in pressure. Lowering the water saturation level raises the permeability of the coalbed to gas and allows the gas to more readily migrate to the wellbore (Lambert and Trevits, 1978).

Vertical boreholes are normally rotary-drilled using portable drilling rigs. Air, mist, water, foam, and mud have been used to carry rock cuttings to the surface. To minimize the possibility of decreasing coalbed permeability by infiltration of drilling fluid, the use of drilling muds should be avoided except where considered absolutely essential. Air or light foam guards against permeability damage and increases drilling speed by reducing pore pressure at the bit-rock interface.

The depth and thickness of the coal units penetrated in boreholes must be known for stratigraphic correlation, well-completion plans, and stimulation design. Evaluation of rock cuttings and drilling time logs, though helpful, do not supply the precise information usually required. The most accurate data are obtained from cores, but these are quite expensive. Geophysical logging is less expensive and can be used to delineate coal zones less than 1 ft (0.3 m) thick. Density logs are typically used in coal identification. The relatively low bulk densities of coal cause distinctive log responses, which are easily recognized. Density logs are obtained for boreholes before they are cased.

If gas is to be drained from several coalbeds, casing is set along the full depth of the borehole, and openings are cut or pierced through the casing at each coalbed horizon. This type of completion allows any number of selected horizons to be produced simultaneously.

Two completion techniques have been used to expose coalbeds through casing: (1) perforating casing using shaped charges and (2) cutting vertical slots through the casing using jetting equipment (Lambert, Trevits, and Steidl, 1980). The perforating technique has been used in a number of different coalbeds. However, even in coals that have been hydraulically stimulated, production from perforated wells has been significantly less than production from wells with open-hole completions. Partial plugging of perforations may be a major cause of the comparatively low flows.

Hydraulic stimulation is a standard technique used to increase well productivity in the oil and gas industry. Modified versions of oil and gas industry hydraulic stimulation techniques have been applied to coalbeds in several areas of the United States. The basic mechanics of stimulation for gas drainage from coalbeds are quite simple (Fig. 22.5.13). Hydraulic pressure, generated at the surface by pumping fluid, is applied to a selected rock unit (coalbed) to widen or extend existing fractures. The fractures are extended into the coalbed by continued injection of fluid. During the treatment, sized particles (e.g., sieved sand) are added to the fluid as a propping agent to hold the fracture open after the hydraulic pressure is released, and the stimulation fluid is drained off. Stimulation creates conductive pathways from the wellbore to points several hundred feet (meters) within the coalbed.

Stimulation treatments are generally categorized according to the type of fluid used. Water, gelled water, and foam are the major types of treatment tested, each type having advantages as well as limitations. Gelled fluid is a water-based, natural guar gum mixture, which serves to carry sand proppant and retard leakoff. After stimulation, gelled fluids are designed to revert or break down to a fluid with a viscosity similar to water.

REQUIREMENTS OF FOAM-TYPE STIMULATION TREATMENTS

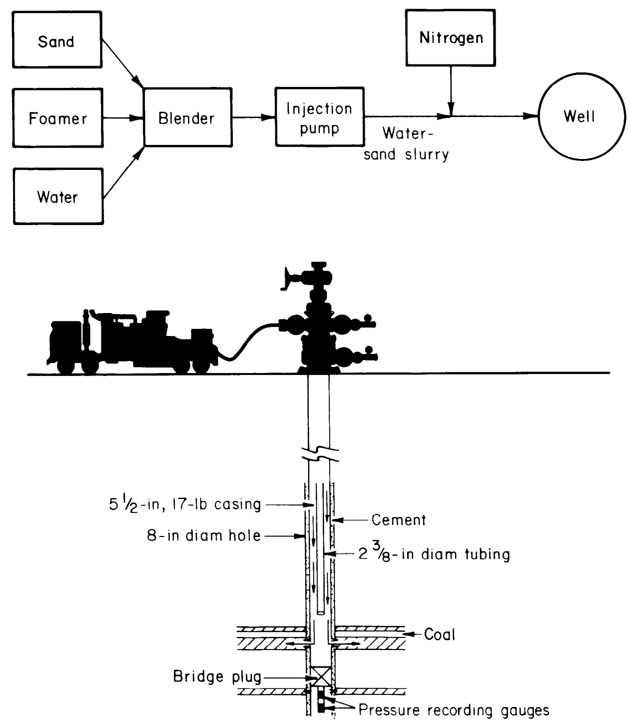

Fig. 22.5.13. Schematic of hydraulic stimulation system for vertical boreholes.

Gel treatments include some type of “breaker” ingredient, which reduces fluid viscosity. The time required after treatment for breakdown to occur depends primarily on the temperature of the coalbed. If adequate breakers are not used or if the in situ temperature of the coalbed is incorrectly estimated, stimulation fluid breakdown may not occur as desired. For example, stable gel or viscous gel residue was observed at well sites several months after stimulation treatment. The presence of gel within coalbed fractures retards fluid flow to the wellbore. Gas and water production have been low where gel has been found.

Foam, as used in coalbed stimulation, is a mixture of fresh water, nitrogen, foaming surfactant, and sand. Various types of surfactant have been used; nitrogen is nonreactive with coal and associated reservoir fluids. Using foam rather than gelled fluids has several advantages. Foam appears to decrease the amount of fluid leakoff to the coalbed. As a result, the occurrence of sandouts has been reduced by more than 50% (Lambert and Trevits, 1979). In foam treatments, problems associated with proper proportioning, mixing, injection, and breakdown of gel are avoided. Foam treatments are very clean; most stimulation fluids are removed from the coalbed in less than one day. This short exposure time limits the possibility of harmful reactions with the coalbed reservoir system, which could reduce fracture conductivity or cause formation damage. Also, site water requirements are about 60 to 80% less than for gel treatments of similar volume. The low liquid content of foam requires reduced hydraulic horsepower for injection, and smaller pumping units can be used for stimulation. Reduced onsite storage and equipment size minimize costs for site clearing operations. Injection rates for foam can be low, 10 bbl/min (1.6 m^3/min) or less. Wells have been stimulated with 50,000 gal (189 m^3) of foam using a single small pump truck normally used for borehole cementing work.

Another coalbed stimulation treatment, referred to as Kiel Frac, is a pressurization-depressurization technique (Kiel, 1976). In theory, this technique creates long fractures along two transverse directions, then props the fractures with reservoir rock material that sloughs off during the treatment. The intermittent pressurization/depressurization is said to produce small horizontal fractures perpendicular to the primary vertical fracture. Water and sand are used in Kiel Frac stimulations. No chemical mixing is required, and potential problems due to fluid formation damage or incomplete gel breakdown are avoided. The sand is supposed to filter-pack the natural joint system and/or the downward extent of vertical rock joints. This limits fracturing to the upper portion of the coalbed, and it is estimated that leakage is inhibited by the overburden.

Experience has shown that single methane drainage boreholes in virgin coalbed areas do not produce high sustained gas flow rates. This indicates that the boreholes cannot reduce coalbed pressure and decrease water saturation rapidly enough and/or to a great enough degree to maintain high flow rates. Given a water-saturated condition and an infinitely large reservoir, fluid drainage can only be effective in an area where permeability to the borehole is greater than the coalbed's natural permeability. At stimulated boreholes, this effective drainage area is directly related to the length and conductivity of the induced fractures. Outside these zones of induced permeability, significant pressure and saturation reductions must occur very slowly since water is supplied by the coalbed at nearly (very sightly below) the same rate as it can be removed (Lambert and Trevits, 1978).

Water production from a single borehole is typically high during the early phases of pumping since water originates primarily from the area of induced permeability around the borehole. Water flow, however, decreases sharply as drainage expands beyond this highly permeable zone. This lower level of water production then very slowly decreases through the remaining life of the borehole (Lambert and Trevits, 1978).

A more efficient method used to reduce pressure and decrease water saturation is to draw fluids from a more limited area of the coalbed. This can be accomplished by completing multiple-borehole patterns or by positioning boreholes close to mine workings. As coalbed fluids are removed from two or more of the boreholes, the drainage areas overlap, which allows the boreholes to remove more fluids from specific coalbed areas. By doing this, coalbed pressure is reduced to a greater degree within a shorter period of time, allowing much more gas to be released from the coal. Water saturation is also lowered to a greater degree which increases gas permeability and thereby enhances flow to the boreholes (Lambert and Trevits, 1978).

Experience with vertical boreholes for methane drainage has indicated that they can be effectively used to reduce methane emissions underground and to produce marketable quantities of gas (Lambert and Trevits, 1978). A number of studies have been conducted, and results of these tests ranged from wells producing 1000 ft^3/day (28 m^3/day) to wells producing more than 100,000 ft^3/day (2831 m^3/day) (Lambert, Trevits, and Steidl, 1980; Deul and Kim, 1988). The production rate was apparently dependent upon factors such as gas content, the efficiency of water removal, proximity of other drainage points, and coalbed characteristics. Cumulative production was related to the average production rate and the duration of the production period.

22.5.4.2 Gob Ventilation Boreholes

Methane gas can be recovered from gob by holes drilled well in advance of actual mining (Deul and Kim, 1988). A zone of tightly compacted rubble is created in areas where virtually all of the coal has been extracted and the roof strata are allowed to fall. Air does not flow easily through this rubble, and large quantities of methane gas may accumulate. On occasions when the barometer falls, the methane-laden air in the gob can expand and enter portions of the mine where a spark source is more likely to be found. Gobs may be sealed; however, these seals cannot be made completely airtight, and so they also can leak methane-laden air as the barometer drops.

Vertical boreholes into gob areas are a common methane drainage technique (Skow, Kim, and Deul, 1980). Usually, two or three holes are drilled into a longwall panel before mining, as close to the panel centerline as the surface terrain allows. The first hole is about 500 ft (152 m) from the point where mining begins; subsequent holes are spaced at 1000- to 1500-ft (304- to 457-m) intervals.

The holes are usually drilled to the coalbed, although recent research has shown that holes completed 100 ft (30 m) above the coalbed to be mined can also be effective. The borehole is cased and cemented through the lowermost water-producing zone, leaving as much open hole at the bottom as possible. The lower 200 to 300 ft (60 to 91 m) of the hole is left open. As the overburden fails, gas emitted from this area is diverted to the hole, rather than entering the mine gob. If casing is used in this part of the hole, it is cut with large slots to avoid restricting gas flow.

When the hole is intersected by the longwall panel, the overburden fractures and flow begins. Production is highest for the first few months; flows of 100 to 200 cfm (2.8 to 5.7 m^3/min) of methane gas are typical, although some holes have produced at twice this rate (Deul and Kim, 1988). Production from gob holes declines, and after a year or two, the flow may be one-tenth of the initial value. The methane concentration varies and may decline as the total airflow declines. Gob gas drainage holes, drilled on a regular basis, drain an average of 40% of the total gas from longwall panels. Drainage from room and pillar panels is less efficient, about 25%. More gas is drained from longwall panels because extraction of the coal is more complete, and the roof caves and fractures to a greater degree.

SURFACE GOB HOLE SYSTEMS. In the central Pennsylvania and northern West Virginia areas, three gob holes are normally drilled into each longwall panel. The first gob hole is drilled about 400 ft (120 m) from the start line of the panel, and the other two are equally spaced over the remaining length of the panel (Deul and Kim, 1988). In the deeper coalbeds where the overlying strata are gassier than at shallow depths, four gob holes are typically used in each panel. In southern Virginia, the first gob hole is located about 300 ft (90 m) from the start line of the panel and the other three holes are spaced at 1200 ft (365 m). In northern Alabama, gob holes are drilled about 500, 1000, 1800, and 2800 ft (150, 300, 540, 840 m) from the start line of the panel.

Gob hole completion differs in various areas because of hole size, size of casing, and depth of hole. In central Pennsylvania, the gob holes are typically 600 ft (182 m) deep. Initially, a 12-in. (304-mm) diameter hole is drilled to 500 ft (152 m). An 8-in. (203-mm) casing is set to the bottom of the hole with a bottom seal arrangement installed. The hole is then extended an additional 100 ft (30 m) with an 8-in. (203-mm) bit. This later section is not cased. The 8-in. (203-mm) diameter casing is sealed with grout at about the 500-ft (152-m) depth.

Gob holes in the northern West Virginia area are typically drilled to a depth of 900 ft (274 m) with an 8.75-in. (222-mm) diameter bit. The hole is then lined with 700 ft (213 m) of 7-in. (177-mm) diameter standard casing and 200 ft (60 m) of 7-in. (177-mm) diameter slotted casing. A formation packer, which is installed above the slotted casing, allows the 700-ft (213-m) length of casing to be grouted in place. The slotted casing that hangs below the formation packer extends to within 20 to 90 ft (6 to 27 m) of the coalbed.

In southern Virginia, average overburden is about 1800 ft (540 m). Gob holes are typically drilled to within 20 ft (6 m) of the coalbed with a 12.25-in. (311-mm) diameter bit. The hole is then lined with 1600 ft (487 m) of 9-in. (228-mm) diameter casing and 180 ft (54 m) of slotted casing. A formation packer, which is installed above the slotted casing, allows the 1600 ft (487 m) of casing to be grouted in place.

Gob holes are about 2050 ft (624 m) deep in northern Alabama. Initially, a 17.5-in. (444-mm) diameter hole is drilled to 150 ft (45 m) and a 13.5-in. (342-mm) diameter casing is grouted in place. Subsequently, the hole is extended about 1880 ft (573 m) to within 20 ft (6 m) of the coalbed using a 12.25-in. (311-mm) diameter bit. The hole is then lined with 1450 ft (441 m) of 9.5-in. (241-mm) diameter slotted casing. A formation packer is installed above the slotted casing, and the top 1450 ft (441 m) of casing is grouted in place.

All gob holes are equipped with exhausters to provide the partial vacuum to draw the gob gas to the surface. The size and capacity of the exhausters are dependent on the relative gassiness of the strata in the study area. The surface venting facilities are fenced and equipped with lightning protectors and flame arresters.

CASE HISTORY 3: POCAHONTAS No. 3 COALBED. The site chosen for this study was approximately 2260 ft (688 m) below the surface in the Pocahontas No. 3 Coalbed in Buchanan County, VA (Maksimovic and Kissell, 1980). The coalbed in this region is approximately 51 in. (1295 mm) thick and has a small shale parting approximately 15 in. (381 mm) below the top of the bed. The horizon above the coal is sandy shale with sandstone streaks and partings. Below the coal is a 10-in. (254-mm) zone of quartzite, and below this is a massive micaceous sandstone.

An 8-3/4-in. (222-mm) diameter gas drainage hole was drilled to a level 4 ft (1.2 m) below the coal. A 7-in. (177-mm) OD casing was placed in this hole and pressure grouted from the surface to within 139 ft (42 m) of the coal. This left an annular space between the lower slotted section of casing and the hole. The grout prevented the entry of water and gas from the upper horizons but permitted the passage of gas through the slots in the lower 120 ft (36 m) of casing and through the end of the pipe.

The gas drainage hole was completed in the spring; mining of the panel was initiated approximately 3-1/2 months later. No significant flow of methane was detected at the top of the casing until after the borehole was intersected by mining.

The longwall panel was 370 ft (112 m) wide and 3920 ft (1194 m) long. A bidirectional plow was used for extraction and the roof was supported by self-advancing, double-frame, heavy-duty hydraulic units; each unit consisted of four double-acting 140-ton (127-t) jacks. The face equipment required a minimum space of 11 ft (3 m) between the coal face and the gob. A single split of air was used to ventilate the longwall panel. The airflow rate past the head end of the face ranged from 77,000 to 82,000 cfm (2180 to 2321 m^3/min) during the study period. Approximately 20% of this air reached the tail end of the face, with the remainder flowing past the roof support units into the gob.

The effect of the borehole was quite evident. Not only was there a decrease in the methane flow rate found underground, but the short-term fluctuations in flow rate were diminished markedly. Presumably, much of the gas emitted by the overlying strata passed through the borehole and was thus prevented from being forced into the mine as the immediate roof and adjacent strata collapsed. Further, there was an associated marked decrease in the maximum measured underground methane concentration. The maximum value recorded before borehole intersection was 1.9% and the maximum value after intersection was 1.3% in approximately 100,000 cfm (2831 m^3/min) of air; this represents a decrease of slightly over 25%, approximately the proportion of methane carried away by the borehole under free-flow conditions. However, an ever larger portion of the gas was removed by the use of an exhaust fan.

The methane concentration of the gas discharged by the borehole during the study period ranged from 77 to 100%. Because the concentration tended to decrease when the exhaust fan (exhauster) was turned on, it was used only intermittently to keep the methane content of the gas flow from the borehole above about 30%. Although the exhauster increased the methane flow rate through the borehole from about 25 to 35% it also entrained some air.

Because gob gas boreholes divert methane from the mine returns, the mining rate can be increased with a particular ventilation system without exceeding the legal methane limit. Higher mining rates were measured after borehole intersection occurred.

22.5.4.3 Horizontal Boreholes from Shaft Bottoms

Construction of shafts for use in methane drainage requires no unusual equipment or techniques. The shaft is drilled to the coalbed and lined. A plate divides the shaft into intake and exhaust sides. Outside the lining, there are two 4-in. (101-mm) diameter pipelines to carry the gas to the surface and one 3-in. (76-mm) pipeline for a power drop. Hoisting facilities and an exhaust fan are located on the surface.

When the shaft is completed, the area within the coalbed is enlarged to provide working space and to accommodate drilling equipment, power supply, stuffing box, water trap, and gas collector. Long horizontal drainage holes are then drilled around the periphery of the shaft. After the holes are drilled, they are connected to a gas-water separator. Since water fills the fractures in the coalbed, it must be removed before gas can flow freely. Generally, as the coalbed is dewatered, the gas flow rate increases (Fields, Cervik, and Goodman, 1976). After the methane is piped to the surface, it can be vented or collected and used commercially. The surface installation depends on the end use. If the gas is to be vented, a stack with a check valve and flame arrester is needed. If the gas is to be added to a commercial gas line, a compressor must be typically used, but other cleanup equipment is usually not needed. The US Bureau of Mines has demonstrated the use of shafts for coalbed gas drainage at the Multipurpose Borehole and Honey Run shaft of the Federal No. 2 mine and at the Maple Meadow mine.

CASE HISTORY 4: FEDERAL No. 2 MINE. In the initial demonstration of the use of shafts for methane drainage in advance of mining, a 4-ft (1.2-m) diameter borehole (designated the “multipurpose borehole” because of its many potential uses) was drilled to virgin coal on Eastern Associated Coal Corporation’s Federal No. 2 mine property, Monongalia County, WV (Fig. 22.5.14) (Fields, Perry, and Deul, 1975). First, an 84-in. (2133-mm) diameter hole was drilled 50 ft (15 m) to bedrock using a rig set up especially for shaft drilling. The surface hole was cased with 74-in. (1879-mm) ID rolled-and-welded casing cemented in place. A 72-in. (1828-mm) diameter hole was then

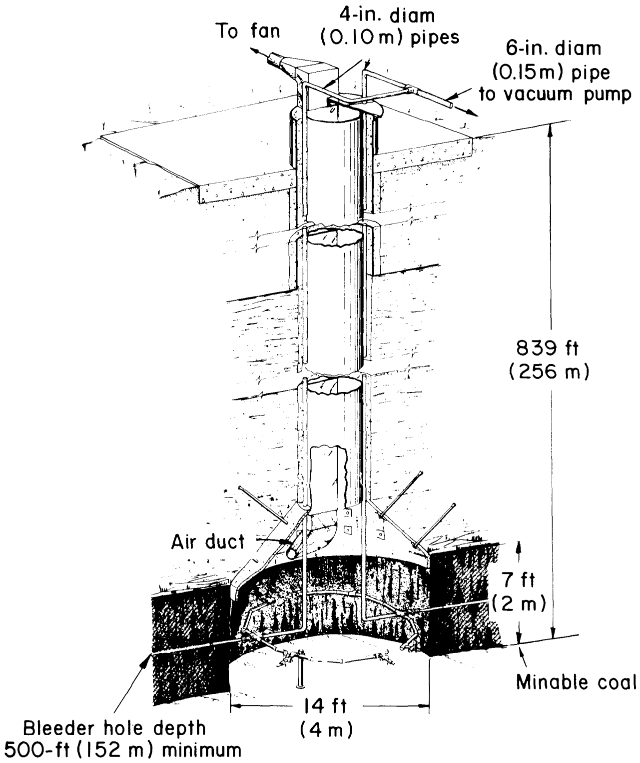

Fig. 22.5.14. Schematic of multipurpose borehole for methane drainage.

drilled to the bottom of the Pittsburgh Coalbed, a depth of 839 ft (255 m), using reverse mud- and air-assist circulation to remove cuttings. After the hole was drilled, a 48-in. (1219-mm) ID casing was grouted in place to a depth of 826 ft (251 m), 8 1/2 ft (2.6 m) above the top of the Pittsburgh Coalbed. The casing had a plate welded in it to divide it into intake and exhaust sides. Outside the casing, two 4-in. (101-mm) ID pipes were tack-welded to be used as gas exhaust lines along with two 3-in. (76-mm) pipes, one for water removal and the other as a power drop. An 8000-lb (3628-kg) capacity, 30-ft (9-m) high tripod headframe and a 5000-cfm (141-m^3/min) exhaust fan were then installed. Within the coalbed, the borehole was enlarged to a diameter of 14 ft (4 m). Seven gas drainage holes, ranging in length from 500 to 850 ft (152 to 259 m) and a 199-ft (60-m) in situ pressure-monitoring hole were drilled from the bottom of the shaft.

The in situ pressure, measured in the monitoring hole, was 203 psi (1399 kPa) just before all the holes went on production. Within 10 days of the initial gas and water production, the pressure had dropped to 18 psi (124 kPa). During the next 4 yrs, the pressure declined gradually to 5.5 psi (38 kPa).

Initial gas flow was 1.2 million ft^3/day (34,000 m^3/day). Within 90 days, total gas flow from all holes declined to 410,000 ft^3/day (11,600 m^3/day). The decline in gas flow paralleled the exponential decrease in waterflow rate. After the immediate area around the hole had been dewatered, gas flow gradually increased over the next year to a maximum of 700,000 ft^3/day (19,800 m^3/day). Gas flow averaged 600,000 ft^3/day (16,900 m^3/day) over the period of 3.5 yrs before the shaft was flooded. After the borehole was drained and gas production resumed, the initial flow rate was 600,000 ft^3/day (16,900 m^3/day), which gradually declined to a level of 370,000 ft^3/day (10,400 m^3/day). Total gas production for approximately 2500 days was 1.10 billion ft^3 (31 Mm^3).

22.5.4.4 Directionally Drilled Slant Holes

Directional drilling for coalbed gas drainage is an outgrowth of the techniques developed for methane drainage. This technique has been modified from horizontal holes drilled underground, either in mines or from the bottom of shafts, and surface vertical boreholes. Horizontal holes were found to have the advantages of relatively low drilling costs and the ability to intersect the coalbed cleat or fracture system, thus increasing permeability to gas flow. However, this method required underground access and facilities that often interfered with the mining cycle. The requirement of access to the coal occasionally limited the value of the horizontal holes because of the limited time and/or distance they could be drilled ahead of mining. Hydraulically stimulated vertical holes eliminated the latter problems but had the disadvantages of requiring large numbers of surface sites, higher costs, and production and maintenance problems.

The concept of directionally drilled degasification holes was originally considered by the US Bureau of Mines as a means of combining the best elements of the surface vertical borehole and underground horizontal drilling techniques (Diamond, Oyler, and Fields, 1977). From a single surface site, a vertical or near-vertical well could be progressively deviated to intersect a coalbed horizontally (Fig. 22.5.15). Several horizontal gas collection holes could then be sidetracked from the original well path in the coalbed. It would also be possible to orient the drill rig in several other directions on the same surface site and drill a succession of directional degasification holes. The cost of site preparation and production facilities would be significantly reduced by having the entire gas flow from a large coalbed area centralized at one surface location.

Thus directional drilling would eliminate the need for underground gas piping systems, would make entry into mines unnecessary, and could remove gas from large areas of coal far ahead of mining. The directional degasification hole could be used at sites where conventional horizontal or vertical holes are not feasible and at sites that are unsatisfactory for other types of methane drainage. Before this technique can be used routinely, problems in bit control, down-hole surveying, and removing water from the horizontal portion of the hole must be solved. The present high cost of the method and the need for a large number of exploratory coreholes may hamper widespread use of directional drilling for coalbed degasification.

CASE HISTORY 5: UPPER FREEPORT COALBED. A hydraulically operated, diesel-powered experimental rig, capable of working at a maximum angle of 10° from the vertical, was used to drill a directionally controlled hole to the Upper Freeport Coalbed horizon in Greene County, PA (Diamond, Oyler, and Fields, 1977). This capability is required in directional drilling to shallow seams (less than 1000 ft or 304 m of cover). At the Greene County site, the cover was 928 ft (282 m) and the rig was tilted to 5°. In the original plan, the hole was to be drilled in several stages. First, a 3-in. (76-mm) diameter pilot hole was to be drilled to the coalbed using BQ drill rod and a 2-3/8-in. (60-mm) diameter Dyna-Drill with a bent sub. Then the hole was to be reamed to 8-3/4-in. (222-mm) diameter, either by the use of a stringer bit or by overreaming the BQ drill string. A 5-1/2-in. (139-mm) diameter casing to the coalbed was then to be cemented in place. Finally, multiple horizontal holes were then drilled in the coalbed from the end of the casing using the Dyna-Drill and BQ drill rod. Owing to discontinuous coal, only the pilot hole was completed in this study.

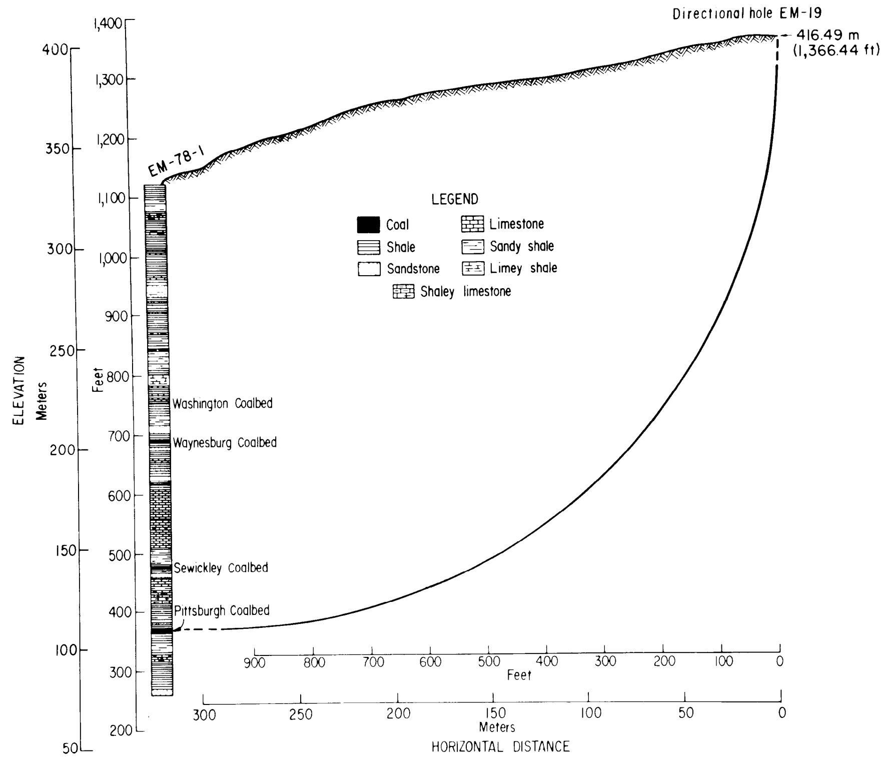

Fig. 22.5.15. Schematic of directionally drilled methane drainage borehole.

At the Greene County site, the 3-in. (76-mm) pilot hole was successfully drilled to a nearby horizontal position at the Upper Freeport Coalbed horizon. The first intercept of the target encountered less than 0.5 ft (0.15 m) of coal. This intercept was only 750 ft (228 m) from an exploratory corehole containing nearly 7 ft (2 m) of Upper Freeport coal. Five additional intercepts of the coalbed horizon encountered 0 to 4.3 ft (0 to 1.3 m) of coal. After approximately 1500 ft (457 m) of horizontal drilling in the target zone, it was decided that sufficient continuous coal thicknesses did not exist at the site to warrant further drilling.

CASE HISTORY 6: PITTSBURGH COALBED. In another test of directional drilling technology, three horizontal drainage holes were drilled in the Pittsburgh Coalbed from a directional surface borehole (Oyler and Diamond, 1982). A 3-in. (76-mm) diameter pilot hole was drilled from the surface to intercept the coalbed horizontally. The directional borehole was located at the top of a ridge to allow sufficient overburden above the coalbed for a circular arc of about 6° per 100 ft (30 m). The vertical distance to the coalbed was 1000 ft (304 m). Actual drilled distance of the directional pilot hole well path was 1652 ft (503 m). The directional drilling was accomplished with a 2-3/8-in. (60-mm) diameter Dyna-Drill down-hole motor and BQ wireline drill rod as the drill pipe. The rate and direction of angle were controlled by using various bent housings. Control of the well path was within specifications, and no remedial drilling was required to complete the pilot hole to the coalbed.

After the pilot hole was completed, a string of BCQ drill rod with a string of 1-in. (25-mm) diameter pipe inside for added stiffness was placed in the hole to be used as a guide string for overreaming bits. A gamma ray log was also run to confirm that the pilot hole had reached the top of the Pittsburgh Coalbed. Reaming operations used 3-1/2-in. (88-mm) diameter pipe with an 8-3/4-in. (222-mm) drillbit.

A vertical borehole was drilled near the coalbed intercept and equipped with a downhole plunger pump for dewatering. The vertical borehole was foam-stimulated to create a path for water to flow from the horizontal holes into the vertical dewatering borehole. Seven other vertical boreholes were planned in order to monitor the progress of gas drainage by observing the changes in hydrostatic pressure at each location. The horizontal portion of the directional hole was drilled to a measured depth of 1732 ft (527 m), slightly past the vertical dewatering hole.

In the initial horizontal drilling, it was necessary to drop through approximately 2 ft (0.6 m) of roof rock to enter the main bench of the Pittsburgh Coalbed. Apparently, the pilot hole had been drilled through the Pittsburgh rider coal into the roof shale above the Pittsburgh Coalbed, and instead of going into the Pittsburgh Coal, the hole had gone back into the rider coal. This error proved to be a major problem for two reasons. First, the vertical dewatering borehole and directional borehole had been placed so that fractures from the stimulation treatment would cross the horizontal hole just beyond the end of the casing. Since the horizontal hole was possibly in roof rock near the anticipated intercept of the stimulated fracture with the horizontal hole, direct communication may not have been achieved. Second, the initial portion of each horizontal hole was in rock that had a tendency to cave or slough, thus blocking the holes.

Continuous drilling of the first horizontal leg, hole 1, used clear water as the drilling fluid to prevent mud additives from plugging the coalbed fracture system. However, the removal of drill cuttings was more difficult because of the low viscosity of water. Fluid loss was also a problem, which could not be solved without the risk of damaging the coalbed's natural fracture permeability. Fluid losses as high as 15,000 gal/day (56 m^3/day) were experienced before hole 1 was completed. The first horizontal leg was completed at 3362-ft (1024-m) measured depth, when it was impossible to reenter the horizontal hole after a pipe trip. The hole had a total horizontal length of 1767 ft (538 m) beyond the end of the casing and was drilled in 26 working days, for an average rate of 67 ft/day (20 m/day).

Leg 2 was completed in 36 working days at a total measured depth of 4802 ft (1463 m). This was the longest horizontal hole drilled from the directional well, reaching 3207 ft (977 m) from the end of the casing. The average daily footage was 89 ft (27 m).

Leg 3 was drilled to a measured depth of 4588 ft (1398 m), a horizontal length of 2993 ft (912 m) from the end of the casing. Leg 3 was discontinued because it could not be reentered, owing to the apparent sloughing of the shale section at the bottom of the casing.

The final drilling operation performed on the slant hole was the underreaming of a horizontal sump 8-1/2 in. (215-mm) diameter from the bottom of the casing to a measured depth of 1742 ft (530 m) along the center leg, hole 1.

After drilling was completed, water production began at a pump capacity of 120 bbl/day (19 m^3/day), and intermittent gas flows were measured up to a rate of 1200 ft^3/day (33 m^3/day). After several days, water production declined drastically without any increase in gas production. It was believed that a barrier to waterflow between the slant hole and the dewatering hole was responsible. Several attempts were made to increase water production by flushing out both the slant and dewatering holes and to extend the dewatering hole's fractures.

Drilling the directional borehole to the Pittsburgh Coalbed demonstrated that it is possible to drill a directional pilot hole to a target coalbed. To determine that the hole actually was in the target coalbed required drilling an additional hole completely through the target coalbed. This was necessary because the accuracy of current directional well surveying is at best 2 to 3 ft (0.6 to 0.9 m) vertically.

The gas production from the directionally drilled borehole was insignificant due to the previously discussed problems. Technically, the drilling operation was successful, but the concept of this method as a methane drainage system remains unproven.

22.5.5 FUTURE TRENDS

Advancements in methane control techniques have been achieved to the extent that a number of different approaches exist for solving gas emission problems. However, additional studies are currently being conducted to address recent technological changes within the industry and within the scientific community. For example, short-radius directional drilling, which has been pioneered by the petroleum industry, is being proposed as a cost-effective method for methane recovery from vertically drilled boreholes. This system would permit a number of horizontal boreholes to be drilled within a coalbed from a single vertical borehole and would have the advantages of the slant hole drainage system without the disadvantages of dewatering problems and extended drilling lengths for the slant hole portion of the system. In addition, the short-radius system could easily be adapted to allow drainage from a number of different horizons such as multiple coalbed deposits.

One trend in the underground coal mining industry is the increased use of larger-dimension longwall panels, as the average panel currently exceeds 650 ft (195 m). This represents an increase of about 20% over the last five years. The implications of this trend on methane drainage systems primarily focus on the expansion of the gob area, Larger gob areas create additional disturbed ground surrounding the longwall panel, which increases the number of methane gas sources exposed to the mine environment and increases the total gas emissions associated with longwall mining. Gob gas drainage by vertical boreholes must become more scientific in nature as more gas must be handled by the drainage system. A basic understanding of the gob gas reservoir system is required to permit proper sizing and siting of the vertical boreholes.

An additional impact on methane control technology from increasing longwall panel dimensions is the importance of the time dimension on installing and operating the drainage systems. Larger longwall panels along with shorter time intervals between panel outline and retreat require methane drainage systems that operate at high efficiency. The drainage boreholes must recover large volumes of methane gas in relatively short periods of time. Drainage enhancement systems must be developed to achieve this goal.

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Chapter 22.6
IN SITU GASIFICATION AND COMBUSTION OF COAL

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22.6.1 INTRODUCTION

Underground coal gasification (UCG) and *in situ coal combustion* (ICC) are processes that can recover the energy of coal seams without the extensive use of traditional mining operations, either underground or surface. In UCG, the primary product brought from underground is a combustible fuel gas of low to intermediate heating (calorific) content, 100 to 300 Btu/standard cubic foot (scf) (3700 to 11,200 kJ/std m^3). In ICC, the primary product brought from underground is hot fully cornbusted flue gas, 1100 to 1800°F (600 to 1000°C), whose sensible heat contains most of the heating value of the coal, 5000 to 13,000 Btu/lb (11,600 to 30,200 kJ/kg).

Both UCG and ICC are particularly applicable to coal deposits that are not economically or technically feasible to mine by conventional methods because of seam quality or quantity, depth, dip, strata integrity, overburden thickness, etc. Actually, the two processes are corollary to each other in that those seam and site conditions that are favorable for UCG (e.g., thick, deep, partially wet seams) often will not promote in situ combustion, whereas conditions that are favorable for ICC (e.g., shallow, rubblized, dry seams) will not promote underground coal gasification.

UCG is the older of the two in situ coal conversion concepts, dating back over 130 years. It has received considerable attention worldwide during this century, particularly by the Soviet Union, and more recently by the United States.[1] Technical specifications and operational procedures for a number of UCG operations have been developed. They are generally site specific; however, the large background of technical data and experience that are available do indicate that the process is ready for commercial use in the United States, at least in the thick subbituminous and lignite seams located in the West.[2] The greater part of the chapter will focus on this aspect of in situ coal recovery.

ICC is a relatively new concept and was first proposed by the US Bureau of Mines (USBM) in 1974 (Chaiken, 1974). The subject of a much smaller development effort than UCG, it has been submitted to only one underground test as part of the development of burnout control, a USBM patented process for controlling fires on abandoned mined lands (Chaiken, 1980, 1983). Although further engineering and field trials will be required before burnout control can be commercialized as an ICC process, it is useful to describe the process here because of its potential application to the relatively thin, shallow bituminous seams (and abandoned mines) that are so prevalent in the eastern United States.

In the following segments, a description of some of the basic physics and chemistry of UCG is given that will allow the mining engineer more fully to understand the site assessment and engineering requirements for a UCG operation, and hopefully to enable him/her to extrapolate those requirements to various sites and coal conditions. This is followed by a discussion of applicable resources and utilization of the fuel gases that can be produced and a description of several types of UCG operations of potential commercial significance and their environmental problems.

Finally, a similar but much shortened approach is taken in describing burnout control as an in situ combustion process.

22.6.2 UNDERGROUND COAL GASIFICATION

22.6.2.1 Basic Processes in Underground Gasification

Gasification Reaction. Coal is gasified underground by injecting air or oxygen/steam through boreholes into a reaction zone formed in the coal seam. The hot gaseous reaction products are forced to migrate though the coal seam to an exit borehole where they are drawn from underground to the surface. Upon reaching the surface, the gases can be cleaned for direct use as a low-to-medium Btu gas, for upgrading to a substitute natural gas (SNG), or for some other chemical feedstock (SynGas). In essence, UCG involves achieving a spatially and thermally distributed reaction zone in the coal seam that consists of overlapping regions of coal oxidation, coal reduction and coal pyrolysis.

This is shown in Fig. 22.6.1, which depicts the basic chemical reactions and temperatures defining the distributed reaction zone. Entering air causes the coal to burn, an exothermic process releasing heat and consuming oxygen. The hot oxidized product gases migrate toward the exit borehole, passing in turn through a reducing region where, in the absence of oxygen, they are converted to combustible gases by a number of endothermic reactions which absorb heat. The somewhat cooler but still hot combustible gases continue to migrate toward the exit borehole, where they cool further as they pass through a coal pyrolysis region. In this region, the coal is heated to drive off its volatile matter, that is, tars, hydrocarbons, and other gases both combustible and noncombustible. The entire gas mixture then exits the seam through a production borehole.

Water plays an important role throughout the entire reaction zone. It is responsible for much of the hydrogen produced in the reduction region, being reduced by carbon in an endothermic reaction:

[1] The history of UCG prior to 1970 has been reviewed quite extensively in several references (Elder, 1963; Little, 1971; Wang et al., 1973). References Edgar and Gregg (1981) and Olness and Gregg (1977) are devoted to the extensive program carried out in the Soviet Union, and reference Gibb et al., 1964 is devoted to the UCG program (1949–1956) carried out in Great Britain. Since 1975, almost all work on UCG, worldwide, has been reported at openly attended annual symposia sponsored by the US Government; the proceedings of which (except for the first meeting in 1975) have been published by the Energy Research and Development Administration (ERDA) and its successor, the Department of Energy (DOE) (Anon., 1976–1988).

[2] It is useful to note that six UCG stations have been operated in the USSR. Starting in 1933, experiments were carried out in the brown coal of the Moscow Basin. Stations operated as research facilities, and some eventually developed into small industrial producers. However, the Angrenskaya station in the lignite fields near Tashkent was apparently established initially in 1952 as a commercial venture. There are currently two other commercial-scale UCG stations operating, one at Shatky (Moscow Basin lignite) and one at Yuzhno-Abinsk (steeply dipping bituminous seams) (Stephens, Hill, and Borg, 1984).

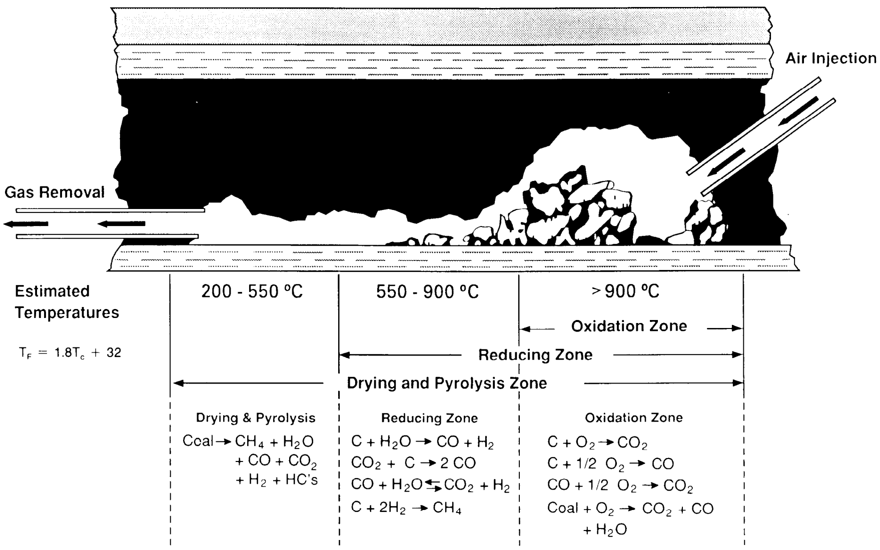


Fig. 22.6.1. Schematic representation of an underground coal gasification reaction zone (Anon., 1977).

$$C + H_2O \rightarrow CO + H_2 - 31.4 \text{ kcal/mol} \qquad (22.6.1)$$

This water reduction reaction enhances the heating value of the product gases. However, further reduction of water by CO in the water-gas shift reaction:

$$CO + H_2O \leftrightarrow CO_2 + H_2 + 9.4 \text{ kcal/mol} \qquad (22.6.2)$$

will detract somewhat from the heating value of the product gas. Here the hydrogen is formed at the expense of carbon monoxide, a somewhat more more energetic fuel gas, and also at the expense of forming additional CO_2 diluent in the product gas. Carbon dioxide is much more difficult to remove from the final product gas during cleanup than is H_2O.

While some water is an important reactant in the underground reaction zone, excess water will behave as an evaporating heat sink, moderating the temperature rates of the chemical reactions, particularly the char gasification reactions in the reducing zone (see Fig. 22.6.1). These reactions generally require temperatures in excess of 1100°F (600°C). With air injection, there will generally be an optimum water/air/coal ratio that yields a maximum heating value for the product gases (Edgar and Gregg, 1981). While some of the water for this ratio will be formed by oxidation of the hydrogen component of the coal, the optimum condition usually requires additional water supplied either by the moisture already in the seam itself or by intrusion from the surrounding strata. Control of this additional water is possible, in part, by varying the pressure of the injection air and in part by increasing the rate of air injection, and hence the gasification rate. However, increased injection pressure will tend to increase gas loss from the underground reaction zone to the surrounding strata.

An example of this is shown in Fig. 22.6.2, which contains data from an experimental Soviet underground gas generator. At constant air injection rate, a linear relationship was observed between gas loss V(loss) and the difference between the square of the absolute pressure P_{av} (averaged over the underground reaction zone) and the square of the atmospheric pressure P_{at}, that is,

$$V(\text{loss}) = C\,[(P_{av})^2 - (P_{at})^2] \qquad (22.6.3)$$

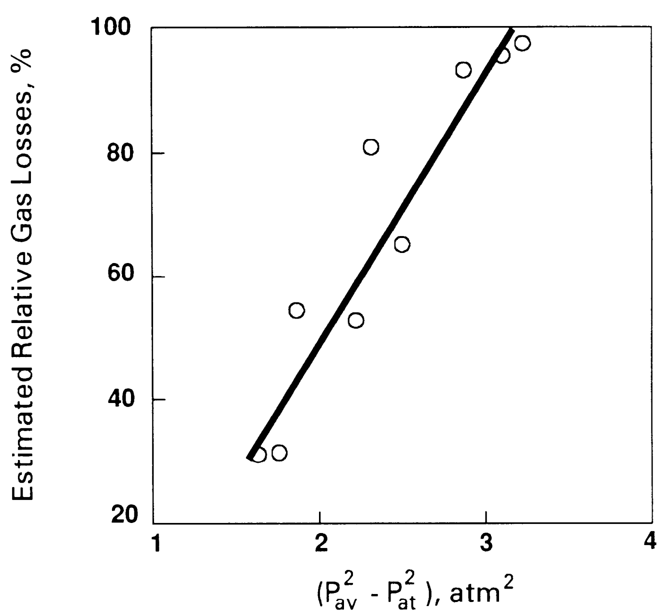


Fig. 22.6.2. Dependence of the relative value of the gas losses on the gas pressure in an underground gas generator (Pitkin, 1960).

The slope C of the straight line in Fig. 22.6.2 is undoubtedly site specific, depending on the permeability of the strata.

Fig. 22.6.3 shows the combined effect of gasification and water intrusion rates on the heating value of the product gases (again based on Soviet data). Here it is seen that excess water alters the heating value of the final product gases—most likely by decreasing the temperature and rate of the char reactions and hence altering the gas composition. This same effect of lower temperature leading to decreased heating values is also reflected in the data shown in Fig. 22.6.4, which show that thin seams, as well as increased water intrusion, decrease the heating value of the final product gas. Heat loss from an underground reaction zone to surrounding rock strata would be expected to vary inversely with seam thickness. Hence, in thin seams, UCG temper-

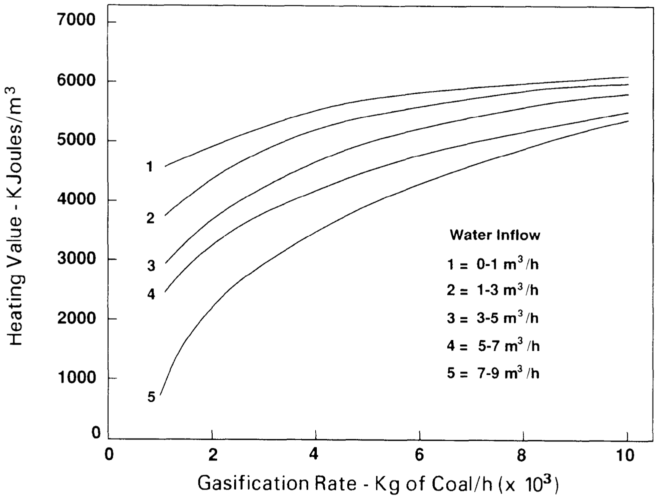


Fig. 22.6.3. Effect of gasification rate on heating value at various water injection rates (Edgar and Gregg, 1981). Conversion factors: 1 lb = 0.4536 kg, 1 Btu/ft^3 = 37.26 kJ/m^3, 1 ft^3 = 0.02832 m^3.

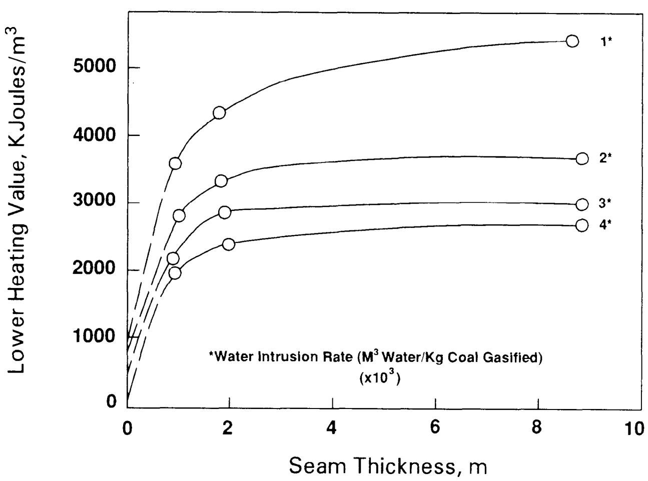


Fig. 22.6.4. Effect of seam thickness and water intrusion on UCG heating values (Anon., 1977). Conversion factors: 1 ft = 0.3048 m, 1 Btu/ft^3 = 37.26 kJ/m^3.

atures may be lower, and chemical reaction rates slower. In Fig. 22.6.4, taken from Soviet studies with brown coals, the heating value of the product gas deteriorates rapidly for those seams less than 6 ft (1.8 m) thick, and gasification can not be sustained for seams less than 3 ft (0.9 m) thick.

In practice, the migrating gases can be subject to high heat loss even in thick seams. Referring back to Fig. 22.6.1, it is easily visualized that the actual flow path of the UCG gases may be directed close to the top of the seam rather than through the main body of the seam. This would tend to enhance heat losses from the reaction zone and could lower product gas heating values. Avoidance of this type of heat loss requires locating the gasification channel(s) properly in the coal seam. This point is discussed below in some detail.

A related point, but one leading to a somewhat different problem, occurs when the path(s) of the underground gas flow enables oxygen from the injected air to reach and mix with the product gases directly in the vicinity of the exit borehole. This results in burning of the product gas prior to it being exhausted from underground, thereby decreasing the heating value of the final product gas, as well as increasing the temperature of production well. The "breakthrough" or bypass of air into the product gas stream often signifies the end of the useful life for a UCG reaction zone. This is seen in Fig. 22.6.5, which depicts heating values and exhaust temperature as a function of time for a UCG reaction zone formed between a single set of injection/production boreholes. The data come from the Hanna II experiment carried out by the USBM.[3]

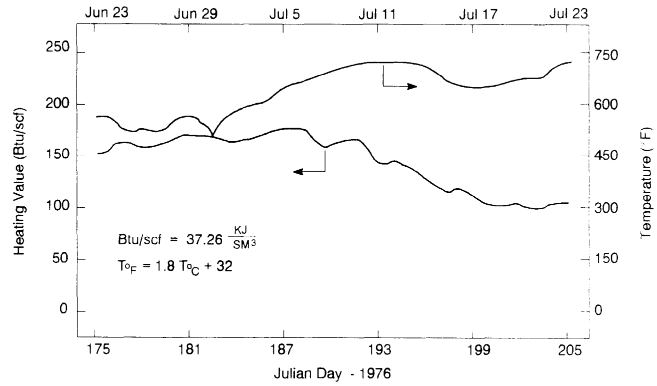


Fig. 22.6.5. Heating values and wellhead product gas temperature at Hanna II field trial (Schrider et al., 1976).

Table 22.6.1 shows some typical gas compositions and heating value data obtained in past UCG experiments (US Bureau of Mines at Gorgas, AL) as compared with results from surface gasifiers. The Gorgas data for UCG in a bituminous seam do not represent the best case for UCG (typically, 80 to 150 Btu/scf, or 3000 to 5600 kJ/std m^3, from subbituminous coal); however, it is seen that the UCG product is similar in composition to the surface gasifier product. The higher CO_2 values in the UCG product probably reflect the greater degree of "breakthrough" or bypass that was experienced in those field trials. More recent higher-quality UCG gases compare quite favorably with Lurgi producer gas (see 22.6.2.3).

Preparation of the Coal Seam. It was described above that proper reaction zone geometry and proper migration of the gas through the reaction zone play a very significant role in a successful underground gasification process. The permeability of the seam must be such that sufficient gas flow occurs, and along paths that ensure (1) minimum gas and heat loss from the seam, (2) a maximum functional life before oxygen breakthrough takes place, and (3) achievement of optimum chemical reactions and a consistent exiting gas composition. Even with an ideal thick seam having no faults, a low level of water influx, and a structurally sound roof strata, the achievement of a successful UCG requires careful seam preparations to obtain a suitable reaction zone.

Early UCG trials were carried out in roadways and drifts prepared by underground mining. The reaction zones established

[3] In 1972, the US Bureau of Mines/Laramie Energy Research Center (USBM/LERC) initiated a second major US developmental effort on UCG at a subbituminous site near Hanna, WY. These efforts followed more closely the Soviet approach than did the Bureau's first experiments in the 1950s at Gorgas, AL (Elder, 1963). In 1975, the UCG program along with all of the Bureau's Energy Research Centers were transferred to the newly formed Energy Research and Development Administration (ERDA), which subsequently (in 1977) was incorporated into the US Department of Energy (DOE).

Table 22.6.1. Data Comparing UCG and Surface Gasifier Products

	UCG			Gasifier		
	Air			Air		
Oxidant	Normal	N_2-free	Oxygen	Normal	N_2-free	Oxygen
CO_2	10.5	32.2	47.8	6.0	13.3	10.1
Illumin.	0.3	0.9	0.3	1.0	2.2	0.5
O_2	0.9	2.8	0.2	—	—	—
H_2	8.4	25.8	24.5	13.0	28.9	35.5
CO	10.7	32.8	21.2	24.0	53.3	52.8
CH_4	1.8	5.5	4.1	1.7	3.8	9.2
N_2	67.4	0	1.9	55.0	0	1.5
Btu/scf	86	—	195	137	—	300

Source: Adapted from Elder, 1963.

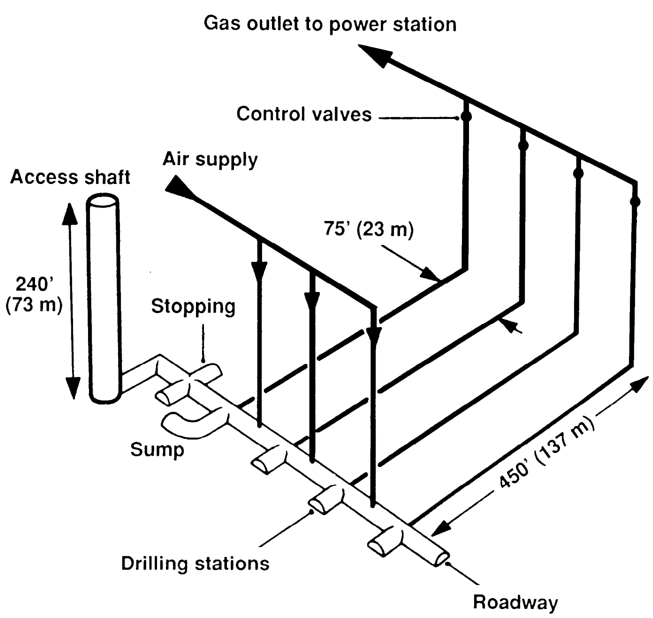


Fig. 22.6.6. Schematic layout of the P5 field trial at Newman-Spinney (Thompson, Mann, and Williams, 1976).

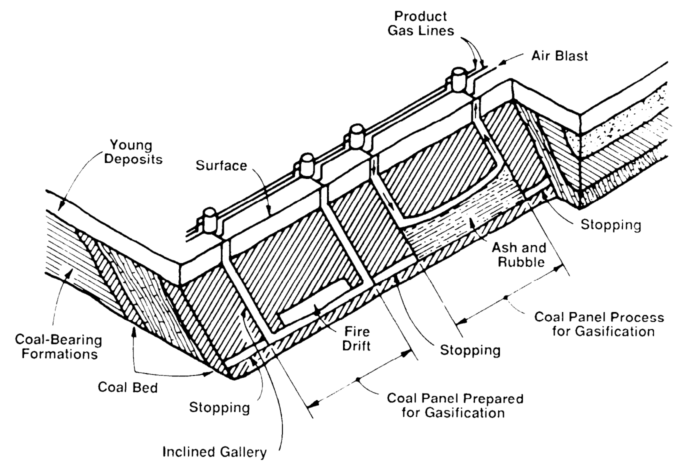


Fig. 22.6.7. Stream method for steeply dipping seams (Elder, 1963).

in the relatively large cross-sectional-area roadways generally did not allow for sufficient interaction between the solid coal and the injected air, thereby leading to early breakthrough of the air. These designs were soon followed by the use of horizontal boreholes drilled from a mined gallery in the case of a horizontal seam (Fig. 22.6.6), or slant boreholes from the surface into a mined gallery in the case of a steeply dipping bed (Fig. 22.6.7). Gasification, fed by injected air and coal progressively spalling from the walls of the channel, would continue until breakthrough occurred. This is depicted schematically in Fig. 22.6.8 for a horizontal seam. The initial injection of air (or oxygen/steam) results in a reaction zone similar to that in Fig. 22.6.1. As coal spalls and is consumed, the oxidation region grows, extending to the roof and floor of the seam, and eventually expands as a reaction wave forward to the production hole, and to a lesser degree, laterally to the sides. The Newman-Spinney P-5 trial (Gibb et al., 1964) is a good example of this UCG process. Fig. 22.6.9 shows plan views of the 2.5-ft (0.8-m) seam before and after the 118 days of UCG. In this case, the final geometry of the reaction zone was verified by excavation.

Progressive UCG along a channel in a steeply dipping seam takes on a somewhat different form as depicted in Fig. 22.6.7. Coal along the horizontal gallery is gasified, leading to a continuing fall of ash and roof coal into the gallery. In this way, the linear reaction zone travels upward through the seam toward the surface outcrop.

The use of these preliminary underground mining methods in preparing a coal seam for UCG had varying degrees of success in several of the early experiments, but their development was discontinued in the late 1950s in favor of surface preparation methods only. This decision was prompted by the desire to have no personnel working underground, by advancements in directional drilling, and by efforts to enhance future application to very deep seams. However, as pointed out in a National Coal Board appraisal (Thompson, Mann, and Williams, 1976), preliminary mining methods can have some advantages, and interest in them will probably be renewed as additional commercial scale UCG plants are developed. Since the thrust of all current UCG projects is to use surface borehole methods exclusively, it is these methods that are emphasized in this segment.

A vertical or slant borehole from the surface is the quickest and simplest way of reaching a coal seam, so it is not difficult to establish an injection borehole and a production borehole by

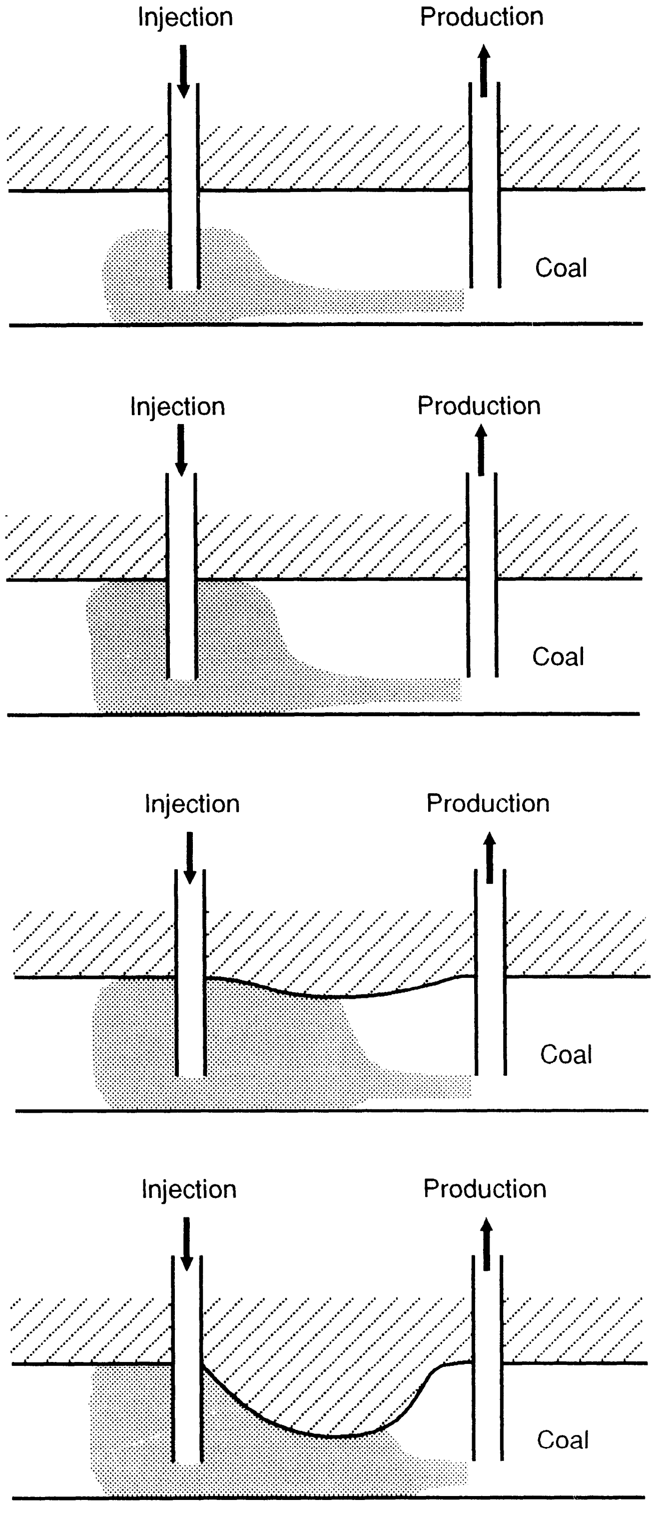


Fig. 22.6.8. Idealized progression of UCG in a horizontal seam (percolation or filtration method).

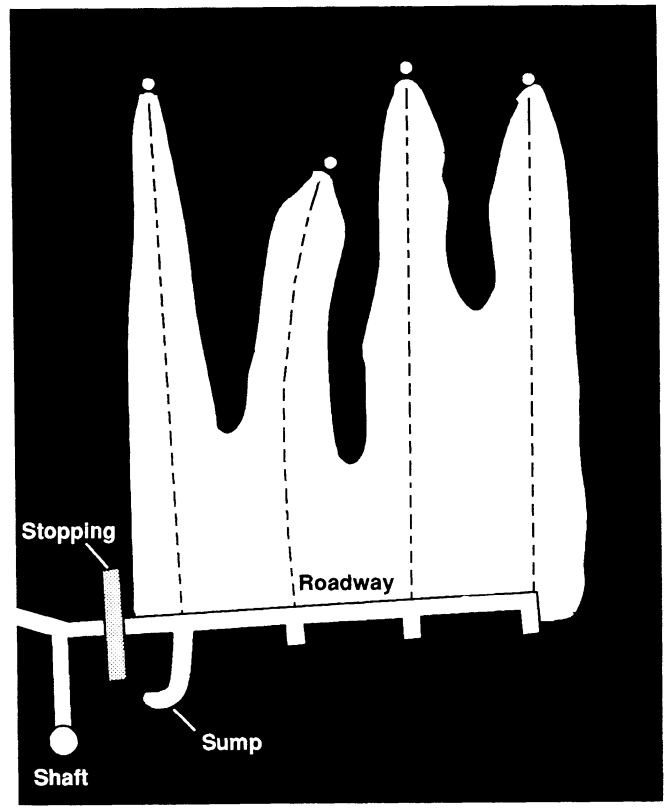


Fig. 22.6.9. Voids left at conclusion of P5 field trial at Newman-Spinney (Thompson, Mann, and Williams, 1976).

surface drilling methods. Fig. 22.6.10 depicts the construction of a typical process well. The problem is how to establish a proper initial underground gas flow link between the ends of the two boreholes, the natural permeability of the coal seam being too low to permit the required gas flows for UCG.[4] There are several methods available to link the injection hole to the production hole, each with their good and bad points. Two of the methods, directional drilling and counterflow or reverse combustion, are in current use and will be described below. The others—electrolinking, hydrofracking, and explosive linking—have been tested with varying degrees of success but are generally no longer of interest, although hydrofracking is still sometimes used in connection with reverse combustion to enhance the initial permeability of a seam.

DIRECTIONAL DRILLING. Adapted from oil well applications, the Soviets have employed *directional drilling* in the USSR since 1940 as a means of preparing the gasification link between injection and production boreholes. More recently, several UCG projects in the United States have also utilized the technique, which has the advantage of establishing a positive link at the base of the coal seam, relatively independent of the geology, dip, and fracture system in the seam. The hole in the coal provides the initial gasification channel, while that portion of the hole leading to the surface may or may not serve as the injection hole. Utilizing down-hole electrically or hydraulically driven cutting tools, curved holes with very small radii of curvature can be drilled. Fig. 22.6.11 shows a representation of a 3-in. (75-mm) directional (or deviated) borehole drilled to the 25-ft (7.6-m) Felix No. 2 subbituminous seam for the Hoe Creek III UCG trial. A $2\frac{3}{8}$-in. (60-mm) diameter Dyna-Drill with mud motor

[4] Permeability measurements on stress-relieved cores have indicated values of 1 to 10 millidarcies (md) for bituminous coal, 100 to 200 md for subbituminous non-shrinking coal, and 1000 to 2000 md for subbituminous shrinking coal (Lien et al., 1977).

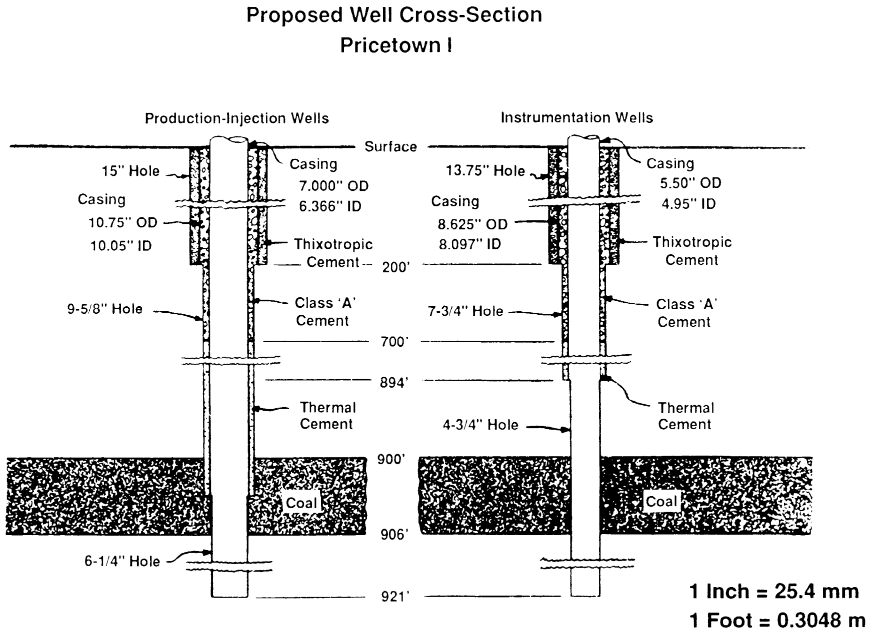


Fig. 22.6.10. Typical construction for UCG boreholes (Strickland, 1977).

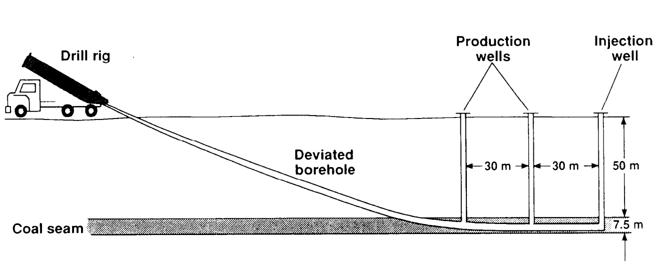


Fig. 22.6.11. Elevation of directional hole at Hoe Creek (Hill, Aiman, and Ganow, 1979). Conversion factor: 1 ft = 0.3048 m.

was used. The initial borehole angle of 30° to horizontal was deviated at a rate of 5°/100 ft (30 m) of travel. The total drilled distance was 705 ft (215 m) with the final horizontal 200 ft (61 m) in the coal about 10 ft (3 m) above the bottom of the seam. Deviated test holes into thinner coal seams have been drilled by experienced drillers; for example, up to 3000 ft (914 m) maintained in a 6-ft (1.8-m) coal seam at a 1000-ft (305-m) vertical depth (Siegel and Chambers, 1982).

REVERSE COMBUSTION. *Reverse combustion linking* (RCL) is a counter-current combustion process that is based on the simple fact that a propagating fire seeks oxygen. As depicted in Fig. 22.6.12, a fire started in the coal seam at the bottom of Well-1 will propagate towards Well-2, following the path of air flowing from Well-2 to Well-l. Unlike co-current burning where fire propagation and airflow are in the same direction, counter-current burning tends to follow a narrowly defined path of enhanced permeability within the coal seam rather than grow into a broad reaction front that expands outward to fill the entire seam (see Fig. 22.6.8). This difference between the two burning techniques is due in part to the direction of the hot gas flow. In co-current or forward burning, hot tarry gases are pushed into the coal ahead of the burn front where they can condense, causing a decrease in seam permeability. In counter-current or backward burning, the same gas products are swept back through the permeable channel of char behind the burn front, and away from

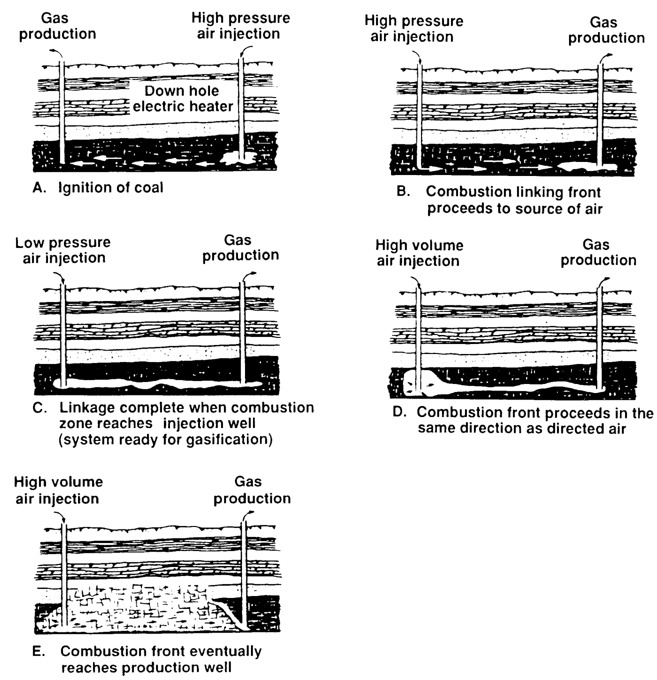


Fig. 22.6.12. Schematic representation of reverse combustion linking and UCG.

the unreacted seam. Thus the reaction front, propagating by thermal conduction from the char front into the seam, tends to grow unidirectionally toward the source of air.

In practice, air injection at high pressure is introduced into one hole until communication is achieved with the ignition hole. A fire is started in the ignition hole using an electrical heater or chemical igniter. Counter-current burning at the high injection pressure continues until linkage is achieved as signified by a

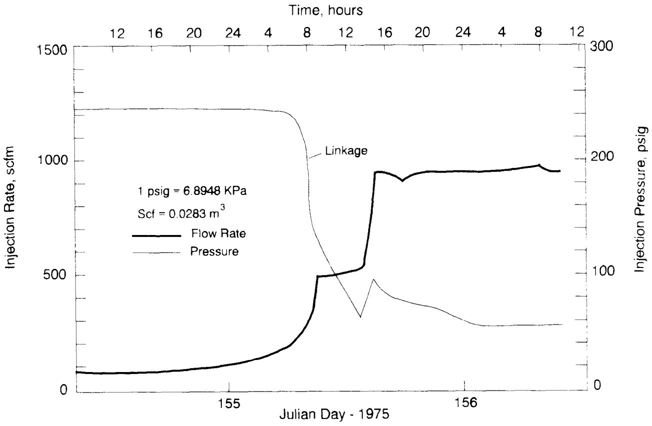


Fig. 22.6.13. Flow parameters as linkage was attained at the Hanna II field trial (Brandenburg et al., 1975).

sudden drop in line pressure (Fig. 22.6.13). In order to maintain the gasification channel on as straight a line as possible between the two boreholes, the holes are aligned with the maximum natural permeability of the seam.

An important use of RCL is in conjunction with directional drilling, namely, to link that part of the directional hole in the coal seam to vertical boreholes drilled from the surface. The vertical boreholes or extended linked wells (ELW) need only to intersect the seam near the horizontal hole, the final connection being made through RCL. The extended linked well would normally serve as the production well for UCG.

22.6.2.2 Resources and Utilization of UCG

COAL RESOURCES IN THE UNITED STATES. The available coal resources for UCG in the United States are tremendous, estimated to be almost 1.8 trillion tons (1.6 trillion t). Fig. 22.6.14 depicts the location of the coal fields on a map of the United States and the potential increase in coal resources through UCG over conventional mining. All regions of the country could benefit. The eastern and midwestern regions, with their large bituminous deposits, represent high concentrations of people and markets for energy, but also present more difficult technical challenges to UCG. The plastic and swelling characteristics of bituminous coal tend to lead to sealing of the gasification channels, and the thin seams of the eastern coal, many of them less than 6 ft (1.8 m), lead to higher heat loss from the seam and more difficult control of the UCG reaction zone.

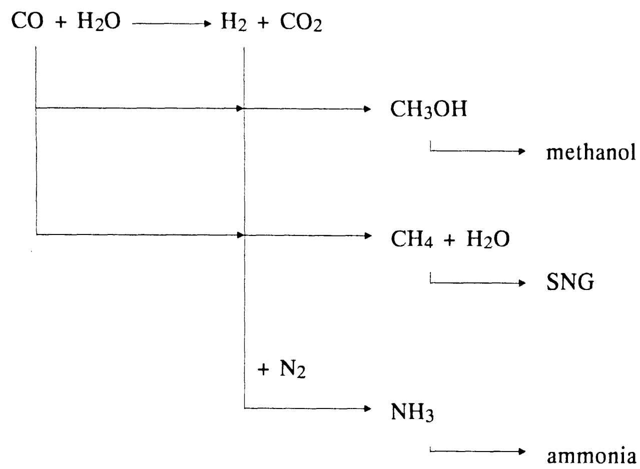
The Rocky Mountain region of the United States with its thick subbituminous seams, both horizontal and steeply dipping, has been the prime focus for UCG development by the US Department of Energy (DOE) and several private companies (e.g., ARCO, Texas Utility, Gulf R&D, Energy International, Gas Research Institute), despite the lower population density and lower market potential of this region. The resource more closely resembles those that the Soviets have exploited most successfully for UCG. Also, many of the thick, deep subbituminous deposits cannot be mined by conventional methods, and the possibility of utilizing this huge unminable resource with low water requirements and less surface disruption is very attractive.

Many of the coal reserves in the western regions of the United States (including Alaska and its 1.8 trillion tons or 1.6 trillion t) are not readily amenable to conventional mining, but a sizable amount may be suitable for recovery by UCG.

UTILIZATION OF UCG. The primary product of UCG with air as the oxidant is low-Btu gas (80 to 150 Btu/scf, or 3000 to 5600 kJ/std m^3), which is best suited for onsite or near-site use as a fuel for boilers, heaters, power generators, and kilns. As a low-heating-value fuel, it contains a sizable quantity of inert nitrogen, and hence is not cost effective for pipe transportation for any distance beyond about 1 mile (1.6 km). However, for onsite use, it is a good source of heat. While UCG fuel gas might have a heating value only 9% that of methane (the major constituent of natural gas), it will, when burned with air, lead to product gases (i.e., the high-temperature working fluid) whose heat content is a respectable 60% of the working fluid obtained when methane is burned with air (Chaiken, 1974).

UCG with oxygen/steam injection leads to a medium-heating-value gas (200 to 300 Btu/scf, or 7400 to 11,200 kJ/std m^3) that contains little N_2. It can be used in the same way as low-Btu gas, basically still onsite, except that the absence of N_2 makes the gas suitable for upgrade through chemical processing to a substitute natural gas (about 900 Btu/scf, or 33,500 kJ/std m^3) and/or to several synthesis gas compositions (see Table 22.6.2).

The synthesis gas would serve as chemical feedstock for making methanol or ammonia, that is,



(22.6.4)

The economics of processing UCG gas onsite in surface facilities appears promising, as shown in Table 22.6.3. However, chemical synthesis plants operated as part of the UCG surface facilities could add considerably to the water requirements at the site, and transportation costs to offsite chemical plants could alter the economics significantly (see Table 22.6.3).

22.6.2.3 Methods for UCG

GENERAL COMMENTS. The recent two decades of research and development in UCG by the US Department of Energy were directed to building a technology base from which one could adapt findings and developments to the development of a commercial UCG facility at any suitable site rather than to build and operate a pilot plant at one given site. A listing of all the appropriate UCG field tests in the Unites States has been compiled by the DOE (Anon., 1986). The recently concluded Rocky Mountain I test represents the latest (1986–1989) and largest of the field trials, and also the one most oriented to commercial interests. The available database on all these tests is well documented by the DOE, with a computerized bibliography stored in the CATAPULT Information DataBase at the Morgantown Energy Technology Center, Morgantown, WV. The UCG methods to be reviewed briefly in this section are described in detail in the documents available from the this center or from the US National Technical Information Center, Springfield, VA.

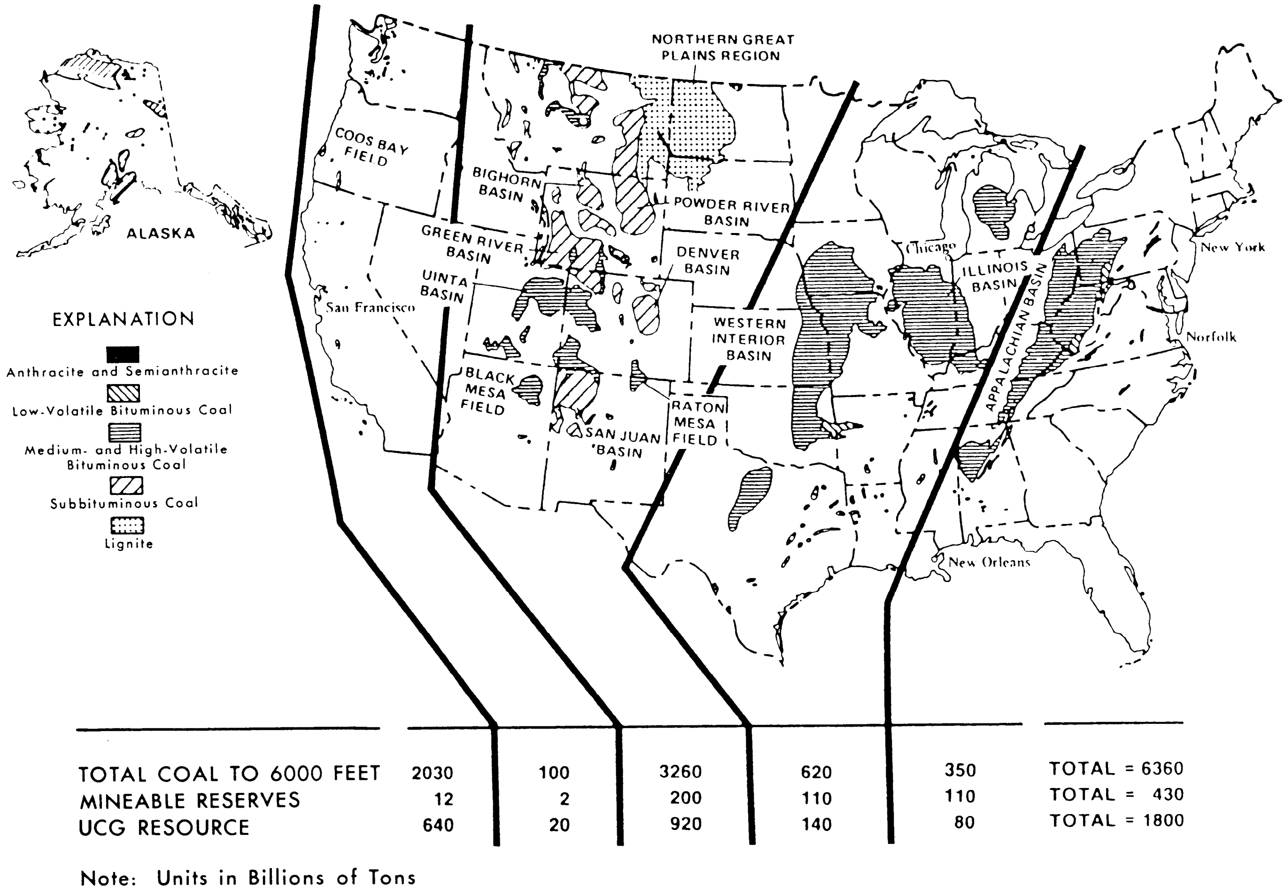


Fig. 22.6.14. Estimated UCG resources in the United States (Anon., 1986). Conversion factors: 1 ton = 0.9072 t, 1 ft = 0.3048 m.

Table 22.6.2. Gas Compositions, % dry

	O_2/steam crude gas	Methanol syn gas	Ammonia syn gas	Byproduct SNG
H_2	38.5	60.1	73.5	2.6
N_2	0.3	0.4	24.5	1.9
CO	19.4	17.6	—	0.1
CO_2	29.4	6.3	—	0.3
H_2S	0.3	—	—	—
CH_4	11.1	14.9	2.0	95.1
C_nH_m	1.0	0.7	—	—

Source: Adapted from Bidlack and Arscott, 1976.

Table 22.6.3. Comparison of Economics of SynGas: UCG (Wyoming and Montana Sites) vs. Lurgi (surface)

	Methanol		Ammonia	
	UCG	Lurgi	UCG	Lurgi
Capital Requirement, 10^6\$	332	511	347	520
Price of SynGas, \$/10^6scf[a]	609	1015	440	882
Added transportation cost, \$/10^6scf[b]	522	—	—	—

Source: Adapted from Bidlack and Arscott, 1976.
Notes: a. Price includes credit for SNG and/or byproducts. Calculated with utility-type financing, 13% capital charges. b. Pipeline to Gulf Coast. Conversion factor: 1 scf = 0.0283 m^3.

The extensive Soviet experience has involved numerous field trials, some of which have evolved into small commercial type UCG plants whose fuel gas output is used by onsite power plants (Olness and Gregg, 1977). These UCG (Podzemgaz) facilities include those sited in the flat beds of lignite near Moscow (Shatskaya) and near Tashkent (Angenskaya), and in the steeply dipping bituminous seams of the Donets basin (Lisichankaya) and Siberia (Yuzhno-Abinskaya). At their peak (in 1966), these plants had an annual production of 70 billion scf (2 G std m^3) of low heating value gas (80 to 100 Btu/scf, or 3000 to 3700 kJ/std m^3), equivalent to 860,000 tons (780,000 t) of coal consumed. This might be compared with the total of 34,000 tons (31,000 t) consumed in all the DOE field trials. UCG production in the Soviet Union has since declined (24 billion scf, or 0.68 G std m^3 in 1977), presumably due to the availability of other plentiful sources of fuel gas. Details on the Soviet experience can be found in the many reports translated into English which are included in the DOE database.

Additional trials, which have been carried out in most coal producing countries, have added to the available UCG database, including the Newman-Spinney P-5 trial in Great Britain where 900 tons (800 t) of a 2.5-ft (0.8-m) thick flat bituminous seam was gasified (Gibb, 1964). The resulting gas (average of 57 Btu/scf, or 2,100 kJ/std m^3) was of poor quality, but the efficiency of coal consumption, as determined by post-burn excavation, was high indicating that gasification and/or combustion reaction zones can be propagated through thin bituminous seams.

What can be assessed from these experiences is an organized way of approaching the development of a UCG facility at a specific coal site, which is the thrust of this segment.

SITE SUITABILITY. The importance of site selection and the subsequent characterization of the site cannot be over emphasized. Since the UCG reactor is bounded only by the surrounding coal and adjacent strata, the results of a UCG operation are subject, to a large degree, to the nature of the environment. For example, tests conducted in the United States have demonstrated that water-bearing strata immediately overlying the coal seam can collapse into the reaction zone, causing degradation of the product gas and even the extinguishment of the gasification front. Faulting in the coal seam and adjacent strata can also result in multiple flow paths, low product gas quality, and excessive gas losses. Even in the absence of faulting, weak and permeable strata above the coal seam have permitted significant gas losses with attendant groundwater environmental problems.

The US Department of Energy has completed an assessment of site selection and characterization needs. The basic requirements and procedures to determine site suitability are soon to appear in a handbook (Oliver and Covell, 1990). In general, cores are extracted and analyzed, well logging and other geophysical and geochemical measurements are performed, and geohydrology tests are conducted.

Geophysical data include seam thickness, seam dip, seam permeability, partings, faulting, depth, seam roof, water-bearing formations, rock strengths, aquifers, and topography.

Geochemical data include chemical composition, heating value, volatiles, ash and water content, shrinking or swelling tendency, agglomeration, sulfur content, and reactivity.

The data are then analyzed and interpreted with respect to the federal, state, and local permitting and environmental requirements/restrictions, and with respect to the planned gasification operations. The results are then used in the design of the plant, and to ensure that the operations are conducted in a manner consistent with the environmental and other permits required for plant construction, gasification, and shutdown of the UCG reactors.

PROCESS WELL LINKAGE. Tests conducted in the United States have established that the gasification reaction front does not progress below the elevation of the linkage path between process wells, and coal lying below the link is not recovered (Oliver, 1987). It is necessary to position the link as close as possible to the bottom of the coal seam to enhance the probability of attaining the highest resource recovery efficiency possible.

Reverse Combustion Linking—Initial experiments in the development of the technology successfully used reverse combustion linkage (RCL) to establish the required linkage paths over relatively short distances (60 ft or 18 m). Subsequent tests, however, met with a variety of problems including (1) loss of control due to faulting, (2) linking established at the coal seam/overburden interface because of the high pressures involved, and (3) plugging of the link due to swelling of the coal and condensation of high boiling point tars.

In the thick subbituminous seams of Wyoming, successful reverse combustion linkages have been achieved at up to 60-ft (18-m) separation between boreholes. The Soviets have reported reverse combustion linkages of up to 100 ft (30 m). However, the average linkage distance at the UCG station at Angren (USSR) was only 57 ft or 17 m (averaged over a total linkage distance of 53,000 ft or 16 km). Typical high-pressure air during linkage was 380 psi (2540 kPa), leading to a channel formation rate of about 2.5 ft/day (0.8 m/day). High-pressure air utilization was about 73,000 ft^3/ft (6800 m^3/m) of channel linked (Stephens, Hill, and Borg, 1984).

The cost of reverse combustion linking is very site selective but can run more than $200/ft ($650/m) of horizontal length in the seam, which is about half the cost of directional drilling (Stephens, 1979). However, the linkage could be seriously impaired by the geologic and chemical factors mentioned above.

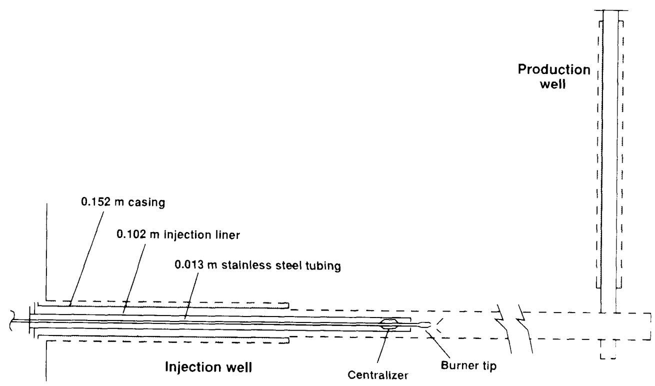


Fig. 22.6.15. Silane/propane igniter installation (Thompson and Skinner, 1982). Conversion factor: 1 ft = 0.3048 m.

Directional Drilling—Directional drilling, which was first advocated by the Soviets for a proposed UCG plant at Kohlmogorsk in the Soviet Union, offered the potential to alleviate the problems that had been experienced with RCL, and to provide a controlled, positive link between the process wells. Two tests, the first at the Hoe Creek site near Gillette, WY (Hill et. al., 1980), and the second at the TONO site near Centralia, WA (Martin and Davis, 1985), demonstrated that wells started at an angle at the surface could be deviated in a controlled manner to place the link parallel to, and at the bottom of the target coal seam. The gasification phase of the tests was subsequently completed with little or no problems related to the linkage path. Also, a directional well drilled earlier near Pricetown, WV, demonstrated that a well could be started at near vertical at the surface, turned nearly 90° over a depth of 900 ft (280 m), and be maintained in a flat-lying, 6-ft (2-m) thick coal seam for a distance of about 500 ft (150 m) (Martin and Liberatore, 1980).

Current directional drilling techniques should enable the proper emplacement of the intial UCG channel in almost any type coal seam. However, the cost of such drilling runs high (about $500/ft of horizontal seam, or $1640/m), and may inhibit its use depending on the specific resource being considered.

PROCESS CONTROL. It has been demonstrated repeatedly in over 21 tests conducted by both the government and industry in the United States that coal seam ignition and the shutdown of the UCG gasification reactor(s), with subsequent cooling of the coal seam to ambient temperatures, can be accomplished reliably. The proper selection of the site so that the reaction zone is below the water table ensures that the exothermic combustion reactions will terminate when oxidant injection is removed. The reducing gases produced from the devolatization of the coal inhibits the combustion process, and water will influx from the surrounding groundwater system to cool the reaction cavity.

Ignition—Reliable coal seam ignition was a problem until researchers advocated the use of a pyrophoric material in the gaseous phase, and devised the silane-methane igniter system (see Fig. 22.6.15). In this system, a small-diameter tube with a burner nozzle on the end is inserted into a larger-diameter tube in which there is flowing oxygen. An inert gas such as nitrogen is injected through the small-diameter tube to purge the line of oxygen. A slug of silane, a pyrophoric gas, is injected immediately behind the nitrogen, and a few seconds later, a combustible gas such as methane is injected. When the silane encounters oxygen, a flame is established which ignites the methane, forming

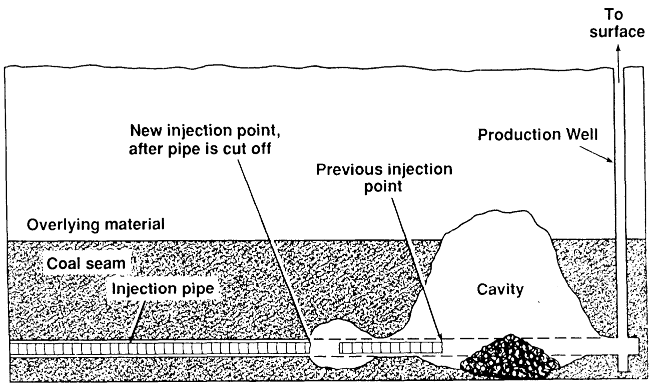


Fig. 22.6.16. Basic design of the controlled retracting injection point (CRIP) system (Hill and Shannon, 1981).

a high-temperature torch. Coal seam ignition has been accomplished in every case that this igniter system has been used.

Gasification—During the gasification phase of the operation, process control can be exercised by varying the gas flow rates and pressures, and by varying the composition of the injected oxidant, such as the steam to oxygen ratio. Test results have demonstrated repeatedly that gas quality degrades as the reactor matures (see Fig. 22.6.5). This has been attributed to the decline of turbulent flow as the reactor becomes larger in volume and to the decline in the cavity temperature when wet, inert overburden material collapses into the cavity.

Controlled Retracting Injection Paint—The controlled retracting injection point (CRIP) technique was conceived and advanced by researchers (Hill and Shannon, 1981) in an attempt to provide a more constant and desirable gas quality from the UCG process. In this technique, the silane-methane igniter is positioned at the point where the coal is to be first ignited (Fig. 22.6.16). The ignition sequence is performed, and when coal seam ignition is assured, the igniter is retracted and positioned within fresh coal at a point outside the predicted zone of influence of the developing gasification cavity. When the gasification reactor is mature and the gas quality declines to a predetermined value, the ignition sequence is repeated at the new igniter location, leading to a new UCG reaction zone (see Fig. 22.6.16). Application of this technique has been demonstrated to ensure the maintenance of the reaction zone at the bottom of the coal seam, and to enhance the production of a gas stream with an acceptable gas quality for the life of the module.

ENVIRONMENTAL RISKS. The environmental concerns associated with UCG processing are no worse than those associated with winning coal by underground or surface mining followed by gasification in a surface gasifier. In both cases, the winning of the coal from underground will result in some subsidence and its accompanying problems. Indeed, in situ processing of coal can be a significant improvement over some aspects of surface processing. For example, the steps usually followed for surface extraction and recovery include (1) mining of the coal, (2) cleaning the coal in a coal preparation plant, (3) transporting the coal to the point of use, (4) storing the coal, (5) preparing the coal for use, and finally, (6) cornbusting, gasifying, or liquefying the coal. Each of these steps produces a variety of solid, liquid, and/or gas residues that must be treated prior to disposal. In addition, a significant amount of potable water is consumed, and this water has to be treated before it can be returned to the environment. UCG, on the other hand, offers the potential to combine several steps such as mining, cleaning, preparation, and pro-

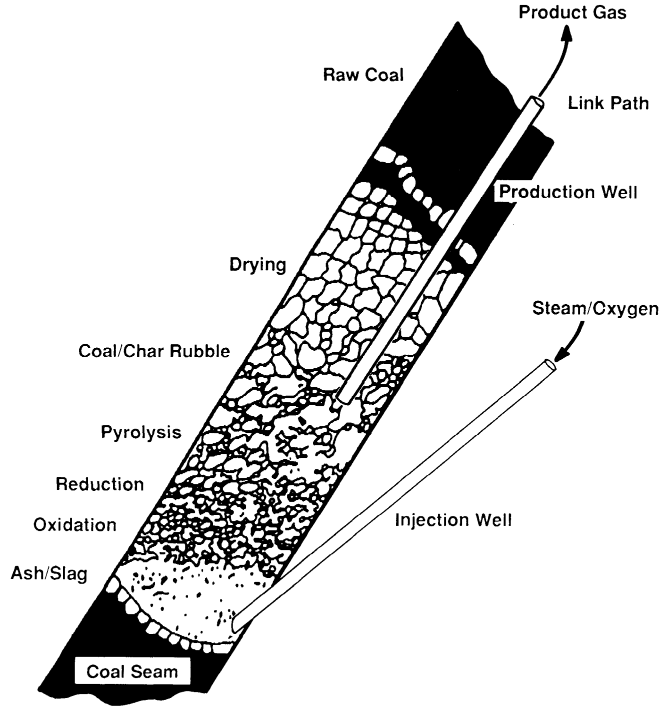


Fig. 22.6.17. Schematic representation of the steeply dipping bed gasification process.

cessing into a single operation—which may well be more acceptable environmentally—and in addition, offers the potential of reduced costs relative to the total costs associated with surface processing.

A possible additional environmental problem associated with UCG is the risk of contaminating the groundwater system. Early UCG tests, which resulted in contaminated groundwater in the unreacted coal as well as in adjacent water-bearing zones, were performed with high cavity pressures to inhibit excessive water influx into the gasification reactor. Subsequent laboratory tests led to the conclusion that high cavity pressures have little effect on the quantity of water that influxes into the reactor during gasification operations (Boysen et al. 1987). As a result of the laboratory studies and modeling of the generation and transport of ground water contaminants, a procedure was formulated (i.e., the "clean cavern concept") to minimize groundwater contamination during and immediately following UCG operations. In this concept, the subsurface reactor pressure is maintained below hydrostatic to minimize the loss of organic laden gases and to ensure a small but continuous influx of groundwater into the gasification cavity. When the gasification operations are complete, steam is then injected into the cavity to promote rapid cooling of the cavity walls and residues, and to "strip" the soluble and volatile organics from the cavity. The steam and contaminating gases are routed through an incinerator before being exhausted to the atmosphere. Operating in this fashion has confined the contaminants from UCG to the gasification cavity, and the contaminated cavity water can then be pumped to the surface for treatment before it spreads to the surrounding groundwater system.

22.6.2.4 Rawlins Steeply Dipping Bed Project

The application of UCG in a steeply dipping (> 45°) coal seam is depicted in Fig. 22.6.17. The oxidant injection well is drilled below the seam and out of the zone where any roof caving or subsidence may occur. The product gas well is drilled through the seam itself, near and parallel to the bottom wall. The oxidant injection well is slant drilled to intersect the coal seam at the prescribed depth. The coal is ignited, and a fire pit is developed at the level where the injection well enters the coal seam. Hot product gas flows upward to and through the unlined product gas well, causing the adjacent coal to dry and undergo devolitization. This process weakens the coal structure resulting in the collapse of a portion of the roof into the fire pit below. The fresh coal is then gasified, continuing the process that in effect becomes self-feeding.

UCG trials were successfully conducted with this configuration near Rawlins, WY, from 1979 to 1981 to demonstrate the viability of the steeply dipping bed technology. The second of two tests conducted at the site was performed about 600 ft (180 m) below the surface in a 23-ft (6.9-m) thick seam of subbituminous coal (Ahner, Bencini, and Bloomstran, 1982). Peak injection pressures in the 63° dipping seam were nearly 160 psi gage (1100 kPa) with oxygen and steam as the injection fluids. During one period of the operation, an average product gas heating value of 356 Btu/scf (13,300 kJ/std m^3) was achieved and maintained for nearly 20 days. Methane concentrations, which were attributed to the high in-seam pressures, of up to 24% were recorded which significantly enhanced the heating value of the gas.[5]

22.6.2.4 Rocky Mountain 1 (RM1) Test Results Summary

The results of the DOE's UCG program investigations were utilized by an industrial consortium (with government support) during the period 1986 to 1988 to design, construct, and operate the RM1 UCG test at a location near Hanna, WY.[6] The borehole linking experiences derived from the UCG program and other technology areas prompted a decision to use directional drilling for process well linking during the RM1 test. The technique known as medium-radius drilling was selected.[7] In this method, the process well was started in a vertical attitude at the surface, drilled to the "kickoff" point, and then turned at a rate of about 21°/100 ft (30 m) of hole drilled to achieve an approximate 300-ft (90-m) radius of curvature. This medium-radius hole was more expensive than a short-radius borehole; however, the short-radius hole would not have accommodated the installation of a casing that is required to isolate the product gas from any subsurface aquifers. Three such wells were successfully placed for the test (Fig. 22.6.18).

The placement of the horizontal and vertical boreholes did not make for physical intersection underground, but they were sufficiently close to enable the final necessary linkages to be accomplished with about four days of reverse combustion. No problems related to the linkages were encountered during the gasification phase of the test. A more complete description of the medium-radius wells completed for the RM1 test can be found elsewhere (Logan, 1987).

As shown in Fig. 22.6.18, two types of UCG modules were evaluated. The ELW module consisted of three process wells: two vertical injection wells (VIW-1 and VIW-2) and a horizontal production well (PW-1). In this module the operation was to inject oxidant initially via VIW-1. When needed, the UCG reaction zone would be extended to new coal by transferring the injection from VIW-1 to VIW-2 (100 ft or 30 m away).

The CRIP module also consisted of three wells: (1) a vertical startup well (CPW-2), (2) a horizontal injection well (CIW-l), and (3) an horizontal production well (CPW-1). CPW-1 was 5 ft (1.5 m) higher in the seam than CIW-1 to allow upward gasification to take place between the points of injection and production. When needed, a new reaction zone would be obtained by withdrawing the CRIP igniter about 100 ft (30 m) and burning off the stainless steel liner.

UCG operations were conducted for a period of 102 days, from Nov. 17, 1987, when the ELW module was ignited using the silane/methane igniter system, through Feb. 26, 1988, when the CRIP module was shut down. Both modules utilized steam-oxygen injection during the gasification phase of the test. A summary of the performance of each module during the gasification phase of the test is given in Table 22.6.4. The complete detailed description of the test can be found in the RM1 Final Operations Phase Report (Anon., 1989).

During the test, the clean cavern concept was followed closely considering the conditions encountered. Operating pressure during most of the gasification phase of the test was maintained between 65 and 90 psi (440 to 610 kPa), which was significantly below the hydrostatic pressure of 120 psi (820 kPa). When each module was ready to be shut down, steam injection was continued for about 10 days after oxygen injection was terminated. The product gas stream was incinerated during this period.

Samples of the cavity water collected after the cavities had refilled exhibited elevated levels of phenols and boron. All other aspects of the water were within the baseline values determined prior to ignition of the coal seam. In addition, three of the twelve groundwater monitoring wells surrounding the cavities showed elevated contaminant concentrations similar to those in the cavity water. These wells were located up dip, and the contaminants were the result of a gas bubble which formed in that area during the test.

Since the shutdown of the gasification phase of the test, cleanup of the cavity water has been completed. The treatment consisted of pumping the water from the cavities to the surface, treating the water in activated carbon adsorption units, and spray evaporating the water through atomizing nozzles. Water from the surrounding groundwater system was allowed to influx into the cavities to "wash" and confine any additional contaminants in the vicinity of the cavities. Water samples extracted from the refilled cavities indicated that the phenols were reduced to baseline levels, but the boron concentrations, although reduced, remained elevated. A second cavity water treatment was performed to insure that the groundwater at the site was returned to levels acceptable by state agencies.

The RM1 UCG test was designed to incorporate the advances developed under the DOE/UCG program, and to serve as the culmination of the technology developed for subbituminous coal seams. The test was conducted at a size scale, and for a time period appropriate for evaluation of the potential for commercializing underground coal gasification of this type of coal resource. At this point, it is believed that industry must decide whether UCG is a viable alternative energy option.

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[5] As a result of the Rawlins test, a Department of Energy contract was awarded (1987) under the "Clean Coal" program to produce synthesis gas by UCG, and to use this gas onsite in the manufacture of anhydrous ammonia on a commercial scale (Davis, Singleton, and Penland, 1988).

[6] The industrial consortium was headed by Gas Research Institute and included Amoco Production Co., Electric Power Research Institute, and Union Pacific Resources Co.

[7] For convenience, directional drillholes can be categorized in terms of three ranges of deviation or curvature: short-radius [1.5 to 3.0°/ft (0.3 m), or 40 to 20 ft (12 to 6 m) radii]; medium-radius [8 to 50°/ 100 ft (30 m) or 700 to 125 ft (210 to 38 m) radii]; and long-radius [2 to 6°/100 ft (30 m) or 3000 to 1000 ft (900 to 300 m) radii].

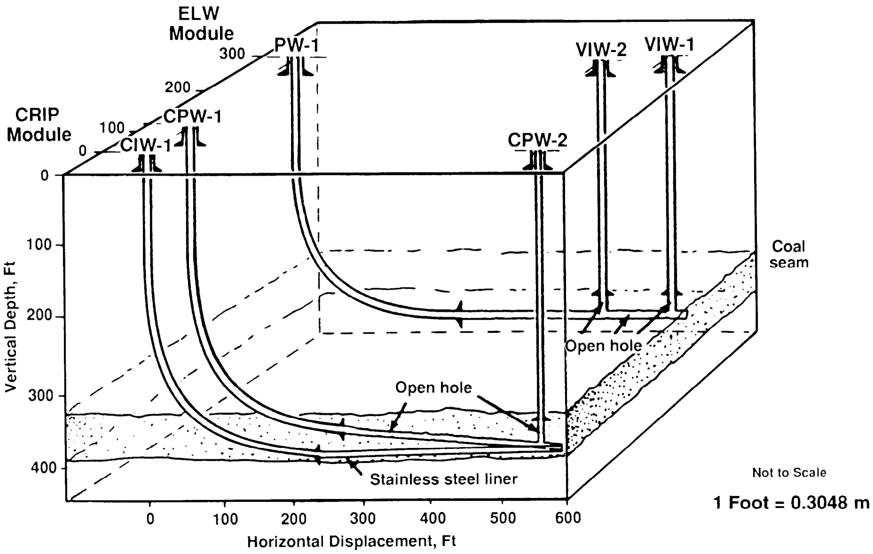

Fig. 22.6.18. Borehole arrangement for the Rocky Mountain 1 project (Tracy, 1988).

Table 22.6.4. Averaged RMI UCG Operational Data

Duration, days	47.0	97.0
Coal Gasified, tons	4,000.0	11,000.0
Injection Rate, scfm	1,790.0	2,005.0
Wet Gas Production Rate,	.0	.0
scfm	4,450	4,900
Steam/O_2 Ratio	1.8	2.0
H_2O/Dry Gas Ratio	0.89	0.46
Dry Gas Heating Value,	.0	.0
Btu/ft^3	265	287
Major Dry Gas Components, Mole %		
H_2	31.0	38.0
CH_4	10.0	9.5
CO	9.2	11.9
CO_2	42.0	38.0

Source: Anon., 1989.
Conversion factors: 1 ton = 0.9072 t, 1 scfm = 0.0283 m^3/min, 1 Btu/ft^3 = 37.25 kJ/m.

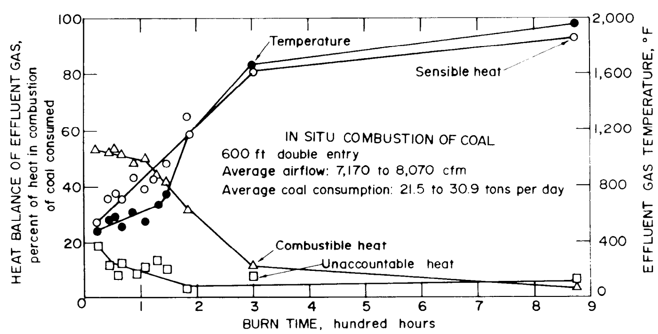

Fig. 22.6.19. Thermal output from UCG experiment at Gorgas, AL (Chaiken, 1974). Conversion factors: 1 cfm = 0.0283 m^3/min, °F = 1.8°C + 32°, 1 ton = 0.9072 t.

22.6.3 IN SITU COMBUSTION OF COAL

22.6.3.1 Introductory Comments

In situ combustion of coal converts the spatially and thermally distributed reaction zone of Fig. 22.6.1 to complete oxidation, thus converting the underground coal seam cavity into a high-temperature (> 1000°C or 1800°F) furnace. The thermal energy release in the cavity furnace is recovered by bringing hot flue gases (mainly N_2, CO_2, and H_2O) to the surface for onsite conversion to steam, electricity, or simply process heat. The advantage of this approach over UCG is the ease of maintaining complete combustion (i.e., chemical equilibrium) as opposed to incomplete combustion (i.e., shifting equilibrium). This was exemplified in the UCG field trial at Gorgas, AL, where gasification was attempted in a double-entry channel, 600 ft (180 m) long, in an 8-ft (2.4-m) thick bituminous seam. Although this 900-hr test was for the purpose of producing combustible gas, it is evident (Fig. 22.6.19) that after 300 hr, the underground combustion reaction went to completion (probably due to "break through" of oxygen) and remained that way. It is also evident that steady burning was achieved where about 90% of the heating value of the coal appeared in the high temperature exhaust gas, at a thermal power level estimated to be 1.5 MW.

The ramification of the Gorgas UCG data in terms of a complete in situ combustion process was reported in a USBM Technical Report (Chaiken, 1974) and was pursued experimentally over the next decade primarily in connection with controlling and making use of the heat from abandoned coal mine fires. There are currently some 64 abandoned coal mine fires in the eastern coal producing states, representing a severe hazard to the environment as well as endangering some 116 million tons (105 Mt) of coal. It has also been estimated that some 500 million tons (450 Mt) of coal are contained in abandoned hilltop seams, mostly in the eastern US, many of which would be amenable to in situ combustion by the Bureau of Mines' burnout control process.

22.6.3.2 Burnout Control

General Description. *Burnout control* is a process that uses exhaust ventilation conditions to promote the complete burning of an underground mine fire while allowing total management of the hot gases produced (Chaiken, 1980). An artist's rendition of a burnout control system is shown in Fig. 22.6.20. An exhaust fan, connected to a ductwork system that leads to the underground mine, pulls a vacuum on the mine containing a

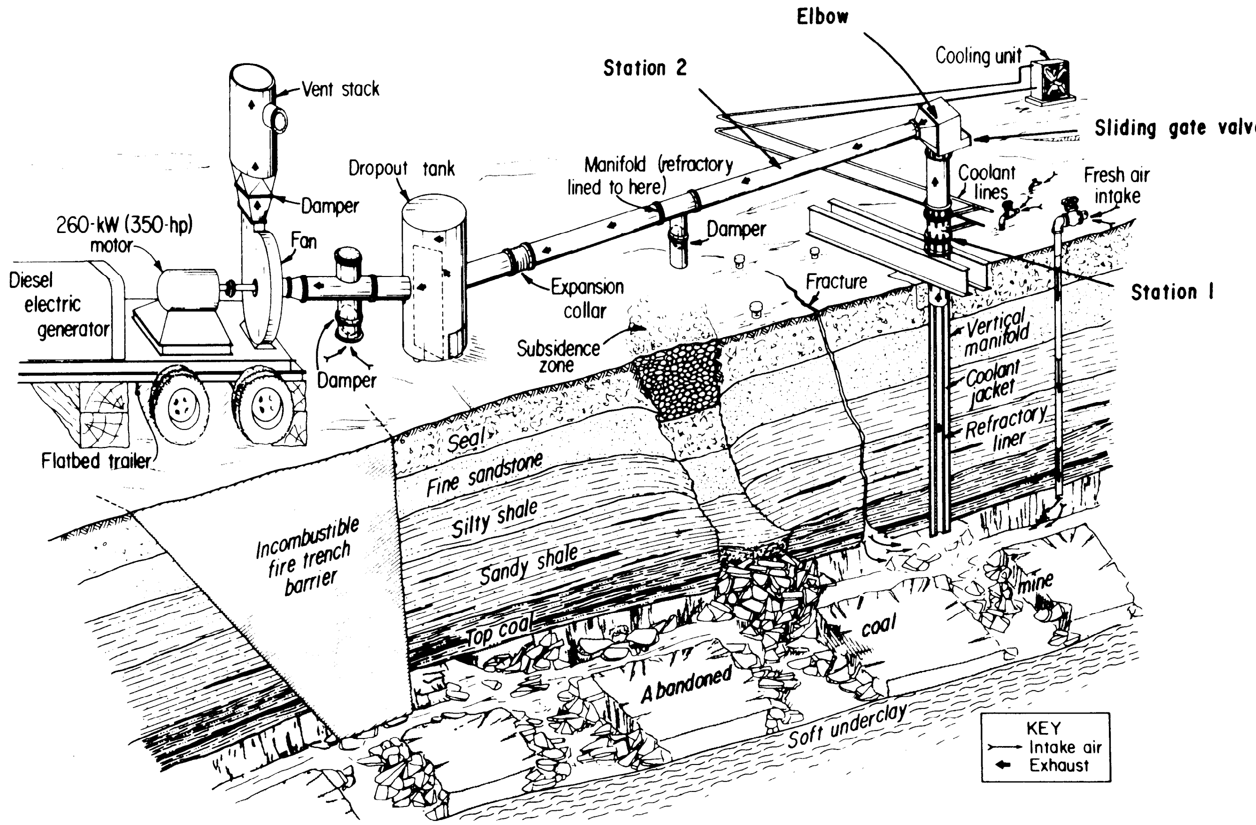

Fig. 22.6.20. Artist rendition of burnout control at Calamity Hollow (Irani et al., 1983).

smoldering fire. This results in air flowing into the mine through natural cracks and fissures and/or from air inlet boreholes set from the surface. The increased airflow intensifies the fire, and all heat and fumes (flue gas) are drawn to the single-fan exhaust point. Fig. 22.6.20 shows the flue gas being cooled by air dilution (via the damper at the end of the refractory lined portion of the manifold) and then simply exhausted to the air. However, instead of cooling by air dilution, the hot flue gas could pass through a waste-heat boiler to recover the sensible heat of the hot combustion products. If necessary, combustion of the hot flue gas can be completed with an afterburner, and air pollutants can be removed with conventional surface scrubbers.

There are several distinct advantages to utilizing the burnout control process for in situ combustion:

1. The affected coal channels are at negative pressure relative to ambient, and therefore all fumes produced will be drawn from underground at a single exhaust point. There will be no loss of gas to the surrounding strata, and conductive heat loss will be relatively small (less than 10%) (Chaiken, 1977).

2. Accumulation of heat and fumes at a single exhaust point enables (a) afterburning to insure complete combustion, (b) on-site utilization of the heat, and (c) the use of conventional or other air pollution scrubbers (mostly for SO_2).

3. The solid residues from high-temperature combustion are relatively inert ash and fused klinkers.

What may or may not be an advantage with burnout control is that the process is more amenable to dry seams with relatively open channels that contain coal rubble. In that sense, burnout control may not be suitable for the type of coal seam favored for UCG (i.e., thick and moist). On the other hand, as the UCG field trials at Gorgas and Newman-Spinney have indicated, burnout control should be applicable to the shallow, thin bituminous seams that are so prevalent in the high-energy demand areas of the eastern United States.

BASIC PROCESSES. The basic chemical process in in situ combustion is complete burning or oxidation of the coal. For stoichiometric burning of moisture free coal, the reaction (with ash taken as SiO_2) might be represented as

$$\underbrace{CH_{0.0856}O_{0.079}N_{0.018}S_{0.01}(Ash)_{0.053}}_{\text{1 mole coal (wt = 14.03 g)}} + \underbrace{1.219O_2 + 4.547N_2}_{\text{5.766 moles air}} \rightarrow$$

$$\underbrace{CO_2 + 0.428\,H_2O + 4.566\,N_2 + 0.005\,SO_2 + 0.053\,Ash}_{\text{5.989 mole gaseous products}} + \underbrace{97.34\text{ kcal}}_{\text{12,500 Btu/lb}} \qquad (22.6.5)$$

Ideally, 4.6 scf (0.13 std m^3) of air will combust 0.031 lb (0.014 kg) of coal to produce 4.79 scf (0.14 std m^3) of flue gas having a sensible heat content of 82.6 Btu/scf. In principle, a 1-MW thermal output can be obtained with a flue gas exhaust flow rate of 690 scfm (19.5 std m^3/min), representing the complete burning of 4.5 lb/min (2 kg/min) of coal.

The minimum fan air horsepower (ahp) to drive the exhaust flow is

$$\text{ahp} = (P)(Q)/6348 = 0.11\,P \text{ for 1-MW exhaust} \qquad (22.6.6)$$

where P is pressure drop across the fan in in. water, and Q is the exhaust gas flow rate evaluated at standard conditions in scfm.

The pressure drop requirements for the fan are site selective depending on the effective permeability of the underground burn zone to gas flow and the resistance of the duct system leading to the fan. The pressure drop due to the duct system alone can be readily estimated from pipe flow formulas; the effective permeability of the burn zone will require direct measurement of induced underground gas flows. However, it is readily seen that even with a fairly high vacuum of 100 in. water (25 kPa), and a fan/motor efficiency of 50% (both achievable with large conventional induction fans), it would require only 16.5 kW of electrical energy to produce 1000 kW of thermal energy. The effective electrical energy gain for operating a small electrical generation plant would then be a factor of 10 to 15, depending on the actual conversion efficiency of the plant. Large coal-burning utilities generally have an electrical energy gain of about 20.

When the underground flow follows Darcy's law, it is possible to develop size requirements for the fan in terms of the underground burning area and the thermal output. For steady radial Darcy flow toward the suction borehole, the exhaust flow rate is described by (Chaiken, 1989) as

$$Q(X) = [2(\pi)(k)(L)(X/\mu)]\ (dP/dX) \qquad (22.6.7)$$

where Q(X) is volumetric flow rate across a cylindrical surface of height L at radial distance X along the flow direction to the suction borehole; k is effective permeability of the mine strata; μ is viscosity of the flowing gas; and dP/dX is pressure gradient along the flow direction.

Integrating this expression with constant Q(X) between appropriate limits yields

$$DP = [(\mu)/2(\pi)(k)(L)]\ \ln(X_o/r)\ Q_{ex} \qquad (22.6.8)$$

where DP is vacuum level in the mine at the suction borehole; X_o is maximum radial distance from the borehole that is affected by the suction; r is inside radius of the suction borehole; and Q_{ex} = Q(X), the exhaust flow rate into the borehole.

Considering only the pressure drop through the underground burn zone, the fan power requirement is obtained by setting P = DP, or

$$ahp = (R)(Q_{ex})^2/6348 \qquad (22.6.9)$$

where $R = [(\mu)/2(\pi)(k)(L)]\ \ln(X_o/r)$ in in. water)/scfm.

The thermal output of the exhaust TP is given by

$$(TP) = (Q_{ex})(C_p)(\rho)(T_{ex} - T_o), \qquad (22.6.10)$$

where T_{ex} is temperature of the exhaust, T_o is reference air temperature (ambient), C_P is heat capacity of the exhaust gas at temperature T_{ex}, and ρ is density of the exhaust gas at the same temperature and pressure conditions as for Q_{ex} (as used here, Q_{ex} is in units of scfm). It is readily seen that the fan power requirement will increase as the square of the thermal output from the mine. Also, through the effective underground flow resistance R, the fan power requirement will increase as the logarithm of the underground area effectively being evacuated.

Some typical values for the parameters involved in the above relationships as determined during the USBM's Calamity Hollow Mine Fire Project (see next segment) are shown in Table 22.6.5. These parameters translate to a fan power requirement of 22 ahp (16.4 kW), which at a 50% motor/fan efficiency would result in an effective electrical gain of 20 to 30. However, this translation ignores the pressure drop requirements for the surface duct system, which were considerable for this particular field trial.

Table 22.6.5. Parameter Values Determined for Burnout Control at an Abandoned Mine Fire Site

Flow resistance, R	0.001 in. water/scfm (at DP = 4 to 16 in. water)
Suction Distance, X_o	160 ft (2 acre area)
Exhaust temperature, T_{ex}	600°C (average)
Exhaust gas flow rate, Q_{ex}	12,000 scfm (at DP = 16 in. water)
Exhaust thermal power, TP	5 MW (at DP = 16 in. water)

Source: Chaiken, 1989.
Conversion factors: 1 scfm = 0.0283 m^3/min, 1 in. water = 8.6 kPa, 1 ft = 0.3048 m, 1 acre = 4047 m^2.

22.6.3.3 Calamity Hollow Field Trial

The first field trial of burnout control was at the site of an abandoned shallow drift mine in the Pittsburgh seam whose outcrop had been stripped and then backfilled after a fire was discovered in the mine. A 1.8-acre (7300-m^2) isolated section of the mine adjacent to the covered outcrop continued to burn, smoldering under 35 ft (10 m) of cover (Fig. 22.6.21). Site evaluation included borehole corings to determine the stratigraphy and the residual coal values: 9000 tons (8000 t) of combustibles per acre with an average heating value of 12,000 Btu/lb (28,000 kJ/kg) (Irani et al., 1983).[8] Borehole temperatures were measured and air communication between boreholes verified utilizing a portable test suction fan (500 scfm capacity at 60 in. water, or 14 std m^3 at 15 kPa).

The burnout control ventilation control system was essentially that shown in Fig. 22.6.20. It was designed to withdraw 1800 °F (1000 °C) gases from the mine at thermal power levels up to 5 MW. At 5 MW output, it would take almost 3 years to completely burn out all the coal at the 1.8-acre (7300-m^2) site. No attempt was made to do this (the trial lasted four months), nor to utilize the thermal power generated. The gases were exhausted to the atmosphere through the fan after first being cooled with ambient air and water introduced directly into the duct system ahead of the drop-out tank (see Fig. 22.6.20).[9] This method of handling the hot exhaust required additional fan power beyond that needed to withdraw the hot gases from the mine, about 10 times the minimum value of 14 ahp estimated from Eq. 22.6.5 (assuming P = 25 in. water or 6.2 kPa). The burnout control suction fan (20,000 scfm capacity at 50 in. water, or 570 std m^3/min at 12.3 kPa) driven by a 350-hp (260-kW) electrical motor exceeded these requirements.

To channel the hot gases up from the mine, a water-jacketed, refractory lined pipe (26-in. or 660-mm ID) was suspended from the surface to the roof of the mine through a 4-ft (1.2-m) hole augered into the mine. This hole was sited at a position that did not reflect high underground temperatures, but was in good air communication with an underground region that did have elevated temperatures. The ground supports for all portions of the duct over the burn zone were floating in the sense that they rested on the ground with a wide footprint, and could be easily shored-up in the event of subsidence. The manifold was con-

[8] The main Pittsburgh seam at a nominal 6-ft (1.8-m) thickness would contain about 9000 tons (8000 t)/acre of minable coal. Conventional mining techniques would normally leave roof and floor coal, pillars, rider seams and carbonaceous shales, all of which would contribute to the total residual coal available through burnout control. Thus, with abandoned coal mines, as much coal (energywise) is left in place as has been mined.

[9] The thermal output of the exhaust could have been utilized if the section of the duct system where cooling air and water were introduced had been replaced with a waste heat boiler.

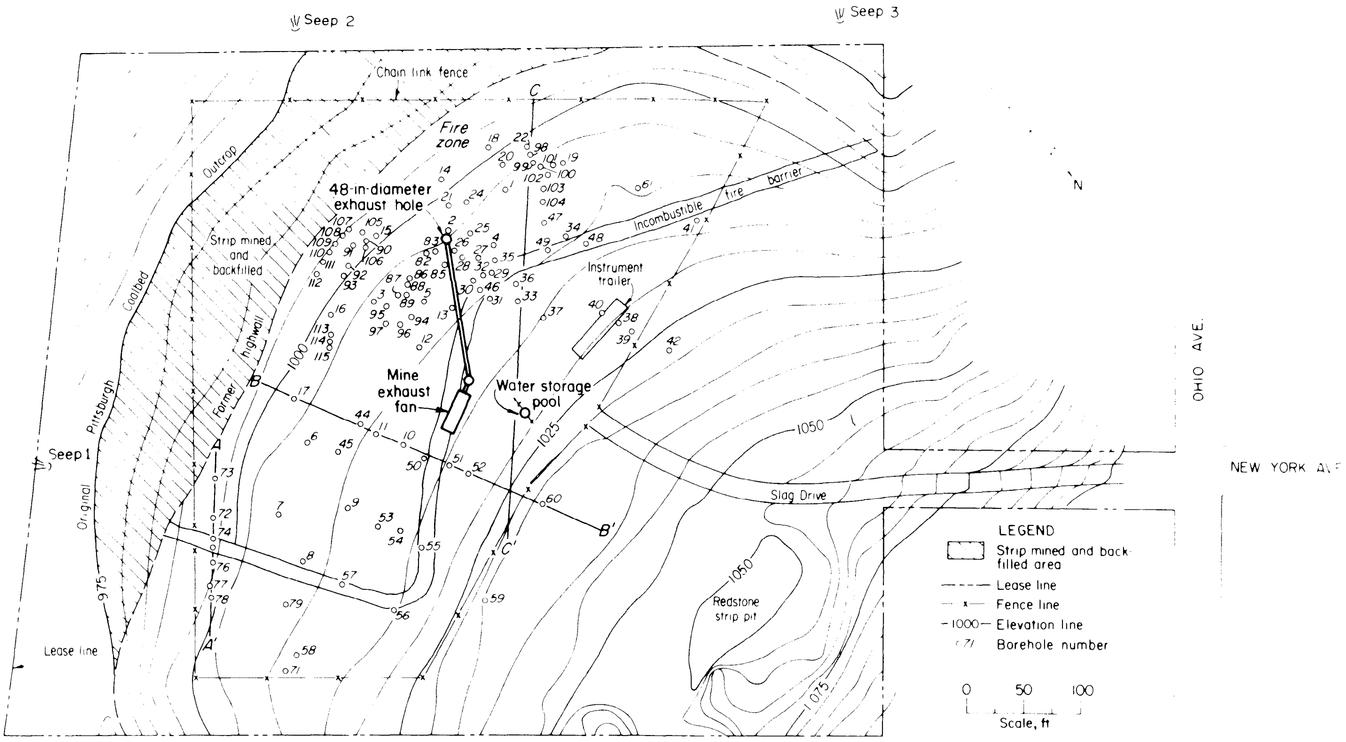

Fig. 22.6.21. Calamity Hollow site plan (Chaiken, Dalverny, and Kim, 1989). Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

nected to the fan through the steel duct system as depicted in Fig. 22.6.20. The refractory-lined portion of the duct system served as an afterburner where small amounts of ambient air could be introduced when necessary. A refractory gate valve (in the elbow section above the manifold) enabled the manifold to be sealed whenever the fan was shut down for any length of time.

Burnout control was carried out over a four-month time period in which the vacuum level in the mine was varied from 2 to 16 in. water (0.5 to 4.0 kPa). Both the gas flow (3000 to 12,000 scfm, or 85 to 340 std m^3/min) and thermal output (1 to 5 MW) from the mine correlated closely with the vacuum changes, while the exhaust temperatures changed only slowly with pressure, presumably due to the very large thermal inertia of the underground burn zone (Fig. 22.6.22).[10] This thermal inertia effect was beneficial for quickly resuming power output after fan shut-down, but made it difficult to alter the exhaust temperatures.

Gas analyses of the exhaust indicated a relatively oxygen-rich environment (13% excess O_2), with less than 0.1% fuel gas (i.e., unburnt CO and hydrocarbons) in the exhaust when its temperature exceeded 1100°F (600°C). When the exhaust temperature was less than 1100°F (600°C), the unburnt fuel content was about 1%. Fig. 22.6.23 depicts data accumulated on NO_X and SO_X produced during the latter part of the trial. The average *air-free* concentrations are 140 ppm for NO_X and 1660 ppm for SO_X. Comparing these observed values with the maximum possible concentration (i.e., 2600 ppm for NO_X and 1540 ppm for SO_X), calculated on the basis of the nitrogen and sulfur content in the residual coal it is seen that in situ coal combustion does bring out the SO_X from the coal, but produces far less NO_X than what might be expected. It can be anticipated that a commercial burnout control system would require air pollution controls for SO_X, but not for NO_X.

During the four-months' operational period of the field trial at Calamity Hollow, 1100 tons (1000 t) of residual coal were consumed to yield a flue gas with an average temperature of 1100°F (600°C) at an average thermal power level of 3.1 MW. A significant finding was that while subsidence occurred with numerous surface cracks and fissures appearing in the ground area above the burn zone, it was not sudden nor did it seriously effect the burnout control process—although it probably contributed to the excess oxygen observed in the exhaust.

22.6.3.4 Utilization of Burnout Control

The potential for burnout control is believed to be quite good, particularly for abandoned hilltop mine sites, which by their nature are isolated (or isolatable) from other coal seams and are not flooded. A number of current AML mine fires probably involve this type of site. In general, hilltop seams are dispersed and variable in size. In terms of utilizing burnout control, these factors call for the use of relatively small onsite power plants that can be relocated as required within a site or to different sites. Power plants up to 5 MW electrical could be mobile and should be suitable for firing by a single burnout control ventilation system.

Based on the Calamity Hollow findings, an abandoned mine in a 6-ft (1.8-m) seam of bituminous coal will contain about 4×10^{12} Btu/30-acre (4×10^{12} kJ/1.2×10^5-m^2) section of the mine, and a 30-acre (1.2×10-m^2) section of a mine could be placed under control from a single burnout control ventilation system. The system should be able to develop the necessary thermal power to operate a 5-MW electrical power plant for about seven years. Thus commercialization is envisioned as one

[10] The absence of data between days 11 to 29 in Fig. 22.6.22 is due to shut down of the fan to replace the diesel generator supplying electricity to operate the fan.

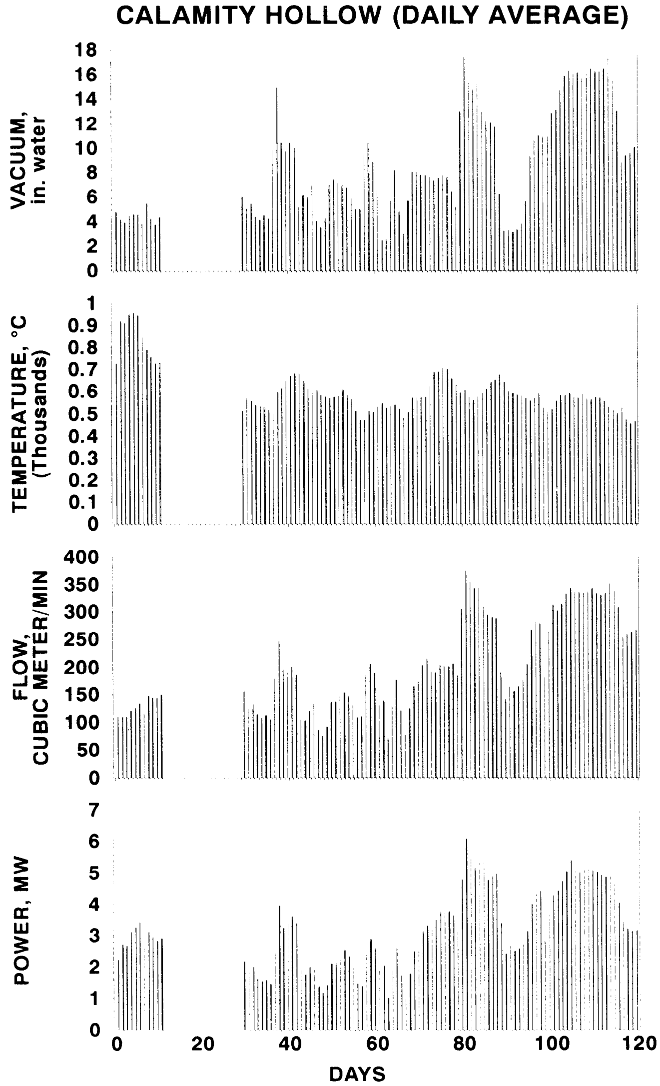

Fig. 22.6.22. Exhaust output, Calamity Hollow field trial (Chaiken, Dalverny, and Kim, 1989). Conversion factors: °F = 1.8°C + 32°, 1 cfm = 0.0283 m^3/min, 1 in. water = 0.249 kPa.

or more burnout control/power plant modules operating as a mobile surface facility, each connecting with the extensive electrical transmission network that covers the eastern United States. As each site or section is completely burned out, the module(s) is moved to a new location.

There are few detailed studies of the economics of burnout control as compared to other in situ combustion processes or even to UCG. However, it is significant that with in situ combustion, the underground burn zone takes the place of coal transportation and coal handling, air and fuel feed systems, fuel burners, and ash handling and cooling—processes which require 22 to 40% of the capital cost of the surface firing unit for a power plant (Maslan, 1980). Also the use of exhaust ventilation to supply air to the underground burning process vs. high-pressure injection (as with current UCG processes) leads to a low-energy expenditure to win the coal energy from the ground. The estimated energy gain from burnout control (10 to 15) at Calamity Hollow was comparable to that obtained with very large utility plants. Whether or not these aspects of a small mobile burnout

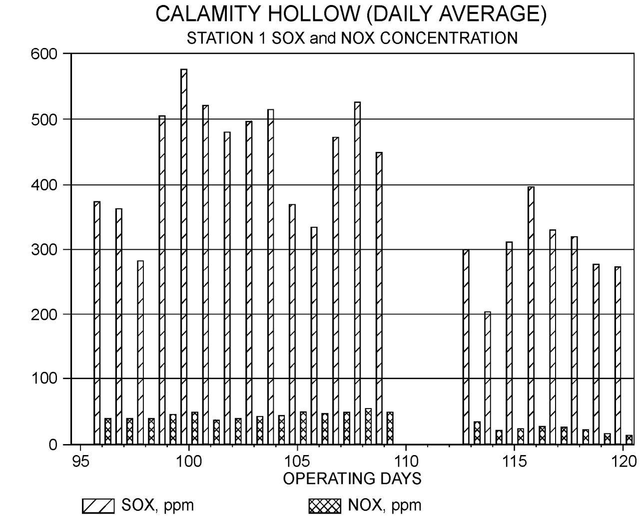

Fig. 22.6.23. Measured SO_X and NO_X in exhaust, Calamity Hollow field trial (Chaiken, Dalverny, and Kim, 1989).

control/power plant module would offset the economy of scale as represented by a 1000-MW (electrical) utility must await further trials of the small-scale in situ combustion process.

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Chapter 22.7
UNDERGROUND RETORTING

THOMAS E. RICKETTS

22.7.1 INTRODUCTION

The oil crisis of the mid- 1970s provided a somber reminder of the dependence of the United States on foreign oil and thus was a very effective incentive for the nation to build a synthetic fuels industry. At the time, the federal government called the need for a synthetic fuels industry the “moral equivalent of war” and formed the Synthetic Fuels Corp., a quasi-government agency to administer a huge synthetic fuels program. A synthetic fuels program was proposed rather than expanding crude oil development since US crude oil reserves are very limited and represent only about 1% of the world’s known total reserves. However, overall US fossil fuel reserves including crude oil, coal, tar sands, and oil shale are a significant part of the world’s fossil fuel reserves and, in fact, at that time represented on the order of 40% of the world’s fossil fuel reserves. This was due largely to massive domestic coal reserves. But the United States also has immense oil shale resources that represent close to 75% of the world’s oil shale reserves; thus oil shale evolved as an important part of the proposed synfuels industry.

Both private industry and the government committed significant resources to the development of shale oil technology and to commercial feasibility and demonstration projects involving various shale oil processes. However, a dramatic drop in oil prices in the mid-1980s due to world overproduction of crude oil essentially ended any major government support for a synthetic fuels industry and resulted in the abolishment of the Synthetic Fuels Corp. This lack of government support along with private industry’s ensuing lack of commitment virtually eliminated all development work in oil shale as well as other synthetic fuel technologies. The major shale oil processes that emerged during this development period are briefly described here. Later segments present the state of the art of the most advanced underground retorting technology that existed at the end of the development period when most work was discontinued.

The two primary types of shale oil processes that evolved were the aboveground retorting process and the underground retorting process. The *aboveground retorting process* consists of a retorting module(s) located aboveground at a mine site. The shale fed to the retorts is mined underground using conventional mining methods. The mined shale has to be handled several times and crushed to a specific size distribution. Ultimately, after processing, the spent shale has to be handled and disposed of in an environmentally acceptable manner. The room and pillar mining method is typically used to produce the oil shale since it can be designed to provide high grade shale for retorting. Room and pillar mining thicknesses of 30 to 60 ft (9 to 18 m) nominally provide oil shale having a grade of about 30 to 40 gallons of shale oil/ton of shale (125 to 170 mL/kg). Other large-scale mining methods such as sublevel stoping can also be used, but are designed for thicker mining intervals and thus cannot provide high-grade shale. For example, 200- to 300-ft (60- to 90-m) thick intervals typically provide oil shale having a grade of about 15 to 20 gallons of shale oil/ton of shale (60 to 80 mL/kg).

The *underground retorting process* involves constructing a large-scale rubble bed underground, then isolating the bed from the mine workings, and finally processing the bed in place. For shallow deposits, the bed can be constructed and processed from the surface using drillholes with no mining required. Thick deep deposits, however, require mining to provide a void volume into which the surrounding shale can be blasted to produce a permeable rubble bed for processing. This latter process is called the *modified in situ* process. For this process, 20 to 30% of the shale within a retort volume is mined with the remaining shale subsequently blasted into this void volume. Typically, the void is in the form of several room and pillar openings 30 to 60 ft (9 to 18 m) high and distributed within the retort volume. Thus the mining is done using conventional room and pillar mining methods but the typical configurations do not provide the most efficient bulk-mining type operations. After blasting the rubble bed, the chamber is sealed off from the mine workings and the bed is processed in place.

The underground retorting process typically provides less yield than the aboveground retorting process, so the former was designed for thicker (on the order of several hundreds of feet, or about 100 m) leaner-grade shale deposits. These, in fact, represent most of the US shale oil reserves. The aboveground retorting process was envisioned for thinner (on the order of several tens of feet, or about 10 m) richer-grade shale operations to take advantage of the higher yield provided by this process. However, the high yields expected for the aboveground retorting process have not yet been demonstrated on a large scale in rich shale. Conversely, excellent yields relative to predicted values have been achieved for the modified in situ process in full-scale demonstrations in lean shale with low-void-fraction rubble beds (Stevens and Zahradnik, 1983). New modified in situ technology for constructing rubble beds has subsequently been developed that is expected to give even higher yields, with increases estimated at 30% or more (Ricketts, 1983; Bickel and Ricketts, 1986; Hommert and Bickel, 1987). A combination of aboveground and modified in situ retorting results in the best total oil recovery and thus resource utilization for a given site. For example, it has been estimated that nearly 5 billion barrels (795 GL) of shale oil can be recovered from a 5000-acre (20.2-Mm^2) government lease tract in Colorado (the C-b tract). This would utilize a combination of the aboveground retorting and modified in situ processes within the nominal 1500-ft (460-m) thick oil shale section that exists in this part of the Piceance Creek Basin.

Three main underground retorting processes have evolved during the synthetic fuels shale oil development program. All three processes were developed as proprietary processes, and thus full particulars are not available, although publications and patents provide some information. The first underground retorting process to be described involves shallow deposits on the order of 100 ft (30 m) deep. For this configuration, the rubble bed is prepared and processed using drillholes from the surface, with no underground mining required. The void into which the shale is blasted is created by initially blasting a wedge of shale upward. The surrounding shale is then blasted into the void that results from this vacated wedge. This process is considered to be more of a “true” in situ method rather than a “modified” in situ process (Tyner et al., 1982; Lekas, 1985). The process has been demonstrated using commercial-sized rubble beds.

The second underground retorting process to be described is for deep deposits and initially involves mining a void room at the bottom of the section to be processed and then blasting layers of oil shale down into the void. The drilling, explosives loading, blasting, haulage, and processing operations can be done from either a mining level within the mine for deep deposits or from the surface for shallow deposits. When necessary, rubble is drawn from the lower void room after a layer is blasted to make room for the blasting of successive layers until the final height of the rubble bed is achieved. This process involves continuous mining operations to construct the rubble bed and results in a high-void-fraction bed that involves a large amount of rock handling and removal. The bed is ultimately isolated from the mine workings and processed in place. This high-void-fraction modified in situ process has been demonstrated using intermediate-sized rubble beds (Anon., 1977; Berry et al., 1982).

The final underground retorting process to be described is for deep deposits and involves mining multiple-void levels within the section to be processed and then blasting the surrounding oil shale into the void levels in a single blasting round. The amount of void mined is designed to produce a low-void-fraction rubble bed to minimize the rock handling, removal, and continuous mining operations required to construct a single rubble bed. The bed is ultimately isolated from the mine workings and processed in place. This low-void-fraction, modified in situ process was developed by Occidental Oil Shale, Inc., a subsidiary of Occidental Petroleum Corp., and represents the most advanced large-scale underground retorting process to evolve during the synthetic fuels development program. It has been demonstrated numerous times using commercial-sized rubble beds (Ricketts, 1980, 1982) and has resulted in excellent oil yields compared to the maximum attainable predicted value. The large-scale rubblization technique for this process, which is the key to any modified in situ process, has been developed into a second-generation technology that has advanced far beyond typical blasting state-of-the-art technology. Consequently, this low-void, modified in situ technology will be the process described in detail in the remainder of this chapter.

It should be noted that this process represents an evolving, state-of-the-art development process and is not an ongoing, routine underground operation; thus there are no generic, handbook-type guidelines for engineers to use. Instead, this chapter presents the latest results of this state-of-the-art development program. This allows an engineer to start at the most advanced stage of this development program and either use the existing technology or know where to begin a new development program.

22.7.2 UNDERGROUND RETORTING DESIGN CONSIDERATIONS

The development of an improved underground retort design and a more advanced blasting technology that resulted from Occidental's modified in situ development program is presented first. These design concepts provide better economics due to less expensive mining and processing operations and higher processing yields than the concepts used in the final retorts constructed in Occidental's development program. The improved retort and blast designs are based on new technologies that have resulted from extensive underground testing and engineering analysis applied to these data. They have not been demonstrated full scale but are based on full-scale test data and have been demonstrated at quarter scale. As such, they represent the concepts that would have been demonstrated full scale if the development program had continued. Other technologies relating to seismic energy prediction and blast damage to underground workings have also been developed based on Occidental's development program, but these are not described here (Redpath and Ricketts, 1987; Ricketts, 1988). Modified in situ processing technology is then presented. This represents the retorting techniques that were used on the final two retorts, Retorts 7 and 8. These were constructed at Occidental's Logan Wash Mine where the modified in situ development program was conducted. The results of these two retorts are covered in more detail as a case history at the end of this chapter.

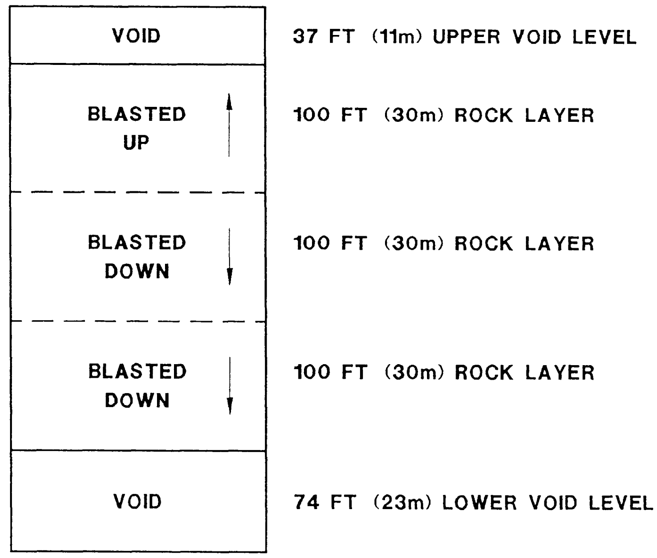


Fig. 22.7.1. Section view, two-void-level retort design (Ricketts, 1989).

22.7.2.1 Two-void-level Modified In Situ Retort Design

The final two retorts constructed in Occidental's modified in situ development program, Retorts 7 and 8, used a three-void-level design that allowed all the rock layers to be blasted into adjacent void levels at the same time (Ricketts, 1982, 1984); thus each rock layer was expanded into the same amount of void. At that time, it was not yet understood how rock bulked when blasted into a limited void volume. Thus a two-void-level retort design as shown in Fig. 22.7.1 was not attempted. This is because the first layer(s) uses up most of the void, if more than one layer is expanded into a single void room. Then, later layer(s) do not have sufficient void volume for expansion, resulting in poorly broken or unbroken sections in the retort. However, "limited void blasting" is now understood based on an analysis of extensive blasting data from Occidental's program. Such an approach can be used to design a two-void-level retort to provide a well-broken, vertically uniform rubble bed (Ricketts, 1989). This is important since the two-void-level design offers significant mining and blasting cost advantages as compared to the three-void-level design.

Limited void blasting refers to rock being blasted into a volume that is not sufficient to let it expand to its free-bulking value. Free bulking occurs when the rock particles have enough space to separate, tumble, and rotate to produce the maximum amount of voidage possible between the fragments when they finally come to rest. Although bulking value depends on particle size, distribution and shape, a typical free-bulking value for uniform, finely broken shale is about 1.67. The modified in situ development work has resulted in a limited void blasting curve,

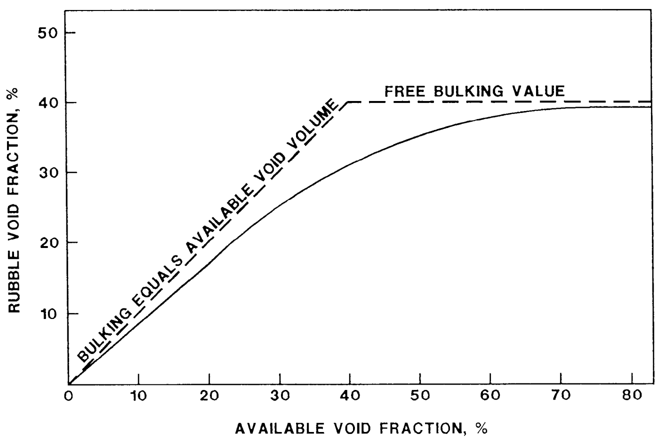


Fig. 22.7.2. Limited void volume blasting curve (Ricketts, 1989).

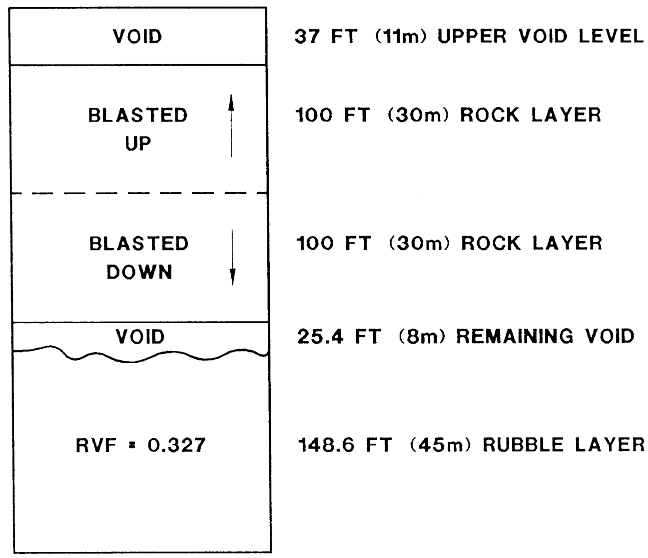


Fig. 22.7.3. Section view, two-void-level retort with bottom rock layer blasted (Ricketts, 1989).

as shown in Fig. 22.7.2. The *available void fraction* is defined as the ratio of the void volume into which a volume of rock is blasted divided by the sum of the void and rock volumes. The *rubble void fraction* is the actual fraction of void contained in a rubble bed and is given by the total rubble volume minus the original in-place rock volume (i.e., the actual void contained in the rubble bed) divided by the total rubble volume. A maximum free-bulking rubble void fraction value of 40% has been used in Fig. 22.7.2 as shown by the horizontal dotted line. *Bulking factor BF* is related to rubble void fraction *RVF* by the relation,

$$BF = 1/(1 - RVF) \qquad (22.7.1)$$

Thus a rubble void fraction value of 0.40 corresponds to a bulking factor of 1.67.

Also shown on Fig. 22.7.2 by a dotted line is the *intuitively expected bulking curve,* which is the bulking behavior that might be expected before actual data becomes available. For example, it might be expected that if rock is blasted into an available void volume that has less than the free-bulking volume, then the rubble would fill the void and have a void fraction value associated with the original volume. This is shown as the dotted line for available void fraction values below 0.40. For available void fraction values above 0.40, free-bulking values would occur that cannot be exceeded, and thus volumes in this range would not bulk full. However, it was found that volumes did not actually bulk full even for available void fraction values below 0.40, based on large retort blasts and extensive underground blasting programs (Ricketts, 1980, 1982). Thus blasting tests were specifically designed during later phases of Occidental's development program to resolve this limited void blasting issue.

The limited void blasting curve will be used in the following to design the most vertically uniform rubble bed possible for the two-void-level in situ retort design given in Fig. 22.7.1. This design consists of three 100-ft (30-m) thick layers of rock to be blasted into two void levels, a 37-ft (11-m) high upper void and a 74-ft (23-m) high lower void. The upper 100-ft (30-m) rock layer is designed to go up into the upper void and the lower 200-ft (61-m) rock layer down into the lower void. The rubble bed is constructed by first blasting the lowest 100-ft (30-m) layer downward, then waiting a sufficient time interval (on the order of 100 ms) before using explosive charges vertically centered in the upper 200-ft (61-m) layer to split this layer 100-ft (30-m) downward and 100-ft (30-m) upward. This splitting concept has been thoroughly tested and successfully demonstrated, and has been reported previously (Ricketts, 1980, 1982). This overall blasting method, however, causes a maldistribution in the vertical void fraction since the first down-layer uses more than its allotted proportion of void leaving less void for the second down-layer to expand into. This is the result of the first layer having an available void fraction value that is larger than the average value for the retort, thus allowing the first layer to bulk more than the average. The limited void blasting curve can be used to minimize the effects of this detrimental feature of the two-void-level retort design.

The overall available void fraction value for the design shown in Fig. 22.7.1 is 0.27. It will now be assumed that the minimum-void-fraction value that will be accepted in any section of the bed is 0.20, which is based on pressure drop and processing experience on previous large-scale retorts. (Note: Pressure drop results from the final two retorts constructed at the Logan Wash mine suggest that even a rubble void fraction of 0.18 is not totally unacceptable.)

This 20% minimum-void-fraction requirement leads to the overall available void fraction value of 0.27 for the two-void-level design. The available void fraction of the first 100-ft (30-m) thick layer blasted down in Fig. 22.7.1 is 0.425 (i.e., 74/174). The limited void blasting curve gives a rubble void fraction value of 0.327, which translates to a bulking factor of 1.486. Thus the 100-ft (30-m) layer expands to 148.6 ft (45.3 m), which results in the hypothetical configuration shown in Fig. 22.7.3. The second 100-ft (30-m) thick layer blasted down thus has 25.4 ft (7.7 m) of void to expand into, which is not symmetrical with the 37-ft (11-m) upper void room into which the upper 100-ft (30-m) thick layer is blasted. The second down layer has an available void fraction of 0.203 (i.e., 25.4/125.4), which results in a rubble void fraction value of 0.18 from the limited void blasting curve. This is below the 0.20 minimum desired rubble void fraction, and thus the chamber cannot be blasted using 100-ft (30-m) rock layers.

A more uniform vertical void fraction distribution is desirable in order to get the center section from a rubble void fraction value of 0.18 up to at least 0.20. This redistribution can be attempted by reducing the available void fraction for the first down-layer so it will bulk less and leave more void for the second

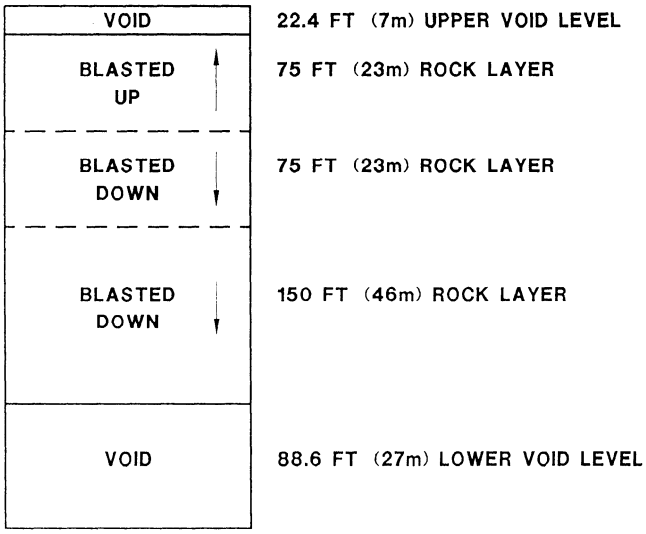


Fig. 22.7.4. Section view, improved two-void-level retort design (Ricketts, 1989).

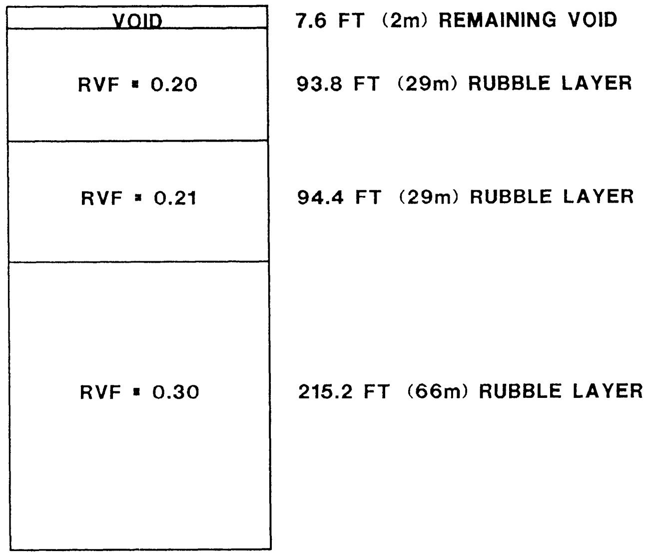


Fig. 22.7.5. Section view, improved two-void-level retort blasting results (Ricketts, 1989).

down-layer. The available void fraction of the first layer blasted down can be reduced by increasing the first layer thickness, say, to 150 ft (46 m). This would leave a 150-ft (46-m) thick upper rock layer to be split up and down into the upper void room and lower void that remains after the first layer is blasted down. This is less rock than that blasted originally, so the upper void room height should be reduced to give a rubble void fraction value at the minimum acceptable value for the upper rock layer. A rubble void fraction value of 0.20 for the uppermost rock layer that is blasted into the upper void level translates to an available void fraction value of 0.23 from the limited void blasting curve. For this value, the upper void room height can be reduced to 22.4 ft (6.8 m). The 14.6 ft (4.5 m) taken from this void level should be put into the lower void level to keep the overall available void fraction value for the retort fixed at 0.27. Thus the lower void room would be 88.6 ft (27 m) high, and the new two-void-level design is shown in Fig. 22.7.4.

The blasting of this improved design begins with a 150-ft (46-m) thick rock layer being expanded down into an 88.6-ft (27-m) lower void level. The available void fraction of this configuration is 0.371 (i.e., 88.6/238.6), which translates to a rubble void fraction of 0.303 using the limited void blasting curve. This corresponds to a bulking factor of 1.435, so the 150-ft (46-m) layer expands to 215.2 ft (66 m). This leaves 23.4 ft (7.1 m) of lower void for the expansion of the 75-ft (23-m) thick second down-layer. This 23.4 ft (7.1 m) of lower remaining void is fairly symmetric with the 22.4-ft (6.8-m) upper void, which gives a fairly uniform rubble bed for this case. The available void fraction for the second down-layer is 0.238 (i.e., 23.4/98.4), which gives a rubble void fraction of 0.205 and a bulking factor of 1.258. Thus this 75-ft (23-m) thick layer expands to 94.4 ft (29 m), which leaves 4.0 ft (1.2 m) of void for this layer. Recall that the upper 150-ft (46-m) thick layer for this design is split up and down at the same time using explosive charges vertically centered in the layer, so the remaining voids from the down layers will be translated to the top of the chamber. The available void fraction for the uppermost rock layer is 0.23, which gives a rubble void fraction value of 0.20 and a bulking factor of 1.25. Thus this 75-ft (23-m) thick layer expands to 93.8 ft (28.6 m), which leaves 3.6 ft (1.1 m) of void from this layer. This results in a total void of 7.6 ft (2.3 m) at the top of the chamber. The overall results of blasting this chamber using the improved blast design are shown in Fig. 22.7.5.

The lateral dimensions of the in situ chambers must now be discussed since it has been found that the lateral dimension actually controls the maximum layer thickness that can be blasted in the design. Extensive testing has shown that the maximum layer thickness that can be blasted using short-delay blasting is about one-half of the minimum lateral dimension of the chamber (Ricketts, 1979). Thus, for example, it would be possible to blast a 100-ft (30-m) thick rock layer for a 200-ft (61-m) square chamber. This maximum layer thickness that can be blasted is tied to lateral confinement of the layer just as in a heading round; that is, if too deep a heading round is attempted, it will not pull to the full depth and could freeze in place. The maximum heading-round depth is related to the drift cross-sectional size, and the larger the opening, the deeper the round that can be pulled. This concept becomes more restrictive as shorter delays are used in the round design. In the above example, a maximum layer thickness of 150 ft (46 m) resulted from the limited void blasting analysis, and thus the chamber should be at least 300 ft (91 m) square. The retort could also have a rectangular cross section, with the smaller dimension of at least 300 ft (91 m).

The use of pillars on the void levels must also be discussed since pillars must be used for large openings. No pillars were used in the above discussion to keep the analysis simple. These pillars support the roof while drilling and blasting operations are being done in the void rooms. However, to properly blast the main rock layers located between the void levels, all the void-level pillars must be removed before the main rock layers are blasted. Thus these pillars are considered temporary support pillars that must be blasted just before the main rock layers are blasted and still be in motion as the main layers expand into the void rooms. This sequence is well understood and has been described previously (Ricketts, 1980, 1982). When these pillars are present in the design, they must be included in the limited void blasting analysis. This is done by blasting them initially and letting them bulk to the free-bulking factor, then accounting for the void they will use up that will not be available for the main rock layers. The addition of pillars thus requires larger void-room heights to account for the expanded pillar rubble in order to keep the overall chamber void fraction at the desired value.

22.7.2.2 Blast Design for Uniform-flow Rubble Beds

The final two retorts constructed in Occidental's development program, Retorts 7 and 8, showed very good reproducibility between rubble beds. They also represented a definite improvement in rubble bed uniformity compared to previous retorts. However, based on thermocouple results during retorting, it was recognized that there was still room for improvement in rubble bed uniformity. One way to accomplish this was to attempt to produce uniform rubble properties across the bed. This had been the approach up to that time. An alternative was to produce a bed with a combination of nonuniform properties designed to provide a uniform flow distribution across the bed. This would result in a flat combustion front and greater oil recovery.

This new approach of constructing nonuniform rubble beds to provide uniform flow was developed using the results of the fragmentation testing and large-scale retort blasts in Occidental's development program (Ricketts, 1983). This concept has subsequently been modified to provide uniform retorting rubble beds rather than uniform flow beds and has been demonstrated on a small scale (Bickel and Ricketts, 1986). This modification causes only minor adjustments to the blast design; thus the remaining discussion follows the development of the uniform flow concept.

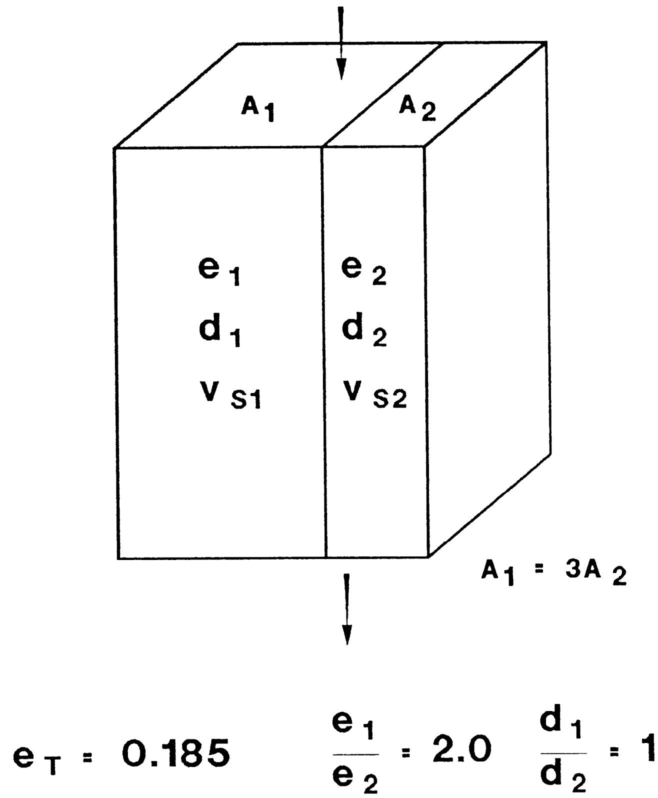
The uniform flow approach provides the design of a nonuniform rubble bed with the proper combinations of void fraction and particle size in different sections of the bed such that the mass flow rate per area during retorting is constant across the bed. This supplies equal oxygen flux to the combustion front all across the bed, which should result in a flat, uniform progression of the combustion front downward through the rubble bed.

The flow in a packed bed can be described by Ergun's equation that relates pressure drop through a bed to particle size and shape, rubble void fraction, and superficial flow velocity (Kunii and Levenspiel, 1977). The rubble beds in Retorts 7 and 8 constructed at the Logan Wash mine can basically be treated as beds with two regions of differing rubble properties, as shown in Fig. 22.7.6. Region A2 is of higher permeability and is found around the perimeter of the bed (Ricketts, 1983). If Ergun's equation is applied to both regions, the equations can be combined and rewritten in terms of particle size ratio. For equal superficial velocities in the two sections of the bed, that is, $Vs_1 = Vs_2$, this relation represents a uniform mass flow rate per area for a bed having two regions. The resulting expression gives the particle size ratio required for perfectly uniform flow in these regions as a function of rubble void fraction. This relation appears as

$$d_2/d_1 = (1 - e_2/1 - e_1)\,(e_1^3/e_2^3)^{1/2} \qquad (22.7.2)$$

where d is mean particle size and e is rubble void fraction.

The rubble void fraction and particle size values in the different sections of the rubble bed can be controlled using fragmentation data obtained in Occidental's modified in situ development program. These fragmentation data are given in Figs. 22.7.7 and 22.7.8. Fig. 22.7.7 shows rubble void fraction (which is related to bulking factor) as a function of powder factor and blasthole spacings. Fig. 22.7.8 gives mean particle size as a function of powder factor and blasthole spacing. Fig. 22.7.7 shows a trend of increasing void fraction for increasing powder factor for all blasthole spacings. The curves also show, for example, that for a fixed powder factor, the bulking can be increased by increasing spacing. Likewise, if a fixed void fraction is desired, a decrease in spacing would require an increase in powder factor. Fig. 22.7.8 shows a trend of decreasing particle size with increasing powder factor for all blasthole spacings. These curves also show, for example, that for a fixed powder factor, the particle size can be increased by increasing the spacing, and that if a fixed particle size is desired, an increase in spacing would require an increase in powder factor.



Fig, 22.7.6. Idealized rubble bed configuration for Retorts 7 and 8 (Ricketts, 1983).

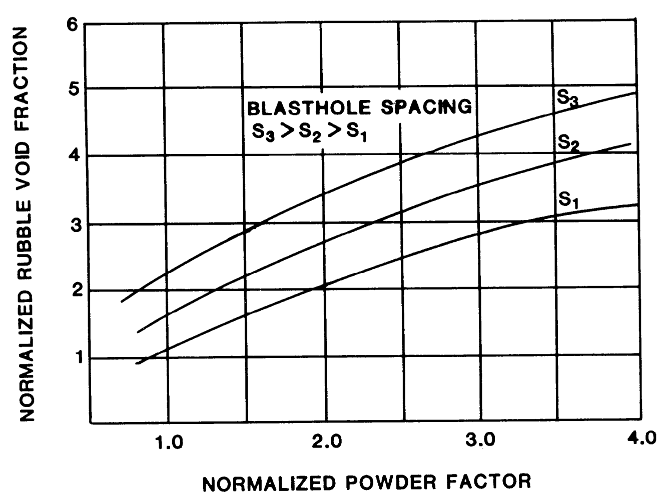


Fig. 22.7.7. Blasting curves relating rubble void fraction to powder factor and spacing (Ricketts, 1983).

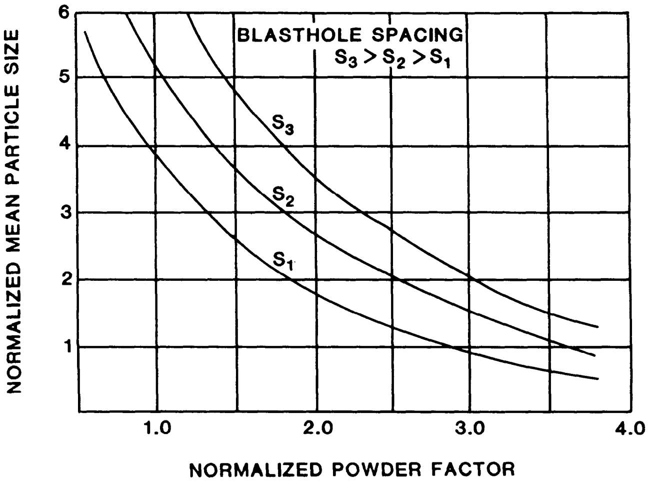


Fig. 22.7.8. Blasting curves relating particle size to powder factor and spacing (Ricketts, 1983).

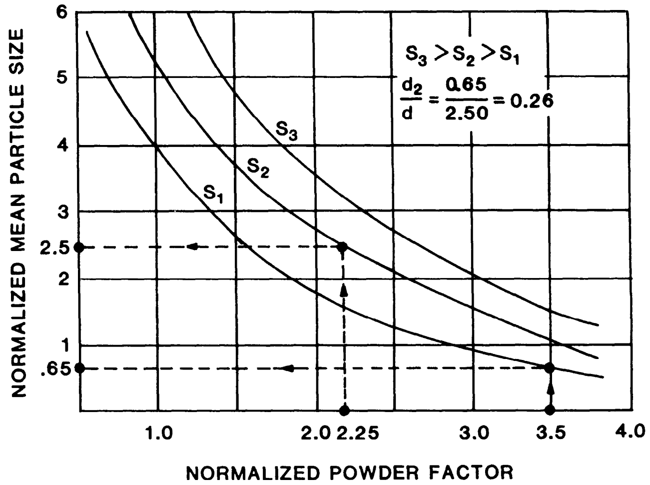


Fig. 22.7.10. Particle size curves for Retorts 7 and 8 uniform flow design (Ricketts, 1983).

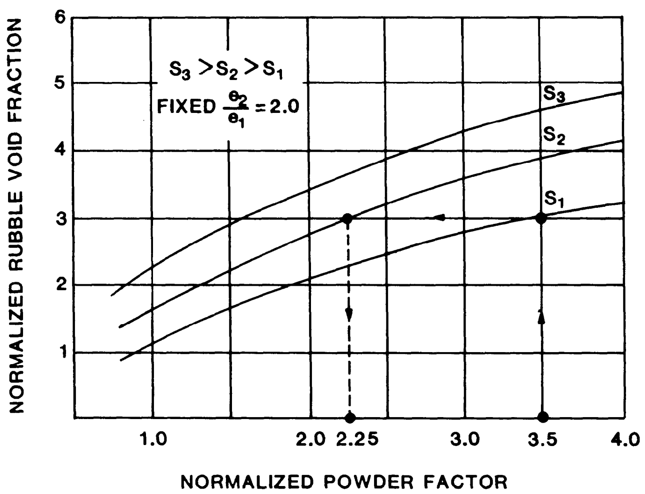


Fig. 22.7.9. Rubble void fraction curves for Retorts 7 and 8 uniform flow design (Ricketts, 1983).

The idealized rubble bed configuration representing Retorts 7 and 8 is shown schematically in Fig. 22.7.6 with rubble properties $e_T = 0.185$, $d_1 = d_2$ and $e_2/e_1 = 2.0$. This rubble bed was blasted using a spacing of 20 ft (6.1 m) and a normalized powder factor of 3.5, which is nominally 1.6 lb of ammonium nitrate-fuel oil (ANFO) explosive equivalent/ton of shale rubblized (0.80 kg/t). The design of Retorts 7 and 8 will be discussed in more detail in the case history, at the end of this chapter.

The two sections of differing rubble properties in Retorts 7 and 8 were divided such that the interior area $A_1 = 3A_2$. Based on 22.7.2, the required value of the particle size ratio d_2/d_1 is 0.29 for perfectly uniform flow conditions in the two sections of the bed, that is, for $Vs_1 = Vs_2$. The rubble void fraction curves in Fig. 22.7.9 show that increasing the spacing to 25 ft (7.6 m) in area A, requires a decrease in normalized powder factor to 2.25 to keep the rubble void fraction constant. This is nominally 1.0 lb ANFO equivalent/ton of shale rubblized (0.5 kg/t).

Keeping the rubble void fraction fixed in area A_1 is not necessary in this analysis. However, it results in the simplest case for designing a uniform flow distribution, since the rubble void fraction in area A_2 then also remains fixed. Fig. 22.7.10 then shows that the normalized particle size increases from 0.65 for the original blast design to 2.5 for the uniform flow design. This gives a ratio of $d_2/d_1 = 0.26$ compared to the desired value of 0.29 associated with perfectly uniform flow. A particle size ratio of 0.26 gives a flow velocity ratio of 0.80 for the two regions. This represents an acceptably uniform flow distribution, which, in fact, is very close to a uniform retorting design (Bickel and Ricketts, 1986). An iterative process could be used at this point to obtain a perfectly uniform flow design from the blasting curves in Figs. 22.7.9 and 22.7.10. However, in this case, construction of a rubble bed that exhibits a slightly nonuniform flow distribution is actually preferred. This is due to the higher density of rubble in the interior region, which requires more air flux per unit volume to retort compared to the perimeter region (Ricketts, 1983; Bickel and Ricketts, 1986).

In summary, the improved uniform flow design developed here involves increasing the blasthole spacing 25% in area A_1 and decreasing the powder factor 36% in this same area as compared to the blast design used in the final two retorts constructed in Occidental's development program. These two factors applied to three-fourths of the total retort cross section represent a significant blasting cost reduction. The flow results for a perfectly uniform flow design along with those of Retorts 7 and 8 are given in Table 22.7.1. These results show the calculated flow properties for the actual rubble beds in Retorts 7 and 8 and the redistribution of flow that occurs by using a uniform flow design. It can be seen that making $d_1 = 3.45\ d_2$ in area A_1, while keeping the void fraction fixed, causes a redistribution of flow such that the flow rate per area is uniform across the bed.

The most significant result of this flow redistribution is the three-fold reduction in pressure drop for the uniform flow design as compared to the Retorts 7 and 8 rubble beds. This translates to a sizable cost savings during commercial processing operations since blower pressure is required for about 300 days to completely retort a commercial rubble bed. Thus the total cost benefits gained by use of the uniform flow design will be significant for a commercial operation. Results include a reduction in rubblizing costs, a decrease in blower operating costs due to a substantially lower pressure drop in the bed, and an increase in the oil yield due to a more uniform retorting front.

Table 22.7.1. Flow Parameters for Retorts 7 and 8 and Uniform Flow Designs

Flow Parameter	Retorts 7 & 8 Design	Uniform Flow Design
Pressure drop per 100 ft (30 m), psi	1.94	0.61
(kPa)	(13.4)	(4.2)
Central region (A_1) flow rate, scfm	3,228	12,005
(scms)	(1.52)	(5.67)
Perimeter region (A_2) flow rate, scfm	12,750	4,001
(scms)	(6.02)	(1.89)
Central region (A_1) superficial velocity, fpm	0.24	0.90
(mm/s)	(1.2)	(4.6)
Perimeter region (A_2) superficial velocity, fpm	2.9	0.90
(mm/s)	(14.8)	(4.6)
Particle diameter ratio, d_1/d_2	1.00	3.45
Void fraction ratio, e_2/e_1	2.0	2.0

Source: Ricketts, 1983.

22.7.2.3 Processing of Modified In Situ Rubble Beds

The processing phase of the modified in situ operation represents an extensive part of the overall process and consists of numerous events that start after the retorts are mined and rubblized. The following processing considerations reflect the technology used on the final two retorts processed in Occidental's development program. A more detailed discussion of what was done on these retorts is given as a case history in 22.7.3.

CONSTRUCTION REQUIREMENTS. After the chambers are rubblized, it is necessary to construct bulkheads in the entry drifts into each void room to isolate the retorts both from the mine workings and from each other. A description of steel bulkheads used for this function, their construction, installation, and insulation, is given elsewhere (Nelson, 1982). To prevent leaks around the edges of the bulkheads, they are typically sealed in place with a blown-in ceramic-masonry grout. For drifts between retorts that are inaccessible after rubblization, it may be necessary to construct barriers to flow between the retorts prior to rubblization. A cost-effective technique of doing this consists of filling the drift with sand, protected at each end by mine-run rock.

Processing facilities typically include blowers and piping needed for supplying process air, and steam to the retorts. Other blowers, operating in the suction mode, can be used to pull the process gases through the retorts. Control valves at the product level bulkheads permit maintaining subatmospheric pressure within each retort, to prevent leakage of retorting gases into the mine ventilation air.

The off-gas from the retorts is transported to the surface processing plant using a cased raise. There, counter-flow direct-water-contact condensers can be used to condense water and C_5 + hydrocarbons from the off-gas.

A water-spray nozzle system can be installed in the collection drift on the product level of each retort. Late in the operation of the retorts, these spray systems are used to cool the high-temperature off-gas to temperatures acceptable to the bulkheads, downstream piping, and processing equipment.

PROCESSING INSTRUMENTATION. Direct periodic measurement of primary variables throughout the processing system and temperatures within and around the retorts should be collected and recorded preferably using a computerized data acquisition system (Kimball and Ogden, 1980). Frequency of measurement depends upon the anticipated rate of change of the variables. On-line analyses of off-gas compositions can be made continuously by specific-gas analyzers (oxygen in the off-gas, for example) and periodically by gas chromatographs. Operating conditions can then be computed and displayed on TV monitors for timely process evaluation and control.

A certain minimum number of key processing measurements are required on all retorts, especially in the inlet and off-gas streams, to safely control the process for maximum oil yield. The initial retorts in a commercial operation will require extensive instrumentation including measurements in the rubble bed so the retorting process can be understood and optimized for maximum oil yield. Thermocouples and gas sample taps are typically placed in the rubble beds after rubblization. Holes are drilled from access drifts through the intact shale surrounding the rubble beds and into the rubble to the desired location with the drill pipe left in place. Prior to installing the thermocouple and gas sample tap assemblies in the drill pipes, directional surveys should be made of the pipes. Survey results and thermocouple/tubing lengths then provide an accurate location of each thermocouple and gas sample tap for subsequent mapping of temperature, pressure, and gas composition distributions within the rubble beds during retorting.

PRE-IGNITION TESTING REQUIREMENTS. After completion of the gas-sample tap and thermocouple installation, and construction of the bulkheads and off-gas handling systems, tests should be run to determine the leak rates, pressure drops, and flow contours within the rubble beds. Results of these tests are used to establish safe and efficient operating conditions as well as any special processing strategy that needs to be used.

Leak tests are first conducted to locate and correct leaks and to minimize uncertainties in subsequent pressure drop tests. With blowers downstream from the retorts in the process gas stream, pressures in the retort chambers are kept subatmospheric during processing; thus any air leakage will be from the mine workings into the retorts. The maximum acceptable leakage rate can be established by the safe upper limit of 2% oxygen in the combustible off-gas during process operations. The pressure drop through the retort rubble is expected to increase with time through the retort like. Likewise, the mean pressure and leakage rate into the retorts is likewise expected to increase with time. Thus it is appropriate to determine cold-flow pressure drop values and from these predict hot-flow pressure drops in the retorts.

It is believed that during retorting, the increased pressure drop is partly due to an increase in gas flow generated from both combustion and pyrolysis, and partly to an increase in viscosity of process gases as the rubble bed temperature increases. The effective permeability also decreases as liquid products compete with process gases for void space, and effective void space also decreases slightly as the shale expands (exfoliates) during the retorting process. No evidence has been found to suggest that the rubble beds slump during retorting. Typical pressure drop values for rubble beds 240 ft (73.2 m) high having an average rubble void fraction of 0.185 are on the order of 15 in. water (3.7 kPa) for cold flow at a rate of 12,000 scfm (5.7 scms) and about 60 in. water (14.9 kPa) during retorting. Most of the pressure drop occurs at the bottom of the retort in the outlet region where turbulent flow conditions exist. For example, the pressure drop for a 200-ft (61-m) section of rubble bed away from any inlet and outlet effects is less than 1.0 in. water (249 Pa) for the void fraction and flow rate values specified above.

RUBBLE BED IGNITION PROCEDURE. The retort startup procedure was developed using a series of laboratory tests and laboratory pilot plant experiments, with the resulting procedure successfully tested underground on intermediate-sized rubble beds before it was used on full commercial-scale retorts. This startup procedure begins with the uniform heating of the top surface of the rubble bed to ignition temperature using an external heat source. The retort and blasting designs have been developed to provide a void room above the rubble bed after the retort has been rubblized. This is specifically for the start-up phase of the processing operation. Open access to the entire top of the rubble bed depends on the roof stability over the bed (which can represent a very large span) both before and during the ignition procedure. It has been found that ignition over the entire top of the bed can still be accomplished after a roof fall due to the large particle size of the roof-fall material and the large voids that exist between these particles. When ignition temperature is achieved, oxygen (air) is gradually supplied to the top of the bed to start combustion within the rubble. When sufficient combustion has been initiated on the rubble surface, the external heat source is turned off.

Typically, a hot inert-gas generator has been used, with shale oil as the fuel, to inject hot (inert) products of combustion (water quenched to 1600°F or 870°C) into the void above the bed. With proper design of the outlet nozzle from the generator, the void above the bed can act as a completely stirred tank that distributes the hot gas uniformly over the rubble. A slight excess of oxygen in the generator can be used as a tracer to detect the onset of combustion at 750°F (400°C), which can be monitored as the disappearance of oxygen in the retort off-gas. At a generator firing rate of 100 million Btu (106 GJ) per hour, about 18 hours will be required for the onset of ignition. The complete transition to air typically takes about 6 hours, at which time the generator can be shut down and moved to the next retort.

RETORTING OPERATION. After ignition, the retorts are typically operated without interruption until gas injection is terminated and water quenching begins. A simplified process flow diagram for operating two retorts is shown in the case history at the end of this chapter. Sumps behind the product level bulkheads are used for the collection, separation, measurement, and sampling of the bulkhead oil and water pumped from the retorts. After transport to the processing facilities, the off-gas can be processed through contact condensers. Suction blowers can be used to withdraw off-gas from the contact condensers and exhaust the gas to the stack, where the gas is discharged to the atmosphere by permit.

The off-gas from the retorts and the stack gas are typically analyzed by on-line gas chromatographs throughout the operation. Standard EPA isokinetic samples of the off-gas can be collected for daily analysis from the retorts at the product level bulkhead and from the off-gas at the base of the raise, top of the raise, exit of each contact condenser, and the stack. Stack isokinetic measurements can be used to determine the daily barrel equivalent of uncondensed C_5+ hydrocarbons exhausted up the stack. Water and light oil recovered at the contact condensers can be measured daily and sampled periodically for analysis.

Near the end of retorting, the off-gas temperature will eventually rise above 250°F (121°C) which represents the safe limit for the product level bulkheads and the retort processing will have to be terminated. If this occurs too early during retorting, it can cause a significant reduction in sweep efficiency and result in a loss of total oil production. It is also possible for oxygen breakthrough to occur, requiring termination of retorting. However, this has never been observed during Occidental's development program. In order to extend the life of the retorts, the off-gas can be cooled to reduce the thermal load on the bulkheads and downstream processing equipment. The off-gas can be cooled by (1) utilizing a water-spray system to cool the off-gas in the production drifts behind the bulkheads; (2) reducing or eliminating steam in the inlet gas; (3) reducing the inlet flow rate; and (4) injecting recycle gas instead of air.

SHALE OIL PRODUCTION. The shale oil recovery from modified in situ retorts can be expected to be on the order of 70% yield of the Fischer assay oil in place in the rubble that was actually blasted. This yield value is for retorts having an average shale grade of 15 to 20 gallons of shale oil/ton of shale (60 to 80 mL/kg). This oil production includes oil pumped from the retort sumps, oil mists and vapors condensed in contact condensers, condensable C_5 + vapors found in the stack gas, and a small amount of oil in the bulkhead water. The partitioning of the total product between these four streams for actual retort production is given in the case history segment (22.7.3). Cumulative production curves for modified in situ retorts are also given in 22.7.3.

It is believed that a yield value of 70% for modified in situ retorts is on the conservative side since one of the final two retorts constructed at the Logan Wash mine provided a higher yield and future retorting operations will always strive for the highest yield value possible. In addition, these final two retorts were constructed using first-generation technology. Improved technology has now been developed (discussed earlier in this segment) that will result in more uniform rubble beds and thus higher oil yields. Sandia National Laboratories has in fact estimated that this technology could provide increases of 30% or more in oil yield, which is significant (Hommert and Bickel, 1987). It is expected that future retorts will be located in richer shale than was available at the Logan Wash mine, and this will also provide higher yields since yield is a function of shale grade and increases with increasing grade. It should be noted that the definition of yield used here is based only on the rubble that was actually blasted in the retorts and does not account for oil residing in roof-sloughing rock and the retort walls that enter into the process.

22.7.3 UNDERGROUND RETORTING CASE HISTORY

22.7.3.1 Modified In Situ Development Background

The development of Occidental's modified in situ technology began in 1972 with the construction and processing of four underground retorts. The first three chambers were about 30 ft (9 m) square and 72 to 113 ft (22 to 34 m) high. These test retorts examined both vertical- and horizontal-mined void configurations at various rubble void fraction values. The first two used a single vertical raise in the center of each retort and a bottom room as the void, while the third used three horizontal rooms located at the top, bottom, and center of the retort volume as the void. The horizontal void test provided a slightly better yield than the other two. The fourth retort was 120 ft (36.6 m) square by 298 ft (90.8 m) high and used two vertical void slots extending parallel to one another across the cross section. The oil yield results for this retort were not as good as the small-scale test retorts. Rock stability problems involving sloughing of rock into the slots played a role in the reduced yield for Retort 4.

Retorts 5 and 6 were then constructed and processed under phase 1 of a joint Occidental/Department of Energy cooperative agreement that started in 1976 (Loucks, 1977). This program compared commercial-sized, vertical-void and horizontal-void retorts. Retort 6, the horizontal-void retort, resulted in significantly more oil recovery than the vertical-void design. This increase occurred in spite of a loss of access to the inlet piping system and subsequent loss of control over the combustion front

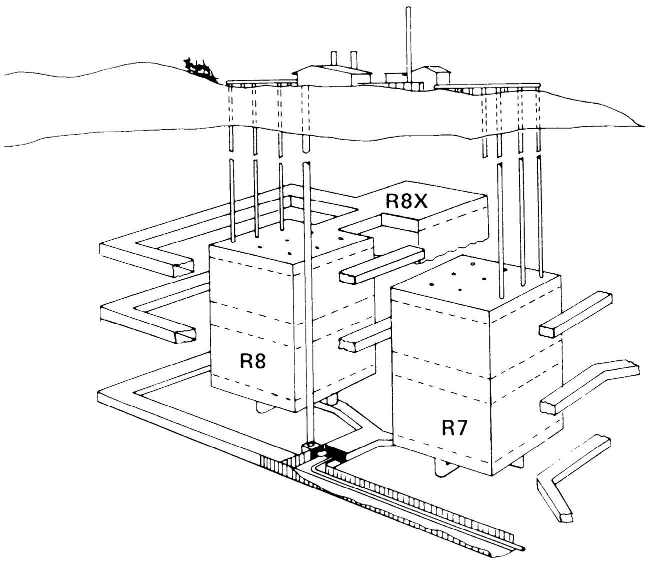


Fig. 22.7.11. isometric view Retorts 7 and 8 and Partial Retort 8X (Ricketts, 1982).

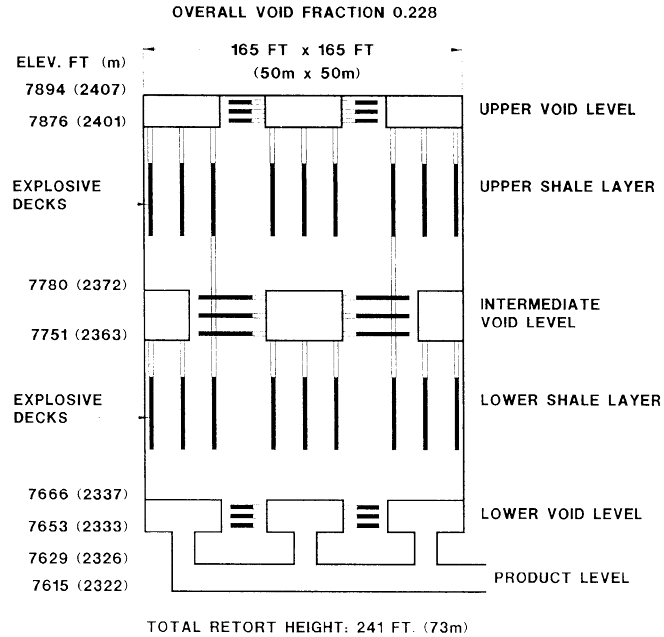


Fig. 22.7.12. Section view Retorts 7 and 8 blast design (Ricketts, 1982).

during the processing of Retort 6. Consequently, the horizontal-void design was used for phase 2 of the cooperative agreement that constructed and processed Retorts 7 and 8 (Nelson, 1981; Romig, 1981). Retorts 5 and 6 are well documented in government reports required under the cooperative agreement, and the processing performances have been modeled by government laboratories (Gregg and Campbell, 1980; Campbell, 1981). The final two retorts completed during Occidental's development program, Retorts 7 and 8, are described in detail in the following case history.

22.7.3.2 Retorts 7 and 8 Objectives

The primary objectives in operating Retorts 7 and 8 were to construct and process two side-by-side commercial-sized retorts in order to (1) establish the reproducibility of the blasting results for two commercial-sized retorts using the same retort design, blast design, and mined-out void volume; (2) start up and process two identically constructed side-by-side retorts connected to a common off-gas line; and (3) obtain rock mechanics and mine design data for a multiple retort system for use in ongoing commercial mine design efforts.

To increase the amount of rock mechanics data relevant to the commercial mine design, a partial height retort, Retort 8X, was added next to Retort 8 (Fig. 22.7.11). This partial retort had a full-scale cross section 165 ft (50 m) square, but was only 63 ft (19 m) high when rubblized as compared to about 240 ft (73 m) for Retorts 7 and 8. The distance between retorts 7 and 8 was about 160 ft (49 m), while the spacing between Retorts 8 and 8X was 50 ft (15 m), both of which were representative dimensions for the clusters of retorts envisioned in the commercial mine design. By adding Retort 8X, additional data on the stresses, deformations, and fracturing for two full-scale openings located very close to one another could be evaluated (Ricketts, 1982). In addition, Retort 8X was rubblized first, which increased the reliability and productivity for the subsequent full-scale blasting operations. The Retort 8X blast involved a total of 205 blastholes loaded with 143,100 lb (64,910 kg) of explosive in ANFO equivalent. This was a large-scale blasting operation even though it was only a partial retort.

22.7.3.3 Retorts 7 and 8 Design

Retorts 7 and 8 used a three void-level-design so that all the rock layers could be blasted into adjacent void levels at the same time. In this way, each rock layer expanded into the same amount of void volume. The design of these retorts (along with the blastholes used for rubblization) is given in Fig. 22.7.12. The retort design for both retorts consisted of three 165-ft (50-m) square horizontal-void levels, an upper level 18 ft (5.5 m) high, an intermediate level 29 ft (8.8 m) high, and a lower level 13 ft (4.0 m) high. There were two shale layers between these void levels, the upper 96 ft (29 m) thick and the lower 85 ft (26 m) thick. There was a product level 14 ft (4.3 m) high for collecting oil 24 ft (7.3 m) below the floor of the lower void level. The product level provided access for the retorting outlet system, which consisted of five 8-ft (2.4-m) diameter raises. One raise was located at the center of the retort cross section with the other four being positioned 24 ft (7.3 m) in from the center of each side wall, as measured perpendicularly in from the side walls to the raise centerlines.

The upper- and lower-void levels each contained two pillars 16 ft (4.9 m) wide by 150 ft (46 m) long. These pillars were oriented parallel to one another and were positioned about 52 ft (16 m) from the side walls that were parallel to the pillars, as measured to the pillar centerlines. The intermediate void levels each contained four 36-ft (11-m) square pillars with one pillar located at the center of each quadrant of the retort cross section. Thus the center of each pillar was located about 43 ft (13 m) in from each of the closest two side walls. These void level pillars were temporary structures used to support the void level roofs while drilling, explosive loading, and blasting operations were performed.

22.7.3.4 Retorts 7 and 8 Blasting Design and Results

The key to Occidental's modified in situ process is in its retort rubblization technology. The rubblization operation provides a rubble bed that is very large and must be uniformly and well fragmented to give a good oil yield from retorting. The blasting is done in a very confined configuration, which makes uniformity especially difficult. The entire chamber is detonated within a single blasting round, so blast effects must be understood and controlled to avoid extensive damage to the surrounding mine structure. Since the blasting is done to a limited void volume, and access to the rubble bed is difficult, the initial blasting round must perform as planned as secondary corrective type blasting is not possible.

The Retorts 7 and 8 blast design was the result of extensive underground blasting programs (Ricketts, 1982, 1984). The blasting tests and retort blasting patterns were designed using cratering characteristics (Redpath, 1977; Ricketts, 1980, 1987) that were related to conventional blasting patterns used during the development program and large-scale retort blasts. The main vertical blastholes for expanding the upper and lower shale layers and the horizontal pillar blastholes are shown in Fig. 22.7.12. The main vertical interior blastholes were 12 in. (305 mm) in diameter and were located in a 20-ft (6.1-m) spacing pattern, while the perimeter holes were $7\frac{3}{8}$ in. (187 mm) in diameter and were spaced 15 ft (4.6 m) apart. The pillar blastholes were 4 in. (102 mm) in diameter and were located in a 5-ft (1.5-m) spacing pattern. In the figure, the black sections in the blastholes represent the explosive columns while the white sections represent stemming material. All powder columns were center-initiated to give symmetric two-way blasting of the main shale layers and pillars. The pillar and main shale blasting sequences had to be coordinated correctly to ensure the pillars were properly removed and were being distributed within the void levels before the main layers were expanded. Thus the pillars were detonated sufficiently (on the order of 100 ms) before the main shale layers to ensure both proper breakage of the main shale layers and uniform distribution of pillar ejecta across the retort cross section. This delay time was verified full scale in the underground blasting programs and was used in the full-scale Retort 6 blast before it was used for Retorts 7 and 8.

The blasting sequence began at time = 0 when zero-delay seismic electric blasting caps on all three void levels were initiated. These caps initiated nonelectric detonating cord systems located on all three levels. The upper two levels had detonating cord networks both on the floor for the main vertical blastholes and on the pillars for the horizontal pillar blastholes. The lower level only required a detonating cord network for the pillars. The detonating cord system on each void level consisted of two independent trunkline systems passing over each hole. Each system was connected to a different downline leading to a delay cap within the powder column; thus all the powder columns were dual primed. The two trunkline networks on each void level were then cross-linked together at numerous locations for an added measure of redundancy. A nonelectric backup system for the electric system was included that connected the trunklines on all three void levels. The detonating cord systems initiated downline cords in the pillar holes and vertical holes within several milliseconds, the process subsequently propagating to the downhole delay blasting caps. These delay blasting caps then sat safely in the holes waiting to detonate at their prescribed times while the pillars were being blasted within the void levels.

The center regions of the pillars on all three levels were initiated first, followed by the upper and lower portions of the pillars 25 ms later. The pillars were blasted in two steps for Retorts 7 and 8 to reduce blasting damage to the retort roof and floor regions since the charges in the pillars closest to these area would have some relief away from these boundary areas if the center regions were blasted first. A delay time of about 100 ms was then provided before charges in the main shale layers were detonated. The first main layer charges were detonated at multiple locations across the cross section to improve the lateral uniformity of permeability in the rubble and reduce mounding of the top of the rubble pile. Subsequent delays occurred at 25-ms intervals at multiple locations across the retort while the final delay interval was along the edges of the retort. This was done so the edge charges would have additional relief away from the chamber boundaries, thus reducing fracturing into the chamber walls.

The results of the blasts are given in Fig. 22.7.13. Table 22.7.2 also presents a summary of these blasts. The average void fractions in the rubble beds and the remaining void heights above the rubble beds were very close to one another and to predicted values. These quantities were predicted using the limited void blasting curve which was described earlier. The tops of the rubble beds were very flat, as desired. Based on the very large rubble volumes and total retort heights involved in these blasts, a difference of only 0.3 ft (0.1 m) in the remaining void heights above the rubble beds is a remarkable result and an excellent demonstration of large-scale blasting reproducibility. These remaining voids are an important part of the design and serve as distribution chambers for the hot inert gases used during the ignition of the rubble beds. The Retorts 7 and 8 blasting results provide a strong

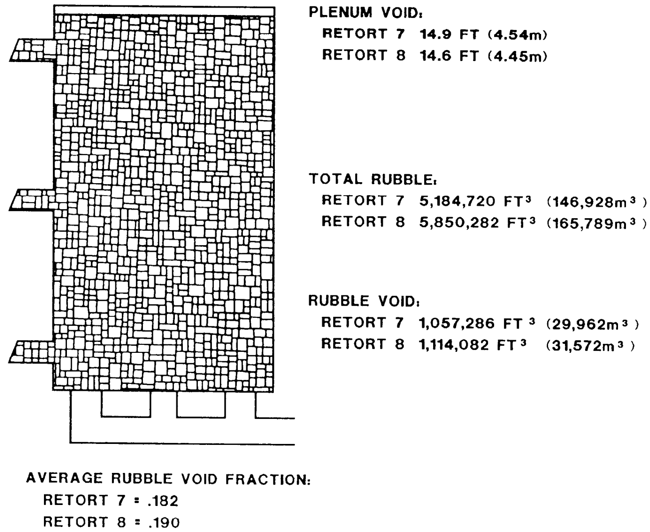


Fig. 22.7.13. Retorts 7 and 8 rubblization results (Ricketts, 1983).

Table 22.7.2. Retorts 7 and 8 Blasting Summary

	Retort 7	Retort 8
Date rubblized	21 Feb 1981	4 Apr 1981
Total number of blastholes	539	554
Main vertical holes	168	168
Pillar horizontal holes	371	386
Explosive weight ANFO equivalent, lb	539,163	545,622
(kg)	(244,564)	(247,494)
Total retort volume, ft^3	6,221,920	6,225,382
(m^3)	(176,185)	(176,283)
Rock blasted, tons	347,655	348,178
(t)	(315,393)	(315,867)
Powder factor, lb/ton	1.55	1.57
(kg/t)	(0.775)	(0.785)

Source: Ricketts, 1982.

endorsement to Occidental's large-scale rubblization technology, which has advanced far beyond normal blasting state of the art and is based on an easy-to-use, practical design tool related to cratering technology (Ricketts, 1987).

The fact that the average rubble void fraction values were fairly close to one another and the tops of the rubble beds were quite flat and level for these two retorts suggested similar permeability distributions existed in these two rubble beds. Flow testing on these retorts discussed in the following sections confirmed the uniformity and similarity of these two beds.

22.7.3.5 Processing of Retorts 7 and 8

A general discussion of modified in situ processing was given earlier. This previously presented material was developed from experience gained with Retorts 7 and 8. This segment presents a more detailed description of the various elements of the processing operation that apply specifically to Retorts 7 and 8.

CONSTRUCTION ACTIVITIES. After blasting, the two retorts were isolated both from the mine and from each other by using steel bulkheads located in the void level entry drifts. Two entry drifts, one running between Retorts 7 and 8 on the lower void level and the other between Retorts 8 and 8X on the upper void level (Fig. 22.7.11), were inaccessible after rubblization; therefore, it was necessary to construct barriers to flow between the retorts prior to rubblization. An effective and low-cost technique was used. It consist of filling the drift with sand, protected at each end by mine-run rock. Because leakage of retort gases was possible through the sand barrier into Retort 8X, and also potentially through the narrow pillar between Retort 8 and Retort 8X, a steel bulkhead was also installed in the entry drift between the Retort 8X upper void room and the upper level mine drift.

Processing facilities were located on the surface over the mine and included blowers for supplying process air, along with steam, to the retorts through pipes from the surface. Other blowers, operating in the suction mode, pulled the process gases through the retorts. Control valves at the product level bulkheads maintained subatmospheric pressure within each retort but neutral at the upper level bulkhead, to prevent leakage of retorting gases into the mine ventilation air.

The off-gas from each retort was separated for a sufficient distance from the product level bulkhead to permit sampling and flow-rate measurements. Then the streams were combined at the base of a cased raise for transport to the surface processing plant. There counter-flow direct-water-contact condensers were used to condense water and C_5+ hydrocarbons from the off-gas. A water-spray nozzle system was installed in the collection drift behind the product level bulkhead of each retort. Late in the operation of the retorts, they were used to cool the high-temperature off-gas to temperatures compatible with the bulkheads, downstream piping and processing equipment.

PROCESS MONITORING INSTRUMENTATION. An extensive network consisting of 1234 thermocouples and gas sample taps was placed in the rubble beds of both retorts after rubblization. Holes were drilled from access drifts through the intact shale surrounding the rubble beds and into the rubble to the desired location with the drill pipes left in place. Thermocouples were assembled around 0.5-in. (13-mm) diameter stainless steel gas-sampling tubes that were inserted into the drill pipes to the desired depths. The annulus between the tubing and the drill pipe was filled with grout to prevent convective heat transfer within the assembly. Provision was made to avoid plugging the open end of the gas-sampling tube with grout. Grout was also extruded into the annulus between the drill pipe and the intact shale surrounding the rubble bed to seal gas leaks between the

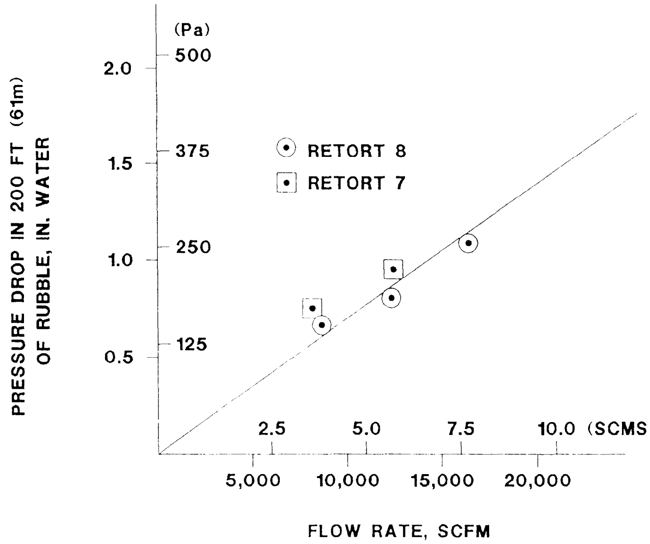


Fig. 22.7.14. Pressure-drop results in the upper 200 ft (60 m) of rubble (Stevens and Zahradnik, 1983).

retort chamber and the mine workings. Drillhole survey results and thermocouple/tubing lengths provided an accurate location of each thermocouple and gas sample tap for subsequent mapping of temperature, pressure, and gas-composition distributions within the rubble beds.

PRE-IGNITION TESTS ON RETORTS 7 AND 8. Pressure drop measurements were made for a series of cold-flow rates through the retorts using pressure taps distributed throughout the rubble beds and at the upper, intermediate, lower, and product-level bulkheads. Fig. 22.7.14 shows the pressure data for both retorts over the upper 200 ft (61.3 m) of rubble, plotted as a function of flow rate for a wide range of flow rates, including 12,000 scfm (5.7 scms), which was the inlet gas flow rate used during processing. The total pressure drop through the 18.5% void rubble is seen to be less than 1 in. water (249 Pa) to within about 40 ft (12.2 m) of the floor of the retorts at the operating flow rate. Initial measurements showed unacceptably high pressure drops within the 20 ft (6.1 m) of rubble above the floor of the retorts. Corrective measures were taken to enlarge the drawports at the bottom of both retorts and reduce the zone of turbulent flow in the region of converging flow lines in the rubble near the drawport exits. The drawports were enlarged until the total cold pressure drop was reduced in Retort 7 to 17 in. water (4.2 kPa) and in Retort 8 to 11 in. water (2.7 kPa) at design flow rates. During retorting, the hot pressure drops, as expected, increased to 75 in. water (18.7 kPa) and 50 in. water (12.4 kPa) for Retorts 7 and 8, respectively.

During retorting, the pressure drop through Retort 8 increased in a predictable manner. The pressure drop in Retort 7 exceeded predictions, indicating that the corrective measures taken in the bottom of Retort 7 were not as thorough as in Retort 8. Although some tracer tests were scheduled and conducted, these analyses were abandoned in favor of mapping the flow pattern using the steam front following ignition (Tyner et al., 1983; Bickel, 1983).

FULL-SCALE IGNITION PROCEDURE. For start-up of Retorts 7 and 8, a hot, shale oil-fired, inert-gas generator was used to inject hot (inert) products of combustion into the void above the bed. The gases were water quenched to 1600°F (870°C). A slight excess of oxygen was used in the generator as a tracer so the

Table 22.7.3. Retorts 7 and 8 Chronology of Processing Events

Event	Retort 7	Retort 8
Rubblization blast	02-21-81	04-04-81
Ignition—First attempt	01-13-82	12-20-81
—Interruption by equipment failure	01-15-82	12-22-81
—Second attempt	01-25-82	12-24-81
—HIGG off, ignition complete	01-29-82	12-28-81
First oil recovered	03-02-82	01-10-82
Steam turned on (design: 20% Steam)	03-13-82	01-22-82
Inlet gas increased to design flow rate (0.54 scfm/ft² or 2.7 mm/s)	03-15-82	03-15-82
Offgas temperature moved above steam plateau	07-15-82	06-15-82
Offgas cooling spray turned on	08-30-82	07-08-82
Injection steam removed	10-02-82	10-01-82
Airflow reduced	10-08-82	10-08-82
Recycle gas injection started	10-21-82	10-15-82
Recycle gas injection terminated and water quenching started	11-19-82	11-09-82
Blowers turned off	01-07-83	01-07-83
Contact condensers turned off	02-28-83	02-28-83

Source: Stevens and Zahradnik, 1983.

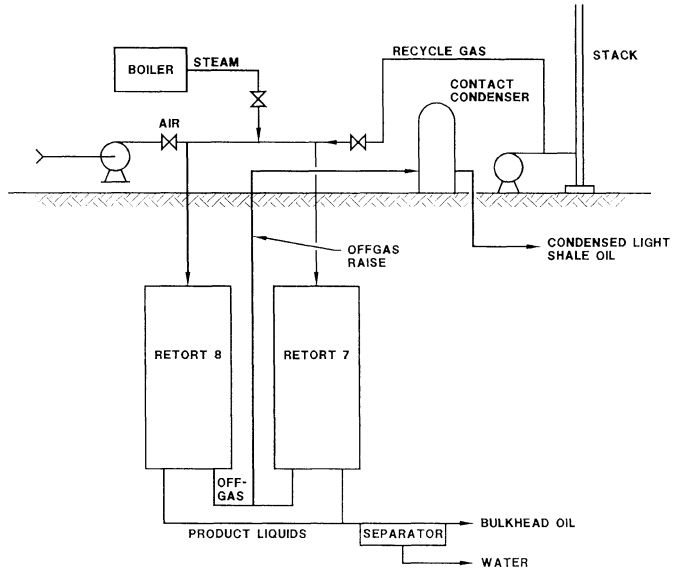


Fig. 22.7.15. Simplified process flow diagram for two retorts (Stevens and Zahradnik, 1983)

onset of combustion at 750°F (400°C) could be monitored by the consumption of the tracer oxygen, that is, the disappearance of oxygen in the retort off-gas. At a generator firing rate of 100 million Btu (106 GJ) per hour, about 18 hours was required for the onset of ignition. The complete transition to air took about 6 hours, at which time the generator was shut down and moved to the next retort.

There were some operational problems experienced during the ignition of both retorts. About 20 hours after starting the hot inert-gas generator on Retort 8 (the first retort ignited), a failure of the blower system was experienced that required 2 days to correct. During this period, the quantity of heat resident in the void above the rubble bed caused the roof to continually slough. Nevertheless, after repair of the blower system, Retort 8 was successfully ignited.

In Retort 7, about 10 ft (3 m) of the roof above the rubble bed fell onto the rubble surface in Sept. 1981, four months before ignition was attempted. Pilot plant tests indicated that start-up procedures would be stretched only a few hours by this occurrence. About 20 hours after starting the generator on Retort 7, leaks caused by corrosion in the flue-gas pipe were discovered. These were located between the generator and the void above the rubble bed and required an ignition shut down for repair. Replacement of this stainless steel pipe required 10 days, during which time extensive additional sloughing of the roof above the bed occurred. As in Retort 8, after repair of the problem, Retort 7 was successfully ignited.

Operation of Retorts 7 and 8. After ignition, the retorts were operated without interruption until gas injection was terminated and water quenching began. A chronology of the significant milestones during processing is given in Table 22.7.3. Fig. 22.7.15 shows a simplified process flow diagram for operating the retorts. Air and steam were fed to the retorts through the inlet pipes from the surface. Sumps behind the product level bulkheads permitted the collection, separation, measurement, and sampling of the bulkhead oil and water pumped from each retort.

Separate off-gas lines from the bulkheads to the common off-gas raise permitted flow rate measurements and sampling of the off-gas from each retort. After transport to the surface, the off-gas was processed through three parallel contact condensers (three different sized units were tested to develop design data for commercial application). Suction blowers withdrew off-gas from the contact condensers and exhausted it to the stack, where the flow rate was once again measured and the gas discharged to the atmosphere by permit.

The off-gas from each retort and the stack gas were analyzed by on-line gas chromatography throughout the operation. Standard EPA isokinetic samples of the off-gas were collected for daily analysis from each retort at the product level bulkhead and from the combined off-gas at the base of the raise, top of the raise, exit of each contact condenser, and the stack. Stack isokinetic measurements were used to determine the daily barrel equivalent of uncondensed C_5+ hydrocarbons exhausted up the stack. Water and light oil recovered at the contact condensers were measured daily and sampled periodically for analysis.

About halfway through the processing life of the retorts, the off-gas temperature began to rise above the steam plateau. This behavior, observed in previous retorts, is essentially a result of nonuniform front advance through the retorts. This phenomenon exists in any retort (even one with an ideal flat reaction front and uniform permeability) with concentrated or localized off-gas drawports, but is accentuated in cases where the reaction front is non-uniform. When the off-gas temperature rises above 250°F (120°C), which represents the safe limit for the product level bulkheads, the retort processing must be terminated. This could result in a significant reduction in sweep efficiency and, therefore, a loss in total retort production. In order to extend the life of the retorts, a plan to cool the off-gas and reduce the thermal load on the bulkheads and downstream processing equipment was developed. The plan consisted of the following steps: (1) utilize a water spray system to cool the off-gas in the drifts behind the bulkheads, (2) reduce or eliminate steam in the inlet gas, (3) reduce the inlet flow rate, and (4) inject recycle gas instead of air. The plan was implemented on both retorts, as shown in Table 22.7.3. The offgas was continuously cooled to about 180°F (82°C). The retorts were thus not shut down because of off-gas over-temperature, and as a result, retorting was extended by about three months and the oil yield increased accord-

Table 22.7.4. Partitioning of Product Oil

	Barrels (ML)	Percent
Bulkhead oil	121,835 (19.4)	60.9
Condensed light oil	18,903 (3.0)	9.5
Uncondensed light oil	54,690 (8.7)	27.3
Oil in water	4,682 (0.7)	2.3
Total	200,110 (31.8)	100.0

Source: Stevens and Zahradnik, 1983.

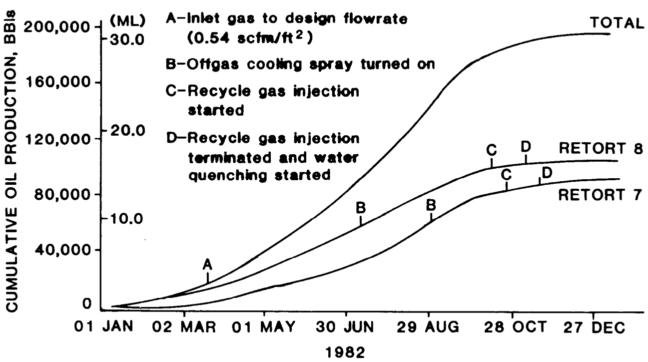


Fig. 22.7.16. Cumulative oil production curves (Stevens and Zahradnik, 1983).

ingly. This was particularly important for Retorts 7 and 8 because the bottom half of the retorts contained the richer "Mahogany Zone" shale.

SHALE OIL PRODUCTION FOR RETORTS 7 AND 8. The combined oil recovery from these retorts totaled 200,110 barrels (31.8 ML), representing over 70% yield of the Fischer assay oil in place in the rubble that was actually blasted. This oil production included oil pumped from the retort sumps, oil mists, and vapors condensed in contact condensers, condensable C_5+ vapors found in the stack gas, and a small amount of oil in the bulkhead water. The partitioning of the total product between these four streams is shown in Table 22.7.4. The properties of condensed light shale oil, collected from the off-gas stream at the contact condensers, were very similar to those of military specifications for No. 2 diesel fuel. During the operation of the retorts, this product was tested and performed satisfactorily as fuel, without upgrading, for diesel-powered mine equipment.

Cumulative production curves for each retort, as well as the total, are shown in Fig. 22.7.16. The bulkhead oil portion of the production was directly measurable for each retort. The production rates measured after combining streams from both retorts (i.e., light shale oil from the contact condensers, uncondensed C_5+ lost to the stack, and oil in the process waters) were partitioned between Retorts 7 and 8 in accordance with the isokinetic sampling results on the off-gas at the exits from the product level bulkheads. These partitioned values were added to the respective bulkhead oil production values to arrive at the curves for Retorts 7 and 8.

Table 22.7.5. Retorts 7 and 8 Retorting and Results

	Retort 7	Retort 8	Total
Superficial gas velocity, fpm	0.54	0.54	—
(mm/s)	(2.7)	(2.7)	
Air/steam ratio, avg	80/20	80/20	—
Fischer assay of rubble blasted, gal/ton	17.3	16.9	17.1
(mL/kg)	(72.1)	(70.4)	(71.3)
Oil-in-place in rubble blasted, bbl	143,200	140,100	283,300
(ML)	(22.7)	(22.3)	(45.0)
Produced oil, bbl	93,101	107,009	200,110
(ML)	(14.8)	(17.0)	(31.8)
Yield, % Fischer assay	65.0	76.4	70.6

Source: Stevens and Zahradnik, 1983.

It is clear that these two retorts performed differently. If the curves are moved along the time axis to account for the nominal one-month difference in ignition time, it becomes evident that the production of Retort 7 deviated significantly from that of Retort 8 after about Oct. 1, 1982, that is, late in the operation of Retort 7. This is attributed to the fact that the efforts to correct the high pressure drop in Retort 7 prior to ignition were not completely successful. This could have caused distortion of the flow lines near the bottom of the retort resulting in reduced sweep efficiency.

It is believed that the stated average yield of 70% for these two retorts is on the conservative side since the results in Table 22.7.5 indicate a higher yield for Retort 8. The definition of yield used here has been used previously by Occidental and others for evaluating modified in situ retorting performance. The definition is based only on the rubble that was actually blasted in the retorts and does not account for oil residing in roof-sloughing rock and the retort walls that enters into the process. In other reports (Nelson, 1981; Bickel, 1983) where material balances are part of the analyses, contributions for the walls and roof shale are incorporated, resulting in somewhat lower yield values.

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Chapter 22.8
MARINE MINING

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This chapter covers briefly the technical aspects of mining in the continental shelf and in the deep ocean. Marine mining, as an industry, is influenced by a heterogeneous mix of international entrepreneurs, advanced engineering companies, politicians, environmentalists, legal experts, long-time dredge operators, and a few old-line mining companies. There is no question that the "industry," such as it is, is here to stay, and that many aspects of the work done to date have been serious and well founded. It is still too early to generalize about marine mining technology, and discussing each activity on a case-by-case basis appears to be the most reasonable approach at this time. Some order has been established, however mineral deposit types have been classified, mining systems have been categorized, and legal regimes have been established. Nevertheless, there is still much to learn, because despite the interest engendered in the last three decades, there are few operating mines offshore, and the likelihood of new ones may be as dependent on the foibles of the politicians as on the market place.

22.8.1 MARINE MINING ENVIRONMENT

The marine environment distinguishes marine mining from conventional mining. Not only are the ore deposits associated with a very dynamic and unpredictable overburden of ocean waters, but the nature of the deposits themselves may be largely a result of interactions between the ocean and geologic systems. Furthermore, the requirement to work in the oceans—about which we still know very little compared to the terrestrial environment—not only makes extraordinary demands on mining engineers but causes intense concern among those who feel that mining may threaten their livelihood or the environment in some way. Thus it is appropriate to present some of the considerations that must be made with regard to the environment (Table 22.8.1), for to ignore these aspects prior to initiation of any marine mining venture is to invite frustration, failure, and despair.

22.8.1.1 Natural Systems

These systems, which include the air, the water, the solid earth, and all living things, are controlled by nature, and activities directed at the extraction of minerals from the marine environment must be designed around them.

Weather is a significant factor in working in the oceans; bad weather can effectively stop marine operations for days or months on end. Its variability and low predictability require that systems be designed for "worst case" scenarios. The potential for weather-induced problems must be factored into technical and economic analyses, and adequate weather and sea-state information must be made available at all times. Climatic conditions at sea, like any other mining environment, range from tropical to arctic. The influence of weather conditions is different if the operation is located in the open ocean or close to shore. Storms may be less of a hazard in the open ocean; during the monsoons in southeast Asia, most nearshore mining operations are closed down. Air quality may be a significant factor in analysis of environmental impact, particularly in operations close to shore where terrestrially derived limits may be applicable.

Hydrologic systems involve not only the open ocean, which covers 70% of the earth's surface area, but also the waters of coastal areas, which may be fresh, brackish, or salt, and may present a wide variety of significant physical conditions. Nearshore conditions may also be influenced by coastal morphology, resulting in such phenomena as tidal bores, longshore currents, breaking waves, and current gyres that could affect operations. Beach and nearshore operations are particularly susceptible to the effects of storms and wave conditions, and tidal activity can present significant variations in depth during each working day as well as increasing the potential for wave and storm damage during the equinox. The presence of fresh water aquifers extending onto the continental shelf, although not themselves presenting any hazard, may constrain a mining operation if there is danger of contaminating the aquifer by the intrusion of saltwater. In the open sea, surface conditions are generally well known and understood in terms of waves and currents; but in deep water, variations occur in the nature, direction, and intensity of water movement throughout the water column. For example, in the Pacific, the Cromwell Current, just north of the equator, flows west at a depth of about 1000 ft (300 m) beneath the easterly flowing surface equatorial current with a net velocity difference of about 6 knots. Other such phenomena exist throughout the ocean's depths and must be recognized and considered in the design of any mining operation. On the bottom, currents with speeds of 1 to 2 knots are not uncommon, even in the deep ocean, and boundary waves are frequently encountered at the interface between stratified water masses of different salinity and temperature. These differences in water properties also affect acoustic and electrical properties utilized in geophysical sensing.

The geologic systems of the world provide a continuum between the seabeds of the deep ocean trenches and the tops of the mountain ranges on the five continents. The processes by which the varying parts are formed and placed in their present juxtaposition is at least partially explained by the modern theories of plate tectonics, confirmed only recently by advances in the technology and understanding of marine geology and geophysics. Traditional theories of local and regional geologic processes have not been upset, however, and are still valid in explaining coastal and submarine morphology, and the formation of mineral deposits on the continental shelf and in the deep oceans (Cruickshank, 1990; also see Chapter 4.2).

Biological systems in the marine environment are extremely diverse and not as well understood as those on land. The major considerations for mining are whether any negative effects of the proposed operations on living creatures will be significant. At the present time it appears that, compared with similar operations on land, marine mining is relatively benign. However, because of the scarcity of data, not only on the ecological systems in question but also on actual mining operations at sea, there is little substantiating evidence to legally support such claims. Extensive research on biological systems has been carried out on the US Continental Shelf under the Environmental Studies Program of the Minerals Management Service, and five environmental impact statements (EISs) addressing the possibility of leasing have been prepared (see 22.8.3.4, for example). Similarly, for the deep

Table 22.8.1. Environmental Systems Elements Affecting the Exploration and Exploitation of Marine Mineral Deposits (examples in parentheses)

Natural Systems

- Meteorologic
  - Climate
    - Temperature
    - Winds
    - Precipitation
  - Air Quality
    - Gases
    - Water vapor
    - Particulates
    - Radiation (Acoustic, seismic, electric, magnetic, nuclear)
  - Natural Hazards
    - (Storms, ice)
- Hydrologic
  - Coastal
    - Surface water
      - Drainage (Rivers, channels)
    - Groundwater
      - Drainage
        - Flow rates (Aquifers, springs)
    - Retention
      - Capacities
      - Leakage
      - Overflow (Underground lakes)
    - Recharge
    - Water quality
    - Physical properties
    - Chemical properties
  - Natural Hazards (Tsunamis, storms, floods)
- Marine
  - Upper boundary layer
    - Sea state (Climate)
      - Waves
      - Swell
      - Current
      - Tides
      - Temperature
    - Boundary gradients (Water quality)
      - Physical properties (Temperature, density, STD)
      - Transmissivity (Optical, acoustic, electric)
      - Chemical properties
        - Dissolved constituents (Salinity, chlorinity, trace elements)
      - Particulates
  - Intermediate Layer
    - Climate
      - Temperature
      - Circulation pattern
      - Mixing pattern
    - Water quality
      - Physical
      - Chemical
  - Bottom Layer
    - Climate
    - Temperature
    - Circulation

Marine (cont.)

- Boundary Gradients (Water quality)
  - Physical
  - Chemical
- Natural Hazards
  - (Storms, turbidity currents, geothermal plumes, submarine waves)
- Geologic
  - Regional geology
    - Morphology
    - Tectonics
    - Petrology
    - Mineralogy
  - Mineral deposits
    - Ore minerals
    - Distribution
    - Origin
    - Ore reserves
  - Geotechnique and geomechanics
    - Index properties (Specific gravity, density, grain size, water content, Atterberg limits)
    - Engineering properties (Compression, shear, permeability, consolidation)
  - Natural Hazards (Earthquakes, slides, plate motion)
- Biologic
  - Diversity and Abundance
    - Floral
      - Terrestrial (Trees, shrubs, grasses, crops)
      - Marine (Algae, seaweed, phytoplankton)
    - Faunal
      - Airborne (Birds, flying insects)
      - Terrestrial (Animals, reptiles, insects)
      - Marine
        - Pelagic (Zooplankton, fish, mammals, reptiles)
        - Benthic (Epifauna, infauna)
  - Ecosystems
    - Zonation
    - Influences (Physical, chemical, trophic)
  - Natural Hazards (Plankton blooms, fish attacks)
- Physiologic
  - Operational Personnel
    - Numbers
    - Ethnics
      - Aptitude

Artificial Systems

- Social
  - Regional Population
    - Distribution
    - Quality of life (Health, education, welfare, income, livelihood)

Social (cont.)

- Social amenities (Housing, schools, fire protection, police, services)
- Transient Population
  - Number of transients
  - Purpose (Vacation, seasonal workers, fishing)
- Economic
  - Resource Value (Minerals, fisheries, real estate)
  - Resource Markets
  - Conflicting uses
    - Airspace
      - Aesthetics
      - Commercial (Transport, communications, research)
    - Water
      - Residential
      - Commercial (Fisheries, transport, farming, ports)
      - Industrial (Cooling, dumping)
      - Recreational
      - Educational
      - Defense
    - Land (Seabed)
      - Residential
      - Agricultural
      - Industrial
      - Recreational
      - Educational
      - Defense
- Legal
  - Ownership (Territorial waters, outer continental shelf, deep sea)
  - Jurisdictional Agencies
  - Restraints (Rights of way, precedential rights)
- Political
  - Existing Regime
    - Policies
    - Representation
    - Relevant functions
- Historical and Archaeological
  - Distribution of Sites and Objects (Ruins, artifacts, shipwrecks)
  - Importance
  - Description
- Technical
  - Materials R&D
  - Design
- Restoration
  - Sealing
  - Removal
  - Restoration
- Accident (Fire, explosion, spillage, inundation, collapse)

Table 22.8.2. Classification of Marine Mineral Resources

Unconsolidated		*Consolidated*		*Fluid*	
Seabed	*Subseabed*	*Seabed*	*Subseabed*	*Seabed*	*Subseabed*
Conshelf	**Conshelf**	**Conshelf**	**Conshelf**	**Conshelf**	**Conshelf**
Industrial Materials sand, gravel shell sands aragonite coral sands *Mineral Sands* magnetite ilmenite rutile chromite monazite	*Mineral Sands* gold platinum cassiterite gem stones *Bedded Deposits* phosphorites	*Outcrops* exposures of veins, etc.	*Vein, Stratified, Disseminated or Massive Deposits* coal phosphates carbonates potash ironstone limestone metal sulfides metal salts	*Seawater* magnesium sodium uranium bromide and salts of 26 other elements	*Freshwater Springs*
Ocean Basins		**Ocean Basins**	**Ocean Basins**	**Ocean Basins**	**Ocean Basins**
Muds or Oozes metalliferous carbonaceous siliceous calcareous baritic *Nodules* manganese cobalt nickel copper		*Crusts* phosphorite cobalt manganese *Mounds and Stacks* metal sulfides	*Vein, Stockwork, Stratabound or Massive Deposits* metal sulfides	*Seawater* magnesium sodium uranium bromine and salts of 26 other elements	*Hydrothermal Fluids*

seabed, EISs and other studies conducted during their preparation are available (Anon., 1981a).

22.8.1.2 Marine Mineral Resources

Known mineral resources in the marine environment are as varied as those on land. In quantitative terms, the lack of three-dimensional data leaves many of the published numbers for marine minerals in the realm of speculation. Nevertheless, it appears that with our rapidly improving knowledge and understanding of earth system dynamics, reasonable extrapolations, at least in orders of magnitude, can be made (McKelvey, 1986; Cruickshank, 1978; Mero, 1965). Recoverable marine mineral resources may well exceed those on land, and may constitute the ultimate source of mineral materials for economically and ecologically sustainable development for survival in the future.

Marine mineral deposits may be classified according to their physical state as in Table 22.8.2, which lists the resources as unconsolidated, consolidated, or fluid, because these properties most influence the technology used in their extraction. More particularly, the unconsolidated deposits are amenable to dredging while the consolidated deposits require the application of some additional energy source for their removal. In the same context, they are also subdivided by being “on” the seabed or “beneath” the seabed. A further dichotomy, based on both technical and legal aspects, describes the mineral targets as being on the continental shelf or in the deep ocean basins. Some minerals, such as phosphorites, are found in both environments. Table 22.8.3 describes some potential resources of minerals that have been indicated within seabed areas under US jurisdiction, and Fig. 22.8.1 shows the extensive distribution of seabed minerals throughout the world’s oceans and exclusive economic zones (EEZs).

For principles of mineral prospecting and exploration as well as project geology applicable to placer deposits, see Sections 4 and 5 and Chapter 15.1, of this *Handbook.*

TARGETS IN THE CONTINENTAL SHELF. The continental shelves are basically submerged continental lands, and contain the complete suite of mineral resources that are found onshore as well as deposits resulting from the overlying marine environment. Mineral deposits that have been worked on the shelf include glacial and alluvial sands and gravels, biogenic materials such as shell and coral sands, detrital mineral sands, phosphorites, indurated sedimentary beds laid down in shallow seas, and other hard rock deposits such as veins and massive deposits typical of a continental environment.

Marine *sands and gravels* are for the most part derived from erosion of the continents followed by transportation of the erosion products to the sea by glaciers and rivers where they are sorted by the action of the waves and currents; other sands are formed by marine processes such as the breakdown of shells or coral beds, or the formation of oolitic deposits of glauconite or phosphorite in shallow waters. All these materials are prolific throughout the marine environment, and shelf deposits of sand and gravel are the bases for major industries in Japan and the European North Sea. Marine *phosphorites* vary in character depending on their genesis and are found as sands, crusts, and nodules in shallow basins, on the slopes of islands, and in tropical lagoons. Very extensive deposits of bedded marine phosphorites of Miocene Age are indicated offshore of the eastern United States and other areas (Table 22.8.4). *Mineral sands* containing gold, diamonds, or minerals of tin (cassiterite), titanium (rutile, leucoxene, ilemenite), and other metals, derived from the breakdown of igneous rocks, are distributed widely throughout the coastal areas of the world, and many of them are recoverable at

Table 22.8.3. Minerals Potential Within the U.S. Exclusive Economic Zone

Potential Seabed Resources of Strategic Minerals Within the U.S. 200-Mile Exclusive Economic Zone

Area	Commodity
1. Offshore Maine	Transition metals, chiefly lead and zinc, in shallow lodes
2. New York Bight and coastal New Jersey	Titanium, chiefly as rutile and ilmenite, possible rare earths, in placers
3. Southeast Atlantic coast	Zirconium, some titanium, rare earths in coastal and buried strand deposits; rock phosphorite below seafloor
4. Blake Plateau	Low-grade, transition metals in ferromanganese nodules; nodules for catalytic and scrubber uses
5. Offshore Texas	Titanium, zirconium, rare earths in tag placers off Corpus Christi; sulfide metals in salt dome cap rock lodes
6. Gulf of Mexico, continental slope	Probable disseminated metal sulfides in intraslope basins
7. Offshore California	Fine gold placers nearshore, particularly Crescent City area; possible platinum off northern California; phosphorite on offshore banks
8. Offshore Oregon-Washington	Chromium and platinum in shallow marine placers; transition metal sulfides in Juan de Fuca-Gorda rift (major exploration in progress, 1983)
9. Alaska coastal and shelf waters	Barite in shallow lodes in inlets of southeastern Alaska; platinum and gold in marine sands, southeastern Alaska; lode copper beneath Prince William Sound; platinum, gold, chromium in coastal placers between Cape Newenham and Kuskokwim River; gold, tungsten, rare earths, and tin in coastal placers, Yukon Delta to Bering Strait; coastal gold placers in Chukchi Sea north of Lapp Lagoon
10. Hawaii, offshore area	Ferromanganese crusts with manganese, possibly transition metals
11. Commonwealth of Puerto Rico, offshore	Disseminated metal sulfides in slope deposits on south side; minor gold placer on north shore
12. American Samoa	Offshore mineral potential unknown, but regional geologic setting suggests possible phosphorite and sulfide metals
13. Isolated Pacific islands, including Line Islands	Cobalt-rich manganese crusts on island flanks; possible transition metals, at depth, in manganese nodules
14. Commonwealth of Northern Mariana Islands and Guam	Sulfide metals off western side; possible sulfur boil deposits of transition metals off Tinnian and Saipan; local phosphorites; volcanic-related minerals in north
15. US waters of the Great Lakes	Lode copper deposits in Lake Superior; ferromanganese pellets in Green Bay, Lake Michigan; other areas unexplored for mineral resources

Source: J.R. Moore in Mangone, 1984.

the present time (Telecki et al., 1985). In many cases, the deposits have been concentrated by high-energy coastal processes and reworked, buried, or exposed during passage of the littoral zone between high and low stands of sea level. Although many nearshore deposits are known, the outer shelves are virtually unexplored, and indications of high-energy activity near the shelf edge during earlier periods of low sea level suggest a high potential for deposit formation in these areas. Improvement in the characterization of still-stands of sea level would assist in the identification of these resources.

TARGETS IN THE OCEAN BASINS. The structure of the ocean basins beyond the continental shelves consists mostly of basaltic seabeds interspersed with oceanic ridges, island arcs, continental remnants, and volcanic islands or seamounts. Overlying sediments are normally sparse in the areas of high relief and thicker in older and deeper areas. They derive mostly from erosion of adjacent continents and islands or from the skeletons of marine creatures, and in deep water may form fine granular deposits of foraminiferal (calcareous) or diatomaceous (siliceous) oozes. Other muds may be formed from the precipitation of minerals dissolved in hydrothermal fluids in the vicinity of diverging oceanic plate boundaries. Mineral deposits such as manganese nodules, cobalt crusts, and metalliferous sulfides are recent discoveries resulting from the study of deep ocean basins and seabed dynamics. Phosphorites in the tropical oceans and other mineral types yet to be discovered may play a significant part in the inventory of marine resources.

The first discovery of *manganese nodules* was made by the expedition of HMS Challenger in 1873. The nodules are potato-like encrustations of iron and manganese oxides, formed around a small nucleus, at depths usually between 17,000 and 20,000 ft (5000 and 6000 m). The minerals carry enrichments of cobalt, nickel, copper, and other metals scavenged from the seawater and represent a significant resource of some of the metals. Distribution of the nodules is widespread throughout all the oceans of the world (Anon., 1981c), but the areas of highest grade known at this time are in the Pacific Ocean between the Clarion and Clipperton fracture zones about 1200 mi (1930 km) west and south of Hawaii (Fig. 22.8.2). Of similar composition but occurring on the submerged portions of islands or seamounts between 2700 and 8000 ft (800 and 2400 m) are encrustations of metalliferous oxides known as *cobalt crusts.* These carry higher values of cobalt and platinum, up to 1.4% and 0.93 ppm, respectively, in certain cases, and they are also associated with phosphorus and rare earth elements. The crust deposits are widely distributed in the Pacific (Fig. 22.8.3) and with thicknesses ranging to 16 in. (400 mm), they may also represent a significant resource of the contained metals (Manheim and Lane-Bostwick, 1989, Table 22.8.5).

More recently discovered and of equal interest are the *metalliferous sulfides* that are formed in the vicinity of active plate boundaries throughout the world's oceans (Fig. 22.8.4). These deposits potentially occur in many different forms from surficial muds to deep seated masses at various depths beneath the seabeds (Table 22.8.6). They are generally formed by the interaction of superheated seawater with fractured crustal rocks, in convection cells caused by the intrusion of new or subducted plate materials, and they may be host to almost any of the metals, including copper, lead, zinc, silver, and gold, each of which has been reported in significant values from sampled outcrops on the seabed (McMurray, 1990; Scott, 1987). Although some of the deposit types postulated have not yet been located, their discovery is highly likely. Mineral deposits preserved within the crust or overlying sediments of moving plates may be carried to other locations within the ocean basins, and many examples are now found on land as the result of upthrusting of colliding plates (Sawkins, 1990).

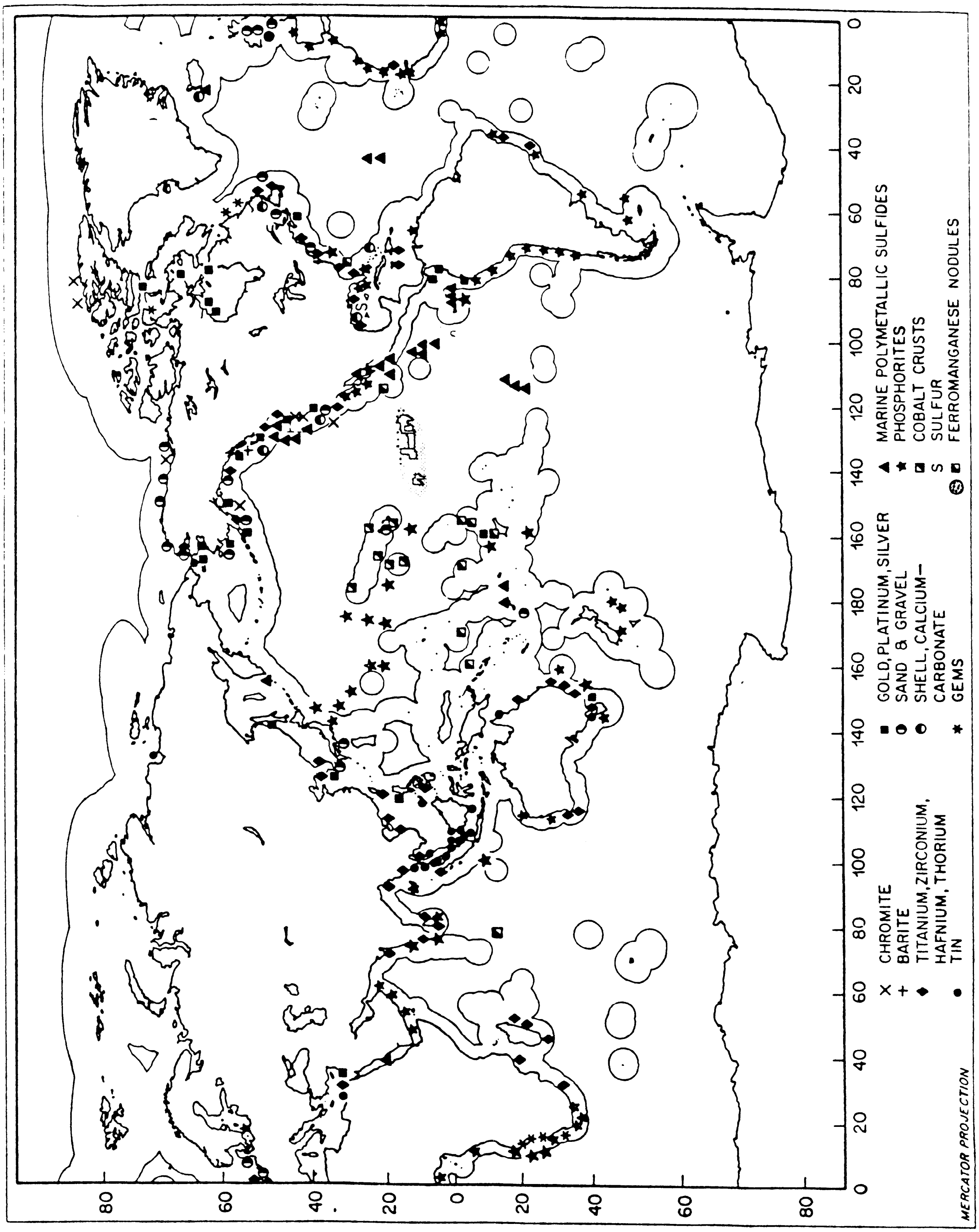


Fig. 22.8.1. World distribution of seabed mineral deposits (after Hoagland and Broadus, 1987).

Table 22.8.4. Distribution of Phosphorites in Miocene Sediments Along Continental Margins

Continental Margin	Region
East Atlantic	Portugal, Northwest Africa through South Africa, and Agulhas Bank
West Atlantic	North Carolina through Florida, Cuba, Venezuela, and Argentina
East Pacific	California through Baja California, Mexico, and Peru through Chile
West Pacific	Sakalin Island, Sea of Japan, Indonesia, Chatham Rise east of New Zealand, and East Australian shelf.

Source: (Teleki et al., 1985).

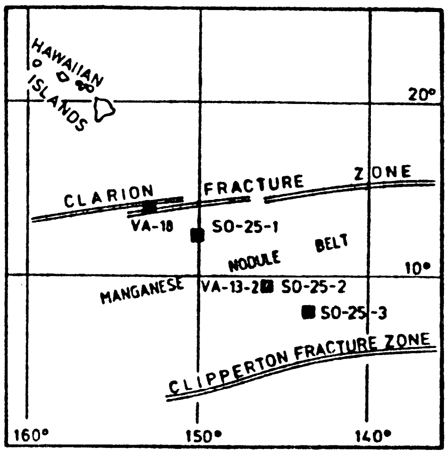


Fig. 22.8.2. Belt of high-grade manganese nodules between Clarion and Clipperton fracture zones in the Pacific Ocean (Teleki et al., 1985).

Ocean waters may represent the most significant source of minerals for sustainable recovery on the planet and include nearly all the known elements in some degree of solution. Fresh water springs are found offshore of many areas on the continental shelf, and high-temperature hydrothermal vents found at spreading centers around 10,000 ft (3000 m) of water depths are potential sources of enriched solutions of metals as well as heat energy. Current production of dissolved materials from seawater is limited to magnesium, magnesium compounds, salt, bromine, and heavy water (deuterium oxide), but the potential for extraction of materials from naturally enriched seawater sources may be high, and the resource is extremely large.

22.8.1.3 Artificial Systems

Unlike the natural systems, which are controlled by nature, artificial systems are artifacts of human society and have evolved in various ways to support human needs. Although theoretically changeable, the process of change may be very cumbersome. Some of the factors that have a strong impact on marine mining development are discussed below.

SOCIO-ECONOMIC FACTORS. Socio-economic factors are generally those supposedly involving the public good and may have very different connotations in a market economy as opposed to a centrally planned economy where the benefits and costs of any action may be evaluated under different rules (Chapter 3.1). Even in the area of environmental concerns, priorities may be set according to different baselines, particularly where poverty is an overriding factor in the economy. With certain bulk commodities such as sand and gravel, the market price is usually set locally based on specific needs. Metal markets, which drive the projected economics of most placer deposits and deep seabed minerals, are world markets that are not generally affected by the output of individual mines (Chapter 2.3). Major exceptions arise, however, in the examination of market effects of manganese nodules and crusts (Anon., 1986).

Without operating data, the benefits and costs of marine mining are not easily computed, as many interacting factors may affect the outcome and many of these cannot be easily quantified. Such analyses are extremely useful, however, and have been well described in the literature (Cruickshank, 1978). In most countries of the world, environmental impact analyses are required before commencing to develop mining operations offshore (Chapter 3.4). In the United States and Europe particularly, strict laws and guidelines prevail. Regulations for the United States under the National Environmental Policy Act (NEPA) are found in *Code of Federal Regulations,* CFR, Title 40, 1500-1508. Examples of impact statements for specific actions are discussed in 22.8.3.4.

LEGAL-POLITICAL FACTORS. Legal and political boundaries may be quite different from geologic and geographical boundaries. The outer continental shelf has several meanings in the United States, depending on whether it is referred to the AGI Glossary of Geology, the 1958 Convention on the Continental Shelf, the United Nations Law of the Sea Convention of 1980, or the 1983 United States Proclamation on the *Exclusive Economic Zone* (EEZ). The most significant legal boundaries of the oceans date from the 1980 Convention and include the 200-nautical mi EEZ, and The Area, (or International Seabeds), encompassing all seabeds beyond the jurisdiction of any state (Holser, 1988). Mining of the resources in The Area will come under the jurisdiction of the International Seabed Authority, headquartered in Jamaica. Examples of new EEZs and mineral occurrences in other countries in the Pacific are given in Cruickshank and Kincaid (1990).

In the United States, offshore mining operations within the EEZ are carried out under the authority of the Outer Continental Shelf Lands Act of 1953 (amended 1978) and accompanying regulations 30 CFR 280 for prospecting, 30 CFR 281 for leasing, and 30 CFR 282 for operating. Activities directed to the development of manganese nodules in The Area are authorized by the Deep Seabed Hard Mineral Resources Act of 1980, and accompanying regulations 15 CFR 971 for exploration, and 15 CFR 270 for commercial recovery. The Acts are administered by the Department of Interior's Minerals Management Service (MMS) and the National Oceanic and Atmospheric Administration (NOAA), respectively. Additional laws and regulations are followed with respect to potential environmental impacts, and these are discussed in the final programmatic environmental impact statements for gold mining off Alaska (Anon., 1990a) and for deep seabed mining (Anon., 1981a).

Typically, the laws of foreign countries relating to marine minerals are much simpler than those of the United States and generally are of the prospecting permit variety. In some states, there are no mining laws, and all concessions are granted on the basis of negotiations with the government. Any extensive research into the mining laws of another country should include contact with the appropriate desk of the country involved in the US Departments of State and Commerce. Consulates and embassies do not usually prove to be a fruitful source of information, particularly for developing nations (Krueger, 1969).

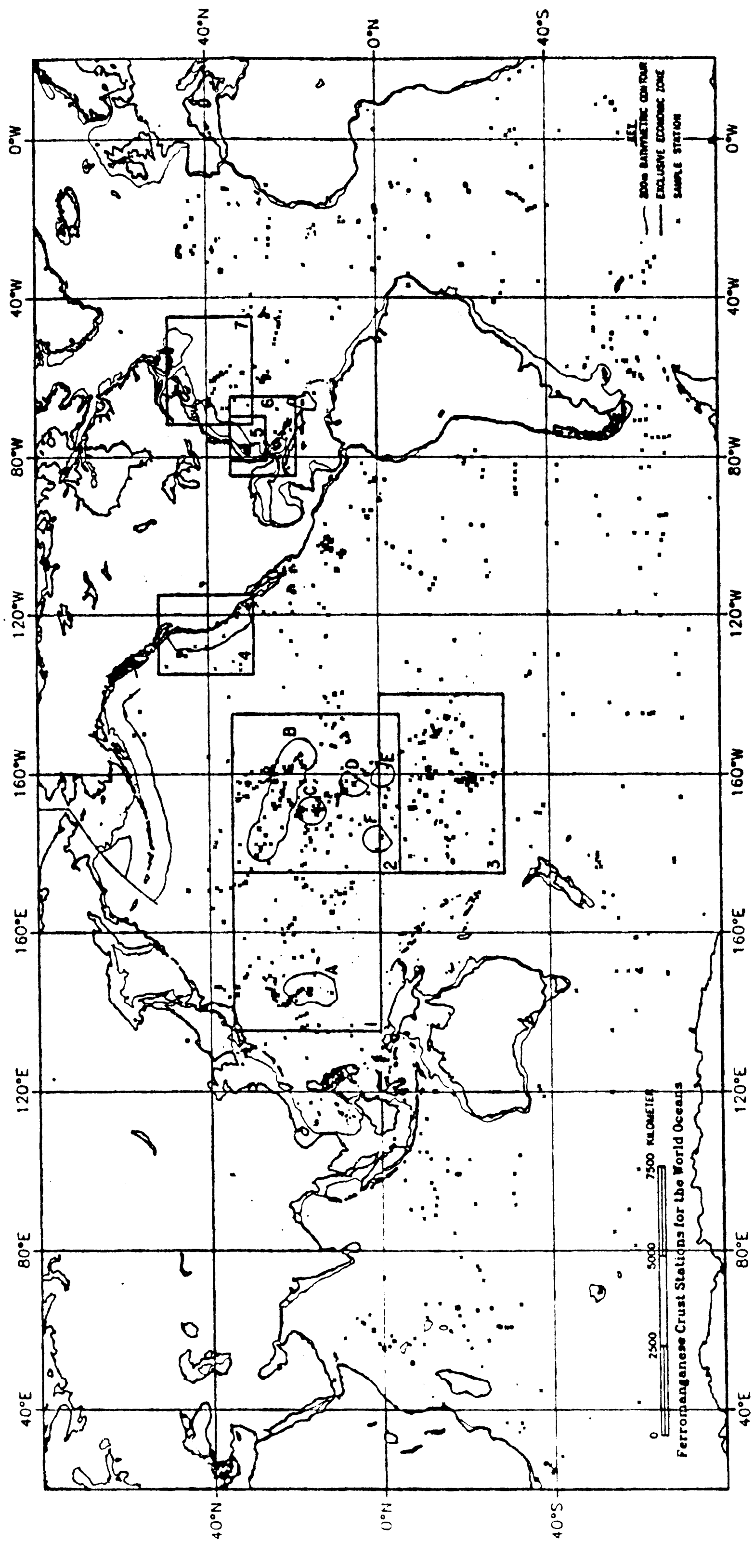


Fig. 22.8.3. Known cobalt crust distribution in the world's oceans (Manheim and Lane-Bostwick, 1989).

Table 22.8.5. Examples of Metal Grades and Values for Crusts and Nodules

Component	Price ($/kg)	MPM crusts % dry	% wet	$/t wet	$/m²	Line Is. crusts % dry	% wet	$/t wet	$/m²	MIDPAC % dry	% wet	$/t wet	$/m²	Nodule (NB) % dry	% wet	$/t wet	$/m²
Co	27.56	0.83	0.54	148.82	2.98	1.00	0.65	179.14	3.58	0.79	0.51	140.56	2.81	0.24	0.16	44.10	0.44
Ni	4.98	0.49	0.32	15.94	0.32	0.55	0.36	17.93	0.36	0.49	0.32	15.94	0.32	1.21	0.79	39.34	0.39
Cu	1.77	0.07	0.05	0.89	0.02	0.07	0.05	0.89	0.02	0.07	0.04	0.71	0.01	1.00	0.65	11.51	0.12
Mo	10.58	0.06	0.04	4.23	0.08	0.06	0.04	4.23	0.08	0.06	0.04	4.23	0.08	0.04	0.03	3.17	0.03
**	1.52	25.0	16.25	247.00	4.94	25.0	16.25	247.00	4.94	24.6	15.99	243.05	4.86	25.2	16.38	248.98	2.49
***	0.58	25.0	16.25	94.25	1.89	25.0	16.25	94.25	1.89	24.6	15.99	92.74	1.85	25.2	16.38	95.00	0.95
Total																	
**				416.88	8.34			449.19	8.98			404.49	8.08			347.10	3.47
***				264.13	5.29			296.44	5.93			254.18	5.07			193.12	1.93

* MPM refers to crusts from the Mid-Pacific Mountains and Line Islands Ridge, at water depths of 2,600 m or less. MIDPAC refers to samples from all depths encompassed in the MIDPAC 81 cruise (n = 61; Halbach et al., 1982). NB refers to abyssal manganese nodules from the nodule belt area in the NE Pacific. Approximate ore abundance is 20 kg/m² for the crust deposits and 10 kg/m² for the abyssal nodules. Molybdenum metal values for the concentrations are taken from previous studies (Craig et al., 1982). Metal prices are for May 1983 (*Source:* Erzmetall 36 (1983) p. 397). Decimals are for computation only.
Recent trace element determinations of ferromanganese crust samples from MIDPAC 81 have shown that platinum may reach concentrations of up to 1.0 g/t (average content 0.5 g/t; range 0.3–1.0 g/t). This might be of additional commercial interest since presently mined platinum-bearing ores have contents of 1–2 g/t of platinum.
** Manganese (99.95%).
***Ferromanganese (78% Mn) recalculated to 100% Mn basis.
Conversion factors: 1 lb = 0.4536 kg, 1 ft = 0.3048 m, 1 ft² = 0.0929 m².
Source: After Teleki et al., 1985.

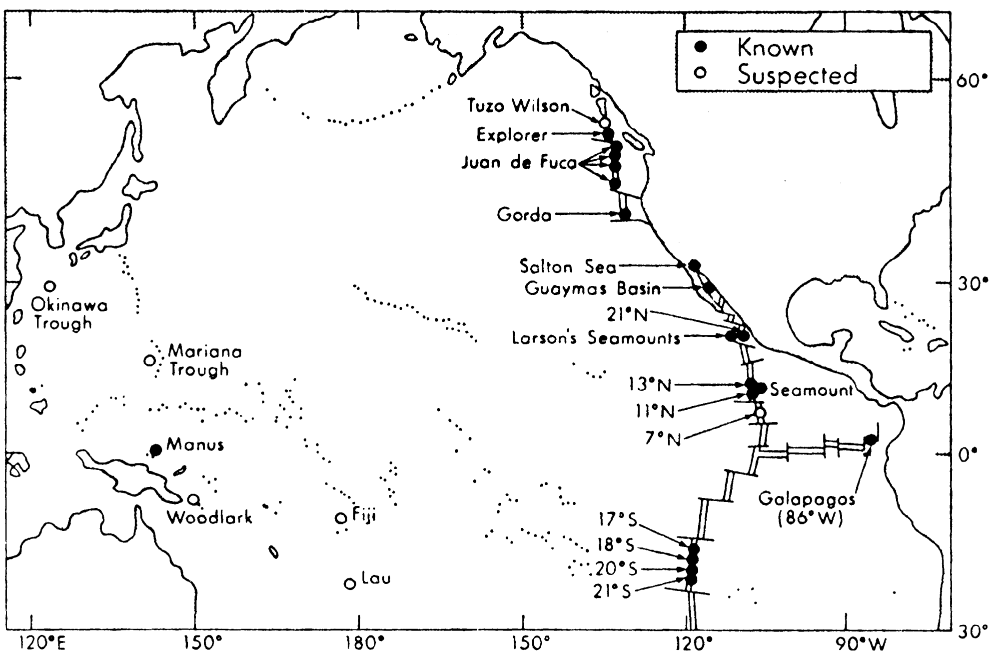


Fig. 22.8.4. Distribution of known and suspected metalliferous sulfide deposits and active hydrothermal vents in the Pacific Ocean (Scott, 1987).

22.8.1.4 Environmental Hazards and Constraints

In the design of systems for marine mining, the possibilities of encountering hazards or other environmental constraints must be assessed early in the design process.

Natural hazards to operating in the marine environment appear in a variety of forms including high winds such as hurricanes or typhoons; freezing rain or snow; heavy swells, breaking waves, tsunamis, waterspouts, submarine countercurrents, and storm seas; slumping, turbidity currents, volcanic activity, and hydrothermal mega-events; fish attacks; and natural seawater toxicity.

Mitigation of potential environmental impacts can be a major concern in the design process and appropriate measures must be designed for and implemented. These matters will be considered in detail in the environmental impact analysis prepared before the commencement of operations. The potential impacts of a deepwater dredging system are illustrated in Fig. 22.8.5, and

Table 22.8.6. Metalliferous Sulfide Deposits Potentially Associated with Moving Plate Boundaries

Nature of resource model	Depth of occurrence	Examples: Seabed	Land	Grade upper 10%	Tonnage (median)	No. of deposits examined[a]
Geothermal energy	0–1.5 km	Active zones	Salton Sea			
Hot-vent fluids	Surficial	Active zones	Salton Sea			
Metalliferous muds	0–30 m	Red Sea	?	Cu 0.45, Zn 2.07, Au 0.5, Ag 30	664×10^6	1
Volcanogenic sediments (Mn)	Surficial	DSDP hole	Numerous	Mn 49, P 0.03	47×10^3	93
Surficial precipitates						
Individual stacks or mounds	Surficial	Active zones	?			
Coalesced stacks or mounds or cemented tallus	Surficial	Active zones	?			
Subseabed precipitates						
Cyprus massive sulfides	0–0.8 km	Postulated	Cyprus	Cu 3.9, Pb __, Zn 2.1, Au 1.9, Ag 3.3	1.6×10^6 ($3 > 25 \times 10^6$)	49
Kuroko massive sulfides	0–0.8 km	Postulated	Kuroko[b]	Cu 3.5, Pb 1.9, Zn 8.7, Au 2.3, Ag 100	1.5×10^6 ($3 > 100 \times 10^6$)	432
Stock works	0.8–1.5 km	Postulated	Cyprus/Kuroko	Included	Included	Included
Vein systems	1–1.5 km	Postulated	Cyprus/Kuroko	Included	Included	Included
Exhalative beds	0.4 km	Escanaba Trough (?)	Besshi	Cu 3.8, Pb __, Zn 0.4, Au 0.76, Ag 9.5	2.2×10^5 ($3 > 25 \times 10^6$)	44
Deep seabed massive differentiates						
Podiform chromatic	8–10 km	Postulated	Numerous	Cr 46, Rh 5, Ir __, Ru __, Pd 2, Pt 4	2.0×10^4	174
Ni-Pt sulfides	> 10 km	Postulated	?			

[a] In USGS (1986).
[b] Kuroko-type deposites are formed in felsic volcanics in back-arc environments.
Conversion factor: 1 mi = 1.609 km.
Source: Cruickshank in McMurray, 1990.

avoidable and unavoidable impacts are summarized in Tables 22.8.7 and 22.8.8. For more detailed information, see the environmental case studies in 22.8.3.4.

22.8.2 TECHNOLOGY FOR MARINE MINING

22.8.2.1 Land-based Infrastructure

Marine operations need land bases, and the development of these follows the same principles applied to the development of any shoreside marine facility and will use existing infrastructure where possible. The availability of utilities, transportation facilities and services, and adequate space for docking, processing plants, and land disposal of waste are critical to marine operations. These are discussed in detail and referred to in a number of documents (Anon., 1983c; Cruickshank, 1988; Anon., 1990a).

22.8.2.2 Prerequisites for Marine Operations

Certain aspects of working in the oceans are common to all activities whether they are carried out for mining, oil and gas recovery, fishing, commerce, or defense. In basic terms, these include a vessel, a source of power, an understanding of the weather, a chart, a way to keep position, and a means of communication. Advances in technology for these systems are mostly driven by activities other than mining, and standard references are available, but where they are specifically relevant to minerals, state-of-the-art systems are discussed (Cruickshank and Marsden, 1973).

SHIPS AND PLATFORMS. There are three basic functions of platforms used in marine mining, namely, the support of equipment for observation and measurement, the support of tools and equipment for working with seabed materials (mining), and the transportation of personnel and materials. Platforms may be (1) airborne, floating, submerged, or in contact with the bottom; (2) self-propelled or stationary; and (3) manned or unmanned. Floating-platform size ranges from massive bucket ladder dredges weighing upward of 20,000 tons (18,000 t) to an individual diver weighing a few tens of pounds (kilograms). The design of marine platforms is a very specialized subject, and consideration of different factors is required for floating platforms and for platforms in contact with the bottom. Detailed design and construction generally are performed by the builder's yard, and include the preparation of working drawings, construction and launching. Roorda and Vertregt (1963) describe the design and construction of floating dredges in Europe for civil engineering purposes, and Scheffauer (1954) deals similarly with seagoing hopper dredges in the United States. There is no such treatise on mining dredges, but the reader is referred to Chapter 15.1 on placer mining and to other references at the end of that chapter dealing with dredge design.

Platforms in Contact with the Bottom—These do not as yet constitute a major requirement in marine mining plant design

Table 22.8.7. Avoidable Impacts of Marine Mining and Responsible Authorities in US

Potential Environmental Impacts	Avoidance/Mitigation Strategies	Relevant Federal Legislation/Authority: Statute	Regulations	Requirement	Agency
1. Deposition of dredged or fill material in waters of the United States and contiguous wetlands	Utilize upland disposal areas, contain spoil	Clean Water Act* (33 U.S.C. 1344)	33 CFR 323	Minimize deposition in wetlands and waters	COE/ EPA/ States
2. Interference with federally designated Marine sanctuaries	Avoid existing or potential Marine sanctuaries	Marine Protection-Research and Sanctuaries Act of 1972 (16 U.S.C. 1432–1433)	15 CFR 922	Preserve and protect Marine Sanctuaries	DOC
3. Interference with state designated Estuarine Sanctuaries	Avoid existing or potential Estuarine Sanctuaries	Coastal Zone Management Act of 1972 (16 U.S.C. 1461)	15 CFR 921	Preserve and protect Estuarine Sanctuaries	DOC/ States
4. Interference with historical and/or archaeological sites, structures, or objects	Perform adequate historic and archeologic surveys, avoid designated sites and structures	National Historic Preservation Act (16 U.S.C. 470f) Preservation of Historical & Archaeological Data Act (16 U.S.C. 469–469b). Exec. Order No 11593	36 CFR 800 36 CFR 800	Review by state and Federal officials to preserve and protect historic and archaeologic sites	State/ ACHP
5. Jeopardizing existence of endangered or threatened species or adversely modifying their habitats	Perform adequate biological surveys, avoid endangered species habitats	Endangered Species Act (16 U.S.C. 1531–1541)	50 CFR 17 50 CFR 222, 226, 227 50 CFR 402	Review to protect endangered and threatened species	All federal agencies FWS/ NMFS/ States
6. Harassment or "incidental taking" of marine mammals	Perform adequate biological surveys, avoid critical areas	Marine Mammal Protection Act (16 U.S.C. 1361–1382)	50 CFR 18 50 CFR 215–225	Protect marine mammals	FWS/ NMFS/ States
7. Increased risk of loss from or damage by flooding	Locate structures outside flood hazard areas	Exec. Order No. 11514 Exec. Order No. 11988	— —	No practicable alternative to encroaching on flood plain	All federal agencies
8. Destruction or modification of wetlands	Locate all development outside wetlands	Exec. Order No. 11990	—	Minimize effect on wetlands	All federal agencies
9. Conflict with designated Wilderness Areas	Locate away from designated Wilderness Areas	Wilderness Act* (16 U.S.C. 1131–1135)	43 CFR 19 36 CFR 293 50 CFR 35	Protect and preserve Wilderness Areas	DOI
10. Conflict with designated wild or scenic rivers	Locate away from designated Wild or Scenic Rivers	Wild and Scenic River Act (16 U.S.C. 1271–1286)		Protect and preserve Wild and Scenic Rivers	DOI
11. Conflict with Prime and Unique Farmland	Avoid areas of designated prime or unique farmland	16 U.S.C. 590 a–f	7 CFR 657	Protect Prime and Unique Farmland	USDA
12. Release of toxic or hazardous materials as a result of flood tsunami, hurricane, or seismic activity	Avoid natural hazard areas	Clean Water Act* 33 U.S.C. 1321 (c)(2) Resource Conservation and Recovery Act (42 U.S.C. 6901)	40 CFR 1510 40 CFR 260–267	Oil and hazardous substances spill contingency planning	EPA/CG

* The reader should note that the following are subject to the EPA consolidated permit program 40 CFR 122–125, 45 F.R. 33287–33558 (May 19, 1980):
—Resource Conservation and Recovery Act (Hazardous Waste Management Program)
—Safe Drinking Water Act (Underground Injection Control Program)
—Clean Water Act (National Pollutant Discharge Elimination System and Dredge and Fill Programs)
—Clean Air Act (Prevention of Significant Deterioration Program)

Source: Anon., 1981a.

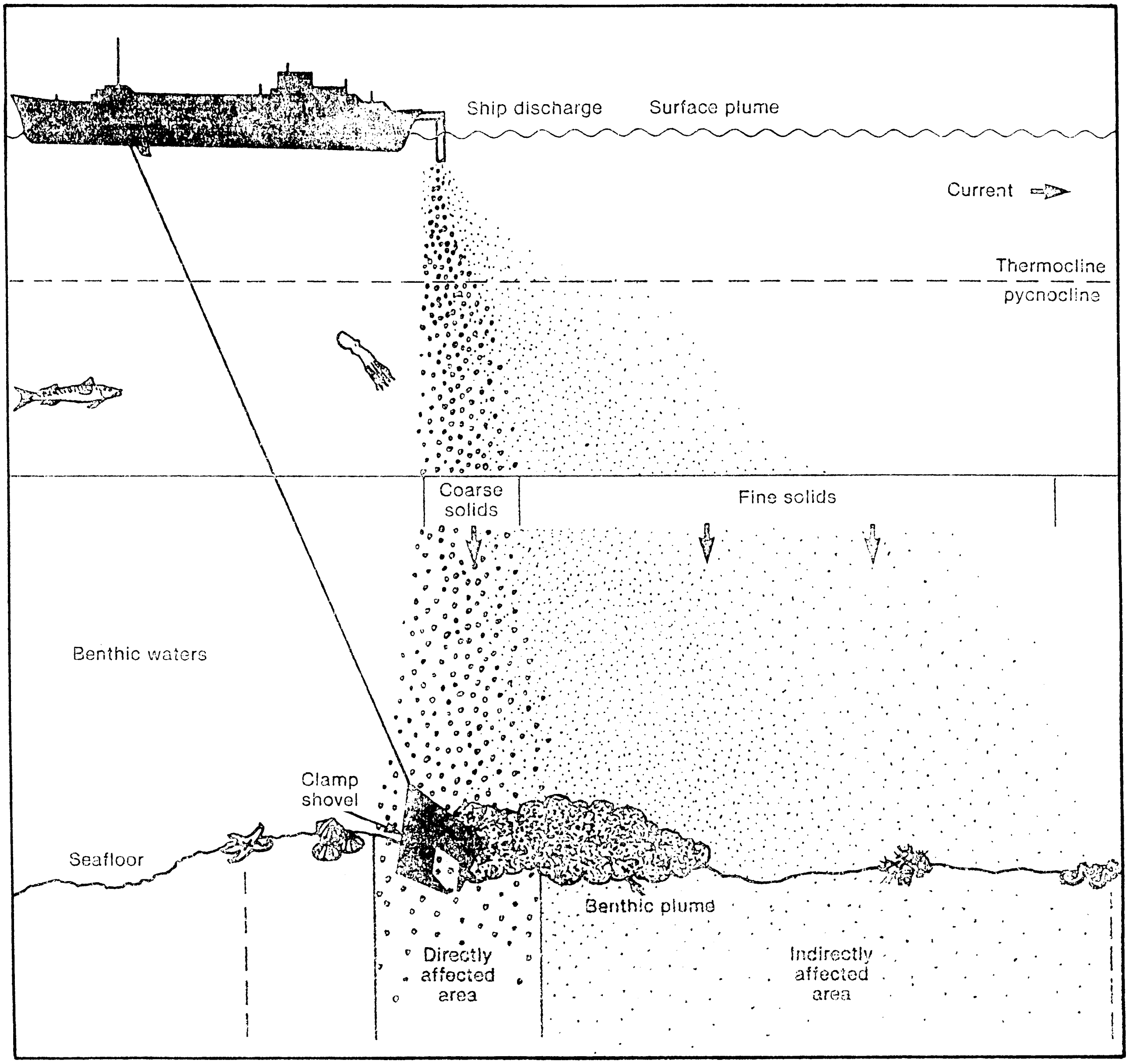


Fig. 22.8.5. Impacts of offshore mining on the marine environment (Anon., 1987a).

but are likely to be more important in the future, particularly for remote mining vehicles in deep water. In shallow water offshore, stationary platforms are used in the extraction of sulfur by the Frasch process and in the development of underground mines from artificial islands. In addition to the environmental considerations discussed for floating platforms, the design of fixed offshore structures should provide for extraneous loads due to earthquakes and for reduction in structural integrity due to fatigue and corrosion (Dean, 1969).

Acquisition of Platforms—Although construction of new platforms to suit each particular task is technically desirable, conversion of existing platforms is often necessary for other reasons. This mostly applies to ships, which may be purchased in used condition, or chartered on a bare boat, fully found or intermediate basis. Chartering of vessels for ocean operations requires specialized knowledge to insure the acquisition of an appropriate vessel and to avoid the assumption of unknown liabilities. Inexperienced charterers can find that they have obligated themselves to assume very large expenses of which they were unaware. Sometimes these may not even be properly chargeable to them. To protect themselves, charterers should always use the services of a competent admiralty attorney before making any commitment (Smith and Holm, 1969).

Conversion of seagoing vessels to mining dredges has been described by Boonyaprabhatsorn (1966), Webb (1965), and others, but none have had notable success. The trade-off of conversion or new construction should be considered very carefully in a production operation where the additional stress of mining excavation is added to those of seakeeping, or vice versa.

The costs of operating platforms at sea vary widely. Table 22.8.9 is useful as an indication of ownership costs for coastal vessels. Costs for mining vessels should be examined on a case-

Table 22.8.8. Unavoidable Impacts of Marine Mining and Responsible Authorities in US

Activity/Facility	Potential Environmental Impacts	Possible Mitigation Strategies	Major Relevant Federal Legislation: Statute	Regulations	Requirement	Agency
1. Transhipment from port to plant						
1.1 Channel and dockage	Marine traffic congestion, accidents	Improved navigation and traffic control	Ports and Waterways Safety Act (33, U.S.C. 1221–1227)	33 CFR 160–165	Conformance with safety regulations	USCG
	Dredging, bottom disruption	Optimum location and timing	Rivers and Harbors Act of 1899 (33 U.S.C. 403)	33 CFR 322	Permit	COE
	Modification of waters		Fish and Wildlife Coordination Act (16 U.S.C. 661–665)	50 CFR Part 470 Regulations reproposed 45 F.R. 83412, 12/18/80	Coordination	COE/FWS/ NMFS States
	Land use conflicts	Conformance with local land use plans	Coastal Zone Management Act of 1972 (16 U.S.C. 1456)	15 C.F.R. 930	Consistency with state Program	DOC/States
1.2 Marine terminal and transportation system	Construction noise	Equipment design and shielding	Noise Control Act (42 U.S.C. 4901–4917)	40 CFR 204 (const. equipment)	Conformance to equipment standards	EPA
	Operating noise	Buffer zones, quiet operations			Local controls	
	Dust	Containment and treatment	Clean Air Act* (42 U.S.C. 7410)	40 CFR 50–81	Conformance to standards, permits, monitoring, modeling.	EPA/States
	Liquid effluents	Containment and treatment	Clean Water Act* (33 U.S.C. 1311–1321, 1341–1345)	40 CFR 104–142	Conformance to standards, permits	EPA/States
	Land use conflicts	Conformance with local land use plans and coastal management programs	Coastal Zone Management Act of 1972 (16 U.S.C. 1456)	15 CFR 930	Consistency with state programs	DOC/States
2. Processing						
	Noise	Equipment design and shielding	Noise Control Act (42 U.S.C. 4901–4917)	40 CFR 204 (constr. equipment)	Conformance to equipment standards	EPA
	Dust, gaseous emissions	Process design, containment, and treatment; avoidance of "non-attainment" areas and Class I (pristine) areas	Clean Air Act* (42 U.S.C. 7401–7643)	40 CFR 50–81	Conformance to standards, permits, monitoring, modeling	EPA/States
	Liquid effluents	Process design, containment, and treatment	Clean Water Act* (33, U.S.C. 1311–1321, 1341–1345)	4 CFR 104–142	Conformance to standards, permits	EPA/States
	Energy consumption	Conservation, use of coal and/or other alternative fuels	Power Plant and Industrial Fuel Use Act of 1978 (92 Stat. 3291–3314)	18 CFR 285	Use of coal and/or other alternative fuels in new large power plants	DOE
	Land use conflicts	Conformance with local land use plans and coastal management program	Coastal Zone Management Act of 1972 (16 U.S.C. 1456)	15 CFR 930	Consistency with state programs	DOC/States

	Exposure to toxic or hazardous materials	Process design, containment, and treatment	Hazardous Materials Transportation Act (49 U.S.C. 1804–1806) Toxic Substances Control Act of 1976 (15 U.S.C. 2604–2627) Resource Conservation and Recovery Act* 42 U.S.C. 6901	41 CFR 171–179 40 CFR 715 et seq. 40 CFR 125? 40 CFR 260–267	Minimize exposure to toxic and hazardous materials	EPA/DOT
	Erosion and sedimentation, particularly during construction	Design and construction standards		40 CFR 260–2667	Local controls	EPA
3. Waste disposal 3.1 Onshore	Aquifer contamination	Optimum disposal area location, design and operation	Safe Drinking Water Act* (42 U.S.C. 300h–00i) Resource Conservation and Recovery Act (42 U.S.C. 6901)	40 CFR 146 40 CFR 260–267	Conformance to standards for underground injections	EPA/States
	Dust	Surface treatment of landfill, containment and treatment	Clean Air Act* (42 U.S.C. 7410)	40 CFR 50–82	Conformance to standards, permits, monitoring, modeling	EPA/States
	Contamination by toxic solid wastes	Optimum disposal area location, design, and operation	Resource Conservation and Recovery Act* (42 U.S.C. 6901–6979)	40 CFR 260–267 seq. 30 CFR 60?	Conformance to standards, permits	EPA/States
	Liquid effluents	Optimum disposal area location, design and operation; containment and treatment	Clean Water Act* (33 U.S.C. 1311–1321 1341–1345) Resource Conservation and Recovery Act (42 U.S.C. 6901)	40 CFR 104–142 40 CFR 260–267	Conformance to standards, permits	EPA/States
	Fossil fuel consumption	Energy conservation	Energy Supply and Environmental Coordination Act 15 U.S.C. 792		Regulate the use of oil and gas by power plants and other major fuel-burning installations	DOE
	Land use conflicts	Conformance with local land use plans and coastal management programs	Coastal Zone Management Act of 1972 (16 U.S.C. 1456)	15 CFR 930	Consistency with state programs	DOC/States
3.2 Offshore	Marine traffic congestion, accidents	Improved navigation and traffic control	Ports and Waterways Safety Act (33 U.S.C. 1221–1227)	33 CFR 160–165	Conformance with safety regulations	USCG
	Water pollution, bottom deposition, benthic smothering	Deposit well offshore in deep water, over infertile bottoms	Marine Protection Research & Sanctuaries Act of 1972 (33 U.S.C. 1411–1418)	40 CFR 220–229	Permits	EPA/COE?

* The reader should note that the following are subject to the EPA consolidated permit program 40 CFR 122–125, 45 F.R. 33287–33558 (May 19, 1980)
—Resource Conservation and Recovery Act (Hazardous Waste Management Program)
—Safe Drinking Water (Underground Injection Control Program)
—Clean Water Act (National Pollutant Discharge Elimination System and Dredge and Fill Programs)
—Clean Air Act (Prevention of Significant Deterioration Program)

Table 22.8.9. Average Cost of Owning and Operating Coastal Marine Equipment in US

Equipment	Average Annual Expense, % of Capital Investment Without Field Repairs: Depreciation	Overhauling, Major Repairs, Painting	Interest, Taxes, Storage Insurance	Total Ownership Expense	Average Use, Months per Year	Expense per Working Month, %	Application of Schedule to Owner's Values: Value, $ (Fill in values)	Expense per Working Month, $
Marine Equipment, Coastwise Craft:								
Derrick boat	10	8	16	34	8	4.3		
Dredge, clamshell	8	6	16	30	8	3.8		
Dredge, dipper	12	16	16	44	8	5.5		
Dredge, hydraulic	10	7	16	33	9	3.7		
Drill boat	8	11	16	35	10	3.5		
Lighter	10	8	16	34	10	3.4		
Mixer boat	12	16	16	44	8	5.5		
Pile driver	12	8	16	36	8	4.5		
Scow	8	6	16	30	10	3.0		
Scow, dump	12	11	16	39	8	4.9		
Tug	10	13	16	39	10	3.9		

Note: Column 3 is a summation of the following: interest, 5%; taxes, 1.5%; storage and incidentals, 4.5%; insurance, 5%.
Source: Anon, 1966.

by-case basis, and the use of used vessels might be considered (see 22.8.3). A new 5500-ton (5000-t) capacity, 330-ft (100-m) suction hopper dredge for sand and gravel mining in the UK was priced at $18 million (Anon., 1990b), whereas total systems costs for manganese nodule mining operations have been projected at upward of $1 billion.

Power Sources. Power sources for use in marine mining operations are subject to many of the same limitations applying to remote areas on land. Selection of the optimum power source must be weighed against the following factors:

1. Amount of power required, including peak power and sustained load.
2. Availability of suitable generating equipment.
3. Cost.

Power sources for marine use are many and varied and are constantly being improved. Most existing mining operations offshore use conventional power sources such as portable diesel electric generators on dredges and central power plants in land-connected undersea mines. There is a trend to the use of gas turbine generators in the offshore oil industry, and these should be considered in any similar mining venture.

Table 22.8.10 indicates the magnitude of the power source required for some actual mining operations compared with some conceptual operations. The trade-offs between high-voltage ac and high-voltage dc also should be considered with regard to transmission loss. The use of nuclear energy has been discussed for future marine mining operations by Duckstad (1959). For submerged instrumentation and measurement, many portable power sources are available (Victor, 1970; Anon., 1968), and a reliable and continuous supply of power aboard floating platforms is available from conventional sources.

Environmental Forecasting. Forecasting of the weather conditions likely to be encountered is essential to the safe conduct of operations. The best long-range estimates of future environmental conditions are provided by statistical summaries of past occurrences. Such climatological information commonly is presented in atlas form. Table 22.8.11 lists some of the more available and useful references that provide similar information for other characteristics and parameters. In addition to material and information contained in the references, more detailed information may be obtained by writing to the director, National Oceanographic Data Center, Washington, DC 20390, or to the director, National Climatic Center (previously called National Weather Records Center), Ashville, NC 28801. If the information is not already available in processed or printed form, and if the observational data are available, processing of data in a required format is available to the requester at cost.

Climatological atlases can provide a general insight into the gross conditions expected in the area but do not provide detailed enough information for design or planning for specific locations.

Table 22.8.10. Power Requirements for Some Dredge and Land Mining Operations in Horsepower per Ton per Hour Throughput

Description	Annual Capacity, Tons	Installed Power, Hp	Hp/Ton*
Conventional Dredging (1–200 ft)			
Bangka 1, bucket dredge (135 ft)	7.6×10^6	3,600	3.4
Franciscan hydraulic "30" (52 ft)	19.0×10^6	10,800	4.4
Dinosaur grab dredge (174 ft)	3.6×10^6	2,000	4.0
Neumann bucketwheel (33 ft)	21×10^6	2,000	5.0
Deep Dredging (200–600 ft):			
Conceptual	$6–10 \times 10^6$	5–10,000	6.0–7.2
Deep Ocean Dredging (10–20,000 ft)			
Conceptual	$2–3 \times 10^6$	85–135,000	300.0
Bottom-Situated Mining System			
Conceptual	2.4×10^6	5,000	12.0
Surface Open Pit			
Toquepala, Peru	5.7×10^6	177,000	6.2
Mt Goldsworthy, Australia	2.5×10^6	NK	3.0
Underground			
Strasser, Sweden	1.5×10^6	—	11.2
Easton mine, UK	0.52×10^6	1,340	19.0

*Throughput per hour.
Conversion factors: 1 ft = 0.3048 m, 1 ton = 0.9072 t, 1 hp = 0.7457 kW.
Source: Cruickshank and Marsden, 1973.

Table 22.8.11. References on Environmental Data and Charts

Type of Map or Chart	Publication Source
Hydrographic surveys, (including copies of original surveys from Archives of NOS)	National Ocean Survey, NOAA, US Dept. of Commerce, Rockville, MD 20852
Climatological and oceanographic atlases: air temperature, barometric analyses, storms and typhoons, precipitation, winds, clouds	US Weather Bureau, National Oceanic and Atmospheric Admin., Dept. of Commerce Silver Spring, MD 20910
Oceanographic data: tides, currents, sea waves, physical properties (temperature, density), ice	National Ocean Survey, US Dept. of Commerce, National Oceanic and Atmospheric Admin., Rockville, MD 20852 US Navy Oceanographic Office, Washington, DC National Oceanographic Data Center
US Coast Pilots (1–9): Atlantic Coast, Pacific Coast	National Ocean Survey, US Dept. of Commerce, National Oceanic and Atmospheric Admin., Rockville, MD 20852
Rivers and harbors, navigable waters	Address respective district offices, US Corps of Engineers, US Army (see Coast Pilot for area of concern).
Safety aids—notice to mariners (weekly publication); local notice to mariners	Jointly published by US Coast Guard and US Naval Oceanographic Office. (Refer to District Commander, local C.G. Dist.)
Charts of foreign waters	US Naval Oceanographic Office, Washington, DC British Admiralty, London, England
Horizontal and vertical geodetic control, triangulation and distance measurement surveys (coastal zone).	National Ocean Survey, NOAA, US Dept. of Commerce, Rockville, MD 20852
Oceanographic charts—sailing directions, list of lights, radio navigation aids, radio weather aids, weather station's index manuals: American Practical Nav., World Port Index, Hydrographic & Geodetic Survey Manual, Handbook of Magnetic Compass Adjustment, Radar Plotting Manual, Manual of Coastal Delineation from Aerial Photos.	US Naval Oceanographic Dist. Office, Clearfield Annex, Ogden, UT 84415 or US Naval Oceanographic Dist. Office, US Naval Supply Depot, 5801 Tabor Ave., Philadelphia, PA 19120 or US Naval Oceanographic Office, Washington, DC
Tables: distances between ports, navigation tables for marine and aviation, dead-reckoning altitude and azimuth tables.	
Atlases: surface current, No. Atlantic Ocean; sea and swell charts, So. Atlantic Ocean; sea and swell charts, No. Atlantic Ocean.	
Plotting charts	

Source: Cruickshank and Marsden, 1973.

Such information usually requires a detailed analysis of original observational data and theoretical computations.

If very specialized information is needed, the services of a competent consultant may be desirable. Some professional societies, such as the American Meteorological Society, maintain lists of certified consultants. When the duration of local observations is not adequate, theoretical diagnostic procedures or "hindcasting" similar to those used in short-range forecasting may be applied. The accuracy of hindcasting can be improved significantly when actual local observations are available for a duplicate time period. These become even more important when short-term forecasting is planned or when high accuracy is needed. Therefore, an observation program should be established as soon as possible. The program may vary somewhat according to the information needed but generally should include a six-hour minimum frequency of observation.

Techniques for marine environmental observation are not standardized and should be developed on an as-needed basis.

NAVIGATION AND SURVEY SYSTEMS. Minerals operations at sea require access to maps and charts at an appropriate scale. These are usually available from local sources or from international suppliers as indicated in Table 22.8.11.

Nearly all navigation and survey at sea is now carried out electronically in real time. Only on very minor projects near to shore, where costs must be held to the absolute minimum, are the time-honored methods of navigation or survey, using compass, sextant, and lead line, utilized. Electronic positioning systems are available in a wide range of capabilities and costs (Anon., 1983a), but continuing advances in global positioning systems (GPSs) may render many of these obsolete.

The GPS is a satellite-based radio navigation system that provides worldwide three-dimensional coordinates and, since its introduction in the United States in 1983, its growth has been phenomenal. It is based on simultaneous range measurement to several satellites, which can provide instantaneous navigational accuracy positions and also, with little additional observational or computational effort, high-precision measurements (Hodges, Johnson, and Bingley, 1988). Due to the high accuracy, simplicity, and relatively low cost (under $5000 for the basic receiver in 1990), the GPS is likely to supersede all other methods used at sea at distances beyond line of sight.

Where accurate positioning is needed for seabed operations, positioning transponders must be located on the seabed and surveyed in from the platform or from a submersible.

POSITIONING AND CONTROL. Maintaining and relocating the position of a vessel or platform at sea to any degree of accuracy is a complicated operation even under ideal conditions. Some of the factors that affect this position maintenance are

1. Sea state (wave heights, periods, etc.).
2. Sea-floor properties (type, strength, etc.).
3. Weather (wind force, etc.).
4. Anchor (type, size, holding power, etc.).
5. Mooring lines (cable and chain, etc.).
6. Mooring configuration (three-point anchoring, eight-point, etc.).
7. Vessel characteristics (size, natural periods, etc.).
8. Tensions in mooring lines.
9. Currents (speed, direction).
10. Electronic equipment (computer, sonar, etc.).

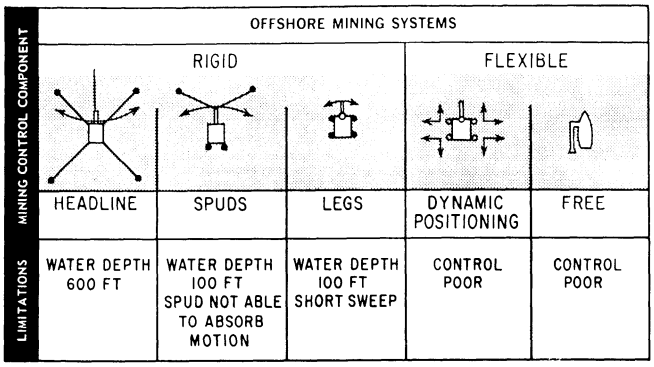


Fig. 22.8.6. Methods of dredge anchoring for mining system control. Conversion factor: 1 ft = 0.3048 in.

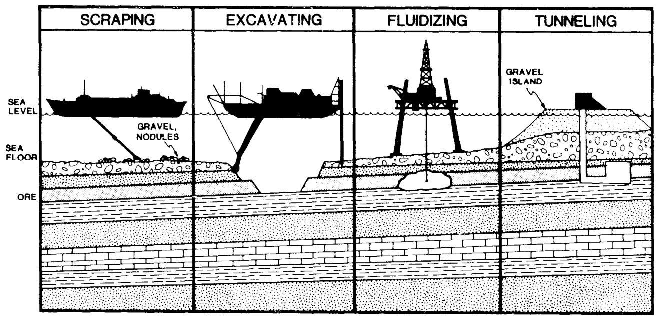


Fig. 22.8.7. Four basic methods of marine mining.

11. Vessel propulsion system (number of engines, horsepower, rating, type, bow thrusters, etc.).

Some of these factors are known, some are estimated, and some will have to be determined with models if the accuracy of the position is to be maintained within a few feet (meters) under a variety of sea and weather conditions.

For more detailed discussion of the different aspects of position control at sea, the reader is referred to Brink and Chung (1981), Graham, et al, (1969), and Macy (1966). Weigel (1964) lists 61 additional references.

Some methods of dredge anchoring for mining systems control are illustrated in Fig. 22.8.6. Deep-ocean mining dredges are not anchored, and because of the great depth, control of the mining device in a towed system is a major problem. The use of a self-propelled miner alleviates most of these problems but requires a much more complex system (see 22.8.3).

COMMUNICATION AND DATA PROCESSING. Communication between the operating platform and all other parts of the operating system, including the shoreside office, are prerequisites for successful operations at sea. Most communication problems have been resolved on the surface by the use of satellites, but some difficulties are still apparent in communication with remotely operated equipment in deep water. The use of fiber optics has been a major advance where tethered vehicles or equipment are employed (Anon., 1988).

Seabed mapping capabilities have been advanced considerably since the 1960s. Wide-swath mapping using side-scan sonar, similar to radar imagery, has been effected over large areas at rates of up to 10,425 mi^2 (27,000 km^2) each day. Detailed bathymetric mapping is now feasible using a variety of multibeam systems, giving horizontal and vertical accuracy within a few feet (meters) at a depth of 17,000 ft (5000 m). These methods are well summarized by the US Office of Technology Assessment (Anon., 1987), and other references are given therein.

Mineral exploration and mineral deposit characterization are both of paramount importance in the development of marine minerals. The reader is referred to some of the excellent treatises on this subject in which advances are constantly being made (Anon., 1987; Teleki et al., 1985; Cronan, 1986; Johnson, 1986).

22.8.2.3 Marine Mining Systems

There are four basic methods of mining solid minerals: (1) scraping or planing the surface, (2) excavating a pit, (3) removing them through a borehole in the form of a slurry or solution, and (4) tunneling into the deposit (Fig. 22.8.7). All deposits on land are mined by one or more adaptations of these methods, and marine mining is amenable to the same basic approaches whether on the continental shelf or in the deep seabeds. Each mining method has variations that may be tailored to a specific situation, and most of the deposit types can be mined by more than one method. Similarly, any one method can be applied to more than one deposit type (Cruickshank, 1978).

Marine mineral deposits, at or near the surface of the seabed, can be gathered by mechanical devices that scrape the seabed and gather the ore for lifting to the surface or for other treatment. In its simplest form, the action is like raking or shoveling or, in some cases, vacuum cleaning. Where a thick layer of hard material is present, the scraping action may be preceded by ripping or cutting. Mineral deposits lying wholly or partially beneath the seabed may be removed by excavation. The applicable mining methods range from those designed to recover free-flowing materials to those for excavating solid rock. Certain types of deposits may be mined by borehole methods using superheated water or high-pressure jets to transform the ore into a fluid or slurry, which is removed by pumping. In other cases, the minerals may be selectively leached and the valuable constituents recovered in solution. Deposits buried too deeply to excavate from the sea floor may also be mined by conventional underground mining methods using shafts and adits for access and ventilation, either from shore or from natural or artificial islands. The seabed location of these mines only slightly adds to the conventional problems of access, safe overhead cover, and ventilation.

This section further explains each of the mining systems listed on Table 22.8.12 and notes types of deposits with which it is normally used. For references describing commercial operations using these mining systems, see Cruickshank and Marsden (1973) and Cruickshank (1973).

SCRAPING. Although there are many variations, there are basically four mining systems that may be used to scrape the surface of the seabed. Two are wholly mechanical and two involve hydraulic action to lift the mined materials.

Dragline Dredge—Dredges of this type are used in offshore mining and deep seabed sampling, as well as in construction. Their use has been advocated for the recovery of seabed nodules and slabs of phosphorites (Mero, 1965). The material would be recovered by large dredge buckets that scrape slabs and nodules from the surface of the deposit and feed them into barges for transportation to shore (Fig. 22.8.8). Annual production at such an outer continental shelf (OCS) mine for phosphorite could be on the order of 400,000 tons (360,000 t), or roughly 180,000 yd^3 (138,000 m^3) of ore. This rate of production could involve a 50-yd^3 (38-m^3) bucket scraping an average of 20 tons (18 t) of phosphorite from the seabed every 20 min in a water depth of about 600 ft (180 m).

At an average abundance of 20 lb/ft (9 kg/m), nearly 2 mi^2 (5.2 km^2) of seabed would be mined annually. At a mine life of

Table 22.8.12. Methods of Mining Used or Proposed to be Used for Marine Mineral Deposits

	Scraping				Excavating						Fluidizing		Tunneling[1]	
Deposit Types	Dragline	Trailing Suction	Crust Miner	Continuous Line Bucket	Clamshell	Bucket Ladder	Bucket Wheel	Anchored Suction	Cutter Suction	Drill, Blast, and Dredge	Slurry Mining	Solution Mining	Store Entry	Artificial Island
Construction Materials														
sands	A	P		F	A	A	A	P	A		A			
gravels	A	P		F	A	A	A	P	A					
shells	A	A		F	A	A	A	A	P					
Placers														
heavy minerals	A	A		F	A	P	A	A	A					
gold	A					P	A							
diamonds							A	P						
Phosphorites														
sands or nodules	P	A			P		A		A					
crusts	P		F											
sediments							P				F			
Metal oxides														
nodules	A	P		F										
crusts			F	F										
Metal sulfides														
vent waters												F		
muds		F		F				P						
mounds & stacks							F			F		F		
veins												F		
deep massive														

[1] These methods have been used in shallow water for hard-rock deposits such as potash, barite, ironstone, and coal.
Symbols: P = principal method used; A = alternate method; F = future use possible.
Source: Anon, 1987b.

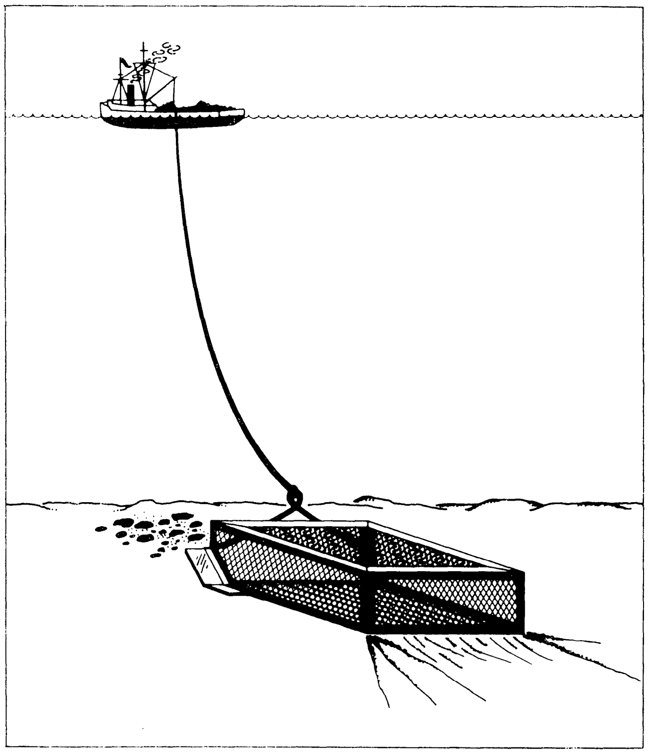


Fig. 22.8.8. Deep-water dragline dredge.

20 years, a total of 40 mi^2 (104 km^2) would be affected if the deposits were confined to the surface of the seabed.

Trailing Suction Dredge—A suction hopper dredge uses a pump to draw a slurry of bottom water and sediment into a riser or pipe leading to the mining vessel. As the sediment accumulates in the hopper, the water decants overboard. This system is used primarily for maintaining harbor channels. However, it is also used extensively for mining sand and gravel in water depths up to 120 ft (36 m) in the North and Baltic Seas (Padan, 1983). New vessels, such as the ARCO Avon launched in 1986, are designed to extend mining capability to depths of 150 ft (45 m) (Drinnan and Bliss, 1986). As its name implies, this type of dredge mines while in motion, creating numerous shallow trenches in the seabed, commonly about 3 ft (1 m) wide and 1 ft (0.3 m) deep (Fig. 22.8.9).

The trailing suction dredge uses one of several types of drag head with a coarse-grid steel framework across the opening of the suction head to prevent large rocks from entering the suction pipe. Coarser particle sizes are screened out and rejected after passing through the pump. Fine materials are washed overboard with the slurry overflow. In some cases, vibrating screens allow part of the sand fraction to be dumped back into the ocean, since the ratio of sand to gravel mined may differ from the desired marketable mix.

The use of sand and gravel as construction material requires that stringent measures be taken to avoid mining any clay layers, since the entire shipload could be contaminated and unmarketable. Additional detail on the equipment used and extensive photographic documentation are given by Hess (1971).

The volume of material that may be mined includes the sum of the marketable material, the fraction to be rejected, and the overburden initially stripped away. For example, in moving 1 million tons (0.9 Mt) of product to shore annually, an equal amount of sand might be rejected to reduce the sand/gravel ratio from 70:30 to a more marketable 40:60 (Anon., 1975). The 2 million tons (1.8 Mt) of annual excavation would result in a gradual lowering of the ocean floor by about 2 ft (0.6 m) over an area of 1 mi^2 (2.59 km^2) or about 2.1 million yd^3 (1.6 Mm^3).

Trailing suction dredges, modified for great depths, have also been used in tests to mine manganese nodules. Operating characteristics for this application are well documented (Anon., 1981a; Anon., 1984) (Fig. 22.8.10). Two basic systems or combinations use airlift or hydraulic lift with towed or self-propelled collectors. The complexities of the design for these systems are illustrated in Fig. 22.8.11 for a hydraulic lift with self-propelled collector as used by Ocean Mining Co. during successful tests in the Pacific.

Continuous Line Bucket Dredge (CLB)—This system (Fig. 22.8.12) has been tested for possible future use in recovering manganese nodules (Masuda et al., 1971) and is described in NOAA's EIS for deep seabed mining (Anon., 198 la). The CLB has also been proposed for mining phosphorite nodules and slabs and for cobalt crust mining; the original CLB system used one ship, empty buckets going down from the stern, and partially filled buckets coming in at the bow. The distance between the downward-moving and the upward-moving parts of the loop of the rope is dependent on the length of the ship and thus cannot be altered. Entanglement can be avoided by achieving an optimum combination of the ship speed and the length of the line or by the use of hydrodynamic deflectors. But the optimum combination may be disturbed by various other factors: the nature of the seabed, underwater currents, variations in the ship's course because of weather heading, or bad weather conditions.

To minimize these disadvantages, and to add to the flexibility of operation, a two-ship system has been designed where the empty buckets go down to the seabed from one ship and are brought up to a sister ship nearby. The distance between the descending and ascending parts of the loop and the curve of the loop can be influenced by the relative positioning of the two ships (Anon., 1984).

Crust Miner—Recent interest in manganese oxide crusts containing relatively high values of cobalt—and in some cases platinum—has led to proposals to develop the deposits. The manganese crusts vary in thickness from mere stains to layers about 15 in. (380 mm) thick and cover a variety of substrate rocks ranging from hard basalt to weak hyaloclastite. The physical properties of the crusts are similar to a hard coal, and they occur extensively in the Pacific on seamounts and submarine ridges at depths between 2600 and 7800 ft (800 and 2400 m). Mining systems proposed for this work include a vessel equipped with hydraulic lift systems and active bottom miners (Halkyard and Felix, 1987). The proposed system would be capable of breaking and removing the thin crust from the underlying rock and feeding it to a hydraulic lift system through a hydrocyclone to separate entrapped substrate (Fig. 22.8.13). The roughly cleaned ore would be pumped to the surface vessel for further cleaning and transport to shore.

Modular Mining System—In this system, an autonomous collector vehicle is launched with ballast material such that the weight in water of the ballast is equal to the weight in water of the nodules to be collected. The collector is designed to have sufficient buoyancy so that the vehicle is weightless in water. Thus, in descent, thrusters propel the unit down steadily against hydrodynamic resistance alone. The collector is propelled over the bottom, and as collection proceeds, ballast material is simultaneously ejected on an equal weight in water basis. In this manner, a small net weight in water of the collector is maintained. Mining is terminated shortly before ballast material ejection. Ballast material ejection is continued until the weight of

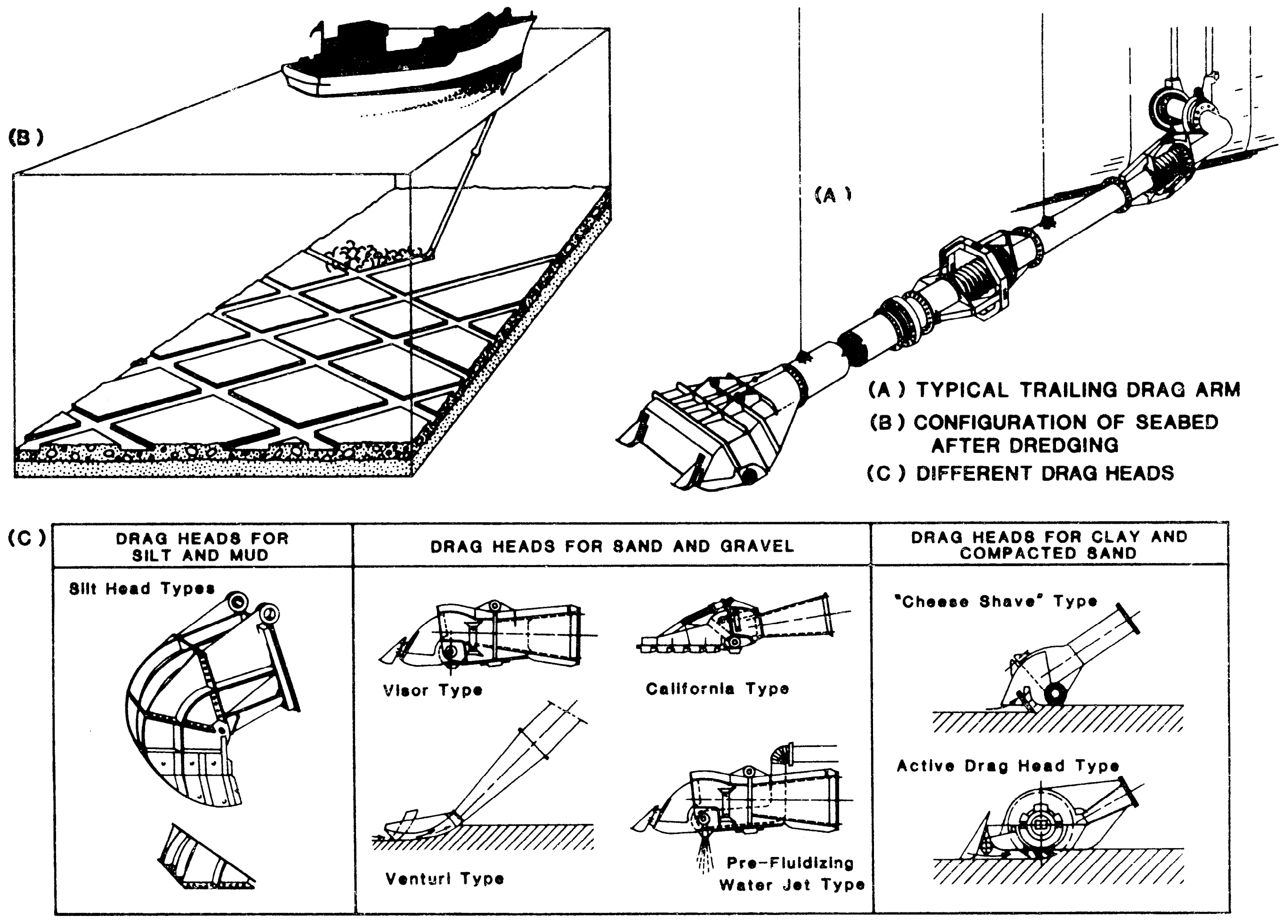


Fig. 22.8.9. Trailing suction hopper dredge and various drag heads.

the vehicle is zero or slightly negative. Finally, the vehicle is propelled by thrusters to the surface, docked with the surface ship, unloaded, serviced, and re-ballasted for a new mining cycle. In theory, very little onboard power is required to collect the nodules because the major source of energy is the potential energy of the ballast material. The operating principle of this system is illustrated in Fig. 22.8.14 (Anon., 1984). Processed tailings have been suggested for use as ballast. Advantage might be taken of ambient currents to propel the vehicle similarly to a sailplane or glider.

EXCAVATING. Mineral deposits that are located mostly within or under the seabed may be removed by excavation. These deposits include thick deposits of sands; metalliferous muds; layered or disseminated deposits of unconsolidated placer minerals or overlying bedrock; and deposits of consolidated minerals in vein, tabular, or massive form, which may extend for considerable distances into the bedrock. The mining system used will depend largely on the ease with which the material may be excavated and removed from its surrounding environment, the water depth, and the ocean climate in the area of operations (Cruickshank et al., 1969). Six examples of seabed mining operations are presented here that range from the excavation of free-flowing materials to the excavation of hard rock (also see Chapter 15.1).

Clamshell Bucket—Clamshell buckets have been used to mine sand and gravel in Japan and tin in Thailand, and to sample phosphorite off New Zealand. The buckets are mechanically actuated to bite into the seabed and remove material (Fig. 22.8.15). The need for multiple cables to actuate the grabs can cause complications, particularly in heavy seas where wave compensating devices may also be needed. Moreover, the clamshell is inefficient in clearing bedrock of fine materials. It is best suited for excavation of large-size granular material where accuracy of positioning and cleanup is not important. The size of buckets may range from a few cubic feet (cubic meters) to as much as 10 yd^3 (7.6 m^3).

Bucket Ladder Dredge—The bucket ladder consists of a chain of closely connected buckets mounted over a heavy digging arm or ladder (Fig. 22.8.16). It is most efficient for excavation of deposits containing boulders, clay, and/or tree stumps and weathered bedrock. Dredges of this type have been used successfully all over the world for mining gold, tin, and platinum placers and diamond deposits, although their use offshore has been limited to gold and tin. They are frequently used for clearing harbors because of their capability for digging into broken rock and coral. The bucket ladder delivers a virtually water-free product to the mineral processing plant on board the dredge. Discharge of water from the shipboard operations is limited to that needed to concentrate the valuable constituents by techniques based on the use of flowing water to remove the less dense materials. In the case of gold, the bulk of the concentrate recovered is only a few parts per million so that virtually all the material removed from the deposit is returned to the seabed.

Considerable turbulence accompanies these operations, which may involve the movement of up to 6.4 million yd^3 (4.9 Mm^3) of material per year. Bucket ladder dredges are limited

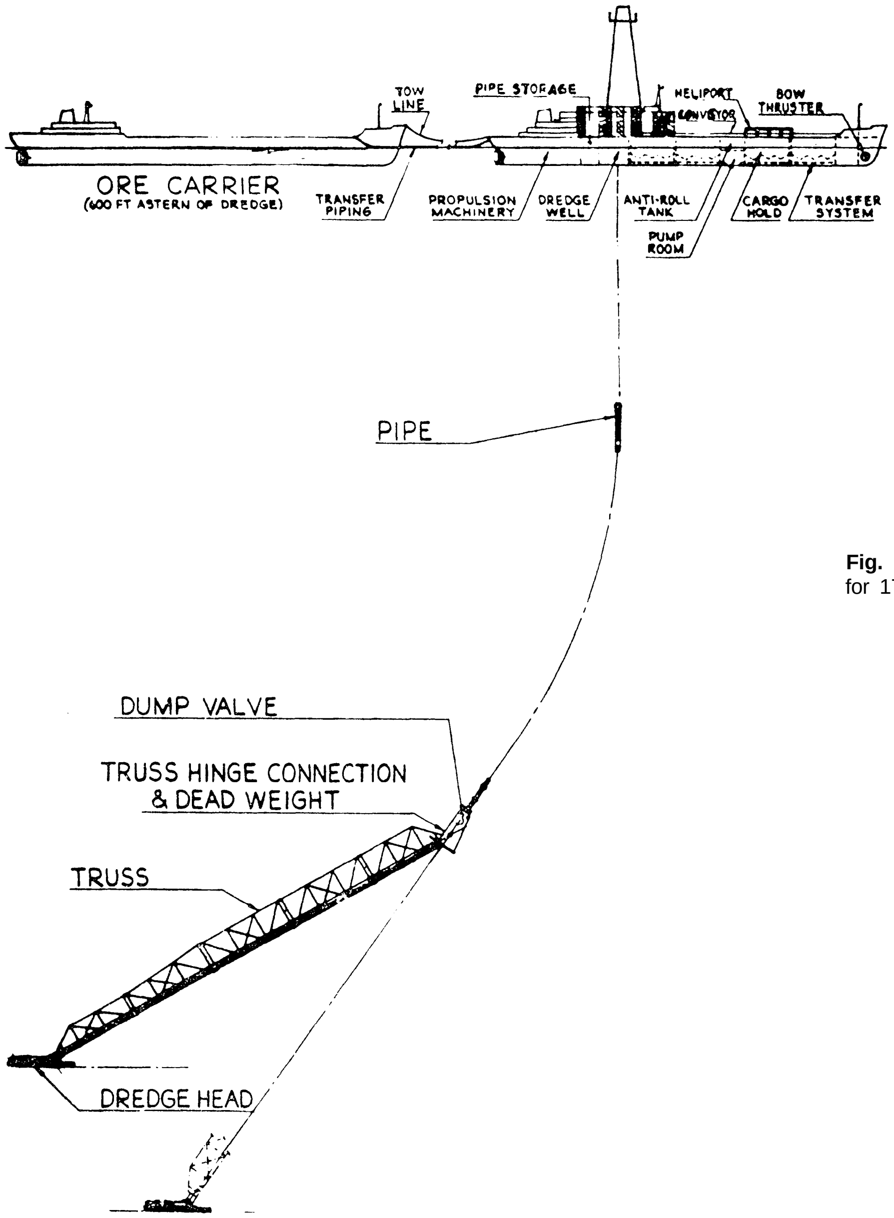


Fig. 22.8.10. Trailing suction hopper dredge for 17,000-ft (5000-m) water depth (Kaufman et al., 1985).

to depths of about 150 ft (45 m) and rarely operate at depths over 65 ft (20 m). Few of these dredges have been built for offshore mining in the last 20 yr, and it is likely that they will be superseded by the bucket wheel suction dredge.

Bucket Wheel Suction Dredge—Bucket wheel dredges use a small-diameter bucket wheel mounted on the suction ladder to excavate material. This combines the best aspects of the bucket ladder and suction dredges (Fig. 22.8.17). Very high torque or digging power can be applied to the wheel, which can deliver the excavated material directly into the mouth of a suction pipe for transport to the sea surface. Digging capability is equal to that of the bucket ladder with respect to ease of digging and bucket capacity, while the depth capability using submerged pumps is almost unlimited. The combination of simultaneous digging and suction at the seabed provides the option to either treat the ore on the vessel or pipe it to shore.

Stationary Suction Dredge—Anchored suction dredges (Fig. 22.8.18) are widely used in Japan for mining sand and gravel at depths less than 100 ft (30 m). These dredges have been used in Britain as well, although the vessels built since 1980 are virtually all trailing suction dredges (Drinnan and Bliss, 1986).

In contrast to the trenches left by trailing suction dredges, anchored suction dredges leave pits in the seabed. An anchored suction dredge has also been tested for mining metalliferous muds at a depth of 6500 ft (2000 m) in the Red Sea (Fig. 22.8.19). Mining involved the conversion of a deep drilling oil exploration vessel (the SEDCO 445) to carry a specially designed mud pump and delivery pipe that was lowered to the seabed in a manner similar to the lowering of a drill pipe. The pump was vibrated into the mud with a waterjet to fluidize the material and the ore pumped to the ship for treatment (see also 22.8.3.2).

Cutterhead Suction Dredge—Typically, cutterhead suction dredges are used to excavate fairly compacted, granular materials in water less than 100 ft (30 m) deep. The rotating cutterhead is usually an open basket with hardened teeth or cutting edges, somewhat like an over-sized dentist's drill (Fig. 22.8.20). The

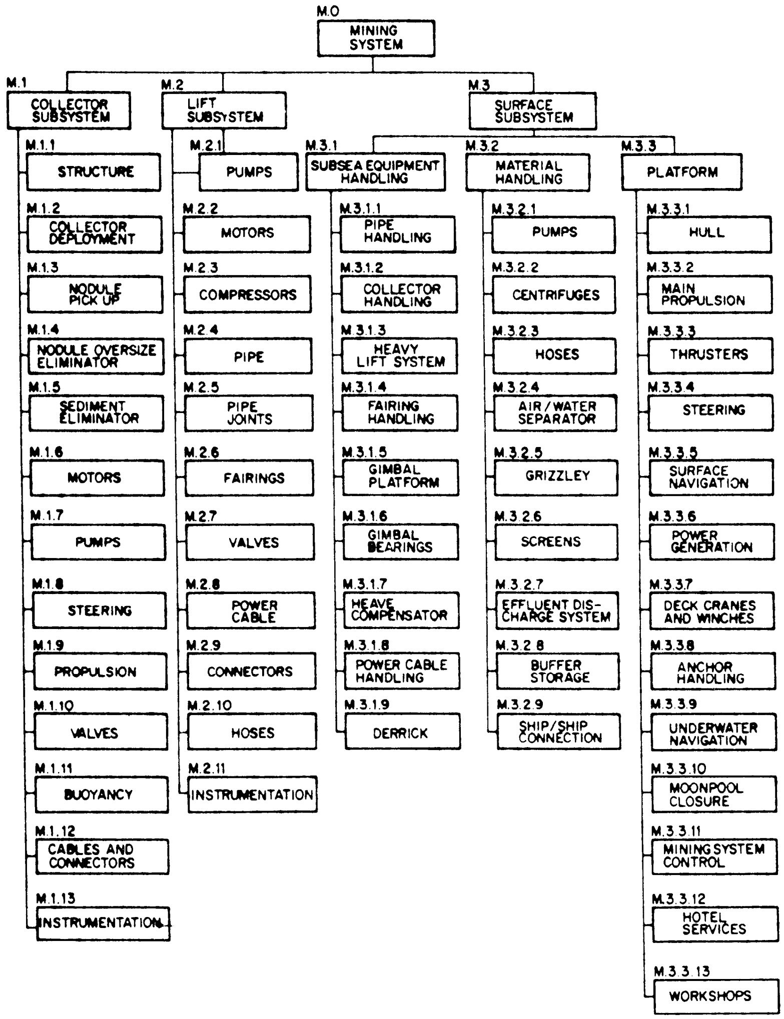


Fig. 22.8.11. Schematic of subsystems required for deep seabed suction dredge with self-propelled miner (Anon., 1984).

end of the suction pipe is normally located within the basket. In standard practice, the dredge is swung back and forth in an arc pivoted from a large post or spud attached to the stern. The dredge cutterhead cuts downward a short distance with each swing. Because the cutter rotates in one direction only, the bite is much stronger on one swing than the other. In mining for heavy mineral sands, the action of the cutter tends to disintegrate the material, allowing heavy minerals to separate, fall below the cut, and be left on the seabed. Cutter suction dredges have never been used successfully for mining gold, although they have been widely used for mining cassiterite (tin placers) in west Thailand where the deposits are rich enough to economically sustain the inefficient cleanup.

Suction dredges circulate large quantities of slurry that must be decanted on board the dredge or pumped ashore by pipeline. In either case, there is a significant discharge of water containing fine particulate materials. Treatment of the decanted solids may be unnecessary for construction sands and gravel but may be required for heavy minerals. The valuable constituent or concentrate from these ores will rarely amount to more than a small percentage of the materials mined. Therefore, more than 95% of the material dredged from such deposits must normally be disposed of either at the mine site or the shoreside treatment site.

The cutterhead suction dredge may also be equipped with a multiblade ripper to cut into moderately consolidated rock. Present use is limited to the excavation of soft rock, such as coal and shale. However, advances in rapid tunneling technology suggest that rock cutterheads could be designed for medium strength rocks, such as sandstone and limestone (Hignett and Banks, 1984; also see Chapters 9.1 and 22.1).

Drilling and Blasting—Deposits that are too hard to excavate by dredging must be broken by other means. The normal system for excavating hard-rock deposits is to drill into the deposit for the placement of explosives. Fracturing using highly pressurized fluids is also possible (Chapter 22.3) but would represent a very special case and is not considered here. Blasting of the material is only an intermediate step and would be followed by the gathering and lifting of the ore by one of the methods previously described. Blasting operations are designed to expend as much force as possible on fragmenting the ore so the water column effects are much lower than those from equivalent unconfined explosions.

With respect to the possibility of mining deep-seabed deposits that require fragmentation, many more aspects need to be examined. Technically acceptable means of drilling and fragmenting hard rock in deep water have not yet been developed. However, methods of gathering and lifting the fragmented material may be assumed to be similar to the methods developed for deep-seabed nodule mining. For drill-blast coverage see Chapter 9.2.

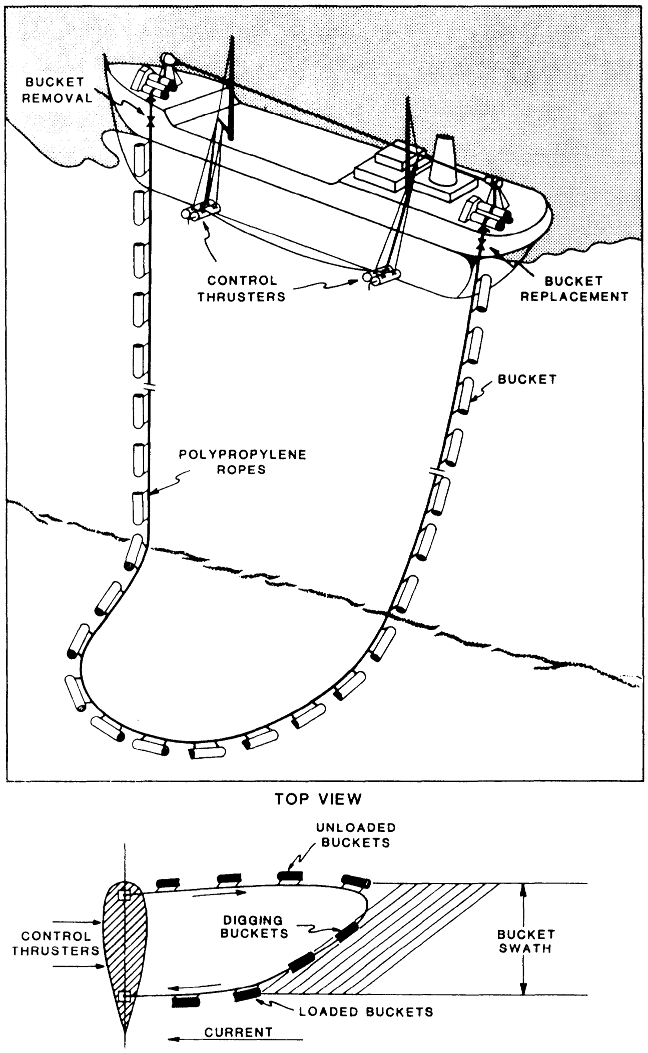


Fig. 22.8.12. Continuous line bucket dredge (Masuda et al., 1971).

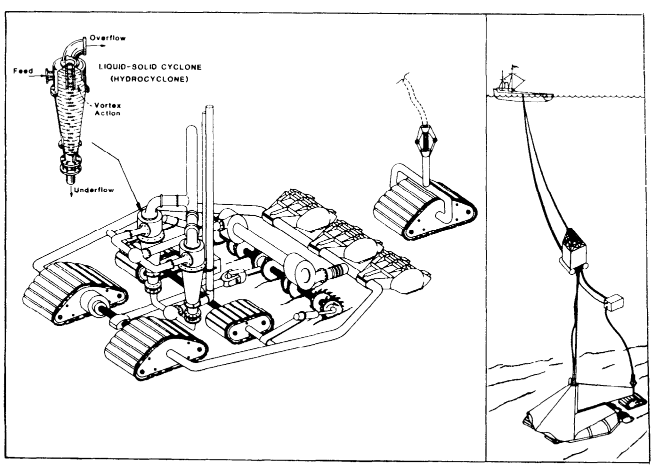


Fig. 22.8.13. Proposed deep-seabed cobalt crust mining machine (after Halkyard and Felix, 1987).

FLUIDIZING. *Fluidizing (Slurries).* Under proper conditions, certain types of unconsolidated or marginally consolidated mineral deposits may be mined as a fluid slurry through drillholes penetrating the sea-bed (Chapter 22.4). Sub-seabed sand was mined in this way in shallow waters offshore Japan in 1974 (Padan, 1983) and recent onshore experiments in Florida proved the capability of this approach in recovering phosphate from beneath thick overburden (Fig. 22.8.21). In the recovery of sulfur, super-heated water is pumped into the deposit to melt the sulfur so that it may be pumped out of the ground (see Chapter 15.3.2).

Fluidizing (Solutions)—Hard rock deposits that are amenable to hydrometallurgical treatment of their ores are potentially extractable by fluidizing methods (Fig. 22.8.22). The valuable constituent is dissolved in place, and the pregnant solution is removed through a borehole (Chapter 15.3). There are major unsolved problems in dealing with toxic or corrosive solvents used to enlarge fractures to provide a flow path for the solvent through the deposit and to selectively extract the desired metals in complex ores.

These problems are being overcome on land, however, and the methods developed there should be applicable in more sophisticated form to seabed deposits.

Effects of accidental spills of solvents would depend on the nature of the solvents and the sites impacted. They could be localized so long as the solvent is water soluble because of the rapid dilution prevailing in the oceans and the buffering capacity of seawater. However, the pollution potential cannot simply be ignored.

TUNNELING BENEATH THE SEA FLOOR. Underground mining by tunneling is commonly practiced in subsurface hard rock. In certain cases, sub-seabed deposits of bedded coal, potash, and ironstone, as well as veins of lead, copper, and tin have been mined by conventional underground methods. Entry to these mines is either from the shore or from natural or artificial islands in shallow waters (Fig. 22.8.23). The location of the mines in the seabed only slightly adds to conventional problems of access, safe overhead cover, and ventilation. The effect on the environment is similar to that for any shoreside mine. The possibility of developing underground access through seabed airlocks has been considered for special cases (Austin, 1967).

MINING OF METALLIFEROUS SULFIDES. Possible methods for mining the deep seabed deposits of metalliferous sulfides are indicated in Table 22.8.13.

22.8.2.4 Processing of Marine Ores

There are two areas in which the processing of marine ores calls for special treatment as distinguished from the processing of terrestrial ores. One is the effect of platform motion on separation processes carried out at sea; the other is in the extraction of metals from specifically marine ores such as manganese nodules and crusts. Some industrial materials such as specialty sands may require washing to remove saltwater, but for the most part, marine ores do not otherwise differ from those on land and use conventional methods of treatment.

Materials handling in the marine environment is complicated by the motions of the platform and the need in most cases for closed cycle systems and stringent controls to avoid spillage or contamination. These issues are normally addressed in the environmental analyses that are a part of the planning process. For offshore dredging systems, the reader is referred to Chapter 15.1 on placer mining; in the case of deep seabed mining, the problems would be addressed on a case-by-case basis.

SEPARATION PROCESSES. Processing technology applied to placer materials has been fairly well established over the years (Chapter 15.1). Due to the conservative nature of the industry,

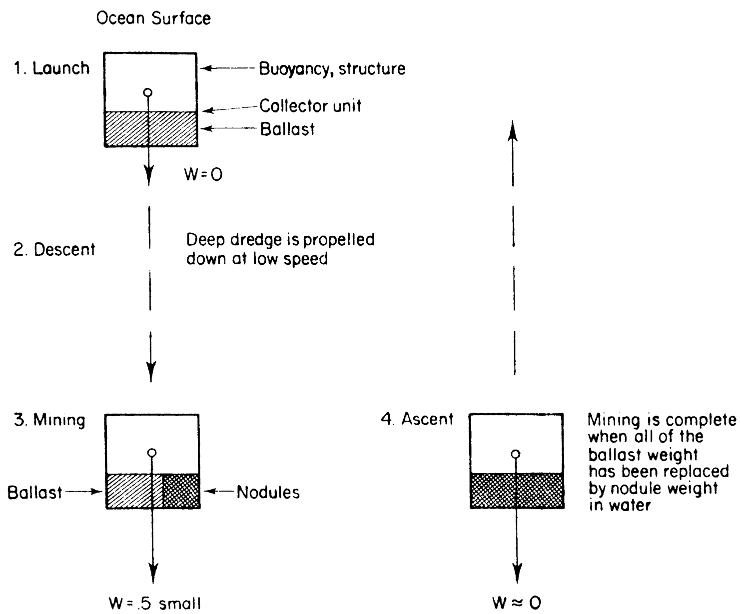


Fig. 22.8.14. Operating principal of a modular mining system for the deep seabed (Anon., 1984).

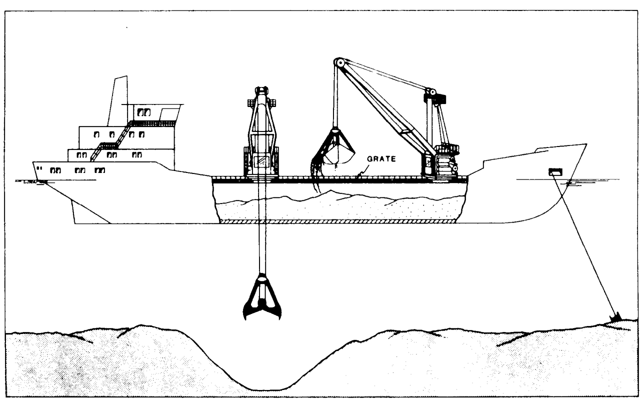


Fig. 22.8.15. Clamshell hopper dredge.

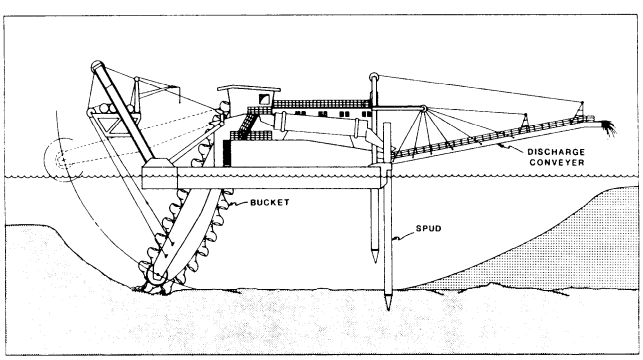


Fig. 22.8.16. Bucket ladder dredge with spuds for mining gold in sheltered waters.

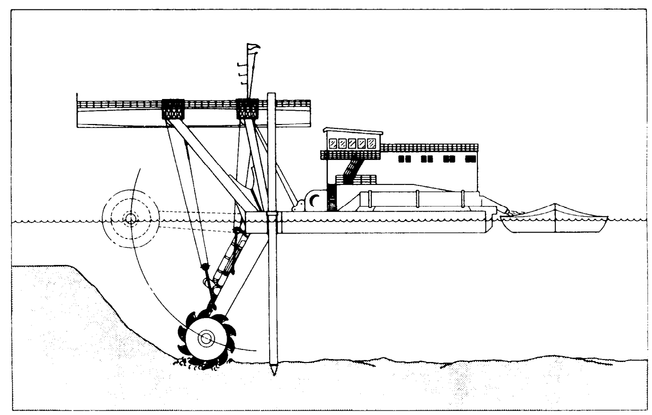


Fig. 22.8.17. Bucket wheel suction dredge with spuds for mining in sheltered waters.

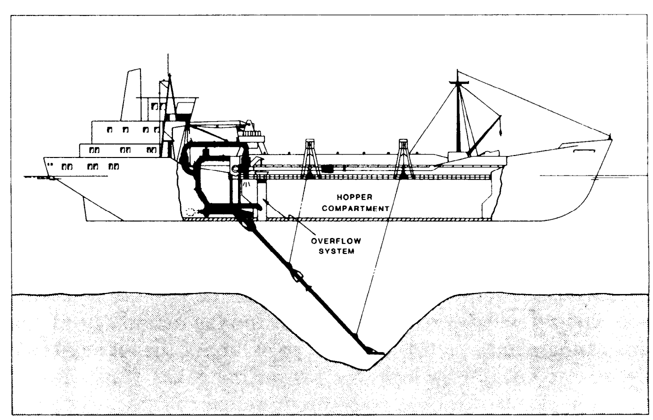


Fig. 22.8.18. Anchored suction hopper dredge for mining in shallow water.

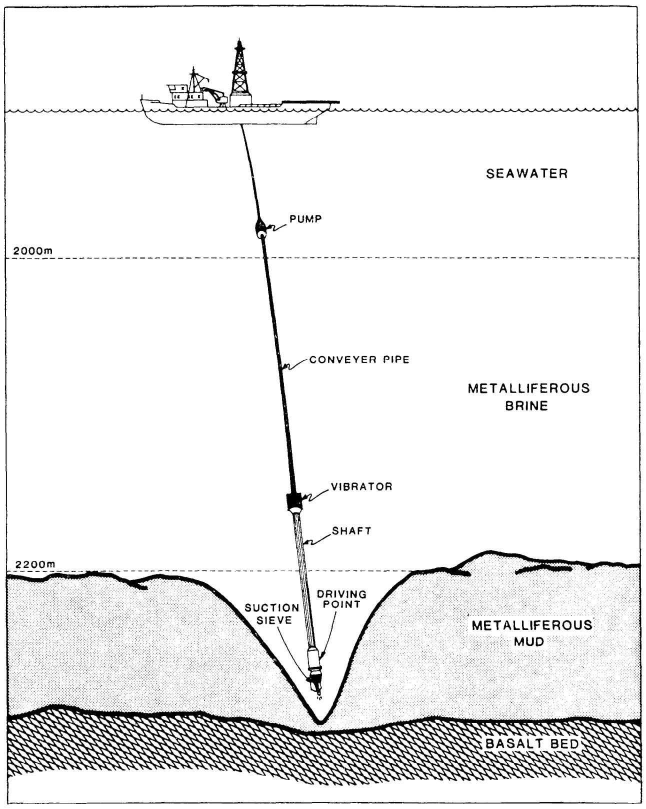


Fig. 22.8.19. Dynamically anchored suction dredge for mining in deep water.

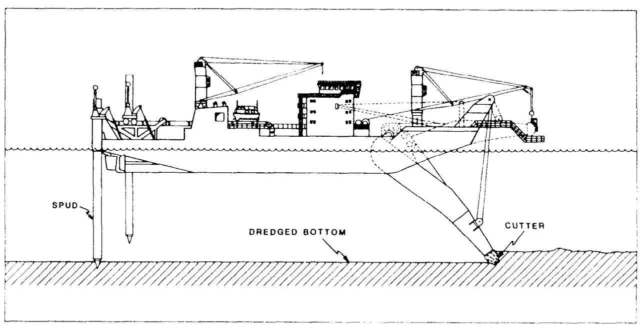


Fig. 22.8.20. Cutterhead suction dredge with spuds.

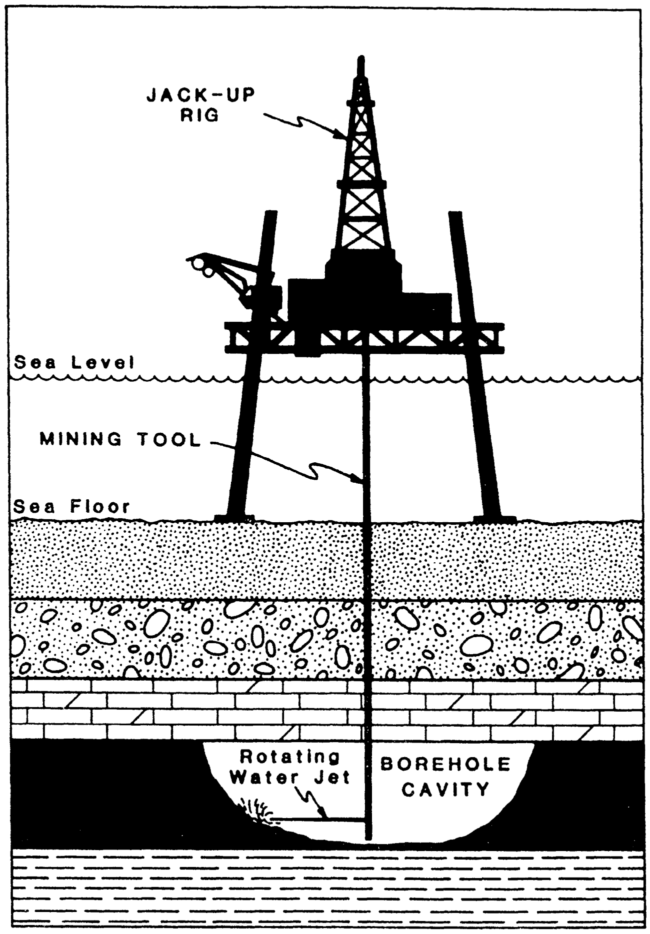


Fig. 22.8.21. Fluid mining by slurrying from a fixed platform in shallow water (after Hrabik and Godesky, 1985).

however, there is a great need for improvement utilizing newer technology, particularly as operators have to recover lower-grade materials and work very fine-grained deposits in the marine environment. The use of placer processing technology on board floating platforms subject to the rigors of the motions of the high seas has been quite limited to date. Thus in the selection of processing equipment for proposed operations at sea, considerable testing will be required to confirm the suitability of the processing plant in terms of both the nature of the material and the environment in which the processing takes place. Initial selection of treatment methods can be indicated from laboratory tests using a few pounds of sample, but in complex ores, which includes most placers, pilot plants to run tests on tens or hundreds of tons of sample are generally required. Table 22.8.14 lists the various options for separation applicable to marine placers (Cruickshank, 1988).

EXTRACTIVE PROCESSES FOR MANGANESE NODULES. The composition of manganese nodules and high-cobalt-manganese crusts is very similar, and extractive processes for nodules are generally assumed to be interchangeable (Haynes and Magyar, 1987).

Of the various processing schemes available for manganese nodule processing, five are considered most probable for first-generation commercial applications:

1. Gas reduction and ammoniacal leach (modified Caron).
2. Cuprion ammoniacal leach.
3. High-temperature and high-pressure sulfuric acid leach.
4. Reduction and hydrochloric acid leach.
5. Smelting and sulfuric acid leach.

The two ammoniacal and high-temperature and high-pressure sulfuric acid processes are designed to recover three metals (Co, Cu, and Ni), and the other two processes are designed to recover four metals (Co, Cu, Ni, and Mn). In the three-metal processes, manganese could also be recovered from the tailings if favorable economic conditions exist. This recovery would alter both the chemical and physical characteristics of the tailings. A variety of wastes is generated during nodule processing with tailings composing the majority of material. Other wastes generated are not generally unique to nodule processing and also constitute a small fraction of the waste. These wastes

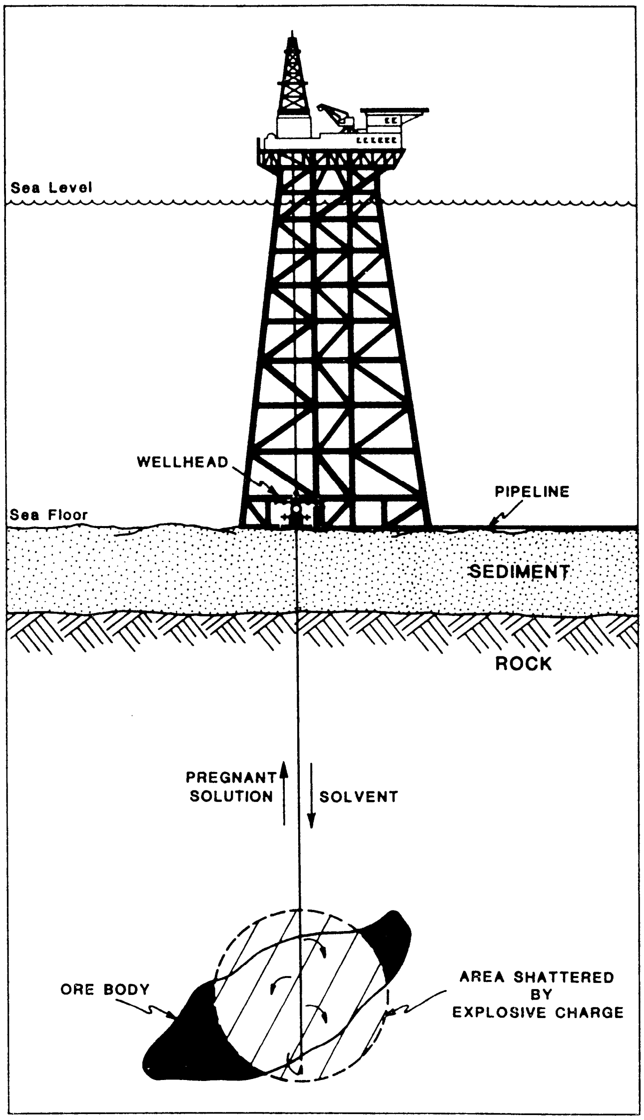


Fig. 22.8.22. Fluid mining by leaching of deep, hard-rock deposits, artificially fractured in place.

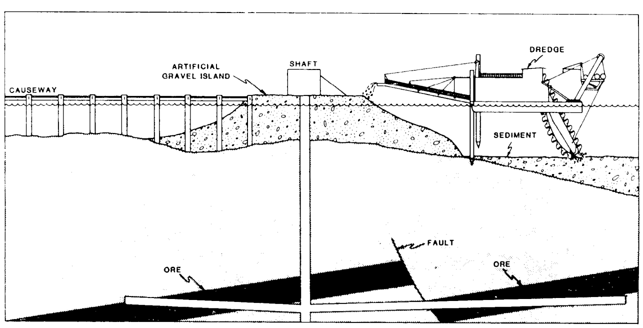


Fig. 22.8.23. Mining of subseabed deposits with access through an artificial island.

include utility and boiler ashes, scrubber sludges, electrowinning sludges, lime boil solids, and other miscellaneous plant wastes. Compositions of these wastes are generally known and not discussed further here. A brief flowsheet of each process is presented in Fig. 22.8.24 a-e. More thorough discussions of these processes are given elsewhere (Anon., 1987).

DISSOLVED MINERALS IN SEA WATER. For discussion on this subject, see Cruickshank and McIlhenney (1973). Salt, magnesium metal and compounds, fresh water, and bromine are recovered commercially. Principal recovery processes include precipitation and evaporation.

22.8.2.5 Technology for Environmental Impact Mitigation

The technology for environmental impact mitigation is developed on a case-by-case basis and is not well established at this time because impacts at sea are not well defined. Most studies done so far have documented discharges of waste materials from land-based mining operations or coastal smelters and refineries (Ellis, 1982). Other studies are listed in environmental impact statements for marine mining activities (see 22.8.3.4).

22.8.3 CASE STUDIES IN MARINE MINING

22.8.3.1 Dredging from the Continental Shelf

Dredging from the continental shelf is quite widespread, and selected operations for mineral sands are listed in Table 22.8.15. A useful reference for potential operations in US seabeds is Anon., 1979. Operations for sand and gravel are discussed in Chapter 15.1.

Tin in Indonesia (Zaalberg, 1970). Dredging for offshore tin deposits has been conducted off the island of Singkep since 1910 and the islands of Banka and Belitung since 1937. The deposits are drowned eluvials and alluvials in an area of still unresolved geomorphology. They are found in watersheds, valley terraces, and in valleys, and are concluded to be residual in character. The dredging operations in the 1950s are very well described by Zaalberg (1970), and specifications of some of the dredges presently operating are given in Table 22.8.15. All are bucket ladder dredges of the headline type. A typical flowsheet for the beneficiation process is shown in Fig. 22.8.25. A considerable auxiliary fleet of tugs, anchor boats, and motor launches is required to maintain the offshore operations, which involve dredging overburden first and then the ore-bearing ground.

The water is generally quite shallow, and in starting a new dredging block, an access channel for the dredge is prepared, and headlines and sidelines laid out with their anchors. After connection with the lines, the dredge usually removes overburden and ore alternately in an arc from side to side across the face determined by the headline. Normal width of cuts is 300 ft (100 m), but using multiple headlines, cuts up to 900 ft (300 m) can be made. At each pass, covering about 35 ft (100 m) in 15 min, the dredging ladder is lowered about 30 in. (800 mm) in normal ground or 18 in. (500 mm) in very hard ground. Care is needed to assure that the overburden in not discharged in a fashion that will allow it to contaminate the unmined ore, and the overburden cut does not usually advance much in front of the ore cut. Forward movement of about 4½ ft (1.4 m) per cut is controlled by markers on the headline or by sextant or electronic positioning. Hourly dredging rates of 600 yd^3 (500 m^3) allow the treatment plant to be put into operation about 18 hr after the commencement of overburden dredging. Cleaning of the bottom may be difficult and time-consuming in the presence of limestone

Table 22.8.13. Methods of Mining Potentially Applicable to Submarine Metalliferous Sulfides

Nature of resource model	Nature of deposit	Depth of occurrence	Scraping	Excavating	Fluidizing: Slurry	Fluidizing: Solution	Tunneling
Geothermal energy	Hot rocks or interstitial fluids	0–1.5 km	na	na	na	[a]	na
Hot vent fluid	Fluids	Surficial	na	na	na	[b]	na
Metalliferous muds	Semifluid	0–30 m	[c]	[d]	[e]	na	na
Volcanogenic sediment (Mn)	Hard rock	Surficial	[c]	[d]	na	na	na
Surficial precipitates							
Individual stacks or mounds	Porous, brittle	Surficial	[c]	[d]	na	na	na
Coalesced stacks or mounds or cemented tallus	?	Surficial	[c]	[d]	na	na	na
Subseabed precipitates							
Cyprus massive sulfides	Hard rock	0–0.8 km	[c]	[d]	na	[f]	[g]
Kuroko massive sulfides	Hard rock	0–0.8 km	[c]	[d]	na	[f]	[g]
Stock work	Hard rock	0.8–1.5 km	[c]	[d]	na	[f]	[g]
Vein systems	Hard rock	1–1.5 km	na	na	na	[f]	[g]
Exhalative beds	Soft to hard rock	0–4 km	[c]	[d]	[e]	[f]	[g]
Deep seabed massive differentiates							
Podiform chromatic	Hard rock	8–10 km	na	na	na	[f]	[g]
Ni-Pt sulfides	Hard rock	>10 km	na	na	na	[f]	[g]

[a] Heat transfer to fluids in boreholes.
[b] Vent capture: minerals already in solution.
[c] Applicable only to materials on or above seabed.
[d] Subseabed materials exposed at or close to seabed.
[e] Unindurated or easily granulated material.
[f] Assumes technical advances in in situ hydrometallurgy.
[g] Applicable only if close to land or if new techniques of seabed entry are developed.
na: Not applicable.
Conversion factor: 1 mi = 1.609 km
Source: Cruickshank, 1990.

Table 22.8.14. Options for Concentration of Marine Placer Minerals

Options	Au	pgm	di	Sn	Fe	Th	Ti	Zr	Cr
Pretreatment									
slurrying	(*)	*	(*)	(*)	*	*	*	*	*
sizing	(*)	*	(*)	(*)	(*)	*	*	*	*
classification	*	*	(*)	*	(*)	*	*	*	*
densification	*	*	*	*	*	*	*	*	*
distribution	(*)	*	(*)	(*)	(*)	*	*	*	*
Gravity Concentration									
riffles	*	*		*					
pinched sluices								*	
spirals				(*)[1]	*	*	*	*	*
jigs	(*)	*		(*)	*				
tables	*		(*)						*
cones				*					
heavy medium	*		(*)						
Magnetic Separation									
wet process					(*)	*	*		
dry process					*	*	*	*	*
Electrostatic Separation									
normal						*	*		*
high tension								*	

* = suitable for normal use
(*) = documented use at sea
[1] = but not in successful production
pgm = platinum group metals

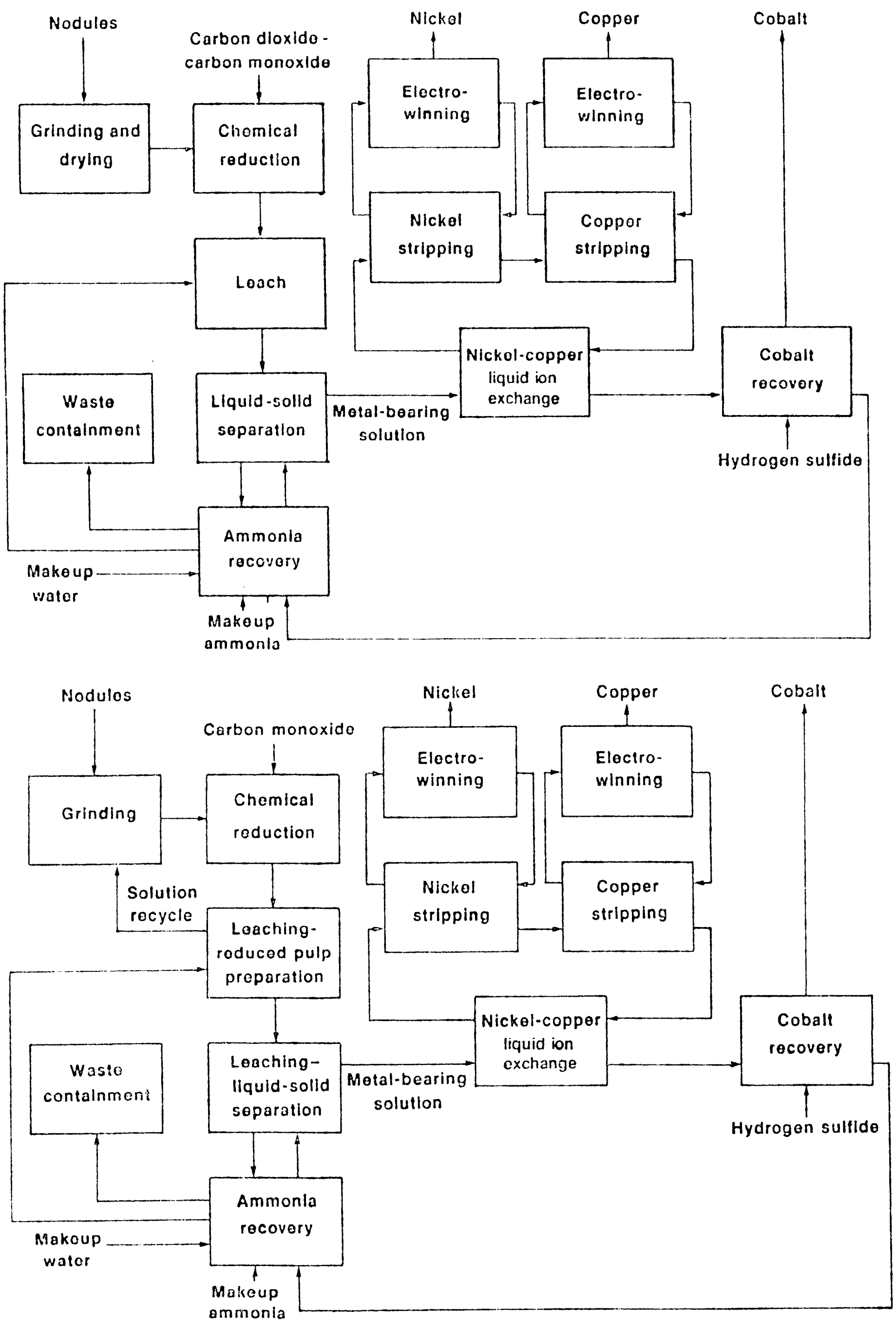
gullies and pillars, which are common in the area. Assessment of the volume dredged is done by measuring the premined water depth and the depth of cut every 30 ft (10 m) and correcting for tidal variation. With the exception of holiday periods, the dredges work a 24-hr day throughout the year with the total downtime of 20% for daily repair and cleaning, holidays, storms, and docking. Recent figures indicate 120 hr/mo for maintenance, and a general overhaul every six to seven years.

Each dredge has a chief dredgemaster and four underdredgemasters. For two or three dredges working in close proximity there is an electrician and a mechanic. The crew consists of 104 men, divided into four watches, each watch or gang consisting of 21 men and a gang of 18 for general daytime work and two men for the motor launch. Each watch is for eight hours, so one watch is free, the gangs changing each week.

Storms are infrequent in Indonesian waters, but when the sea becomes too rough, the dredge is stopped and the ladder raised. If required, wedges can be placed between ladder and hull. If the ladder were to bump against the bottom during a gale, the large ladder bearings near the upper tumbler probably would be destroyed. Connection with the shore by launch is impossible during storms, so the crew has to stay on board to await calmer weather—usually no more than 24 hr.

Dredging costs attributable to salaries and wages are 34% materials including fuel 30%, transportation 11%, repair shop 7%, and miscellaneous 18%.

A yearly throughput of 4 million yd^3 (3 Mm^3) with a tin richness of one (0.1 oz/ft^3 or 1 kg/m^3) would produce 3300 tons (3000 t) of tin in ore. Present-day workings are, in general, not so rich, and values as low as 0.02 oz/ft^3 (0.2 kg/m^3) have been mined. To arrive at the direct revenue for the dredge, the costs of transportation, smelting, and selling of the product have to be deducted. In addition, overhead expenses, such as general office, hospital and so on, and a depreciation rate of 10%, have to be considered, though dredges last much longer than the 10 years originally estimated. Because of this, the fully amortized



a.

b.

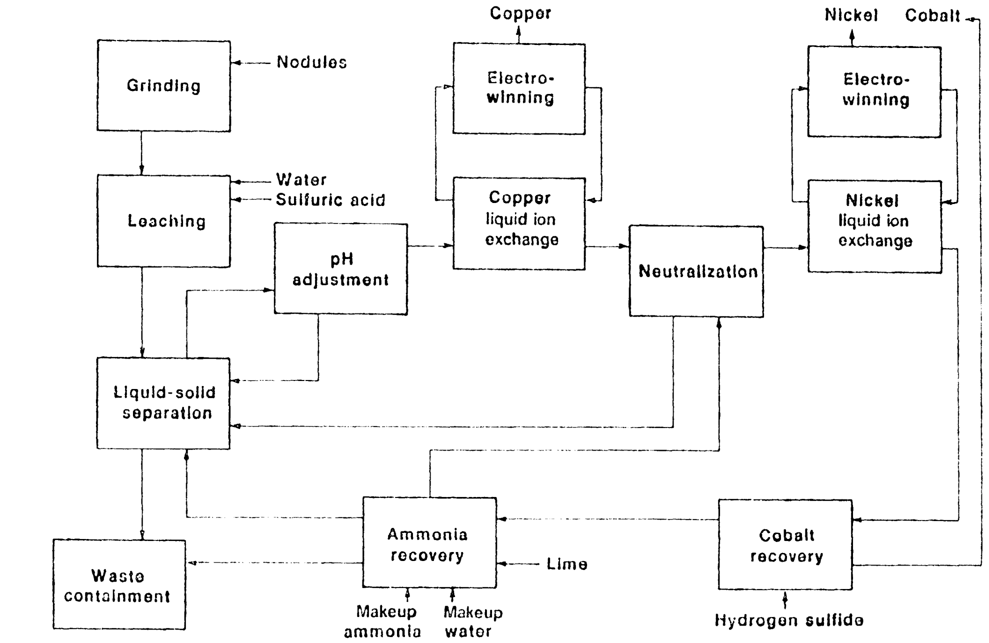
Fig. 22.8.24. Processing of manganiferous oxides.
a. gas reduction and ammoniacal leach.
b. cuprion ammoniacal leach.
c. high temperature/pressure sulfuric acid leach.
d. reduction and hydrocloric acid leach.
e. smelting and sulfuric acid leach.

older dredges have been able to survive some of the later price shocks in the tin industry.

GOLD IN THE UNITED STATES (Anon., 1987). Gold-bearing beach sands were discovered and mined at Nome, AK, in 1906. Mining gradually extended inland from the current shoreline to old shorelines now above sea level. By 1906, about 4.5 million oz (128,000 kg) of alluvial gold had been mined from a 55 mi^2 (142 km^2) area. Early miners recognized that the Nome gold placers were formed by wave action and that additional deposits, formed when sea levels were lower, should be found in the adjacent offshore area (Fig. 22.8.26).

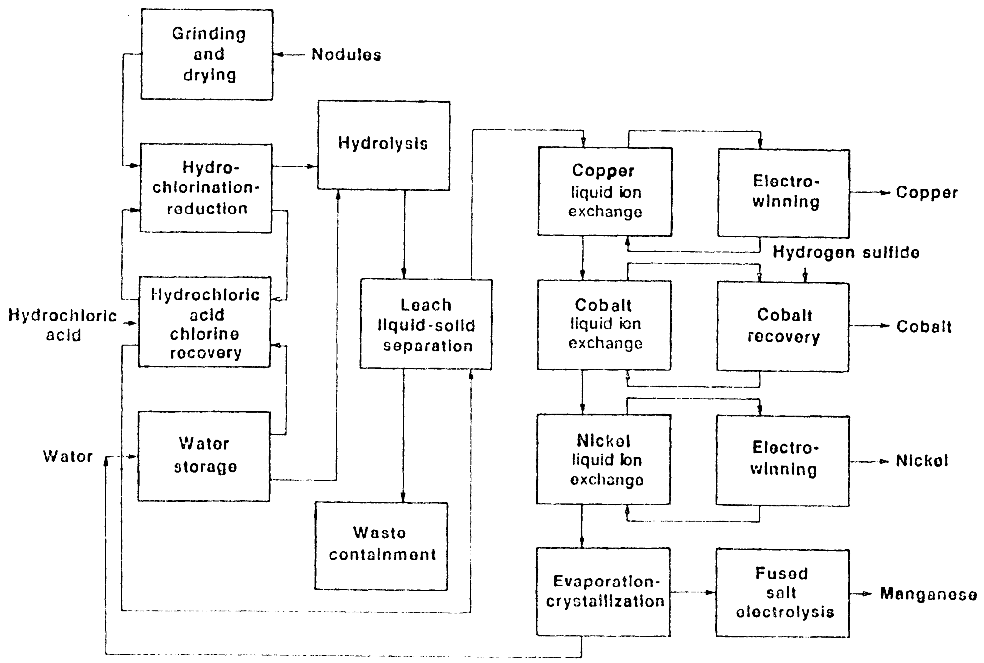
Two US companies, ASARCO and Shell Oil Co., sampled offshore deposits near Nome in 1964 and recovered alluvial gold. By 1969, proven offshore reserves of approximately 100 million yd^3 (76 Mm^3) of ore had been established. The rights to these reserves were acquired in 1985 by Inspiration Resources, which then began a pilot mining and testing program. This program was followed by mining tests with a bucket ladder dredge in 1986. All operations to date have taken place about 1½ mi (2.4 km) offshore in waters under the jurisdiction of the State of Alaska, although gold resources have been identified out to about 10 mi (16 km).

Water depths in the sound do not exceed 100 ft (30 m). Ten miles offshore, water is only 60 ft (18 m) deep. Gold-bearing sediments are a maximum of 30 ft (9 m) thick and consist of bedded sands, gravels, and clays alternating with occasional beds



c.

Fig. 22.8.24. cont.



d.

of cobbles and boulders. These sediments were sampled from the ice out to about 1½ mi (2.4 km) from the coast. Gold has been found further offshore, but reserves have not yet been fully delineated. Current mining sites are located 1½ mi (2.4 km) out in water depths averaging 30 ft (9 m) and in formations 6 to 30 ft (2 to 9 m) thick.

Norton Sound is ice-free and accessible to floating vessels only between June and October. During the winter, thick pack ice forms over the Sound. Waves reportedly do not exceed periods of 7 sec, but occasional sea-swells with longer periods may come from the west and southwest. Predominant winds are from the north and northeast. Currents and longshore drift are westward. Maximum tides are 6 ft (2 m).

Mining Technology—The bucket ladder dredge, Bima, was designed and built in Singapore in 1979 as a seagoing vessel for mining tin offshore Indonesia. The vessel was brought to Nome in July 1986 for preliminary tests, modified in Seattle, and began operation in July 1987. Its hull is 361 ft (110 m) long, 98 ft (30 m) wide, and 21 ft (6 m) deep. The entire vessel is of steel construction and weighs about 15,000 tons (13,500 t), including the dredging ladder and machinery. Freeboard is 10 ft (3 m) and draft 15 ft (4.5 m) with the ladder retracted.

The Bima is not self-propelled. It must be moved to and from the mining site by a tugboat. On site, the dredge is kept in position by five mooring lines attached to 7-ton (6-t) Danforth anchors. This anchoring arrangement allows the dredge to swing

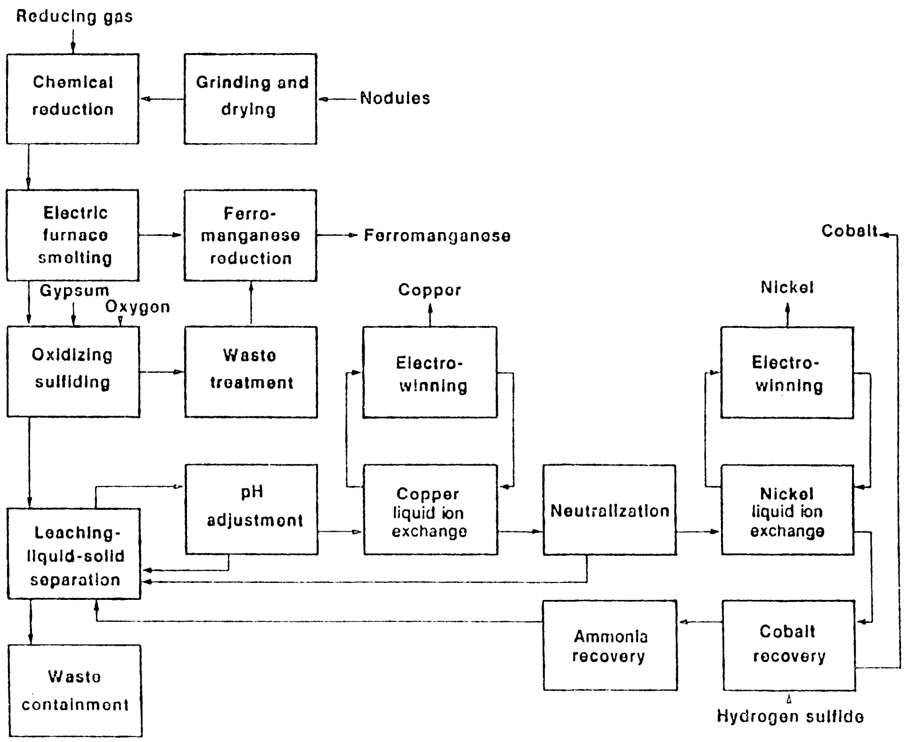


Fig. 22.8.24. cont.

e.

600 ft (180 m) from side to side on the headline and to advance while digging. The anchors are positioned and moved by a special auxiliary vessel.

A 15,000-hp (11,000-kW) diesel-electric power plant is used to operate the bucketline, the ore processing plant, the anchor winches, and the auxiliary systems. There is fuel storage on board for 2½ months of operation.

The Bima's dredge ladder and bucketline were originally designed to operate in 150 ft (45 m) of water. The dredge is able to mine from 25 to 100 ft (8 to 30 m) below the water line at the rate of 33 yd^3/min (25 m^3/min) or approximately 2000 yd^3/hr (1500 m^3/h). The Bima was designed to enable the mass of the ladder and bucketline to be decoupled from the motions of the hull by an automated system of hydraulic and air cylinders that act like very large springs. This feature keeps the buckets digging against the dredging face on the seabed while the hull may be heaving or pitching due to the motions of passing waves. During the trials of the Bima in Norton Sound from July to October 1986, it was not necessary to activate the system.

At-sea Processing—The Bima is equipped with a gravity processing plant to make a gold concentrate at the mining site. The throughput capacity of the plant is 2000 yd^3/hr (1500 m^3/h). The plant consists of two parallel inclined rotary trommels, 18 ft (5.4 m) in diameter and 60 ft (18 m) long. After removal of any large boulders, ore brought up by the dredge bucket slides down the trommels under the spray of powerful jets of seawater. The water jets are used to break up the clay and force sand and gravel smaller than 3/8 in. (9.5 mm) to pass through the trommel. Material coarser than 3/8 in. (9.5 mm) is discharged over the stern.

Material retained by the trommel is distributed in a seawater mixture to three circuits of jigs, beginning with six primary circular jigs, each 24 ft (7.2 m) in diameter. The concentrates from the circular jigs are then fed to cross-flow secondary and tertiary jigs. The jig concentrates are further refined on shaking tables before transport to shore for final gold separation and smelting into bullion. It is expected that about 22 lb (10 kg) of concentrate containing the gold will be produced by mining and processing approximately 50,000 tpd (45,000 t/d) of ore. The actual amounts of concentrate produced will depend on the quantity of heavy minerals associated with the gold at each location.

Environmental Effects—The ore processing plant on the Bima returns over 99.9% of the processed material to the seabed as tailings. Since the tailings do not undergo chemical treatment, local turbidity caused by particles that may remain in suspension is likely to be the most significant environmental impact. During pilot plant tests in 1985, Inspiration Resources found that turbidity could be minimized by discharging fine tailings through a flexible pipe near the seabed. Other potential environmental impacts could occur if diesel fuel is spilled, either as it is being transferred to the Bima or as a result of accidental piercing of the hull.

Operating Conditions—The Bima operates only between June and October (five months per year) because ice on Norton Sound prohibits operations during the winter months. Thus, without breakdowns or downtime due to weather and other causes, a theoretical yearly production of about 7.5 million yd^3 (5.7 M m^3) of ore is possible. During tests in 1986, Bima operated only a small fraction of the time available. This was due more to the nature of the trials than to downtime related to winds and waves. Assuming a mining efficiency (bucket filling) of 75% and operating efficiency of 80% (allowing for time to move and downtime due to weather), yearly production is limited to 4.5 million yd^3 (3.4 M m^3). If gold grades of 0.012 to 0.016 oz/yd^3 (488 to 650 mg/m^3) of ore are assumed, the yearly gold production would vary between 1.75 and 2.20 tons (1.6 t to 2 t) before any losses due to processing and refining.

The Bima has a crew averaging 12 persons per watch, three watches per day. Personnel are transported to and from Nome daily by helicopter. The operation also requires extensive maintenance, supply, and administrative facilities onshore. These facilities are manned by an additional 46 persons during the operating

Table 22.8.15. Data on Selected Marine Mining Operations

Operating Company	Name of Dredge	Type	Hull	Total Installed HP	Tonnage	Depth Range, ft	Mooring	Rated Capacity
US Army Corp of Engineers	Varina	Hydraulic dredges 8–20 in. (20)	Ship, 177–525 ft 500–8,000 yd^3 cap.	pump, 175–1,850; ship, 800–8,000	NK	19–41	Free	Variable
Hall & Co., Ltd.	AA Raymond	28-in. hydraulic suction	270 × 46 × 23 hopper dredge	3,000	5,900	50	Free	15,000 tpd
Iceland Gov't Cement Works	Sansu	24-in. hydraulic suction	Ship, 150 ft	1,000	1,100	140	Free	9,600 tpd
Aokam Tin Mining Ltd.	Dredge No. 3	15-cu ft bucket-ladder	Barge, 270 × 80 × 12	2,154	3,975	140	6-anchor	420,000 cu yd/mo
Indonesian State Tin Mining Enterprises	"Bangka 1"	18-cu ft bucket-ladder	Barge, 300 × 80 × 16	4 × 900 diesel	NK	135	6-anchor	420,000 cu yd/mo
Marine Diamond Corp.	Pomona	5 × 18-in. suction	Barge, 285 × 60	2,100	4,800	<150	Anchor	15,000 tpd
Marine Diamond Corp.	Colpontoon	$\frac{1}{16}$- & $\frac{1}{18}$ in. flexible suction line with water jet	Barge, 260 × 45 × 13	1,365 total; 765 pump		120	6-anchor	24,000 tpd, 15,000 gpm
Terra Marina	Ontginner 1	12-in. air-jet lift & water jet cutter	Tanker, 180 × 49 × 15	2,040	981	<150	4-anchor	2,600 tpd
Matachewan Canadian Gold Ltd	—	Suction	Ship	NK	NK	<60	NK	150 cu yd/hr
Westgold Ltd.	Bima	30 cu ft bucket ladder	Barge 361 × 98 × 21	15,000	15,000	<150	5-anchor	2000 cu yd/hr
Deep Sea Beds								
Deepsea Ventures Inc.	R/V Deepsea Miner	10-in. airlift	Converted CI-M-AVI cargo vessel 320 × 50 × 18	2,400	7,500	3,000	Dynamic	500 tpd
Japan Resources Assoc.	Chiyoda Maru No. 2	Continuous bucket-line	Ship conversion, 73 × 12 × 6	Dredge, 150	1,391 gross	2,000 m	Dynamic	NK
Ocean Mining Association (OMA)	DeepSea Miner II	13 in. airlift	Bulk carrier conversion 855 × 106 × 63	28,000	20,000 dwt	5,000 m	Dynamic	70 tph
Preussag AG	Sedco 445	5-in. 6 stage centrifugal slurry pump	Drill ship	NK	NK	2,500 m	Dynamic	33–70 m^3/hr
Ocean Management Inc. (OMI)	Sedco 445	Pump Airlift	Drill ship	NK	NK	5,000 m	Dynamic	NK
Ocean Mining Co. (OMCO)	Glomar Explorer	Suction	Mining test ship	NK		5,000 m	Dynamic	NK
Afernod (France)	CLB	Continuous bucket line	2 ship test	NK	NK	NK	Dynamic	NK

Conversion factors: 1 in. = 25.4 min, 1 ft = 0.3048 m, 1 yd^3 = 0.7646 m^3, 1 hp = 0.7457 kW, 1 ton = 0.9072 t.

season. During the winter months, the Bima is brought in to the Nome breakwater and protected by an ice caisson sprayed into place around it.

Capital and Operating Costs—Capital and operating cost estimates are based on a number of assumptions and must be considered first-order approximations. The estimates rely in part on published information that the Bima gold mining project will have a life of 16 years and will recover about 48,000 oz/yr (1500 kg/yr) of gold at operating costs of less that $200/oz ($6.43/g).

The Bima was constructed at a cost of $33 million in 1979. It is assumed that its purchase in 1986 as used equipment (sold because of the fall in the price of tin) is on the order of $5 million. Also assumed is that other capital costs, including ancillary facilities onshore, pilot-plant mining tests in 1985 and trials in

Table 22.8.15. Data on Selected Marine Mining Operations (cont.)

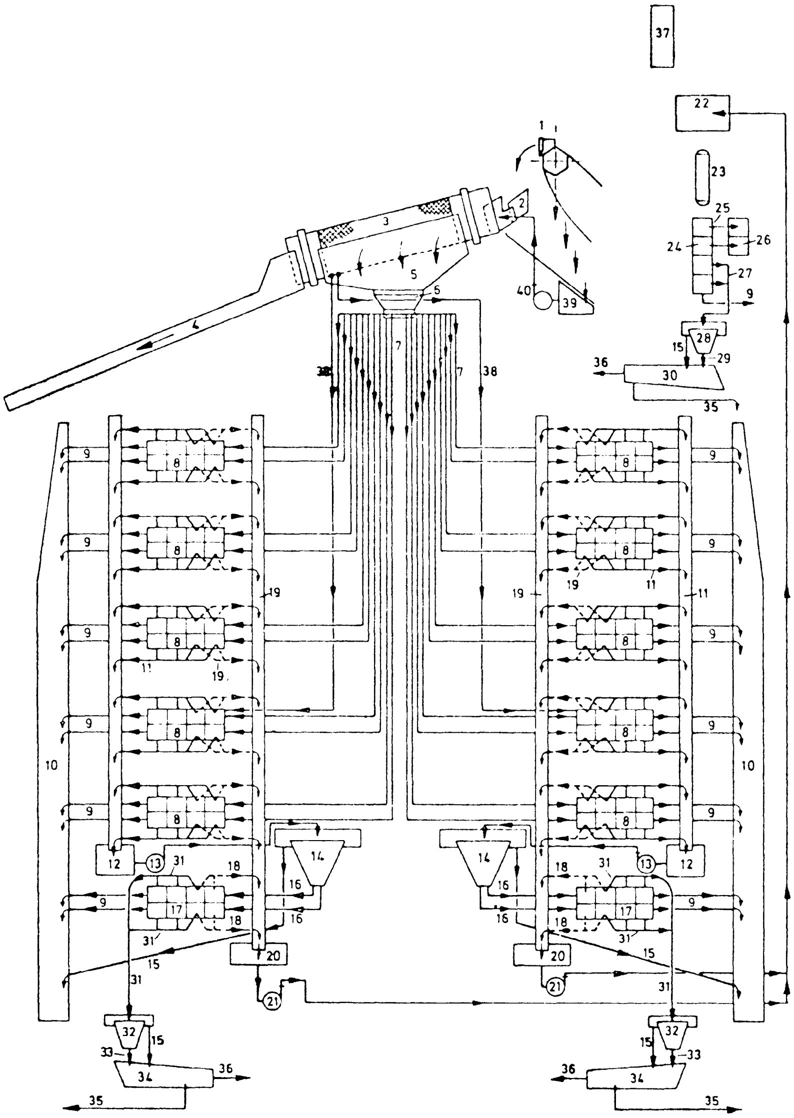
Capital Cost, $	Period of Data	Location	Operating Depth, ft	Deposit Type	Monthly Throughout, yd^3	Cost per yd^3, ¢	Deposit Value, $ per yd^3	Remarks & References
465,000–5,029,000	1949	Coastal U.S.	0–40	Sands & clays	74,000–741,000	6.9–36.5	NK	Scheffauer, 59
NK	1967	North Sea	NK	Gravel	60,000	NK	NK	M&ME, 39
NK	1964	Akranes Bay, Iceland	130	Shell sand	10,000 mt	NK	NK	Vestdal, 68
NK	1969	Phuket, Thailand	140	Tin alluvials	217,000	NK	1.94 lb per yd^3	MMTC, 1970
8,000,000	1966	Bangka	82	Tin alluvials	NK	NK	0.74 lb per yd^3	*Mining Journal*
2,100,000	7/64–6/65 1/68–6/68 1969	S.-W. Africa	< 100	Diamond gravel	18,450 8,940; 13,200	$41.6 –36.2 $6	39.6–33.2	All dredges *World Mining,* Oct. 1968
NK	1965	S.-W. Africa	55	Diamond gravel	84,000	NK	63*	*Screened gravel Nesbitt, 1966
1,400,000 + 500,000 refit	1960	S.-W. Africa	< 100	Diamond gravel	42,000	2.30*	Extremely variable	*estimate from $120,000/ m per mo; Hollandship, 1966; MMTC 1970
NK	1969	Nova Scotia	60	Gold placer	—	—	0.30 reported	Experimental, no record of success; Ubby, 1969
33 million (1979)	1986–89	Nome, Alaska	25–100	Gold placer	1.4 million	155	NK	Converted from Indonesian tin dredge, 1986
2,000,000	1970	Blake Plateau	2,600	Manganese nodules	Exceeded rate	NK	NA	Prototype tested, 1970 Kaufman, 1970
250,000	1970	S. Pacific	3,760 m	Manganese nodules	NA	NA	NA	Prototype tested, 1970; MMTC
NK	1977–79	N.E. Pacific	4,800 m	Manganese nodules	Test	NA	NA	¼–⅕ Commercial scale test
20 million (reported)	1979	Red Sea	2200 m	Sulfide muds				
137 million (for mining system)	April 1978	N. Pacific	16,500 ft	Manganese nodules				⅕ scale model
NK	1978–79	N.E. Pacific		Manganese nodules	Test	NA	NA	
NK	1975–90	Pacific	NK	Manganese nodules	NK	NA	NA	

Conversion factors: 1 in. = 25.4 min, 1 ft = 0.3048 m, 1 yd^3 = 0.7646 m^3, 1 hp = 0.7457 kW, 1 ton = 0.9072 t.

1986, auxiliary vessels for prospecting and for tending anchors, shipyard modifications and alterations to the processing plant, and the cost of shipment of the Bima from Indonesia to Nome and to and from the shipyard near Seattle amount to another $10 to $15 million. Total capital costs are thus assumed to be between $15 and $20 million.

Annual operating costs for fuel, maintenance, insurance and administration, and personnel and overhead are estimated (to an accuracy of 25%) to be $7 million. At a production rate of 48,000 oz/yr (1500 kg/yr), a cash operating cost on the order of $150 to $175/oz ($4.82 to $5.63/g) is implied. At a mining rate of approximately 4.5 million yd^3/yr (3.4 M m^3/yr), direct costs would amount to $1.55/$yd^3$ ($2.00/$m^3$).

Assuming the price of gold to be $400/oz ($12.86/g), the projected pretax cash flow on production of 48,000 oz/yr (1500 kg/yr), would be approximately $12 million (after subtracting operating costs) on an investment of $17 million. Although this figure does not include debt service, it nevertheless indicates that



1, Bucket; 2, chute; 3, revolving screen; 4, chute for oversize material; 5, revolving screen tank; 6, distributor; 7, pipes to jigs; 8, primary jigs; 9, launders from jigs; 10, tailing launders; 11, middling launders with settling tanks; 12, collecting tanks middlings; 13, middlings pumps (12–14); 14, clean-up thickeners; 15, discharge thickener overflow water; 16, middlings to clean-up jigs; 17, clean-up jigs; 18, discharge half concentrate clean-up jigs; 19, discharge half concentrate primary jigs; 20, half concentrate receivers; 21, half concentrate pumps (20–22); 22, dewatering tank; 23, bucket elevator (half concentrate from 22 to 24); 24, fine-wash jig; 25, discharge concentrate fine-wash jig; 26, concentrate receiver (two parts); 27, discharge half concentrate fine-wash jig; 28, small thickener under fine-wash jig; 29, half concentrate to shaking table; 30, shaking table; 31, discharge half concentrate clean-up jig; 32, thickeners above shaking tables; 33, half concentrate to shaking tables; 34, shaking tables; 35, discharge tailings from shaking tables; 36, discharge concentrate from shaking tables; 37, concentrate box; 38, overflow (from 5 to 8); 39, save-all; 40, save-all pump (from 39 to 2)

Fig. 22.8.25. Typical flowsheet for Indonesian tin dredge.

the Bima offshore gold mining project at Nome shows good promise of profitability if the operators are able to maintain production. This scenario illustrates that offshore gold mining is economically viable and technically feasible using a bucketline dredge under the conditions assumed.

DIAMONDS IN SOUTH WEST AFRICA (Webb, 1965; Nesbitt, 1967). Original concessions were granted to Marine Diamond Corp. by the South West African government from the Orange River to Deay Point, 176 mi (280 km) north, reaching from low-water mark to 3 nautical miles (5.5 km) seaward. The sea diamonds originated from kimberlite bodies within the drainage basin of the Orange River. They were transported by the river and deposited in economic concentration up and down the coast by the action of the sea. Diamondiferous gravels generally occur

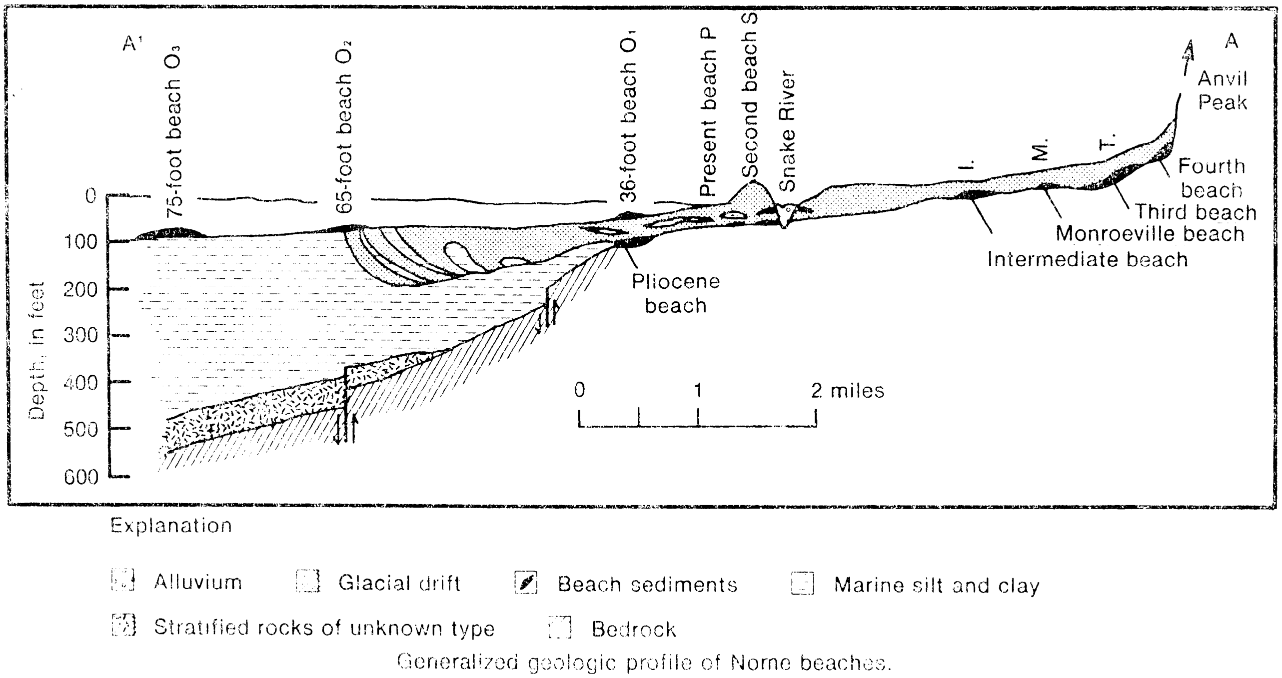


Fig. 22.8.26. Generalized profile of gold-bearing beaches at Nome (after Cobb 1973). Conversion factors: 1 ft = 0.3048 m, 1 mi = 1.609 km.

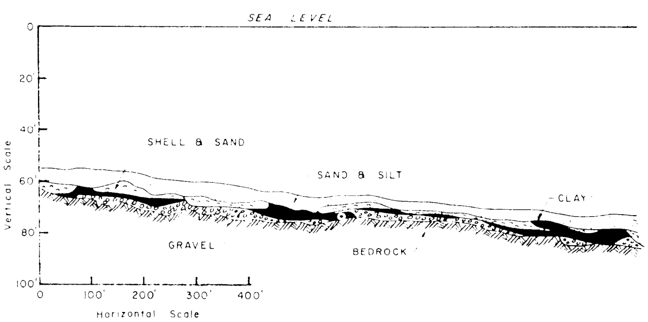


Fig. 22.8.27. Generalized profile of diamond-bearing beaches at Chameis Bay (after Nesbitt, 1967). Conversion factor: 1 ft = 0.3048 m.

immediately overlying the bedrock or in gullies or potholes (Fig. 22.8.27). Webb and Nesbitt described the sea mining operations and commensurate problems in some detail.

Present mining is conducted immediately to seaward of the breaker line. In this area, the waves are starting to form, thus increasing the swell frequency. From Aug. 1965 to May 1966, swells exceeded 5 ft (1.5 m) during 61% of the period, and exceeded 8 ft (2.4 m) during 41% of the period. During Mar. 1966, the highest mean daily significant height (MDSH) was 29 ft (9 m), and it was over 15 ft (5 m) during five days. To counter these difficult conditions, the mining barge Colpontoon is held by six anchors, two of these being classified as deep-sea anchors, which always are positioned seaward and to the stern of the barge. These deep-sea anchors serve to pull the barge away from the shore when the weather deteriorates. The other four are shallow maneuvering anchors. The anchor winch is situated mid-barge and the cables are taken along the decks through sheaves and fairleads out to the anchors.

The anchor spread embraces an area 3300 by 3300 ft (1000 by 1000 m), which lasts between one and two months. Positioning is by hydrodist .

On the Colpontoon, the flexible suction hoses are 12, 16, or 18 in. (304, 406, or 457 mm) in diameter. A smaller hose carries high-pressure water used to jet and disturb the sediments.

Using an airlift, the technique is to blow for 25 sec, then shut off and fill a receiver for 15 sec. One reason for doing this is to clear boulders away from the nozzle of the lift, the boulders being cleared when the load in the pipe drops away. Another reason is the inability of the airlift to raise the load when it becomes too dense.

Dredge pumps later replaced the airlifts with an 80% increase in volumetric efficiency.

The present method of spot dredging with flexible rubber hoses is the only one that has proved practical so far, but it suffers grave disadvantages, especially in clay or boulder horizons, as generally, much of the deposit is left behind, and little penetration is made to bedrock.

Powered by a 765-hp (570-kW) diesel, the 18-in. (457-mm) dredge pump can deliver as much as 13,000 gpm (820 L/s). Unfortunately, with the present unguided flexible hose, most of this 13,000 gpm (820 L/s) is water, not sediment. Data on the operation in Table 22.8.16 include some performance figures for the dredge Colpontoon.

In the treatment plant, dredge-pump delivery is piped to an 84-in. (2.1-m) diameter dewatering cyclone, designed to accept 13,000 gpm of pulp carrying 660 tph (600 t/h) of solids ranging in size from a maximum of 9 in. (228 mm) to micrometer slimes.

Water rejection from the cyclone averages 90%, while all material coarser than 0.04 in. (1 mm) reports as concentrate. A very low cyclone inlet pressure of 3 to 5 psi (21 to 34 kPa) is essential to minimize wear. A prototype cyclone constructed from ¾-in. (19-mm)-thick steel wore out in 24 hr; 3/8-in. (9.5-mm) Bennox lasted a week. An 8-in. (203-mm)-diameter cyclone was later cast from 1-in. (25-mm) Scaw metal.

Table 22.8.16. Typical Monthly Performance Figures for the Dredge Colpontoon

Average solid sediment dredged, tph	133
Operating time, hr	633
Lost time, hr	33
Total feed to heavy-media cyclone (−25 + 1.5 mm), tons	8,400
Average feed to heavy-media cyclone (−25 + 1.5 mm), tph	13.3
Highest recorded feed to heavy-media cyclone (−25 + 1.5 mm), tph	30
Lowest recorded feed to heavy-media cyclone (−25 + 1.5 mm), tph	1.5
Total carats produced	7,700
Total area worked, sq m	5,000
Maximum estimated wind force, mph	45
Maximum estimated swell height, ft	21

Conversion factors: 1 in. = 25.4 mm, 1 ft^2 = 0.0929 m^2, 1 ft = 0.3048 in., 1 mi = 1.6093 km, 1 ton = 0.9072 t.
Source: Nesbitt, 1967.

Large and very small diamonds are uncommon. Accordingly, the apertures of the primarily scalping and washing screens are, respectively, 1 in. (25 mm) and 0.06 in. (1.5 mm), and any solids outside this range are rejected immediately as waste. In most areas, the bulk of the plant feed to the heavy-media cyclone unit comprises seashells. At other times, a tenacious, sticky green clay constitutes the major component.

Together with a dense slurry of powdered ferrosilicon and water, the – 1-in. (25-mm) + 0.06-in. (1.5-mm) sediment is pumped into the heavy-media cyclone where the denser mineral components, including the diamond, are centrifuged and then escape through the cyclone spigot. Over 90% of the feed has a density less than diamond and can be rejected as waste from the cyclone overflow. Owing to the flat shape of seashells, shell rejection from the conventional design of cyclone is inefficient. This was corrected by redesigning the cyclone with a 60° frustum in place of the conventional 20°.

Heavy-media cyclone concentrate either is fed to a ball mill or treated directly by Pleitz jigs, the concentrates from which are finally sorted by hand for lack of a better method.

Although very efficient in diamond recovery, the heavy-media cyclone process has proved costly not only because of expensive capital equipment but also high operating costs. An alternative primary gravity concentrating system used on the research vessel, Barge 177, is a 5-ft (1.5-m) circular Cleaveland jig, which also is very suitable for the removal of seashells.

A typical processing flowsheet is given in Fig. 22.8.28, and Table 22.8.17 lists the equipment used on the Colpontoon.

Mining of the foreshore between high and low tide provides most of the production during the development of improved offshore techniques. Borchers, et al. (1970) describe in detail the construction of seawalls and the various mining methods.

22.8.3.2 Dredging from Deep Seabeds

Several hundred millions of dollars were spent in the 20 years since 1968 on research and development on deep-seabed mining systems. No full-scale systems have yet been built, but pilot tests have been run on several systems for the mining of manganese nodules at 17,000 ft (5000 m), a system for the collection of a bulk sample of metalliferous muds at 7300 ft (2200 m), and conceptual systems for cobalt crusts at about 3300 ft (1000 m). Some of these are described now in more detail.

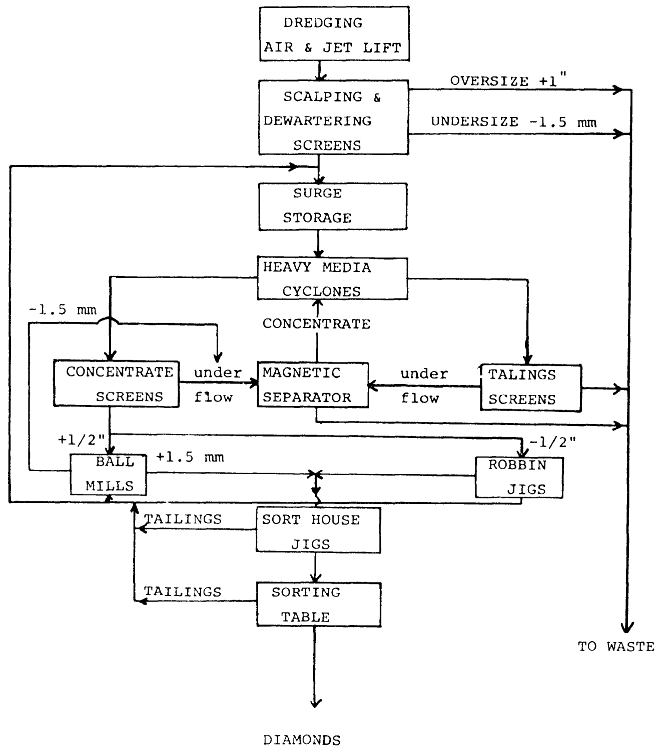


Fig. 22.8.28. Typical processing circuit for marine diamonds (Nesbitt, 1967). Conversion factor: 1 in. = 25.4 mm.

Table 22.8.17. Listing of Beneficiation Equipment Used on the Diamond Dredge Colpontoon

Unit	Dimensions
Dewatering cyclones	84 in.
Scalping screens	25-mm top deck, 1½-mm bottom deck, 6 × 16 ft
Washing screens	1½-mm deck, 6 × 16 ft
Bin capacity	40 tons
Heavy-media preparation sumps	—
Cyclone pumps	8 @ 6 in.
Heavy-media cyclones	20 in. × 60°
Float screens	6 × 16 ft
Sink screens	3 × 12 ft
Magnetic separators	36 × 72 in. 36 × 48 in.
Ball mills	6 × 8 ft 8 × 10 ft
Bobbin Pleitz jigs	—
Blind Pleitz jigs	—

Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 ton = 0.9072 t.
Source: Nesbitt, 1967.

R/V Deep Sea Miner II (Kaufman and Latimer, 1971; Kaufman, Latimer, and Tolefson, 1985). The OMA Deepsea Miner II tests spanned a two-year period (1977-1978) and included four major sea test campaigns. The tests were conducted at a test site within the OMA license exploration area, which is located about 1100 nautical mi (2037 km) southwest of San Diego, CA. Fig. 22.8.29 shows the location of the test site as well as boundaries of the OMA, OMI, and OMCO license areas that have been

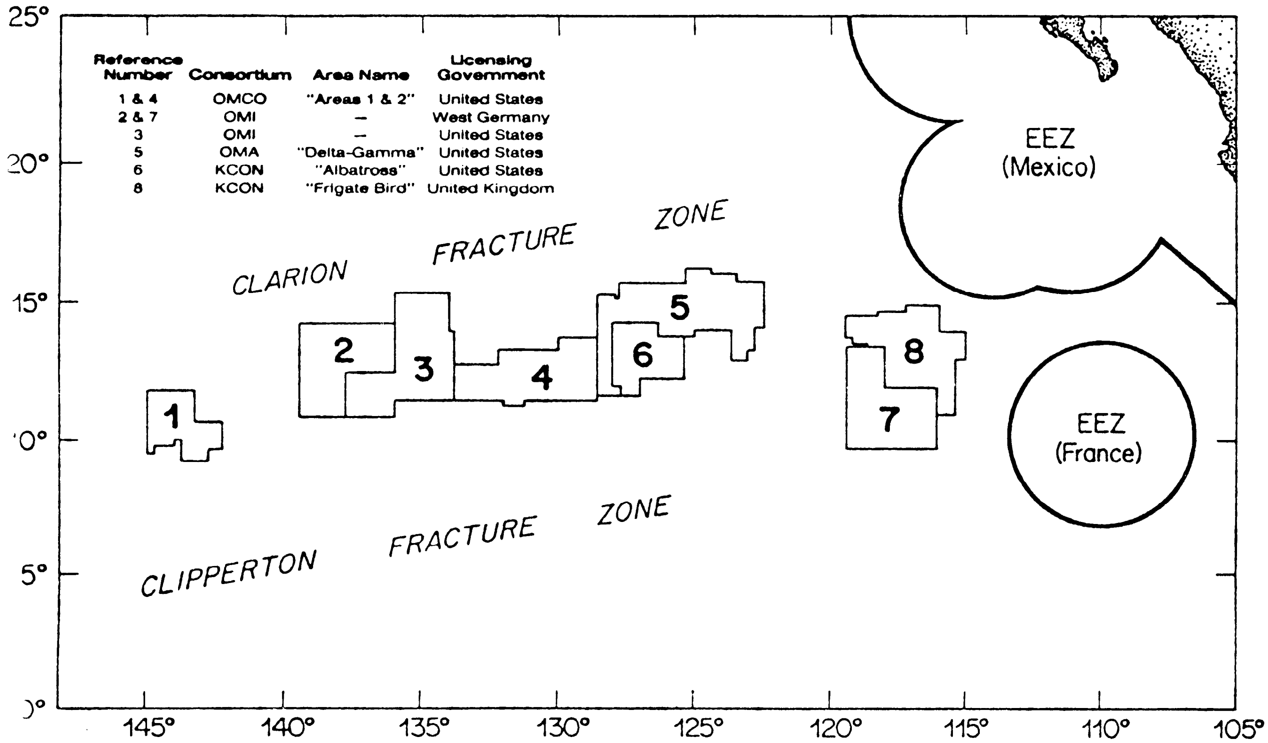


Fig. 22.8.29. Site location of manganese nodule tests.

issued by the US government pursuant to the Deep Seabed Hard Minerals Resources Act.

The mining system was divided into five principal subsystems:

1. Collector.
2. Mining Production Control/Integrated Navigation/Instrumentation.
3. Nodule Conduit/Nodule Lift.
4. Mining Platform including pipe handling and suspension and pipe storage.
5. Ore Transport and Transfer at Sea.

All subsystems were incorporated into the R/V DEEPSEA MINER II pilot tests except ore transport and transfer at sea.

Initial evaluation included study of the five fundamental engineering approaches: 1) the continuous line bucket, 2) the autonomous shuttle, 3) wireline basket dredging, 4) containers in a pipe, and 5) hydraulic dredging. The continuous hydraulic dredging approach was determined to be the most effective system for efficient first generation commercial mining.

Some of the major features of the continuous hydraulic test mining system are

1. High-strength steel dredge pipe (conduit) with hinged splitter plates for vortex suppression towed through water.
2. Pipe suspended at all times by a gimballed holding device located in a midship internal well (moonpool).
3. Gimbal and associated derrick structure passively aligned with the upper portion of pipe by means of rollers or a similar device installed on the lower portion of the derrick that projects below the gimbal.
4. Hydraulic main hoist cylinders mounted on the gimballed suspension device, in such a manner that the hoisting force is always approximately coincident with the axis of the upper portion of the pipe.
5. Airlift pump system.
6. Deadweight at the lower end of pipe to reduce curvature and bending stress from collector towing force.
7. A flexible hose connecting the lower end of pipe via a universal joint to the collector to accommodate variations in seabed relief.
8. A nodule collector or dredge at the end of the hose that is towed at the average speed of ship. The collector removes nodules from the seabed by hydraulic sluicing, but provides essentially clear, sediment-free water to the hose suction.

Some of the many alternative design features that were evaluated, or developed and evaluated, in arriving at the system configuration are provided on Table 22.8.18. It should be noted that self-propulsion of the collector was not a test or a commercial requirement. Steering of the collector is a commercial requirement but was not provided on the test.

The R/V Deepsea Miner II was approximately a one-fifth to one-quarter production-scale version of the commercial system. The ground rules for the design of the test apparatus did not require modeling of all components of the commercial system, but only the major ones and other devices that could not be analytically sized and designed with a high degree of confidence. Notwithstanding such latitude, the end product was very similar to a commercial design. The major areas of difference were the life span of some components as effected by corrosion, omission of some aspects of automation that would be provided on the commercial ship to reduce manning, reduction of pipestring deployment and retrieval time, and capability to operate and survive under all weather conditions.

Glomar Explorer Test (Welling, 1981). The basic limitation in seabed mine production is caused by nodule abundance. At an average nodule abundance of 2.04 lb/ft^2 (10 kg/m^2) and a miner speed of 3.2 fps (1 m/s), one can see that a 33-ft (10-m)-wide miner would deliver 220 lb/min (100 kg/s), 8640 tpd (8784 t/d) or 3,153,600 tpy (3,206,308 t/a) if 100% efficient. Of

Table 22.8.18. Design Options Evaluated by Deep Sea Ventures Inc. for Nodule Mining Test

ALTERNATIVE DESIGN FEATURES EVALUATED	
Pipe material:	Steel of various types.
	Aluminum.
	FRP.
	Composite.
Pipe structure (body):	Stepped in steel area.
	Stepped in steel area and flow area (airlift only).
Pipe joint installation:	Welded to upset pipe.
	Welded to plain end pipe.
	Threaded attachment.
Type of joint make-up:	Threaded (tool joint).
	Clamp ring.
	Others.
Main hoist force device:	Wireline.
	Hydraulic rams.
	Hydraulic cylinders.
	Load screw devices with recirculating ball bearings.
Main hoist sequence:	Hand over hand.
	Stop and go.
Main hoist yoke location:	Above gimbal.
	Below gimbal.
Gimballed derrick:	Passive alignment with pipe.
	Active (power driven alignment with pipe).
Pipe transfer:	Horizontal attitude.
	Vertical attitude.
Pipe stowage:	Above deck.
	Below deck.
Dredging methods:	Continuous slow traverse with collector following at same average speed.
	Stop and go with collector running a pattern on bottom.
	Continuous slow traverse with collector running pattern on bottom.
Collector nodule pickup:	Pick up of nodules and sediment with subsequent jetting to separate sediment.
	Pick up of nodules without sediment.
Nodule pickup mechanisms:	Scraper.
	Fixed tines or rakes.
	Rotary tines.
	Hydraulic sluicing (suction).
	Magnetic.
Nodule transport (collector):	In-line pump; and ductwork.
	Jet pump; and ductwork.
	Conveyors (includes mechanical belts).
	Hoppers/Feeders.
	Crushing/No crushing on bottom.
Collector steering:	Fins.
	Thrusters.
	Rudders.
Collector ground support:	Runners or skids.
	Track laying devices.
	Large fluid inflated rollers.
	Others.
Main pumping system:	Airlift.
	Submerged electrically driven centrifugal or multi-stage pumps along the line.
	Large multi-stage centrifugal pump at bottom.
	Reciprocating devices.
Airlift features:	Depth of injection.
	Pressurized or atmospheric discharge.
	Extraction and recompression.
Motor-pump features:	Pass nodules through pump.
	Nodules injected downstream by alternating holding chamber or rotary feeders.
	Others.
Mining platform:	Semisubmersible platforms.
	Swath.
	Catamaran.
	Flip ship.
	Conventional ship form.

Source: Kaufman and Latimer, 1971.

course, there are many things wrong with these simple figures: abundance is given in wet tons, production is given in dry tons, and wet tons contain 30% less metal than do dry tons. Fig. 22.8.30 illustrates the many considerations that contribute to production efficiency. The two factors considered are rate on the vertical axis and time on the horizontal axis. Rate times time describes the production or the area of the large rectangle. The losses in rate are pickup efficiency, fines losses, collector-down time, and interaction of the various systems. This can amount to a loss of one-third to one-half of the rate. The losses in time are due to such things as the mining ship in drydock, weather delays, breakdowns and repairs, logistic delays, and losses during turns. This can amount to one-quarter to one-third of the time.

Seafloor Characteristics—Knowledge of the terrain and nodule distribution of a mine site can only be known on a statistical basis. The three important factors—nodule grade, abundance, and terrain characteristics—determine the value and economics of a mine site. While preliminary investigations indicate some relationships between these three factors, there appear to be enough variations that one should strive for a mining system with wide operational characteristics.

Deep-Ocean Miner Characteristics—To obtain maximum efficiency, one of the functions considered essential was good mobility. This means control of speed and direction with minimum turning radius and maximum trafficability over varying soil properties.

Good miner mobility requires a flexible link between the miner and the end of the heavy pipe suspended from the surface ship, approximately 17,000 ft (5000 m) above. Because of the mass and the drag of the pipe and ship, the end of the pipe can appear quite rigid to the miner, and therefore, the pipe cannot act as a flexible link.

Another requirement was buffer storage of the nodules prior to pumping of the nodules up the pipe. Slurry systems require steady state conditions for maximum efficiency. Buffer storage in conjunction with miner speed variations will allow a greater variation in nodule abundance without loss of mining efficiency.

Additional requirements set forth were (1) raising and lowering of the mining head to allow for control of depth of cut and retraction for passage over small obstacles, (2) washing of the nodules to minimize the amount of clay pumped to the surface, (3) crushing of the nodules to establish an upper maximum size for slurry transport, (4) high-resolution sonar and TV systems for observation of the surrounding terrain, and (5) the necessary command and control system for real time data acquisition and control functions.

Instrumentation and Sea Tests—Extensive instrumentation and data recording were employed. Furthermore, every future large-scale operational system was incorporated into the test miner. The primary power was obtained by use of a high-voltage electrical cable strapped to the pipe and terminated in a transformer located in the equipment section on the bottom of the buffer. Electrical power of reduced voltage was then supplied from the transformer to a large electrical motor that drove a hydraulic pump, all located in the equipment section. The hydraulic power was in turn supplied to the miner by hydraulic lines forming a part of the umbilical lines. A series of electrically controlled valves on the miner metered the hydraulic power to the various motors on the miner. Fig. 22.8.31 is a photograph of the test miner as it was stowed in the large center well of the Glomar Explorer. The test miner's physical characteristic is dominated by (1) the Archimedes screw propulsion system, (2) the television and light boom extending forward, (3) the umbilical lines extending upward back and down to the bottom of the buffer, and (4) the buffer and equipment section located inside the two horizontal rings.

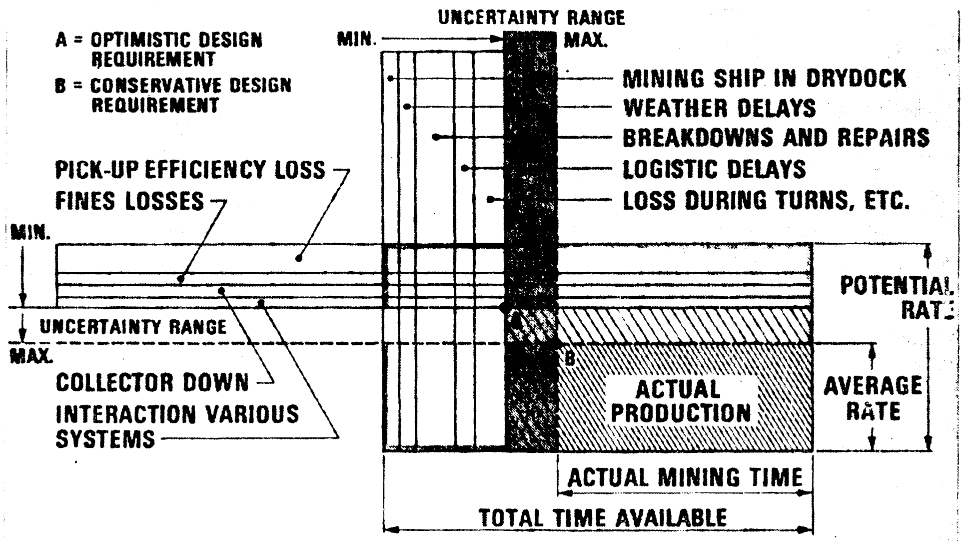


Fig. 22.8.30. Production efficiency for deep seabed mining system (after Welling, 1981).

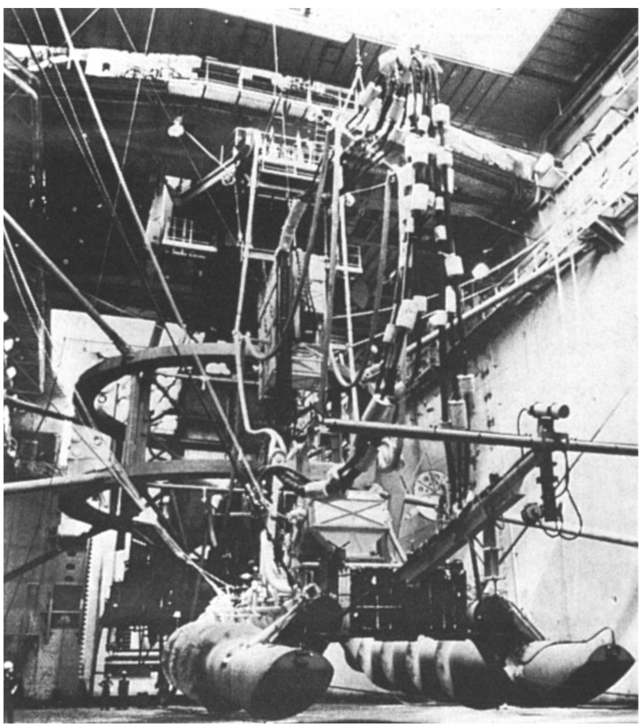


Fig. 22.8.31. Photograph of Lockheed test miner (*courtesy:* C.G. Welling).

The handling technology used for launching and retrieving the miner and buffer is unique. After the well is flooded, and the large doors are open, the miner is lowered until it hangs by the umbilical cord approximately 160 ft (50 m) below the keel. The whole system is then lowered to the ocean floor by makeup of 60-ft (18-m) sections of pipe. The launching and retrieval of the units weighing a total of several hundred tons was accomplished without major incident, even in the presence of sea surges of 20 ft (6 m) or more in the well.

Bottom Navigation Control—The principal information sent to the control room was obtained by use of sonar and television. Mounted on the buffer equipment section, which was approximately 60 ft (18 m) above the ocean floor and 60 ft (18 m) ahead of the miner, was a high-resolution scanning sonar. Also sonar systems are used for range and bearing from the miner as well as height from the ocean floor. Several sets of fixed and pan and tilt TV cameras are installed on both the buffer and miner providing detailed video information, which was fed to the main control consoles in the control room aboard ship. Multiple recording systems were employed so that a permanent record of the outputs of all sensors employed was obtained. This included video, sonar, electrical, hydraulic, and mechanical data. Post review and analysis of all the data reveal that sufficient engineering and operational information was obtained to give confidence in the design and expected performance of future pilot and commercial systems.

CONCLUSIONS. 1. A self-propelled vehicle in the weight range of 100 tons (90 t) or more can be successfully operated on the soft pelagic clays of the deep ocean floor.

2. The tests proved that sufficient control of the speed and manueverability of a large vehicle on the deep ocean floor can be achieved to allow efficient mining of manganese nodules.

3. Dynamic positioning of the surface ship obtainable by use of computer-controlled thrusters is sufficiently accurate to allow coordination of movement between the surface ship and the miner.

4. Large electrical and hydraulic power systems can be reliably operated at a depth of 17,000 ft (5000 m) by cable from a surface ship.

5. By the use of multiple sensors that measure range and bearing, speed and direction, voltage, amperage, pressure, rpm, position, and angle, sufficient information can be obtained on the mining status to allow adequate operational control of the miner.

6. All of the necessary data on the operation of the ship, pipe, and miner system can be displayed in a control room of the surface ship in such a manner as to allow timely control decisions.

7. Large systems with dimensions of 60 ft (18 m) or more and weighing several hundred tons can be successfully launched, placed upon the ocean floor, and retrieved in the sea state normally encountered in the mid-Pacific.

METALLIFEROUS SULFIDE MUDS FROM THE RED SEA (Anon., 1981b; Anon., 1986; Amann, 1989). Lying below 6700 ft (2000 m) of ordinary sea water at 68°F (20°C) and 670 ft (200 m) of concentrated brine at 1 ton/yd^3 (1.2 t/m^3) and 140 to 149°F (60 to 65°C), a metalliferous mud deposit extends over an

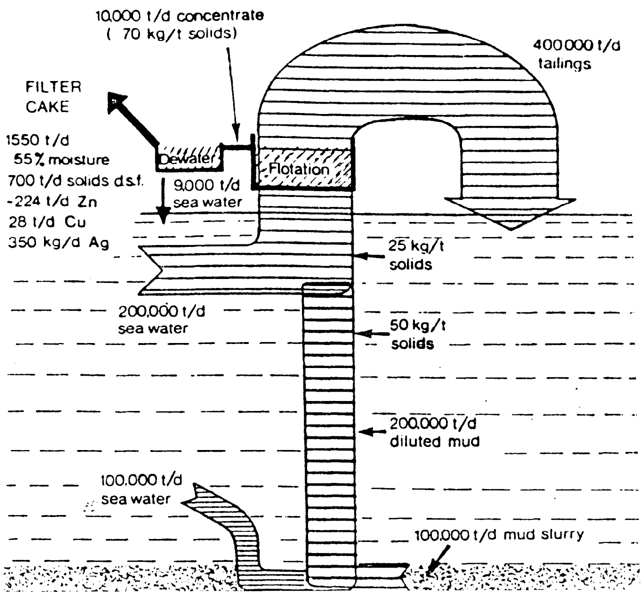


Fig. 22.8.32. Slurry cycle in Red Sea mining tests (after Anon., 1981 b). Conversion factors: 1 lb = 0.4536 kg, 1 ton = 0.9072 t.

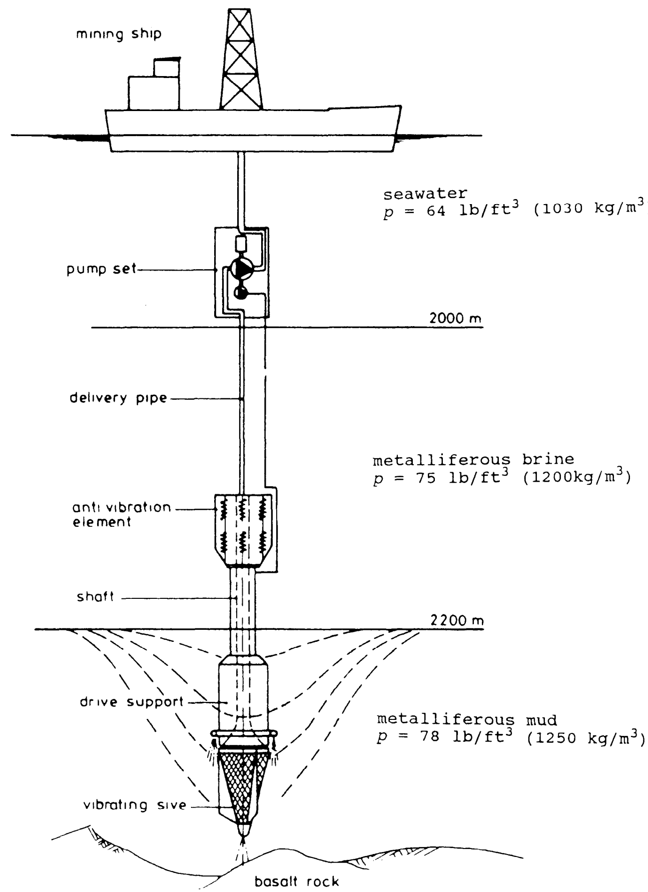


Fig. 22.8.33. Mining system components for Red Sea test on metalliferous muds (after Hahlbrock, 1979). Conversion factor: 1 ft = 0.3048 m.

area of 25 mi^2 (65 km^2) in the Red Sea. Although the surface is fairly even, the thickness varies from 7 to 100 ft (2 to 30 m), and the strata present may include two or three metal-enriched units. The consistency of the mud varies with depth but is most commonly similar to shoe polish. It is thought that deposition has resulted over a period of at least 25,000 years from the metal enrichment of seawater in contact with magma along a crustal fissure, followed by precipitation during cooling.

Pilot Mining Test: Concept to Trial—It was required to develop a three-month initial mining test leading to a pilot operation. Although the sediment is compacted to "shoe polish" at depth, on the surface it is quite soft, and the water content can be as high as 200%, the wet density 78 lb/ft^3 (1250 kg/m^3) and the shear strength 8.7 to 36 psi (60 to 250 Pa). Given this consistency pattern and an economic output requirement of around 220 tpd (200 t/d) contained metal, the production and pumping to surface of a mud slurry was seen to be the most effective route. However, because of the fine particle size of the sediments, slurrification of the muds in seawater yields a viscous suspension with unusual rheological properties: the muds are thixotropic and Bingham plastic, having a high initial shear strength. Therefore, considerable dilution is necessary to permit pumping to surface. To mobilize the required daily metals throughput, it would be necessary to extract 11,000 tpd (10,000 t/d) of in situ mud, creating 11,000 tons/day (10,000 tpd) of slurry in the process. As shown in Fig. 22.8.32, this 110,000 tons (100,000 t) must then be diluted with an equal amount of sea water before it can be pumped to the ship. Two options were initially considered, a plunger pump and a vibrating suction sieve, and the latter was selected.

The suction head designed by Orenstein and Koppel comprises a conical vibrating screen, which acts as a sieve vibrating in the axial direction, fitted to bars at the drive support which is configured to form a suction mouth. The slurry is directed from the head via a 67-ft (20-m) shaft, fitted with a vibration damping device, to the conveyor pipe string. The pressurized water used to help break up and then slurry the sediment is supplied by a pump set located 670 ft (200 m) higher with the slurry pumping equipment. The positioning equipment is fitted to the pipestring immediately above the damping device (Fig. 22.8.33).

Using this head, mining is affected by lowering the sieve through the sediment to a position just above the basalt base, loosening mud in the process by the combined effect of the waterjets and the vibrating screen. To accommodate different mud consistencies, the water pressure and vibration frequency can be adjusted to a certain extent, which also permits control of the slurry concentration. The vibration and downward taper of the sieve prevent clogging by oversize solids. As the head proceeds downward through the sediment, the loosened material flows over its top, sealing out the brine. The slurry density produced by the mining action is less than the 0.9 tons/yd^3 (1.1 t/m^3), which characterizes the brine zone, so that the vertical lift is effectively supported.

The major components of the pump unit are a main mud pump and electric motor, a flush pump to supply the pressure water to the suction head jets via a hose, and an electronic control unit with boxes to distribute power and provide connections between the main coaxial cable and the data transfer systems. A specially designed deep-sea power cable links the control unit and the suction head motor. This cable and the water supply hose are attached to the pipe string and shaft. The power for the underwater electric motors, instrumentation and control devices was to be provided by a 1000-kW, frequency-regulated generator and 440/3000-V transformer assembly on the mining vessel.

The mud pump eventually selected was a modified Worthington 4 UX-1, six-stage, centrifugal pipe-line-type pump, with oil-filled pressure-compensated bearings outside the pump body. The centrifugal type slurry pump first tested suffered severe bearing damage after two weeks of pumping Red Sea mud. Slurry, at 78 lb/ft^3 (1250 kg/m^3) average, could be pumped at 43 to 92 yd^3/hr (33 to 70 m^3/h) with a pump rotation speed of 2850 to 3560 rpm. The pump was driven by a water-filled, pressure-compensated 3000-V, 535-kW Pleuger motor, through a water-lubricated toothed coupling. The flush pump chosen was an 8-stage Pleuger submersible unit.

For transporting the abrasive, corrosive mud slurry to the mining ship and, at the same time, supporting the mining head, pump unit, and measuring devices, Preussag evaluated fiber-reinforced plastic pipes, high-pressure hoses, and steel oil drilling pipe. The plastic pipe interconnections proved unsatisfactory; the high-pressure hoses were expensive, did not easily withstand the starting of the submerged motors, and had not been used in large-diameter sizes. To support the 110-ton (100-t) static load, withstand the starting moments, and handle 39 to 92 yd^3/hr (30 to 70 m^3/h) slurry, an API Standard 5A drill pipe (25 Cr, 5 Mo steel) of 5-in. (127-mm) diameter was selected.

To permit proper control of the mining operation and to monitor operating conditions, a comprehensive instrumentation system was designed, to be installed at three locations: the mining head, in the pump unit, and at the storage tank entry on board ship.

Recovered as a relatively dilute slurry, containing not more than 5.8 lb/ft^3 (100 g/L) solids of fine particle size (80% at -2μm), the metalliferous muds are unlike any of the feeds usually supplied to mineral processing facilities. Yet since the muds contain only 3 to 6% zinc, up to 1% copper, and about 0.03 oz/ton (50 g/t) silver, some concentration prior to extractive metallurgy is essential.

Because it is not possible to blow air bubbles through a slurry containing more than about 3 lb/ft^3 (50 g/L) solids in the manner used in froth flotation, the application of that process was at first ruled out and direct leaching investigated instead. However, not only were attempts to achieve selective metals recovery unsuccessful either at atmospheric pressure or using oxygen pressure, the transport of large volumes of dilute slurry to the necessary land-based plant would be very costly. Similarly, thermal beneficiation would require prohibitively expensive dewatering and drying, and both these routes were abandoned.

When flotation processes were investigated, early success was, in fact, achieved (at Clausthal University) by the simple expedient of further diluting the slurry with sea water and limiting the solids content to less than 1.7 lb/ft^3 (30 g/L). Such a dilution immediately prior to the slurry entering the flotation circuit poses no particular problems on the mining vessel. The reagents used in the process are xanthate collectors with copper sulfate and sodium sulfide activation, standard and similar to those used for bulk sulfide flotation of complex sulfide ores. The process initially differed only in the use of sea water, elevated temperatures, and the dilute pulp; later it was discovered that elevated temperatures were unnecessary.

To assess the effects of ship motion on flotation, Preussag enlisted the help of the Warren Spring Laboratory at Stevenage, UK, where a heave motion simulator was available. The results of tests under rather severe simulated sea conditions of heave, roll, and pitch indicated that flotation was not adversely affected by ship motion. It was concluded that design would have to provide for pulp levels in the cell such that spillage of the froth was prevented, and that to produce 90,000 tpy (80,000 t/a) zinc in concentrate would require a total cell volume in the region of 176,550 ft^3 (5000 m^3), assuming a 70% recovery.

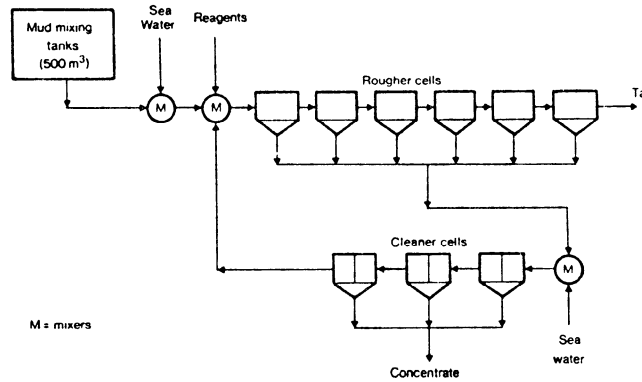


Fig. 22.8.34. Flotation plant on-board test vessel, SEDCO 445 (after Anon., 1981 b). Conversion factor: 1 ft^3 = 0.0283 m^3.

To perform the actual prepilot mining test, it was necessary to find a ship that could be positioned accurately above the chosen mining sites. The sophisticated drill ship *SESCO 445* was chosen because it was known to have an efficient navigation system, good stability at sea, and an experienced drill crew. The rig on board was able to lower the pipestring into the mud deposit in approximately 12 hr and retrieve it in 8 hr. Sediments were successfully pumped aboard for the first time on May 1, 1979, and it was established that the system could deliver mined slurry at a rate of 131 yd^3/hr (100 m^3/h). At first the density of the slurry was rather low and variable, but with experience, it became possible to deliver slurries that contained up to 5.8 lb/ft^3 (100 g/L) solids. The method of dynamic position fixing used for *SESCO 445*, with both surface and seabed radio beacons, proved to be highly accurate, and it is believed that this is the first time that the technique has been used for mining at depths greater than 3300 ft (1000 m). Several methods of mining were tested (crater, pattern, and trench) with use of both a reciprocating movement of the suction head and controlled ship movement. The data gathered were stored on computer for subsequent evaluation.

The flotation circuit supplied by Krupp for use on *SEDCO 445* was typically arranged as shown in Fig. 22.8.34, though several modifications were tried. The plant operated satisfactorily from shortly after start-up at a slurry feed rate of about 212 ft^3/hr (6 m^3/h) and a pulp density of about 1.7 lb/ft^3 (30 g/L). With experience, it was possible to obtain equally good results irrespective of the zinc grade in the mud (which varied according to the site mined). The concentration target was easily achieved and frequently exceeded, with grades of about 40% being recorded on several occasions. The concentrate tonnage target was also achieved, with recoveries being generally about 60% Zn.

On the basis of water oxygen content measurements in the Atlantis II Deep area and observations of the distribution of life forms in the Red Sea, it was decided to discharge tailings from the shipboard flotation plant at a depth of 1300 ft (400 m); at this depth, oxygen is at a minimum, and life lower down is very limited, while current speeds are also at a minimum so that the sediment could be expected to sink. The probable behavior of the tailings plume was modeled at Imperial College, London, prior to the test. To monitor the movement of the plume in practice, *Valdivia* was able to use its ARNAV navigational system in conjunction with a 30-kHz echo sounder, although during the day, plankton movements obscured the readings from the plume so that experiments were restricted to nighttime when the plankton stay near the surface. These observations showed that the plume behaved much as expected, though deep penetration was better than predicted by the model. The discharge penetrates to a depth of 4000 ft (1200 m) within 20 min, with a marked concentration occurring at 2700 ft (800 m). The solids are rapidly diluted, and after a few days the concentration in the plume may be less than twice the normal for Red Sea waters.

Extractive Metallurgy—Under subcontract to Preussag, Lurgi and Davy International prepared shortlists of potentially economically viable process technologies that might be used for zinc, copper, and silver extraction. Cyclone smelting was selected as the most attractive pyrometallurgical route but was abandoned because of the need for a low-salt feed. Four hydrometallurgical routes have been considered as well. Three of the processes involve chloride leaching, solvent extraction, and electrowinning steps according to rather different flowsheets. These are Duval's CLEAR process, modified to include a zinc recovery step; the Minemet process; and the Elkem process. The fourth route involves oxygen pressure leaching in the presence of chloride ion followed by the Comprex process being developed by Tecnicas Reunidas.

22.8.3.3 Mining of Subseabed Deposits

Undersea mining has been carried out by conventional methods from shoreside entries for at least 3000 yr since the Laurium mines were developed under the sea in ancient Greece. Underground mines with shaft entry from the coast or islands have recovered coal, iron ore, nickel-copper ores, tin, and limestones off the coasts of Alaska, Canada, England, Finland, France, Greece, Ireland, Japan, Spain, Taiwan, Turkey, and the United States. Although conventional mining methods prevail, solution mining methods are applicable to many deposits (see Chapter 15.3). Advances in rapid tunneling (Howard, 1976) and subsea technologies (Austin, 1967; Warfield and Parkinson, 1968) will give added incentives to the development of hard-rock deposits offshore (Wang and Cruickshank, 1969). Some current operations are listed in Table 22.8.19.

In general, underground mining is little affected by the marine environment, and additional costs commonly prevail only in the exploration and development phases. As deposits further offshore are discovered and developed, the nature of the environment will become more significant (McMurray, 1990).

22.8.3.4 Environmental Impact Assessment

A listing of completed environmental assessment documents on marine mining in the United States follows, with references:

1. Sand and Gravel, Beaufort Sea, Alaska (Anon., 1983c).
2. Metalliferous sulfides, Gorda Ridge, USA (Anon., 1983b).
3. Manganese Nodules, LOS Negotiations, US/UN (Anon., 1974b).
4. Manganese Nodules, NE Pacific, USA (Anon., 1984b,c,d,e).
   b. Kennecott Consortium.
   c. Ocean Management Inc.
   d. Ocean Mining Associates.
   e. Ocean Minerals Company.
5. Cobalt Crusts, Hawaii-Johnston, USA (Anon., 1990c).
6. Gold, Nome, Alaska (Anon., 1990a).
7. Programmatic OCS, USA (Anon., 1974a).
8. Programmatic Deep Seabed, USA (Anon., 1981a).

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Anon., 1983c, "Final Environmental Impact Statement for Proposed Outer Continental Shelf Arctic Sand and Gravel Lease Sale," US Dept. of the Interior, Minerals Management Service, Mar.

Anon., 1984a, *Analysis of Exploration and Mining Technology for Manganese Nodules,* Vol. 2, United Nations, Ocean Economics and Technology Branch, Graham & Trotman, London.

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Table 22.8.19. Data on Underground Mines Operating Offshore

Name of Mine	Location	Company	Minerals	Depth below Sea Level ft	Max. Dist. From Shore	Access	Method of Working	References
Collieries (5)	NS, Canada	Dominion Coal Co. Ltd.	Coals	300–1000	5 mi (reserves)	Adit on shore	Room and pillar, 300–1000 ft	
			5-ft 7-in. seam @ 4–24°				Longwall below 1000 ft	MMTC
Chien Chi coal	Taiwan	Chien Chi Coal Mine Co.	Coal, 0.3–3.3 m @ 9–47°	0–1500	9000 ft	Adit on shore	Longwall	MMTC
Kozlu coal	Turkey	Eregli Coal Mines Ltd.	Coal, 19 seams @ 45° 3–60′	1500	1000 m	Vert. shaft on shore	Stepped longwall	MMTC
Collieries (11)	Japan	Mitsui Mining Co. Ltd.	Coal	600–2400	5 mi	Inclined & vert. shafts on shore & artificial islands	19 operating mines	MMTC RRI 1970
Collieries (31)	UK	National Coal Board	Coal	300–8800	5 mi	29 shafts on shore, 1 natural island, 1 artificial island	Longwall	Armstrong, 1965
Lotaschwager	Chile	Lotaschwager Coal Co.	Coal	3000	4 mi	Shaft on shore	NK	MMTC
Jusarro Gruva	Finland	Oy Vuoksenniska Ab	Magnetite quartz banded iron ore	30 m, steep dipping	1000 m	Shaft on shore	Shrinkage stoping	MMTC
Levant	Cornwall, UK	Geevor Tin Mines	Tin, vert. lodes	Holed through	1 mi	Shaft on shore	Underhand stoping	Batchelor, 1969
Grand Isle	LA, US	Freeport Sulphur Co.	Sulfur, 220–425 ft dome	2000	5 mi	Drillholes	In situ Frasch process	Lee, 1960
Camenada*	LA, US	Freeport Sulphur Co.	Sulfur	—	—	Drillholes, offshore platform	In situ Frasch process	MJ, 5/66; EMJ, 8/67; WSJ 2/69
Castle Island	Alaska	Alaska Barite Co.	Barite	50	1 mi	Sea surface	Blast and dredge	Thompson, 1970
Yorkshire Potash†	England	Rio Tinto Zinc Corp.	Potash	3500	5 mi	Shaft on shore	Room and pillar	MJ, 5/69
Goderich	ON, Canada	Sifto Salt (1960) Ltd.	Rock salt, 75 ft, 0.2°	1170	2500 ft	Vert. shaft on fill	Room and pillar	Mamen, 1969
Cape Rosier	ME, US	Callahan Mining Corp.	Cu/Zn sulfides	Tidal	—	Land surface	Open pit	Beck, 1970

* Closed temporarily, 1969.
† Under development. Production scheduled 1970.
Conversion factors: 1 ft = 0.3048 m, 1 mi = 1.609 km.
Source: Cruickshank and Marsden, 1973.

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Chapter 22.9
OTHER NOVEL MINING METHODS

PETER G. CHAMBERLAIN, ROGER J. MORRELL, DAVID A. LARSON, STERLING J. ANDERSON, DAVID P. LINDROTH, GEORGE A. SAVANICK, AND EGONS R. PODNIEKS

22.9.1 INTRODUCTION

PETER G. CHAMBERLAIN

Previous chapters of this section feature new methods that are beginning to play a strong role in mining. Each chapter presents a novel mining method that is moving from development into commercial practice. Other less-developed methods, however, are slowly gaining acceptance in the mining community. Some of these may have an important role in future mining. Others may vanish. With their development still in infancy, it is simply too early to determine their future impact on mining. The purpose of this chapter is to present enough information on these other methods to identify possible applications. Sources of further information are listed at the end of the chapter.

The following six methods are presented in this chapter:

22.9.2 A ripper miner concept of mechanically cutting mine development openings with one large, single bit.

22.9.3 A radial-axial rock-splitting device for mechanically creating development and production openings in mines.

22.9.4 Rock extraction with thermal energy.

22.9.5 Shock wave rock breakers for splitting oversize rock.

22.9.6 Lunar and planetary mining concepts.

22.9.7 Nuclear-assisted mining.

These methods all have a single common characteristic: they are not yet commercially available. Some have been extensively tested and may be commercially available before the end of the century. Others are still preliminary concepts and will not be practiced until well into the 21st century, if at all.

With such diverse methods, the scope of the information provided in this chapter varies considerably with the method. Since these methods are new, the information is based on experimental results rather than case histories. Where possible, the chapter presents the following information for each method:

1. Concept.
2. Background.
3. Implementation: mining operations, equipment design, factors influencing use, nature of the rock product.
4. Cost estimates.
5. Case histories (primarily field tests).
6. Future trends.

Insufficient information exists to provide a “cookbook” recipe for implementing these methods. Rather, the chapter attempts to provide background material that will stimulate potential users to seek updated data on particular applications.

For additional coverage of the fundamentals of rock breakage, the reader is referred to Chapters 9.1, 9.2, and the previous chapters of this section.

22.9.2 RIPPER MINER

PETER G. CHAMBERLAIN, R.J. MORRELL, AND D.A. LARSON

22.9.2.1 Concept

The US Bureau of Mines (USBM) is presently testing a concept for cutting and excavating underground openings in rock with a cutting head consisting of a single, large (12 to 24 in., or 0.3 to 0.6 m, wide) bit. The concept has been termed the *ripper miner method* (Morrell, Larson, and Schmidt, 1989).

The method features a steel or tungsten carbide bit that operators can easily remove from the cutting head for resharpening. During a cutting cycle, push-pull hydraulic cylinders force the bit from its starting position upward through an arc of about 185° (Fig. 22.9.1) (Schmidt, 1989). After each cut, the cutting head returns the bit to its starting position, rotates sideways to the next cutting position, and again moves the bit upward through the cutting arc. When the cutting strokes have completed one pass across the face, the cutting head advances the depth of cut desired and the process begins again.

The ripper miner cuts most rock, but is better suited for soft- and medium-strength rocks, such as sandstone, limestone, and shale. Equipped with a 130-hp (97-kW) hydraulic motor and a 12-in. (300-mm) bit, the machine is capable of advancing 5 fph (1.5 m/h) in a 10- by 10-ft (3- by 3-m) heading in rock with a compressive strength of 18,000 psi (120,000 kPa). The ripper miner is energy efficient because the large, slow-moving bit breaks rock in tension rather than crushing it as do other large drilling and boring machines.

22.9.2.2 Background

Where engineers can blast large volumes of rock at one time, the drill-blast-muck cycle provides a rapid and economical method of excavating rock. In small and confined underground openings, however, blasting efficiency and economy drop. The cyclic movements of personnel and equipment in and out of the openings during the performance of drilling, loading, blasting, roof support, and mucking operations limit productivity. Furthermore, blasting may damage the surrounding rock, increasing the necessary rock support.

These problems spawned much research over the past several decades to develop alternative mining methods. Mechanical excavation techniques have received the most attention. In essence, researchers hoped to introduce a mechanical-mining concept for hard rock similar in principle to the continuous-drum mining machines that have revolutionized coal mining. The costs, advance rate, and flexibility of blasting for excavating rock remained the standard of comparison. Methods studied in the Bureau and in private industry included tunnel boring machines (TBMs), roadheaders, continuous coal mining machines, raise borers, impactors, saws, and drills. These methods either could not fragment hard rock or could not do it as economically as could blasting.

USBM researchers sought a new approach to cut rock. They wanted to fracture the rock in tension rather than crushing it in compression to take advantage of rock’s inherently weak tensile strength. Bureau researchers developed the concept of cutting with a large, slow-moving bit that would allow large chips to spall ahead of the bit as it moved through the rock (Morrell and Larson, 1985). Researchers next tested the concept using 9-in.-(230-mm) wide bits at full-scale cuts in a 6- by 6-ft (2- by 2-m)

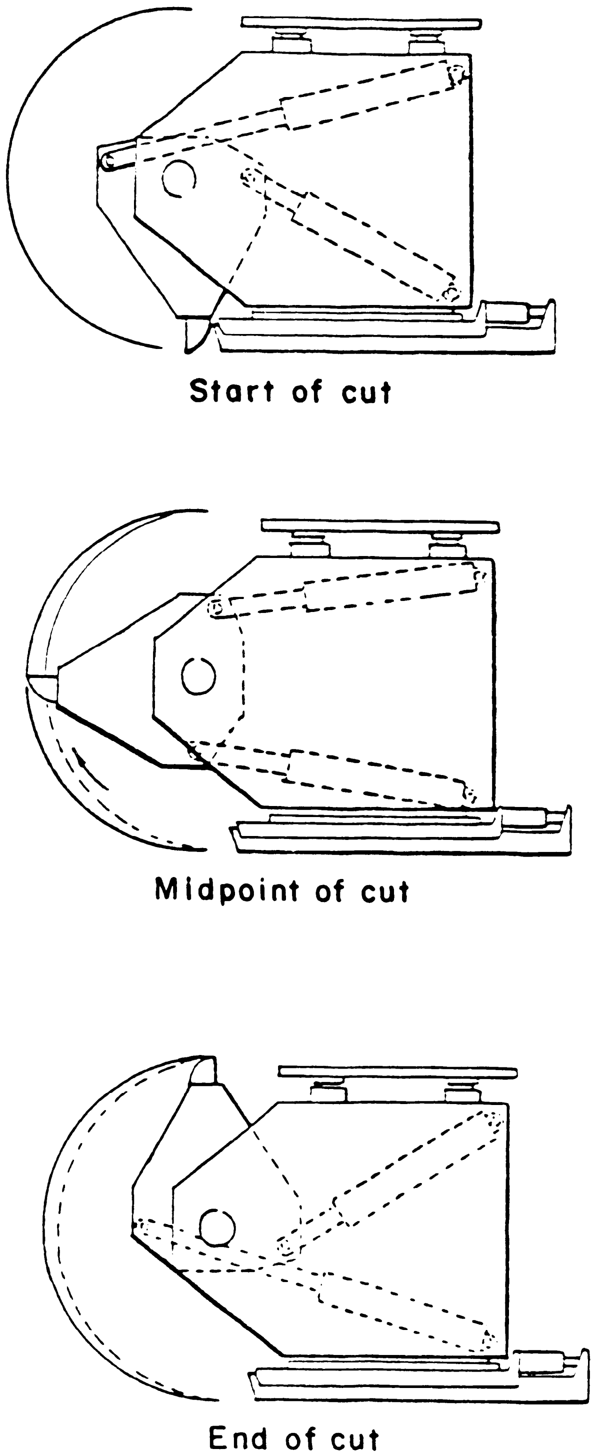

Fig. 22.9.1. Ripper miner concept showing cutting action (Larson et al., 1987).

simulated mine opening (Fig. 22.9.2). Researchers also fitted the machine with subsystems to rotate the cutting head sideways after each cutting sweep and to advance the cutting head after the bit had cut the full face. This configuration allowed research-

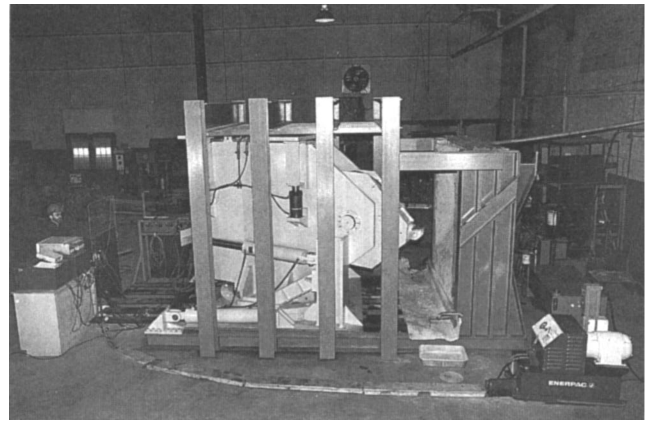

Fig. 22.9.2. Full-scale ripper miner cutting head cutting a concrete block during laboratory tests (Larson et al., 1987).

ers to establish cutting forces and energy requirements for confined side-kerf cuts as well as cuts in the central portion of the face.

These tests provided performance data on various cutting bits and demonstrated desired operating sequences (Larson, Morrell, and Swanson, 1987). The cutter cut rock more efficiently than could TBMs by cutting more of the rock in tension. This also yielded less dust, which could easily be suppressed with water sprays.

22.9.2.3 Implementation

MINING APPROACH. USBM engineers believe that the first commercial machines will be designed to excavate underground mine openings up to about 20 by 20 ft (6 by 6 m) in rock with 5000 to 30,000 psi (30,000 to 200,000 kPa) compressive strength. To establish the desired operating parameters, an engineer first determines the size of opening desired and the type of rock anticipated. The rock type may dictate the type of bit; full-scale laboratory experience indicates that the width of the bit should be about one-tenth the width of the opening.

To operate the machine in an underground heading, an operator would advance the cutting system by its thrust cylinders to a depth equal to the desired depth of cut. The depth of cut is usually one-third to one-half the width of cut.

The cutting head is then rotated horizontally to position the bit where the first cut begins. This can either be in the center or on one side of the face (Fig. 22.9.3).

The cutting system consists of push-pull cylinders that force the bit upward through an arc of about 185°. The rock pieces spall from the face ahead of the bit and fall onto the cutterhead. When the upward cut is completed, the operator swings the bit back down to its starting position. The rock cuttings drop into a chute. Each cut requires about 15 sec to complete. Repositioning between cuts requires an additional 15 sec for a total cycle time of 30 sec.

Although the cutter breaks the rock primarily in tension, some dust is generated by crushing during operation. Water jet nozzles mounted so that the spray covers the leading and trailing edges of the bit effectively reduce the dust to an acceptable level.

EQUIPMENT. The first commercial ripper miners will probably be built to cut rectangular openings 10 by 10 ft (3 by 3 m). Engineers believe that ripper miners eventually could be built to cut 20- by 20-ft (6- by 6-m) openings. The system for 10-ft (3-

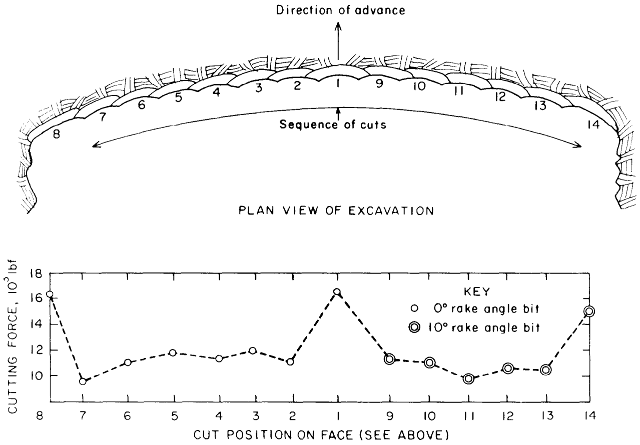

Fig. 22.9.3. Sequence of cutting positions for the ripper miner and corresponding cutting forces (Larson et al., 1987).

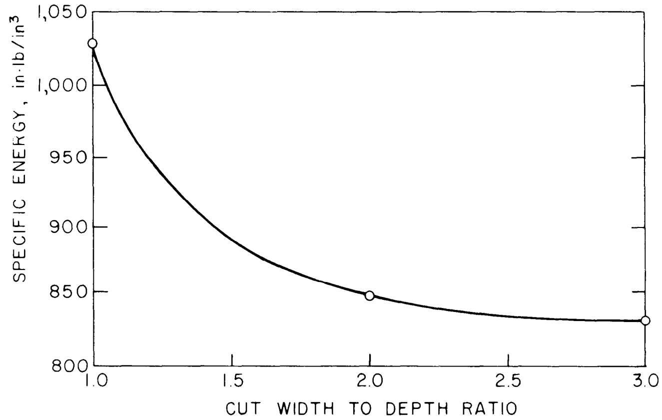

Fig. 22.9.5. Specific energy required to cut Indiana limestone as a function of the length-to-width ratio of the cut (Morrell, Larson, and Schmidt, 1989). Conversion factor: 1 in.-lbf/in.^{3} = 6.667 kJ.

Table 22.9.1. Ripper Miner Specifications

Subsystem	Specifications
Cutting	Two 8-in. diameter bore hydraulic cylinders; 150,000 lbf maximum at 3000 psi
Cutting speed	3½ in./sec at 30 gpm
Thrust	Two 6-in. diameter bore hydraulic cylinders; 600,000 lbf maximum at 10,000 psi
Rotation	Two 6-in. diameter bore hydraulic cylinders; 600,000 lbf at 10,000 psi
Hold-down	Four 100-ton jacks; two forward, two rear
Cutter	Each 9- to 12-in. wide; crown type, bolted on, heat-treated tool steel
Dust suppression	Water spray

Conversion factors: 1 in. = 25.4 mm, 1 ton = 0.9072 t, 1 lbf = 4.448 N, 1 psi = 6.895 kPa, 1 gpm = 0.0631 L/s.
Source: Morrell, Larson, and Schmidt, 1989.

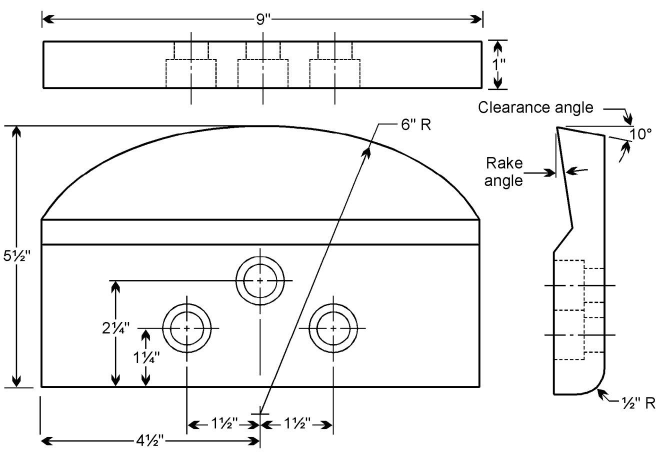

Fig. 22.9.4. Bit configuration for ripper miner (Larson et al., 1987).

m) headings will require approximately 200 hp (150 kW) to operate the cutting head with an 18-in. (460-mm) bit. Specifications for the various cylinders that comprise the operating systems of the full-scale ripper tester are provided in Table 22.9.1.

The heart of the cutting system is the bit. Drag bits 9 to 18 in. (230 to 460 mm) wide will be mounted in modest-sized machines. Bits up to 24 in. (610 mm) wide could be mounted in larger machines (Fig. 22.9.4).

The bits would be bolted to the cutter arm by three bolts to allow easy exchange for sharpening. Engineers estimate that the ripper miner could cut soft rock and coal for an entire shift before the operator must change bits. In hard, abrasive rocks, the bit may require changing five or more times. For most sedimentary rock, miners would change bits once or twice during each shift. One operator can change a bit in 5 min. Bits can be resharpened five to ten times.

Four water-spray jets comprise the dust-suppression system. These jets would be pointed toward all four sides of the bit to assure maximum dust suppression.

ROCK HANDLING AND PROCESSING REQUIREMENTS. The ripper miner cutting system produces coarser muck than would a TBM. Larger pieces result from the ability of the large, slow-moving bits to spall much of the rock with tensile forces. The largest pieces are about the same size as the cut; a 2-in. (50-mm) deep cut with a 9-in. (230-mm) wide bit would produce a maximum chip of about 2 by 9 in. (50 by 230 mm). By weight, more than 40% of the muck would be retained on a 2-in. (50-mm) screen and more than 70% on a 0.5-in. (13-mm) screen (Larson, Morrell, and Swanson, 1987).

Muck is dumped in a hopper in the machine at the end of each stroke. Materials handling would be most efficient if the machine were linked to a continuous conveying system. The muck could also be dumped in a shuttle car for transportation out of the heading.

FACTORS INFLUENCING USE. Rock strength controls the applications of the ripper miner concept. The cutting system has been tested on rocks with strengths over 45,000 psi (310,000 kPa). It appears most suited for sedimentary rocks with strengths between 5000 and 30,000 psi (30,000 and 200,000 kPa).

The ratio of the depth of cut to its width greatly influences the energy expended in cutting (Fig. 22.9.5) (Morrell, Larson, and Schmidt, 1989). The USBM believes that cutting at a depth at one-fourth to one-third of the bit width provides the best cutting efficiency.

Bit wear is governed by forces on the bit, speed of the bit, hardness of the material, and abrasiveness of the material being cut. Of these factors, the operator can most easily control bit speed. Wear studies on drag bits indicate that best life can be achieved at bit speeds of around 12 in./sec (300 mm/s) (Haspert, 1978).

Table 22.9.2. Ripper Miner Cost Calculations

Assume a 10- by 10-ft ripper mining machine cost of $250,000 with a 5-yr and 2500-shift life and 18,000-psi rock.

Ownership Costs	
1. Amortization of ripper miner $250,000/2500 shifts	$100/shift
2. Interest, taxes, and insurance—20% (average investment per year)/500 shifts	$60/shift
Operating Cost	
3. Power for ripper 200 hp = 148 kW at $0.05 for 5 hr	$37/shift
4. Maintenance cost 60% (of 1)	$60/shift
5. 18-in. wide cutter cost (assume two cutters/shift @ $150 each)	$300/shift
Labor Cost	
6. One operator $20/hr by 8 hr	$160/shift
TOTAL COST	$717/shift
TOTAL COST/TON ($717/317 tons)	$2.26/ton

Source: Larson, Morrell, and Swanson, 1987.

22.9.2.4 Costs

With reasonable allowances for delays, productivity of the ripper miner for driving a 10- by 10-ft (3- by 3-m) opening with an 18-in. (460-mm) bit moving at a speed of 12 in./sec (300 mm/s) and with a 6-in. (150-mm) depth of cut is estimated at about 300 tons (272 t)/shift. This was based on the following performance data: cutting and positioning time per stroke 30 sec; cutting time for one pass across the face (7 strokes) 3.5 min; instantaneous advance rate 8.5 fph (2.6 m/h); advance rate (assuming 5 hr operating time/shift) 42 ft (13 m)/shift; and tonnage rate 317 ton (288 t)/shift (Morrell, Larson, and Schmidt, 1989).

Capital costs for the cutting machine are estimated at $250,000 in 1989 dollars. This cost is based on a machine capable of cutting 10- by 10-ft (3- by 3-m) openings and fitted with a 200-hp (150-kW) motor to power the hydraulic cylinders. The machines are expected to last 5 years or 2500 shifts in 18,000-psi (120,000-kPa) rock.

Bits cost about $150 each. Cutters are expected to wear twice as fast as laboratory bits, and so two bits per shift is the basis for estimating bit costs.

Detailed estimated cost calculations are shown in Table 22.9.2. The 1987 cost per ton of rock excavated, $2.26, compares favorably with drilling and blasting costs of $2.60 for the same size opening ($2.49 vs. $2.87/t).

22.9.2.5 Future Trends

Manufacturing companies have expressed interest in the ripper miner concept. The USBM would like to award an exclusive license to manufacture the equipment. Hopefully, ripper miner systems will be commercially available in the 1990s. The first machines will probably target the tunneling and development markets.

Researchers continue to investigate improvements to the system that will facilitate cutting very hard rock. Several bits that cut hard rock much more efficiently are being tested. Thermal softening of rock ahead of the bit may also offer improved cutting speeds with lower forces and reduced bit wear. Kerfing with water jets may eventually progress to the point where operators could save energy and reduce cutting forces particularly at the edges of the openings.

Fig. 22.9.6. Drill-split fragmentation apparatus.

22.9.3 DRILL-SPLIT FRAGMENTATION WITH THE RADIAL-AXIAL SPLITTER

PETER G. CHAMBERLAIN AND STERLING J. ANDERSON

22.9.3.1 Concepts

Another new method of mechanically extracting rock is called *drill-split fragmentation.* The heart of this system is a unique splitter. This splitter applies both an axial and a radial load to a rock mass from within a drillhole (Fig. 22.9.6). In operation, the combined radial and axial forces fracture a plug of rock from the face. This system is an outgrowth of radial-loading splitters that miners use to trim mine and tunnel openings.

Since the hydraulic cylinder contains the reaction forces of excavation, the splitter can be light. Operators can carry the compact, light splitter with a drill on a common frame anywhere underground that they can carry a drill. Drill-split methods offer the same flexibility as do drill-blast methods to meet the excavation requirements. Drill-split equipment can excavate in any direction and create openings of any size or shape.

Developers designed the radial-axial splitter to perform as a primary underground rock excavation machine. It is especially well suited to applications where blasting excessively damages the rock wall or where blasting causes unacceptable vibrations or noise. The splitter has operated effectively in bedded and fractured rock without any apparent loss in efficiency. The splitter has proven effective in a broad range of rock types. In the laboratory, it has extracted plugs from large blocks of concrete, limestone (13,000 psi or 90,000 kPa, compressive strength), and granite (26,000 psi or 180,000 kPa, compressive strength). It performed well during tests in an underground limestone mine.

The drill-split method offers several appealing attributes. The splitter is easy to operate, requiring a skill level comparable to drilling. One operator can both drill and split rock from a single setup, reducing both the crew size and excavation time. Miners can excavate rock with a simple two-step, repetitive process. A nonviolent process, the drill-split method is compatible with nearby mining operations. Being essentially continuous, it avoids the production delays attendant in drill-blast methods. Drilling and splitting eliminates the safety problems associated

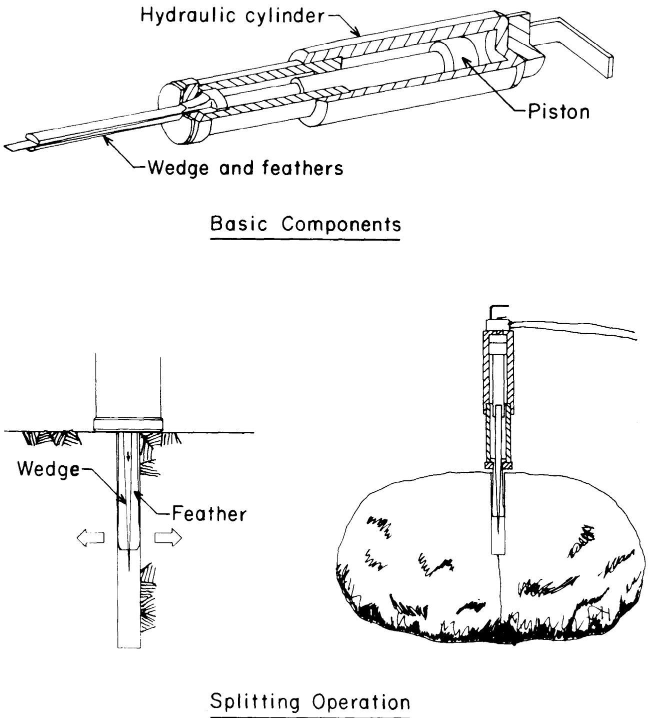

Fig. 22.9.7. Radial-demolition splitter (Anderson and Swanson, 1982).

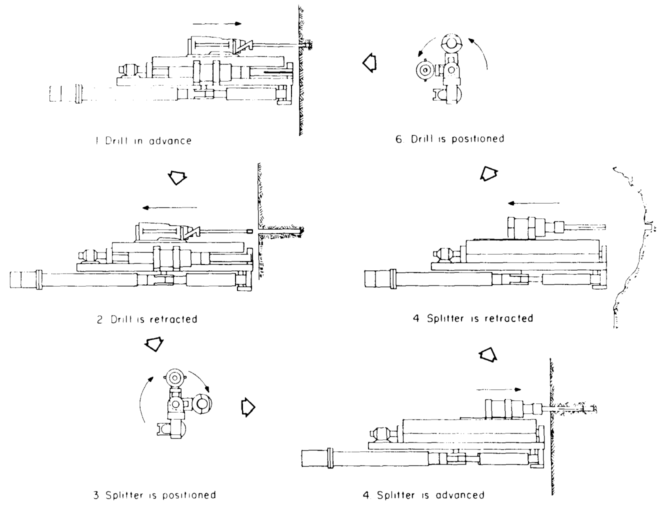

Fig. 22.9.8. Drill-split excavation sequence (Anderson and Swanson, 1987).

with blasting. Finally, the mine environment is healthier owing to the lack of blasting fumes and the reduction in dust.

22.9.3.2 Background

The basic concept of splitting rock with wedges in hole is not new. Workers in prehistoric societies split rock along pre-existing cracks with hammer-driven wedges. Commercially available splitters use hydraulically activated wedges to split boulders, split slate, trim mine openings, and demolish concrete structures.

These commercial splitters consist of a double-acting hydraulic cylinder, twin feathers, and a wedge (Fig. 22.9.7). During operation, a piston drives the wedge between the tapered feathers in a drillhole, creating radial loading against the wall of the hole. As the force increases, a fracture initiates and eventually propagates to the nearest free face.

Several investigators (Choulette et al., 1976; Clark and Maleki, 1978), and the USBM (Anderson and Swanson, 1982) attempted to develop primary mining schemes with these radial splitters. The researchers tried to fracture rock to free surfaces that were perpendicular to the face and parallel to the axis of drillholes. They wanted to excavate a longwall face with a series of side-by-side splits. Each operation of the splitter would hopefully break the rock to the free face created by the previous split. These experiments all met problems in fracturing the rock and in dislodging the fractured material. Such problems became more prevalent during experiments on driving headings.

An experimental tool developed by Institute CERAC in Switzerland combined radial and axial forces in a single drillhole. With this tool, an operator could fracture the rock without a free face parallel to the drillhole (Cooper, 1978; Cooper et al., 1980). USBM researchers began studying the technology to determine its mining potential. Using reduced-scale laboratory tests, the USBM measured the influence of various rock and operating parameters on the ability of the tool to split rock from a confined face (Anderson and Swanson, 1982).

From those experiments, the USBM conceived a mining system based on radial-axial splitting. The USBM incorporated specific unique features into the system to make it reliable, easy to operate, and economically viable. Full-scale tests at an operating limestone mine provide confidence that the method will soon appear in operating mines (Anderson and Swanson, 1987).

22.9.3.3 Implementation

MINING OPERATION. Mining crews could drill-split rock in many applications. Beside excavating development openings, operators could mine rock with the splitter in shrinkage stopes and underhand stopes. Operators could also mine rock in room and pillar and cut and fill operations with the splitter. The most likely early applications, however, will be driving development openings and specialized underground construction projects.

The sequence of operations for a drill-split excavation begins with the operator drilling a hole into the face (Fig. 22.9.8). The operator must drill the hole to the correct depth so the rod can push against the hole bottom. After drilling the hole, the operator rotates the drill out of the way and inserts the splitter. The operator then activates the cylinder. The radial force anchors the splitter, much like a rock bolt anchor, and generates the loads required to fracture the rock. These loads create a tensile stress field that fractures the rock in a bowl shape centered at the drillhole (Fig. 22.9.9). The fracture begins at the place in the hole that the feather is forced into the hole wall. It propagates in a curved path until it intersects the rock surface. The fracture usually meets the face at a distance from the hole of roughly 3½ times its depth. After the rock breaks, the operator again rotates the tools to drill the next hole. In large headings or for production applications, an automated excavation system may feature several such drill-splitter units mounted on one large carrier.

The required loads will vary with rock type. The operator must match the loads with each rock type encountered in a drift or tunnel.

Miners can drive blind headings in drifts with several configurations. Figs. 22.9.10 to 22.9.12 illustrate several possibilities. In the full-face technique, the center of the drift is advanced with

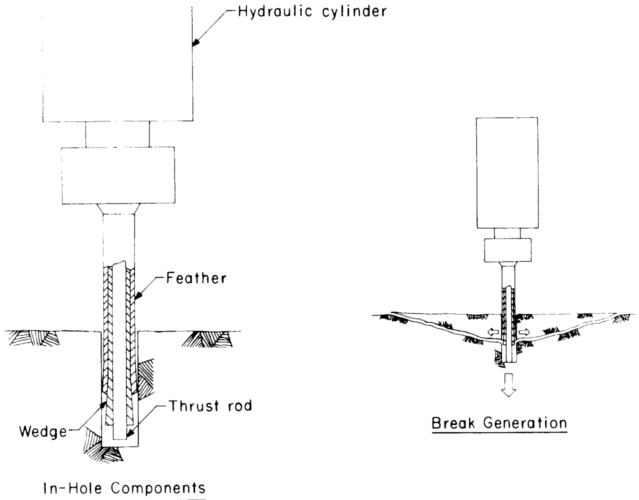

Fig. 22.9.9. Radial-axial splitter (Anderson and Swanson, 1982).

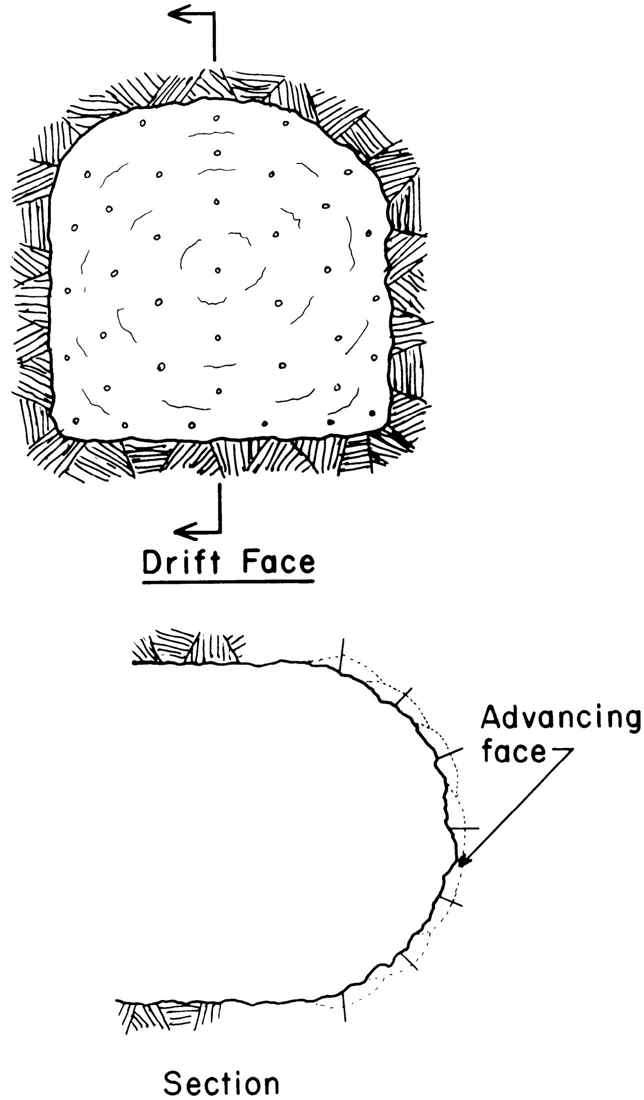

Fig. 22.9.11. Hemispherical-advance mining sequence (Anderson and Swanson, 1982).

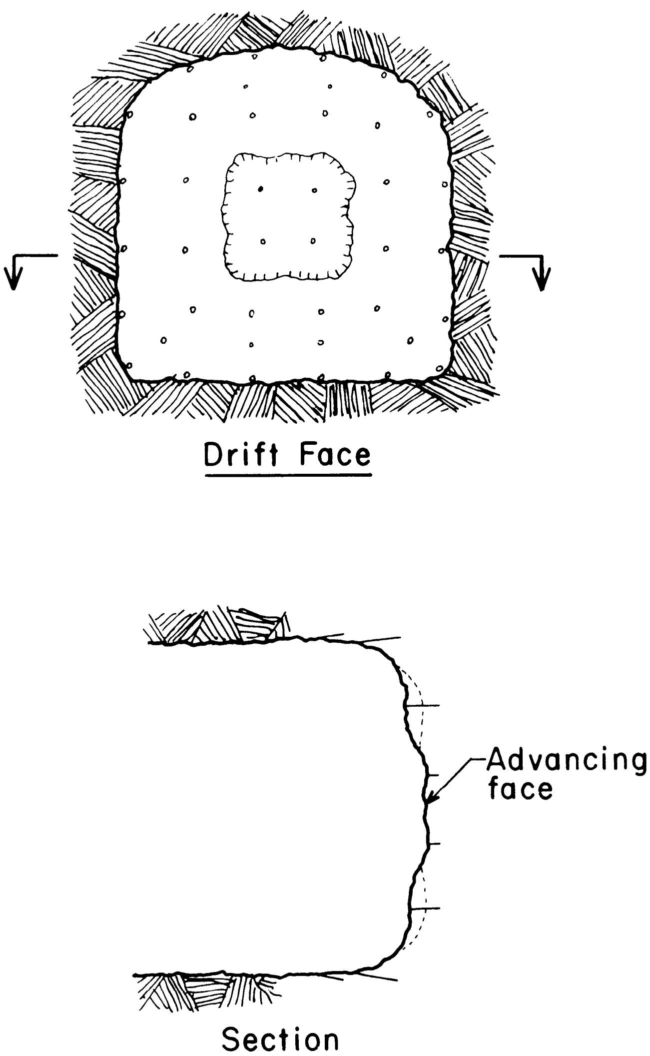

Fig. 22.9.10. Full-face-advance mining sequence (Anderson and Swanson, 1982).

the first break. Subsequent breaks work outward to the desired perimeter. All holes are drilled in the direction of advance. In the hemispherical technique, the operator again advances the center of the drift first. In this technique, however, the operator keeps the profile of the advance in a hemispherical shape by drilling the successive holes perpendicular to the curved surface of the advancing face. The third configuration, stepped-face advance, also begins with a central break. Instead of completing all the successive breaks to the perimeter, however, the operator begins a second advance as the first one widens sufficiently. The face eventually assumes a wide conical shape rather than a sharply stepped profile. The extraction forces and efficiency are roughly comparable among each configuration.

Operators must maintain the desired size and shape of opening. This focuses attention on excavating the highly confined corners of the drift. Two methods are available for removing the rock from sides and corners of the drift. First, the operator can slab the rock by drilling the perimeter hole at a slight outward

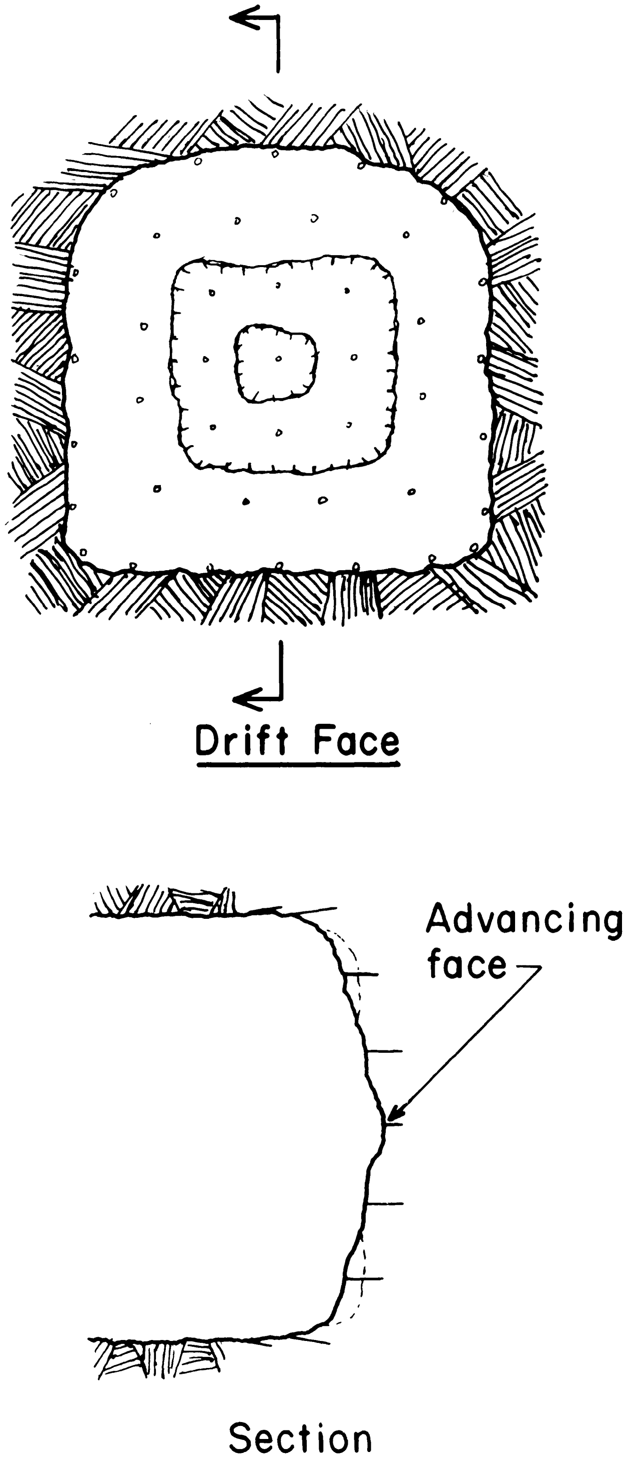

Fig. 22.9.12. Stepped-advance mining sequence (Anderson and Swanson, 1982).

angle. When the tool is operated in the radial mode, the rock spalls into the drift. Second, the operator can drill shallow holes perpendicular to the wall and pull the rock in the tool's normal radial-axial mode. Both methods work better if the face has advanced one or more rounds beyond the trimming activity.

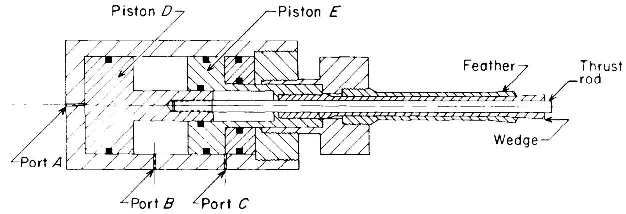

Fig. 22.9.13. Radial-axial splitter detail (Anderson and Swanson, 1982).

Operators can advance the face by following a prescribed pattern of holes. Or they can judge individual hole positions for the most efficient breakage, depending on the nature of the preceding breaks. New operators and automated systems require planned drilling patterns. To accommodate local geologic variations and varying quality of breaks, the planned drillholes will be spaced closer together than may be necessary. Experienced operators will develop the ability to position holes visually. They will be able to use fractures and bedding to help break the rock.

EQUIPMENT. The radial-axial splitter consists of a special hydraulic cylinder containing two independent pistons and the in-hole rock-breaking parts. The latter include the feather, wedge, and thrust rod (Fig. 22.9.13). The design of the rock-breaking parts is critical to successful operation of the splitter.

The wedge, concentrically located between the thrust rod and the feather, resembles a thick-walled tube with a cone-shaped forward end. Attached to one of the cylinder's two pistons, this wedge is drawn into the feather when the cylinder retracts its piston.

The feather is divided axially into four equal segments that expand radially as the wedge is withdrawn. It, too, has a cone-shaped forward end that has been constructed with various angles. In laboratory tests, this angle affects the operator's ability to anchor the tool in rocks of different strengths (Anderson and Swanson, 1987).

Relative hardness and brittleness of the feather and wedge must be balanced. A soft wedge combined with a hard feather wears very rapidly, and the wedge is soon rendered useless by excessive material creep. A hard wedge with a hard feather may cause brittle failure of either piece at stress concentration points. If the feather is too soft, its contact with the rock wears rapidly, eventually affecting its ability to hold. A reasonable compromise seems to be a wedge hardness of 56 Rockwell C and a feather hardness of 50 Rockwell C.

The thrust rod is attached to the cylinder's other piston. When the cylinder pushes this rod against the bottom of the hole, it produces an axial load in the rock mass.

Typically, the hydraulic system used to power the splitter operates up to 3000 psi (20,000 kPa) pressure. A splitter weighing just 80 lbm (36 kg) that works in a $1\frac{5}{8}$-in (40-mm) drillhole can generate up to 71,000 lbf (320 kN) of axial and 345,000 lbf (1550 kN) of radial load. This tool combined with a jackleg can be hand manipulated, yet break out rock to depths exceeding 7 in. (18 mm). Such a splitter typically produces about one-quarter ton (one-quarter t) of rock per break. A larger splitter that works in a 3-in. (76-mm) drillhole can generate up to 270,000 lbf (1200 kN) of axial and 650,000 lbf (2900 kN) of radial load. This tool by comparison can extract 4 tons (4 t) of rock per break.

The large splitter and drill are boom mounted on an air-powered, crawler-driven chassis that provides a rugged, mobile platform. The chassis holds the drill and splitter so an operator can align the splitter precisely over the previously drilled hole.

Table 22.9.3. Drill-Split Operating Schedule

Drilling operation:		
Collaring and drilling (penetration rate 50 in./min or 1.27 m/min)	min/hole	0.35
Drill repositioning	do	.27
Drill retraction	do	.10
Splitting operation:		
Indexing (includes rotation to splitter, insertion of splitter, retraction of splitter, rotation to drill)	do	.30
Splitting	do	.10
Total time to drill and split one hole	do	1.12
Total time to drill and split one round (28-hole round worked by a 3-boom jumbo)	min/round	11.20
Jumbo repositioning operation: (10-min repositioning time, every fourth round)	do	2.50
Total time for round	do	13.70
Advance rate (calculated): (based on 6 working hours per shift)	ft/shift	26.28

Assumptions: Splitter operating in a 2.5-in. (62-mm) hole, 15 in. (380 mm) deep, breaking at a depth of 12 in. (300 mm).
Source: Anderson and Swanson, 1982.
Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

The operator aligns the splitter and drill with a rotary actuator and mechanical stops. Alignment is very important; the efficiency of the drill-split method relies on the rapid alignment of the splitter with the drillhole.

Table 22.9.3 provides an analysis of the time devoted to each step of the operation. For a 10- by 10-ft (3- by 3-m) blind heading, an advance rate of 26 ft (7.8 m)/shift should be expected (based on 6 hr actual working time per shift). This advance rate is from a system consisting of three drill-split units on separate booms. Two miners could operate such a system in one heading.

The rock produced by drill-split methods may be mucked by various methods. Miners may attach muckers that use gathering arms or similar loading systems to drill-split equipment to produce a continuous miner. Other mining scenarios lend themselves to separate excavation and mucking equipment.

Factors Influencing Use. The amount of confinement affects the efficiency of excavation with splitters. Confinement is influenced by proximity to the excavation's perimeter, topography of the excavated face, and nearness to preceding breaks. Laboratory evidence points to a considerable decline in the amount of rock excavated per break as the confinement increases. This is difficult to quantify and outside the control of the operator. The effect of confining pressure caused by proximity to the excavation's perimeter or free faces is clearly shown in the shape of the crater (Fig. 22.9.14). Under greater confinement, the tool is less efficient. With less confinement, efficiency increases.

The insertion angle is under the operator's control and influences the amount of rock broken by the splitter and the forces required to break it (Fig. 22.9.15). At shallow angles or near a free face, the radial-axial splitter works as a simple radial splitter. It can be used to trim, shape, or enlarge openings. As the angle of insertion increases to vertical, the relative axial force necessary to extract the plug of rock increases.

Splitting efficiency and the fluctuations caused by confinement should be presented in the context of specific energy. A 3-in. (76-mm) splitter operating in an underground limestone mine produced an average of 1.25 ton (1.1 t) per break. The 3-in. (76-mm) diameter drillhole required for the splitter's operation averaged 15 in. (380 mm) deep. The overall specific energy for the drill-split operation was 180 in.-lbf/in.^{3} (1200 kJ). This number is a marked improvement over the equivalent number for other mechanical cutting methods, which can require 1000 in.-lbf/in.^{3} (7 MJ).

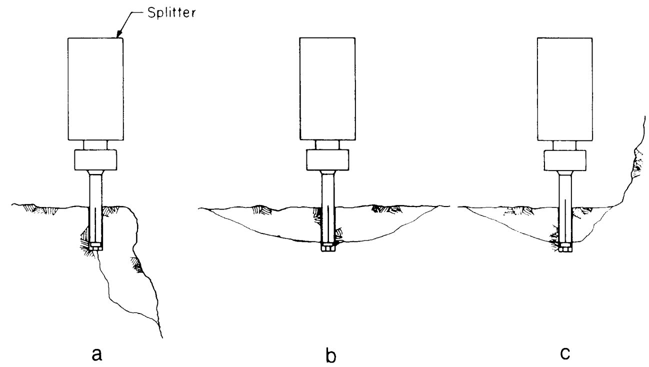

Fig. 22.9.14. Crater shape as affected by confinement. a) Bordered by a free face; b) Planer; c) Bordered by a confining wall (Anderson and Swanson, 1987).

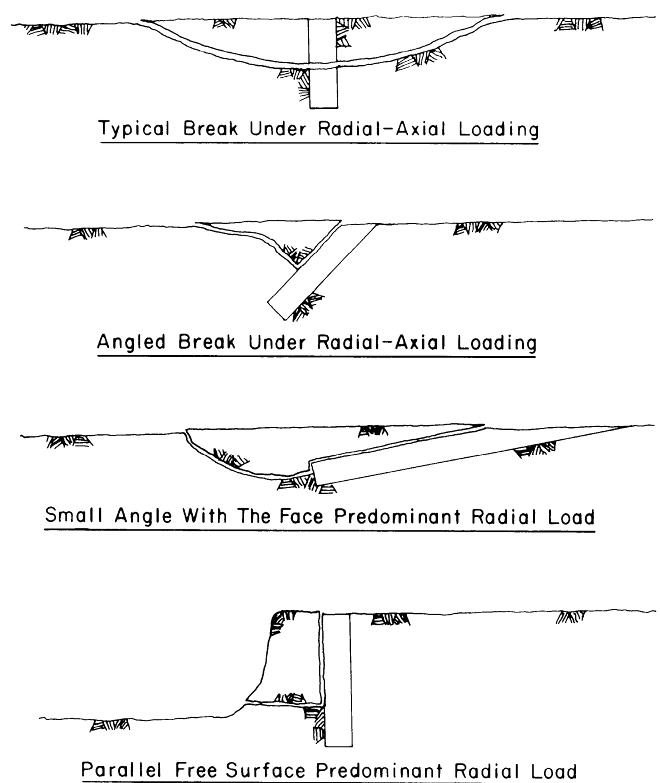

Fig. 22.9.15. Effect of insertion angle on crater shape (Anderson and Swanson, 1982).

Depth of break and rock toughness both exert influence over the forces necessary to excavate rock with the splitter. In tests on large limestone and granite blocks, the relationship between depth of break and axial force was defined by the following for limestone and granite, respectively:

$$A = 4{,}152D^{1.36} \quad (22.9.1)$$

$$A = 6{,}282D^{1.66} \quad (22.9.2)$$

where A is maximum axial force in lb, and D is depth of break in in.

The relationship between depth of break and radial force was defined for limestone and granite, respectively, as follows:

$$R = 13{,}134D^{1.40} \tag{22.9.3}$$

$$R = 24{,}212D^{1.60} \tag{22.9.4}$$

where R is maximum radial force in lbf.

These empirical relationships support the hypothesis set forth by Rhyming, Cooper, and Berlie (1980) that the breaking force is proportional to $D^{1.50}$.

Depth of break also governed the effectiveness of the splitter. For the limestone and granite tests at depths ranging from 1 to 7-½ in. (25 to 190 mm) and 0.75 to 4 in. (20 to 100 mm), respectively, the equation describing this effectiveness for limestone and granite was as follows:

$$W = 0.85D^{2.77} \tag{22.9.5}$$

$$W = 1.34D^{2.97} \tag{22.9.6}$$

where W is quantity of material removed in lb.

At any given depth, the volume of material removed is independent of the forces. The volume removed appears to be strictly a geometric consideration independent of the rock type. It can be approximated by the volume of a cone with a base radius 3½ times its height. The height equals the depth of break.

22.9.3.4 Costs

To help analyze the costs of the radial-axial splitter, USBM engineers estimated production rates for driving a 10- by 10-ft (3- by 3-m) opening. The excavation system consisted of three drill-split units on one chassis, run by two operators. The analysis assumed that 15-in. (380-mm) deep holes would be drilled for the splitters. It also assumed that operators would anchor the splitters to break the rock at an average depth of 12 in. (300 mm). About 28 breaks and 13.7 min were required to advance the full face 12 in. (300 mm). After every fourth advance of the face, the operator moved the chassis forward 4 ft (1.2 m). Based on 6 hr actual working time available per shift, the advance rate per shift averaged 26 ft (7.9 m). This translates to an excavation rate of 27 tph (24 t/h) (Anderson and Swanson, 1982). Table 22.9.4 provides an analysis of capital and operating costs for the radial-axial splitter in 1982 dollars. Inflating to 1988 dollars gives an ownership cost of $29.25/hr and total operating costs of $78.00/hr. The inflated cost becomes $4.26/ton ($4.70/t).

The radial-axial splitter cost $10,000 (1988 dollars) to build, and the entire system including the splitter, indexer, and crawler modifications cost under $25,000 (1988 dollars).

22.9.3.5 Case History

The USBM tested a radial-axial splitter in the Linwood Mining and Minerals Corp. underground limestone mine near Davenport, IA. Engineers operated the splitter in a blind heading on the lower level, 170 ft (51 m) below the surface. The roof was 23 ft (7 m) high, and the 40-ft (12-m) rooms were separated by 30-ft (9-m) pillars. The company continued normal mining operations during the testing.

In this field test, the USBM used the previously described crawler-mounted radial-axial splitter that operates in a 3-in. (76-mm) diameter drillhole. This tool and an air-powered rotary percussive drill were mounted together on a chassis. The chassis contained an indexing mechanism to permit the operator to rapidly align the splitter with the hole.

A diesel-driven compressor that generates 750 cfm (0.35 m^3/s) of air at 100 psi (700 kPa) produced the air needed by the machine. The splitter power was generated by a 2-hp (1.4-kW), air-driven hydraulic pump that was added to the crawler unit.

Operating through about 100 cycles, it mined approximately 125 tons (113 t) of ore. The average depth of break was 10 in. (250 mm), and loads required to fracture the rock averaged 65,000 lbf (250 kN) for the axial and 170,000 lbf (760 kN) for the radial modes. The 1¼-ton (1.1-t)/break average was larger than that predicted by the empirical relationships developed in the laboratory. Most rock pieces produced by the tool weighed from 20 to 300 lbm (9 to 140 kg).

The heading was mucked with the mine's normal mucking equipment, front-end loaders with 6-yd^3 (4-m^3) buckets. Miners mucked the face when it became difficult to work over the muckpile. During mucking, miners pulled the crawler out of the heading.

The field operation erased concerns about the tool's ability to excavate fractured and fissured rock and its ability to completely free the broken pieces from the rock mass. Occasionally the splitter lost its grip on part of the fractured rock plug when a majority of it fell away. If that happened, a normally spaced, subsequent break removed the remaining rock with no difficulty.

As a part of this test, the USBM completed a time study for the drill-split method. USBM researchers classified the drill-split operations into four categories: drilling, indexing (including all the time taken to replace the drill in the hole with the splitter), splitting, and repositioning the machine. The average times devoted to these operations were as follows:

Operation	Time (sec)
Drilling	50
Indexing	15
Splitting	150
Repositioning	60
Total	275

These operations averaged a total of 275 sec for the entire drill-split cycle with splitting consuming the most time. The splitting operation was slower in these tests than would be expected in commercial operations because an undersized hydraulic pump was used to power the tool. Splitting speeds of 15 sec are reasonable, and at such a rate, the average total time consumed would be reduced to 140 sec. In laboratory tests, the speed of the splitting process did not affect the quantity of rock excavated. The best total time achieved during the field test was 206 sec for a 10-in. (250-mm) break depth in which the splitting operation consumed 68 sec. The USBM believes that with operator experience and upgraded equipment, the average total time can be reduced further, to about 80 sec.

The USBM conducted the entire test in a blind heading. Holes were drilled with water flushing to control the dust. The splitting operation produced almost no dust. Although falling rock produced small amounts of dust, it caused no problem for the operators. This underground field test proved that the radial-axial splitter is an efficient primary rock excavator. Operators experienced little difficulty learning to operate the machine in these underground mining conditions.

22.9.3.6 Future Trends

Engineers anticipate that a variety of sizes will be manufactured for different excavation requirements. The tool may be

Table 22.9.4. Drill-Split Cost Schedule (1982 Dollars)

Machine designation:	
Assumed price (FOB-factory)	\$350,000
Freight charges at $\frac{\$11}{\text{cwt}} \times 430 \frac{\text{cwt}}{\text{unit}}$	4,730
Delivered price (assumed price + freight charges)	354,730
Tire cost at \$440/tires × 4 tires/unit	1,760
Salvage value at 10% delivered	35,473
Depreciable value	319,257
Ownership costs:	
Average investment (delivered price) $\$354{,}730 \times \frac{5+1}{2(5)\text{ yr}} \times \frac{\text{yr}}{4{,}000\text{ hr}}$	\$53.21 /hr
Depreciation (straight line) 5 yr at 4,000 hr/yr $\$319{,}257 \times \frac{1}{5\text{ yr}} \times \frac{\text{yr}}{4{,}000\text{ hr}}$	15.96
Interest, insurance, taxes \$53.21/hr × 0.20	10.64
Total Ownership Cost (depreciation + interest, insurance, taxes)	26.60
Operating costs:	
Fuel (diesel) \$0.87/gal × 2.5 gal/hr	2.18
Lube and preventive maintenance \$2.18/hr (fuel cost) × 0.45	0.98
Maintenance and repair $\frac{\$350{,}000\text{ (list price)}}{5\text{ yr (depreciation)} \times 4{,}000\text{ hr/yr}} \times 0.6$	10.50
Tire cost $\frac{4\text{ tires (\$440/tire, new)} + 4\text{ recaps (\$260/recap))}}{3{,}200\text{ hr}}$	1.85
Bits and steel \$0.26/ft × 150 ft/hr	39.00
Total Operating Cost	54.51
Labor cost:	
Operator cost (wages and fringes) \$14.15/hr × 2 workers	28.30
Cost per ton:	
$[\$26.60\text{/hr (ownership)} + \$28.30\text{/hr (labor)}] \times \frac{8\text{ hr}}{\text{shift}} \times \frac{\text{shift}}{26.28\text{ ft adv}} \times \frac{1\text{ ft adv}}{8.25\text{ tons}} +$	
$\$54.51\text{/hr (operating)} \times \frac{6\text{ hr}}{\text{shift}} \times \frac{\text{shift}}{26.28\text{ ft}} \times \frac{1\text{ ft}}{8.25\text{ tons}} =$	\$3.53/ton

Source: Anderson and Swanson, 1981.
Conversion factors: 1 ft = 0.3048 m, 1 ton = 0.9072 t, 1 gal = 3.785L, 1 lb = 0.4536 kg.

used singly or in multiple units. It break rocks nonviolently with little impact on the surrounding rock. Drill-split fragmentation is a flexible excavation method that can be adapted to meet a broad range of mining requirements.

The radial-axial splitter seems ready for commercial application. First uses in mines will probably concentrate on special underground construction projects. Rock excavation operations in heavily populated zones that preclude or restrict blasting also seem ideal early applications. Development openings in rock especially susceptible to blast damage provide yet another excellent opportunity to use the splitter.

22.9.4 THERMAL FRAGMENTATION AND EXCAVATION

PETER G. CHAMBERLAIN AND DAVID P. LINDROTH

22.9.4.1 Concept

Although mechanical mining and tunneling machines have dominated research on methods of extracting rock without blasting, *thermal fragmentation systems* hold promise for certain applications. Attempts to excavate rock with greater than 20,000 psi (140,000-kPa) compressive strength using continuous mechanical mining machines have met with marginal success. Commercially available machines excavate rock primarily with high compressive forces that crush the material at the rock-bit interface. Cutting hard rock with such machines requires excessively large forces. Bits wear and break rapidly under these conditions. Methods previously described in this chapter introduce the concept of fracturing rock in tension during excavation. Thermal methods present an alternate way to induce tensile stresses in the rock as a primary excavation technique or to augment mechanical excavation.

Although early experiments showed that thermal energy broke rock inefficiently, recent developments indicate more efficient applications. One such development is *microwave heating.* Microwave heating can fragment hard rock. It can also weaken rock to improve other mining or secondary breakage methods. Microwaves can penetrate rock and, as electromagnetic energy is converted to heat by rapid vibration of molecules at a location inside the rock, can create a thermal inclusion (hot spot) inside the rock. The hot rock then expands according to its specific thermal expansion, whereas the surrounding cool rock does not. The resulting swelling of the hot spot inside the rock causes tensile stresses in the surrounding rock, eventually fragmenting it. With rapid heating at high energy levels, microwave heating can fracture the rock sufficiently for mucking. The rock can also be heated with lower energy to induce just enough fractures to make it easy for mechanical methods such as a ripper miner machine (see 22.9.2) to complete the excavation.

Additional applications of high temperature could appear in future mines. Laser technology allows rapid kerfing that could make traditional mechanical excavation more efficient. With continuous miners and tunneling machines, the mechanical kerfing tools are subjected to extremely high wear because the rock is confined. A thin kerf, such as could be produced by lasers, relieves the confining forces. This allows the machine to cut the peripheral rock as easily as it cuts the interior rock.

Microwave researchers have fragmented extremely hard rocks in laboratory settings (Thirumalai, 1970). Technical feasibility is thus not a question. Before the methods can move to commercial application, however, researchers must complete considerably more fundamental testing and evaluation.

Spin-off microwave heating technology is also showing considerable promise. Microwave heating efficiently dries coal and limestone (Lindroth, 1986a). Microwaves could reduce metallurgical processing time and cut costs by heating ore particles in a shorter time than fossil fuel heating now does (McGill, Walkiewicz, and Smyltes, 1988).

22.9.4.2 Background

Mining with heat dates to the earliest prehistoric societies. The first hard-rock metal miners in this country mined copper from northern Michigan 2000 years ago. Their technique was simple; they built fires in mine pits. When the surrounding rock was hot, they doused it with cold water. They then beat the fractured ore with mauls to free it. The hundreds and perhaps thousands of mine pits dotting the region when the first Europeans arrived offered silent tribute to their success and industriousness.

Miners no longer heat rock to excavate it. Energy provided by chemicals (blasting) and mechanical methods break large volumes of rock efficiently and economically. Special problems, however, have periodically inspired research on ways to use high temperature to break rock. Such problems usually involve excavating very hard rocks that cannot be readily excavated with mechanical machines and where blasting is not desired.

The early research on excavating rocks by applying high temperatures focused on surface applications of heat. A rock surface was heated with high-temperature sources such as fossil-fuel torches, ionized-plasma torches, jet piercers, electron beams, and lasers. Heat was transferred to the interior of the rock by the slow process of conduction. The resultant thermal stresses caused some rocks to spall (those high in quartz, for instance) but most melted.

The high amounts of energy consumed to melt rock (without yielding appreciable thermal stresses) and the difficulty in removing molten rock from a face eliminated surface heating as a viable excavation technique for rocks that did not readily spall. A few applications for surface heat did reach operating mines. Miners “drilled” blastholes with jet piercers for several years in the extremely hard taconite ore of northern Minnesota. The high energy costs, however, doomed jet piercers when better rotary bits were developed for such rock.

Researchers began considering methods of internally heating rock to induce tensile stress without wasting energy by heating the surface. USBM and Soviet researchers investigated *high-frequency dielectric heating* for internally heating rocks that did not readily spall (Thirumalai, 1970). The first USBM experiments were conducted on rock blocks by inserting an array of copper electrodes. The thermal inclusion developed inside the electrode array. Blocks of basalt and granite fractured with a thermal inclusion that represented less than 2% of the rock volume. This fragmentation consumed 4 to 8 kWh/m^3 (14 to 29 J/m^3). Although the authors proposed a stoping method where large thermal inclusions would be created in the rock, no practical way of creating the inclusions using direct-contact electrodes was ever advanced.

The USBM is now investigating methods of dielectrically heating rock remotely using microwaves. Microwave energy is transferred to rock by a radiant-energy beam. Recent studies showed that the dielectric heating more effectively fractures rock at higher frequencies than those used in the early studies. USBM researchers presently conduct microwave research with a microwave source of 25 kW. It operates at a frequency of 2.45 billion cps (2.45 GHz).

Lasers have evolved over the past several decades for many industrial applications. High-power CO_2 lasers are routinely used to cut, weld, and machine metals. USBM research (Lindroth, 1984) has provided several possible applications of CO_2 lasers in mining. Lasers may be used to rapidly spall specific rock types such as quartzite or for scribing and lettering of building stone and monuments. One promising application is to cut a deep kerf in a rock face and then break out the rock inside the kerf by other means. For example, a laser could be used so that the beam would act as a gage cutter in a hard-rock tunneling system.

22.9.4.3 Possible Mining Operations

Electromagnetic energy in the form of microwaves offers several possible mining applications. Research and development efforts are still defining basic parameters, and specific systems for excavating rock have not been developed. Experience with fracturing rock using microwaves and the acquisition of the necessary engineering parameters, however, permits researchers to establish likely operating guidelines.

The core of any microwave excavation system will be the microwave generation unit. Microwaves generated at a frequency of 2.45 billion cps (2.45 GHz) with a power rating of 25 kW fracture large blocks of rock in laboratory settings and

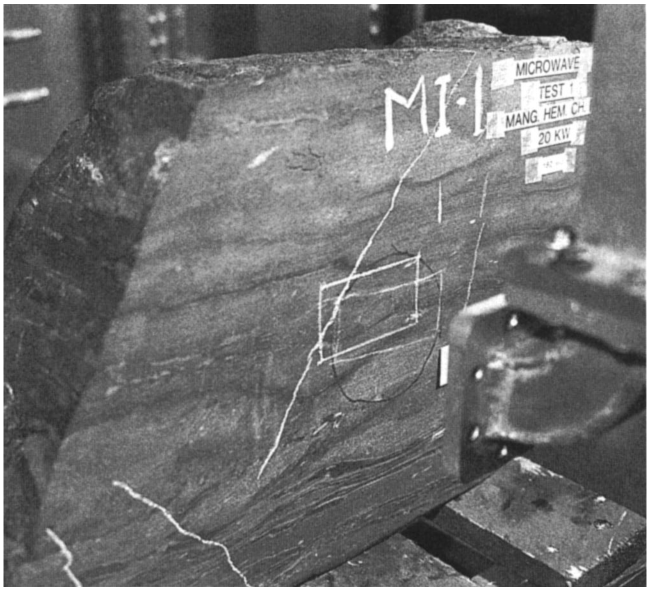

Fig. 22.9.16. Block of manganiferous iron ore in microwave irradiation laboratory test facility.

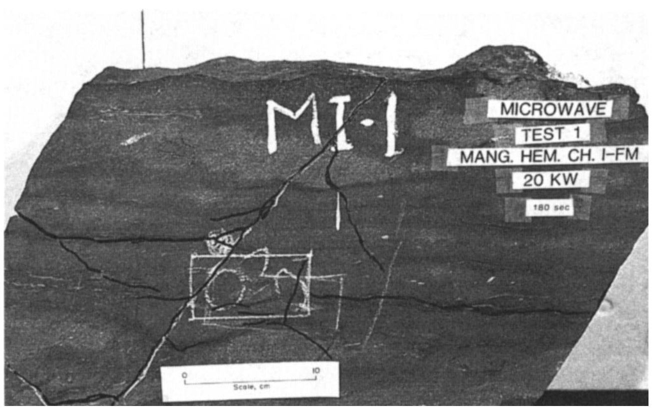

Fig. 22.9.17. Fracturing created in block of manganiferous iron ore after microwave irradiation for 180 sec.

represent a reasonable guide for size. Fig. 22.9.16 shows a 180-lbm (82-kg) rock being irradiated by this microwave device. Note the stand-off distance of about 6 in. (150 mm) in front of the open waveguide for the test. Fig. 22.9.17 shows the 180-lbm (82-kg) block of manganiferous cherty iron formation (Cuyuna Range) fractured after irradiation for a little over 20 sec. The black lines are the microwave-induced fractures, and the white are mineralization lines.

The microwaves penetrate the rock to a depth that is determined by the frequency of the electromagnetic wave and the rock's dielectric and magnetic properties. The objective is to rapidly create a hot inclusion inside the rock at a sufficient depth from the surface. The stresses created at the contact between the boundary of the inclusion and the surrounding cold rock are governed by the following relationship (Thirumalai, 1970):

$$\sigma = \mu\, E\, a\, T/(1\text{-}p) \tag{22.9.7}$$

where σ is stress, μ is a constant related to the shape of the inclusion and varies from 1 to $\frac{2}{3}$, E is Young's modulus, a is an expansion coefficient, T is temperature rise for the inclusion, and p is Poisson's ratio.

Thus, the stresses are independent of the size of the inclusion. Since the stresses induce tensile and shear fractures, the ability of microwaves to fracture rock also does not depend on compressive strength. This unique feature allows microwaves to fracture extremely hard rock not amenable to mechanical excavation.

The ability of the thermal inclusion to fracture rock grows as the temperature of the inclusion increases up to a maximum temperature. The internal temperature at which the stresses are maximized vary with rock type. Thirumalai (1970) found that for basalt, this maximum temperature averages about 930°F (500°C); for granite it averages 570°F (300°C). At higher temperatures, the Young's modulus falls and this rock softens, thus outweighing any benefit from a higher temperature rise. The hotter inclusion essentially dissipates the stresses without fracturing the rock.

Researchers have accumulated enough evidence to support the technical feasibility of fracturing many very hard rock types with microwave energy. Until further research establishes the engineering parameters of the optimum frequency, power, and energy density, the design of an economical primary excavation system remains conjectural.

A more energy-efficient application of microwaves may be to soften hard rock in advance of mechanical excavation. In this application, the rock is irradiated with microwave energy immediately ahead of the mechanical bit. As described above, the microwaves develop thermal stresses inside the rock, which generate fractures. As the bit passes through the rock, the rock is excavated with a lower force because of the network of thermal-generated fractures. Preliminary experiments revealed a 70% reduction in cutting force on drag bits when the rock was irradiated with microwave energy. USBM researchers are still studying this approach. They hope to develop data on proper combinations of mechanical cutting mode, cutting tool configuration, and microwave energy that will reduce mechanical cutting forces and energy consumption.

Research on the use of high-power lasers in mining showed a promising concept featuring the laser as a kerfing device. By cutting a deep kerf in hard rock, an operator could more easily break the rock between kerfs with mechanical or other primary excavation methods. For example, a laser beam could act as a gage cutter on TBMs (Carstens, 1972). Laser beams can make thin deep cuts by melting and vaporizing rock. Lasers easily cut kerfs in even the hardest rocks. High-energy lasers readily penetrate rocks with compressive strengths up to 73,000 psi (500 MPa). Even relatively low-power CO_2 lasers of a few hundred watts can create narrow cuts by spalling in certain rock types. For example, the USBM measured the mass of rock removed by a 100-W laser beam as a function of the power density (up to 570 W/mm^2) and beam scan rates for a quartzite (Fig. 22.9.18). The highest spalling rate was observed at a power density of only 2.5 W/mm^2 and a beam scan of 0.14 in./sec (3.5 mm/s).

For lasers up to 5-kW output (Carstens, 1972) and scanning speeds in the range of 0.83 to 3.35 in./sec (21 to 85 mm/s), the depth of penetration is linearly proportional to the beam power. Penetration depth is inversely proportional to the scanning speed. Within this range of scanning speeds, the product of speed times depth of cut was a constant for a given laser-beam output. Researchers expect that full-scale laser cutting tools will operate with 15-kW output.

These high-power lasers appear well suited as gage cutters for large tunneling machines in rock with compressive strengths over 40,000 psi (275,800 kPa). Present mechanical gage cutters

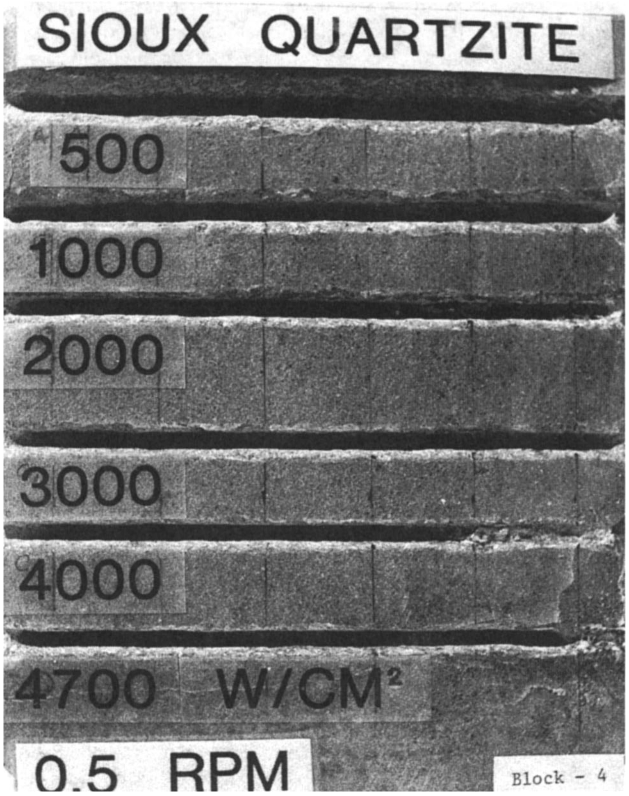

Fig. 22.9.18. Slots in quartzite created by scanning with 100-watt CO_2 laser beam at various power densities.

operate with a component of thrust acting radially outward; they must cut both the face and the wall of the tunnel. This complex loading shortens their life considerably. Much machine downtime is attributed to rapid wear of gage cutters. A single, highly focused laser beam, rotating at the same speed as the interior mechanical cutters, could cut a deep kerf and eliminate the need for gage cutters. Laser light remains coherent for long distances so that the laser would be located in a protective housing at a remote distance from the cutting head.

As an example of this application, a 10-ft (3-m) boring machine rotating at 9 rpm can penetrate 5 fph (1.5 m/h) in 40,000-psi (275,800-kPa) rock. This rotation causes a gage cutter linear velocity of 4.7 fps (1.436 m/s). To maintain a kerf depth corresponding to the advance rate achieved by the machine, a laser beam would have to cut a kerf 0.1 in. (3 mm) deep at that velocity. This is now possible with the experimental 4-kW laser previously described. With 15-kW lasers, such a kerfing system may be economically viable.

22.9.4.4 Costs

Microwave generators presently cost $2000 to $3000/kW. High-powered, CO_2 laser generators presently cost about $40,000/kW. As the high-power laser industry matures, the costs can be expected to drop considerably.

22.9.4.5 Future Trends

Key information to be determined before thermal fragmentation methods are applied to mining includes matching microwave frequency and power density for efficient fragmentation. As the fundamental data on frequency and power density become available, additional effort will be necessary to conceive practical means of applying the energy to the rock.

Thermal fragmentation suggests some interesting potential applications in the mining and minerals industry. Microwave energy applications in fragmenting and processing could blossom as equipment costs are reduced and efficiencies are increased. Many state-of-the-art mechanical excavation techniques may be enhanced by combining them with electromagnetic energy.

22.9.5 SHOCK WAVE ROCK BREAKERS

PETER G. CHAMBERLAIN AND GEORGE A. SAVANICK

For many years, engineers have known that *shock waves in water* cause considerable damage to structures. Nearby exploding ordinance, for example, can damage ships and submarines even without a direct hit. At least two devices are sold to break rock based on shock waves generated in water. One uses shotgun cartridges to create the shock wave, the other high-pressure gas cylinders.

22.9.5.1 Shotgun Cartridge Rock Breaker

CONCEPT. Hudco, Inc., Mentor, OH, markets a pulse breaker called the *water wedge* to break oversized rock and demolish concrete structures. To use the water wedge, an operator drills a hole in the rock to be broken and fills it with water. The tool is then inserted in the hole and charged with a shotgun cartridge. The cartridge is fired remotely with a rope lanyard. The explosion generates a stress wave in the water column. The resulting high pressure splits the rock in tension.

OPERATION. The pulse breaker is designed to break large boulders or oversize muck as well as demolish concrete structures where blasting is not desirable. To prepare an oversized rock for the pulse breaker, an operator drills a 1⅝-in. (40-mm) hole 15 or 21 in. (390 or 540 mm) deep. The operator then fills the hole with water using any convenient container.

The operator inserts the lower end of the breaker into the hole and charges the breaker with a 10- or 12-gage shotgun cartridge containing smokeless powder. The operator then retreats to a safe position and fires the cartridge using a 60-ft (18-m) rope lanyard. When the cartridge fires, it drives a plug down a transfer tube in the water-filled hole. The resultant high pressure (40,000 psi, or 275,800 kPa, with the 12-gage model) in the water transfers high tensile stresses to the walls of the hole. The rock breaks in tension along fracture planes parallel to the axis of the hole (Fig. 22.9.19).

The pulse breaker can break hard rocks having up to 62,000-psi (427,490-kPa) compressive strength. Its simple operation enables even unskilled workers to operate the breaker with little training.

EQUIPMENT. The water wedge pulse breaker is a portable, hand-held tool. It consists of a cartridge holder, firing mechanism, and downhole transfer tube assembly packaged into a unit weighing 22 lbm (10 kg). The operator may choose either a 15- or 21-in. (380- or 530-mm) tube assembly, depending on the size of rock typically split. Hole depth must be matched with tube length.

The stem of the transfer tube contains holes pointed in a radial direction. When the cartridge fires, the high pressure in the water forces the sleeve against the wall of the hole. This prevents the tool from being ejected from the hole.

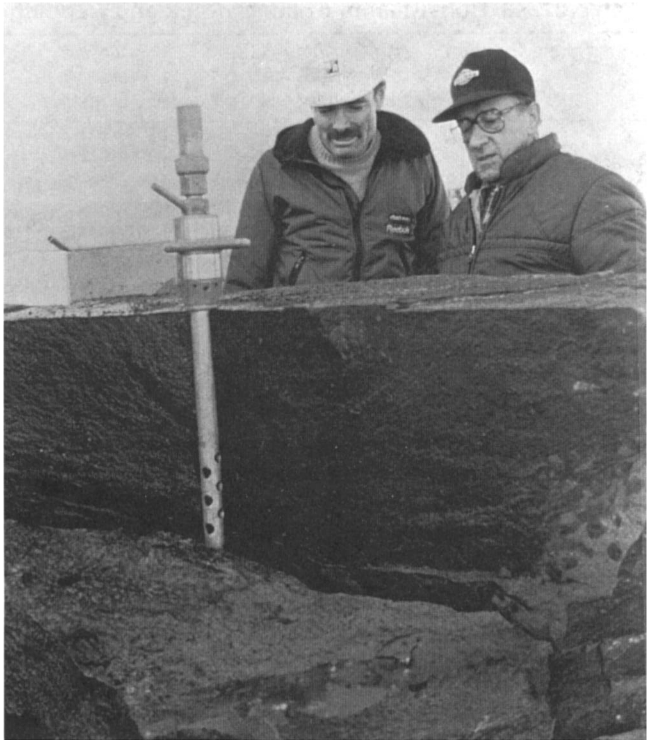

Fig. 22.9.19. Dresser basalt broken by pulse from 12-gage cartridge.

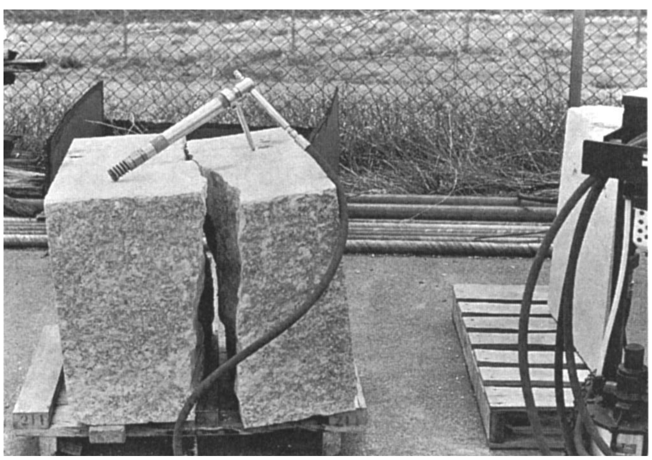

Fig. 22.9.20. Block of dolomite broken with a pulse from a water-charged accumulator.

22.9.5.2 Cheney Rock Breaker

CONCEPT. The *Cheney rock breaker* is another apparatus that splits rocks with high-pressure water pulses. First patented in Australia (Cheney, 1979), it breaks rock in a controlled manner from one or more holes drilled in rock. The operator inserts the rock breaker tool in a hole, locks it in place, and fills the hole with water through the tool's hollow stem. The operator then applies a sudden surge of pressure to the water by releasing a valve connected to a high-pressure accumulator. The pressure pulse in the drillhole splits the rock (Fig. 22.9.20). A mechanical feather/wedge assembly accentuates the pressure and directs the forces so that the operator can orient the fracture plane.

Cheney developed the rock breaker to overcome the problems of flyrock and eliminate the other hazards of blasting by controlling fracture. The method is particularly applicable to breaking oversize rock remaining from typical quarry blasting. Most quarries break oversize rock by detonating a blasting agent placed on the rock's surface. Such methods may cause flyrock and considerable air-blast noise. The Cheney rock breaker operating in one hole in a boulder represents one commercially available alternative.

The rock breaker can also presplit rock masses by using it in a line of holes. The operator drills a line of holes along a desired fracture plane. The tool, operated in successive holes, splits the rock along the plane containing the holes.

MINING OPERATIONS. To split rock with the rock breaker, an operator first drills one or more holes into the rock to be split and then inserts the downhole portion of the tool into the hole. The downhole portion consists of a hollow stem, expanding collet, and seal. By tightening a nut, the operator expands the seal and the split collet to secure the tool in the hole. Water is added through the hollow stem to fill the hole. The operator then applies a modest pressure to the water in the hole before sealing it. The unit is then charged by pressurizing a water-filled accumulator connected to the hollow stem. When the operator suddenly releases this higher pressure to the water in the hole through the hollow stem, the pressure pulse fractures the rock.

The pressure pulse in water rapidly transmits force to the rock. Since the water is nearly incompressible, however, the pressure almost instantly drops when the fracture initiates. The fracture thus propagates with much more control than fractures generated by conventional blasting practices.

To presplit a rock mass via a line of holes, the operator sequentially activates the rock breaker in each hole. A feather and wedge assembly forces the collet against the wall of the hole with the same mechanism as contained in a rock bolt anchor. The rock fractures along the plane of the split in the collet.

EQUIPMENT. The Cheney rock breaker is built by Almet Enterprises, Pty., and distributed by Laundry Investments, Pty., both of Australia.

The downhole assembly fits in a $1^1/_2$-in. (37.5-mm) hole. It consists essentially of a hollow tube or stem, a split expandable collet, and a seal. A bleed valve on top permits the operator to fill the hole with water so no air bubbles are trapped inside. A ratchet attached to a large nut outside the hole squeezes the seal until it securely anchors in the hole.

The power unit, connected to the downhole unit with a high-pressure hose, is completely portable. It consists essentially of a compressed air supply, pressure intensifier, and an accumulator half filled with water. These components are mounted on a two-wheeled cart. The pressure intensifier builds compressed air pressure in the upper half of the accumulator. When the operator releases a ball valve on the lower end of the accumulator, the accumulator discharges a pressure pulse of about 350 bar to the water in the hole. This pressure pulse generates about 90,000 lbf (400 kN) of radial force/foot (0.3 m) of hole.

CASE HISTORY. The Cheney rock breaker successfully split huge granite blocks during restructuring of a dockyard in the port of Gibraltar. Shand, Ltd., used the rock breaker while refurbishing an existing naval dockyard for commercial shipping. The company was hired to remove the four large steps on the walls of the dock against which vessels were previously moored. The granite steps were constructed in the 1890s. They extended for a length of 500 ft (150 m). Shand was not permitted to blast the steps. Other forms of demolition cost too much and consumed excessive time.

To remove the four granite steps, the contractor drilled a line of holes on 12-in. (300-mm) spacings over the length of the

Table 22.9.5. Design Production from Lunar Mining Operations for Eight-Man Lunar Base

Product	Amount tpy (t/a)
Oxygen	165 (150)
Hydrogen	16.5 (15)
Carbon	6 (5.4)
Other volatiles	Capture all available

Source: McKay, 1990.

Table 22.9.6. Constituents of Lunar Rock and Regolith

Lunar Maria (Basalt Lavas) (*Apollo 11* Site)	Weight % as Oxides		Weight % as Elements	
Silicon (Si)	SiO_2	41.3	Si	19.3
Titanium (Ti)	TiO_2	7.5	Ti	4.5
Aluminum (Al)	Al_2O_3	13.7	Al	7.3
Iron (Fe)	FeO	15.8	Fe	12.3
Magnesium (Mg)	MgO	8.0	Mg	4.8
Calcium (Ca)	CaO	12.5	Ca	8.9
Sodium (Na)	Na_2O	0.41	Na	0.30
Potassium (K)	K_2O	0.14	K	0.12
Manganese (Mn)	MnO	0.21	Mn	0.16
Chromium (Cr)	Cr_2O_3	0.29	Cr	0.14
Oxygen (O)	—	—	O	42.2
TOTAL		99.9		100.0
Lunar Highlands (Anorthosites) (*Apollo 16* Site)	**Weight % as Oxides**		**Weight % as Elements**	
Silcon (Si)	SiO_2	45.0	Si	21.0
Titanium (Ti)	TiO_2	0.29	Ti	0.17
Aluminum (Al)	Al_2O_3	29.2	Al	15.5
Iron (Fe)	FeO	4.2	Fe	3.3
Magnesium (Mg)	MgO	3.9	Mg	2.4
Calcium (Ca)	CaO	17.6	Ca	12.6
Sodium (Na)	Na_2O	0.43	Na	0.32
Potassium (K)	K_2O	0.06	K	0.05
Manganese (Mn)	MnO	0.06	Mn	0.05
Chromium (Cr)	Cr_2O_3	0.08	Cr	0.04
Oxygen (O)	—	—	O	44.6
TOTAL		100.8		100.0

Source: Taylor, 1982.

steps and halfway in from the outer step. The Cheney rock breaker was activated in alternate holes, generating a hairline fracture. After the holes were drilled, operating the rock breaker consumed only a few minutes per hole. The contractor easily removed the steps with an impact hammer.

FUTURE TRENDS. Shock wave breakers offer a safe, portable, and inexpensive alternative to blasting in certain mining and quarrying operations. They seem ideally suited to breaking oversize rock in quarries, a duty frequently assigned to relatively inexperienced miners. They also function well in certain presplitting applications.

The advantages suggest that shock wave breakers will appear more frequently in open-pit mines and quarries. As their popularity expands, a broader range of configurations should become available.

22.9.6 LUNAR AND PLANETARY MINING

PETER G. CHAMBERLAIN AND EGONS R. PODNIEKS

22.9.6.1 Concepts

Mining in space, once strictly the realm of comic strips, now looms as a distinct possibility. *Astro-miners* could be mining on the Moon or on Mars and its moons by early in the 21st century. Two critical needs have drawn mining into future space planning—materials to support space missions and commercial exploitation of materials unavailable on Earth.

In 1989, the National Aeronautics and Space Administration (NASA) developed a new mission—the Space Exploration Initiative. Its goal would be to establish a sustained, self-sufficient presence in space. The effort would focus first on a permanent lunar outpost, with expansion to Mars and its moons. This outpost or base would require local life support and construction materials. The base would also function as a refueling station for extended space travel, requiring locally produced propellent components.

MATERIAL NEEDS. A prime requirement of a lunar outpost would be a liquid oxygen plant. Workers on the moon will need oxygen to breathe plus oxygen and hydrogen to make water. Initial oxygen requirements would average 165 tpy (150 t/a) for a 6- to 8-person crew (Table 22.9.5) (McKay, 1990). NASA estimates that 1100 tons (1000 t) of oxygen will be required each year to support full-scale lunar base activities. Lunar mining and processing operations can provide this oxygen and some hydrogen.

Extraterrestrial mining could provide other materials needed for outpost bases in space. Construction materials, for example, will be extremely expensive to transport from the Earth to the Moon. NASA contractors are evaluating the feasibility of preparing sintered blocks from lunar regolith. Liquid hydrogen and oxygen are the key ingredients of propellants for space travel. Astronauts could refuel at outpost bases with locally mined ingredients. Astronauts traveling to Mars would also benefit by being able to restock oxygen and water during a stop at the Moon. Without this capability, much of the payload contained in space launches from Earth must consist of life-support materials.

Beside mining and processing to support space missions, at least one possible commercial mining enterprise is being considered. Scientists are studying the fusion of deuterium and helium-3 as well as the more probable deuterium/tritium fusion for future US nuclear power plants. Apparent safety and efficiency benefits may favor fusion with helium-3. Unfortunately, very little helium-3 exists on earth. An extraterrestrial source is needed if deuterium/helium-3 fusion plants are to become a commercial reality. Helium-3 is deposited on the lunar surface by solar winds. Although the low concentrations of helium-3 on the Moon minimize chances of mining significant amounts of helium-3, at some future time the economics of energy supplies may warrant such production.

DEPOSITS. Two types of deposits offer hope for lunar mining. The first is oxide minerals in mare regions. Moon rock and regolith contain FeO, SiO_2, Al_2O_3, and TiO_2 (Table 22.9.6). Most attractive mining candidates for oxygen would be regions of mare containing concentrations of ilmenite, $FeO \cdot TiO_2$. Early mining operations for oxygen will likely focus on the loosely bound regolith. Although the regolith appears to contain 45% oxygen, only about 10% of the regolith is ilmenite. NASA scientists assume that the recoverable oxygen from these deposits would average about 3% (McKay, 1990).

The second type of deposit comprises the ions of hydrogen, helium, and helium-3 impinged upon the Moon's surface by solar winds. Solar winds are essentially ions of light gases—mostly hydrogen—streaming away from the sun. The ions deposited on

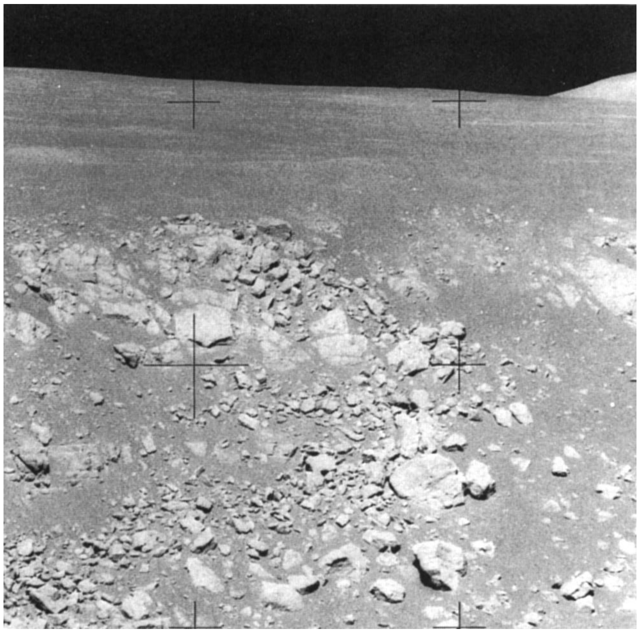

Fig. 22.9.21. Typical lunar regolith (*courtesy:* NASA, Houston, TX).

the Moon are more concentrated on the surface of small particles (less than 100 μm). Fortunately, approximately 80% of the soil is 8 to 125 μm (Wittenberg, Santarius, and Kulcinsky, 1986). Continual bombardment of the lunar surface by meteorites has stirred the particles to a depth of several meters. Mining operations would concentrate on the top 6 to 10 ft (2 or 3 m) of regolith (Fig. 22.9.21).

Hydrogen is the most abundant ion deposited by the solar winds. Scientists estimate that the regolith contains 20 to 200 ppm of hydrogen (Taylor, 1975). The hydrogen is easily driven off the surfaces of regolith particles at temperatures above 752°F (400°C). The big operational difficulty would be to trap the gases in the vacuum that exists at the moon's surface. For planning purposes, NASA scientists assume that recoverable hydrogen in the upper 10 ft (3 m) of regolith averages 50 ppm (McKay, 1990).

In regolith samples, the ratio of hydrogen to helium averages about 8.5:1. Although the ratio of helium-3 to helium is generally low, Apollo and Luna mission data indicate that the lunar regolith at some locations may contain 7 to 30 ppm helium-3.

22.9.6.2 Mining Activities in Space

MINING CONSIDERATIONS. Mining operations on the Moon or planets will be highly automated extensions of mining practices on earth. The most likely methods will be surface mining operations. Mining the unconsolidated regolith will essentially consist of digging and hauling material to a processing site (Register, 1989; Siekmeier and Podnieks, 1990). This mining will include the means to handle the boulders apparent in the regolith. Underground mining may evolve later if the benefits of protection from radiation, temperature extremes, and bombardment by solar wind and meteorites outweigh the lower productivity (Podnieks and Schmidt, 1990).

Fig. 22.9.22. Surface mining concept for lunar oxygen.

The lunar environment imposes severe restrictions on possible mining operations. Even the simplest tasks are difficult under the conditions present on the moon. Terrestrial mining equipment will not function well under these circumstances. Rather, completely new technology must be developed. Development of successful mining operations in space truly represents a monumental challenge.

OXYGEN MINING. Oxygen requirements have driven planning efforts to date. Although several processes for recovering oxygen have been proposed, the current thinking favors hydrogen reduction of ilmenite. The first mining operations on the moon would be to provide oxygen for an 8-person lunar base requiring a 165-ton (150-t) liquid oxygen extraction system. To annually recover 165 tons (150 t) of oxygen, miners would excavate 50,000 tons (45,000 t) of regolith per year.

Such amounts of "ore" could be provided from a surface mining operation. The mines would appear similar to sand and gravel operations on earth (Fig. 22.9.22). Since scientists believe that ilmenite is rather broadly dispersed in maria, astro-miners could select sites in easily accessible regions, relatively free from boulder fields.

For safety, miners would operate the equipment remotely from protective enclosures. They would go outside of shelter only to repair minor equipment malfunctions or to retrieve the equipment for major repairs.

Oxygen mining operations in regolith could also provide construction materials. For instance, tailings from the oxygen recovery process could be sintered into construction blocks. If tailings material does not sinter well, the bulk regolith could be mined, screened, and fed to a sintering process.

Underground mining operations could also provide oxygen for lunar base life support. These would be shallow, probably collared in escarpments at the edges of maria. Drifts in hard rock would allow miners access to draw down the loose regolith through a series of raises. Alternately, astro-miners could excavate ore in a simple room and pillar sequence with the drill-split tool described in 22.9.3. The USBM conceived an idea of excavating with a drill-split tool, then sealing the openings with microwave-induced melting to provide a safe working environment (Podnieks and Schmidt, 1990). Working in the drifts would protect the miners from radiation and micro-meteorite impacts. Temperature could also be controlled. Particularly if combined with construction activities, underground excavations for oxygen could also provide shelters and habitation sites for lunar base inhabitants.

HYDROGEN/HELIUM MINING. Considerably larger mining operations would be needed to recover sufficient hydrogen and/or helium than would be needed to recover oxygen. Mines for solar-wind implanted ions also may be configured differently than oxygen mines. Since the ions are concentrated in the upper few feet of the regolith, astro-miners would excavate only the upper 6 to 10 ft (2 to 3 m) of regolith.

For early activities on the moon, an eight-person base would require 16.5 tons (15 t) of hydrogen. At a hydrogen concentration of 50 ppm, workers would need to process 300,000 tons (270,000 t) of regolith to produce the required hydrogen. Miners would likely screen the regolith so only the material less than 100 μm would be processed. Since 80% of the regolith is less than 100 μm, miners would have to excavate an extra 25% of regolith. Mining operations thus must produce 375,000 tons (340,000 t) of regolith per year.

The retained regolith would be preheated and then placed in a solar-heated retort. At temperatures of around 1112°F (600°C), hydrogen, helium, carbon dioxide, carbon monoxide, and nitrogen trapped on the particles would be driven off and collected. Considerable study would be required to develop technology for collecting these gases under the lunar conditions of ultrahigh vacuum.

A fully operational lunar base, staffed by 15 to 20 people, would require about 130 tons (120 t) of hydrogen for water and for propellent (McKay, 1988). Such hydrogen production would require a mine producing 3 million tons (2.7 Mt)/yr of regolith. Sand and gravel operators on Earth commonly achieve such production rates. A crew of 15 to 20 people expected to perform other duties, however, will have a difficult time attaining these rates in the hostile lunar environment. The ability of lunar mining operations to provide adequate hydrogen for self-sufficiency may lag that of oxygen. Evidence of higher concentrations of hydrogen deposited on the surface of Mars and Phobos may make such mining operations more practical on those bodies.

Any commercial exploitation of helium-3 deposits on the Moon would require mining at a very large scale. One estimate of the helium-3 requirements predicts that the demand could reach 1 tpy (0.9 t/a) by 2030, 10 tpy (0.9 t/a) by 2035, and 200 tpy (180 t/a) by 2050. For 1 ton (0.9 t) of helium-3, about 120 million tons (110 Mt) of regolith must be mined (Wittenberg, Santorius, and Kulcinsky, 1986). The scope of operation needed for these requirements seems beyond those that will be available on the Moon early in the 21st century.

Although present efforts to develop mining concepts for space environments are based on conventional terrestrial technology, engineers will likely develop completely new concepts for recovering materials needed on the moon. At this time, such concepts are too speculative to include in this Handbook.

EQUIPMENT. Scientists have proposed every imaginable sort of mining equipment for lunar mining. Since thorough mining engineering studies have not been completed, however, any discussion on mining equipment is premature. Highly automated vehicles that dig, load, and transport the regolith to a storage pile would probably form the heart of the excavation system. They would function in a manner similar to scrapers or front-end loaders in construction activities on earth (Register, 1989; Siekmeier and Podnieks, 1990).

Alternative excavation equipment considerations would include separate vehicles for digging and for transporting the regolith. In such a system, excavating vehicles could dig the regolith similar to backhoes or shovels on Earth. Separate haulage vehicles like trucks would then haul the regolith to storage locations. Engineers will also evaluate the merits of closed conveyors instead of trucks. For early oxygen and hydrogen mining operations, engineers estimate that excavation machines would require about 9 tons (8 t) of capacity. One excavating machine could feed two or more haulage vehicles.

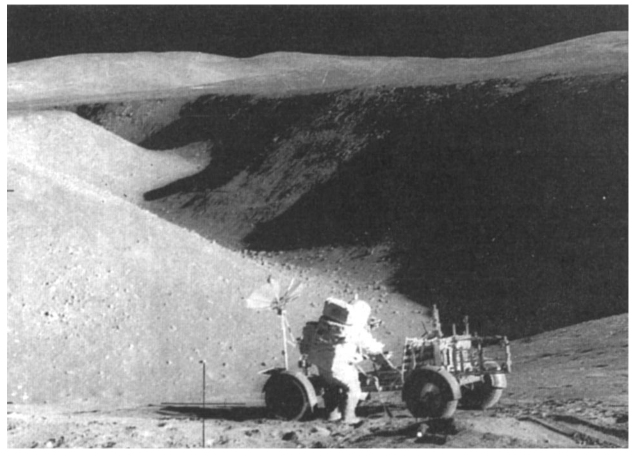

Fig. 22.9.23. Rover vehicle deployed on Moon (*courtesy:* NASA, Houston, TX).

Miners will probably size the regolith before it is processed. Grizzlies to reject boulders would be attached to excavating vehicles. Workers would size material down to ¼ in. (6 mm) with vibrating screens. Further sizing would be accomplished with cyclones. Workers would need a small conveyor to move the sized material into the processing plant.

Very little information is available on beneficiation. Researchers are investigating magnetic and other dry techniques for separation of ilmenite from other minerals in regolith.

Manufacturers will construct most equipment from advanced materials. Although terrestrial mining equipment does not contain much advanced material, the completely different economics of mining in space will force engineers to build equipment with high-strength, low-weight materials. The lower cost of transporting a lighter payload to the Moon will override higher equipment construction costs owing to advanced materials.

The great temperature extremes between day and night will influence equipment design, especially for surface operations. Since it will be difficult for equipment to efficiently operate in both extremes, engineers will probably design equipment to operate efficiently in one. At this time, designers favor high temperature operations. The surface mining equipment thus would operate during the 14-day lunar "day" and undergo maintenance during the 14-day lunar "night."

Experience in developing roving vehicles for sampling on the Moon and Mars will help engineers to design vehicles for excavating and moving lunar regolith (Fig. 22.9.23). Designers will include features that allow miners to operate the equipment with little actual human control. These vehicles would be highly automated and perhaps tele-operated (controlled by operators with television and "joy sticks"). Advanced artificial intelligence systems will be integrated into this equipment (see Chapter 22.2).

Engineers have not yet completed specific locomotion designs for space mining equipment. In addition to locomotion, excavation vehicles must also provide the traction necessary to excavate the regolith, perhaps via a stub inserted into the ground. NASA contractors are presently developing navigation and control systems for automated and/or remotely controlled sampling rovers, which could be adapted to mining machines. Present motor design studies should also benefit future mining vehicles. Mining vehicles would be electrically powered. NASA contractors are designing 1-MW nuclear-powered generators for lunar operations. Solar cells would be a key energy source for auxiliary equipment. Direct conversion of solar energy to provide heat for processing operations is a distinct possibility.

Mining operations to recover hydrogen and helium ions from the regolith could be several orders of magnitude larger than oxygen mining operations. Sufficient hydrogen for early lunar bases could be mined with the equipment described above. For larger amounts of hydrogen and for proposed amounts of helium-3, mining equipment becomes considerably more conjectural. The mines would cover a vast area but to a shallow depth. No mine on Earth has such a configuration. The large-volume mines on Earth use bucket wheel excavators and huge shovels to move material. Such equipment is not mobile enough to mine large areas each year. No large earth-moving equipment seems readily adaptable to the lunar environment. It is simply too premature to propose specific mining equipment for large-scale recovery of hydrogen and helium.

FACTORS INFLUENCING MINING EQUIPMENT AND METHODS. The lunar environment will profoundly influence the selection of mining equipment and methods (Podnieks, 1988). Apollo mission reports contain much information regarding potential environmental effects on lunar operations.

Dust was a significant problem for Apollo astronauts. Dust kicked up by walking, driving, and from engine exhaust obscured vision and coated equipment. A coating of dust, for instance, made it extremely difficult for an astronaut to remove a nut from a bolt during the Apollo 14 landing. The problems of the ubiquitous dust on bearings and other moving surfaces is readily apparent.

Operating at a gravity one-sixth that of Earth poses problems for operators and equipment. Body movements are completely different than movements on Earth. Equipment is much less stable at the lower gravity.

Without an atmosphere, the Moon suffers extreme temperature variations between night and day. At the Apollo 17 site, the surface temperature during the day was 231°F (111°C); during lunar night, temperatures at the same site were –275°F (–171°C). Such temperatures will pose extremely difficult operating conditions for mining equipment. Early miners will probably work only during the day because equipment design problems for operating in hot temperatures seem easier to overcome than those associated with cold temperatures. Surveyor III equipment left on the Moon was brittle (wire coatings) and weak (aluminum structures) when inspected by astronauts 2½ yr later.

The lack of atmosphere will pose additional problems in collecting gaseous products during any regolith processing. Operators will require completely closed systems to prevent the gases from escaping.

Radiation produced by cosmic rays, solar winds, and solar flare particles poses serious safety problems and could damage improperly shielded equipment. Micro-meteorite bombardment will also create a continual "sandblasting" attack on exposed equipment.

The combined effect of all aspects of the hostile lunar and planetary environment create a monumental challenge to design engineers. The development of mining equipment that functions well in this environment will be a noteworthy accomplishment.

22.9.6.3 Mining Costs

Costing a lunar or planetary mining operation differs completely from terrestrial mine cost analyses. Life support materials will be mined on the Moon if transportation costs exceed mining and processing costs. Given the high cost to transport cargo to the Moon, a mining operation may provide sufficient oxygen for life support at a lower cost. Engineers are determining the equipment, energy, and human resources necessary to mine oxygen on the Moon. Once these are determined, it will be possible to more accurately cost a mining operation.

Establishing costs for mining helium-3 and other solar-wind-deposited ions from the lunar regolith is almost impossible at this time. The technology is too doubtful to warrant speculation. Engineers have estimated that 1 ton (0.9 t) of helium-3 would replace an equivalent $1 billion in fossil fuel costs for electric power generation. This provides guidance for estimating the value of helium-3. Another way of examining the economics of mining helium-3 is to consider the value of a unit volume of regolith. At a helium-3 value of $1 billion/ton ($1.1 billion/t), the value of regolith averages about $8/ton ($9/t). This is not particularly impressive considering the great difficulties of mining.

22.9.6.4 Future Trends

Mining and processing oxygen seems the most likely mining operation to first appear in space. Local mining of construction materials—sintered blocks—also seems like a realistic possibility. Less likely but still under consideration are possible mining operations for recovering hydrogen, helium, and other ions deposited on the surface of the Moon, Mars, and Phobos.

Engineering feasibility studies are beginning to consider in more detail the possibility of lunar oxygen mining. Considerable research will be required before equipment can be developed to operate in the hostile lunar atmosphere.

22.9.7 NUCLEAR-ASSISTED MINING

PETER G. CHAMBERLAIN

22.9.7.1 Concept

During the 1960s, scientists and engineers considered a variety of techniques for applying the tremendous energy in *nuclear explosives* to mining systems. The first edition of the *SME Mining Engineering Handbook* (Cummins and Given, 1973) contained a section on these techniques (Russell, 1973). Since that time, little additional research on nuclear mining techniques has emerged. Public concern over environmental risks associated with nuclear energy, the nuclear test ban treaty, and uncertain availability discouraged further studies. Nevertheless, the concentrated energy and potentially low unit energy costs available in nuclear explosives may renew interest in using these devices in future mining operations. Consequently, selected portions of the *SME Handbook*, 1st ed., are reproduced below.

NUCLEAR EXPLOSIVES. In a nuclear explosion, the energy produced results from formation of different atomic nuclei by the redistribution of the protons and neutrons within the interacting nuclei. Nuclear energy is of much higher magnitude than chemical explosive energy considering equal masses and is released many times more rapidly. Nuclear explosions are produced by either fission or fusion. Fission occurs when a free (or unattached) neutron enters the nucleus of a fissionable atom. It can cause the nucleus to split into smaller parts, releasing large quantities of energy. Complete fission of 1 lbm (0.4 kg) of uranium can produce the energy equal to the detonation of about 9000 tons (8000 t) of TNT.

In nuclear fusion, a pair of light nuclei unite (or fuse) to form the nucleus of a heavier atom. Thus two deuterium (heavy hydrogen) nuclei may unite to form the nucleus of the heavier element helium. Such a reaction from 1 lbm (0.4 kg) of deuterium could theoretically release energy equivalent to about 20,000 tons (18,000 t) of TNT.

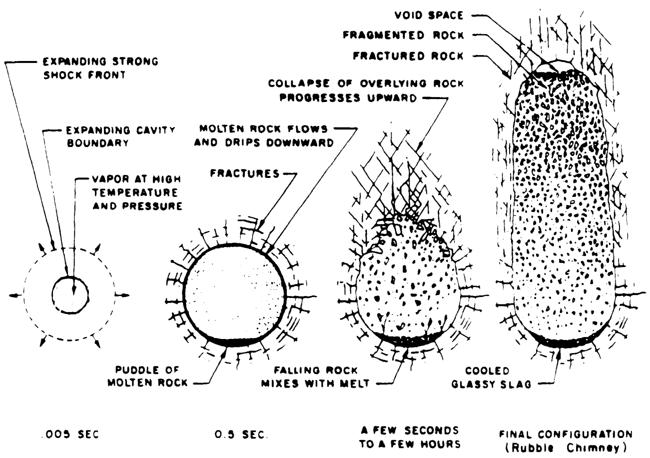

Fig. 22.9.24. Cavity growth during nuclear explosion for mining purposes (Russell, 1973).

By custom, the power of a nuclear explosion is expressed in terms of its energy release, or yield, compared with that from TNT. A nuclear explosive energy of 1 kiloton is defined as the prompt release of 10^{12} calories (4×10^9 Btu) of energy, and it is approximately equivalent to the energy released by detonating 1000 tons (900 t) of TNT.

UNDERGROUND MINING CONCEPTS. Formation of a rubble chimney, through collapse of the initially created cavity, is the basic concept for all current underground mining proposals using nuclear explosives. Both fission and fusion explosives have been used to produce rubble chimneys in a variety of rock types. Fig. 22.9.24 shows the sequence of events in cavity-chimney formation. The fractured zone surrounding the rubble chimney is vital to some proposed uses and is believed to extend from one to four cavity radii, depending on geology, depth of burial of explosive, and explosive yield. The technical and economical advantages of nuclear explosives appear to restrict their use in underground mining to large massive ore deposits where such large-scale blasts can be utilized to advantage. Proposed underground mining applications include the fracturing and fragmenting of large volumes of ore in preparation for block-cave-type mining and in situ leaching.

The term block caving has been used to describe ore extraction from nuclear-produced rubble chimneys. The success of this method assumes that ore can be extracted from such chimneys in formations that are not economically viable with other methods. Both the existence of such deposits and this cost relationship have yet to be proved.

The principle of in situ leaching has been proved and is widely used today (see Chapter 15.3). Recovery of valuable metals through ore broken or fractured by nuclear explosives offers many advantages, such as permitting large-scale mining operations in deep low-grade ore deposits with minimum disturbance of the surface topography. Such factors are becoming increasingly important as the need for natural resources increases and as any pollution of the environment becomes less acceptable.

CAVITY AND CHIMNEY FEATURES. Experimentation shows that cavities produced by nuclear explosives are roughly spherical in shape and are so considered when calculating dimensions or volumes. Cavity- and chimney-dimension formulas were derived empirically. Each formula contains a constant chosen according to the physical properties of the medium tested.

Cavity expansion probably continues until the internal gas pressure equals the pressure exerted by the surrounding rock mass, and this assumption was used in deriving the formula for cavity radius which follows (Anon, 1967):

$$R = C \times W^{1/2}/(\rho h)^{1/4} \qquad (22.9.8)$$

where R is cavity radius in ft; C is a constant for rock media; W is equivalent yield of nuclear explosive in kilotons of TNT; ρ is density of rock media in g/cm^3; and h is depth of burial in ft.

This formula is considered accurate to ±20% in changeable host rocks and to ±8% in one specific-type host rock. The following experimental values have been determined for the constant C in specific rock media (Hardwick, 1967):

Alluvium	86.6
Granodiorite	79.6
Salt	85.7
Tuff	104.6

The graphs in Russell (1973) indicate cavity radii for contained underground nuclear explosions of various yields and depths of burial in granite rock.

Present knowledge indicates little correlation between the size of the cavity formed and the physical and elastic properties of the surrounding rock. Porosity has a negligible effect on cavity size, but the presence or absence of pore water is significant—the higher the pore water content the larger the cavity.

In nearly all cases, the chimney formed above the initial nuclear explosive takes a roughly cylindrical shape slightly larger in diameter than the cavity. Unless collapse reaches the surface, the chimney roof usually is domed and contains a small rubble-free void. Chimney height H may be determined by the formula,

$$H = K R \qquad (22.9.9)$$

where K is a factor determined from rock type and depth of burial, and R is the cavity radius (Williams, Russell, and Sheridan, 1969). For granite, K = 4.35. A rule-of-thumb approximation is that chimney height is equal to 2½ cavity diameters. True height, of course, varies with rock type. Chimney heights for contained underground nuclear explosions of various yields and depths of burial in granite rock are indicated graphically in Russell (1973).

Estimates of rock broken by deeply buried nuclear explosives can be made. The graphs shown in Russell (1973) indicate the theoretical amount of rock broken by intermediate underground explosions at various depths of burial. Approximate tonnages of broken rock may be determined from these graphs, but variance occurs with the type of host rock. Rubble fragment sizes for two contained shots in granite are given in Table 22.9.7.

In the design of any underground mining system, both the yield and the emplacement depth of the nuclear explosive must be such that the resulting chimney does not extend to the surface. No method of equating these design criteria to all geologic conditions is presently known. However, experimental results indicate that an emplacement depth of at least 10 diameters of the designed cavity provides a suitable containment safety factor. Because cavity diameter depends on a variety of engineering and geologic factors, this 10-diameter rule is an approximation only. Where geologic conditions indicate strong fracturing or faulting, special detailed studies will be required to evaluate containment

Table 22.9.7. Fragment Size Distribution in Rubble Chimneys From Two Contained Nuclear Explosions

Sieve Size	% Passing	Sieve Size	% Passing
6ft	100	2 in.	20
5ft	95	1½ in.	16
4ft	88	1 in.	14
3ft	75	¾ in.	12
2ft	60	½ in.	10
1ft	40	⅜ in.	9
6in.	30	#4 (0.187 in.)	7
4in.	25		

Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m
Source: Russell, 1973.

problems. Recent studies by the Lawrence Radiation Laboratory have established that a buffer zone of rock 300 to 500 ft (90 to 150 m) thick (depending primarily on the rock characteristics) above the top of the predicted rubble chimney is sufficient for containment up to yields of several hundred kilotons.

[Author's and editor's note: Section 21.10, 1st ed. of the *Handbook,* provides additional information on specific proposed applications of nuclear explosives including block caving, in situ leaching, and in situ retorting. Cost estimates for the nuclear explosives, assuming that they could be sold for mining purposes, are also provided. So little interest is presently evidenced in nuclear-assisted mining, primarily because of radiation hazards, that the editors feel it inappropriate to provide more detail in this current edition.]

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Section 23 Evaluation of Mining Methods and Systems

JAN M. MUTMANSKY, ASSOCIATE EDITOR AND SECTION COORDINATOR

Chapter 23.0
INTRODUCTION

JAN M. MUTMANSKY

Since 1973 when the first edition of the *SME Mining Engineering Handbook* was published, many significant changes have occurred in the US mining industry. The industry went through a fairly prosperous period immediately after 1973, but then many economic pressures came to bear on mining companies, particularly in the metal and coal branches of the industry. These pressures changed both the mining methods used and the general economics of the mining methods. As a result, the industry today is different in many ways than it was in 1973. A general summary of some of the more important changes that have come about will be provided here as well as some assessment of the effects of these changes on the choices of mining methods.

In the metal mining industry, the most significant trends were increased surface mining, the use of larger and more efficient equipment, and increased technology and automation. The primary impetus for these changes was unprecedented competition from foreign producers. In many cases, this competition came about because of the nationalization of foreign mines and the tendency for these mines to become a source of foreign exchange. Under these circumstances, the mines could be operated even though they were not profitable. As a result, domestic metal producers were unable to readily compete on the world metals market.

In response to this challenge, US producers became more competitive in a number of ways. One way was to close unprofitable mines and to emphasize more efficient mining methods. Consequently, many of the mines using traditional selective stoping methods were either closed or redesigned to produce in a more efficient manner. This led most metals companies toward bulk mining methods, either underground or surface. During much of the 1980s, little in the way of exploration or development of metals deposits was being accomplished except in the precious metals areas. In precious metals, change was also the order of the day as the use of open pit mining and heap leaching were greatly expanded.

In the coal industry, the early to middle 1970s was characterized by growth and generally high prices with the Arab oil embargo helping to elevate energy prices. Many new coal mines were initiated during this time period. Toward the end of the 1970s, however, the costs of production were continuing to rise significantly while the market prices for coal remained relatively stable. As a result, many smaller and less efficient coal operators were eliminated as productive capacity was significantly higher than the demand. During the late 1970s and the 1980s, the coal industry in general responded to economic pressures by substantially increasing productivity. From 1977 to 1987 alone, productivity nearly doubled in the underground coal industry as the producers fought to maintain their share of the competitive marketplace.

The strong prevailing shift of the previous decades towards greater percentages of coal production from surface mines ended during this period, apparently as a result of environmental problems and a reduced inventory of coal that could be easily extracted from surface operations. Instead, increased use of longwall mining, more reliable and productive continuous miners, improved roof bolters and bolting practices, use of dust scrubbers and remote control on continuous miners, and use of improved systems such as supersections all contributed to better productivity. In addition, the labor force matured and became sensitized to the need for productivity improvement during this period. As a result, the ways in which coal mining operations were managed changed more than the mining methods themselves. In addition, most coal mining companies concentrated on productivity improvements instead of new operations.

During the same period of time, the industrial minerals producers suffered fewer market problems than the other segments of the industry, probably due primarily to relatively strong domestic demand for industrial minerals and the inability of foreign producers to readily compete within the US due to transportation costs. However, foreign competition did hurt domestic production in some categories such as in the cement and fertilizer industries. One significant trend in the industrial minerals industry has been larger companies and larger mines, primarily as a result of the acquisitions tendency in industry in recent years. In addition, a trend toward more surface mining and less underground mining was also prevalent.

In terms of the selection of mining methods and systems, the period from 1973 to the present represents a problem due to the intense competition and a general lack of mine development. As a result, the record of costs for development of new mines is quite sparse and somewhat more difficult to interpret. Furthermore, the technology of mining has changed significantly since the last surge in minerals development projects. Thus cost prediction and the selection of mining methods based upon past costs may be more difficult than before. The only significant developments in recent years have been the surface gold leaching operations that have emerged throughout the western states. In addition, technological advancements have occurred with substantial applications of automation, monitoring, and computer control becoming commonplace in all segments of the industry.

Though automation and technology have been instituted by the mining companies as a necessary and desirable part of their operations, an additional change has been brought about by the public. This change is the emphasis on social and environmental concerns that all mining companies must now satisfy. Water pollution, air pollution, waste disposal, and reclamation requirements are all important considerations in the planning and development of future mining operations and will have a major bearing on the methods chosen and the estimation of costs. In addition, these considerations become important in the choice between surface and underground operations.

While the cost prediction phase of mining method selection has been affected, the other variables associated with the choice of a mining method have not changed significantly. Geologic and rock mechanics variables will normally be of prime importance in the elimination of unsuitable mining methods and in the final selection of the most efficient and safest methods. The geologic and mechanics parameters of the selection process currently may be more crucial, however, as the choice of the method will certainly affect the costs and the competitive nature of the marketplace requires that the most efficient method be chosen.

During the past two decades, the selection of mining methods and systems has certainly been impacted by the changes that have occurred. Many new social and environmental concerns have entered into the picture as have several new areas of technology. The choice of a mining method and other important decisions to be made before a mine can be developed are therefore more important than before if only because of the increasing number of social and governmental concerns that must be met. It is therefore imperative that minerals engineers use every means at their disposal to improve their decision-making procedures so that mining methods are properly chosen before mining is undertaken.

23.0.1 SECTION TOPICS

This section of the *Handbook* deals with the evaluation of mining methods, methods of providing costs for the methods, and selection of mining methods to suit the orebody conditions and related social and environmental constraints. Chapter 23.1, Selection Variables, enumerates the major variables that must be considered during the selection process for a mining method. This includes the geologic parameters and rock properties that are important in mining method selection and also a review of the social, political, labor, and environmental variables as well.

Chapter 23.2, Surface vs. Underground Methods, discusses in some detail the economics and practical variables involved in making the choice between surface and underground methods in an orebody where either can be used. Numerous engineering considerations used to make the decision, such as productivity, labor, safety, energy consumption, and environmental concerns, are also discussed. The chapter then emphasizes the economics of the decision with a detailed example to illustrate the various economic factors to be considered. In addition, it economically defines the proper point in the lifetime of a surface mine when conversion from surface to underground mining is warranted.

Several additional economic considerations related to evaluation of mining methods and systems are discussed in Chapter 23.3, Cost Comparison and Control. This chapter deals primarily with cost estimation methods, sources of cost data, cost updating, and previously published costs for various mining methods. In addition, the results of two cost surveys conducted in 1989 for inclusion in this chapter are presented. The two cost surveys cover coal mining methods and noncoal methods in separate subsections of the chapter. These costs are then compared with other cost values that have appeared in the literature. Finally, some concepts of cost control are presented.

The final chapter in this section, Chapter 23.4, Selection Procedure, outlines several aspects of the selection procedure for mining methods. The specific subtopics discussed within the chapter include a logical sequence of events for choosing a mining method, previous strategies used in the selection of a mining method, and some detailed ranking procedures for use in categorizing the suitability of mining methods under differing conditions. Finally, the chapter concludes with a logic flow diagram to be used in the process of selecting a mining method.

Chapter 23.1
SELECTION VARIABLES

DAVID E. NICHOLAS

23.1.1 GENERAL

Selection of an appropriate mining method is a complex task that requires consideration of many technical, economic, political, social, and historical factors. Mining methods considered in this chapter include traditional surface and underground methods, as well as non-traditional mining methods such as hydraulic mining, in situ leaching, gasification, robotics, and the like. The variables involved in a general consideration of mining method selection are very similar to those already discussed specifically in Section 16 for surface methods and in Section 21 for underground methods. This chapter concentrates mainly on technical and economic aspects of the selection process. However, the mining engineer needs to recognize and include the other considerations—political, social and historical—as well.

The most critical prerequisite for performing mining method selection is a good database. This becomes imperative for high-capital-cost operations where a decision based on inadequate data can be a costly mistake. Low-capital operations may require technical evaluation, but it is difficult to justify the cost of data collection and engineering.

The past approach of adopting the same mining method as that of a neighboring operation is not as appropriate as it once was. However, this does not mean that one cannot learn from or quantitatively compare mining plans to those of existing operations in the district or to similar deposits. The importance of a good database is even more critical for the evaluation of the more novel mining methods, as there is not the well of past experience to draw on that exists for traditional mining methods.

Although experience and engineering judgment still provide major input into the selection of a mining method, subtle differences in the characteristics of each deposit can usually be perceived only through a detailed analysis of the available data. It becomes the responsibility of the geologist and engineer to work together to ensure that all factors are considered in the mining method selection process.

The mining method that is ultimately selected should be one that the management is comfortable with and believes is feasible. In some cases, this may not correspond to the best method based on the ground conditions. In the early mining stages a more costly method might be chosen either because it has a higher reliability of meeting production requirements or because the labor force or operator is familiar with that method. As mining progresses, the method should be continually improved and refined to reflect the experience gained in daily mine operations.

23.1.2 INPUT PARAMETERS

There are six major parameters that should be considered when evaluating a mining method:

1. Physical and geologic characteristics of the deposit.
2. Ground conditions of the hanging wall, footwall, and ore zone.
3. Mining and capital costs.
4. Mining rate.
5. Availability and cost of labor.
6. Environmental regulations.

Of these parameters, physical and geologic characteristics and ground conditions should be the primary parameters considered in determining the most appropriate mining method(s).

23.1.2.1 Physical and Geologic Characteristics

A physical description of the deposit should include

1. General shape.
2. Ore thickness.
3. Plunge.
4. Depth below the surface.
5. Grade distribution.

These characteristics are best understood by the preparation of geologic sections and level maps from the drilling program. These maps should identify major rock types, alteration zones, and major structures, such as faults, veins, and fold axes. The level and cross-sectional maps should be drawn to the same scale as mine planning maps, and sections should be drawn without vertical exaggeration. A good general rule is that the area included on the maps should extend 1.75 times the depth beyond the limits of the ore body. Separate overlays depicting alteration zones and grade contours or ore blocks usually provide the best picture of spatial relationships in the ore body.

Geologic data are the most critical in terms of determining economic viability. In most cases, the engineer has developed a target model for the deposit prior to exploration, and certainly after the first ore intercept. During ore delineation drilling, the geologist not only must collect assay data but also metallurgical, geotechnical, hydrologic, and environmental data.

The basis for all understanding of the ore deposit is the geology, that is, rock type, alteration, and major geologic structures. The known geology should be put on one set of sections and level maps, and the interpreted geology shown on another set of sections and level maps. These interpreted maps are the more useful to the engineers. The geologist needs to make some interpretations even if the database is limited; on the other hand, the engineer should discuss these interpretations with the geologist, making it clear which geologic interpretations are well understood and which interpretations are conjecture based on limited data.

The assay data needed will depend on the type of material to be mined. Not only should the primary economic mineral be assayed or tested, but those less important but recoverable associated minerals should be tested, as well as minerals that can be detrimental to processing. The deposit should be zoned or contoured based on the mineral grade. This can be done using a block model or manually. The data should be interpreted in section as well as in plan, and related to the geology.

The metallurgical, geotechnical, hydrologic, and environmental data to be collected during drilling for ore definition varies greatly. Metallurgical data do not usually impact the type of mining method, although the data may impact whether the mining method must be able to provide a blended ore.

Hydrologic data do not usually impact the choice of mining method, except in determining the feasibility of in situ leaching; they do however, impact the design of the mining system. The primary hydrologic data needed are elevation of the groundwater table. Slug tests (Bouwer and Rice, 1976) in exploration drillholes can be used to provide a reasonable estimate of the transmissivity of the rock. This transmissivity data should be contoured and correlated with the geology.

Some of the geotechnical data, such as oriented core data, RQD (rock quality designation), fracture frequency, and estimated strength, can only be obtained from drill core (Chapter 10.5). This is easily collected (23.1.2.2). Most of the drilling in a deposit is in or near the mineralization. However, the mining method may be dictated by the ground conditions above, next to, or below the deposit. Consequently, it is necessary that drilling be done beyond the lower limit of the deposit by 50 to 100 ft (15 to 30 m). Holes drilled outside the limit of the deposit are very important from a geotechnical point of view and should be logged in as much detail as ore holes.

The above work applies to base metal, ferrous, and nonmetallic deposits. Each of these deposits is unique in terms of its geologic database and assay data requirements. For some of these deposits, it is not affordable to collect all of these data early on, or at least it is not common to do so. Therefore, drill core should be photographed before it is assayed, and whole core should be preserved for possible physical testing. Photographing the core also provides a permanent document of the core so that if some new understanding of the deposit occurs later in the mine life, the old holes can be reexamined via the photos.

23.1.2.2 Ground Conditions

Ground conditions of the ore zone, hanging wall, and footwall should be defined for the deposit. The basic components that define the ground conditions are

1. Rock substance shear strength.
2. Natural fracture and fault shear strength.
3. Orientation and location of major geologic structures (faults, folds, bedding).
4. Orientation, length, and spacing of joint sets.
5. In situ stress.
6. Hydrologic conditions.

The combination of all these parameters defines how the rock mass will behave. The rock mass can be described quantitatively by determining the individual parameters, or it can be classified by an RQD value. Neither approach is completely satisfying in determining the rock mass behavior. The classification approach is relatively easy to perform, but the choice of class is somewhat subjective, and each classification system has been aimed at predicting certain aspects of traditional mining. The biggest advantages of classifying are that it can be done as part of the geologic mapping and logging, and that a number of empirical correlations have been made from these classifications. Taking measurements of each of the components of the rock mass is the best way of determining the actual properties. However, the art/science of rock mechanics is still working on ways of combining the properties to come up with the rock mass strength.

Existing classification methods are listed in Table 10.5.1 in Chapter 10.5. They include Deere, 1968 (Table 23.1.1), Coates, 1970 (Table 23.1.2), Bieniawski, 1973 (see Table 10.5.2 in Chapter 10.5) Barton et al., 1974 (Fig. 23.1.1) and Laubscher, 1977 (Fig. 23.1.2). Classifications with the better empirical correlations are those of Bieniawski, Barton, and Laubscher.

ROCK SUBSTANCE SHEAR STRENGTH. This represents the strength of the rock *between* geologic structures. The intact rock shear strength is determined from uniaxial and triaxial compression tests (Fig. 23.1.3). It can be estimated from uniaxial compression and disk tension tests. At least three test specimens, preferably six per rock type per test type, are required. By selecting three specimens of each rock type from each drillhole, it is possible to build up a collection of specimens for later detailed testing. In addition, the elastic properties of Young's modulus and Poisson's ratio should be measured during the uniaxial test.

Point load testing of drill core can be done during the drilling program to provide a measure of the uniaxial compressive strength variability. This is most useful in deposits where geologic structures are limited and the rock strength is low enough to fail, that is, in the case of coal, potash, or uranium deposits.

Special evaluations such as creep testing may be required for material that flows, such as salt and potash.

NATURAL FRACTURE AND FAULT SHEAR STRENGTH. These are determined by direct shear tests (Fig. 23.1.3). Most failure paths will be controlled by these pre-existing geologic structures, making this test critical.

MAJOR GEOLOGIC STRUCTURES. These should be identified as to location, orientation, and thickness. This should be done as part of the normal geologic mapping and interpretation.

ROCK FABRIC. Those geologic structures that are too numerous and must be dealt with statistically should be mapped to determine the type of structure, orientation, length, and spacing. This can be done using cell mapping (Call, 1979), the detail line mapping technique (Call et al., 1976), or some similar method.

When only drill core is available, oriented core can be used, which will provide orientation and spacing data (Call et al., 1987). This technique is relatively inexpensive and requires only a full-time geologic technician on the drill rig to collect the data. This technique is difficult in highly fractured ground and may not provide sufficient data for its cost. Not all drillholes are oriented; usually three to six holes per deposit are required. For those drillholes not oriented, a measure of the fracture intensity is needed. The most common measure is rock quality designation. RQD is a modified core recovery where the sum of all pieces greater than two times the core diameter is divided by drill interval. RQD is commonly used, but one must be cautious in interpreting it. An RQD rating of 100% means only that all the pieces are at least 4 in. (100 mm) long for NX core; this can be attained either by all pieces being 4 in. (100 mm) long or all pieces being 1 m (3.3 ft) long. These two conditions represent different types of ground, but similar RQD values. Another measure of fracture intensity is the fracture frequency and percentage-broken zone. The fracture frequency is the number of whole core samples divided by the recovered drill interval, minus the interval of broken zone. The broken zone is that interval of non-whole core plus unrecovered core.

IN SITU STRESS. In situ stress measurements provide the orientation and magnitude of the stresses in the ground (Chapter 10.3). The parameters discussed above are needed to determine the strength of the rock mass, while the in situ stress provides the basis for determining the driving stress that could cause failure. Acceptance of the importance of taking in situ stress measurements is increasing; measurements are required for deep pits where stresses are high, and for any underground operation. Methods of measuring stress include the USBM overcoring (Hooker and Bickel, 1972), South Africa's CSIR "doorstopper" or triaxial cell (Leeman, 1969), and Australia's CSIRO triaxial cell (Fama and Pender, 1980). There are pros and cons to each of these methods, and improvements are being made constantly. The orientation of the stress is generally more accurate than the stress magnitude for any of the three techniques. The magnitude provides the relative difference in stresses, while the absolute value has a high variability. If stress measurements are not made, they can be inferred based on the geologic history, rock type, and proximity to geologic structures. Abel (1981) has developed a flow chart that provides a guesstimate of the in situ stress based on correlation between rock type and geologic structure, and stress measurement results.

Table 23.1.1. Deere's Classification of Rock

I. INTACT ROCK STRENGTH

Class	Description	Uniaxial Compressive Strength psi	Uniaxial Compressive Strength MPa
A	Very High Strength	> 32,000	> 220
B	High Strength	16,000–32,000	110–220
C	Medium Strength	8,000–16,000	55–110
D	Low Strength	4,000–8,000	27–55
E	Very Low Strength	< 4,000	< 27

II. INTACT ROCK MODULUS RATIO (E_t/σ_c)

Class	Description	Modulus Ratio
H	High Modulus Ratio	> 500
M	Average (Medium) Ratio	200–500
L	Low Modulus Ratio	< 200

III. TERMINOLOGY FOR JOINT SPACING

Description	Spacing of Joints	
Very Close	< 2 in.	< 50 mm
Close	2 in.–1 ft	50–300 mm
Moderately Close	1–3 ft	300 mm–1 m
Wide	3–10 ft	1 m–3 m
Very Wide	> 10 ft	> 3 m

IV. ROCK QUALITY DESIGNATION (RQD)

Description	% RQD
Very Poor	0–25
Poor	25–50
Fair	50–75
Good	75–90
Excellent	90–100

Source: Deere, 1968.

Table 23.1.2. Coate's Classification of Rock

I. SUBSTANCE STRENGTH

Description	Uniaxial Compressive Strength MPa	Uniaxial Compressive Strength psi
Strong	≥ 70	≥ 10,000
Weak	≤ 70	≤ 10,000

II. FORMATION (FRACTURING)

Massive:	Joint spacing > 6 ft (2 m)
Layered:	Bonding between layers is less than within layer
Blocky:	Joint spacing is between 1 ft (0.3 m) and 3 ft (1 m)
Broken:	Joint spacing is less than 1 ft (0.3 m)
Very Broken:	Joint spacing is less than 3 in. (80 mm)

Source: Coates, 1970.

HYDROLOGIC CONDITIONS. Hydrologic conditions contribute to the rock mass strength to the extent that, if water exists, the design must allow for a drainage program (Chapter 12.1). Most mining methods, with the exception of some hydraulic methods, do not deal well with water. The hydrologic data needed include elevation of the water table, transmissivity, and the storage coefficient.

23.1.2.3 Mining and Capital Costs

By nature, some mining methods have higher unit costs than others. The perceived relative cost is based on the fact that the deposit was mined using the method that was appropriate for its conditions. For example, block caving is generally considered the cheapest underground method. However, if mining a large deposit with widely spaced jointing and high strength, the cost of block caving would probably be prohibitive because the production rate would be too low for the number of men required. As our technology improves, what was once an expensive method may become less expensive. An example of this is underhand cut and fill. With the recent improvements in placing dense sandfill, underhand cut and fill should not require the labor or the amount of cement it did in years past to create a competent back to mine under. This may make the smaller deposits with moderate grade and poor ground more feasible. Consequently, although there may be an overall difference in mining costs, the mining method should still be chosen based on the ground conditions and ore geometry first; if the method is too expensive, then cheaper methods should be considered, realizing that the cheaper method may require new or unproven techniques, and that the operating costs may be higher than normal for that method. Table 23.1.3 shows an approximate ranking of different mining methods based on their relative operating costs (Hartman, 1987).

Capital requirements vary depending on the mining method, size of deposit, access requirements, and production rate. Unfortunately, most of the decisions on how to bring a deposit into

RQD: ROCK QUALITY DESIGNATION

J_n: Joint Set Number - Nine categories that define the number of joint sets ranging from massive to three joint sets to crushed rock.

J_r: Joint Roughness Number - Nine categories that describe the physical nature of the joint set ranging from discontinuous, to rough and planar, to filled with clay gouge so that the rock walls are not in contact.

J_a: Joint Alteration Number - Sixteen categories that describe the alteration along the joints ranging from tightly healed joints to filled with clay gouge, to zones of crushed rock.

J_w: Joint Water Reduction Factor - Six categories that define the water pressure ranging from dry to high flows.

SRF: Stress Reduction Factor - Sixteen categories that define the stress conditions for rock. Related to compressive strength and principal in situ stress.

ESR: Excavation Support Ratio - Six categories that define the type of opening being designed ranging from mine opening to underground nuclear power stations.

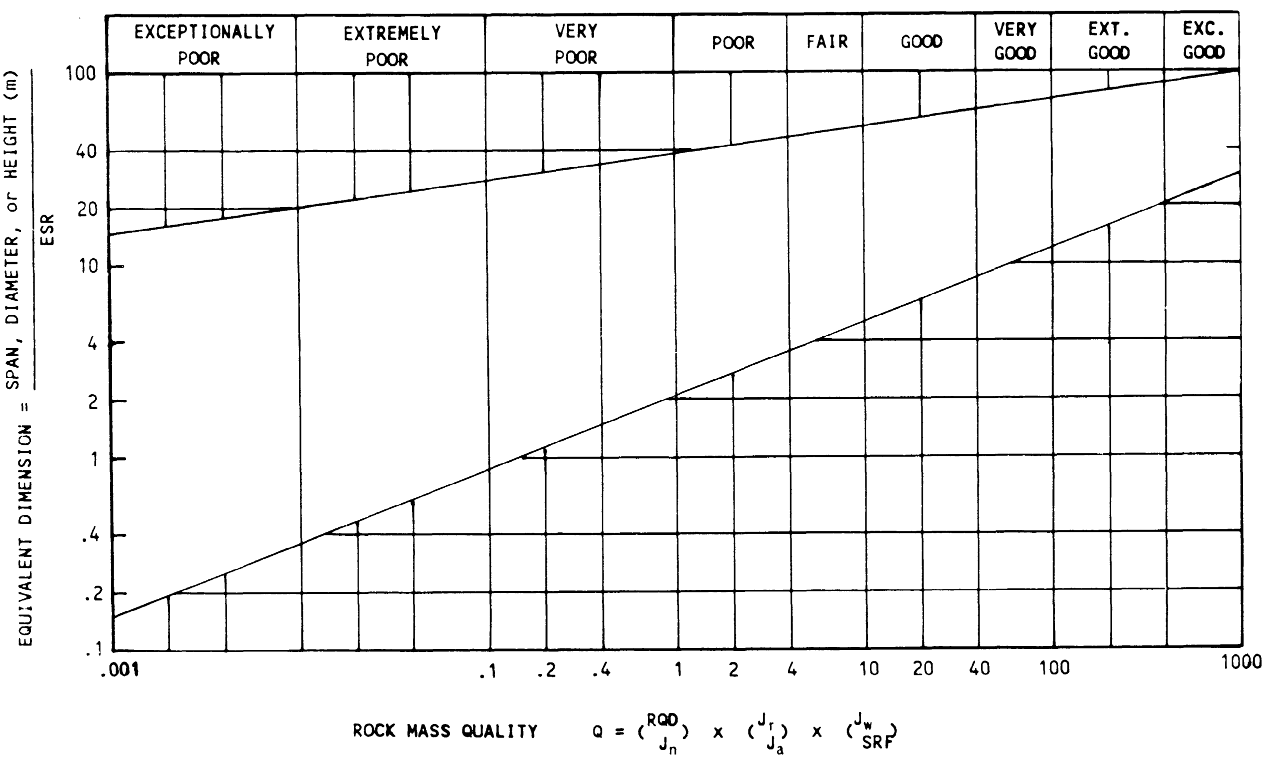


Fig. 23.1.1. Description of the NGI classification system (Barton et al., 1974).

production are controlled by the time value of money and not the nature of the deposit. The more capital required to start the project, the better the grade of the deposit must be and/or the faster it must be mined. The capital usually requires that the high-grade ore be mined first, which is not always the best sequence from a geotechnical or metallurgical point of view. In a block cave, one should start in the easy caving ground to ensure meeting production schedules. In an open stope, the usual mining sequence is to retreat from one direction to another or mine from the center out.

Often no one mining method is the most appropriate for a deposit, but rather two or three may be feasible. These methods should be economically evaluated to determine which will provide the best return on investment. In this way, one can define

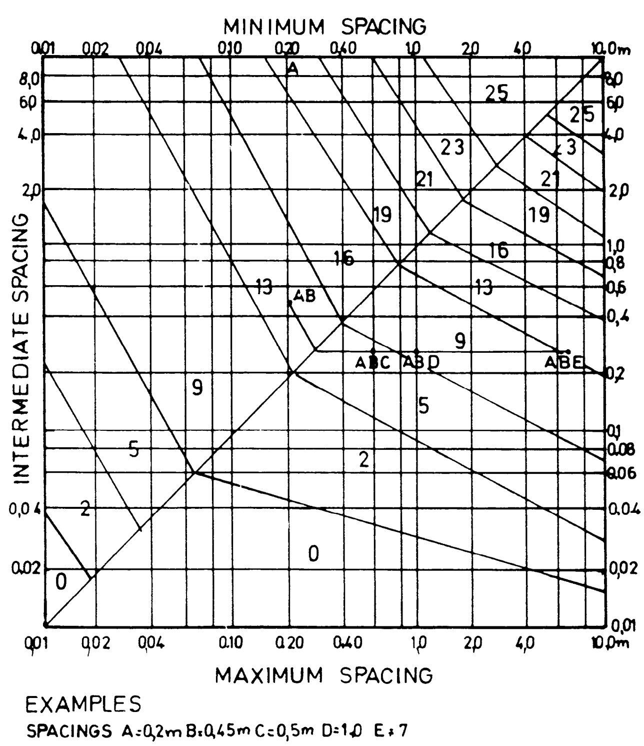
A. MEANING OF THE RATINGS

CLASS	1		2		3		4		5	
	A	B	A	B	A	B	A	B	A	B
RATING(Σ1-4 OF B)	100	– 81	80	– 61	60	– 41	40	– 21	20	– 0
DESCRIPTION	VERY GOOD		GOOD		FAIR		POOR		VERY POOR	

B. BASIS OF THE CLASSIFICATION

1	ROD %	100 — 97		96–84	83 –71	70 – 56	55 – 44	43 –31	30 – 17	16 – 4	3 — 0	
	RATING (= ROD % x 15/ 100)	15		14	12	10	8	6	4	2	0	
2	IRS (MPa)	>185	184-165	164 145	144 125	124-105	104-85	84-65	64- 45	44 - 25	24 -5	4 -0
	RATING (= 0,1xMPa)	20	18	16	14	12	10	8	6	4	2	0
3	JOINT SPACING	REFER C (BELOW)										
	RATING	25 ⟵⟶ 0										
4	JOINT CONDITION INCL GROUND WATER	REFER D (BELOW)										
	RATING (40xA×B×C×D/10[8])	40 ⟵⟶ 0										

C. RATINGS FOR MULTI JOINT SYSTEMS



EXAMPLES

SPACINGS A=0,2m B=0,45m C=0,5m D=1,0 E=7

RATINGS A=19 AB=13 ABC=5 ABD=9 ABE=13

D. ASSESSMENT OF JOINT CONDITIONS

ACCUMULATIVE % ADJUSTMENT OF POSSIBLE RATING OF 40

PARAMETER	DESCRIPTION		DRY COND.	WET CONDITIONS MOIST	MOD.PRES 25-125 1/m	SEV.PRE >125 1/m
A JOINT EXPRESSION (large scale irregularities)	WAVY	MULTI DIRECTIONAL	100	100	95	90
		UNI DIRECTIONAL	95 90	95 90	90 85	80 75
	CURVED		89 80	85 75	80 70	70 60
	STRAIGHT		79 70	74 65	60	40
B JOINT EXPRESSION (small scale irregularities or roughness)	VERY ROUGH		100	100	95	90
	STRIATED OR ROUGH		99 85	99 85	80	70
	SMOOTH		84 60	80 55	60	50
	POLISHED		59 50	50 40	30	20
C † JOINT WALL ALTERATION ZONE	STRONGER THAN WALL ROCK		100	100	100	100
	NO ALTERATION		100	100	100	100
	WEAKER THAN WALL ROCK		90	90	90	90
D JOINT FILLING	NO FILL - SURFACE STAINING ONLY		100	100	100	100
	NON SOFTENING & SHEARED MATERIAL (CLAY OR TALC FREE)	COARSE SHEARED	95	90	70	50
		MED. SHEARED	90	85	65	45
		FINE SHEARED	85	80	60	40
	SOFT SHEARED MATERIAL (eg TALC)	COARSE SHEARED	70	65	40	20
		MED SHEARED	65	60	35	15
		FINE SHEARED	60	55	30	10
	GOUGE THICKNESS <AMPLITUDE OF IRREG.		40	30	10	
	GOUGE THICKNESS >AMPLITUDE OF IRREG.		20	10	FLOWING MAT. 5	

† IGNORE THIS FACTOR FOR STRAIGHT, POLISHED OR STRAIGHT SMOOTH JOINTS

Fig. 23.1.2. Laubscher's classification system (Laubscher, 1981).

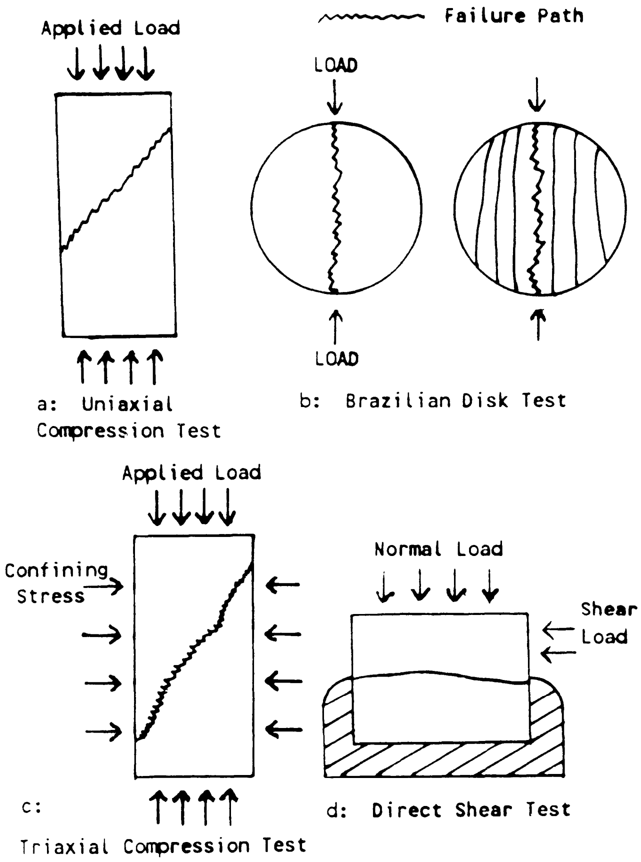


Fig. 23.1.3. Typical types of physical testing for determining rock strengths.

Table 23.1.3. Ranking of Mining Methods Based on Relative Operating Costs

Mining Method	Relative Cost (%)
Hydraulicking, Dredging, In situ Leaching	5
Open Pit/Open Cast	10
Block Caving, Longwall	20
Room and Pillar, Stope and Pillar	30
Sublevel Stoping	40
Shrinkage Stoping, Sub-level Caving, Induced Caving	50
Cut and Fill Stoping	60
Square Set Stoping	100

Source: Hartman, 1987.

those methods that are technically feasible and those that are the most economic. When considering less conventional techniques, the uncertainty of meeting production is high. Therefore the approach might be to choose a more traditional method, and at the same time begin a test area of the novel mining technique to prove its viability. If a novel method (Section 22) must be used, management needs to be willing to take the risk of developing a new method, and have the time and/or funding needed to make the method productive.

23.1.2.4 Mining Rate

The mining or production rate is a function of the capital costs, the size and grade of the deposit, and the location of the deposit. The mining rate does not usually have a major impact on choosing the mining method. The mining rate has more impact on how the ore is transferred out of the mine. Given that the deposit is physically large enough, the mining method can usually be expanded to meet the production needs.

When mining in countries with marginally stable governments, companies may wish to mine the deposit more quickly. This may mean high-grading the deposit rather than mining it to maximize the profits. In these situations, the mining method should be designed to high-grade and recover the lower-grade material later. Some commodities, such as industrial minerals, do not require a large production rate, so that even though a fast mining rate is possible, there is no sense in spending the capital if the rate will never be mined.

23.1.2.5 Availability and Cost of Labor

The abundance or lack of trained labor can be significant in the mining method selection process. It is important that the management and engineering staff be able to visualize the operation as a whole and deal with any problems that may develop in implementing it. The mine employees must be capable of performing the required tasks of the method, or suitable training programs must be planned. In some cases, the location of the deposit will have a significant impact on labor resources and thus the type of mining methods that can be considered. In general, where labor rates are high but the labor is well-educated/trained, techniques requiring sophisticated equipment should be possible (although labor must still be trained and supervised). Given these conditions, robotics and computerized equipment can be used. This will provide a high production rate per employee-shift and minimize operating costs. However, in countries where the cost of labor is low and education is limited, a method that requires a large labor force is feasible and probably preferable.

23.1.2.6 Environmental Regulations

Environmental regulations and requirements may limit the types of mining methods that can be selected (Chapters 3.4 and 7.3). Consideration of subsidence restrictions, groundwater contamination, tailings and waste disposal, and reclamation issues will all have to be taken into account during the method selection process. Although surface mining methods are generally lower in cost than underground methods, in the future the mining industry may be forced into choosing underground methods more often due to the fact that they generally have less impact on the surface environment. Surface subsidence limits are imposed on those deposits under existing non-mine structures and in wilderness areas. Improvements in backfill materials should help reduce this problem and allow maximum recovery of the deposit. Environmental regulations are likely to be one of the motivating forces for developing new mining methods. However, probably before this will happen, those deposits outside the United States will be mined using the more traditional mining methods. Groundwater will be more of an issue as our water sources are depleted in the United States (Chapter 12.1); this problem is not likely to occur in most of the developing countries. Dealing with the water regulations will not change the mining method, but rather its feasibility.

Other particular considerations in the mining method selection will usually be site specific. These considerations can include excessive ground temperature, proximity to inhabited areas or large bodies of water, and earthquake hazards.

23.1.3 SUMMARY

It is important that the geologists and engineers evaluating the deposit consider each of the selection variables before making a decision regarding mining method. In most situations, the controlling parameters that will define the feasible mining methods are

1. Tonnage, grade, shape, and depth of the deposit.
2. Ground conditions of the ore zone, hanging wall, and footwall.
3. Capital and operating expenses.

The evaluating engineer must recognize these controlling parameters and ensure that adequate data are available for the feasibility analysis to be performed.

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Chapter 23.2
SURFACE VS. UNDERGROUND METHODS

DAN NILSSON

23.2.1 INTRODUCTION

Many deposits can be mined entirely with surface methods; others must be worked underground from the very beginning. Under similar circumstances, surface mining is normally regarded to be more advantageous than underground mining. In a choice between surface and underground methods, there are many factors to be taken into account, such as,

1. Size, shape, and depth of the deposit.
2. Rock conditions.
3. Productivities and machinery capacities.
4. Capital requirements and operating costs.
5. Ore recoveries and revenues.
6. Safety and injuries.
7. Environmental aspects.

Still other deposits can be mined first as open pits but must be worked underground later. The question arises: what is the optimal depth for changing over from surface to underground mining?

In this chapter, these factors will be discussed briefly. Some costs will be cited. Please note that although these costs are in the right order of magnitude at the time of this writing, they cannot be applied to any single mineral producer.

Table 23.2.1. Number of Noncoal Mines in the Western World by Production Rate

	Production Category (1000 tons of ore/yr)	Year 1967	1977	1987
Surface Mines	More than 3000	102	138	176
	1000–3000	109	142	116
	500–1000	81	64	86
	300–500	61	63	63
	150–300	68	53	81
	Total	421	459	522
Underground Mines	More than 3000	29	56	62
	1000–3000	144	140	125
	500–1000	116	119	121
	300–500	108	121	105
	150–300	116	157	178
	Total	563	593	592
	Grand Total	984	1052	1114

Note: The number of commodities in the surveys has changed over the years, so the figures above should be used with some caution. *Source:* Anon., 1968, 1978, 1988 (*Mining Magazine's* Annual Surveys). Conversion factor: 1 ton = 0.9072 t.

23.2.2 SURFACE VS. UNDERGROUND MINING TRENDS

In 1987, it was estimated that there were 1114 noncoal mines in the non-socialist world, producing an average each of more than 165,000 tons (150,000 t) of ore/year. These mines account for approximately 90% of the total output of ore in these countries. In addition, it was estimated that there were 6000 to 7000 smaller mines. Of these 1114 noncoal mines, 522 were surface mines and 592 underground. The average surface mine was much larger in production than the average underground mine. During the last two decades, the number of noncoal surface mines has increased from 421 to 522. Most of these new mines are large (Table 23.2.1). The number of underground mines has not changed as much.

In 1987, there were 1750 hard-coal and lignite mines in the non-socialist world, each with a mine production of more than 220,000 tons (200,000 t) of run-of-mine coal/year. These mines accounted for 80% of the output. About 60% were underground mines. The number of smaller mines was in the thousands. The number of US coal mines in 1985 producing more than 10,000 tons (9000 t) of clean coal annually was 3355. Of these, 1660 or 50% were surface mines. During the last decades, there has been a sharp reduction in the number of smaller coal mines, especially underground. Many large strip mines have been opened, especially in the western states. Most US underground coal mines are located in the eastern states.

In many countries, open pit mining is the predominant mining method. In 1987, some 90% of US crude ore production in metallic mines came from open pits, and production of industrial minerals was as much as 95%. The corresponding percentage of coal mines that were surface operations was 60%. During the last decade or so, the proportion of coal mined by surface methods has increased. In Canada, 72% of the 1986 metallic crude ore production came from surface mines. On the other hand, other countries depend heavily on underground mining. In Sweden, for example, some 80% of the annual crude ore production comes from underground mines.

Different mining methods are used for different minerals. Iron and copper ores are mostly mined in open pits. Most industrial minerals are mined in open pits (also called quarries). Lead, zinc, etc., are mined principally underground. Gold until now has been mined mostly underground, such as in South Africa, but during recent years, there has been a strong trend to surface gold mines, especially in the United States.

Large amounts of waste are also handled in surface mines. About twice as much waste as ore has been mined in US open pit metal mines during the recent decades, although stripping activities declined in the mid-1980s. In 1986, the average stripping ratio was 1.5 tons/ton (t/t). In 1986, the average stripping ratio of Canadian metal mines was 0.8 tons/ton (t/t).

23.2.3 MINE DEVELOPMENT

Surface mining of coal involves the removing of cover layers of soil and rock to expose the coal and then replacing and revegetating the soil when mining is completed. Surface mining is utilized when the coal seam is located relatively close to the surface, usually within 200 ft (60 m). Removal of the overburden takes place only a short time before coal mining takes place. For larger horizontal deposits, covered by large amounts of overburden and waste, much waste rock must be stripped away before ore production can start.

For steeply dipping ore bodies occurring near the surface, open pit mining can start with only a minimum of stripping, but when mining goes deeper, more and more waste rock must be removed. Waste rock in the open pit often must be removed many years before mining of the corresponding amount of ore at deeper benches is possible. The ultimate pit limits must therefore be known at an early point in time, and the interest rate cost for waste rock removal must be included in the economic evaluation of the ultimate pit depth.

Waste rock stripping should be delayed as long as possible to avoid high interest cost for all the money spent in waste stripping activities. The waste rock within the planned ultimate pit limits should, if possible, be mined out in a sequence of push-backs or expansions. Each push-back will make the pit deeper or larger and the pit life longer. The exact point in time when a new push-back must be started depends on machinery capacity, etc. More than one push-back can be in progress at the same time. Waste stripping activities can vary from year to year.

In an underground mine, hoisting shafts, ventilation shafts, and other major access openings require long-range planning from the very beginning if they are to meet the requirements of future workings at deeper levels. Underground mining is a complex undertaking, which includes the development of drifts, manways, rock support, etc. A large capital investment is often necessary before production can start.

It is very important that the underground mine design and the machinery capacities are properly chosen from the very beginning. By the time one discovers that the mine layout is wrong, or the capacity not what was expected, it is often too difficult to do anything about it. There are three reasons that explain how this may happen: (1) rock conditions often limit the possibility for reconstruction underground; (2) long construction and development times often mean that it is too late to correct mistakes after mining begins; and (3) the fact that mining is a highly capital-intensive business means that a planning mistake can destroy the profitability of the whole venture.

The development of a large underground mine can take from five to ten years. The interest cost during the construction time will thus be high and can account for up to 30 to 40% of the capital requirement before mining can start.

The possibility of using trucks for hauling the broken ore through inclined underground ramps up to the crusher on the surface should be examined in an actual operation. Such a solution can be very advantageous, especially in small- and medium-sized underground mines, to achieve early production and reduce capital requirements and interest costs.

23.2.4 PRODUCTIVITY, TIME UTILIZATION, AND LABOR COSTS

Normally, productivity in a surface mine is much higher than in a comparable underground mine (Table 23.2.2). This means that underground mines account for a disportionately larger percentage of the work force. In 1987, the average productivity in US open pit metal mines was about 53 tons (48 t) of waste and ore/employee-hr. The corresponding value underground was only 3.8 tons (3.4 t). The coal industry produced 5.4 tons (5.0 t) of clean coal/employee-hr in the surface mines in 1987, compared with only 2.4 tons (2.2 t) underground.

Productivity, especially in open pits, has improved significantly during recent decades, mostly as a result of the introduction of larger and more flexible equipment. Large open pits normally have a productivity that is much higher than smaller ones. Rock conditions and limited space constrain the use of larger equipment underground. Table 23.2.3 shows productivity in US coal mining by production size groups. The table shows that there are no major differences in productivity between surface mines and underground mines, or between different production size groups, as long as production is less than 1 million tons (0.9 Mt)/year. However, for a higher annual production, the surface mines show a considerably higher productivity.

Productivity, especially underground, strongly depends upon the mining method, the rock conditions, the kind of mineral, the degree of automation, etc. (Table 23.2.4). The huge Swedish underground iron ore mines have a productivity of approximately 12 to 15 tons (11 to 13 t) of ore/employee-hr. One reason behind the higher productivity in surface mines is that less working time is normally consumed for such things as transportation between the mine mouth and the working face. The higher productivity in surface mines thus makes the labor cost per ton produced lower than underground.

Wage agreements can make a difference in the daily, weekly, and yearly working time on the surface and underground. If an underground miner has a weekly working time of 40 hr/wk, his/

Table 23.2.2. Productivity in US Metal and Coal Mines in 1987

	Million Short Tons	Million Employee-hr	Short Tons/ Employee-hr
METAL MINES			
Surface			
Crude ore	500*	22.9	22.0
Waste and ore	1200*	22.9	53.0
Underground			
Waste and ore	60*	16.2	3.8
COAL MINES			
Surface			
Clean coal	533	98.4	5.4
Underground			
Clean coal	365	153.3	2.4

*Preliminary. Conversion factor: 1 ton = 0.9072 t.
Sources: Anon., 1987d (*Mineral Yearbook,* US Bureau of Mines); Anon., 1987b (*Mine Injuries and Worktime Quarterly,* MSHA).

Table 23.2.3. Productivity in US Coal Mines by Production Rate in 1987

Yearly Coal Production Thousand Short Tons	Short Tons of Coal/Employee-hr Surface	Underground
0– 50	2.0	1.8
50– 100	2.7	2.6
100– 150	2.9	2.9
150– 250	3.1	2.8
250– 500	3.6	2.6
500– 750	3.5	2.4
750–1000	3.9	2.2
1000–	8.3	2.8
Average	5.4	2.6

Source: Anon., 1987a (*Injury Experiences in Coal Mining 1977,* MSHA). Conversion factor: 1 ton = 0.9072 t.

Table 23.2.4. Production and Employment in the US Mineral Industry in 1987 by Commodity Mined

	Metal Mines	Non Metals	Stone, Sand and Gravel	Coal Mining
Production, Million tons of ore, coal, etc.	500**	1400*		898
Percentage mined by:				
Surface	90	95*		60
Underground	10	5*		40
Average number of employees, thousands	42	32	121	159
Percentage assigned to:				
Surface	29	22	55	32
Underground	21	12	2	52
Plants	39	52	28	10
Shops, offices	12	14	15	6
	100	100	100	100

* Combined figures for nonmetals, stone, sand and gravel
** Preliminary. Conversion factor: 1 ton = 0.9072 t.
Source: Anon, 1987b (*Mine Injuries and Worktime Quarterly,* MSHA).

her colleague on the surface may be required to work 42 hours. This difference affects the size of the working force. Lower utilization of the working time underground works in the same direction.

Many surface mines operate 24 hr/day and almost 365 days/yr. The high capital requirements for purchase of open pit equipment make it important to use as many hours per year as possible so the number of machines can be reduced. In the case of a machinery breakdown, it is often easy to move the vehicle and replace it with a spare, so that ore production can resume. Underground mines, on the other hand, often must limit their daily, weekly, and yearly operating times because of the need for ventilation and maintenance. In the case of underground coal mines, tradition and union attitudes have so far prevented work crews from being rotated into weekend time slots. This makes the number of mines greater than what it could be with a higher equipment time utilization and the capital requirement higher per ton of capacity.

Often miners are also paid more per hour underground than for a similar job on the surface. This, in combination with better productivity and time utilization on the surface, will often result in a much higher labor cost per effective working hour underground than on the surface.

In surface mines, the largest part of the work force is normally employed in the transportation operations, especially in a large pit in which the haul distance between the face and the primary crusher on the surface can be long. In underground mines, especially in smaller ones, a larger part of the work force is engaged in drilling and blasting.

23.2.5 SAFETY AND INJURY RATES

Concern for health and safety in a mine is a prime responsibility of the mining engineer. During the last decade, the injury rate in the US mineral industry has been cut by some 50%, from some 40 injuries/million employee-hr to 20. The injury rate has dropped in surface mines as well as in underground mines (Table 23.2.5). But the injury rate is much higher underground than on the surface, some 40 compared to less than 20 injuries/million employee-hr. However, 1987 was a year with a high incidence rate.

Table 23.2.5. Injury Rate in the US Mineral Industry in 1987

	Injuries All Occurrences	Injury Rate/ Million Hours	Injury Rate/ Million Tons*
METALLIC MINES			
Surface	736	32.1	1.5
Underground	969	60.0	16.0
NONMETALLIC MINES			
Surface	332	25.6	N/A
Underground	344	43.7	N/A
COAL MINES			
Surface	2,710	27.9	5.1
Underground	12,124	79.0	33.3

* Tons of crude ore or clean coal, respectively.
Note: MSHA normally shows the incidence rate per 200,000 hours.
Source: Anon, 1987b (*Mine Injuries and Worktime Quarterly,* MSHA).

In addition, one may wish to account for the difference in productivity. If one takes this into account, one finds that the injury rate per million tons of product produced is 5 to 10 times higher underground than on the surface and 10 to 15 times higher per ton handled. From the standpoint of human welfare, it would thus be very advantageous if even more ore could be mined by surface methods.

23.2.6 ENVIRONMENTAL ASPECTS

A surface mine normally makes a deeper and more visible encroachment on the environment than an underground mine. The often large amount of waste and overburden must be placed somewhere. Noise and vibration often disturb the environs as well. It can be necessary to place a safety zone around a pit to protect people and equipment from fly rock.

Surface coal mine operators must comply with the strict requirements and regulations of the Federal Surface Mining Control and Reclamation Act of 1977. Coal industry reclamation must return surface mined land to productive use, such as agriculture, wildlife areas, and recreation sites.

Underground, there are environmental problems with respirable dusts, diesel fumes, gases from explosives, methane gas in coal mines, etc. The cost for ventilation often accounts for a considerable part of the cost for underground mining. The need for ventilation after blasting sometimes necessitates a shutdown of production for several hours. Only in very cold or hot climates do miners find it better to work underground than in a surface mine.

23.2.7 ENERGY CONSUMPTION

As a result of the soaring price of petroleum products in the early 1970s, the interest in energy conservation has been especially high during the last decade. Although the price has declined somewhat today, it is important to keep up the interest in energy consumption.

Published data indicates that most surface operations require 5 to 10 kWh energy equivalents/ton of rock handled. Most of this is diesel fuel, although the use of electric power has increased in recent years because of trolley-assist trucks and in-pit crushing and conveying. Underground mining normally requires an average of 15 to 30 kWh of energy equivalent/ton, with up to 50 kWh in smaller cut and fill operations. A large part of this is electricity. During recent years, electric trucks and load-haul-dump (LHD) equipment have been developed.

Although energy consumption/ton handled is much lower in a surface mine than in an underground mine, energy costs often account for a larger part of the total mining cost per ton of ore or per unit of metal extracted in a surface mine than in an underground mine. Future increases in the price of energy, especially of petroleum products, will reduce the competitiveness of surface mining methods as a consequence.

23.2.8 CONSUMPTION OF EXPLOSIVES

Removal of the overburden and coal in surface coal mining sometimes requires no drilling and blasting, but most rocks require that drilling and blasting be employed before materials handling occurs. The costs for drilling and blasting, however, are normally lower in a surface mine than underground. There are several reasons for this difference. First, surface haulage equipment, especially rear-dump trucks, can normally handle coarser material than underground equipment. This makes it possible to minimize the powder factor in surface mines. Also it is often possible to use larger hole diameters and spacing in surface drilling than in underground drilling. In surface operations, hole diameters of 10 to 15 in. (250 to 400 mm) are common, while most underground mines, typically use holes less than 6½ in. (165 mm). Tunneling and drifting usually are performed using even smaller hole sizes.

In a surface mine, one normally has multiple free surfaces or faces to blast to, while underground, only one face may be available. This makes a lower powder factor more feasible in surface mines. In those US surface metal mines where drilling and blasting are required, the average powder factor has been 0.3 to 0.5 lb/ton (0.5 to 0.7 kg/m^3) of blasted rock in situ in recent years. Underground, often twice the powder factor is required. But conditions vary significantly from mine to mine and with the type of rock.

The coal industry as a whole is the largest consumer of explosives, using some 70% of the total US industrial consumption. Nearly all of this is used in surface coal operations.

23.2.9 MINING COSTS

In surface mines, especially larger ones, the haul distance from the face to the primary crusher can be long, and the haulage cost will thus account for a large part of the total costs. In underground mines, especially smaller ones, the costs for development drilling and blasting are often more important. Drilling, blasting, loading, and haulage are examples of direct costs in a mine. In a cost calculation, it is important to also include auxiliary costs.

In a surface mine, auxiliary costs like rock support, water pumping, and road maintenance are normally small and account for about 10% of the total mining costs. Underground such auxiliary costs are higher, often 20 to 40% and include costs for ventilation, lighting, rock support, etc. The total cost per ton of material handled is normally much lower in a surface mine than in an underground mine.

In an estimation of the total cost per ton of ore or coal produced, one also has to take waste rock mining into account, especially in a surface mine where the stripping ratio can be high.

Table 23.2.6. Costs in a Surface Mine

Annual production: 5.0 million tpy (4.5 million t/yr)	
Corresponding to: 15,000 tons/day (13,500 t/day)	
Investment in mining equipment:	
Drilling and blasting	\$ 1.3 million
Loading	\$ 3.6 million
Haulage	\$ 7.1 million
	\$12.0 million
Annual capital cost (10%, 5 yr):	
\$12.0 million × 0.26 =	\$ 3.1 million/year
Capital cost/ton:	\$ 0.60/ton (\$0.67/t)
Operating cost:	
Drilling and blasting	\$ 0.30/ton (\$0.33/t)
Loading and haulage	\$ 0.80/ton (\$0.88/t)
Auxiliary costs	\$ 0.10/ton (\$0.12/t)
Total mining cost:	\$ 1.80/ton (\$2.00/t)

Note: 1988 costs.
Source: Calculated from Anon., 1987c (*Cost Estimating System Handbook*, US Bureau of Mines).

23.2.9.1 Surface Mining

The cost for surface mining varies from mine to mine. Larger mines have costs per ton handled which are considerably lower than in smaller operations. Table 23.2.6 shows an estimation of the total operating cost, including capital costs for mining equipment, for a surface mine. Of the total cost, \$1.80/ton (\$2.00/t) handled, loading and haulage accounts for 74%.

23.2.9.2 Underground Mining

Underground mining costs vary significantly from mine to mine and depend on such factors as production levels, mining method, rock support, etc. Cut and fill stoping is an expensive method. Block caving, on the other hand, can sometimes compete with surface mining. Table 23.2.7 shows an estimation of the operating costs in a medium-sized open stoping mine, using long, parallel, large-diameter blastholes. It is based upon material published by the Swedish Mining Research Foundation from its Luossavaara Research Mine in northern Sweden. The costs apply to the mining of a 330-ft (100-m) high block of this deposit. On the top of the block, a drilling level was developed. At the bottom of the block, there was a drawpoint level, where LHD equipment loaded the ore into trucks for haulage via an inclined ramp to the surface (Fig. 23.2.1).

Table 23.2.7 illustrates that a calculation of the costs for underground mining includes the estimation of the costs for many different items. The total cost is \$14/ton (\$15/t) of crude ore, of which development in this mine accounts for as much as 43%. The table also illustrates that underground mining, on a unit cost basis, is much more expensive than surface mining, although the difference in production volume, etc., make future conclusions difficult.

In a surface mine, a large part of the operating costs is variable. If a surface mine operator must decrease production, it is often possible to sell the mining equipment, because there is a large market for such equipment that is also used by other industries.

Underground mining requires more mine-specific equipment. In addition, large amounts of money are often invested in shafts and haulage systems. It is, therefore, more likely that an underground mine will keep producing as long as the revenues cover the variable operating costs, even though the operation is not profitable on an overall cost basis.

Table 23.2.7. Operating Costs in an Underground Mine

Production data summary:			
— Total amount of recoverable ore: 1.50 million tons (1.36 Mt) of which 100,000 tons are from development			
— Waste rock dilution: 10%	0.17 million ton (0.15 Mt)		
— Total amount of crude ore:	1.67 million tons (1.51 Mt)		
— Lifetime: 4 yr			
— Annual amount of crude ore:	0.42 million ton (0.38 Mt)		
— Annual amount of ore excluding waste:	0.38 million ton (0.34 Mt)		
Development costs:			$ Million
— Example drilling	18,000 ft ×	$30/ft	
	(5,500 m ×	$100/m)	= 0.6
— Drifts	10,000 ft ×	$400/ft	
	(3,000 m ×	$1,300/m)	= 4.0
— Raises	2,000 ft ×	$350/ft	
	(600 m ×	$1,200/m)	= 0.7
— Cable bolting	70,000 ft ×	$8/ft	
	(21,000 m ×	$30/m)	= 0.6
— Instruments			= 0.1
— Undercut level and draw points			= 0.6
— Misc.			= 0.4
			7.0
Annuity (4 yr, 10%)			$ Million/year
	$7.0 million × 0.32		2.2
Development cost/ton of ore (excluding waste)			$/ton ($/t)
	$\frac{\$2.2 \text{ million/yr}}{0.38 \text{ million tpy}} =$		5.8 (6.4)
Operating Costs:			$ Million
— Drilling	150,000 ft ×	$10/ft	
	(45,500 m ×	$33/m)	= 1.5
— Blasting	1.40 million tons ×	$0.5/ton	
	(1.26 Mt ×	$0.3/t)	= 0.7
— Loading	1.57 million tons ×	$1.6/ton	
	(1.41 Mt ×	$1.8/t)	= 2.5
— Haulage	1.57 million tons ×	$1.6/ton	
	(1.41 Mt ×	$1.8/t)	= 2.5
— Crushing	1.67 million tons ×	$0.1/ton	
	(1.51 Mt ×	$0.1/t)	= 1.7
— Hoisting	1.67 million tons ×	$0.2/ton	
	(1.51 Mt ×	$0.2/t)	= 0.3
— Auxiliary Costs Underground, 25%			= 2.4
— Misc.			= 0.4
			12.0
Operating cost/ton of ore (excluding waste)			$/ton ($/t)
	$\frac{\$12.0 \text{ million}}{1.5 \text{ million tons}} =$		8.0 (8.8)
Total cost/ton of ore (excluding waste)			
	5.8 + 8.0 =		13.8 (15.3)

Note: Capital costs for mining equipment included in unit costs. No investment costs for haulage and ventilation system, crushers, etc., included.
Source: Nilsson, 1985.

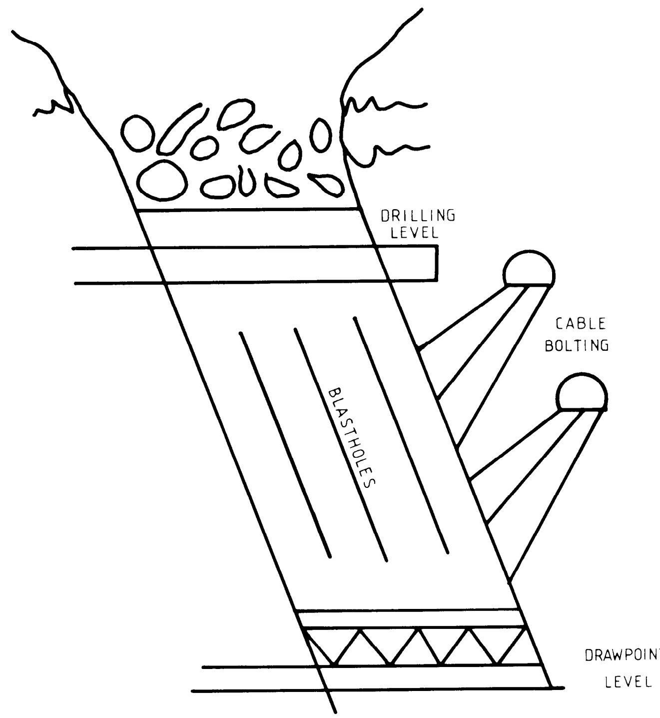

Fig. 23.2.1. Cross section through a medium-sized underground mine.

23.2.10 ORE RECOVERY, CUTOFF GRADE, AND WASTE ROCK DILUTION

In a surface mine, it is normally possible to recover almost all the mineral or coal one wants to mine. The recovery underground is usually lower and depends upon the mining method used. With cut and fill stoping, it is often possible to get a high recovery, but the cost is also high. With caving methods, it is normally not possible to extract all the blasted ore, because waste rock dilution will be too high (15 to 25% of the ore is normally left). With stoping methods and room and pillar mining, 15 to 25% of the ore is often left in supporting pillars.

Underground coal mining recovers an average of 50% of the coal with the room and pillar system and up to 80% using the longwall system. Underground mine costs are normally higher than surface mining costs, which means that a lower cutoff grade can normally be used in a surface mine. As a general rule, surface mining allows extraction of a larger part of a deposit than if underground methods are used.

Cut and fill mining is a selective but expensive mining method that makes high ore recovery and low dilution possible. This is also possible in the primary stages of room and pillar mining and open stoping. Other underground methods may result in waste rock dilution from caving roofs and hanging walls. Often 20 to 30% of the extracted tonnage is country rock, resulting in extra costs. Often such dilution has a coarse fragmentation, which causes extra costs and loss of capacity in underground mines.

23.2.11 GENERAL CONSIDERATIONS IN THE CHOICE BETWEEN SURFACE AND UNDERGROUND MINING

If an ore body or coal seam is situated at a relatively great depth, an experienced mining engineer can readily evaluate the

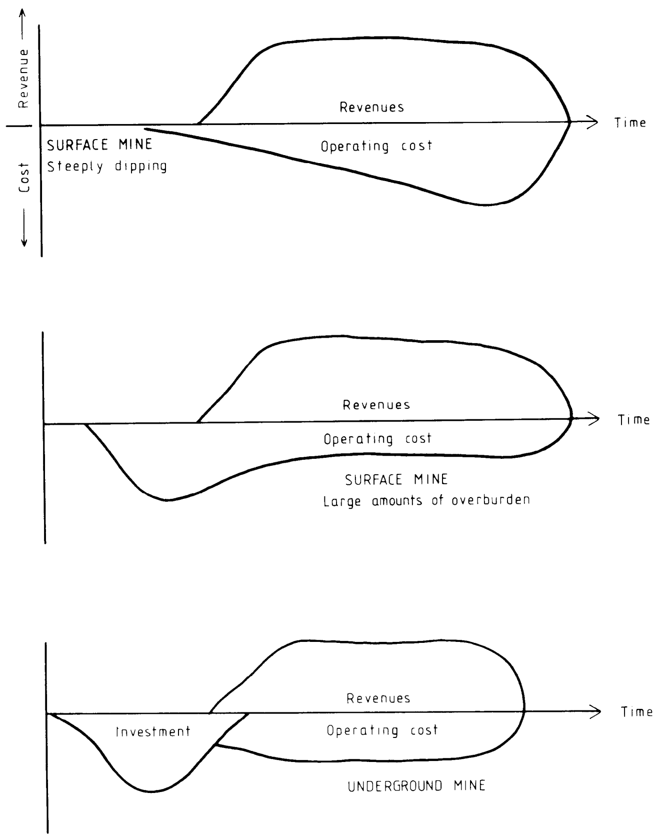

Fig. 23.2.2. Examples of cash-flow profiles for surface vs. underground mines.

possibility of surface mining by estimating the total stripping ratio for a surface mining venture. If the estimated stripping ratio essentially exceeds that of similar mines, surface mining can probably be ruled out. However, there are examples where bad rock and working conditions underground can make surface mining necessary in spite of a high stripping ratio. There are also examples where consideration of the environment around a mine site necessitates underground mining.

Secondly, there are cases where one has to go through a feasibility study before one can decide whether to use surface or underground methods. An economic comparison between open pit or underground mining of a deposit should begin with sound reserve and grade estimation. Cutoff grades, ore recovery, etc., will be different.

At the next stage, surface mine designs as well as underground mine designs must be studied; waste rock stripping, dilution, etc., should be estimated. A reasonable annual production rate must be selected, considering the market, the size of the reserve, capacity aspects, etc. Machinery, energy, and labor requirements should be studied for surface as well as for underground mining. Reasonable capital and operating cost estimates must be made, revenues estimated, and cash-flow profiles shown for each alternative.

These cash-flow profiles can be quite dissimilar, and the choice between a surface venture and an underground operation can thus affect a company's financial situation for many years, even if the final profitability for the two alternatives is the same. Fig. 23.2.2 illustrates how the cash-flow profiles can differ. The first case describes a steeply dipping ore body near the surface. In the beginning, waste stripping activities and thus the operating costs are low, especially if the equipment can be leased or a contractor used. As mining progresses deeper, more and more waste must be removed.

In the second case, a large amount of overburden and waste must be stripped before ore mining can start. This will require much capital in the early stage.

The third case describes an underground mine where large capital requirements are required before any revenues accrue. The life of the mine will also be shorter because of a lower recovery. The financial strength of a company can thus affect the choice. With a shortage of funds, it can be necessary and attractive to choose a solution that requires less capital.

Finally, surface and underground mining can be compared by calculating total mining costs, net present values, etc. In such a comparison, all factors and costs that are different between the two alternatives, as well as a discount rate, must be included. The possibility of future increases in productivity, automation, etc., should be studied and the effects of additional regulations and environmental restrictions discussed. Sensitivity and risk analyses should be used to show what happens in the case of future drops in demand, labor shortages, increased energy costs, etc.

23.2.12 STUDY OF HYPOTHETICAL DEPOSITS

To illustrate how to compare open pit with underground mining, two simplified examples are shown in the following. The first case is a horizontal deposit covered by large amounts of overburden. This deposit can be entirely mined out by open pit or underground methods. The second case is a steeply dipping deposit, which will initially be mined as an open pit but mined underground later.

The size of the deposit is the same: 3300 by 330 by 1650 ft (1000 by 100 by 500 m). This corresponds to a total amount of 165 million tons (150 Mt), given by:

$$3300\,\text{ft} \times 330\,\text{ft} \times 1650\,\text{ft} \times 0.09\,\text{tons/ft}^3 = 165 \times 10^6\,\text{tons}$$

$$(1000\,\text{m} \times 100\,\text{m} \times 500\,\text{m} \times 3.0\,\text{t/m}^3 = 150 \times 10^6\,\text{t})$$

The copper content averages 2% and is constant in all parts of the ore body. There are no byproduct metals present. The yearly crude ore production will be assumed to be 5 million tons (4.5 Mt)/yr.

With open pit mining, diesel trucks will haul the rock up to the surface. All open pits will have a slope angle of 45°. The crude ore will be milled in a plant on the surface. The' copper concentrate from the flotation plant will be transported to a smelter and refinery in the area for further processing, resulting in refined copper. The annual output of refined copper will be 75,000 tons, given by:

$$\begin{aligned}&5 \text{ million tpy} \times 2\% \text{ Cu} \times 75\% \text{ recovery}\\&= 75{,}000 \text{ tons/yr (68 kt)}\end{aligned}$$

In the case of underground mining, sublevel stoping with high blocks will be used with drawpoints and a loading level at the bottom of each block. It will be assumed that a haulage level is developed at the bottom of the ore body. The broken rock will be dumped from the loading levels into ore passes. From the bottom of the ore passes, trucks will haul the ore to a primary crushing station. The ore will then be hoisted up to the surface through a vertical shaft. The following costs will be used:

Waste rock stripping	\$1.80/ton (\$2.00/t) of waste
Open pit ore mining	\$1.80/ton (\$2.00/t) of ore
Underground mining	\$13.50/ton (\$15.00/t) of ore

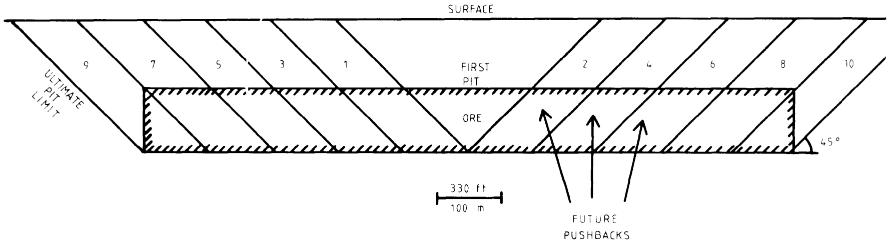

Fig. 23.2.3. Longitudinal section through hypothetical copper deposit.

Investment in underground mine	$75 million
Ore beneficiation and misc.	$40 million/year
Investment in mill and misc.	$200 million
Discount rate	10%

For the sake of simplicity, the mining cost assumed here will be independent of depth. Two different copper prices will be used: $2 and $3/kg of refined copper (corresponding to approximately $1 and $1.50/lb).

23.2.13 OPEN PIT OR UNDERGROUND MINING OF HORIZONTAL DEPOSIT WITH VARYING OVERBURDENDEPTH

Example 23.2.1. The deposit has a length of 3300 ft (1000 m), an average width of 1650 ft (500 m), and a thickness of 330 ft (100 m). Three cases will be assumed in which the deposit is covered by 330 ft (100 m), 500 ft (150 m), or 660 ft (200 m) of overburden. This deposit can be entirely mined by open pit or underground mining, and one has to choose the mining method from the very beginning.

To get open pit production started, a large amount of waste must first be removed. Additional parts of the ore area will later be available after new push-backs. These waste quantities will be higher the deeper the deposit is situated. Fig. 23.2.3 shows a longitudinal section through the deposit in the case of open pit mining. If the amount of overburden over the deposit is too thick, underground mining is of interest. The question is: How thick can the overburden be before that happens?

Table 23.2.8 shows some mine planning data. Before open pit ore production can start, a total of 81,145 and 230 million tons (73,132 t and 208 Mt) of waste must be removed. Assuming that an average two 55-ft (17-m) benches of overburden can be removed per year, waste rock stripping must start 3, 4.5, and 6 years before ore mining can be initiated.

Table 23.2.9 shows the calculation of the net present values (NPVs) for open pit mining and different thickness of the waste over the deposit; a comparison of results is as follows:

	NPV ($ Million)	
Waste Thickness	$1.00/lb ($2.00/kg)	$1.50/lb ($3.00/kg)
330 ft (100 m)	−84.9	+561.0
500 ft (150 m)	−340.4	+305.5
660 ft (200 m)	−658.9	−12.9

With $1.00/lb ($2.00/kg), all values are negative, which means that it will not be profitable to invest $200 million to get production started. But the values also show that, with a copper price of $1.50/lb ($3.00/kg), it would be profitable to invest $200 million, even if the deposit is covered by almost 660 ft (200 m) of overburden.

How will the possibility of using underground mining affect the conclusion? Table 23.2.10 shows the calculation of the NPV of the deposit mined underground. The table shows that, with a copper price of $1.00/lb ($2.00/kg), the revenues will not be high enough to cover operating costs. But at a price of $1.50/lb ($3.00/kg), it will be profitable to make all the investments and start underground mining.

If one compares the results from Tables 23.2.9 and 23.2.10, one can conclude that, as long as the deposit possesses less than

Table 23.2.8. Summary of Mine Planning Data for a Horizontal Deposit

	Waste Thickness		
Open Pit Mining	330 ft (100 m)	500 ft (150 m)	660 ft (200 m)
Waste, million tons			
— first pit	81.1	144.9	230.0
— in each push-back 1–8	26.8	40.8	56.6
— in each push-back 9–10	34.2	49.3	63.9
	363.8	570.0	810.5
Ore, million tons			
— first pit	16.5	16.5	16.5
— in each push-back 1–8	16.5	16.5	16.5
— in each push-back 9–10	8.3	8.3	8.3
	165.0	165.0	165.0
Average stripping ratio	2.2/1	3.4/1	4.9/1
Annual ore production million, tpy	5.0	5.0	5.0
Lifetime, yr	33.0	33.0	33.0
Year when waste rock stripping must start	−3.0	−4.5	−6.0
Waste rock stripping million, tpy			
— yr − 6			38.3
— yr − 5		16.1	38.3
— yr − 4		32.2	38.3
— yr − 3	27.0	32.2	47.7
— yr − 2	27.0	41.2	47.7
— yr − 1	27.0	41.2	47.7
— yr 1–30	9.4	13.6	18.4
Underground Mining			
In Situ Ore, million tons	165.0	165.0	165.0
Lifetime, yr	26.5	26.5	26.5

given by: $$\frac{165.0 \text{ million tons} \times 0.8 \text{ recovery}}{5.0 \text{ million tons/yr}}$$

Conversion factor: 1 ton = 0.9072 t.

Table 23.2.9. Net Present Values of Open Pit Mining of a Horizontal Deposit With Different Thicknesses of Waste

	Waste Thickness		
	330 ft (100 m)	500 ft (150 m)	660 ft (200 m)
Revenues per year, \$ million, 75,000 tons × \$1800/ton	135.0	135.0	135.0
Operating cost per year (excluding waste), \$ million			
Ore mining			
5.0 million tons × \$1.8/ton	−9.0	−9.0	−9.0
Ore beneficiation and misc.	−40.0	−40.0	−40.0
Smelting and refining			
75,000 tons × \$540/ton	−40.5	−40.5	−40.5
Annual income, \$ million (excluding costs for waste rock)	+45.5	+45.5	+45.5
Net present values, \$ million (10%)			
NPV of annual income, excluding waste rock (33 yr)			
\$45.5 million tpy × 9.57	435.4	435.4	435.4
NPV of costs for waste rock stripping			
\$1.80/ton × (27.0 × 3.31ˣ + 9.4 × 9.42ˣˣ) =	320.3		
\$1.80/ton × (16.1 × 1.46ˣˣˣ + 32.2 × 1.33 + 32.2 × 1.21 + 41.2 × 1.10 + 41.2 × 1.00 + 13.6 × 9.42) =		−575.8	
\$1.80/ton × (38.3 × 7.72ˣ + 9.4 × 3.31ˣ + 18.4 × 9.24ˣˣ) =			−894.3
Capital investment	−200.0	−200.0	−200.0
Grand total, \$1.00/lb (\$2/kg)	−84.9	−340.4	−658.9
Grand total, \$1.50/lb (\$3/kg)	+561.0	+305.5	−12.9

ˣ uniform series future-value factors.
ˣˣ uniform series present-value factors.
ˣˣˣ single present-value factors.
Conversion factor: 1 ton = 0.9072 t.

Table 23.2.10. Net Present Value of Underground Mining of Horizontal Deposit

Revenues per year, \$ million	
75,000 tons × \$1800/ton	135.0
Operating cost per year, \$ million ore mining	
5.0 million tons × \$13.50/ton	−67.5
Ore beneficiation and misc.	−40.0ˣ
Smelting and refining	−40.5ˣ
Annual income, \$ million, at \$1.00/lb (\$2/kg)	−13.0
Annual income, \$ million, at \$1.50/lb (\$3/kg)	+54.5
Net present values (10%), \$ million	
NPV of annual income, \$1.50/lb (\$3/kg) (26.5 yr)	
\$54.5 million/year × 9.20 =	+501.4
Underground investment	−75.0
Capital investment on surface	−200.0
Grand total, \$1.50/lb (\$3/kg)	+226.4

ˣ Same costs used as for open pit mining. Waste rock dilution can increase these costs. Conversion factor: 1 ton = 0.9072 t.

550 ft (165 m) of waste, open pit mining is preferable. If the deposit is situated deeper, underground mining should be used. This solution indicates that open pit mining should be used as long as the average stripping ratio is less than 3.5 tons/ton (t/t). If more waste rock must be removed per ton of ore, underground mining is preferable. By simply comparing the capital values of all costs only, an approximate solution to a problem of this sort can be made. But, because of the difference in ore recovery, it is recommended also to include the revenues in any economic analysis.

23.2.14 OPTIMIZING DEPTH IN CONVERTING FROM OPEN PIT TO UNDERGROUND

When a surface mine is deepened, more and more waste rock must be removed. Surface mining can continue as long as the cost for mining another bench is lower than underground mining of that ore quantity and as long as the costs are covered by revenues. To understand how to optimize the depth in changing from open pit to underground mining, a simplified example is utilized.

Example 23.2.2. The deposit is a hypothetical, steeply dipping copper deposit, with the same dimensions as in the previous example: length of 3300 ft (1000 m), average width of 330 ft (100 m), and depth of 1650 ft (500 m). This deposit should first be mined with open pit methods. Four different open pit depths will be studied: 330, 500, 660, 825 ft (100, 150, 200 and 250 m). Table 23.2.11 summarizes some planning data for mining down

Table 23.2.11. Summary of Mine Planning Data for Steeply Dipping Deposit

	Final Open Pit Depth			
Open Pit Mining	330 ft (100 m)	500 ft (150 m)	660 ft (200 m)	825 ft (250 m)
Waste, million tons	40.2	98.0	173.4	281.4
Ore, million tons	33.0	50.0	66.0	83.0
Stripping ratio, average	1.2	1.9	2.6	3.4
Stripping ratio, for deepest part	2.5	3.3	4.7	6.5
Annual ore production, million tpy	5.0	5.0	5.0	5.0
Lifetime, yr	6.7	10.0	13.3	16.7
Waste rock stripping, million tpy	6.0	9.8	13.1	17.0
Underground				
In situ ore, million tons	132.0	115.0	99.0	82.0*
Lifetime, yr	21.3	18.7	16.0	13.1**

*given by: 165.0 − 83.0 = 82.0 million tons

$$** \text{ given by: } \frac{82.0 \text{ million tons} \times 0.8 \text{ recy.}}{5.0 \text{ million tons/yr}} = 13.1 \text{ yr}$$

Conversion factor: 1 ton = 0.9072 t.

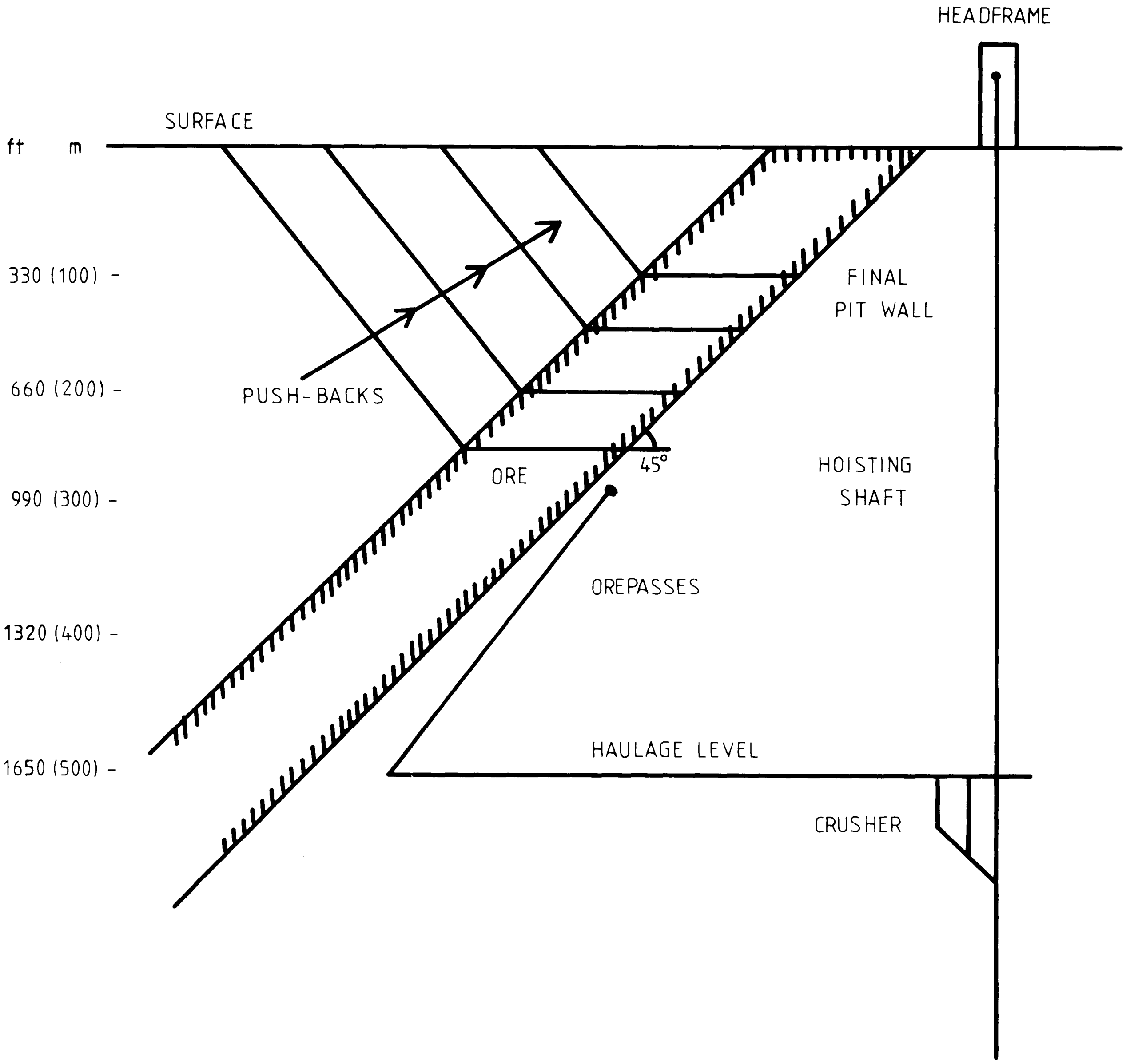

Fig. 23.2.4. Cross section through hypothetical steeply dipping copper deposit.

to the 1650-ft (500-m) depth. Fig. 23.2.4 shows a cross section through the ore body with alternative open pit depths as well as underground installations.

In an actual situation, detailed open pit layouts should be developed for each ultimate depth. As a next step, detailed mining plans, especially for overburden stripping, should be developed. (In the example below, however, simplified mining plans are used to avoid a lengthy description.)

There are two alternative ways to calculate the final pit depth:

1. Calculate the profit and cost of the ore at different ultimate pit depths. When the cost of deepening the pit exceeds the cost for underground mining, one should change to underground mining. If the cost for underground mining is higher than the corresponding revenue, underground mining is not profitable.

2. Calculate the NPV of the cash flow of the whole mine for different ultimate pit depths. The ultimate depth that yields the highest value will be the best alternative.

These two methods will yield approximately the same result, although each method has its advantages and disadvantages.

The disadvantage of the first method is that the values do not tell the reader anything about the total profitability of the whole mine, and that they cannot be compared with the capital requirement for ore processing plants, etc.

To determine total profitability, the NPV must be calculated. The first method also does not fully take into account the differences in ore recoveries. The second method will therefore be used here.

Table 23.2.12 shows the calculation of the NPV for different open pit alternatives:

	NPV ($ Million)	
Open Pit Depth	$1.00/lb ($2.00/kg)	$1.50/lb ($3.00/kg)
330 ft (100 m)	−16.4	301.5
500 ft (150 m)	−28.7	385.8
660 ft (200 m)	−42.5	442.8
825 ft (250 m)	−81.4	455.9

Table 23.2.12. Net Present Values of Different Open Pit Alternatives for Steeply Dipping Deposit

	Final Open Pit Depth			
	330 ft (100 m)	500 ft (150 m)	660 ft (200 m)	825 ft (250 m)
Revenues per year, $ million				
75,000 ton × $1800/ ton	135.0	135.0	135.0	135.0
Operating cost per year, $ million				
Waste rock stripping on average				
6.0 million tons × $1.80/ ton	−10.8			
9.8 million tons × $1.80/ ton		−17.6		
13.1 million tons × $1.80/ ton			−23.6	
17.0 million tons × $1.80/ ton				−30.6
Ore mining				
5.0 million tons × $1.80/ ton	− 9.0	− 9.0	− 9.0	− 9.0
Ore beneficiation and misc.	−40.0	−40.0	−40.0	−40.0
Smelting and refining 75,000 tons × $540/ton	−40.5	−40.5	−40.5	−40.5
Annual income, $ million	+34.7	+27.9	+21.9	+14.9
Net present values, $ million				
NPV of annual income				
$34.7 million/year × 4.71ˣ	184.4			
$27.9 million/year × 6.14ˣ		171.3		
$21.9 million/year × 7.18ˣ			157.5	
$14.9 million/year × 7.96ˣ				118.6
Capital investment	−200.0	−200.0	−200.0	−200.0
Grand total @ $1.00/lb ($2/ kg)	−16.4	−28.7	−42.5	−81.4
Grand total @ $1.50/lb ($3/ kg)	+301.5	+385.8	+442.8	+455.9

ˣ Uniform series present value factors for 10% and 6.7, 10.0, 13.3, and 16.7 yr, respectively. Conversion factor: 1 ton = 0.9072 t.

Table 23.2.13. Net Present Values of Underground Mining of Steeply Dipping Deposit

Revenues	
75,000 ton × $1800/ton	= $ 135.0 million/year
Operating costs per year	
Ore Mining	
5.0 million tons × $13.50/ton	= $ −67.5 million/year
Ore beneficiation and misc.	= $ −40.0 million/year
Smelting and refining	= $ −40.5 million/year
Income per year, at $1.00/lb ($2/kg)	$ −13.0 million/year
Income per year, at $1.50/lb ($3/kg)	$ +54.5 million/year
Net present values at $1.50/lb ($3/kg)	
Net present values when underground mining starts	
330 ft (100 m):	$54.5 million tpy × 8.68ˣ − 75 = $398.0 million
500 ft (150 m):	$54.5 million tpy × 8.31ˣ − 75 = $378.0 million
660 ft (200 m):	$54.5 million tpy × 7.82ˣ − 75 = $351.2 million
825 ft (250 m):	$54.5 million tpy × 5.62ˣ − 75 = $231.3 million
Net present values when open pit mining starts at $1.50/lb ($3/kg)	
330 ft (100 m):	$398.0 million × 0.53ˣˣ = $210.9 million
500 ft (150 m):	$378.0 million × 0.39ˣˣ = $147.4 million
660 ft (200 m):	$351.2 million × 0.28ˣˣ = $ 98.3 million
825 ft (250 m):	$231.3 million × 0.20ˣˣ = $ 46.3 million

ˣ Uniform series present value factors for 10% and 21.3, 18.7, 16.0, and 13.1 yr, respectively.
ˣˣ Single present value factors for 10% and 6.7, 10.0, 13.3, and 16.7 yr, respectively.

Table 23.2.14. Net Present Values of Open Pit and Underground Mining of Steeply Dipping Deposit

	$ Million, at $1.50/lb ($3/kg)		
Depth	Open Pit	Underground	Total
330 ft (100 m):	301.5	210.9	512.4
500 ft (150 m):	385.8	147.4	533.2
660 ft (200 m):	442.8	98.3	541.1
825 ft (250 m):	455.9	46.3	502.2

At a price of $1.00/lb ($2.00/kg), all values are negative, which means that it will not be profitable to invest $200 million to get production started. The open pit can only be opened if ore beneficiation capacity, etc., is already available. In such a case, the pit should be limited to a depth of less than 330 ft (100 m).

The values also show that if the copper price increases by 50%, to $1.50/lb ($3.00/kg), the open pit can be deepened to 825 ft (250 m) in depth, and it will be profitable to invest money in the mill, etc., to get production started.

An important and often forgotten factor to include is the discount rate. If no discount rate had been included in the calculations above, one would incorrectly draw the conclusion that open pit mining would be economic to much deeper levels. A steeper final pit slope will reduce the necessary waste rock removal and make a deeper final pit possible.

It is now necessary to take the effects of future underground mining into account. Table 23.2.13 provides a study of the underground mining operation. With a $1.00/lb ($2.00/kg) price, the revenues are not high enough to cover the operating costs. Capital values for underground mining, at the point in time when underground mining can start, are as follows

	Capital Values ($ Million)	
Open Pit Depth	$1.00/lb ($2.00/kg)	$1.50/lb ($3.00/kg)
330 ft (100 m)	N.P.	398.0
500 ft (150 m)	N.P.	378.0
660 ft (200 m)	N.P.	351.2
825 ft (250 m)	N.P.	231.3

N.P. = Not Profitable

These cash flow values show that if the copper price (in the future) is only $1.00/lb ($2.00/kg), it will not be profitable to start underground mining, even if open pit mining has been used and the beneficiation plants, etc., are constructed and available. With a copper price of $1.50/lb ($3.00/kg), it will clearly be profitable to start underground mining when open pit mining is completed.

However, these values cannot be compared with the corresponding values for the open pit mining period. To do that, the underground values must be discounted to the same point in time.

The net present values for underground mining, when open pit mining starts, are as follows

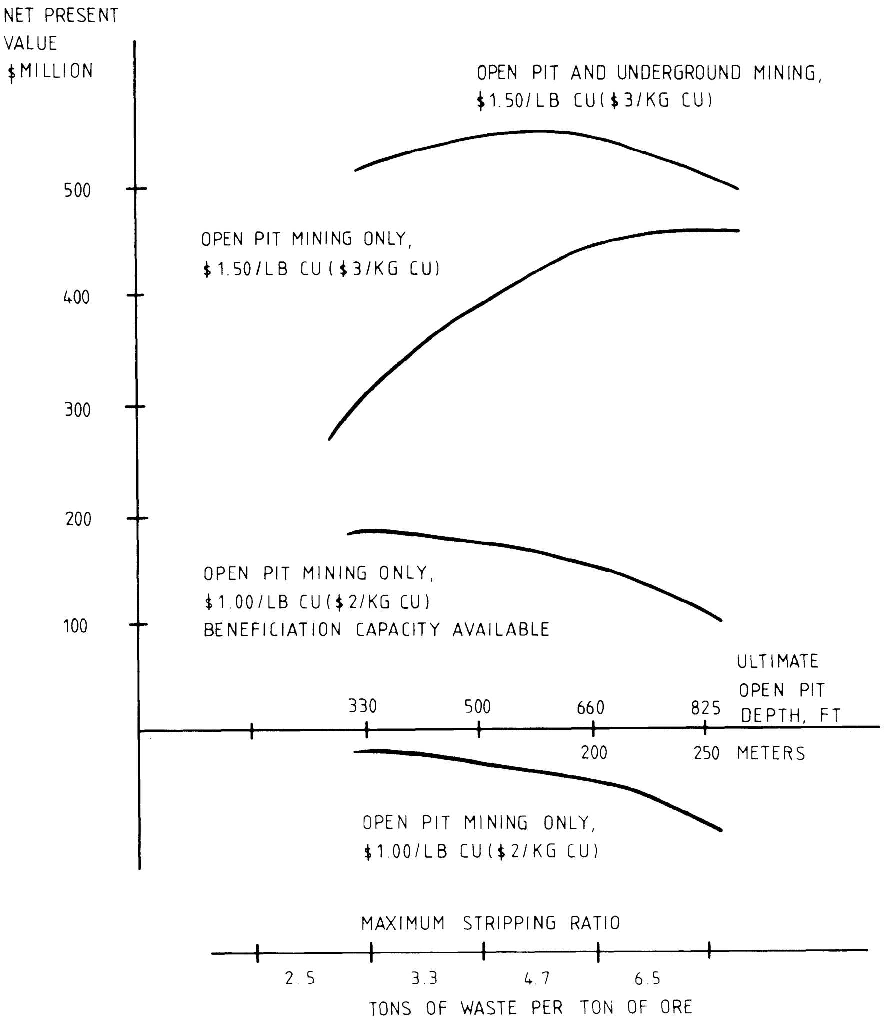

Fig. 23.2.5. Net present value of hypothetical steeply dipping copper deposit for different ultimate open pit depths.

	NPV ($ Million)	
Open Pit Depth	$1.00/lb ($2.00/kg)	$1.50/lb ($3.00/kg)
330 ft (100 m)	N.P	210.9
500 ft (150 m)	N.P	147.4
660 ft (200 m)	N.P	98.3
825 ft (250 m)	N.P	46.3

These values show that although underground mining at $1.50/lb ($3.00/kg) is quite profitable, the NPV is not very high because of the effects of the discount rate.

Finally, the NPV for the whole deposit can be found by adding the value of open pit and future underground mining (Table 23.2.14):

	NPV ($ Million)	
Open Pit Depth	$1.00/lb ($2.00/kg)	$1.50/lb ($3.00/kg)
330 ft (100 m)	−16.4	512.4
500 ft (150 m)	−28.7	533.2
660 ft (200 m)	−42.5	541.1
825 ft (250 m)	−81.4	502.2

The summary above shows that at $1.50/lb ($3.00/kg) Cu, the possibility of a profitable future underground mine will limit open pit mining to a final depth of 500 to 660 ft (150 to 200 m). This means that, in a case like this, one can accept a stripping rate of up to approximately 4.5 tons of waste/ton of ore (4.5 t/t) before one should change to underground mining.

The result is also shown in Fig. 23.2.5; this figure shows that the curves are rather flat around the optimal point, which means that it is not so important to choose a depth that is exactly optimal. With this material as a base, different types of sensitivity analyses can be performed and the effects of future changes in revenues and costs studied.

In the optimization above, the effects of increasing costs when mining at deeper levels has not been taken into account. However, these will have only a minor effect on the conclusions.

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Chapter 23.3
COST COMPARISONS

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23.3.1 INTRODUCTION

Evaluation and selection of mining methods is an involved process that requires a knowledge of mining methods as well as a strong working knowledge of the methods of cost estimation. Knowledge of both areas is necessary because the work of providing cost comparisons requires that the cost estimator be familiar with mining methods to provide accurate cost predictions. The choice of a mining method and the decision whether to pursue the development of a mining property are closely interrelated. Thus this chapter on cost comparisons can be applied to either of these engineering problems.

To introduce the topic of cost comparison, it is useful to establish a set of definitions for the types of cost estimates used in the various stages of consideration of a project. The American Association of Cost Engineers (AACE), an association of cost engineers and related personnel, has established one such set of definitions (Humphreys and Katell, 1981). The AACE estimates are given the following designations:

Type of Estimate	Accuracy	
Order-of-magnitude estimate	–30%	+50%
Preliminary estimate	–15%	+30%
Definitive estimate	–5%	+15%

In this classification scheme, the order-of-magnitude value is a very rough estimate, the preliminary estimate is normally associated with the budget authorization, and the definitive estimate is normally associated with the cost control on the project.

The US Bureau of Mines (USBM) has established a somewhat more complicated cost estimation classification (Lemons, 1988). Its estimation categories have the following definitions:

Type of Estimate	Accuracy
Order-of-magnitude estimate	± 35 to 50%
Factored estimate	± 30%
Budget authorization estimate	± 20%
Definitive estimate	± 10%
Detailed estimate	± 5%

This classification scheme employs similar definitions of accuracy to those of the AACE but adds the factored and detailed estimates. It should also be noted that the USBM scheme defines the accuracy in a somewhat different fashion, keeping positive and negative deviations from the estimates equal.

In our discussions here, we will refer to the three primary definitions outlined above, that is, to order-of-magnitude, preliminary, and definitive estimates. We favor the AACE definitions primarily because the positive deviations are greater than the negative deviations. It is our belief that this classification method is more realistic because cost underestimates are more common than overestimates. Individual estimators may wish to follow the USBM classification scheme.

23.3.2 COST ESTIMATION METHODS

Estimation of mineral industry costs is a specific field of endeavor pursued, in general, by engineering or financial personnel. In many ways, the discipline of cost estimation utilizes the general methods of cost estimation practiced in other fields. There are, however, some important differences that can complicate minerals cost estimation. First, cost information is not as available in the minerals field as it is in other areas. The construction and electronics industries are two examples where a considerable amount of cost information is available in the public domain. This is not true in the minerals industry. Second, the subject of cost engineering is complicated by the variability of the geologic environment and the inability of the minerals engineer to choose his working materials. This results in the actual costs being more variable and difficult to predict than costs for projects where the engineer can more completely control his engineering materials and components.

One additional complicating factor in mine estimating projects may be the remoteness of a given project. Unlike many engineering projects that are often located in urban areas with many competitive contractors, mining projects are often located at remote sites lacking in skilled labor and other resources. This remoteness may add both significant costs and the additional element of uncertainty to the project.

In spite of the differences that exist between minerals and other engineering projects, many of the cost estimation (or cost engineering) methods that are useful to the minerals estimator are procedures that come from the general field of cost estimation. Some of the more useful methods will be outlined in the next segments. Also to be discussed here are methods of updating costs, which should be a major help in projects where past data are being used to estimate a current project.

The three primary methods that are commonly used in the cost estimation field are (1) rough estimation methods (often used for order-of-magnitude estimates), (2) statistical or econometric methods for cost prediction (generally used for preliminary estimates), (3) accounting cost estimating methods that involve a detailed component-by-component approach to estimating, and (4) hybrid methods that involve a mixture of estimating methods. Accounting cost methods are often used for providing definitive estimates of a project. The following sections will outline some of the characteristics and features of these methods.

23.3.2.1 Rough Estimation Methods

Rough estimation methods can take a number of forms but are often performed using previous mining costs. The data are often utilized without discrimination based upon the specific parameters of the mining process. In other words, data from a mining method are simply lumped together without attempting to relate them to the precise conditions of mining, the specifics of the equipment used, or the degree of management exercised. The primary estimation method used may be based on the historical record for a particular category of mining. One such method has been called the *unit cost method* by Gentry and O'Neil (1984)

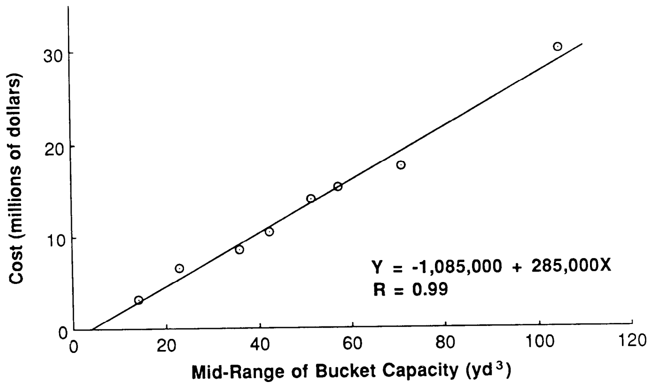

Fig. 23.3.1. Relationship between walking dragline mid-range bucket capacity and total cost (Schumacher, 1989, by permission). Conversion factor: 1 yd^3 = 0.7646 m^3.

because the costs are normally expressed in terms of dollars per annual ton of capacity of the operation or in other linear expressions based upon units of size or capacity. This method is often utilized simply because data necessary for a more complete analysis are not easy to obtain. In addition, it may be that the cost estimate does not require great accuracy. It should be noted, however, that rough estimation methods of this type will generally be used only for order-of-magnitude estimates. While rough estimation methods may be used for preliminary estimates, the accuracy may not be sufficient in all cases for preliminary estimates.

23.3.2.2 Statistical Cost Methods

A *statistical* or *econometric model* is any model that attempts to predict a dependent variable as a function of its independent variables (also often called its parameters). A statistical cost equation is one such econometric relationship that can be utilized in a cost estimation problem. A regression equation of this type will generally be of the form,

$$\text{Cost of an entity} = f(N \text{ parameters}) \tag{23.3.1}$$

in which cost of the entity is a function of N independent parameters of the entity, and N can be any number. To establish this type of a cost relationship, a variety of procedures and methods can be used with the emphasis on providing the equation from a statistical analysis of the data. For example, assume that the data plotted in Fig. 23.3.1 (Schumacher, 1989) represent the total costs of purchase and erection of walking draglines. If the data are plotted as shown, one might be justified in simply using the linear regression line as a cost estimation equation for the machines in a preliminary estimation. The regression equation can be used for any size of dragline with the understanding that some errors are inherent in the estimate. The correlation coefficient R will represent an important measure of the accuracy of the estimate because, when squared, it represents the percentage of the variation in the data explained by the regression. The unexplained variation will be the variation around the regression line and can be due to any number of variables such as manufacturing differences, design variations, or pricing idiosyncrasies.

A number of cost models have been published that may be quite useful to the engineer or accountant involved in cost estimation. Some of these are outlined in the upcoming paragraphs. The cost model that has been applied to the widest variety of cost relationships in both mining and mineral processing of metallic deposits is that outlined by the USBM (Anon., 1987a, 1987b). An additional publication (Stebbins, 1987) provides similar cost relationships for small placer mines. These cost estimating systems handbooks utilize a generalized approach based on the exponential regression function,

$$Y = a X^{b} \tag{23.3.2}$$

where X is the independent variable upon which the cost relationship is based, and a and b are the constants that are determined as a result of regression of X on Y. Some of the primary advantages of such a relationship are essentially that the equation can be applied to a wide variety of relationships, and that the regression line will appear as a straight line on log-log paper with the slope determining the parameter b. An additional advantage is that the cost models have been formatted into a Lotus 1-2-3 spreadsheet program for the convenience of users, and the spreadsheet program is available from the USBM.

The USBM states that their exponential cost regressions are based upon 5- to 7-point estimates of the cost that are derived using uniform cost data and that the estimates have an accuracy of ± 25%. A wide variety of cost relationships have been provided by the USBM in its publications including capital costs; costs of supplies, labor, and equipment operation for many underground and surface mining methods; exploration and development costs; transportation and mine plant costs; and capital, operating, and equipment costs associated with typical metallic mineral processing operations. One disadvantage of these publications is that no raw data or correlation information is presented along with the models. More will be said about these cost models later in this chapter.

Other cost models of the statistical type have been presented by O'Hara (1980, 1982, 1987). The models presented in these publications are often exponential like those of the USBM, but they tend not to be tied to one specific equation type. In addition, O'Hara also uses the econometric approach to choosing such variables as the manpower, equipment size, and other decision variables often considered at the same time as the cost estimate. Readers can assess the usefulness of these cost models by referring to Chapter 6.3 of this *Handbook* where examples of the cost equations have been provided.

In the area of coal and other sedimentary mineral beds, cost models have been constructed in the past by Toth (1981a, 1981b) and Toth and Annett (1981a, 1981b). These models cover the primary methods of mining coal by both underground and surface methods. The models tend to combine some of the statistical procedures with more detailed accounting cost models. While these models are now out of date due to changes in the technology of coal mining, they are of general interest because of their construction. The basic underground mining cost model utilized six of the major variables that affect the cost of underground coal mining: seam depth, seam thickness, seam grade, floor conditions, and methane liberation. The primary statistical equation used to initiate the model for a continuous miner cost estimation is as follows:

$$TPMS = (aH^2 + bH + c)\,(OEF)\,(AFT/390) \tag{23.3.3}$$

where $TPMS$ is output in tons per machine shift, H is seam thickness in feet, OEF is operator efficiency factor, AFT is available face time/shift in minutes, and the parameters a, b, and c are coefficients of a second-order polynomial obtained by regression. (Note: Most of the equations presented in this chapter are regression equations based upon English units. Because the equations do not apply to SI units, these units are not provided here.) This statistical relationship considers only the seam height as an independent variable, and so a separate regression was required for each combination of depth, grade, floor condition, roof condition, and methane emission considered. Data points for these regression relationships were generated using a computer simulation program. Similar relationships were generated for conventional and longwall mining as well. After the TPMS values are obtained, the overall cost prediction for the mining system is determined by calculating the capital requirements of the system, the labor and operating costs, and then the discounted cash flow of the proposed mining system.

Because prediction of the productivity of coal mine working sections is a major part of the battle in prediction of costs, a model that will accurately predict production or productivity is often as useful as a model that will estimate costs. Suboleski (1978) determined a multi-variable stepwise regression model for predicting productivity that extended the concept of regression models to include geologic, mining plan, equipment, geographic, and labor variables in the prediction process. The data for this model were obtained from questionnaires reporting the production from 326 operating coal mine sections using continuous miners. Depending on which variables were used and how the equations were constructed, the multiple-variable regressions were able to explain from 27 to 63% of the variance in the section production. Eight of the physical variables (seam height, roof quality, methane liberation, bottom quality, water, grades, depth of the seam, and hardness of the seam) were considered to have an effect on production. These variables are not under control of the engineer or mine manager and thus must be overcome through good planning and management of the mining operation.

Another statistical method that can be applied in estimation work is the method often called the cost ratio method (Gentry and O'Neil, 1984, pp. 120–126). In this procedure, a detailed capital cost estimate is prepared for certain essential categories of capital equipment necessary to equip the mine. Other costs (e.g., secondary equipment costs, development costs, costs of supplies, manpower costs, etc.) are then estimated from these capital costs by using cost ratios that relate other cost categories to the cost of the selected primary capital equipment. In most cases, this method is used primarily to estimate the capital costs of a project. However, the method can be applied to the overall operating costs if the labor and other operating costs can be reliably predicted as cost ratios of the primary capital equipment. A model by Ramani, Murray, and Manula (1977) performed this type of cost estimation for surface coal mining operations. Based on the models developed, the method was able to predict selling price of the coal within 15% for six varied surface mining operations.

In the statistical methods discussed in the preceding paragraphs, the cost estimations that result will all provide estimates in the preliminary estimate class if properly applied. However, each suffers from some shortcomings, and the cost estimator should be well aware of these so that no undue errors from the estimation procedure are introduced. If sufficient data are not available, or if the models are not appropriately applied, the accuracy of estimation may slip into the order of magnitude range.

23.3.2.3 Accounting Cost Models

An *accounting cost model* is a more detailed procedure for estimating costs based upon a component-by-component breakdown of the cost elements. The methods involve a complete accounting for all capital costs, labor costs, supply costs, power costs, etc., for each year of operation and discounted cost or cash flow analysis of the cost or income data. The cost estimator who uses this approach to estimation must have a thorough knowledge of the mining system and its component elements so that the cost computation is complete and accurately reflects the mining system. Under proper conditions of use, the accounting cost model can and does produce definitive estimates of the capital and operating costs of a mining operation.

Accounting cost models have been used and outlined in a number of publications of the USBM. Bureau personnel have published a series of analyses of coal mining costs that cover the room and pillar method using continuous miners (Katell, Hemingway, and Berkshire, 1975, 1976a), the longwall mining method (Duda and Hemingway, 1976a, 1976b), and strip mining (Katell, Hemingway, and Berkshire, 1976b). The approach outlined in these publications is to develop a detailed mining plan and then provide a set of calculations that provide a capital investment summary, labor requirements, depreciation schedule, power and water costs, estimated annual production cost, estimated development cost, estimated working capital and total capital investment, plus the discounted cash flow summary.

The primary advantage of such a costing method is the fact that it breaks the costs into elements and provides a cost estimating method and a cost control method simultaneously. The general method is easily computerized and can be readily understood by nearly everyone on the mine staff. The most crucial limitation in the use of this type of model in the cost estimation for a projected mine is the need for accurate labor requirements and productivity estimates to achieve a cost estimation. The accuracy of the method hinges primarily on these variables. While capital and supply costs are important in the determination of overall costs, they can normally be determined through standard cost reconnaissance methods. The accuracy of these estimates can probably be determined within 10%. However, due to the complexities of geologic and human variables, it is generally not possible to determine the productivity as accurately. Therefore, the accuracy of the productivity estimate is highly likely to determine the overall accuracy of the cost estimation method.

Accounting cost methods are often used to prepare definitive estimates for financing or budgeting purposes. It would therefore be important to keep estimation accuracy to ± 10%. This is very difficult to achieve before the mine is in operation. However, the same method of accounting for costs can be used after the mine is in operation with better results as then the planning engineers can more accurately predict how geologic and human variables will affect the productivity.

23.3.2.4 Hybrid Models

Many cost estimating procedures pursued during the exploration or prefeasibility stages of a mineral development project will require estimating accuracy somewhere between that provided by statistical models and that provided by detailed accounting models. When estimating in these situations, hybrid models that utilize both statistical and accounting procedures are often used. These procedures may use statistical models for estimating some of the component costs while accounting procedures may be used for others.

Using these procedures, the accuracy of the estimation will ordinarily be better than that for pure statistical methods due to the basic property of costs that are cumulative. This basic property states that the expected deviations in the estimated total cost of a project will be the square root of the sum of the squares of the deviations of the individual component costs. Using this procedure, the negative deviations in component costs have a tendency to negate positive deviations in other component costs. Thus the accuracy of hybrid models will have a tendency to increase if a more detailed approach is used and to decrease if a less detailed approach is used.

23.3.3 SOURCES OF COST INFORMATION

The minerals engineer working in mine planning and cost estimation does not have a large number of cost sources to help in the construction of cost estimates. There exist no comprehensive sources of cost data, and the information that is available often tends to be out of date by the time of publication. As a result, planning and cost engineers often try to maintain their own files on costs that they derive from various unpublished sources. However, several cost sources are worthy of note and these are outlined here.

The first source of information is the USBM cost-estimating system handbooks (Anon., 1987a and 1987b; Stebbins, 1987). These references contain many cost functions that cover capital and operating costs related to both mining and mineral processing of metal and nonmetal deposits. These handbooks are based entirely upon the exponential regression cost function (Eq. 23.3.2) and have been published with cost relationships that were in effect in 1984. As a result of the type of data used in the compilation of the cost functions and the need to update the data for current use, the cost relationships can be used only for rough or order-of-magnitude estimates. Nonetheless, the form of the regression relationship is useful, and the wide coverage of the USBM handbooks make them generally useful.

Additional cost information is found in the publications of O'Hara (1980, 1982, 1987) and in his work in Chapter 6.3 of this *Handbook*. O'Hara has developed a number of the important statistical cost relationships that exist in the minerals industry. To be used for cost estimation purposes, his cost functions also require updating. One of the advantages of his work is that it often includes relationships to accomplish the sizing of the equipment and the work force, an important consideration during the planning and cost estimation process for a mining operation.

An industry group that has published a considerable amount of cost information is the Northwest Mining Association. This group has published two volumes (Hoskins, 1982; Hoskins and Green, 1977) that contain a variety of papers on the cost elements of a mining operation. Many of these published presentations cover cost topics that have not been well covered elsewhere. Therefore, they may be of particular interest to minerals cost personnel for this reason.

An additional source of general information on cost estimation is the American Association of Cost Engineers, an international professional society headquartered in Morgantown, WV. This group has a monthly journal titled *Cost Engineering* and is dedicated to the dissemination of information on cost estimation, cost/schedule control, and project management. In addition, they maintain a certification program for cost engineers, conduct an annual meeting, and publish annual transactions. Advantages of membership in this association are that a large reference library is maintained and a computerized reference system that will provide information from the literature on a variety of cost engineering and related topics is available.

Another resource of interest to cost estimators is a subscription service on mining costs. *Mining Cost Service* (Schumacher, 1989) is a monthly publication of Western Mine Engineering of Spokane, WA, that provides cost information to subscribers on a variety of topics including electrical power, natural gas, transportation, labor, cost indexes, supplies, equipment, smelting, taxes, published cost models, and special articles. The data are gathered through a series of ongoing surveys of a network of mining companies, smelters, equipment and supply vendors, utilities, transportation companies, and government agencies. The cost service updates a category of costs each month, and every topic is updated each year. Therefore, the data are relatively recent in each category. An additional advantage of the service is that many of the data are provided on a regional basis (e.g., electric power and natural gas costs) so that the data will be relatively accurate for products and services that are location-dependent. *Mining Cost Service* also provides a selected list of vendors and other experts who have agreed to discuss equipment costs with subscribers whether or not a sale is imminent.

Finally, it should be mentioned that accurate cost estimates can often be achieved only through direct communication with the vendors who provide equipment, supplies, and services to the mining industry. In most cases, this can be performed as the projects move to their preconstruction stages and more definitive cost estimates are necessary. Equipment costs vary considerably over time as they frequently depend on market conditions and the size of the equipment purchase. Thus vendor quotations should be sought when detailed planning is complete. Accounting cost models require that definitive estimates be obtained from suppliers.

23.3.4 COST UPDATING METHODS

One of the problems that must constantly be dealt with in cost estimating is the updating of basic cost data to present cost values. This problem arises because costs for equipment and operating costs change over time, primarily as a function of inflation. The engineer or others involved in cost estimation will often find the need for updating old prices or costs taken from various sources and estimating current prices or costs from them. To perform this function, the practicing cost engineer must have cost indexes available that reflect the inflation in costs or prices over time. A cost index provides a means of comparing the costs or prices of a specific category of goods or services at specific points in time and thus will allow updating by relating costs to the index. The general equation to update a cost value is

$$\text{Current Cost} = \text{Past Cost} \times (\text{Current Cost Index/Past Cost Index}) \qquad (23.3.4)$$

In utilizing this relationship, it is necessary to know the date when the past cost was effective so that the past cost and the past cost index are matched in time. An application of this principle will be illustrated in examples later.

A variety of cost indexes are available to the minerals engineer for use in cost estimating work. Most of the indexes are not specific to the mining industry but contain index values that are applicable to the plants, equipment, and services used in the mining industry. An outline of several of the more important price indexes is provided here.

The *Producer Price Indexes* is a monthly publication of the US Department of Labor and covers a rather comprehensive array of products used in industry. Some of the prices are provided on an industry or commodity basis while others are provided on a stage-of-production basis. The index values are updated monthly based upon an extensive set of price quotations collected from suppliers of the products. As such, they probably contain more basic background data and are more useful than any other set of indexes. Table 23.3.1 contains a number of the index values from 1967 to the present for commodities that have

Table 23.3.1. Mining and Related Cost Indexes

Year	Mine Labor	Construction Labor	Construction Machinery & Equipment	Mine Machinery & Equipment	Mine Machinery Parts	Mine Drill Bits	Iron & Steel	Lumber	Refined Petroleum Products	Explosives	Truck Tires	Rubber & Plastic Products	Electric Power	Industrial Chemicals
1965	2.92	3.70	27.2	26.2			28.9	28.9	12.3	33.3	36.4	39.7	21.2	27.7
1966	3.05	3.89	28.1	27.0			29.1	30.4	12.8	32.8	37.3	40.5	21.1	27.9
1967	3.19	4.11	29.1	27.8			29.5	31.1	13.1	33.5	36.8	41.4	21.1	28.4
1968	3.35	4.41	30.7	28.7			30.1	37.5	12.9	34.2	37.8	42.8	21.3	28.6
1969	3.60	4.79	32.1	29.6			31.6	41.8	13.1	35.0	36.2	43.6	21.6	28.4
1970	3.85	5.24	33.7	30.9			34.0	35.2	13.3	35.7	38.8	44.9	22.5	28.6
1971	4.06	5.69	35.4	31.9			35.9	44.0	14.1	37.9	40.6	45.2	24.8	28.9
1972	4.44	6.06	36.6	32.5			37.9	52.1	14.3	38.5	41.0	45.3	26.2	28.7
1973	4.75	6.41	38.0	33.6	29.2		40.2	66.6	16.9	40.2	42.6	46.6	28.0	29.3
1974	5.23	6.81	44.3	39.9	35.0		52.7	65.7	29.3	50.2	52.1	56.4	36.4	43.0
1975	5.95	7.31	53.8	51.2	47.2		59.3	62.4	33.8	59.5	57.2	62.2	44.3	58.7
1976	6.46	7.71	57.8	58.9	60.0		63.7	77.1	36.3	62.6	63.6	66.0	47.9	62.2
1977	6.94	8.10	62.1	63.5	64.2		68.0	92.5	40.5	64.9	66.9	69.4	54.3	63.5
1978	7.67	8.66	67.7	69.3	70.2		74.8	107.6	42.2	69.8	70.7	72.4	59.1	64.0
1979	8.49	9.27	74.5	75.8	77.4		83.6	118.1	58.4	75.5	80.9	80.5	64.5	74.9
1980	9.17	9.94	84.2	85.2	88.1		90.0	107.3	88.6	84.0	92.1	90.1	77.8	91.9
1981	10.05	10.80	93.3	93.2	94.3		98.5	106.6	105.9	96.7	99.5	96.4	89.2	103.1
1982	10.78	11.62	100.0	100.0	100.0		100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
1983	11.30	11.91	102.3	102.3	102.4		101.3	115.0	89.9	101.1	95.7	100.8	103.1	97.3
1984	11.63	12.12	103.8	104.1	104.5		105.3	110.0	87.4	103.6	93.4	102.3	108.4	96.8
1985	11.98	12.32	105.4	105.4	104.8		104.8	106.6	83.2	105.0	90.5	101.9	111.6	96.0
1986	12.46	12.48	106.7	106.1	105.4		101.1	107.2	53.2	103.6	88.0	101.9	112.6	91.5
1987	12.54	12.71	108.9	106.5	104.9	100.0*	104.6	112.8	56.8	107.3	87.7	103.0	110.6	95.5
1988	12.80	13.08	111.8	110.2	108.0	103.3	115.7	118.9	53.9	109.0	92.5	109.3	111.2	106.8
1989	13.26	13.54	117.2	116.3	113.0	106.6	119.1	126.7	61.2	117.7	96.3	112.6	114.8	114.8
1990	13.69	13.78	121.6	121.0	116.2	113.1	117.2	129.7	74.8	125.6	93.8	113.6	117.6	113.2
1991	14.21	14.01	125.2	125.2	117.2	120.6	114.1	132.1	67.2	132.1	95.7	115.1	124.3	111.8

* Value as of December 1987.

Notes: (1) Labor costs are $/hr values and were obtained from Anon., *Monthly Labor Review,* US Department of Labor, various issues.
(2) Commodity costs are relative values based upon 1982 = 100 (except for mine drillbits which are based upon December 1987 = 100) and were obtained from Anon., *Producer Price Indexes,* US Department of Labor, various issues.
(3) Some table values were updated using US Department of Labor computer outputs.

some relationship to mining costs. Table 23.3.1 also contains some labor indexes that are published monthly in the publication of the US Department of Labor entitled *Monthly Labor Review.* To use the price and labor indexes, it is generally necessary to simply reflect the price increases for the correct category of commodity.

Example 23.3.1. A given size and type of front-end loader was known to have a price of $330,000 in 1985. What is its estimated cost in 1989?

Solution. The 1989 cost for the loader can be estimated using the price index for construction machinery and equipment index or the mining machinery index from Table 23.3.1 However, loader prices may more accurately be reflected by the construction equipment index, and so the estimated 1989 cost is

$$\begin{aligned} \text{1989 Cost} &= \text{1985 Cost} \times (\text{1989 Index}/\text{1985 Index}) \\ &= \$330{,}000 \times (117.2/105.4) = \$367{,}000 \end{aligned}$$

In updating the cost of a loader, the updating procedure is relatively straightforward. In other estimations, a component-by-component procedure must be used to provide a current cost. An example of this procedure follows.

Example 23.3.2. A zinc mining company has a 1984 cost estimate for mining by the vertical crater retreat (VCR) method (Anon., 1987a, pp. 558–560) that indicates the following operating costs for a 2000-ton/day operation.

Labor cost	$1.00/ton
Supply cost	$1.31/ton
Equipment cost	$0.29/ton
Total operating cost	$2.60/ton

How should these costs be updated to 1989 values if the component-by-component procedure is to be used?

Solution. The basic procedure calls for updating each cost category in the estimation procedure using the most applicable cost indexes available. In studying the cost categories in Table 23.3.1, it is obvious that labor costs can best be updated by using the mine labor cost index. The equipment costs are updated using the mine machinery parts index as repair parts make up 48% of the equipment cost category. The cost of supplies can be related to the cost index for bits and steel as this constitutes 70% of the costs involved. However, the index is not continuous back to 1984 and the machinery parts index will be used instead. The updating proceeds as follows:

1989 Labor Cost = 1984 Labor Cost × (1989 Labor Index/1984 Labor Index)
= $1.00/ton × (13.26/11.63) = $1.14/ton

1989 Supplies Cost = 1984 Supplies Cost × (1989 Mine Machinery Parts Index/1984 Mine Machinery Parts Index)
= $1.31/ton × (113.0/104.5) = $1.42/ton

1989 Equipment Cost = 1984 Equipment Cost × (1989 Machinery Parts/1984 Mine Machinery Parts Index)
= $0.29/ton × (113.0/104.5) = $0.31/ton

1989 Total Operating Costs = 1989 Labor Cost + 1989 Supplies Cost + 1989 Equipment Cost = $2.87/ton

While *Producer Price Indexes* provides a measure of the fluctuation in the prices of individual products, it it often helpful to have an index that reflects the cost of erecting a completed processing facility. One index that provides a measure of this type of cost is the plant construction cost index that is published each month in *Chemical Engineering.* This is a composite index that reflects the prices of products used in constructing a plant, building materials, construction labor, plus staff labor costs. The index is designed to reflect the cost of construction of a typical chemical engineering plant and thus may not be equally accurate when projecting the cost of providing a minerals processing plant or a smelter. As a result, other indexes may be preferable for this purpose.

The Marshall and Swift Index is also published monthly in *Chemical Engineering.* The monthly values from this list provide indexes for the installed price of various categories of equipment. Like the *Chemical Engineering* index, it is compiled as a composite of numerous costs to reflect the equipment, materials, and labor required to install equipment in a given processing industry. One of the reasons that it is of interest to mineral engineers is that it provides a specific cost index for the mining and milling industry and thus provides useful information when projecting mine plant, processing, and smelting facilities.

Several other indexes are available as well. The *Engineering News-Record* index and the Nelson refinery construction cost index (published in *Oil and Gas Journal)* are described by Gentry and O'Neil (1984). Other indexes are also available to reflect various costs, particularly construction costs. The basic problem with most of these indexes is that they may or may not parallel the cost of minerals projects. The downturn in the minerals industry that occurred in the late 1970s and early 1980s was evidence of this as costs of mining equipment and construction did not rise as fast as that for other industries due to the competition that developed among vendors when mineral development projects became scarce. Thus an index should be chosen carefully so that it correctly reflects the costs of the minerals industry. Otherwise, alternative methods of updating should be developed.

23.3.5 MINING METHODS COST ESTIMATION

To provide working data on current operations, the authors conducted two surveys of mining companies. The first survey was oriented towards coal mining methods; the second was oriented towards mining methods applied to the noncoal minerals. The response to the questionnaires was good in some categories and poor in others, but a great deal of useful data was derived from the effort. A summary of each of the data sets is provided in the following, along with cost models and statistical results derived from the data. (Readers may also wish to refer to the topics on cost estimation in Chapter 6.3 and to the treatment on cost updating provided in 23.3.4.)

23.3.5.1 Cost Estimation in Coal Mining

The precise estimation of costs in coal mines can be a difficult task. While estimation may be readily accomplished to obtain a cost that is within $2 to $3/ton of the correct value, this range of error can affect the decision to proceed or not to proceed with a mining project. Thus final cost estimation should be done in great detail before proceeding with a project. The method presented here is one to obtain preliminary costs and should be considered as no more that ± 20% in accuracy, roughly the equivalent of a preliminary estimate. This segment offers some guidelines for estimating the operating costs of surface and underground coal mines.

Cost estimation in coal mining differs considerably, of course, for underground and surface operations. Within these two broad categories, costs tend to be largely a function of productivity. Thus the key in obtaining accurate estimates of operating costs is the accurate estimation of productivity. This is preferably in terms of tons per employee-hour, but initially must begin as tons per machine shift (or its surface mine equivalent, tons per "spread" per shift).

BACKGROUND ON OPERATING COST. Operating costs are usually categorized into fixed and variable costs, with the latter characterized as remaining constant per unit of production (within certain limits) and the former characterized as remaining constant per time period, regardless of the production (again, within limits). For both surface and underground mines, the list of "pure" variable costs is relatively short. These may be categorized as follows:

1. Royalty and wheelage.
2. Black lung excise tax.
3. Reclamation excise tax.
4. Severance tax.
5. Sales commission.
6. Most operating supplies.
7. Contractors paid by the ton.
8. Energy component of the power bill.
9. Maintenance parts and supplies.

Listing maintenance parts and supplies as a variable cost may be somewhat questionable as it may be argued that maintenance costs will fall as productivity rises. The counter argument is that machines only incur the need for maintenance as a result of usage. The conservative approach is to consider this category as a variable cost; it is probably more accurate to consider it as partially variable.

At the time of this writing (1989), most coal mines incur total costs of $20 to $35/ton, with total variable costs in the range of $8 to $13/ton. Therefore, given the large value of fixed costs per ton, it seems reasonable to expect that the total cost per ton would track productivity.

Productivity in many cases functions as an indicator of other mining costs as well, including variations in the total variable cost among mines. For example, in underground mines, low productivity may often be caused by poor roof conditions, necessitating additional cost in roof support. While roof support supplies are variable costs within both high- and low-productivity mines, the roof support cost will tend to be higher (per ton) in the low-productivity mine.

It is misleading, however, to consider that all differences in costs between mines can be explained by differences in productivity. Some of the other factors contributing to differences in costs among mines with identical productivities are outlined in the following.

Union/Nonunion Status—Items such as pension funding, benefit levels, pay rates, and retiree costs will cause differences in costs even among mines with the same productivity levels.

Severance Tax—Severance tax varies from 0 to levels approaching a 20% effective rate, depending upon state law.

Workmen's Compensation—Compensation rates vary considerably between states, and may vary among mines within the same state, depending upon the accident rate of the mine.

Plant Recoveries—Most preparation plant costs, most supply costs, and items such as trucking costs are a function of raw rather than clean production. Thus a difference in recovery will cause cost differences in addition to those caused by clean coal productivities.

Management Practices—Items such as choice of supplies, supply wastage, amount of overtime pay, etc., can affect costs independently of productivity.

Use of Contractors—Coal produced or trucked by contractors will not increase reported company employee-hours worked (i.e., reported productivity), but will add to the cost. Many companies also use maintenance and other direct-labor contractors that add to the cost but not the employee-hours worked by company personnel.

Other factors, such as the age of the mine, capital replacement policy, etc., also affect total cost. Thus it is important to keep in mind the location and operating practices of the mine for which the cost is to be estimated. The cost estimation procedures discussed later in this segment can then be adjusted for these specific features of the mine being analyzed.

COST SURVEY. To verify the results of the cost models presented within this section, a survey of costs and productivities at operating coal mines was conducted in midyear 1989. Results were obtained from 86 underground mines in eight states, and from 39 surface mines in seven states. Both union and nonunion mines responded to the survey. The survey asked for the following costs to be reported on a per-ton basis:

1. Mine cost.
2. Plant cost.
3. General and administrative costs.
4. All other costs.
5. Royalties.
6. Depreciation.
7. Total cost.

Data was also gathered on the specific types of costs included within each of these categories. In general, there was such a large variation between companies that only total cost, royalty, and depreciation were useful in generating cost models. Plant costs also showed a reasonable degree of consistency.

Information about a number of other items that were judged to be potentially significant in determining costs for each mine was also requested. These items, some of which are shown in summary form in Table 23.3.2, were then used to construct regression models of mine cost and productivity. A discussion of these models follows.

In general, the productivities reported by the mines responding to the survey tended to be higher than the average productivities reported for 1988 (the latest year available) by the Energy Information Administration (EIA) for most of the states (Anon., 1989). However, this does not detract from the cost models presented since cost is presented as a function of productivity and a wide range of productivities was represented in the data. The models that were developed to predict productivities are likely to be biased toward the high side, however. Table 23.3.3 shows the range of values for each important variable for underground mines. Fig. 23.3.2 shows a plot of costs as a function of productivity for continuous miners. The general trend in the data indicates the influence of productivity on cost. However, it also shows that while cost decreases with an increase in productivity, there are significant variations caused by geologic, equipment, and human variables.

Average productivity of the underground mines responding was 3.00 tons per employee-hour (tpeh), compared to the national average of 2.38 tpeh reported by the EIA for 1988. The average mining cost for the mines responding to the survey was $21.66/ton, but the average was clearly different for eastern and western mines, with the western mines being those in the Far West and the Illinois Basin. Table 23.3.4 shows the difference in cost for these regions and for certain states. The east/west difference is taken into account in the cost models by using a dummy variable to differentiate eastern and western mines. Note that the western mines also have higher productivities, and that this difference is greater than that reported by the EIA for these states.

For the surface mines, the reported average productivity was 5.58 tpeh versus a national average of 5.43 tpeh. The average total cost of mining was $23.13/ton, but this varied widely by region and has little meaning by itself.

In underground mines, nonunion mines were found to outperform union mines, with productivities of 3.72 and 2.65 tpeh, respectively. In addition, mines that paid incentives were also found to outproduce those that did not, with respective figures of 3.89 and 2.55 tpeh. Most union mines do not pay incentives, and most nonunion mines do; thus it is difficult to separate these effects. However, as shown in Table 23.3.2, there is some indication that incentives may be more significant in determining productivity than unionization.

UNDERGROUND COAL-MINE COST MODEL. This segment discusses two methods for estimating costs in underground coal mines. The first method is a more general procedure, while the second is based upon the results of the regression models developed from the cost survey described above. Both models attempt to estimate the total cash cost of production less royalty. Both assume that this cost is primarily a function of productivity, but that it will also be dependent upon other factors. In the case of the regression model, these factors are presence or absence of an incentive system, the preparation plant yield, and the average depth of the mine.

Notice that, in the case of the regression model, the yield is determined by engineering data, the average depth of the mine can also be readily ascertained, and the use of an incentive system is the choice of management. Thus estimating the cost of a mine continues to revolve about the determination of productivity. This is now discussed in further detail, both in general and as a separate regression model obtained from survey results.

Estimating Productivity—Productivity is the key to cost estimation and must be estimated as accurately as possible. One method is to estimate productivity from a regression model that has also been developed from the cost survey results; however, it should be kept in mind that the productivity of the average mine in the survey was greater than that reported by the EIA for the mining industry. Use of the model is likely to predict a productivity somewhat higher than that achieved in practice. Thus an alternative procedure is discussed. In this method, the following steps are used:

1. Estimate average tons per unit shift for the mine.
2. Estimate the number of employees to be used and the number of hours to be worked per year or per day.
3. Estimate the number of unit shifts to be worked per day or year, allowing for unscheduled downtime.
4. Calculate the annual production using the above estimates.
5. Calculate the tons per employee-year or tons per employee-hour for the operation. Once step 5 has been accomplished (or the tons per employee-hour has been determined by other means), then the regression model for mine cost may be used to determine total cost as follows:
6. Apply the productivity and the other required input to the regression model to predict the cash cost per ton (less royalty) of the operation.
7. Add the royalty and depreciation to the above number to obtain the total cost.

Step 1. Calculate the average tons per unit shift. Probably the best method of estimating tons per unit shift and consequent tons per day is to look at similar mines in the area or similar mines operated by the company because earlier studies have shown that management practices and regional factors may be as important as physical conditions in determining productivity (Suboleski and Manula, 1979). The figures from these similar mines should then be adjusted to account for the differences in

Table 23.3.2. Summary Data from the Underground Coal Cost Survey

Variable	All Mines	Longwall	Room and Pillar	
			Conventional	Continuous
No. of mines	73	15	6	69
States represented	8	7	2	8
Mean seam thickness (ft)	5.30	6.84	5.52	5.01
Mean seam depth (ft)	534	723	572	483
Age of mine (yr)	15	29	11	12
Mean mine size (million tons/yr)	0.8	1.8	1.5	0.6
Haulage distances (mi):				
Face to portal	2.5	4.7	4.7	1.9
Portal to plant	4.7	1.1	2.8	5.4
Plant to loadout	0.9	1.1	3.6	0.8
Total haulage distance	8.1	6.9	11.1	8.1
Plant yield (%)	68	70	71	67
Unit shifts/unit day:				
Longwall		2.62		
Conventional/continuous		1.69	1.87	1.81
Mean no. of employees		350	281	126
Mean hourly employees—underground		266	233	93
Mean salaried employees—underground		52	31	17
Total employees/unit shift		38.2	54.8	22.6
Underground employees/unit/day		34.8	51.4	20.1
Clean tons/employee-hour (CTEH)—all mines	3.00	2.62	2.82	3.06
CTEH—mines using incentives	3.89	3.60	3.50	4.04
CTEH—mines not using incentives	2.55	2.44	2.15	2.53
CTEH—union mines	2.65	2.44	2.15	2.71
CTEH—nonunion mines	3.72	3.60	3.50	3.68
% of mines using incentives	35%	20%	50%	38%
% of mines unionized	65%	73%	50%	65%
Clean tons/employee–year	4966	5249	5205	4511
Tons/unit shift		2298	759	505
Apparent employee–hours/year [(tons/ employee–yr) ÷ (tons/employee–hr)]	1655	1956	1829	1474
Mining cost/ton	21.66	20.64	20.01	21.55
Total cost/ton	28.78	29.61	25.19	28.70
Total cash cost—royalty	25.97	27.19	23.18	25.80

Conversion factors: 1 ft = 0.3048 m, 1 mi = 1.609 km, 1 ton = 0.9072 t.
Note: Data from 86 mines were received. However, only 73 of the data sets were sufficiently complete for this analysis. Also, some mines reported data for more than one method.

conditions expected between the proposed mine and the existing mines. For a possible methodology, see Suboleski and Manula (1979) and Manula and Suboleski (1979).

For continuous miner operations where comparable operations are not available, the following equation may be used. This equation is derived from regression analysis on the clean tons per unit shift (tpus) for continuous miners reported by 67 of the 86 mines participating in the cost survey. The regression has an unadjusted R^2 value of 0.57 (i.e., statistically the equation can account for 57% of the observed variation in tpus). Recall that the average productivity of the mines participating in the survey was higher than the average reported by the EIA, and thus the tons per unit shift predicted may well be higher than average as well, though this cannot be verified.

$$\begin{aligned} \mathrm{TPUS_c} &= 5.83(\mathrm{THICK}) + 267(\mathrm{INCEN}) \\ &- 49(\mathrm{UNION}) - 2.34(\mathrm{YIELD}) \\ &- 104(\mathrm{EAST}) - 67.2\,[\ln(\mathrm{DEPTH})] - 94 \end{aligned} \quad (23.3.5)$$

where $\mathrm{TPUS_c}$ is tons per unit shift for continuous miners, THICK is seam thickness in in., INCEN is 1 if an incentive plan is in place or 0 otherwise, UNION is 1 if the mine is unionized or 0 otherwise, YIELD is prep plant yield in % (use YIELD = 100.0 if no plant), EAST is 1 if the mine is in Appalachia or 0 if in the Illinois Basin or the Far West, DEPTH is depth in ft, and ln is the natural logarithm.

The mean value of productivity in the 67 mines with complete data used to develop the above equation was 503 clean tpus. The entire data set (for the 84 mines reporting sufficient data) showed productivity to be slightly higher in non-longwall mines as follows:

	No. of Mines	Mean Tons Per Unit Shift for Continuous Miners
All Mines	84	505
Longwall	15	408
Non-longwall	69	537

Example 23.3.3. Compare the expected productivity (for 1989) in tons per unit shift of an 8-ft nonunion mine with 100% yield (no cleaning) in the western states with a 6-ft union mine in Appalachia with no incentive plan and a 70% yield if both average 700 ft deep.

Table 23.3.3. Statistical Summary of Selected Survey Variables for Underground Coal Mines (Total Number of Mines = 86)

Variable	Mean	Standard Deviation	Range
Seam thickness (in.)	61.5	20.2	30–120
Depth (ft)	534	384	120–2000
Haulage distance (Portal to Plant)	2.46	2.14	0–12
Yield (%)	67.6	17.0	40–100
Longwall tons/shift (N = 16)	2298	1207	320–4500
Continuous miner tons/shift (N = 80)	505	306	160–1500
Conventional tons/shift (N = 6)	759	350	440–1325
Tons/employee–hour (all mines)	3.00	1.37	1.1–8.7
Annual production (million tons)	0.85	0.79	0.063–3.40
Total employees	163	153	10–720
Salaried employees (underground)	24	29	1–212
Hourly employees (underground)	118	109	8–484
Preparation cost ($/ton)	3.71	1.64	0.34–8.42
Total cost ($/ton)	28.79	6.52	15.06–44.31
Depreciation ($/ton)	1.76	1.13	0.07–5.54
Royalty ($/ton)	1.46	1.01	0.02–4.45
Cash cost* ($/ton)	25.57	6.39	12.40–42.39

*Cash cost = total cost − depreciation − royalty
Conversion factors: 1 ft = 0.3048 m, 1 in. = 25.4 mm, 1 ton = 0.9072 t.

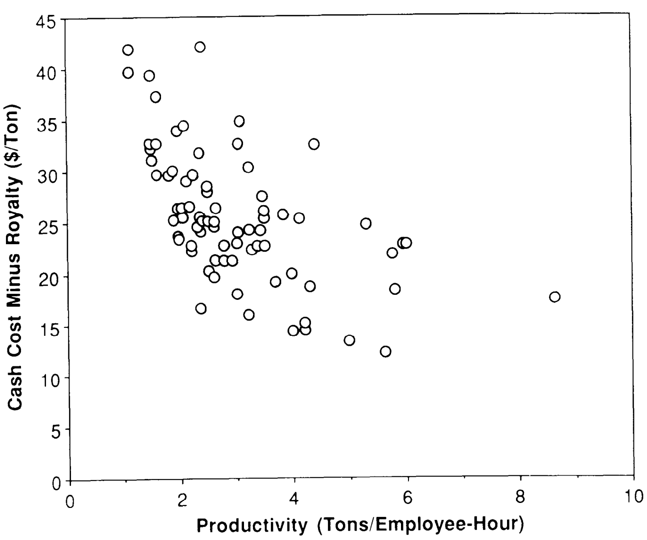

Fig. 23.3.2. Plot of cost survey results showing cash costs minus royalty vs. productivity for continuous miners. Conversion factor: 1 ton = 0.9072 t.

Solution.

For the 8-ft seam:

$$
\begin{aligned}
TPUS_c &= -94 + (5.83)(96) + (267)(1) - (49)(0) \\
&= -(2.34)(100) - (104)(0) + (67.2)(\ln 700) \\
&= -94 + 560 + 267 - 0 - 234 - 0 + 440 \\
&= 939 \text{ tpus}
\end{aligned}
$$

For the 6-ft seam:

$$
\begin{aligned}
TPUS_c &= -94 + (5.83)(72) + (267)(0) - (49)(1) \\
&= -(2.34)(70) - (104)(1) + (67.2)(\ln 700) \\
&= -94 + 420 + 0 - 49 - 164 + 440 \\
&= 553 \text{ tpus}
\end{aligned}
$$

Table 23.3.4. Differences in Underground Coal Mining and Productivity by State and Region

State	No. of Mines	Mean Cost ($/ton)	Mean Productivity (tons/employee–hour)
Pennsylvania	18	29.42	1.88
West Virginia	27	25.12	3.41
Kentucky	18	23.59	3.62
Virginia	10	27.96	2.69
Illinois	4	18.87	2.31
Colorado/Utah*	6	16.26	4.17
Region	**No. of Mines**	**Mean Cost ($/ton)**	**Mean Productivity (tons/employee–hour)**
East (PA, WV, KY, VA)	73	26.19	2.93
West (CO, UT, IL)	10	17.30	3.63

* Combined to provide a larger sample size.
Conversion factor: 1 ton = 0.9072 t.

Data were also gathered on longwall productivity. Here the mean value for the 13 mines with complete data was 2037 tpus; however, the values ranged from a minimum of 500 to a maximum of 4500. The mean value for shifts operated per day per unit was 2.62, indicating that most mines schedule three production shifts for their longwalls. While regression models were developed, it is believed that the wide range in values seen in longwall mining today is largely due to the rapidly changing technology in longwall mining; thus such a model may be misleading and is not presented here.

Step 2. Estimate number of employees and hours. One method that may be used to estimate the number of underground employees at newer mines is to calculate the number of face employees and apply a 1:1 ratio of outby to face employees. To this must be added the number of outside employees.

A second method is to use a "rule-of-thumb" to estimate total employees based on the number of unit shifts worked. Suggested employee numbers for several mine sizes are shown for mines using continuous miners:

One-section mine: 15 to 20 employees/unit shift (usually no plant)

Multi-section mine, less than 10 years old: 20 to 30 employees/unit shift (with plant)

Multi-section, 10 or more years old: 25 to 35 employees/unit shift (with plant)

The average number of employees per unit shift, based on the survey results, is shown in Table 23.3.2. It shows 35 underground workers and 38 total for longwall mines and 20 and 23 for all-continuous mines. For either type of mine, about 90% of the workers are underground, and 5 to 5.5 hourly workers are employed for each salaried employee.

A prediction equation for employees as a function of seam thickness, incentive plan and annual production was also developed using regression analysis, as follows:

$$
\begin{aligned}
TOTE_u &= 146.7(\ln(TPY)) - 72(INCEN) \\
&\quad - 1.7(THICK) - 1638 \qquad (23.3.6)
\end{aligned}
$$

where $TOTE_u$ is total number of employees for underground mines, TPY is annual production in tons per year, INCEN is 1 if an incentive plan is in place or 0 otherwise, and THICK is seam thickness in in. Note that neither the age or depth of the mine nor the total unit shifts per day proved to be significant variables for predicting total employees. This equation was developed for both longwall and non-longwall mines using 72 samples and has an R^2 value of 74%. The mean number of employees for the data set is 180.

Example 23.3.4. Estimate the total employees in two l-million-tpy mines, one in a 6-ft seam with no incentive system and the other in a 5-ft seam with an incentive plan.

Solution .

For the mine in the 6-ft seam and no incentive system:

$$\begin{aligned} TOTE_u &= (146.7)(1n\ 10^6) - (72)(0) - (1.7)(72) - 1638 \\ &= 267 \text{ employees} \end{aligned}$$

For the mine in the 5-ft seam with an incentive system:

$$\begin{aligned} TOTE, &= (146.7)(1n\ 10^6) - (72)(l) - (1.7)(60) - 1638 \\ &= 215 \text{ employees} \end{aligned}$$

Note that, at 220 days/year and 500 tpus, we would expect a 1-million-tpy mine to work 9 to 10 unit shifts/day. Thus the above figures are equivalent to 22 to 24 and 27 to 30 employees/unit shift.

Step 3. Calculate the tons per employee-hour. Calculation of tons per employee-hour may be accomplished in several ways. First, it may be estimated by adjusting the employee figure by a suitable absenteeism and overtime figure to arrive at average hours worked per day. This value is then divided into the average daily production. Both scheduled and unscheduled absenteeism must be accounted for, and both daily and extra-day (weekends, holidays) overtime must be counted.

A second, perhaps more accurate, method is to calculate tons mined per year and hours worked per year. This method allows for days in which the production unit is scheduled but does not work. This can be especially significant for longwall where two moves of one to two weeks or more may be encountered in a year. It is also significant, though to a lesser degree, in many continuous miner operations.

For 1988, the Energy Information Administration (EIA) has estimated that the average employee-hours worked per year was 1891. From the current survey, the employee-hours worked per year was estimated by dividing (tons/employee-year) by (tons/employee-hours). The calculated average was 1655 hours, but results varied widely by mining method, presence or absence of an incentive plan, etc.

Given the scheduled holidays, vacation, personal days, etc., the expected hours worked per year without overtime is 1688, as shown in Table 23.3.5, for a mine covered by the United Mine Workers of America (UMWA) agreement. Using this number and the EIA value for employee-hours worked per year, an expected overtime pay of 18% of straight-time pay is estimated. In practice, overtime is usually 10 to 15% of the straight-time pay.

The daily production must also be adjusted to full-year production by deleting scheduled and unscheduled idle days, and recalling that all sections are not in production on all scheduled days. From the survey, the reported tons per unit shift and reported tons per year were used to calculate the theoretical days worked per year. While this number varied widely, 220 days appeared to be the most frequent value. The exception to this rule was in non-union mines that paid incentives where an approximate average of 227 days was calculated. Note that these values are calculated and fluctuated widely.

A regression model was also developed from the results of the survey to estimate TPEH. The model has an R^2 value of only 46%.

Table 23.3.5. Expected Days to be Worked and Paid at a UMWA Coal Mine

Available days = 52 × 5 = 260			
Less: Holidays		11	
Regular vacation		10	
Floating vacation		4	
Graduated vacation		8	(Average?)
Sick and personal days		5	
Training		2	
Jury and bereavement		2	(Average?)
Idle days reporting, military leave		3	(Average?)
		45	
Days scheduled to be worked = 260 − 43	=	217	days
Unscheduled absenteeism @ 3.5%	=	8	
Days expected	=	211	
Hours expected = 211 × 8	=	1688	
Actual average hours worked (EIA, 1989)	=	1891	
Expected % overtime — 12			
Expected % overtime pay = 12 × 1.5	=	18%	
Expected overtime days = (1891 − 1688) ÷ 8	=	25	
Or, to summarize:			
Days paid per year			
Work days		211	
Scheduled days off		43	
Overtime days × 1.5 (25 × 1.5)		38	
Other*		2	
		294	days paid

* Paid 12 days for 10-day regular vacation period.

$$\begin{aligned} TPEH_u &= 5.01 + 0.029(THICK) + 1.13(INCEN) \\ &\quad - 0.016(YIELD) \\ &= -0.213[\ln(TPY)] - 0.177[\ln(AGE)] \qquad (23.3.7) \end{aligned}$$

where $TPEH_u$ is tons per employee-hour for underground mines, AGE is mine age in years, and the other variables are the same as before.

Step 4. Calculate cash costs of operations. After labor productivity has been established, the remainder of the costs may be estimated. To simplify this, the costs are categorized as follows:

1. Total labor.
2. Other mine cash costs.
   a. Operating and maintenance supplies.
   b. Power.
   c. Miscellaneous.
3. Other preparation plant cash costs.
4. General and administrative costs, miscellaneous other cash costs.
5. Royalties, taxes, and other costs based on the sales price.
   a. Royalties.
   b. Severance tax.
   c. Reclamation excise tax.
   d. Black lung excise tax.
   e. Sales commissions.
   f. Wheelage.

Total labor costs may be obtained by dividing the average cost per employee-year (approximately \$57,000 to \$60,000/year in 1990; see Table 23.3.6) by the tons per employee-year or, alternatively, the average cost per hour *worked* by the tons per employee-hour. At \$59,000 and 1890 hours, this is \$31.21/hour in 1990. Thus, it can be estimated that, at 2.38 tpeh, the average mine would have a total labor and benefits cost of about \$13.00/ton.

Other mine costs are simply operating and maintenance supplies, power, and miscellaneous expenses. In 1990, these likely

Table 23.3.6. Estimate of Cost per Employee–Year in Underground Coal Mines (1990)

Paid to Worker	
1. Average rate/year—$129.64/day × 294 days =	$38,114
2. Clothing allowance	$ 180
	$38,294
Payable on Wages	
1. Unemployment tax—3.5% × $8,000	$ 280
2. Social security—7.8%	$ 2,987
3. Workmens comp.—traumatic and black lung (estimated at 20% of wages)	$ 7,659
Subtotal	$10,926
Other	
1. Pension & benefit trusts $2.64/hr × 1891 hrs	$ 4,992
2. Training fund—$0.05/hr	$ 95
3. Medical & dental, life & disability insurance	$ 5,000
	$10,087
Labor cost for active employees	$59,307
Other potential labor costs:	
1. Incentive pay and bonuses	
2. Retiree medical	
3. Self-insured black lung	
4. Black lung excise tax	
5. Life and disability insurance for retirees	

ranged from a low of $4.50 to $5.00/ton to a high of $8.00 to $8.50/ton. Whether the mine falls within this range is largely determined by productivity, but is also affected by seam thickness and geographical location (with costs usually higher in the East). Longwall mines often have supply costs $2.00/ton lower than all-continuous operations.

Other plant costs are operating supplies, power, and miscellaneous costs. These costs vary from $0.75 to $2.00/ton depending on plant type, age, and size.

The average preparation cost in the survey was $3.71/ton including labor and benefits, with a minimum of $0.34 and a maximum of $8.42. The standard deviation was $1.64/ton. Trucking costs to and from the plant must also be accounted for, but will normally be a function of distance or other variables.

General and administrative costs usually range from $1.00 to $3.00/ton depending upon the mine size, the functions undertaken, and the total company size. The age of the mine is also important in that retirement benefit costs may be included here.

Insurance and property taxes may be hard to estimate in practice. However, each is ordinarily estimated at 1 to 2% of the value of the property on an annual basis.

Royalties and other costs based on the sales price are generally as follows:

1. Royalty.	4–8%
2. Severance tax.	2–20%
3. Reclamation excise tax.	$0.15/ton
4. Black lung excise tax.	$1.10/ton
5. Sales commissions.	1–3%
6. Wheelage.	$0.10/ton − 0.5% of the sales price

Regression Model—A regression model for adjusted cash cost was developed from a sample set of 68 mines and has an R^2 value of 0.69.

The model predicts the adjusted cash cost, which is defined as the total cash cost less royalties. To get total cost, the depreciation and royalty must be added to the number:

$$\mathrm{ACC_u} = 31.08 + \frac{27.85}{\mathrm{TPEH}} - 0.153(\mathrm{YIELD}) - 3.27(\mathrm{INCEN}) - 0.664[\ln(\mathrm{DEPTH})] \qquad (23.3.8)$$

where ACC_u is adjusted cash cost for underground mines.

Example 23.3.5. Estimate the adjusted cash cost of a mine averaging 3 tons/employee-hour, 70% yield, at a depth of 600 ft, with and without an incentive plan.

Solution.

$$\begin{aligned}\mathrm{ACC_u} &= 31.08 + \frac{27.85}{3} - (0.153)(70) - (3.27)(1) - (0.664)(\ln 600)\\ &= 31.08 + 9.28 - 10.71 - 3.27 - (0.664)(6.4)\\ &= \$22.14/\text{ton with an incentive plan}\end{aligned}$$

and $22.14 + 3.27 = \$25.41$/ton without an incentive plan.

Note that the model predicts that cost will decrease as depth increases. This is obviously seldom true in practice, but the reason for this anomaly was not readily apparent from the data. It may be related to the fact that longwalls are typically deeper than non-longwall mines, but this could not be verified from simple calculations.

Example 23.3.6. Compare the above cost to a prediction using the method discussed earlier. First, note that no adjustment is made for the presence or absence of an incentive system in this estimate.

Solution.

Total labor cost—$31.21/3:	$10.40/ton
Other mine cost—$4.50–$8.50/ton-pick lower end of range:	$4.50/ton
Other plant cost—$0.75–$2.00/ton-pick $1.50/ton (larger plant—complex cleaning):	$1.50/ton
Trucking—say, $2.00/ton:	$2.00/ton
General and administrative—$1.00–$3.00/ton, choose midpoint:	$2.00/ton
Royalties and other—omit royalties, Excise taxes—$1.25/ton Severance tax @ 4%—say, $1.20/ton:	$2.45/ton
Predicted cash cost less royalties:	$22.85/ton

Here the difference in estimates is $22.85/ton vs. $25.41/ton or about $2.50/ton. To resolve this difference, detailed mine planning must be undertaken.

SURFACE COAL-MINE COST MODEL. Underground coal mines tend to differ mainly in terms of seam thickness with virtually the same type of equipment and mining plan in each (the exception being the division between longwall and non-longwall mines). Surface mines, on the other hand, tend to differ by mining methods (single-pass contour, multi-pass contour, mountaintop removal, and area stripping, for example) as well as by mix of equipment (such as large draglines vs. small draglines, truck-shovel, front-end loaders, dozers, or combinations of this equipment in the same operation). Operating methods add more variety. For example, the dragline may work on the highwall side or the spoil side; it may also, through the use of extended benches, effectively work in between. Likewise, spoil may be casted using explosives, a dragline bench may be dug entirely by the dragline, or a significant part may be prestripped.

More examples may be offered, but the point is that it is very difficult to provide a relatively simple set of guidelines as were

Table 23.3.7. Statistical Summary of Selected Variables for Surface Coal Mines (Total Number of Mines = 39)

Variable	Mean	Standard Deviation	Range
Seam thickness (in.)—all seams	177	221	25–1320
Stripping ratio (yr^3/ton)	12.8	4.6	1.5–22
Distance to plant (mi)	0.3	0.7	0–3.0
Yield (%)	87.5	17	50–100
Dragline size (yd^3)	56.6	29.6	7–105
Overburden shovel size (yd^3)	21.4	5.4	13–27
Overburden loader size (yd^3)	13.0	1.0	10–15
Coal truck size (tons)	55.0	50.0	22–170
Tons/employee–hour	5.6	3.9	1.6–20.1
Tons/year (10^6)	1.5	3.0	0.021–13.5
Yd^3/year (10^6)	11.5	14.7	0.24–64.4
Hourly employees	59	57	3–204
Salaried employees	23	55	1–253
Mines on incentives (%)	58.3		
Union mines (%)	10.5		
Cost of preparation ($/ton)	1.91	1.04	0.28–4.30
Total cost ($/ton)	23.13	8.72	3.53–43.92
Depreciation ($/ton)	1.74	1.16	0.10–3.45
Royalties ($/ton)	1.48	1.22	0–5.19
Cash cost* ($/ton)	19.95	7.51	3.14–38.36

*Cash Cost = Total Cost − Depreciation − Royalty
Conversion factors: 1 in. = 25.4 mm, 1 mi = 1.609 km, 1 yd = 0.9144 m, 1 ton = 0.9072 t.

given for underground coal mines. Instead, some general discussion of the survey results will be given, followed by a discussion of regression models to predict manning.

Discussion of Survey Results—Complete, usable responses were obtained from 39 operations in eight states. These data are summarized in Table 23.3.7. (It should be noted that no responses were obtained from Illinois or Indiana, two of the primary area-stripping states.) The following primary stripping equipment was used (some mines used multiple stripping equipment):

	No. Mines
Large draglines	12
Stripping shovels	4
Loading shovels	14
Overburden front-end loaders	32

Most of the mines produced coal from seams of modest thickness, and only four, or 10%, were union. Of the nonunion mines, about two-thirds had an incentive plan. Only 16 washed their coal, and the average yield of those washing was 71%. The average cost of preparation was $2.10/ton, considerably less than the deep-mine average, reflecting perhaps the coarser size consistency and ability to do some in-pit separation with surface mining. The total cost vs. stripping ratio and the total cost vs. productivity for surface mining operations are shown in Figs. 23.3.3 and 23.3.4, respectively. Fig. 23.3.3 shows the dependence of the mining cost upon stripping ratio with the expected result that costs increase as the stripping ratio increases. In addition, cost decreases as the productivity increases, as illustrated in Fig. 23.3.4. These plots show, however, that there exists significant variation about the trend, probably as a result of different methods, equipment, and mining conditions.

The average strip ratios by state (for those states with more than one response) are shown in Table 23.3.8. Note that because the operator can chose his maximum stripping ratio with all but the area method, these may be an indicator of the maximum

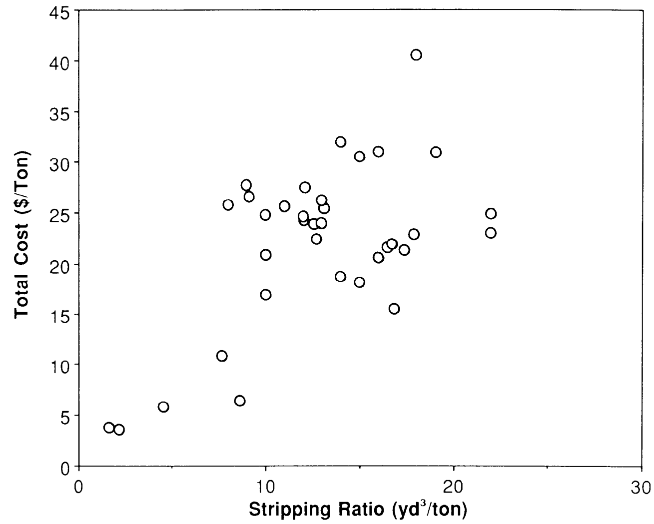

Fig. 23.3.3. Plot of cash survey results showing total cost vs. stripping ratio for surface mining operations. Conversion factors: 1 ton = 0.9072 t, 1 yd^3/ton = 0.8428 m^3/t.

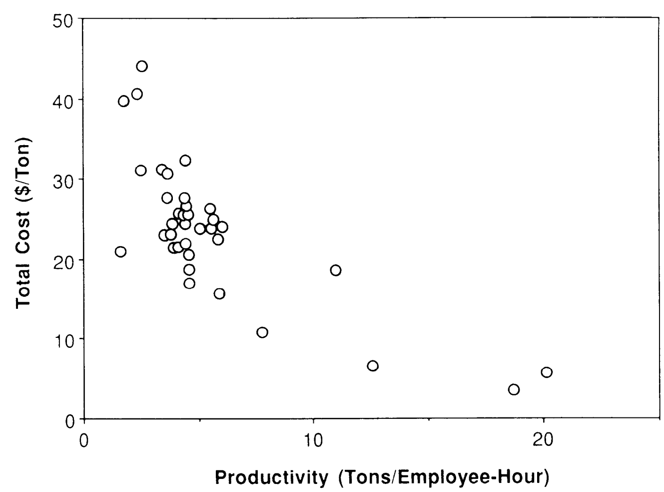

Fig. 23.3.4. Plot of cost survey results showing total cost vs. productivity for surface mining operations. Conversion factor: 1 ton = 0.9072 t.

Table 23.3.8. Average Stripping Ratio (yd^3/ton) for Surface Coal Mines Reported by State.

State	Number of Mines	Average	Range
Kentucky	17	13.7	10–19
North Dakota	3	6.9	4–9
Pennsylvania	4	17.3	10–22
Virginia	7	15.4	9–18
West Virginia	5	9.6	8–12
Wyoming	2	1.8	NA

Conversion factor: 1 yd^3/ton = 0.8428 m^3/t.

ratio economically minable at today's prices. It should be noted that the stripping ratios appear to reflect the energy content and the geographic location of the coal seams.

Regression Models—Regression models were developed to predict the total employees at a mine, the likely productivity in tpeh, and the cash cost of production less royalties. These are discussed here.

Total employees was found to be a function of the number of tons mined per year and whether or not a dragline larger than 20 yd^3 was used. The regression equation is as follows:

$$TOTE_{s1} = 64.8\,(BIGDG) + 30.28\,[\ln\,(TPY)] - 338.7 \qquad (23.3.9)$$

where $TOTE_{s1}$ is total employees for surface mines (method l), and BIGDG is 1 if a dragline larger than 20 yd^3 is used and 0 otherwise. This model predicted the employees with an R^2 value of 74%. The mean number of employees at the mines surveyed was 70.

A second model was also developed which takes stripping ratio into account; it also has an R^2 value of 74%:

$$TOTE_{s2} = 62.7(BIGDG) + 31.37[\ln\,(TPY)] + 0.66(RATIO) - 360.9 \qquad (23.3.10)$$

where $TOTE_{s2}$ is total employees, surface mines (method 2), and RATIO is yd^3 of overburden moved per ton of coal mined.

Example 23.3.7. Estimate the employment at a mine with an 18:1 strip ratio using a large dragline and producing 1 million tpy.

Solution.

$$\begin{aligned}\text{Using Eq. 23.3.9: } TOTE_{s1} &= 64.8(1) + 30.28[\ln(10^6)] - 338.7\\ &= 64.8 + 418.33 - 338.7\\ &= 144 \text{ employees}\\ \text{Using Eq. 23.3.10: } TOTE_{s2} &= 62.7(1) + 31.37[\ln(10^6)] + 0.66(18) - 360.9\\ &= 62.7 + 433.4 + 11.9 - 360.9\\ &= 147 \text{ employees}\end{aligned}$$

Compare this to a mine producing 500,000 tpy of coal, a 12:1 stripping ratio, and no dragline.

$$\begin{aligned}\text{Using Eq. 23.3.10: } TOTE_{s2} &= 62.7(0) + 31.37[\ln\,(500{,}000)] + 0.66(12) - 360.9\\ &= 0 + 411.7 + 7.9 - 360.9\\ &= 59 \text{ employees}\end{aligned}$$

An estimator for productivity was also developed from the survey data. The average productivity for all mines surveyed was 5.8 tpeh. In this model the key independent variables are mine size (tons per year), strip ratio, plant yield, and presence or absence of a union. This equation has an R^2 value of 75%:

$$TPEH_{s1} = 1.37[\ln(TPY)] - 0.60(RATIO) + 0.069(YIELD) - 3.27(UNION) - 10.24 \qquad (23.3.11)$$

where $TPEH_{s1}$ is tons per employee-hour for surface mines (method 1). Note that the sign on the coefficient of RATIO is opposite to that which would logically be expected.

An alternative equation, using mining plans as independent variables has an R^2 value of 73%

$$TPEH_{s2} = 1.27[\ln(TPY)] - 0.51(RATIO) + 0.058(YIELD) + 1.28(AREA) - 4.04(SPC) - 9.78 \qquad (23.3.12)$$

where $TPEH_{s2}$ is tons per employee-hour or surface mines (method 2), AREA is 1 if this is an area mine or 0 otherwise, and SPC is 1 if this is a single-pass contour mine or 0 otherwise.

Example 23.3.8. Estimate the tpeh for the two mines described above if they are union, the 18:1 ratio is an area mine, and the 12:1 ratio is a mountaintop removal operation, and both produce direct-shipping coal.

Solution.

For mine 1 using Eq. 23.3.11:

$$\begin{aligned}TPEH_{s1} &= 1.37(\ln(10^6)) - 0.60(18) + 0.069(100) - 3.27(1) - 10.24\\ &= 18.93 - 10.80 + 6.9 - 3.27 - 10.24\\ &= 1.52 \text{ tpeh}\end{aligned}$$

For mine 1 using Eq. 23.3.12:

$$\begin{aligned}TPEH_{s2} &= 1.27(\ln 10^6) - 0.51(18) + 0.058(100) + 1.28(1) - 4.04(0) - 9.78\\ &= 17.55 - 9.18 + 5.80 + 1.28 - 4.04 - 9.78\\ &= 1.63 \text{ tpeh}\end{aligned}$$

For mine 2 using Eq. 23.3.11:

$$\begin{aligned}TPEH_{s1} &= 1.37(\ln\ 500{,}000) - 0.6(12) + 0.069(100) - 3.27(1) - 10.24\\ &= 17.98 - 7.20 + 6.90 - 3.27 - 10.24\\ &= 4.17 \text{ tpeh}\end{aligned}$$

For mine 2 using Eq. 23.3.12:

$$\begin{aligned}TPEH_{s2} &= 1.27(\ln\ 500{,}000) - 0.51(12) + 0.058(100) + 1.28(0) - 4.04(0) - 9.78\\ &= 16.67 - 6.12 + 5.80 + 1.28(0) - 4.04(0) - 9.78\\ &= 6.57 \text{ tpeh}\end{aligned}$$

For the latter mine, the estimates are more uncertain.

The adjusted cash cost less royalties (ACCS) was found to be a function of the stripping ratio, yield, and mining plan. It may be noted that the average value of adjusted cash cost was $19.78/ton. The following model has an R^2 of 81%:

$$ACC_s = 39.99 + 0.69(RATIO) - 0.14(YIELD) - 1.45[\ln(TPY)] + 3.39(MTR) + 23.06(SPC) \qquad (23.3.13)$$

where ACC_s is adjusted cash cost for surface mines, MTR is 1 if this is a mountaintop removal mine or 0 otherwise.

Example 23.3.9. Using the two previous mines,

$$\begin{aligned}\text{Mine 1: } ACC_s &= 39.99 + 0.69(18) - 0.14(100) - 1.45(\ln 10^6) + 3.39(0) + 23.06(0)\\ &= \$18.87/\text{ton}\\ \text{Mine 2: } ACC_s &= 39.99 + 0.69(12) - 0.14(100) - 1.45(\ln 10^6) + 3.39(1) + 23.06(0)\\ &= \$18.12/\text{ton}\end{aligned}$$

Eq. 23.3.13 should probably not be used for single-pass contour mines because of a limited number of observations of this type of mining in the data set.

Table 23.3.9. Open Pit Noncoal Mine Cost Data (Effective Mid-1989)

Large-Size Mines (> 20 million tons/yr of Ore and Waste):

Mine	Commodity Group	Truck Size (tons)	Haul Length (mi)	Stripping Ratio		Cost Estimates ($/ton moved)						
				yd³/ton	tons/ton	Drilling	Blasting	Loading	Haulage	Auxiliary Cost[1]	Direct Mining[2]	Total Mining[3]
A	Metallics	190	1.2		1.85	0.030	0.056	0.085	0.328	0.125	0.62	0.73
B	Metallics	170	0.7		3.01	0.024	0.058	0.148	0.257	0.154	0.64	
C	Metallics	170–190	2.0		1.12	0.035	0.056	0.133	0.247	0.161	0.63	
D	Metallics	170	7.0	0.11		0.020	0.275	0.076	0.156	0.062	0.59	1.96
E	Metallics	120–170	18.0		2.10	0.023	0.038	0.125	0.276	0.050	0.51	0.73
F	Metallics		0.1		1.35	0.034	0.046	0.078	0.294	0.115	0.57	0.76

Medium-Size Mines (2 million tons/yr < Production of Ore and Waste < 20 million tons/yr):

Mine	Commodity Group	Truck Size (tons)	Haul Length (mi)	Stripping Ratio		Cost Estimates ($/ton moved)						
				yd³/ton	tons/ton	Drilling	Blasting	Loading	Haulage	Auxiliary Cost[1]	Direct Mining[2]	Total Mining[3]
G	Industrial Minerals	35		12.50		0.139	0.221	0.194	0.281	0.291	1.13	1.89
H	Metallics	100	0.5	2.44		0.090	0.150	0.110	0.250	0.320	0.92	1.45
I	Metallics	75–85	0.5–1.5	4.00		0.060	0.050	0.130	0.330	0.260	0.83	1.01
J	Industrial Minerals	50	1.5	0.00		0.183	0.161	0.103	0.742		1.19	1.49
K	Industrial Minerals	50	2.0	0.06		0.040	0.040	0.340	0.400		0.82	0.95
L	Metallics		2.9		11.5	0.021	0.029	0.073	0.203	0.067	0.39	0.54
M	Metallics	85	5.0	2.00		0.116	0.087	0.147	0.208	0.131	0.69	0.95

Small-Size Mines (< 2.0 million tons/yr of Ore and Waste):

Mine	Commodity Group	Truck Size (tons)	Haul Length (mi)	Stripping Ratio		Cost Estimates ($/ton moved)						
				yd³/ton	tons/ton	Drilling	Blasting	Loading	Haulage	Auxiliary Cost[1]	Direct Mining[2]	Total Mining[3]
N	Industrial Minerals	50	11.0	0.00		0.113	0.033	0.083	2.472		2.70	3.00
O	Industrial Minerals	50	2.5	0.00		0.123	0.027	0.218	0.879		1.25	1.55
P	Industrial Minerals	50	2.0		0.31	0.090	0.170	0.170	0.920		1.35	3.65
Q	Industrial Minerals	60	1.0		0.50	0.240	0.180	0.310	0.450	0.05	1.23	1.73
R	Industrial Minerals	50–85	1.0	0.78		0.100	0.210	0.250	0.810	0.12	1.49	3.82
S	Industrial Minerals	35	1.0	0.02		0.000	0.000	0.500	0.510		1.01	1.46
T	Industrial Minerals	35 –50	2.0	0.00		0.190	0.180	- - - 0.43 - - -		0.70	1.50	
U	Metallics		0.5–5.0	2.33		1.000	0.400	0.900	1.900		4.20	5.00

Note: Missing values not provided by the source.
[1] Cost of support, pumping, roads, etc.
[2] Sum of drilling, blasting, loading, haulage, and auxiliary costs.
[3] Total mining costs include development, exploration, services, and administrative costs.
Conversion factors: 1 mi = 1.609 km, 1 yd = 0.9144 m, 1 ton = 0.9072 t.

23.3.5.2 Cost Estimation in Noncoal Mining

Costs related to the surface and underground mining of metals, industrial minerals, and other categories of noncoal minerals are not easy to find in the literature. To provide some useful cost data for this segment, questionnaires were prepared for both underground and surface mining. These were sent to a large number of mining companies with the help of the American Mining Congress. The authors acknowledge the AMC's considerable help in the provision of the costs below.

SURFACE MINING. Open pit mining has become the primary method of mining many of the base metals, gold, and industrial minerals in the United States and the costs reflect the obvious reason for this trend. The various categories of costs for 23 different open pit mines are presented in Table 23.3.9. It should be noted that data have been omitted from the table if they seemed to provide little enlightenment or if they had the potential for identifying the individual mines. The mines are quite well distributed geographically and represent both the metal and industrial minerals segments of the mining industry. The cost patterns in this table show a great deal about the cost of mining by the open pit method. First, it is rather clear that the costs of producing minerals by the open pit method have not risen very rapidly over the last two decades if the data are compared with open pit mining costs in Michaelson and Hammes (1968). Several reasons for this modest cost increase are apparent including a period of intense foreign competition plus continued increases in equipment size and efficiency.

Second, it seems apparent from the raw data that a significant decrease in costs occurs when the size of the mine is increased. This is shown in Table 23.3.10 where the average costs as a function of mine size are presented. The data thus reveal a trend that speaks highly of the desirability of larger operations. This is particularly striking when it is realized that the costs are shown to decrease with an increase in size for each category of cost (drilling, blasting, loading, hauling, and auxiliary costs) as well as for the overall direct mining costs. This appears to be primarily a result of the efficiency of larger equipment, though better engineering, planning, maintenance, and management may play a part as well.

Other attempts at finding significant statistical trends in the data by analyzing mining costs vs. truck size, mining costs vs. haul distance, and a number of other such relationships were not

Table 23.3.10. Open Pit Noncoal Mine Costs as a Function of Pit Production (Effective Mid-1989)

	Average Costs from Table 23.4.2 in $/ton mined					
	Drilling	Blasting	Loading	Haulage[1]	Auxiliary Costs[2]	Direct Mining[3]
Large-Size Mines (Production > 20 million tons/yr)	0.028	0.088	0.108	0.260	0.111	0.59
Medium-Size Mines (Production from 2 million to 20 million tons/yr)	0.093	0.105	0.157	0.345	0.214	0.85
Small-Size Mines (Production < 2 million tons/yr)	0.265	0.171	0.347	1.134	0.290	1.84

[1] Comparison of haulage costs may not be meaningful as haulage lengths vary.
[2] Cost of support, pumping, roads, etc.
[3] Sum of drilling, blasting, loading, haulage, hoisting, and auxiliary costs.
Conversion factor: 1 ton = 0.9072 t.

as successful. It may have been that there were too many variables related to the costs to easily find such trends in the data.

Another surface mining method for which some data were obtained is the stripping of phosphates with hydraulic transport of the ore to the processing facility. The costs reported for this mining method ranged from $6.00/ton to $12.50/ton. The extraction method used in these phosphate operations is more like a typical coal method than a metal or nonmetallic mining method. It is, however, unique to the phosphate industry that hydraulic transport is routinely used to move the ore material from the mining bench to the processing facility. It should be noted that the costs are based upon just a few mines. Thus the costs should be used with caution.

UNDERGROUND MINING. In attempting to gather cost data on underground mining, costs were solicited on a variety of methods using a common cost questionnaire. As might be expected, the amount of cost information received varied considerably with the mining method type. More information was received on room and pillar/stope and pillar mining methods than on other underground methods. The data are presented in Table 23.3.11 with the mines with a regular pillar pattern (room and pillar) separated from the mines with a random pillar pattern (stope and pillar). It should be noted that costs for mines with a regular pillar pattern are quite low, ranging from $2.90/ton to $7.00/ton. In general, these low costs can be attributed to the nature of the mineral deposits mined in these mines. The mines are primarily relatively soft evaporites that can be mined using continuous miners or conventional equipment. As a result of the productivity of that equipment and the general uniformity of conditions in these evaporite mines, mining costs are often among the lowest in the underground mining category.

The data in Table 23.3.11 relating to random pillar mines show costs ranging about twice as high as that for regular pillar mines with the range of direct mining costs being $4.07/ton to $9.81/ton for most of the mines. It should be noted, however, that costs for one mine using slushers in the stopes are considerably higher. In general, when one compares the costs for stope and pillar operations with those for room and pillar operations, it is clear that stope and pillar mines will have higher costs because of the harder rock, less uniform conditions, and other factors that make hard-rock mining more difficult than soft-rock mining. Nonetheless, it is difficult to draw any broad or sweeping conclusions because of the limited number of mines represented here.

One other mining method for which meaningful results were obtained is the cut and fill method. These data are presented in Table 23.3.12. While the number of mines reported is small, a general trend in the data is evident. When the data are divided into mines with veins narrower than 10 ft (3 m) and mines with veins greater than 10 ft (3 m), some fairly obvious cost differences are apparent. The mines exploiting veins greater than 10 ft (3 m) have costs from $35/ton to $41/ton while those exploiting veins less than 10 ft (3 m) have costs from $63/ton to $116/ton. Apparently, the vein or deposit size has the potential to significantly affect the costs due to the productivity improvement in larger stopes, particularly if more efficient load-haul-dump equipment can be utilized. These costs also tend to show one of the reasons that the mining of narrow-vein deposits is no longer widely practiced in this country. The costs of producing ore from these veins has been rising faster than other mining methods due to the labor-intensive nature of the methods and the inability to ease that situation through the use of larger, more efficient equipment.

Several other mining methods were represented in the data returned from the survey. However, none had more than a handful of mines represented in the final tally of cost figures. These are presented in Table 23.3.13 in summary form. Block caving still represents the least expensive of the traditional metal mining methods with costs reported in the $2.50/ton to $8.00/ton range (specific cost values were omitted here to prevent identification of individual mines). Methods with moderate costs in the survey results were the vertical crater retreat method ($20/ton to $30/ton), shrinkage stoping ($30/ton to $40/ton), and slusher stoping ($30/ton to $40/ton). Slusher stoping is defined here as the mining of open stopes with dips greater than that for which rubber-tired equipment can be used and less than that for which gravity can be used to extract the ore from the stope. This method is practiced in a few mines where other methods are generally not suitable. Because of the small number of observations for each mining method represented in Table 23.3.13, the costs should be used with great caution.

23.3.6 METHOD SELECTION CONSIDERATIONS

The selection of a mining method is a procedure that will involve many variables and considerations. Chapter 23.1, Selection Variables, outlines many of these variables in a general fashion. For surface methods, the geologic and geometric variables of the ore body will often be of most importance. For underground methods, particularly those aimed at metallic deposits, rock mechanics variables will often play a very big part in the choice of a mining method. This subject is an important part of the discussion in Chapter 23.4, Selection Procedure. However important geologic and geomechanics variables are in the choice of a method, economics and cost variables are also

Table 23.3.11. Room and Pillar/Stope and Pillar Noncoal Mine Cost Data (Effective Mid-1989)

Mines With a Regular Pillar Pattern (Primarily Evaporite Mines):

Mine	Million Tons/Yr	Equipment Type	Cost Estimates ($/ton) Drilling	Blasting	Loading	Haulage	Hoisting	Auxiliary Costs[1]	Direct Mining[2]	Total Mining[3]
A	>5	Continuous Miner			0.76	0.49	0.34	0.64	2.23	3.08
B	>5	Continuous Miner			1.36	1.49	0.34	0.64	3.83	5.18
C	1 to 5	Continuous Miner			0.64	0.41	0.41	0.64	2.10	2.90
D	1 to 5	Continuous Miner			1.25	1.41	0.41	0.64	3.71	5.00
E	>5	Conventional	0.12	0.29	0.18	0.43	0.43	1.16	2.80[4]	3.47
F	1 to 5	Conventional	0.18	0.43	0.55	0.23	0.43	0.80	2.62	7.00

Mines With a Random or Unknown Pillar Pattern (Primarily Metal Mines):

Mine	Million Tons/Yr	Equipment Type	Cost Estimates ($/ton) Drilling	Blasting	Loading	Haulage	Hoisting	Auxiliary Costs[1]	Direct Mining[2]	Total Mining[3]
G	<1	LHD	- - - 2.14 - - -		- - - 0.50 - - -		0.22	3.33	6.19	
H	<1	LHD	- - - 1.56 - - -		- - - 0.50 - - -		0.19	3.96	6.21	
I	<1	Slusher	6.58	1.10	7.24	1.45	1.21	36.09	53.67	86.00
J	1 to 5	LHD	- - - 1.74 - - -		- - - 0.84 - - -		0.26	2.78	5.62	
K	<1	LHD	- - - 2.57 - - -		- - - 0.89 - - -		0.38	5.97	9.81	11.37
L	1 to 5	Loader/Truck	1.10	0.68	0.51	1.03	0.34	4.24	7.90	
M	1 to 5	LHD	0.65	0.53	- - - 0.17 - - -		0.22	2.94	4.51	
N	1 to 5	LHD	1.13	0.53	- - - 0.11 - - -		0.26	5.04	7.07	
O	1 to 5	Loader/Truck	1.12	0.54	0.49	1.00		0.95	4.10	7.46

Note: Missing values not provided by the source.
[1] Auxiliary costs include support, ventilation, pumping, power, etc.
[2] Sum of drilling, blasting, loading, haulage, hoisting, and auxiliary costs.
[3] Total mining costs include development, exploration, services, and administrative costs.
[4] Includes undercutting costs.
Conversion factor: 1 ton = 0.9072 t.

Table 23.3.12. Cut and Fill Mine Stoping Costs (Effective Mid-1989)

Mine	Ore Width (ft)	Equipment Type	Costs ($/ton) Drilling	Blasting	Loading	Haulage	Hoisting	Auxiliary Costs[1]	Direct Mining[2]	Total Mining[3]
A	>10	LHD	- - - - - - 17.37 - - - - - -			3.02	3.00	2.97	26.36	35.35
B	<10	Slusher/LHD	- - - - - - 34.00 - - - - - -			2.00	4.00	28.00	68.00	115.00
C	>10	LHD	- - - 2.30 - - -		3.30	2.92	1.57	10.73	20.82	41.31
D	<10	Slusher	- - - 5.26 - - -		3.67	5.33	4.65	25.04	43.95	80.35
E	<10	Slusher	- - - 4.15 - - -		3.74	3.08	2.18	20.95	34.10	62.68
F	<10	Slusher	- - - - - - 29.50 - - - - - -			4.50	4.30	18.50	56.80	116.00

[1] Auxiliary costs include support, fill, ventilation, pumping, power, etc.
[2] Sum of drilling, blasting, loading, haulage, hoisting, and auxiliary cost.
[3] Total mining costs include development, exploration, services, and administration costs.
Conversion factors: 1 ft = 0.3048 m, 1 ton = 0.9072 t.

Table 23.3.13. Costs Related to Other Mining Methods

Method	Direct Mining Cost ($/ton)
Block Caving	2.50– 8.00
Vertical Crater Retreat Mining	20.00–25.00
Shrinkage Stoping	30.00–40.00
Slusher Stoping	30.00–40.00

Note: These costs are based upon very small numbers of observations. The costs should therefore be considered as order-of-magnitude estimates only.
Conversion factor: 1 ton = 0.9072 t.

certain to be near the top of the list. It is therefore important that the mining engineer have some idea at the beginning of the method selection procedure as to how the various mining methods rank in the scale of costs. To provide some idea of how the methods compare in cost, Table 23.3.14 was compiled using two sources. The first source was the cost survey performed by the authors for this chapter. These costs were representative of the situation in mid-1989. The second source was the cost models of the USBM cost estimating system (Anon., 1987a). The costs derived from these models were calculated and updated to December 1988 and published by *Mining Cost Services* (Schumacher, 1989).

To render the cost values in Table 23.3.14 as comparable as possible, both sets of costs were reduced to the direct costs of mining that included the drilling, blasting, loading, haulage, hoisting, and auxiliary costs directly connected with the mining

Table 23.3.14. Summary and Comparison of Direct Mining Costs[1]

Mining Method/Conditions	Direct Mining Costs ($/ton) Handbook Survey (Mid-1989 Costs)	Mining Method/Conditions	Direct Mining Costs ($/ton) Cost Estimating System[2] (December 1988 Costs)
Metals and Nonmetallics:			
Open Pit Mining:		Open Pit (1:1 Stripping Ratio):	
Large Mines (> 20 million tons/yr)	$0.51–0.64	Large Mines[3] (> 20 million tons/yr)	$0.78
Medium Mines (2 to 20 million tons/yr)	0.39–1.19	Medium Mines[3] (2 to 20 million tons/yr)	0.84–1.13
Small Mines (< 2 million tons/yr)	1.01–4.20	Small Mines[3] (< 2 million tons/yr)	3.65–6.68
Phosphate stripping with hydraulic transport	6.00–12.50		
Room and Pillar Mining (Shaft Mines)	2.10–3.83	Room and Pillar (Adit Mine, 5500 to 16,500 tons/day)	2.00–2.89
		Room and Pillar (Shaft Mines, 5500 to 16,500 tons/day)	2.11–3.66
Stope and Pillar Mining (Diesel Equipment)	4.10–9.81		
Stope and Pillar Mining (Slusher)	53.67		
Cut and Fill Stoping (< 10-ft Vein, Slushers)	34.10–68.00	Cut and Fill (Adit Mine, 1100 tons/day)	34.31
Cut and Fill Stoping (> 10-ft Vein, LHDs)	20.82–26.36	Cut and Fill (Shaft Mines, 1100 tons/day)	42.10–43.13
Block Caving	2.50–8.00	Block Caving (44,000 tons/day)	1.86–2.44
Vertical Crater Retreat Mining	20.00–25.00	Vertical Crater Retreat (Adit Mine, 880 tons/day)	6.65
		Vertical Crater Retreat (Shaft Mine, 880 tons/day)	8.90–9.40
Shrinkage Stoping	30.00–40.00	Shrinkage (Adit Mine, 110 tons/day)	38.54
		Shrinkage (Shaft Mine, 110 tons/day)	64.85–65.10
Slusher Stoping	30.00–40.00		
Coal:			
Stripping	3.00–34.00		
Room and Pillar Conventional Mining	15.00–30.00		
Room and Pillar Continuous Mining	13.00–40.00		
Longwall Mining	12.00–34.00		

[1] Direct mining costs include drilling, blasting, loading, haulage, hoisting, and auxiliary costs. Administration, development, and processing costs have been excluded.
[2] These costs were calculated using costs supplied by *Mining Cost Service* (Schumacher, 1989, used with permission). The costs are based upon the 1984 cost models of the Bureau of Mines' Cost Estimating System. The costs were updated to December 1988 using updating techniques. Those values were then converted to direct mining costs per ton to make comparison possible.
[3] Yearly tonnages were calculated using daily tonnage and multiplying by 250 days/yr.
Conversion factor: 1 ton = 0.9072 t.

process. All administration, development, and processing costs have been omitted from the cost figures. In addition, an attempt was made to make the costs compatible in terms of the size of the operation and the mining variables (equipment utilized, type of mine opening, depth, vein width, etc.). The USBM costs were available only for noncoal mining methods, and thus no comparison is available for coal mining methods.

Comparison of the costs in Table 23.3.14 shows that the two cost sources are in good agreement on some of the methods and more variable on others. For example, both sources seem to provide smaller estimates for the costs for room and pillar and cut and fill mining. The costs for open pit mining and shrinkage stoping were somewhat different in the two sources, while the costs for block caving and the VCR method are relatively inconsistent. A number of possible reasons for these differences exist. First, most of the costs are based upon relatively few mines. Second, in our survey the mining companies were asked to provide costs that were free of development costs. Because these may not have been available, the cost values may not have been entirely consistent. Finally, variations in mining and accounting practices may contribute to the variations in the costs reported.

The costs reported here, while generally useful for general comparison purposes, must be considered as order-of-magnitude estimates only. They may help in the early evaluation of mining methods but must be reevaluated and made site- and condition-specific at the time that the methods have been narrowed down to a few candidates. At that time, both the operating and capital cost estimates must be assembled, and the estimates can be made using detailed estimating procedures that will allow both specifics of the method and specifics of the geology and the rock mechanics to be considered in the estimation process. In particular, it is important to construct a detailed capital requirements list to accurately estimate the capital cost of a new mine. Since there are few good sources in the literature that will provide much of this information, detailed listing and costing of capital goods will normally be necessary.

23.3.7 COST CONTROL

Due to the difficult situation in the minerals industry during the 1980s in which the industry has had to deal with foreign competition of an unprecedented level and intense domestic competition for the remaining market, cost control and cost reduction methods have become an important part of many mining operations. These endeavors often span the accounting, management, and engineering functions within an organization. Over the years, many different procedures have thus emerged with the industrial engineers leading in the development of procedures among the engineering disciplines. Cost control procedures have taken many different forms and variations. It would thus be impossible to mention them all here; instead, only the most important procedures are reviewed. Readers who are interested in more detailed lists of procedures should refer to the section by Scott (1971) in the *Industrial Engineering Handbook.*

23.3.7.1 Cost Control Procedures

COST ACCOUNTING AND ANALYSIS. Cost accounting is a procedure that allows an organization to make some measurement of performance that may help in the period-to-period management of costs. Nearly all minerals companies utilize cost accounting as a routine procedure so that unusual cost shifts can be studied for potential savings, so that production enhancement methods can be evaluated, and so that outstanding performances can be analyzed for their potential in other areas of the organization. Along with cost accounting procedures, it is often necessary to perform some analysis of the trends so that problem areas can be pinpointed, waste can be controlled, and better performance can be recognized. While this area of endeavor does not appear very glamorous, application of cost control procedures through cost accounting can provide an important tool to keep costs within limits.

BUDGETARY CONTROL. The use of budgets as a cost control procedure is, in many ways, similar to cost accounting in that it provides a procedure for measuring cost performances on a period-to-period basis in an organization. The budgetary control procedure, however, provides a target (the budget allocated) in order to provide a further means of control. The budgeting process will help to detect costs that are out of control and will enable different production units within the organization to compare costs and revise cost control methods. This would be particularly applicable in minerals companies where similar production units within an organization are performing the same job on a parallel basis.

METHODS ENGINEERING. Methods engineering is here meant to be the area of endeavor in which jobs are analyzed for their elements, measurements are made on the elements, and job improvements are made through standardization, job simplification, worker education, and other related methods. This area, which has been the province of the industrial engineer for many years, has seen many forms of application in the minerals industry over the years. The use of time and motion studies has, over time, produced more efficient and more productive operations. Other types of related activities, including better worker education, have also been emphasized at times. The emphasis in this area seems to shift from one type of activity to another, but the general strategy continues to be to analyze the activities involved and to make the most of the resources available.

OPERATIONS RESEARCH. Operation research, systems analysis, and statistical analysis are all related methods that have been used to control prices. In general, these methods are used to control prices through the use of the available resources in an optimal manner such that the costs are minimized or the profits maximized. These methods differ from the traditional cost control or cost reduction methods in that they make use of mathematical or statistical methods, sometimes quite complicated, to achieve the optimization of the system under study. While the methods are often classified as optimization methods rather than cost control methods, their influence on costs is obvious. Readers interested in a more complete description of these methods should refer to Chapter 8.3 of the *Handbook*, Systems Engineering.

PRODUCTIVITY IMPROVEMENT. One of the most important cost control and cost reduction techniques that has been practiced in the minerals industry has been productivity improvement. This has come about due to the necessity of remaining competitive in the minerals marketplace. Productivity improvement has come about through the use of larger or otherwise improved equipment, use of improved methods for performing certain mining procedures, and through better use of the available personnel in the mines. The value of productivity improvement as a cost reduction technique will be illustrated in the next section.

Table 23.3.15. Mining Costs at the Mine in Ex. 23.3.10

		Cost ($/ton)
Mine Cost		
1. Labor ($58,000)/(4158 tons/miner-yr)		$ 13.95
2. Supplies		
A. Operating	$2.50	
B. Maintenance and Repairs	$2.50	$ 5.00
3. Power		$ 0.70
4. Miscellaneous		$ 0.10
	Subtotal	$ 19.75
Plant Cost		
1. Labor ($58,000)/(46,200 tons/miner-yr)		$ 1.25
2. Supplies		$ 1.00
3. Power		$ 0.20
4. Contract Trucking		$ 1.00
	Subtotal	$ 3.45
General and Administrative		$ 2.00
Miscellaneous Variable Costs		
1. Reclamation Excise Tax		$ 0.15
2. Black Lung Excise Tax		$ 1.10
3. Royalty		$ 1.00
	Subtotal	$ 2.25
Depreciation and Amortization		$ 2.55
	Total	$ 30.00

23.3.7.2 Application in Minerals Production

The control and reduction of costs has been a primary concern of the minerals industry in the United States throughout the 1980s. A look at some of the possibilities available to mine management under strong competition can illustrate the procedures for cost reduction in the minerals industry. This will be accomplished through the use of an example that applies to the underground production of coal by the room and pillar mining method.

Example 23.3.10. A room and pillar mining operation is operating in an Appalachian coal seam that cannot be mined easily by surface or longwall mining methods. The company operating the mine has basically been pleased with the operation and its management over the past few years. The equipment is relatively new and well maintained and the nonunion workforce has been cooperative with management suggestions for productivity improvement. An abbreviated cost sheet showing various categories of costs on a per-ton basis is provided in Table 23.3.15.

The company has recently had to accept a one-year sales contract at $28/ton due to the competition in their area for the available coal market. To institute a cost reduction program, a new mine superintendent is appointed. He is charged with bringing the costs down below $28/ton within one year, with the threat that the mine will be closed after a year if the reduction cannot be achieved. The superintendent carefully studies the cost

Table 23.3.16. Mining Costs at the Mine in Ex. 23.3.10 After Supply and General and Administrative Cost Reductions

		Cost ($/ton)
Mine Cost		
1. Labor ($58,000)/(4158 tons/miner-yr)		$ 13.95
2. Supplies		
A. Operating	$2.00	
B. Maintenance and Repairs	$2.50	$ 4.50
3. Power		$ 0.70
4. Miscellaneous		$ 0.10
	Subtotal	$ 19.25
Plant Cost		
1. Labor ($58,000)/(46,200 tons/miner-yr)		$ 1.25
2. Supplies		$ 1.00
3. Power		$ 0.20
4. Contract Trucking		$ 1.00
	Subtotal	$ 3.45
General and Administrative		$ 1.50
Miscellaneous Variable Costs		
1. Reclamation Excise Tax		$ 0.15
2. Black Lung Excise Tax		$ 1.10
3. Royalty		$ 1.00
	Subtotal	$ 2.25
Depreciation and Amortization		$ 2.55
	Total	$ 29.00

data to determine where he can cut costs to achieve his goal. He targets two areas of costs that appear to him to be high: (1) operating supplies and (2) general and administrative costs. One of his top assistants is retiring, and one of the mine engineers has resigned. This gives him some opportunity to reduce the administrative and general staff, a strategy that he feels is both possible and desirable. In addition, he transfers a couple of other general staff members to other mines and achieves a 25% reduction in general and administrative costs that results in a $0.50/ton reduction in the cost of the coal.

At the same time, he institutes a program of holding down costs of operating supplies. He lets the work force know of the financial problems of the company and encourages them to help out by eliminating unnecessary waste of supplies. In addition, he has one of his staff members keep careful control over the supplies used and makes him responsible for ensuring that supplies are not abandoned underground.

After six months of intensive efforts, the reduction in supplies is 20% with an accompanying $0.50/ton savings in the cost of producing coal. The costs after the savings are achieved are given in Table 23.3.16. While the savings have helped, the overall goal of achieving a profit has not been achieved.

At this point, the superintendent clearly needs to make some other strategic move to further reduce his costs. Assuming that he targets productivity improvement as his only other feasible cost reduction technique, several possibilities may exist. First, he can appeal to the work force to improve productivity, using the (valid) threat of closure as incentive, or he can offer the workforce some financial incentive. The next example illustrates the logic of the second strategy.

Example 23.3.11. After considerable thought, the superintendent decides that the work force is doing productive work and that he feels the need to provide a monetary reward to the workers as the mine-saving strategy. He communicates the following to the workers. If the cost of $28/ton is bettered, the workers will share the profits with the company on a 70/30 basis with the workers sharing 30%. To determine what the results will be under different productivities, he prepares Table 23.3.17 which follows. The table clearly shows that it will take about a 5% increase in productivity to break even. Because the bonus is based on total costs, an improvement in productivity over 5% will provide both a profit and an incentive payment for the work force. The superintendent shares the information with everyone at the mine and hopes for the best.

Table 23.3.17. Costs and Related Information as a Function of Productivity Improvement (Ex. 23.3.11)

Productivity (tons/manshift)	Labor Cost ($/ton)	Cost Before Incentive ($/ton)	Incentive Per Employee ($/yr)	Cost After Incentive ($/ton)
17.0 (Status Quo)	$15.20	$29.00	$ 0	$29.00
17.9 (+5%)	14.48	28.00	0	28.00
18.7 (+10%)	13.28	27.09	1904	27.55
19.5 (+15%)	13.22	26.26	3811	27.13
20.4 (+20%)	12.67	25.50	5717	26.75
21.3 (+25%)	12.16	24.80	7623	26.40
22.1 (+30%)	11.69	24.16	9530	26.08
23.0 (+35%)	11.26	23.56	11436	25.78
23.8 (+40%)	10.86	23.01	13342	25.50

Assumptions:
1. Base supply cost is 20% fixed; base power cost is 70% fixed.
2. Incentive per employee based on 224.5 miner shifts per year.

The previous example shows several interesting results. First, the cutting of costs can produce some results, but a large leverage is often achieved when increasing productivity. As shown here, using typical coal mine costs, a 20% reduction in some costs does not greatly help the bottom line, but a 20% increase in productivity produces a much more significant cost reduction. Second, the use of incentives can be a powerful force to reduce costs. The use of incentives has often been achieved without any increase in accidents, but this must be emphasized in any incentive program through education of the work force and use of safety performance in the calculation of incentives so that the increase in productivity is not accompanied by an increase in accident rates.

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Chapter 23.4
SELECTION PROCEDURE

DAVID E. NICHOLAS

23.4.1 GENERAL

The purpose of this section is to present the logical sequence of events to follow in choosing a mining method. Characteristics that have a major impact on the determination of the mining method (as discussed in Chapter 23.1, Selection Variables) are

1. Physical and geologic characteristics of the deposit.
2. Ground conditions of the hanging wall, footwall, and ore zone.
3. Mining and capital costs.
4. Mining rate.
5. Availability and cost of labor.
6. Environmental considerations.

What follows is a discussion of the work that should be done at each stage of developing the deposit to choose the appropriate mining method. There is no single appropriate mining method for a deposit; there are usually two or more feasible methods. Each method entails some inherent problems. Consequently, the optimum method is that method with the least problems.

Although this chapter is aimed at determining the appropriate mining’ method, the ultimate goals are to maximize company profits, maximize recovery of the mineral, and provide a safe environment for the miners (not necessarily in this order). These ultimate goals may appear to be mutually exclusive, but in fact are not. Maximizing the profit meets the business requirement of the company. Maximizing the recovery benefits the mining company as well as society. The more tons recovered, the less capital cost per ton, which is good for the company; maximum recovery of our natural resources means society has an adequate supply. Providing safety for the miners means the work force will be able to attain the productivity called for in the feasibility study.

For additional discussion of method comparisons, see Section 16 for surface mining and Section 21 for underground mining.

23.4.2 MINING METHODS FOR CONSIDERATION

Before the discussion of the actual process of mining method selection, it is important to understand the key characteristics required for each mining method. The following briefly summarizes the mining methods. (For more detailed descriptions of these methods, refer to Sections 14, 15, 18, 19, 20, and 22.) Readers who are already familiar with the various methods can proceed to 23.4.3, Selection Method Techniques.

OPEN PIT/OPEN CAST MINING. There are surface mining techniques in which the waste material is removed to uncover the ore. This category includes strip mining and quarrying. The equipment used to extract the waste and ore is generally mechanical but differs somewhat for each of these methods. The parameters of stripping ratio and slope angle are critical to all in determining whether the method is applicable. Open pit usually refers to metal mines (Chapter 14.1), open cast or strip mining (Chapter 14.3) generally refers to coal, and quarrying (Chapter 14.2) refers to construction materials and dimension stone such as limestone and granite.

PLACER MINING. Another surface method, placer mining is distinctive in that the ore is mined and initially concentrated using gravity and water. This method requires that adequate water is available, that the material to be mined is gravel-like in character, and that the mineral to be mined can be separated based on its density differences. Examples of placer mining are dredging and hydraulicking (Chapter 15.1).

SOLUTION MINING. Solution mining requires that the mineral can be recovered using water or some type of acid, with a minimum amount of crushing and no grinding. In situ solution mining (Chapter 15.3) involves mining the ore in place. A solution is injected into the ore formation via boreholes and recovered either from boreholes or underground drifts. Borehole mining is commonly used for salt and sulfur-type deposits. Surface leaching (Chapter 15.2) is a solution method where the ore is either recovered in place in dumps or is drilled and blasted and stacked in a heap or a vat. In order for the ore to be leached, it must have the correct chemical makeup and possess adequate porosity and permeability; good fragmentation is required so that the solutions can pass through and there is sufficient surface area.

BLOCK CAVING (Chapter 20.3). An area of rock is undercut so that it caves under its own weight; the overburden waste material is expected to cave as well. Any rock can be made to cave, but it must be determined how much area is needed in order to cave, what the size of the caved blocks will be, and whether the rock will cave fast enough. Therefore, the size distribution of the caved rock and the size (plan dimensions) of the deposit are critical parameters. This category includes panel and continuous caving. Within the bounds of block caving, there is still a choice to be made as to whether to use gravity, slusher, or LHD (load-haul-dump) extraction drifts. The ground conditions and the degree of fragmentation are the primary parameters in determining which of these approaches should be used.

Recent longhole drilling has resulted in a method called *forced caving* that is similar to block caving except that widely spaced sublevels are developed from which the rock is drilled and blasted, that is, induced caving. It is applied to those deposits that may not cave for the dimensions of the deposit.

LONGWALL/SHORTWALL MINING (Chapter 20.1). The deposit, generally a coal seam, is removed in a continuous operation along an extensive working face using a long series of props over the face and the working area. The ground above the mined out area caves and results in subsidence. This method requires relatively low strength ore that is nearly flat lying and extends over a large area.

ROOM AND PILLAR/STOPE AND PILLAR MINING (Chapters 18.1 and 18.2). In these similar methods, a grid of rooms or stopes is developed, leaving pillars to support the roof or back. The difference between these two methods is the height of the zone to be mined. The rooms/stopes and pillars may or may not be uniform in size, and the pillars may or may not be recovered at a later date. The ore geometry, which is a flat-lying deposit that is not much over 150 ft (50 m) thick, is the primary controlling factor of this method. The room/stope size and pillar size are determined based on the depth and strength of the ground.

SUBLEVEL STOPING (Chapter 18.4). An overhand mining method in which the ore is blasted by longholes from sublevels.

This method differs from stope and pillar in that the deposit is so large or thick that sublevels are required and the blasting requirements approach those of open pit operations. The ore is drawn off as it is blasted, leaving an open stope. The stopes are separated by pillars. The stopes may or may not be filled after mining is completed, and if filled, the pillars are usually recovered using the same method or some type of cut and fill stoping. The parameters that control this method are an appropriate geometry and competent enough ground to leave open stopes. As in room and pillar, the stope and pillar widths are determined by rock strength vs. depth as well as the stope height.

SUBLEVEL CAVING (Chapter 20.2). Sublevel caving is an induced caving method in which the ore is blasted by ring drilling from sublevel drifts. The overlying and hanging wall rock is expected to cave as the ore is drawn. This method is chosen when the ground does not cave on its own either because of the limited width of the deposit or incompetent rock. Draw control and maintaining the brow of the face are the key concerns of this method.

SHRINKAGE STOPING (Chapter 18.3). A stoping method in which the ore is blasted, with most of it being left in the stope to accumulate until blasting is completed. The broken ore is then drawn off all at once. This method is usually used in narrow, steep deposits where the walls are not competent enough to stand without some support, which is provided by the blasted muck.

CUT AND FILL STOPING (Chapter 19.1). A stoping method in which each slice of rock is removed after blasting and replaced with some type of fill material. There are a multitude of variations on this method such as overhand cut and fill, underhand cut and fill, and post-pillar cut and fill, each of which is used under different conditions. Overhand cut and fill is used in narrow, steeply dipping deposits where the ore is moderately to highly competent, the walls are weakly to highly competent, and the cost of fill is less than the cost of leaving pillars. An underhand cut and fill is used when the ore zone is an incompetent material and the back cannot be kept open using normal ground support. Post-pillar cut and fill is an overhand method that uses a room and pillar pattern; it is used to obtain higher productivity than is possible with overhand cut and fill. The pillars are designed to support only the immediate roof.

SQUARE SET STOPING (Chapter 19.2). Prisms of timber are formed to replace the rock mined and to support the surrounding rock. Associated with this category are other timbered stoping methods such as stull stoping. This method is used in cases where the ground is incompetent and the ore geometry is too wide and/or irregular for cut and fill stoping to be applied. This method is not commonly used today because of the high cost of timber and the intensive labor required.

TOP SLICING (Chapter 19.2). This method, too, is not often used today because of the high cost of labor to lay the timber mat. It has been more or less replaced by the cheaper method of sublevel caving. It is discussed here because some of the mining selection techniques include this method as an option. Mining is done from sublevel drifts where the rock is blasted. Overlying rock is separated from the ore by a timber mat. The overburden is expected to cave.

NOVEL MINING METHODS (Section 22). These mining methods are called "novel" because the technology is different or more innovative than that used in traditional methods. Novel methods that have already been considered and are either in the research and development stage or have actually been tried include these:

1. Rapid excavation.
2. Automation and robotics.
3. Hydraulic mining.
4. Methane drainage.
5. Underground gasification and combustion.
6. Underground retorting.
7. Marine mining.
8. Nuclear mining.

Rapid Excavation—This is a method in which the drill, blast, muck, and support cycle is replaced by continuous operation. This method requires that the ground be weak enough that it can be cut at a rate that will exceed normal mining. Rapid excavation could be applied to most of the mining methods discussed and especially to mine development and access drifts. Continuous mining of coal is the forerunner to rapid excavation in rock.

Automation and Robotics—This is not really a new method, but rather a new technology that could be applied to existing mining methods. The use of automation/robotics is not really as dependent on the geological parameters and ground conditions as it is on the knowledge of the engineers and labor force.

Hydraulic Mining—This method is dependent on the ground being able to be cut by a high-pressure water jet. Transporting the material once it is cut is one of the main technological problems.

Methane Drainage—The success of this method is a function of the "physical properties of the coal seam (diffusivity, reservoir pressure, permeability, and gas content), mining method (if in progress), and drainage method" (Hartman, 1987). The method is used in Europe but only now is gaining widespread acceptance in the United States to produce coalbed methane. This method is similar to borehole mining in which the wells are used to recover the methane.

Underground Gasification—This method is applied to coal and again is related to borehole mining, in which the coal is burned at one end and the gases given off are recovered at another borehole. Use of this method is based on whether the cost of burning the coal and recovering the gases is cheaper than traditional mining. The key parameters that impact the method are the fracturing of the coal and the chemical composition of the coal. This method may become more feasible in cases where the coal seam is too narrow for traditional methods, or in recovery of multiple seams, where the second seam is too close to the first to be recovered in a traditional fashion.

Underground Retorting—This method is being tried with oil shales and tar sands. The area is mined to some extent using traditional drifting techniques and pillar designs. Then the rock in the stope (retort) is blasted in place. Using the oil, it is released from the rock and recovered under the stope. This method would be chosen based on the retorting characteristics rather than on the mining parameters. The critical factor from a mining perspective would be the cost and methodology of fracturing the ground. The degree of fragmentation will impact the percentage recovery of the oil, which is probably the most critical concern nowadays.

Marine Mining—Marine mining is much like placer mining, in which the recovery is of a mineral that has been concentrated mechanically and can be mined without drilling and blasting. From a mining point of view, the key determinant is whether the material is low enough in strength that it can be dug or vacuumed. The political problems involved will probably be greater than the technical ones.

Nuclear Mining—The main advantage of this method is the highly efficient fracturing of the rock that can be achieved. The main drawbacks are the questionable working conditions for the miners, environmental pollution concerns, and political treaty prohibitions. This technique could be used to mine competent rock in a block caving style, similar to a forced or induced caving method.

23.4.3 SELECTION METHOD TECHNIQUES

In order to determine which mining method is feasible, we need to compare the characteristics of the deposit with those required for each mining method; the method(s) that best matches should be the one(s) considered technically feasible, and should *then* be evaluated economically. The selection techniques discussed deal with the first two parameters that determine a mining method, (1) the physical and geologic characteristics of the deposit, and (2) the ground conditions of the hanging wall, footwall, and ore zone. The techniques for evaluating mining methods are only attempts at defining and quantifying in a written format what miners in years past determined through discussion, previous experience, and intuition. Therefore, each of the method selection schemes presented here is similar and yet different, reflecting personal preferences in their emphasis. The purpose of discussing these techniques is not to critique them but simply to present the alternatives available to aid in selecting the most appropriate.

Most of the schemes are aimed at determining the appropriate *underground* method, as there are so many possible choices. However, the purpose of this chapter is to discuss the selection of the *best* mining methods, including *surface,* hydraulic, and other more novel methods. The method selection process should first determine whether the deposit should be mined using a more traditional surface, underground, or in situ leach mining method. A novel method should only be considered if traditional methods are not economically or technically feasible. To start a mine with a novel mining method requires adequate funding and an enormous commitment from the board of directors to technical development; the board must also have the patience to work out the technical problems.

If the deposit cannot be mined using a surface method, then an underground method should be considered. The mining method selection techniques are limited in that selection is based solely on the known physical parameters and rock strength characteristics. Sometimes several mining methods may appear to be equally feasible. In order to further determine which method(s) is the most suitable, the input variables of mining costs, mining rate, labor availability, and environmental regulations should be considered in more detail.

Note: None of the method selection systems deal with in situ stress. The techniques do account for the vertical stress via depth, but none of the methods discuss how a high horizontal stress impacts the choice of the mining method.

23.4.3.1 Boshkov and Wright

The classification system proposed by Boshkov and Wright (1973) in the *SME Mining Engineering Handbook,* 1st ed. (based on Peele, 1941), was one of the first qualitative classification schemes developed for underground method selection (Table 23.4.1). Their system assumes that the possibility of surface and underground mining has already been eliminated. It uses general descriptions of the ore thickness, ore dip, strength of the ore, and strength of the walls to identify common methods that have been applied in similar conditions. The results of this classification provide up to four methods that may be applicable.

23.4.3.2 Hartman

Hartman (1987) has developed a flow chart selection process for defining the mining method, based on the geometry of the deposit and the ground conditions of the ore zone (Fig. 23.4.1). This system is similar to that proposed by Boshkov and Wright, but is aimed at more specific mining methods. Hartman admits the method is qualitative and should be used as a first-pass approach. This classification includes surface and underground methods, coal, and hard rock.

23.4.3.3 Morrison

The classification system proposed by Morrison (1976) divides underground mining into three basic groups: (1) rigid pillar support, (2) controlled subsidence, and (3) caving (Fig. 23.4.2). General definitions of ore width, support type, and strain energy accumulation are used as the criteria for determining a mining method. This classification helps to demonstrate the selection continuum, choosing one method over another based on the various combinations of ground conditions. In this system, the ground conditions have already been evaluated to determine the type of support required.

23.4.3.4 Laubscher

The selection of an appropriate mass underground mining method has been presented by Laubscher (1981). The selection process is based on his rock mass classification system (see Chapter 23.1, Fig. 23.1.2), which adjusts for expected mining effects on the rock mass strength. Laubscher's scheme is aimed at the mass mining methods, primarily block caving vs stoping; his main emphasis is on cavability. The two parameters that determine whether a caving system is used over a stoping system are the degree of fracturing, RQD (rock quality designation), joint spacing, and the joint rating, which is a description of the character of the joint, i.e., waviness, filling, and water conditions (Fig. 23.4.3). This scheme puts emphasis on the jointing as the only control for determining cavability. Laubscher (1990) has recently modified the classification to relate his rock mass rating to the hydraulic radius (Fig. 23.4.4). By including the hydraulic radius, cavability becomes feasible for more competent rock if the area available for undercutting is large.

23.4.3.5 Nicholas

The classification proposed by Nicholas (1981) determines feasible mining methods by numerical ranking and thus is truly quantitative. The first step is to classify the ore geometry and grade distribution using Table 23.4.2. The rock mechanics characteristics of the ore zone, hanging wall, and footwall are similarly classified using Table 23.4.3. A numerical ranking is then performed by adding up the values of each mining method, using Tables 23.4.4 and 23.4.5. The values of the tables represent the suitability of a given characteristic for a particular mining method. A value of 3 or 4 indicates that the characteristic is preferred for the mining method. A value of 1 or 2 indicates that a characteristic is probably suited to that mining method, while a value of 0 indicates that a characteristic will probably not promote the use of that mining method, although it does not rule it out entirely. A value of –49 would indicate that a characteristic will completely eliminate consideration of that method. A recent modification to the system is the weighting of the categories for the ore geometry, ore zone, hanging wall, and footwall. To give each of these categories equal weight, the ore zone, hanging wall, and footwall need to be multiplied by 1.33 ($^{16}/_{12}$). However, the importance of each category is not equal; the ore geometry is more important than the ore zone, the ore zone more important than the hanging wall, and the hanging wall more important than the footwall. The proposed weighting for each category is summarized in Table 23.4.6; this weighting can be changed based on personal experience. The net weighting is then multiplied by each of the categories. Those two or three mining methods that have the highest overall positive numerical ranking should be economically analyzed.

The proposed values for the characteristics can be changed as our technical expertise with mining equipment and mining processes improves. In addition, each individual has a different point of view as to the relative importance of the various characteristics for each method.

Table 23.4.1. Applications of Underground Mining Methods

Type of Ore Body	Dip	Strength of Ore	Strength of Walls	Commonly Applied Methods of Mining
Thin beds	Flt	Stg	Stg	Open stopes with casual pillars
				Room and pillar
				Longwall
		Wk or Stg	Wk	Longwall
Thick beds	Flt	Stg	Stg	Open stopes with casual pillars
				Room and pillar
		Wk or Stg	Wk	Top slicing
				Sublevel caving
		Wk or Stg	Stg	Underground glory hole
Very thick beds				Same as for masses
Very narrow veins	Stp	Stg or Wk	Stg or Wk	Resuing
Narrow veins	Flt			Same as for thin beds
(widths up to	Stp	Stg	Stg	Open stopes
economic				Shrinkage stopes
length of stull)				Cut and fill stopes
			Wk	Cut and fill stopes
				Square set stopes
		Wk	Stg	Open underhand stopes
				Square set stopes
			Wk	Top slicing
				Square set stopes
Wide veins	Flt			Same as for thick beds or masses
	Stp	Stg	Stg	Open underhand stopes
				Underground glory hole
				Shrinkage stopes
				Sublevel stoping
				Cut and fill stopes
				Combined methods
			Wk	Cut and fill stopes
				Top slicing
				Sublevel caving
				Square set stope
				Combined methods
		Wk	Stg	Open underhand stopes
				Top slicing
				Sublevel caving
				Block caving
				Square set stopes
				Combined methods
			Wk	Top slicing
				Sublevel caving
				Square set stopes
				Combined methods
Masses		Stg	Stg	Underground glory hole
				Shrinkage stopes
				Sublevel stoping
				Cut and fill
				Combined methods
		Wk	Wk or Stg	Top slicing
				Sublevel caving
				Block caving
				Square set stopes
				Combined methods

Wk = weak; stg = strong; flt = flat; stp = steep.
Source: Boshkov and Wright, 1973 (permission: Society for Mining, Metallurgy, and Exploration, Inc.).

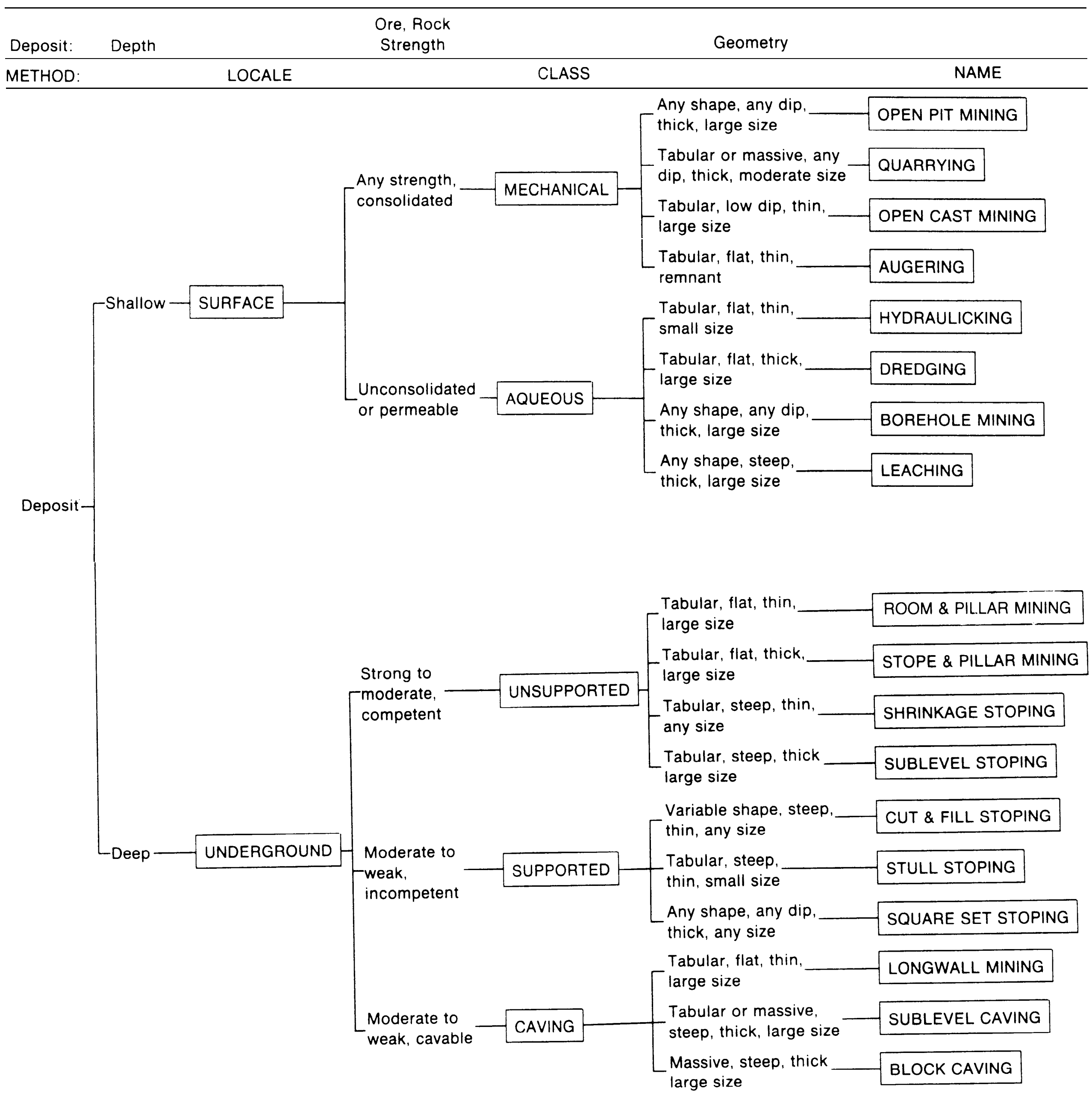


Fig. 23.4.1. Hartman's chart for selection of a mining method (Hartman, 1987).

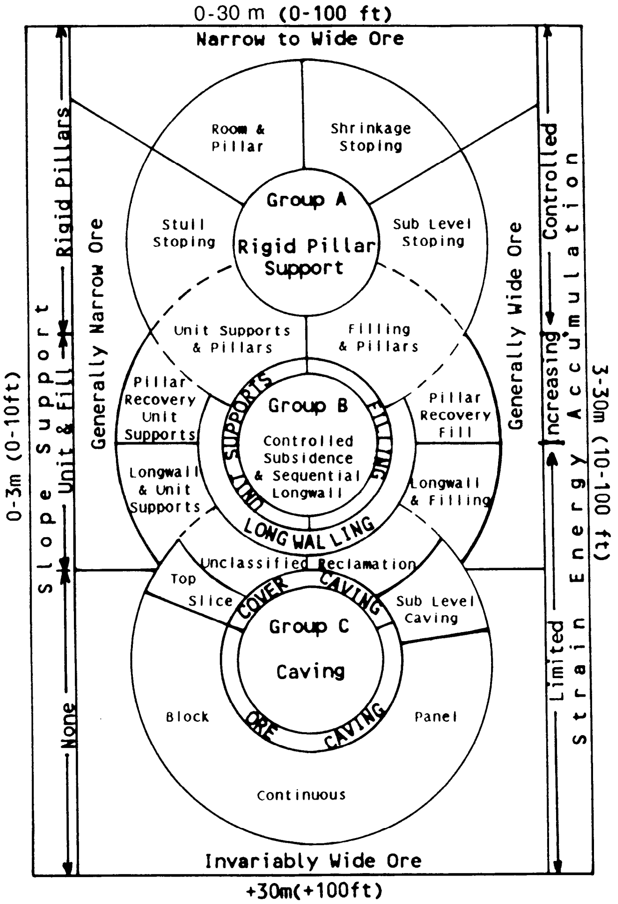


Fig. 23.4.2. Morrison's chart for selection of a mining method (Morrison, 1976).

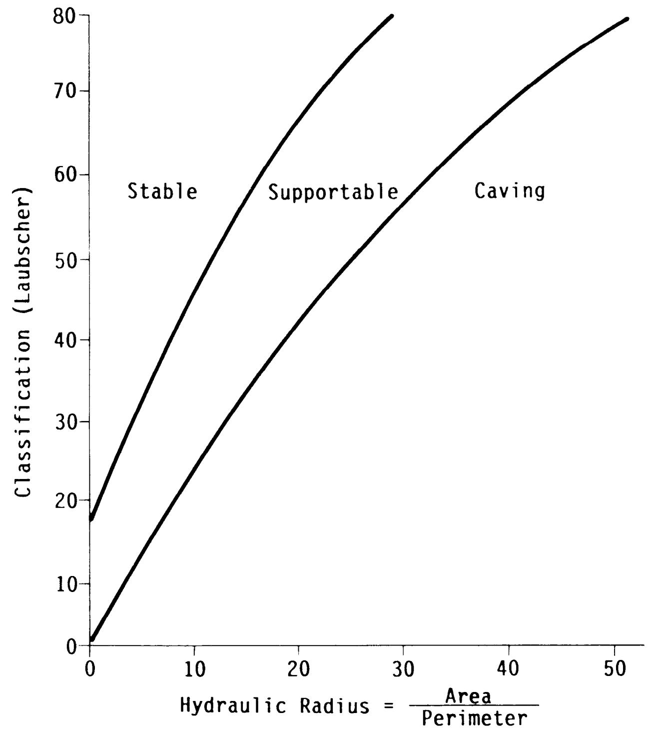


Fig. 23.4.4. Laubscher's cavability based on hydraulic radius and classification.

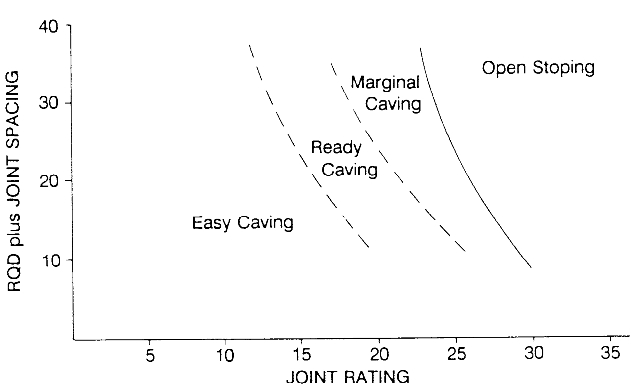


Fig. 23.4.3. Laubscher's 1981 classification for cavability evaluation.

Table 23.4.2. Definition of Deposit Geometry and Grade Distribution

1) *General Shape/Width*	
equi-dimensional	all dimensions are on the same order of magnitude
platy—tabular	two dimensions are many times the thickness, which does not usually exceed 325 ft (100 m)
irregular	dimensions vary over short distances
2) *Ore Thickness*	
narrow	<30 ft (<10 m)
intermediate	30–100 ft (10–30 m)
thick	100–325 ft (30–100 m)
very thick	>325 ft (>100 m)
3) *Plunge*	
flat	<20°
intermediate	20°–55°
steep	>55°
4) *Depth Below Surface*	
provide actual depth	
5) *Grade Distribution*	
Uniform. The grade at any point in the deposit does not vary significantly from the mean grade for that deposit.	
Gradational. Grade values have zonal characteristics, and the grades change gradually from one to another.	
Erratic. Grade values change radically over short distances and do not exhibit any discernible pattern in their changes.	

Source: Nicholas, 1981.

23.4.3.6 Example of Use of Selection Schemes

The following is an example and description of how to use the various classification schemes. The data presented are fabricated and aimed at being suitable input for Nicholas' scheme, but can be used for the other methods as well.

Input data	Description
General Deposit Shape	tabular or platy
Plan width	330 ft (100 m)
Ore thickness	300 ft (90 m)
Ore plunge	15°
Grade distribution	primarily disseminated
Depth	1150 ft (350 m)
Ore Zone	
Rock substance strength	12,000 psi (83 MPa)
Fracture spacing	3 frac/ft (10 frac/m)
RQD	40
Fracture shear strength	clean joint with a rough surface
Hanging Wall	
Rock substance strength	16,000 psi (110 MPa)
Fracture spacing	1.2 frac/ft (4 frac/m)
RQD	65
Fracture shear strength	clean joint with a rough surface
Footwall	
Rock substance strength	7,000 psi (48 MPa)
Fracture spacing	4.3 frac/ft (14 frac/m)
RQD	40
Fracture shear strength	joints coated with thin clay

Boshkov and Wright's method would classify this deposit as either "very thick beds" or "massive," with a "weak" ore and a "weak/strong" wall rock (Table 23.4.1). Based on this classification, feasible mining methods include top slicing, sublevel caving, block caving, square set stoping, and combined methods.

Hartman's method selection chart would show this deposit as either shallow or deep in depth. The ore strength would be moderate to weak, and the ore geometry would be tabular and massive, or thick (Fig. 23.4.1). The possible mining methods would be open pit mining, sublevel caving, and block caving.

Table 23.4.3. Rock Mechanics Characteristics

1) *Rock Substance Strength* (uniaxial strength/overburden pressure)		
weak	<8	
moderate	8–15	
strong	>15	
2) *Fracture Frequency*	No. of Fractures per (ft) (m)	% RQD
very close	>5 >16	0– 20
close	3–5 10–16	20– 40
wide	1–3 3–10	40– 70
very wide	<1 <3	70–100
3) *Fracture Shear Strength*		
weak	clean joint with a smooth surface or fill with material with strength less than rock substance strength	
moderate	clean joint with rough surface	
strong	joint is filled with a material that is equal to or stronger than rock substance strength	

Sources: Nicholas, 1981.

Morrison's method would classify this deposit as "invariably wide." Based on this, the mining methods possible are a caving method, top slicing, and sublevel caving.

Laubscher's system requires more information than that provided, but a guesstimate can be made from the data given. In actuality, one would have looked at the drill core and could therefore make the necessary measurements. Using Laubscher's classification (Fig. 23.1.2), the rating of the rock is 48, as follows:

RQD	6
Intact rock strength	8
Joint spacing	5
Joint condition	29
TOTAL	48

Using his first method selection, which is based primarily

Table 23.4.4. Ranking of Geometry/Grade Distribution for Different Mining Methods

	General Shape			Ore Thickness				Ore Plunge			Grade Distribution		
Mining Method	M	T/P	I	N	I	T	VT	F	I	S	U	G	E
Open Pit Mining	3	2	3	2	3	4	4	3	3	4	3	3	3
Block Caving	4	2	0	−49	0	2	4	3	2	4	4	2	0
Sublevel Stoping	2	2	1	1	2	4	3	2	1	4	3	3	1
Sublevel Caving	3	4	1	−49	0	4	4	1	1	4	4	2	0
Longwall Mining	−49	4	−49	4	0	−49	−49	4	0	−49	4	2	0
Room and Pillar Mining	0	4	2	4	2	−49	−49	4	1	0	3	3	3
Shrinkage Stoping	2	2	1	1	2	4	3	2	1	4	3	2	1
Cut and Fill Stoping	0	4	2	4	4	0	0	0	3	4	3	3	3
Top Slicing	3	3	0	−49	0	3	4	4	1	2	4	2	0
Square Set Stoping	0	2	4	4	4	1	1	2	3	3	3	3	3

M = Massive
T/P = Tabular or Platy
I = Irregular

N = Narrow
I = Intermediate
T = Thick
VT = Very Thick

F = Flat
I = Intermediate
S = Steep

U = Uniform
G = Gradational
E = Erratic

Table 23.4.5. Ranking of Rock Mechanics Characteristics for Different Mining Methods

Key:

Rock Substance Strength
W = Weak
M = Moderate
S = Strong

Fracture Spacing
VC = Very Close
C = Close
W = Wide
VW = Very Wide

Fracture Strength
W = Weak
M = Moderate
S = Strong

5a: Ore Zone

Mining Method	Rock Substance Strength			Fracture Spacing				Fracture Strength		
	W	M	S	VC	C	W	VW	W	M	S
Open Pit Mining	3	4	4	2	3	4	4	2	3	4
Block Caving	4	1	1	4	4	3	0	4	3	0
Sublevel Stoping	−49	3	4	0	0	1	4	0	2	4
Sublevel Caving	0	3	3	0	2	4	4	0	2	2
Longwall Mining	4	1	0	4	4	0	0	4	3	0
Room and Pillar Mining	0	3	4	0	1	2	4	0	2	4
Shrinkage Stoping	1	3	4	0	1	3	4	0	2	4
Cut and Fill Stoping	3	2	2	3	3	2	2	3	3	2
Top Slicing	2	3	3	1	1	2	4	1	2	4
Square Set Stoping	4	1	1	4	4	2	1	4	3	2

5b: Hanging Wall

Mining Method	Rock Substance Strength			Fracture Spacing				Fracture Strength		
	W	M	S	VC	C	W	VW	W	M	S
Open Pit Mining	3	4	4	2	3	4	4	2	3	4
Block Caving	4	2	1	3	4	3	0	4	2	0
Sublevel Stoping	−49	3	4	−49	0	1	4	0	2	4
Sublevel Caving	3	2	1	3	4	3	1	4	2	0
Longwall Mining	4	2	0	4	4	3	0	4	2	0
Room and Pillar Mining	0	3	4	0	1	2	4	0	2	4
Shrinkage Stoping	4	2	1	4	4	3	0	4	2	0
Cut and Fill Stoping	3	2	2	3	3	2	2	4	3	2
Top Slicing	4	2	1	3	3	3	0	4	2	0
Square Set Stoping	3	2	2	3	3	2	2	4	3	2

5c: Footwall

Mining Method	Rock Substance Strength			Fracture Spacing				Fracture Strength		
	W	M	S	VC	C	W	VW	W	M	S
Open Pit Mining	3	4	4	2	3	4	4	2	3	4
Block Caving	2	3	3	1	3	3	3	1	3	3
Sublevel Stoping	0	2	4	0	0	2	4	0	1	4
Sublevel Caving	0	2	4	0	1	3	4	0	2	4
Longwall Mining	2	3	3	1	2	4	3	1	3	3
Room and Pillar Mining	0	2	4	0	1	3	3	0	3	3
Shrinkage Stoping	2	3	3	2	3	3	2	2	2	3
Cut and Fill Stoping	4	2	2	4	4	2	2	4	4	2
Top Slicing	2	3	3	1	3	3	3	1	2	3
Square Set Stoping	4	2	2	4	4	2	2	4	4	2

Table 23.4.6. Weighting Factors

Ore Geometry	1.0	1.0	1.0
Ore Zone Ground Conditions	1.33	0.75	1.0
Hanging Wall Ground Conditions	1.33	0.6	0.8
Footwall Ground Conditions	1.33	0.38	0.5

on jointing, the ground would be considered either "marginal caving" or open stoping" (Fig. 23.4.3). The newer selection scheme, which uses the total mass rating and the hydraulic radius, indicates that a hydraulic radius of 28 is required for the deposit to cave (Fig. 23.4.4). A hydraulic radius of 28 is equivalent to a square area of 370 ft (112 m) or an area of 330 by 410 ft (100 by 125 m). This means that for our example, the deposit

Table 23.4.7. Example of Numerical Model Selection Process

Geometry/Grade Distribution	(Column 1)	(Column 2) open pit	(Column 3) block caving etc.
		(values from Table 23.4.4)	
General shape	tabular or platy	2	2
Ore thickness	thick	4	2
Ore plunge	flat	3	3
Grade distribution	uniform	3	4
Depth (used later)	425 ft (130 m)	- -	- -
		12	11
Rock Mechanics Characteristics		(values from Table 23.4.5)	
Ore Zone			
Rock substance strength	strong	4	1
Fracture spacing	close	2	4
Fracture strength	moderate	3	3
		9	8
Hanging Wall			
Rock substance strength	strong	4	1
Fracture spacing	wide	4	3
Fracture strength	moderate	3	2
		11	6
Footwall			
Rock substance strength	moderate	4	3
Fracture spacing	close	2	3
Fracture strength	weak	2	1
		8	7

Characteristic Values Totaled for Different Mining Methods

Mining Method	Geometry/Grade Distribution	Rock Mechanics Characteristics				Grand Total
		Ore	HW	FW	Total	
Open Pit Mining	12	9	11	8	28	
Block Caving	11	8	6	7	21	
Sublevel Stoping	11	5	7	2	14	
Sublevel Caving	13	7	6	3	16	
Longwall Mining	−37	8	5	6	19	
Room and Pillar Mining	−38	7	8	3	18	
Shrinkage Stoping	11	6	6	8	20	
Cut and Fill Stoping	7	8	7	10	25	
Top Slicing	14	6	6	7	19	
Square Set Stoping	8	8	7	10	25	

must be undercut full width, and for a length greater than the width. This may mean that the production schedule will have to allow for additional time to undercut the large area or for some type of inducement. Either that or a sublevel cave operation may be used instead.

Nicholas' example is summarized in Table 23.4.7. The first step is to list the geometry/grade distribution and rock mechanics characteristics of the deposit (Table 23.4.7, column 1). The columns of characteristics in Tables 23.4.5 and 23.4.6 are then identified for the deposit, and the values summed for the geometry/grade distribution, ore zone rock mechanics, hanging wall rock mechanics, and footwall rock mechanics for each mining method (Table 23.4.7, columns 2 and 3).

The ore zone, hanging wall, and footwall should be multiplied by the weighting factors and then summed. This total should then be added to the geometry/grade distribution sum (Table 23.4.8). Using this type of characteristic grouping, one can see which grouping(s) reduce the chance of using a particular mining method, or, in cases where the total sum is nearly equal, one can determine which characteristics are the most suitable for the mining method. For the example given, the preferred mining methods would be open pit mining, top slicing, square set stoping, block caving, and sublevel caving.

All of the method selection techniques arrive at similar conclusions (Table 23.4.9). One might surmise that the conclusions were obvious from the beginning. This may be true for this example but is not so in all cases. Important factors may be overlooked; using a method selection scheme, one is forced to consider all of the parameters. As is shown by Nicholas' scheme, in which the main drawback to using sublevel caving is the poor ground conditions of the footwall, one unfavorable factor does not necessarily eliminate a method. Some technical problems can be overcome. Sublevel caving requires extensive drifting and ore pass work in the footwall, so the poor ground conditions will call for additional ground support, meaning development may take longer. This does not rule out the feasibility of the method;

Table 23.4.8. Characteristic Values Multiplied by Weighting Factors

Mining Method	Geometry/Grade Distribution	Rock Mechanics Characteristics				Grand Total
			HW	FW	Total	
Open Pit Mining	12	9	8.8	4.0	21.8	33.8
Block Caving	11	8	4.8	3.5	16.3	27.3
Sublevel Stoping	11	5	5.6	1.0	11.6	22.6
Sublevel Caving	13	7	4.8	1.5	13.3	26.3
Longwall Mining	−37	8	4.0	3.0	15.0	−22.0
Room and Pillar Mining	−38	7	6.4	1.5	14.9	−23.1
Shrinkage Stoping	11	6	4.8	4.0	14.8	25.8
Cut and Fill Stoping	7	8	5.6	5.0	18.6	25.6
Top Slicing	14	6	4.8	3.5	14.3	28.3
Square Set Stoping	8	8	5.6	5.0	18.6	26.6

Table 23.4.9. Summary of Methods Selected

	Boshkov and Wright	Hartman	Morrison	Laubscher	Nicholas
Open Pit Mining		X			1
Block Caving	X	X	X	X	3
Sublevel Caving	X	X	X		5
Top Slicing	X		X		2
Square Set Stoping	X				4
Cut and Fill Stoping					6

Table 23.4.10. Ranking of Mining Methods Based on Relative Operating Costs

Mining Method	Ranking Cost (Percent)
a. Hartman (1987) Ranking	
Hydraulicking, Dredging, Leaching	5
Open Pit/Open Cast Mining	10
Block Caving, Longwall Mining	20
Room and Pillar Mining	30
Stope and Pillar Mining	
Sublevel Stoping	40
Shrinkage Stoping, Sublevel Caving, Induced Caving	50
Cut and Fill Stoping	60
Square Set Stoping	100
Source: Hartman, 1987.	
b. Morrison (1976) Ranking	
Open Pit Mining	Lowest
Block Caving	
Sublevel Stoping	
Sublevel Caving	
Longwall Mining	
Room and Pillar Mining	
Shrinkage Stoping	
Cut and Fill Stoping	
Top Slicing	
Square Set Stoping	Highest

it simply means that when performing the cost analysis, one should allow for a higher cost in the footwall development work.

None of the method selection systems deal with in situ stress. The techniques account for the vertical stress via depth, but none of the methods discuss how a high horizontal stress impacts the choice of the mining method.

Although the mining methods resulting from the selection process are all technically feasible, their mining costs may be significantly different. Based on Hartman's and Morrison's (Table 23.4.10) relative rankings of mining costs, those methods with the potentially lowest operating cost can be identified. However, these cost rankings represent averages, and the estimated cost provided the method is *appropriate* for the ground conditions; therefore the rankings should be used with caution.

On the basis of relative operating costs, the methods would be ranked as follows:

1. Open pit mining.
2. Block caving.
3. Sublevel caving.
4. Top slicing.
5. Square set stoping.

With this ranking established, we must next determine whether the deposit can be mined from the surface or if it must be mined underground.

23.4.4 STAGES IN THE SELECTION PROCESS

The mining method selection process is iterative, with each iteration correlating to the stages of development of the property. A deposit that is mined will go through roughly five stages:

1. Initial discovery of the mineralized zone.
2. Developmental drilling to outline reserves and a feasibility study.
3. Design and development of the deposit.
4. Revising and updating during production.
5. End of mine life.

The preceding section described how the key physical parameters of the deposit and the ground conditions are used to determine a feasible mining method. These techniques can be used in each iteration; what will change is the database, the degree of certainty in the economic analysis, and the consideration of the other parameters that determine the mining method such as costs, labor, and environmental regulations.

23.4.4.1 Stage 1: Initial Discovery

The exploration for a deposit is guided by corporate philosophy. The corporation should provide the exploration department with the rough target size and grade. (Exploration techniques are discussed in Section 4, Prospecting and Exploration.) Although the exploration geologist is trained in the required exploration techniques, he/she may not understand the economics behind the search for a particular mineral(s) with a minimum grade and/or tonnage. The basic economics should be discussed with the geologist so that he/she understands how the target was defined. What is found may not match the target exactly, but he/she will be able to assess how to change the exploration program to meet the corporate needs. This will also open the lines of communication between the geologist and the mining engineers for future mine planning.

As soon as a mineralized zone is discovered, an initial prediction of potential mining methods should be made. Primary considerations for the method choice should be the expected ore geometry and ground conditions, combined with engineering judgment and comparisons to nearby operations. The use of Boshkov and Wright's and/or Hartman's system is reasonable during this stage. Using these techniques should help identify which characteristics are critical for the mining method, so that the necessary data can be collected during the development drilling. The other method selection techniques could be used but the database would probably not be adequate.

In order to plan a better development drilling program, the possible mining methods should be costed out for capital and operating expenses at various production rates so that the size and grade of the target can be better defined in order to identify drilling limits.

23.4.4.2 Stage 2: Development Drilling and Feasibility

In stage 2, development drilling is performed to determine the tonnage and grade of the deposit as well as for the purpose of collecting metallurgical, geotechnical, hydrologic, and environmental data to determine mineral processing and mining methods. During or near the end of the development drilling program, a quantitative analysis of potential mining methods can be performed. This is usually a conceptual mine plan or a feasibility study.

The development drilling program should be designed to define the target size and/or the limits of the deposit. Therefore, the drill spacing is wide and a number of the holes are outside of the minable limits. The holes outside the deposit provide the rock mechanician with information on the area where the pit walls or access facilities will be. The geologist must find out from the metallurgist, rock mechanician, and mining engineer what data they each require from the drill core or cuttings. The drill pattern, drilling method, and logging techniques have to be determined based on the particular type of deposit; this is further discussed by Call (1979), and Peters (1978).

By stage 2, the database should be adequate to use Laubscher's or Nicholas' method selection technique. For a first pass, the more traditional mining methods should be considered. If conditions warrant investigation of more novel methods, then they should be considered after the traditional methods have been eliminated. As shown previously, a deposit may be mined either from the surface or underground. Chapter 23.2, Surface vs. Underground Methods, discusses in detail the process of choosing between these two. In general, if at all possible, a surface method should be chosen over an underground method, as surface mining involves fewer technical problems and generally yields more efficient productivities than underground methods. The decision between a surface and an underground method is usually an economic one, although environmental regulations may require the use of an underground method, even when it is less economic. Fig. 23.4.5 provides a rough guide as to when an underground method would be preferable to a surface method, based on mineral value and stripping ratio.

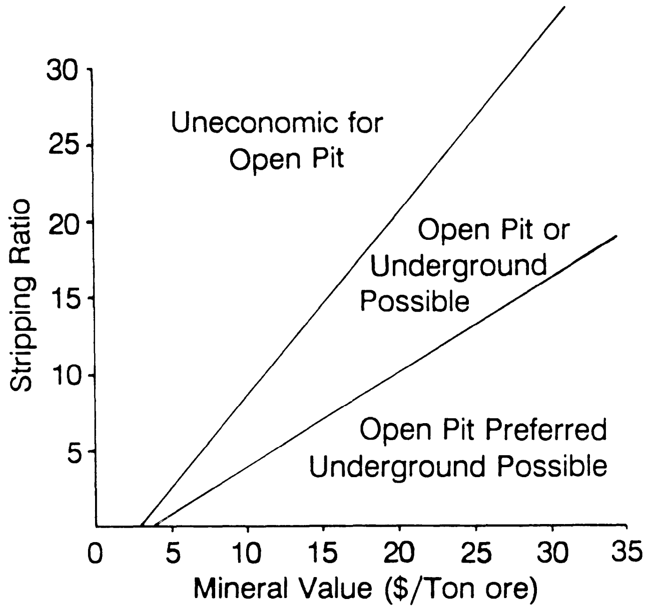


Fig. 23.4.5. Stripping ratio vs. mineral value.

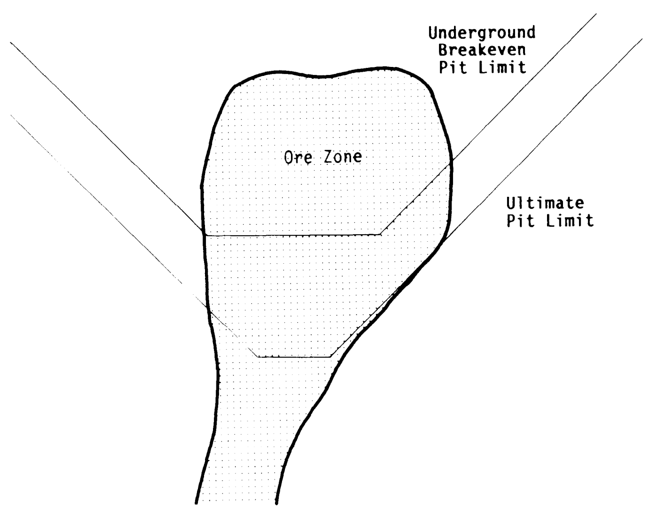


Fig. 23.4.6. Ultimate pit limit vs. underground breakeven pit limit.

A situation that is becoming more common is for the deposit to be mined as an open pit initially, recovering the deeper reserves later using an underground method. One of the key factors in planning this sequence of mining is to make sure there are adequate reserves *after* the open pit is mined to pay for the capital required to go underground. This can be done by mining the pit to the underground/open pit "breakeven" point rather than to the ultimate pit limits (Fig. 23.4.6).

If an underground method appears to be preferable, then one of the method selection techniques discussed in 23.4.3 should be used to provide two or three possible mining methods. These methods should then be economically analyzed to determine which method nets the best return on investment. The input

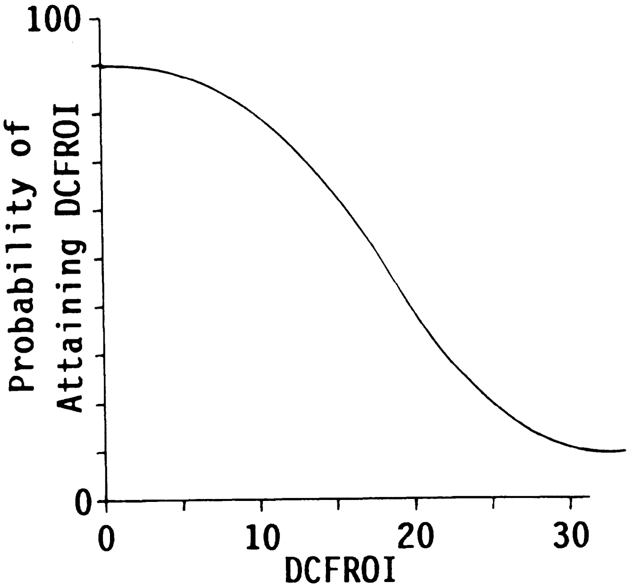


Fig. 23.4.7. Probability of attaining DCFROI.

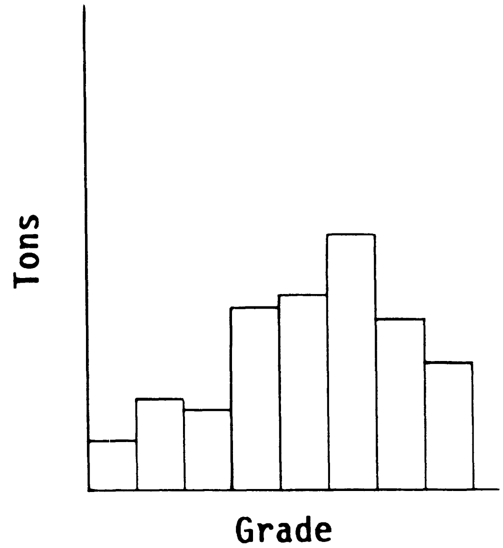
parameters for this cost analysis require some rough mining dimensions. These dimensions can be based on either the classification work or the judgment of the engineers doing the analysis. The key parameters for some of the main mining methods are discussed in 23.4.4.3.

The remaining parameters should be considered and the pros and cons of the feasible methods should be weighed based on the mining rate required by the corporation, the availability and type of labor, and the environmental regulations/considerations.

23.4.4.3 Stage 3: Design

In stage 3, the design stage, the mine plans for the most feasible method, as determined in stage 2, are evaluated. The calculation of minable reserves and determination of cutoff grades allows a detailed economic analysis to establish capitalization requirements and return on investment. With this information, final decisions on appropriate mining methods can be made. While this level of detailed planning on paper may appear to extend start-up time, it will always be more cost effective to discover mistakes on paper than to find errors in the field after mining has already begun. Still, a project can be evaluated "to death" or beyond the limits of the data. The evaluation needs to be based on an understanding of the variability of conditions and the quantity and quality of data. Mining is a risky business. Bankers understand risk analyses, and evaluations should be done in their language, presenting *distributions* of possible outcomes. As long as the probability of attaining a certain minimum DCFROI (discounted cash flow return on investment) (Fig. 23.4.7) is met, the property will be considered a viable proposition (Chapter 6.5). The probability required is a function of the corporate philosophy. The point here is that the accuracy of the design should not be any greater than the accuracy of the ore reserve or the mine design.

MINABLE RESERVES. The minable reserve defines what ore is recoverable for the possible mining methods. These reserves are the result of defining the mineral inventory, cutoff grade, and limitations imposed by ore thickness, lateral extent, or depth. The mineral inventory is the result of the drillhole data and geologic interpretations. Numerous methods exist to interpolate and extrapolate the ore grade from the drillholes, such as kriging, inverse square, and assigning the grade to an area of influence (see Chapter 5.6). Common methods used to calculate the mineral inventory include polygons, triangles, cross sections, and block models.



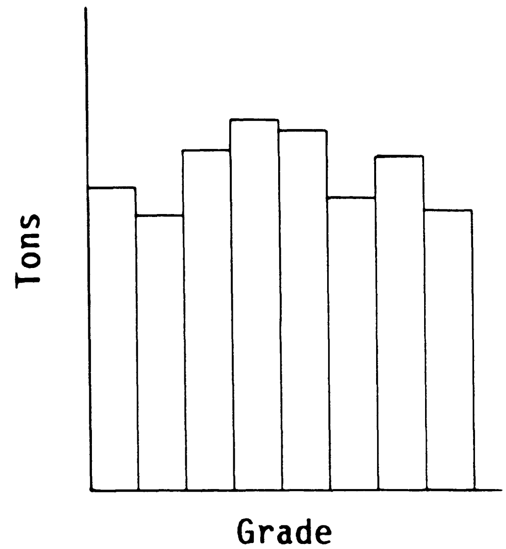


Fig. 23.4.8. Tons vs. grade frequency plot.

Determining the cutoff grade is an iterative process and can have a major impact on the minable reserves. The primary inputs required to determine the cutoff grade are the capital and operating costs. This information is not always available or accurate in the early phases of defining the minable reserve. The geologist should generate a grade and tonnage distribution to determine whether there is a natural grade cutoff, that is, a point at which mining below a certain grade does not add significantly to the tonnage (Fig. 23.4.8). If this is not the case, then estimates of operating and capital costs will have to be made. There are a number of philosophies on whether capital should be included in the cutoff grade (Marek and Welhener, 1985). Initially, cutoff grade should be determined with and without the capital.

CRITICAL DESIGN PARAMETERS. For each of the mining methods, several critical design parameters must be evaluated as part of the method selection process in order to make the economic analysis.

Open Pit Mining—The critical design parameter for the selection of an open pit mining method is the slope angle. This is because the slope angle and limit of the ore zone will permit the estimation of the stripping ratio, which will ultimately impact the economics of the selection (Soderberg and Rausch, 1968). An assessment of the final slope angles can be made by defining potential failure geometries from the orientation of the geologic structures and then choosing a slope angle that minimizes the number of daylighted structures. If structure, shear strength, length, and spacing are available, a stability analysis can also be performed. Chapter 10.4, Slope Stability, discusses in more detail the approaches to determining slope design.

Solution Mining—The critical parameter for solution mining is the recovery of the mineral. That is, the metallurgical constraints are more critical than the mining constraints. For in situ or surface leaching, the permeability and porosity are the limiting factors; these parameters can be improved by increased fragmentation from blasting. In borehole mining, design of the pump sizes and hole spacings are required. Vat or heap leaching is also controlled by fragment size distribution. The design of the crusher size and/or blasting and fracture spacing determine ultimate fragment size distribution.

Block Caving—The cavability of the deposit should be examined in greater detail than in stage 2. Once cavability is determined, the minimum drawpoint spacing, supportable drift size, and subsidence limit should also be defined. Any ground can be caved; cavability is a matter of how much area has to be undercut to sustain the cave and what the fragment size distribution of the caved muck will be. If the area of the deposit is too small to sustain the cave, then some type of inducement will be required.

Using Laubscher's classification or the pressure arch concept (Alder, Potts, and Walker, 1951), the undercut width required to sustain a cave can be estimated. Laubscher provides the hydraulic radius required based on his classification system. In the pressure arch concept, the rock is considered to have a maximum distance that it can transfer load. Although each deposit has its own transfer distance, a correlation between depth and maximum transfer distance has been determined (Fig. 23.4.9). Based on the pressure arch concept, if the undercut width does not exceed twice the maximum transfer distance, only the rock under the arch has the potential for caving. If the deposit is too small to sustain the cave, then some form of inducement, such as longhole blasting from sublevels or boundary weakening, may be needed.

Drawpoint spacing is primarily a function of the ore, along with overlying waste fragment size distribution and pillar strength. The smaller the fragment size, the closer the drawpoint spacing must be to minimize dilution. If the ore can be mined uniformly, the impact of the fragment size will be minimized. The rock mass between the drawpoints can be considered a pillar and used to determine the minimum drawpoint spacing. Loading on the pillar is difficult to determine, but estimates by Kendorski (1975) or Panek (1978) can be used in conjunction with a pillar analysis (Wilson, 1972) to determine load-carrying capacity. With the fragmentation curve and pillar design, a comparison of drawpoint spacing with existing operations can be made (Fig. 23.4.10).

The ore-gathering drift size and support required are important in estimating the cost of the mining method. Equipment selection may influence this decision, but the characteristics of the rock mass will indicate what is feasible and what is not. Use of the classification and support approaches (Barton, Lien, and Lunde, 1974) may provide some estimates of support requirements.

The subsidence limit should be defined for locating buildings and shafts that are to last the life of the deposit. In the absence of any major geologic structure, a 45° angle projected from the bottom of the ore zone is usually considered the closest to the deposit that one should locate long-term facilities. However, most actual ground movement takes place within a 60° angle from the deposit, unless a major fault exists which will control the limit of subsidence.

Longwall/Shortwall Mining—The critical parameters that must be analyzed in the feasibility of a longwall mining system are the expected stress field in the mining zone, the size of entry and barrier pillars, and the support of the entries. Brady and Brown (1985) emphasize how understanding the stress field will have an impact on all aspects of the mine plan (also see Chapter 10.5). The length of the face, advance rate, face support requirements, and behavior of the caving waste will all be influenced by the nature of the developing stress field. By understanding this stress field, it becomes possible to design the entry size, support requirements, and pillar sizes.

Room and Pillar/Stope and Pillar Mining—The three critical parameters in a stoping or room and pillar method are the width of the stopes or openings, the pillar size, and backfill requirements. In sublevel stoping, the width of the stope is a function of the immediate and intermediate roof (Alder, Potts, and Walker, 1951). The intermediate roof is characterized by the pressure arch concept. The maximum mining width without a barrier pillar is twice the maximum pressure arch. Barrier pillars spaced at twice the arch distance must be able to carry the tributary area load to the surface. The immediate roof is the ground under the pressure arch which will act as a beam, plate, or arch. Joint orientation, spacing, and length can be used to define the stope width. The pillars within the arch have to carry this rock mass only (Fig. 23.4.11).

Backfill is not commonly used in room and pillar mining because the pillars are designed to carry the load. However, if a large area is being mined, backfill could be used to minimize a failure area or to prevent the pillars from deteriorating with time.

Sublevel Stoping—The critical design parameters are similar to those for stope and pillar, except that the design must consider what the sublevel interval should be, if the stopes are to be filled, and whether sill pillars are needed. The sublevel interval is determined by the height of the ore and the pillar width. Backfilling is implemented so that pillars can be recovered, or to reduce the impact of a failed pillar; backfill also helps to minimize the required pillar width. A sill pillar is necessary in cases where the pillar width required is large, the sublevel interval is greater than 170 ft (50 m), or long-term access of the sublevel is required. Whenever possible, rock fill should be used, as it makes a solid, stiff fill. Tailings can be used for fill as long as they can be dewatered, preferably before they enter the stope. The addition of cement reduces dilution while mining the pillar. Adding cement probably does not improve the load-carrying capacity of the fill unless it is dewatered prior to placement.

Sublevel Caving—The critical parameters for sublevel caving are the cavability of the hanging wall, the sublevel drift size and required support, and the spacing between the sublevel drifts. Janelid and Kvapil (1966) and Kvapil (1982) have presented guidelines for the layout of a sublevel mine. The hanging wall must come in behind the ore zone; otherwise sublevel caving will not work. Using analyses similar to those used for block caving will provide an estimate of the dimensions needed to initiate the cave, as well as the impact of the fragment size distribution.

Vertical spacing of the drifts is mainly a function of equipment, but the horizontal spacing between drifts is determined by the width of the draw ellipsoid and the stability of the rock. Janelid and Kvapil related drift spacing to the distance between

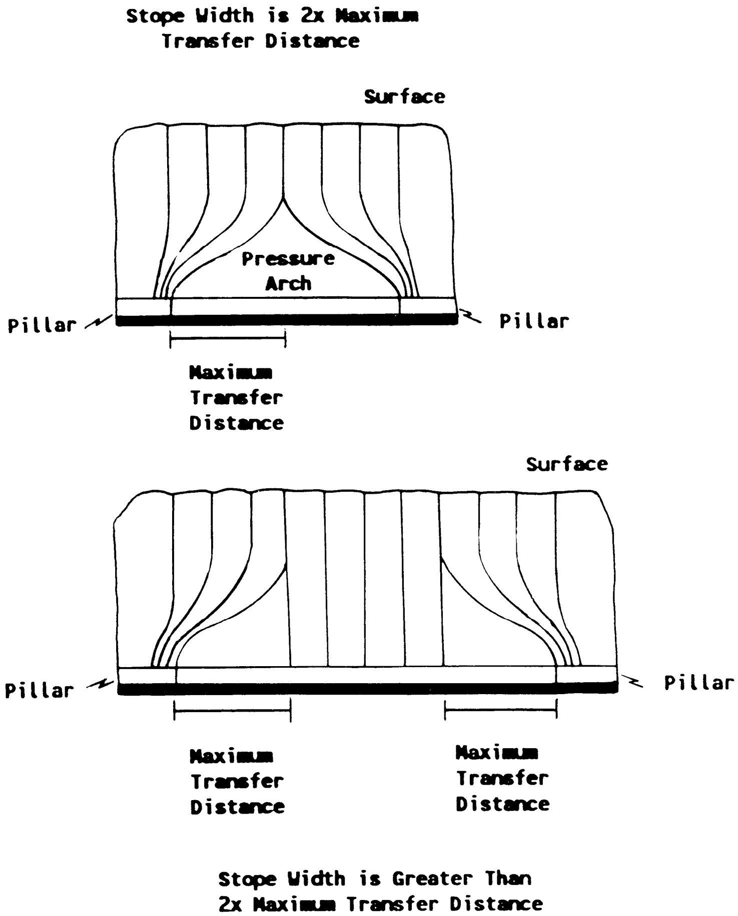


Fig. 23.4.9. Pressure arch concept.

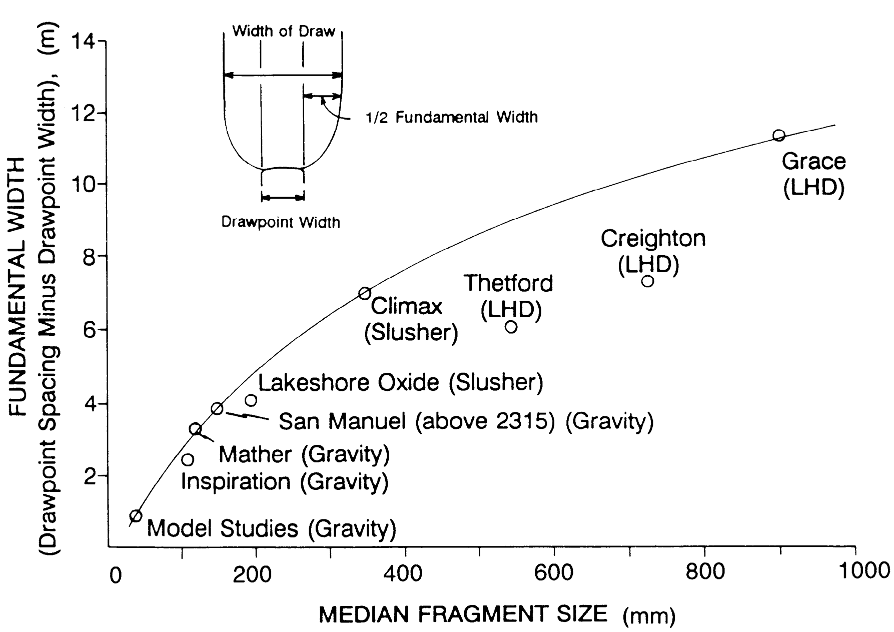


Fig. 23.4.10. Fundamental width vs. fragment size. Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

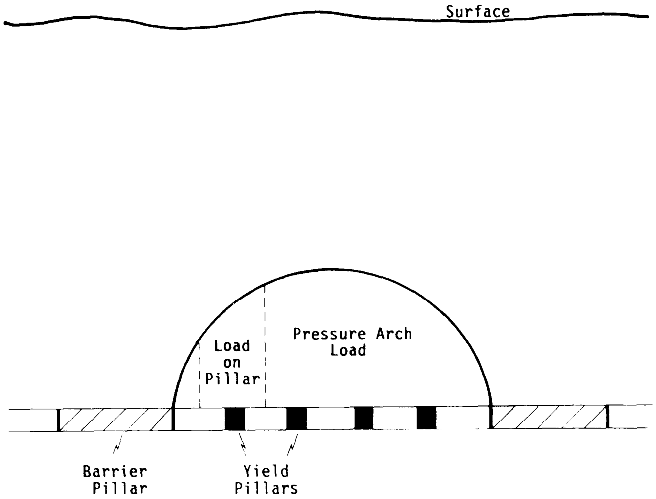


Fig. 23.4.11. Load on yield pillars.

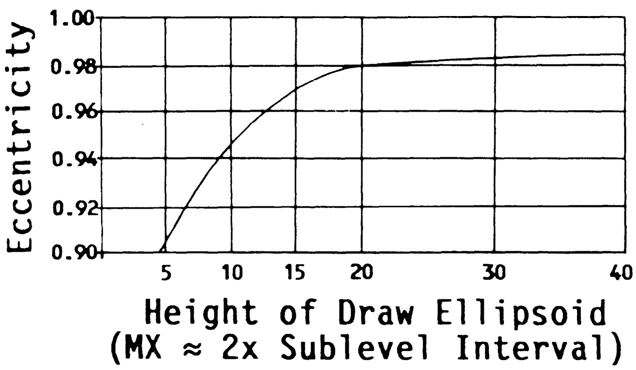


Fig. 23.4.12. Eccentricity vs. height of draw ellipsoid.

sublevels and the eccentricity of the ellipsoid (Fig. 23.4.12). The rock masses between the drifts can be considered pillars and analyzed as such.

Shrinkage Stoping—For shrinkage stoping, the same type of analysis must be made, but the cavability of the overlying rock must be evaluated as well. Support requirements can be estimated, as discussed under block caving.

The considerations for designing a room and pillar mine are basically identical to those made when designing a sublevel stoping method.

Cut and Fill Stoping—The critical parameters to consider in both overhand and underhand approaches to cut and fill stoping are the rock mass conditions in the ore zone, hanging wall, and footwall. MacMillan and Ferguson (1982) have presented guidelines for cut and fill mine planning.

For a very weak ore zone, it is necessary to design an artificial back in order to perform the mining process. If the ore zone is relatively competent, overhand mining must be designed to take into account the maximum stable exposure of the hanging wall. In both cases, the joint characteristics and stress field will play a critical role in predicting the rock's behavior.

In some cases, it may be necessary to leave pillars behind to aid in supporting the ore zone. Pillars used in cut and fill stoping should be designed on a yield concept to permit transfer of loads to the backfill material. The backfill material must be designed to be of sufficient density or stiffness to carry transferred loads after an acceptable deformation in the pillars and surrounding wall rock has occurred.

Square Set Stoping—The key to square set stoping is good timbermen. The timber sizes and spacing between timber supports can be designed. However, the design specifications should be considered an average, as adjustments will be made as mining progresses. The most difficult part is determining the load on the timbers, which can be estimated using one of the numerical analyses. Because of the high labor cost, materials handling becomes a major component of the operating cost.

24.4.4.4 Stage 4: Modification During Production

During initial development and production, the ground conditions and the ore geometry are better defined. Because of this additional information, the mine design should be modified to account for these changes. Just as the ore limits cannot be redefined without additional drilling, the ground conditions cannot be evaluated without additional data collection. This data collection should include additional structure, rock strength, and in situ stress measurements, as discussed in Chapter 23.1, and the ground response to mining should be monitored.

Monitoring can identify areas of potential instability much sooner than they can be observed by the naked eye. By the time cracks are visible, the rock mass strength may be reduced to the point where the movement cannot be stopped. Monitoring should be done using the simplest device possible, such as the tape extensometer, borehole extensometers, surveying, borehole inclinometers, or pressure cells. The tape extensometer and an EDM (electronic distance measurement) survey system are two of the easiest methods to use; the survey system is usually used in the open pit, while the tape extensometer is used predominantly underground. The EDM utilizes x, y, and z coordinates and plots the bearing, plunge, and amount of displacement. The tape extensometer measures the movement between two points, for example from rib to rib, back to floor, or rib to back. The data collection should include the stations monitored, and the date, time, and extent of mining.

The following discussion on data interpretation applies specifically to the tape extensometer, but is also applicable to survey monitoring. The data should be plotted as a cumulative displacement plot and a velocity plot (Fig. 23.4.13). The shape of the curve is more significant than the absolute value. On the cumulative displacement plot, a flat line means no movement, a constant sloping line means there is movement but the rate of movement (velocity) is constant; an increase in the slope of the line means the velocity is increasing, and a decrease in the slope of the line means the velocity is decreasing. The velocity plot usually shows more variability than the cumulative displacement plot. On a velocity plot, a flat line means the velocity (rate of movement) is constant, a constant slope means the velocity is either increasing or decreasing at some constant rate (i.e., constantly accelerating or decelerating), and a changing slope means the acceleration is increasing. By correlating the ground movement with the mining activity, we can determine when ground is being loaded to its limits. Unfortunately, it seems that unless there is an operational concern regarding ground movement, the monitoring does not get done. It should be part of the normal follow-up on the design, and used to compare with design changes.

Even though at this stage we know more about the deposit, making changes in the mining system is not necessarily easy. The access drifts and/or haulage systems can have a lead time requiring a three- to five-year delay before any changes in the mining system can actually be implemented. A change in the slope angle can be made at most any time during the operation.

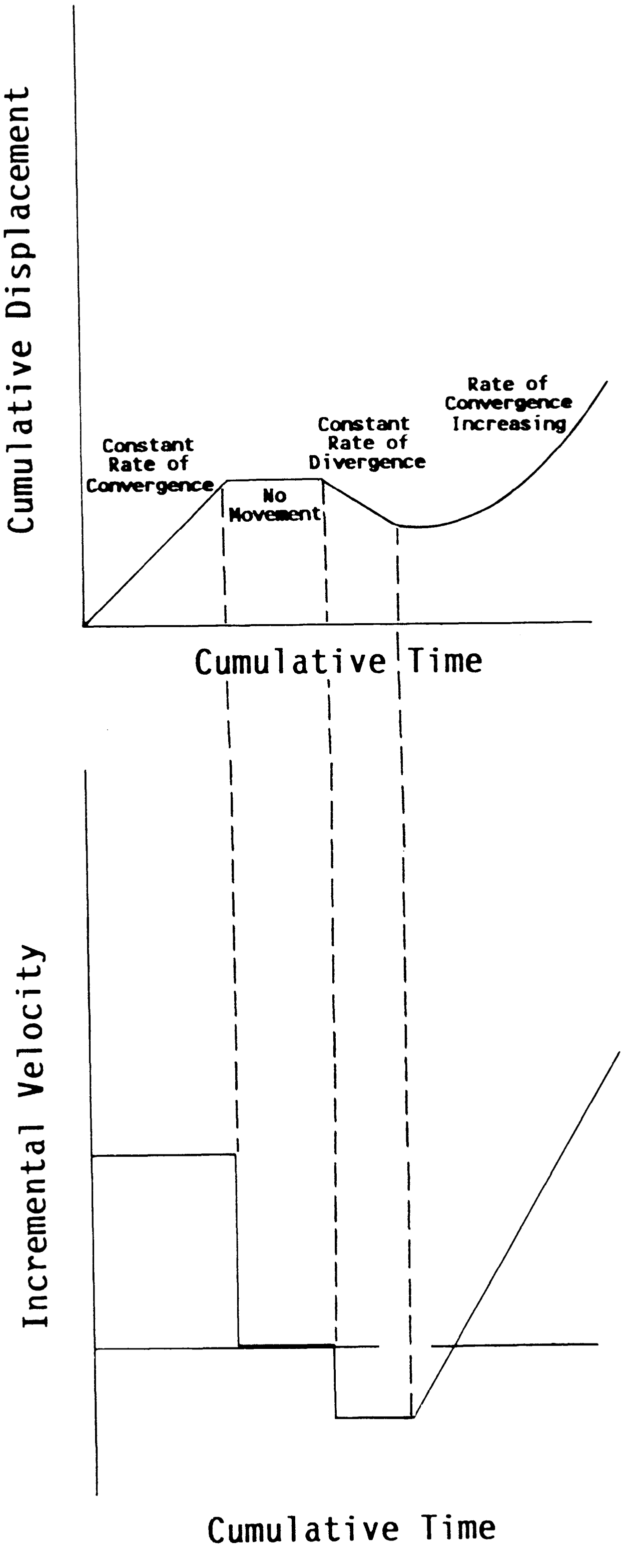


Fig. 23.4.13. Cumulative displacement plot and velocity plot.

Even though the design has been engineered in the feasibility study, the engineering approach at this point should be to broaden and improve the database and to re-evaluate the mine design during development and production.

23.4.4.5 Stage 5: End of Mine Life

By the end of the mine life, the approach to ore recovery has changed because the capital has been paid off and the main objective is simply to extend the mine life as long as possible. Because of this, there is more willingness to mine at a higher operating cost, a lower grade, and a lower chance of success. Therefore, the process of choosing how to mine the remaining ore (or ore not included in the original minable reserves) is similar to that discussed in stages 2 and 3; the difference lies in the level of risk, because the only alternative is to shut the operation down.

23.4.5 SUMMARY

In performing the mining method selection task, it is important to remember that no one method is able to meet all of the requirements and conditions. Rather, the appropriate mining method is that method that is technically feasible for the ore geometry and ground conditions, while also being a low-cost operation. This means that *the best mining method is the one with the technical problems that are the cheapest to deal with.* The mining engineer must balance all of the input parameters and select that method that appears to be the most suitable, making method selection both an art and a science. Still, by ensuring the maximum use of available data and performing detailed economic analyses on paper, the chances of making the most appropriate selection are vastly improved.

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Section 24 Openings for Nonmining Purposes

MADAN M. SINGH, ASSOCIATE EDITOR AND SECTION COORDINATOR

Chapter 24.0
INTRODUCTION

MADAN M. SINGH

This section is unique to the Handbook in that it is the only section that deals with underground excavations for nonmining purposes. The editors were of the opinion that the section should be incorporated because the subject matter covered by the chapters is closely related to mining, and many mining engineers are currently involved in these activities. As we enter the decade of the 1990s activities in this category will increase, enhancing the relevance and utility of the material discussed.

As the population increases, especially in metropolitan areas, the necessity to go underground will be greater. At the present time, not only are subsurface shopping malls and pedways connecting buildings becoming commonplace, but structures as diverse as the Civil and Mining Engineering Building on the University of Minnesota campus, a road-and-fleet maintenance facility in Multnomah County, OR, and even an ice cream parlor (Dairy Queen) in northern Wisconsin have been constructed underground. Placing utilities below the surface has numerous advantages and is now customary. Rapid transit and highways through tunnels in urban areas have been the norm for many years, but subterranean stormwater reservoirs, power plants, industrial warehouses, and food storage silos are relatively new innovations. The philosophy of underground defense installations is ancient, but their requirements and designs have transformed radically as technology advances. The use of underground space for disposition of wastes is modern, since it is recently that environmental concerns and waste disposal costs have assumed increased significance.

As the reader peruses the following chapters, it should be borne in mind that many of the principles and technologies discussed therein in connection with a particular usage are applicable in numerous other situations. As our understanding of these factors improves, novel nonmining uses of subsurface space will emerge (e.g., the placement of an underground cable-operated people-mover in the Austrian ski village of Serfaus), thereby widening the scope of engineers trained in the fields of mining, geology, and geotechnology. Among the items of importance common to several underground usances are the effects of heat, water, and vibrations. Thus the effect of high temperatures on the surrounding strata are not only of significance to nuclear waste disposal and defense, but also critical to in situ coal gasification and retorting of oil shales. The better understanding of the flow of water through the ground is helpful for locating subterranean structures, modeling of toxic waste sites, and planning (effects on local aquifers) of new underground mines.

Civil works tunnels are reviewed in Chapter 24.1. These are located primarily in congested areas and, therefore, subject to constraints in location, geology, size, method of excavation, and support design and requirements. These may be built for a variety of different uses: rail transit, highways, water supply, wastewater storage and transport, utilities, pedestrian paths, and other purposes. Complexities in the design of these tunnels pose a challenge to the ingenuity and ability of the engineer.

Chapter 24.2 discusses the use of underground space for the purpose of bulk storage of materials (solid, liquid, and gaseous) and for generation of electrical power. Siting and design of the caverns for these uses requires detailed knowledge of the geology and support requirements, and special needs relevant to the purpose for which these are excavated. Pumped hydrostorage power plants, such as those in Bath County, VA, and near Rome, GA, have been constructed and successfully used. Schemes have also been suggested for the storage of pumped hydroelectric, compressed air, and superconducting electromagnetic energy. Unique facilities, such as for the superconducting super collider (SSC) proposed for construction near Fort Worth, TX, and the large electron-positron collider (LEP) and super proton synchrotron (SPS) operated by the European Organization for Nuclear Research (CERN) have been designed for research or other special uses.

A discourse on high-level nuclear waste repositories is presented in Chapter 24.3. In addition to engineering parameters, a host of societal factors (including regulatory and political issues) govern the siting and design of such facilities. The idea involved in the design is to encapsulate the hazardous nuclear waste in solid form in hermetically sealed canisters, and emplace it in a suitable. geologic formation, nearly 1000 to 3000 ft (300 to 1000 m) below the ground surface, so as to isolate it from the envirosphere and minimize the potential for human intrusion. Hydrological, thermal (generated by radioactive decay), geochemical, and long-term stability (including tectonic behavior) considerations are significant and need to be taken into account. The design life of the repository may be divided into the operational phase (first 100 years) and an isolation phase (10,000 years). Hence the adoption of mining methods that do not introduce potential flow paths for gaseous and fluid emissions and appropriate sealing of the openings created assume added importance. Regulations also require that the material be able to be retrieved, if desired, during the operational phase. Although the scope of this chapter is limited to high-level radioactive waste, repositories suitable for the underground disposal of low-level nuclear wastes, toxic wastes, and mixed wastes remain possibilities in the future.

The use of underground facilities for defense and security is a notion long used by mankind, although this has taken on renewed significance with the advent of nuclear weapons in this century. The concepts are noted in Chapter 24.4, although many aspects of the facilities remain classified. Even the use of a London underground tunnel as a factory for the production of vital machinery during World War II was disclosed to the public only after 40 years. The Scandinavian countries have adopted the use of underground shelters in the planning of communities, and have combined the military function with peacetime civilian uses to justify the costs. In the United States, several facilities have either been built (such as the communications and command center at Cheyenne Mountain in Colorado Springs, CO, the satellite tracking station near Spokane, WA, and scattered missile silos in the western states) or envisaged (such as deep basing for the MX-missile). Ingenious excavation and hardening requirements have been developed for utilization in such facilities.

Because the principles of excavation are covered so well elsewhere in this *Handbook* (Sections 9 and 10), the emphasis of the current chapters is on applications.

Chapter 24.1
CIVIL WORKS TUNNELS FOR VEHICLES, WATER, AND WASTEWATER

J.E. MONSEES AND W.H. HANSMIRE

24.1.1 INTRODUCTION

As the population and congestion of urban areas increase, the demand for civil works tunnels to provide transportation, convey water and wastewater, and serve as passageways for pedestrians and utilities also increases. No two tunnels are ever the same because of the wide variation of conditions applying to individual tunnels:

1. They may be in soft ground, soft rock, hard rock, or a mixed-face combination of these.
2. They may have different geologic conditions such as ground water, in situ stresses, or geologic structures.
3. They may be built by different methods and equipment and supported and lined by different methods.
4. They may have different end uses.

For all of these reasons, and more, it is not possible to discuss in a single *Handbook* chapter all of the considerations that go into planning, designing, constructing, and operating tunnels for civil works. (Indeed, it could be argued that to attempt to do so might mislead some into thinking that civil tunnels are simple structures.) In this chapter, the goal is to discuss some of the major considerations that go into a civil tunnel system and thereby to raise the level of awareness of the reader that such tunnels are very complex structures. If this goal can be met, then the reader's appreciation of at least the general considerations relating to civil tunnels will be heightened, and he/she will better understand where to turn for more information and for resolution of those considerations. [One publication that all readers of this chapter will want to have in their libraries is *Guidelines for Tunnel Lining Design* (O'Rourke, 1984).]

In the usual case, civil tunnels are built to serve the populace. Thus they must be built where the people are, and their end points are generally fixed by existing infrastructure. The reader should contrast this situation to many other tunnel systems, for example, waste repositories, military installations, and power tunnels. In these latter cases, it is usually part of the designer's challenge to locate the tunnels in the best possible geology.

Given that civil tunnels must be built between existing end points, the challenge is to optimize the design to accommodate conditions encountered in between those end points. This challenge can be very demanding on the engineer's training, experience, and ingenuity because the optimization process can require making difficult decisions concerning trade-offs among options that usually are not simple to characterize.

24.1.2 GEOTECHNICAL INVESTIGATIONS

Geotechnical investigations play a principal role in the development of a civil tunneling project. These investigations provide the description of the general geologic conditions, estimates of soil or rock properties, approximations of the loading and interaction conditions applicable to the tunnel, estimates of the ground behavior during construction, and estimates of the impact of tunneling on the nearby surface and subsurface.

Geotechnical investigations should be conducted in a stepwise fashion, proceeding through reconnaissance, a preliminary program, and a final program. Areas of special concern during the investigation are summarized in Table 24.1.1. More detailed discussions of methods appear in Sections 4, 5, and 10.

24.1.2.1 Reconnaissance

Reconnaissance is the first step in the geotechnical exploration process. It should start with a visit to the site by the geotechnical engineer or engineering geologist, during which the whole alignment is inspected, preferably on foot. Outcrops, existing cuts, drainage, existing improvements, and existing tunnels should be inspected. Geophysical exploration methods may be helpful, as are aerial or satellite photographs, ground surveys, and historical records of other soil exploration programs and construction projects in the same geology. From this reconnaissance, the preliminary boring program is planned.

24.1.2.2 Preliminary Boring Program

This program should provide borings on a general pattern with extra borings at special features (e.g., faults) that were identified during reconnaissance and at any special construction feature (e.g., shafts). The borehole exploration program should be coordinated with the needs of the designer and supervised by a geotechnical engineer. Frequently, borehole programs are preceded by plotting and evaluating the available borings of others. Standards are given by the American Society for Testing and Materials (Anon., 1983) for borehole sampling and testing methods that include auger borings, split-spoon, and thin-walled "undisturbed" sampling, as well as the standard penetration and field vane shear tests. Additional in situ tests in soil include cone penetration and pressuremeter tests.

Coring with double-tube or triple-tube core barrels is used to investigate rock conditions. The core is evaluated on the basis of primary fracturing, strength, and weathering. Special borehole methods in rock include oriented core sampling, borehole photography, gas monitoring, overcoring grout-injected specimens to evaluate joint characteristics and small-scale voids, various overcoring and hydrofracturing techniques to determine in situ stresses, as well as water pressure tests to estimate permeability.

Boreholes provide essential information about the groundwater, including perched, artesian, and depressed water levels. Piezometers can be installed in selected boreholes to determine initial conditions and monitor changes in groundwater levels related to construction (O'Rourke, 1984).

Field pumping tests provide valuable data on dewatering requirements and potential construction effect; on local groundwater. Deep test pits provide for direct observation of subsurface conditions and the collection of block, or hand-cut, samples. Shallow test pits at existing foundations and utilities can be used to evaluate structural response to tunneling and to monitor movements and distress at critical locations. Exploratory or pilot tunnels, adits, and shafts are used on large projects when difficult tunneling conditions are anticipated. In some cases, geophysical techniques may be appropriate to delineate the top of rock between boreholes or to distinguish different types of subsurface materials with downhole applications of the technique (O'Rourke, 1984).

Table 24.1.1. Areas of Special Concern in Geotechnical Investigation for Tunneling

Planning, Design, Production of Contract Documents		Construction Methods		Influence on Adjacent Structures and Environment	
Selection of Tunnel Alignment and Profile	• Overall subsurface conditions. • Major discontinuities, contacts, unconformities. • Adjacent buildings and underground facilities.	Excavation and Disposal of Muck	• Variations of in situ structure and rock hardness, obstructions and cementation in soft ground. • Effect of mixing with water, contamination, leachates.	Hazardous Conditions	• Sudden falls, running, or flowing material. • Explosive or toxic gases. • Corrosive soil or groundwater. • Ground impregnation by waste leachate, leaking petroleum, chemicals, and sewage. • Contaminated tunnel soil.
Prediction of Ground Behavior	• Soil-like or rock-like performance. • Ground loads, stability, and deformation. • Permeability, groundwater inflows. • Joints, slickensides, shears, and faults.	Ground Stabilization	• Inherent stability and stand-up time. • Permeability and suitability for grouting or freezing. • Blow-out potential in compressed air.	Effects on Adjacent Structures	• Ground loss from tunneling, consolidation, blasting vibrations. • Disruption of or interference with utilities.
Selection of Tunnel Cross Section and Lining	• Relative stiffness of lining and ground. • Stability of crown, invert, and walls. • Effect of in situ structure on opening shape. • Tunnel orientation relative to in situ stresses.	Groundwater Problems	• Quantity and flow concentrations. • Running or piping of materials. • Need for control measures: drawdown, compressed air, grouting.	Changes in Groundwater	• Effects on wells, plantings, and natural vegetation. • Widespread consolidation by drawdown. • Long-term changes of groundwater level.
Decision Between Water Pressure Resistance or Relief	• Groundwater amount and quality. • Special watertightness membranes, leakproofing. • Corrosion, precipitates, clogging by fines.	Variation in Ground Conditions	• Variations from average conditions. • Strata changes, elevation changes along contact surfaces. • Number and orientation of discontinuities and faults. • Mixed-face conditions.		

Source: O'Rourke, 1984.

Note: Potential hazards should be identified according to natural and man-made origins. Natural hazards include toxic and explosive gases and large groundwater inflow. Man-made hazards include leachates, petroleum and chemical wastes, sewage, and buried or abandoned structures. In general, man-made hazards are most likely to be encountered in urban projects.

24.1.2.3 Final Boring Program

The final boring program is the third step in the exploration process, but often takes the form of a continuation and expansion of the preliminary boring program. The major change is the increase in number of borings (or decrease in average boring spacing). Typically, the final boring program includes borings at a maximum average spacing of 300 ft (90 m), but for some projects this average spacing may be 100 ft (30 m). Note that borings should have closer spacing in reaches of changing or troublesome geology, and may have wider spacing elsewhere. From this final boring program, the geotechnical engineer should have the data needed to prepare subsurface investigation and interpretation reports containing the information summarized in Tables 24.1.2a and 24.1.2b.

24.1.2.4 Borings in Rock

Borings in rock are usually continuous and are taken by means of a rotary drill rig using multiple-tube core barrels. A geotechnical engineer or engineering geologist should be in direct supervision and control of all boring operations and prepare (or supervise preparation by a qualified technician) a drill log of every hole drilled. This drill log should generally include the following information:

1. Drilling depth and/or elevation.
2. Sample number and conditions.
3. Detailed description of stratigraphy and lithology.
4. Water level.
5. Core recovery.
6. RQD (rock quality designation).
7. Weathering.
8. Bedding and/or joint spacing.
9. Discontinuity description:
   a. Bedding or jointing.
   b. Dip.
   c. Filling—type and degree.
   d. Surface—condition and roughness.
   e. Permeability.

Table 24.1.2a. Information to be Considered for Subsurface Investigation Report

A. *Reconnaissance and Geologic Background*
- Identify and describe lithological and structural units from maps, outcrop examination, geologic background.
- List appropriate references on local or regional geology, groundwater studies.

B. *Boring Log, Exploration, and Field Test Information*
- Log or record made by professional on site including quantitative data on depths, sampling, and recovery.
- Methods and equipment used in drilling and sampling.
- Descriptions of each sample or core and overall strata descriptions.
- Observations made during drilling: rate of advance, drill water conditions, bit pressure and rpm.
- Water level measurements: including times, dates, correlation with drilling operations, rainfall or open water.
- Special details of rock cores: character of discontinuities, joint and fault attitudes, core sketches and core photographs (in color where appropriate).
- Data on in situ testing: pump or packer tests, geophysical tests, deformation or strength tests.
- Presence of boulders and other obstructions.

C. *Laboratory Test Information*
- Diagrams and data sheets detailing test procedures and test results.
- Correlations between various properties, such as sand shear strength and relative density, uniaxial compressive strength and rock index properties.
- Properties of soil or rock grouped by strata or lithological units.

D. *Background Reference Information*
- Listing of technical references.
- Listing of drawings and information on adjacent structures, utilities, preexisting conditions.
- Sources of boring and testing information from other projects.
- Availability for examination of rock cores and soil samples.

Source: Modified from O'Rourke, 1984.

Table 24.1.2b. Information to be Considered for Geotechnical Interpretation Report

A. *Geological and Geotechnical Conditions*
- Interpretive geologic maps, cross sections, and soil profiles, separating and describing principal strata as determined from the field and laboratory investigations.
- Generalized properties of the strata recommended and utilized in design.
- Conclusions as to tunneling conditions expected, based on the geotechnical interpretation.

B. *Explanation of Design Assumptions*
- Criteria, assumptions, properties, and constraints utilized in design.
- Explanation for derivation of features of the technical specifications and limitations on construction procedures.
- Geotechnical criteria to be met by contractor's submittals for temporary support, cofferdams, shop drawings, or value engineering proposals.

Source: Modified from O'Rourke, 1984.

f. Slickensides.

10. Remarks, observations.

In addition to these data, it may be desirable to obtain other properties, such as rock-mass moduli, in specially designed field tests.

24.1.2.5 Borings in Soil

Borings in soil should be taken with a modern sampling barrel that incorporates a sample catcher, and should be as nearly undisturbed as technology permits. As a minimum, tube samples should be taken at intervals of 5 ft (1.5 m) or at every change of geology. As with rock borings, soil borings should be under the direct supervision and control of a geotechnical engineer or engineering geologist. This supervisor should also prepare or supervise the preparation of boring logs that contain the following as a minimum:

1. Drilling depth and/or elevation.
2. Soil description following the Unified Soil Classification System.
3. Water level.
4. Penetration resistance.
5. Pocket penetrometer or shear vane.
6. Remarks, observations.
7. Permeability.

24.1.2.6 Laboratory Tests—Rock

Laboratory tests on rock samples generally should consist of the following:

1. Unit weight.
2. Unconfined compressive strength.
3. Triaxial strength parameters.*
4. Shear tests on joints and joint fillings.*
5. Modulus.*

*as project conditions require

Other special tests, such as hardness, slaking, or swelling tests, may be required for specific project or geologic conditions. The requirement for such special tests must come from the design and geotechnical engineers.

24.1.2.7 Laboratory Tests—Soil

Similarly, laboratory tests on soil samples generally should consist of the following:

1. Moisture content.
2. Atterberg limits.
3. Grain size distribution.
4. Unit weight.
5. Unconfined compressive strength.
6. Direct shear strength.*
7. Triaxial strength.*
8. Modulus.*
9. Consolidation.*

*as project conditions require

As above, special tests may be required by the geotechnical or design engineers.

24.1.3 SOFT-GROUND TUNNELING

Soft-ground tunneling is defined as tunneling in ground that could be excavated by hand tools, although such hand excavation rarely occurs in today's world (McCusker, 1982).

Table 24.1.3. Ground Classifications for Soft-ground Tunneling

Classification	Behavior	Typical Soil Types
Firm	Heading can be advanced without initial support, and final lining can be constructed before ground starts to move.	Loess above water table, hard clay, marl, cemented sand and gravel when not highly overstressed.
Raveling, slow, fast	Chucks or flakes of material begin to drop out of the arch or walls sometime after the ground has been exposed, due to loosening or to overstress and "brittle" fracture (ground separates or breaks along distinct surfaces, opposed to squeezing ground). In *fast raveling* ground, the process starts within a few minutes; otherwise the ground is *slow raveling.*	Residual soils or sand with small amounts of binder may be fast raveling below the water table, slow raveling above. Stiff fissured clays may be slow or fast raveling depending upon degree of overstress.
Squeezing	Ground squeezes or extrudes plastically into tunnel, without visible fracturing or loss of continuity, and without perceptible increase in water content. Ductile, plastic yield and flow due to overstress.	Ground with low strength. Rate of squeeze depends on degree of overstress. Occurs at shallow to medium depth in clay of very soft to medium consistency. Stiff to hard clay under high cover may move in combination of raveling at execution surface and squeezing at depth behind face.
Running: cohesive, running	Granular materials *without cohesion* are unstable at a slope greater than their angle of repose (±30°–35°). When exposed at steeper slopes they run like granulated sugar or dune sand until the slope flattens to the angle of repose.	Clean, dry granular materials. Apparent cohesion in moist sand, or weak cementation in any granular soil, may allow the material to stand for a brief period of raveling before it breaks down and runs. Such behavior is *cohesive running.*
Flowing	A mixture of soil and water flows into the tunnel like a viscous fluid. The material can enter the tunnel from the invert as well as from the face, crown, and wall, and can flow for great distances, completely filling the tunnel in some cases.	Below the water table in silt, sand, or gravel without enough clay content to give significant cohesion and plasticity. May also occur in highly sensitive clay when such material is disturbed.
Swelling	Ground absorbs water, increases in volume, and expands slowly into the tunnel.	Highly preconsolidated clay with plasticity index in excess of about 30, generally containing significant percentages of montmorillonite.

Source: Modified from Terzaghi by Heuer, McCusker, 1982.

24.1.3.1 Ground Classification

Ground classifications for soft-ground tunneling were proposed by Terzaghi (1977) and have been modified by several authors, notably Heuer and Virgens (1987). These classifications are described in Table 24.1.3 and approximately correlated to grain size in Fig. 24.1.1. For further discussion of the stability and behavior of soils in soft-ground tunnels, the reader should refer to McCusker (1982), Clough and Schmidt (1981), and Terzaghi (1977).

24.1.3.2 Tunneling Methods

Soft-ground tunnels are nearly always excavated by means of a tunnel shield that is usually, but not always, circular. Within the shield, which can be thought of as a "cookie cutter" pushed horizontally through the ground, the workers excavate the ground and erect the initial support system in relative safety. Shields were first designed by Brunel in London in 1825. The modern shield takes numerous forms:

1. Simple Brunel-like cylinders using shove jacks in the rear and breast jacks in the face, with excavation in free air by hand.
2. Mechanized digger shields using hydraulic diggers much like a backhoe to excavate, and breast jacks and tables to control the face.
3. Mechanized digger shields using rotating cutterheads to excavate and variable "shutter" or "guillotine" doors to control the face.
4. Slurry face machines that maintain pressure on the face with a slurry of excavated soil or outside material (bentonite) to support the face, with excavation by a rotating cutter.
5. Earth balance machines that support the face by trapping and controlling the material removed from the face, thereby maintaining a soil pressure on the face equal or nearly equal to its in situ pressure, with excavation by a rotating cutter.

Because of a myriad of geologic and tunneling conditions, soft-ground tunneling employs a corresponding myriad of combinations of tunneling equipment. Often, for example, the shield is coupled with compressed air to tunnel in conditions where the soil strength (and stability) is low or the water table is high or where both obtain. The compressed air improves the face stability and keeps water out, but, depending on pressure level, can lead to severe working-time restrictions due to decompression requirements. For a more thorough discussion of soft-ground tunnel equipment types and operations, readers are referred to Stack (1982). It is noted that US practice tends to lean toward the more basic tunnel shields summarized in items 1, 2, and 3 above, while European and Japanese practice tends to lean towards the more mechanized equipment summarized in items 4

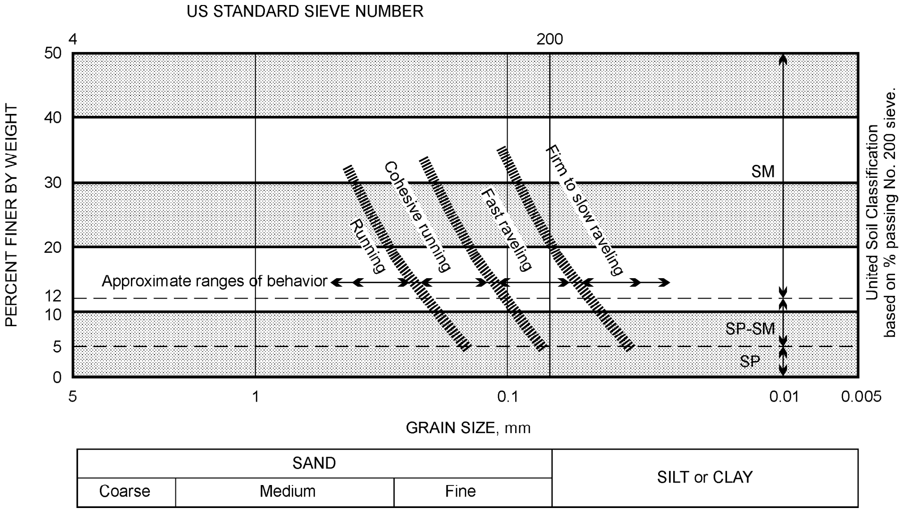

Fig. 24.1.1. Anticipated ground behavior based on D_{10} size. Shown for dense soil N 30, above water table (adapted from Heuer and Virgens, 1987).

and 5. Undoubtedly, the latter are the trend of the future and will be used more in the United States as time passes.

24.1.3.3 Soft-ground Tunnel Lining and Support

Soft-ground tunnels are typically supported by one of two general systems: (1) segmental precast concrete tunnel liners serving as both initial support and final lining, and (2) initial support by ribs and boards or low-cost precast concrete segments followed by a final lining of cast-in-place concrete. In some instances, but rarely in current times for extensive works, segmental steel or cast iron segments may be used as both initial support and final lining.

Precast Concrete Tunnel Liners. This lining serves as both the initial support and final lining. Construction is usually by traditional shield and the most severe loading the lining may experience is often at the time of construction. Fig. 24.1.2 illustrates one representative segment, its reinforcing, and bolt pocket layout.

The criteria for structural adequacy of the segmental lining are as follows:

1. Flexible behavior. The lining must be sufficiently flexible to be able to deform in response to an unbalanced load. Passive support pressures in the ground must be mobilized through ring deformations that do not induce unacceptable effects in the lining.

2. Sufficient strength for combined thrust and bending. Reinforced concrete design methodology is used to determine the design strength of the lining. In establishing the required strength, the behavior of the lining must be considered as being interactive with the surrounding ground. Special attention must also be given to the effective stiffness of the lining.

3. Constructability. Before the lining is subjected to earth loading, adequate provisions must be made for element handling, erection, bolting, shoving of the shield, expanding (sometimes), and grouting.

Design Approach. Structural capacity of the lining is determined in accordance with ACI 318; the lining is generally considered flexible, as originally outlined by Peck (1969). Hansmire et al. (1989) give an overview of design trends for flexible tunnel linings. Over time, research and experience have contributed to quantifying, refining, and broadening the lining design approach. For nearly all soft-ground tunnels, the tunnel lining is considered flexible, that is, one that deforms and interacts with the surrounding ground. The measure of flexibility is the flexibility ratio F, which is defined as

$$F = \frac{\dfrac{E_m}{(1 + \nu_m)}}{\dfrac{6EI}{(1 - \nu_\ell^2)} \dfrac{1}{R^3}} \tag{24.1.1}$$

where E_m is modulus of soil surrounding the tunnel, E is modulus of tunnel lining, I is moment of inertia of tunnel lining per unit length, $\nu_{m,\ell}$ is Poisson's ratio of soil and lining, respectively, and R is tunnel lining radius.

The relationship of moment in the lining to lining flexibility is illustrated for theoretical cases in Fig. 24.1.3. The moment has been made nondimensional as a moment coefficient by dividing by the factor γHR^2. For most practical applications, the lining can be considered flexible when $F > 10$.

The lining as installed has nonuniform properties about its periphery because of such items as bolt pockets and staggering of joints between rings. Joints between the segments act like partial hinges. The value of the moment of inertia varies from that of the solid section to a lower value at the nonuniformities. In view of these considerations, the lining usually does not behave as an intact, elastic ring of uniform stiffness.

Muir Wood (1975) suggests that owing to the influence of joints, the stiffness of the tunnel lining might be reduced to one-fourth the full lining value for an eight-segment lining. Analytical studies by Paul et al. (1983) of the effects of joints in segmented linings show the effective stiffness to be from 30 to 95% of the intact ring case. The reduction in stiffness was greatest for lower flexibility.

A further consideration in the stiffness of the tunnel lining is that reinforced concrete behaves in a nonlinear fashion. As the lining is subjected to thrust and bending, it cracks, behaves in a less stiff manner, and helps redistribute moment. Analysis, however, usually makes an assumption of linear properties. The result is that the linear analysis suggested herein typically underestimates the load-carrying capacity of the lining.

For analysis of strength required, the authors normally assume that the stiffness of the lining (EI) for the full section could be halved ($EI/2$). Parameter studies and design analysis use various values to test sensitivity to this assumption.

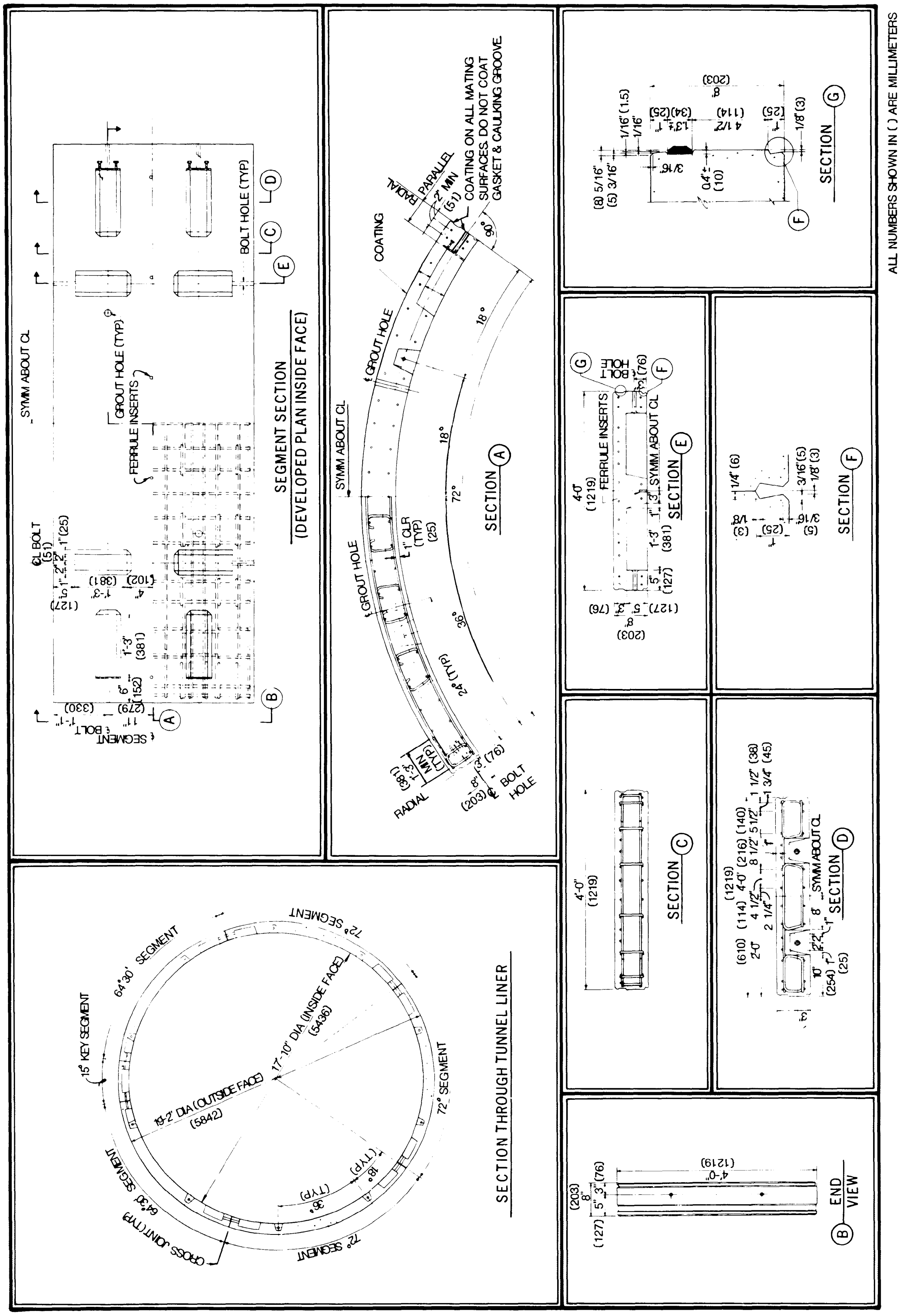

Fig. 24.1.2. Precast concrete segmented liners. Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

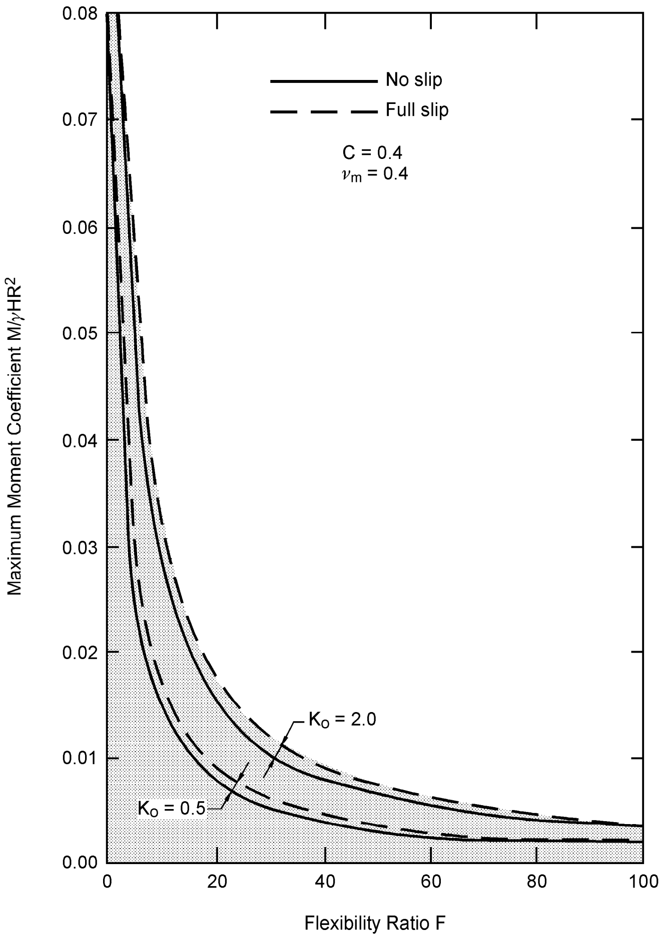

Fig. 24.1.3. Maximum moment coefficient as a function of the flexibility ratio (adapted from O'Rourke, 1984).

In all but the empirical methods (discussed later) for tunnel lining design, an assumption must be made about the state of stress in the ground prior to tunneling (K_o) or the ratio of unbalanced loads on the tunnel lining (K). As used herein, K_o is defined as the ratio of horizontal to vertical stress in the soil prior to tunneling. The parameter K is the ratio of unbalanced load (horizontal to vertical) that produces moment in the tunnel lining, and, except for a perfectly rigid lining, generally will be between the value of K_o and 1.0.

Most design approaches recognize that the lining must be able to withstand an "unbalanced loading" between vertical and horizontal forces. In conjunction with relative lining stiffness, the assumed final unbalanced loading will control the structural requirements for earth loading.

The assumption of a stress ratio K_o in two-dimensional analyses is inconsistent with the stress relief that results from excavation prior to lining installation. Stress is first relieved directly by excavation at the tunnel face. Furthermore, the lining is not installed at the face, but about one tunnel diameter behind. In a sand, or almost any soil except a soft clay, the actual loading may be a modest gravity loading of soil dropping into the tail void and would have little relation to the original state of stress in the ground.

If it is assumed that the initial stresses are unloaded during construction and will not be reestablished in the long term, extreme values of K_o cannot be used. In a sense, then, the selection of K_o is a judgmental factor that fits the designer's perception of an adequate design. For most projects, an unbalanced loading ratio of K or $K_o = 7/8$ is judged to be adequate for analysis (Schmidt, 1984; Hansmire et al., 1989).

The design must recognize that loading of a tunnel lining is different in nature from that of aboveground structures. The loading a tunnel lining will experience is often dictated by geologic details. The design approach is to select loadings consistent with geologic conditions and construction procedures. Maximum, or extreme, loadings are assumed to have little uncertainty. In such cases, there is no need to include an additional margin of safety by applying load factors. In the analysis of less extreme cases, a load factor would typically be used. This approach generally follows the philosophy expressed in O'Rourke (1984).

For analysis of most loadings, a load factor of 1.4 is recommended where deemed appropriate. When an earth loading equal to full overburden is used, the authors believe a load factor of 1.4 is excessive because loads generally cannot exceed full overburden.

DESIGN ANALYSIS. Realistic assumptions must be made in the analyses. Assumptions of various parameters must recognize not only the intrinsic properties of the lining but also the limitations of the analytical model in representing the true physical conditions.

The *empirical design approach* proposed by Peck (1969) assumes a lining load corresponding to full or partial overburden, and a bending moment corresponding to a diametric distortion of the lining. Moment in the lining is computed by the relationship

$$M = \frac{3EI}{R}\frac{\Delta D}{D} \tag{24.1.2}$$

where ΔD is diameter change and M is moment, with other terms as defined before.

The value of diametric distortion ($\Delta D/D$) is assumed on the basis of experience. A value of 0.5% was originally proposed, based on experience with segmented metallic linings. In general, a precast lining that is analyzed as an intact ring cannot be designed within ACI code requirements for such a distortion in the low-thrust range common to shallow tunnels where bending strength is exceeded without the beneficial action of lining thrust.

Schmidt (1984) recommends several values that depend on soil type. For dense sands, a range of $\Delta D/D$ from 0.05 to 0.25% is given. For recent subway project analysis purposes, the value of 0.5% $\Delta D/D$ has been used, but $EI/2$ was used for lining stiffness.

The *empirical method* with an assumed diameter change and full (or partial) overburden thrust is often regarded as the primary method of analysis. Using an adjusted lining stiffness, the lining must accordingly be designed for the resulting bending moment with the appropriately selected thrust. In general, for most civil tunnels, full overburden thrust is readily accommodated by a precast concrete lining and high thrust is desirable to offset the tensile component of bending. The more critical case is for low thrust, which may exist during construction immediately behind the shield, or in tunnels with very low cover. In such cases it may be necessary to control the ovalling distortion of the lining until grouting is complete.

Closed form solutions from continuum mechanics exist for computing thrust and moment in a tunnel lining. As presented in Ranken et al. (1978), this is termed "excavation loading." Compared to other approaches, this approach usually represents a lower bound. The value of such solutions is largely to illustrate the significance of lining flexibility to bending moment. The analysis is limited in that K_o must be assumed, and computations are usually directly proportional to the assumed depth of tunnel. This analysis best characterizes lining behavior where the lining is installed at the tunnel heading with minimal loosening of the ground. The closed-form interaction analyses poorly model the conditions in which loosening loading is expected to be the primary loading. On the other hand, the analyses may more closely represent long-term conditions for tunneling in weak ground where the lining may receive its major loading from long-term conditions in the surrounding ground.

Analyses using *closed-form* continuum mechanics equations provide a convenient tool for parameter studies. For the modest lateral earth pressure coefficients assumed, computed thrusts and bending movements should be well within the strength capacity of the lining.

Ground-lining interaction may also be modeled by a series of *beams and springs.* In a general structural frame analysis computer program, the lining is represented as a series of straight beams. Horizontal and vertical (or radial and tangential) springs model reaction of the soil. Assumed loads from earth and groundwater are input as forces at the joints. A gravity loading of soil acting as a dead load on the lining is simulated. Lateral loading is input at the tunnel sides as required to simulate an unbalanced loading on the tunnel lining. A soil reaction develops around the lining in accordance with the relative stiffness of lining and soil. The final loading on the lining is determined from the addition of the forces input and the forces resulting from lining-soil interaction. The final unbalanced loading on the lining generally will be less severe *(K* higher) than that assumed for input forces only.

Beam-spring methods of analysis are outwardly attractive because they simulate the real physical structure. Loads are imposed as in the analysis of an aboveground structure. Both gravity loads from disturbed soil above the tunnel, and groundwater loads are simulated directly.

One of the greatest areas of confusion in the beam-spring type of analysis is distinguishing between what causes thrust and what causes bending in the lining. Full overburden thrust does not necessarily mean the lining is carrying the dead weight of all directly overlying soils (a gravity load). Arching theories indicate that in a deep tunnel, it is not possible for a loosening load, as in a sand, to be as great as full overburden pressure. Significant thrust can come from interaction of the lining with a stressed medium, such as that existing near the tunnel heading. This is generally called "excavation loading." For high-input loads, the beam-spring analysis produces unrealistic results because the computed bending moment is directly proportional to the tunnel depth. A full overburden ground load therefore is not appropriate, particularly for many civil tunnels.

The conclusion regarding the beam-spring type of analysis is that loading must be restricted to a loosening load. This loosening load will be less than full overburden for tunnels with more than about two diameters of soil cover. With the beam-spring analysis restricted in this way, computed results for bending moment are similar to those obtained in the empirical approach.

Construction loads often represent some of the most severe loadings the tunnel lining experiences. Handling loads take place during construction and are not the result of earth loading. Several stages of handling of segments are required from the time of casting through final erection of segments into a ring within the tail of the shield.

A significant jacking load occurs on the circumferential joint of the segments as the tunnel shield is advanced by shoving on the previously erected tunnel lining. In general, reinforcing of the lining must be designed to accommodate the planned shield jacks about the tunnel periphery. Final design of the tunnel shield, including the shield jacking system, is the responsibility of the tunneling contractor. The contractor must, therefore, verify that the precast lining is compatible with the jacking system.

The ability of the lining to carry the jacking load is highly affected by details of the lining joint configuration and by eccentricity of the shield jack loading. Over the gross thickness of the lining, the average stress from jacking is usually acceptable. However, the requirements for a gasket and caulking groove in the lining may leave much less area for seating of the shield jack reaction. At bolt pockets, a locally weaker section exists. Finally, shield steering requirements often place very unbalanced jacking loads on the lining. To meet all these conditions, higher strength concrete and strict limitations on the center of force of the shield jacks may be required.

Grouting or segment ring expansion is required to fill the tail void that is created as the tunnel lining emerges from the tail of the shield. Typical grout holes should be uniformly spaced and grouting pressures controlled; grouting produces thrust that is beneficial in accommodating bending, but uneven grouting can produce bending. Experience shows the lining can usually withstand the loading from such grout pressures as a temporary loading.

CAST-IN-PLACE TUNNEL LININGS. The cast-in-place lining design recognizes that the tunnel is built in two stages. At the time of excavation, an initial support, consisting of simple precast segments or steel ribs with timber lagging, is erected and expanded against the soil immediately behind the tail of the shield. At that time, the initial supports absorb initial ground loads and undergo distortion. The final cast-in-place concrete lining is placed weeks or months later. Thus, at least initially, the final lining will not take load directly from the ground. The most significant loadings will be from contact grouting or from future stress changes such as those resulting from building construction over or near the tunnel.

Criteria for structural adequacy of the cast-in-place liner are as follows:

1. Sufficient strength for contact grouting.
2. Sufficient strength to accommodate future stress changes.
3. Constructability (minimum thickness for concreting).

It is recognized that cast-in-place linings have significant strength even without steel reinforcing. In pure thrust, the lining generally has more capacity than is required for loads from grouting, from other future influences, or from the extreme case of full overburden weight of soil. Bending resulting from unbalanced loading on the lining is usually the largest factor.

Steel reinforcing is sometimes placed at springline as an outside ring to give bending strength for the common case of slight vertical squatting deformation of the tunnel. If the lining were carrying significant thrust, the lining would theoretically have adequate capacity to accommodate bending without this reinforcing. Placement of the reinforcing is considered a conservative measure. The reinforcing does have a beneficial action in controlling cracking should bending take place.

In the crown over a 60° arc about the centerline, similar reinforcing is sometimes placed at the inside face of the lining. This reinforcing is included for structural reasons for the bending induced by squatting of the tunnel. As with the sidewalls, the reinforcing reduces or distributes cracking. The reinforcing is also desirable to maintain the integrity of the lining and to prevent the extremely remote possibility of a wedge or slab of concrete falling from the crown.

The general considerations summarized in the discussion regarding analysis for the precast lining apply in varying degrees for the cast-in-place lining. Two different approaches to analysis of the cast-in-place lining are recommended and are summarized in the following paragraphs.

Based on the premise that future construction may take place above or near the tunnels, it may be assumed in the *empirical approach* that a $\Delta D/D$ of approximately 0.25% may occur. This diameter change is assumed to occur because of a change in stress conditions resulting from excavation and later loading by building foundations or by other construction. Since the future buildings may be above or to the side of the tunnel, it is assumed that the $\Delta D/D$ might impact the tunnel from any direction. When making calculations involving $\Delta D/D$, an effective modulus of elasticity for concrete of $E/2$ may be assumed because any such deformations would occur over a long period of time. Creep experienced by the concrete as the result of this slow deformation would make the lining act with reduced stiffness. A soil loading of approximately one tunnel diameter is usually used, arguing that a properly installed initial support system will not permit significant loosening or other load development on the final lining.

Alternative analyses are also performed to evaluate the structural adequacy of the lining. A *beam-spring model,* similar to that discussed above, may be used in a structural analysis program to simulate the lining. Vertical and horizontal loadings must be assumed. Computations for several assumptions produce thrusts and moments that are analyzed by means of an interaction diagram.

For all of the analyses outlined above (i.e., precast and cast-in-place linings), values of moment and thrust in the lining are estimated, and it is necessary to compare them with allowable values. The recommended method for making this comparison is to use the moment-thrust diagram as summarized in the next section.

NONLINEAR RESPONSE OF CONCRETE LININGS. When the thrust and moment around the lining have been calculated, it is necessary to evaluate these quantities in comparison with allowable values. Normally, it is necessary to make this comparison only at locations where one of the quantities is maximum or where there is an abrupt change in the lining section. Moment and thrust interact strongly, so it is customary to check these parameters together by using the *moment-thrust (M-T) interaction diagram* to represent the allowable combinations. The *M-T* interaction diagram can be drawn for each section of the lining and depends only on the section dimensions and material properties.

One way to obtain an *M-T* interaction diagram is to use the procedure of the ACI Code (Anon., 1987). In this procedure, the combinations of moment and thrust, which cause theoretical failure of the section under unconfined conditions, are computed and shown on a diagram in which thrust and moment are the axes. A general *M-T* diagram for a tunnel lining is shown in Fig. 24.1.4. This diagram may represent the full lining section if it has constant dimensions and composition, or several such diagrams may be used to represent different lining sections.

To determine whether the section for which the *M-T* diagram in Fig. 24.1.4 is adequate, the moment and thrust combination obtained in the analysis should be plotted on a project-specific diagram like that shown. The material factor in the ACI Code procedure is used as a safety measure to cover uncertainties in material properties, determination of section resistance, and the difference between concrete strengths from cylinder tests and the structure. If the moment and thrust combination lies inside the diagram, the section is adequate. As discussed earlier, the loads on the lining may be multiplied by a load factor to give the moment and thrust combination an additional margin of safety. Readers should refer to O'Rourke (1984), Paul (1983), and Hansmire (1984) for a more thorough discussion of the *M-T* diagram and its application.

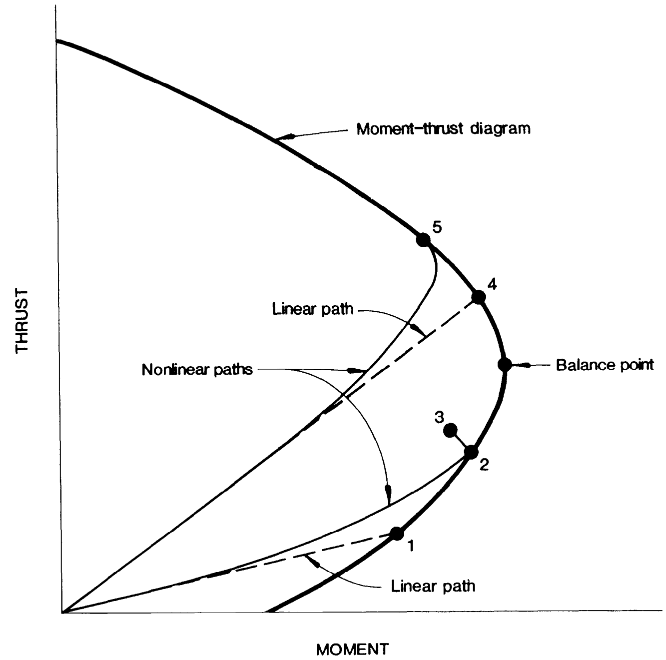

Fig. 24.1.4. General moment-thrust (*M-T*) diagram for a reinforced concrete lining with linear and nonlinear moment-thrust paths (adapted from O'Rourke, 1984).

24.1.4 SETTLEMENTS

It is a fact that ground movements take place during soft-ground tunnel construction. For most civil projects, the tunnels are shallow enough to result in surface settlements. However, few subsurface measurements were made about soil tunnels until the early 1970s. Since then, much has been learned from the Washington, DC, and Baltimore, MD, transit tunneling experiences about how ground movements around a tunnel are complexly related to surface settlements. In understanding these settlements, it is considered important to distinguish "volume of lost ground" from the "volume of settlement observed at the surface." Fig. 24.1.5 illustrates the distinction between the volume of lost ground and volume of surface settlements. Definitions are as follows:

1. *Volume of lost ground* (V_L): The sum of all ground movements taking place immediately about the tunnel periphery.
2. *Volume of surface settlement* (V_s): The sum of settlement at the ground surface.
3. *Volume change* (ΔV): Increases or decreases in soil volume above and adjacent to the tunnel.

For general usage, equating the volume of surface settlement to the volume of lost ground is acceptable. However, in many materials, increases in soil volume (or bulking) for a single tunnel can be very significant. Without allowance for volume increases, the surface settlements may be significantly overstated.

24.1.4.1 Volume of Lost Ground.

Lost ground occurs about the tunnel in three forms:

1. *Face loss:* Lost ground that occurs ahead of the shield in the form of soil squeezing, flowing, or running into the heading.
2. *Shield loss:* Lost ground that takes place from the time the leading edge of the shield passes a section until the tail of the shield passes that section. Shield loss takes place on account

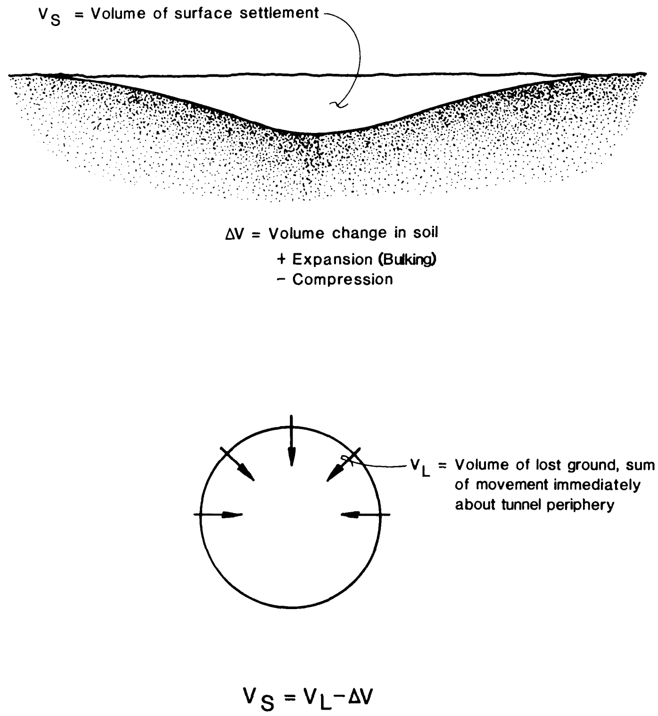

Fig. 24.1.5. Terminology for tunneling settlement and lost ground.

of shield actions (plowing, changes in pitch or yaw), from the void created by overcutters, or by overexcavation.

3. *Tail loss:* Lost ground that occurs after the tail of the shield passes a section. Tail loss results from insufficient filling of the tail void because of incomplete grouting or insufficient expansion of support members.

This segment presents judgmental expressions of the quality of the ground or workmanship in relation to tunneling. The following definitions are established:

1. *Good tunnel ground:* In most instances, this is synonymous with "firm ground" as defined by Peck (1969) or Terzaghi (1977).

2. *Good tunneling practice:* Expression used to describe the high quality and appropriateness of both tunneling procedure and workmanship for a particular ground condition. In general, this means

a. Tunnel face is supported as required to prevent loss through front of shield.

b. Shield design that minimizes ground disturbances (minimal overcutters, minimal overexcavation, easily steered, which means length/diameter < 0.9, unless articulated).

c. Tail loss minimized by preventing closure of tail void (such as by use of thin tail piece, good expansion of lining, and grouting behind lining during and after shove).

Two means are frequently used to estimate the volume of lost ground anticipated. The first uses past field measurements as precedents. The second uses estimates based on the source of lost ground.

Field measurements from past tunneling experiences in the United States are mostly from Washington, DC, and Baltimore, MD. Tunnels constructed with good procedures for tunneling in sand were able to achieve approximately 1% volume of lost ground. Average values measured were in the range of 1.5 to 3%.

Table 24.1.4. Volume of Lost Ground and Quality of Tunneling Practice

Case	V_L, %
1. Good practice in firm ground —Applies to better soils and excellent ground control	0.5
2. Good practice in slow raveling ground —Considered good ground	1.5
3. Fair practice —More shield and tail loss	2.5
4. Poor practice —Yet more shield loss —Tail void mostly unfilled	4.0

As an alternative to using past field measurements, estimates based on the source of lost ground may be used. As previously discussed, lost ground occurs in three forms: face loss, shield loss, and tail loss. The amount of lost ground is dependent on several factors relating to construction method and workmanship, as well as tunnel ground conditions. Estimates for volume lost for several cases are given in Table 24.1.4.

Larger ground losses can be encountered if large losses occur at the face. For example,

1. Boulders caught in the cutterhead of a shield with a rotating excavator could result in very large volumes of lost ground.

2. A run or flow of soils at the face of a shield could also result in significant ground losses.

3. Uncontrolled squeezing of clays or silts occur.

4. Consolidation settlements occur.

Such events are considered the result of poor workmanship (poor practice) and are to be avoided by strict contractual requirements.

On the basis of the foregoing concepts, a volume of lost ground equal to 1.5% of the tunnel volume is often used as a first approximation for analyses of tunneling settlements. This value applies to good tunneling practice in good tunneling ground and must be evaluated for each tunnel project.

For most practical computation purposes, the settlement analyses make several assumptions. For the individual tunnel, bulking is assumed for high soil cover areas. For shallow cover, volume of surface settlement for the individual tunnel is assumed equal to lost ground. Total settlements calculated for parallel tunnels are assumed equal to the straight addition of individual settlements for the first and second tunnels. No extra settlements are assumed for the pillar effect.

DISTRIBUTION OF SETTLEMENT. Generally, the shape of the settlement trough at the ground surface resembles that of an inverted bellshaped probability curve. This concept was used by Peck and others (including Schmidt, 1969) to correlate field measurements of trough width for several cases. Geometry of the settlement trough is illustrated in Fig. 24.1.6. In all cases in the calculations, the ground surface is assumed at the bottom of the building footing, and the influence of building stiffness is ignored.

The width of the settlement trough is measured by an "i-value," which is theoretically the horizontal distance from the location of maximum settlement to the point of inflection of the settlement curve. The maximum value of the surface settlement is theoretically equal to the volume of surface settlement divided by 2.5i.

Fig. 24.1.7 illustrates connections that have been made between settlement trough width and ground type through the use of the bell-shaped probability curve. The tunnel depth and settlement trough widths are simplified as ratios of the tunnel

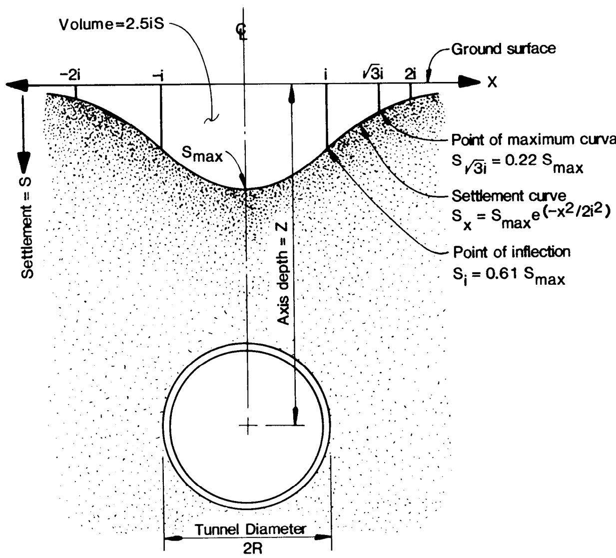

Fig. 24.1.6. Properties of probability curve as used to represent cross section of settlement trough above tunnel (adapted from Peck, 1969).

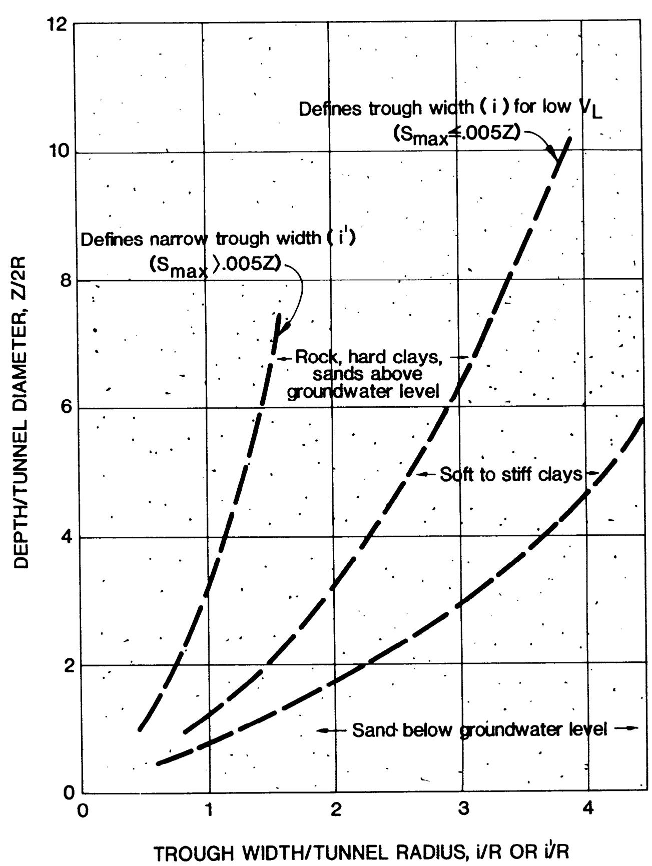

Fig. 24.1.7. Assumptions for width of settlement trough (adapted from Peck, 1969).

radius ($Z/2R$ and i/R). The trends in Fig. 24.1.7 illustrate that generally wider settlement troughs are associated with soft clays, while narrower troughs occur in stiffer materials such as dense sands. The deeper the tunnel, the wider the settlement trough.

Detailed field measurements have illustrated that the real and complex nature of tunnel settlements cannot be reduced to a simple precise correlation. For example, settlements will vary in a nonlinear way with the absolute magnitude of settlement (such as an ultimate case of forming a sinkhole at the ground surface). Also, geologic details such as stiff or loose layers of soil will strongly affect how ground loss about a tunnel becomes surface settlements. Detailed examples of actual measurements of ground movement about tunnels in sand are given in Cording and Hansmire (1975), and in clay in Palmer and Belshaw (1979, 1980). However, despite the inherent complexity, the correlations in Fig. 24.1.7 make possible a preliminary estimation of the settlement resulting from tunneling.

INFLUENCE OF SETTLEMENT—POTENTIAL FOR STRUCTURE DAMAGE. Having estimated the settlements (and recognizing that the estimates are often quite crude), the engineer should then evaluate the possible impact of that settlement on overlying structures. For reference in discussion, the following definitions from Boscardin and Cording (1989) of categories of building damage are made:

1. *Architectural damage:* Damage affecting the appearance of structures, usually related to cracks or separations in panel walls, floors, and finishes. Cracks in plaster walls greater than $\frac{1}{64}$-in. (0.4-mm) wide and cracks in masonry or rough concrete walls greater than $\frac{1}{32}$-in. (0.8mm) wide are considered to be representative of a threshold where damage is noticed and reported by building occupants.

2. *Functional damage:* Damage affecting the use of the structure, usually related to jammed doors and windows, cracking and falling plaster, tilting of walls and floors, and other damage that would require nonstructural repair to return the building to its full service capacity.

3. *Structural damage:* Damage affecting the stability of the structure, usually related to cracks or distortions in primary support elements such as beams, columns, and load-bearing walls.

Alternative methodologies exist for evaluating the influence of tunneling settlement on buildings. Many works exist on the subject, but a widely referenced one in civil works is Skempton and MacDonald (1956), who studied data from 98 buildings, 40 of which were damaged to some extent. The study included both framed and bearing-wall structures. Grant, Christian, and Vanmarke (1974) extended the study to 95 additional buildings, 56 of which had some damage. These results, along with the work of other investigators, were combined in a study by Wahls (1981). The results are astonishingly close in all cases, and permit evaluation of the influence of settlement on buildings by means of empirical correlation of building distortion (relative settlement between columns) to building damage. Such a correlation is given in Table 24.1.5.

It is generally agreed by all investigators, including the Soviets (Polshin and Tokar, 1957), that bearing-wall structures and plaster partitions are first cracked at an angular distortion of approximately $\frac{1}{300}$. Considering all of these factors, and to be conservative, it is recommended that the angular distortion be limited to one-half this amount, or $\frac{1}{600}$, and that absolute settlement of a column or footing be limited to 1 in. (25 mm).

The first line of defense in limiting possible damage from settlements induced by tunneling is to require and use tunneling equipment and methods that reduce the lost ground, including:

1. Full and proper face control at all times, especially during shoving of the shield.

Table 24.1.5. Limiting Angular Distortion

Category of potential damage	Angular Distortion
Danger to machinery sensitive to settlement	1/750
Danger to frames with diagonals	1/600
Safe limit for no cracking of buildings	1/500
First cracking of panel walls	1/300
Difficulties with overhead cranes	1/300
Tilting of high rigid buildings becomes visible	1/250
Considerable cracking of panel and brick walls	1/150
Danger of structural damage to general buildings	1/150
Safe limit for flexible brick walls, L/H 4[a]	1/150

Source: Wahls, 1981.
Note: [a] Safe limits include a factor of safety.

2. Limiting the length-to-diameter ratio for the shield.
3. Avoiding overexcavating or overcutting.
4. Rapid installation of ground support.
5. Rapid expansion, pea-gavelling, and/or contact grouting of ground support.

In special cases, other steps must also be considered, including:

1. Grouting or freezing the ground before tunneling.
2. Compaction grouting.
3. Underpinning structures by any of a group of methods.

24.1.5 ROCK TUNNELING

Rock tunneling is defined as tunneling in ground that could not reasonably be excavated by hand methods, that is, ground that requires excavation by drill and blast or tunnel boring machine methods. Readers will find the following discussion of rock tunneling less extensive than the preceding discussion of soft ground tunneling. This de-emphasis is warranted by extensive coverage of rock tunneling elsewhere in the *Handbook.*

24.1.5.1 Ground Classification

The pioneering systematized classification of rock for tunneling purposes was proposed by Terzaghi (1946). That classification is summarized in Table 24.1.6.

In 1968, Deere defined rock quality designation (RQD) as the core recovery (in percent) of sound pieces of core of 4 in. (100 mm) or greater in length (see Deere and Deere, 1989, for the latest illustration of RQD). For the first time, this permitted rock support to be based on a measurable quantity. A comparison of rock loads for the RQD system gave loads approximately 20% below those by Terzaghi (Table 24.1.7), and recommended loads for machine-driven tunnels at 25% below those for drill and blast tunnels (Deere et al., 1969). With this new system, approximate requirements for support systems could be given in terms of the RQD, as illustrated by Table 24.1.8.

In recent years, new classification systems have been proposed that add other factors to the RQD to develop support recommendations for rock loads (Barton, Lien, and Lunde, 1974, 1977; Bieniawski, 1974(b); Einstein, 1983; Hoek and Brown, 1980). The most important of these new systems are summarized in Table 24.1.9.

The writers have found that all systems require interpretation and judgment based on experience. It is recommended that initial decisions be based on the RQD. Final design may then incorporate a check by one or more of the more detailed methods based on the engineer's evaluation of the specific site conditions and requirements.

Table 24.1.6. Rock Load H_p, in feet or meters of Rock, on Roof of Support in Tunnel with Width B and Height H at Depth of More than 1.5C, where $C = B + H$

Rock Condition	Rock Load H_p	Remarks
1. Hard and intact	Zero	Light lining required only if spalling or popping occurs
2. Hard stratified or schistose	0.0 to 0.5B	Light support
3. Massive, moderately jointed	0.0 to 0.25B	Load may change erratically from point to point
4. Moderately blocky and seamy	0.25B to 0.35C	No side pressure
5. Very blocky and seamy	0.35C to 1.1C	Little or no side pressure
6. Completely crushed but completely intact	1.1C	Considerable side pressure. Requires continuous support for lower ends of ribs or circular ribs
7. Squeezing rock, moderate rock	1.1C to 2.1C	Heavy side pressure
8. Squeezing rock	2.1C to 4.5C	Circular ribs recommended
9. Swelling rock	Up to 200 ft (60 m) irrespective of C	Circular ribs required. May require yielding support

Source: Modified from Bickel and Kuesel, 1982.
Note: H_p, B, H, and C are inconsistent units; feet or meters.

24.1.5.2 Tunneling Methods

Civil tunnels in rock are driven either by tunnel boring machines or drill-and-blast methods. Depending upon the tunnel, *tunnel boring machines* (TBMs) come in numerous configurations, but all contain the same basic components (Bickel and Kuesel, 1982):

1. Excavation system (cutterhead).
2. Machine body.
3. Muck system.

These machines are capable of excavating rock with compressive strength up to approximately 30,000 psi (207 MPa). They generally cut a circular tunnel with diameters up to approximately 35 ft (10 m). Their selection and design is a highly specialized area and beyond the scope of this chapter.

Drill and blast methods may be required as a matter of practical choice or for special conditions, including the following:

1. Large openings.
2. Other than circular shapes.
3. High-strength or abrasive rock.
4. Highly variable rock.
5. Mixed-face conditions.
6. Short tunnel runs.

Blasting techniques are very similar to those explained elsewhere in this *Handbook* (see Chapter 9.2). There are, however, some differences that may apply to civil tunnels:

1. The structures are designed for a long life, and, therefore, rock damage may be of greater concern. Thus such items as control of powder factors and use of smooth-wall blasting methods may be more important.

Table 24.1.7. Comparison of Terzaghi and Deere Rock Loads for Tunnels Supported with Steel Sets

RQD	Joint Spacing mm	Joint Spacing in.	Description	Deere Rock Load For Tunnels Up to 40-ft (12-m) Diameter: Conventional	Deere Rock Load For Tunnels Up to 40-ft (12-m) Diameter: Boring Machine	Terzaghi Classification: Description	Rock Load: Initial	Rock Load: Final
		24				Hard and Intact	0	0
	500		Excellent	0 to $0.3B$	0 to $0.2B$	Hard, stratified or schistose	0	$0.25B$
90		12						
				$0.3B$ to $0.6B$	$0.0B$ to $0.4B$	Massive, moderately jointed	0	$0.5B$
80	200		Good			Moderately blocky and seamy	0	$0.25B$ to $0.35C$
		6						
70								
60			Fair	$0.6B$ to $1.3B$	$0.4B$ to $1.0B$	Very blocky, seamy and shattered	0 to $0.6C$	$0.35C$ to $1.1C$
50	100	4						
40			Poor	$1.3B$ to $2.0B$	$1.0B$ to $1.6B$			
30								
20				Excluding squeezing or swelling ground: $2.0B$ to $2.8B$	$1.6B$ to $2.2B$	Completely crushed, gravel and sand and squeezing and swelling rocks	$0.54C$ to $1.2C$	$0.62C$ up to 250 ft (75 m)
10	50	2	Very Poor	In squeezing and swelling ground: Rock load up to 250 ft (75 m)	Rock load up to 250 ft (75 m)			
		1						

Source: Modified from Bickel, 1984.
Note: B = width of tunnel
C = width plus height of tunnel

Table 24.1.8. Support Recommendations for Tunnels in Rock (20 to 40 ft, or 6 to 12 m in diameter)

Rock Quality	Tunneling Method	Alternative Support Systems: Steel Sets**	Rock Bolts*	Shotcrete
Excellent* RQD >90	A. Boring machine	None to occasional light set. Rock load 0.0 to 0.2*B*.***	None to occasional.	None to occasional local application.
	B. Conventional	None to occasional light set. Rock load 0.0 to 03.*B*.	None to occasional.	None to occasional local application.
Good* 75 < RQD < 90	A. Boring machine	Occasional light sets to pattern on 5 to 6 ft (1.5 to 1.8 m) center. Rock load 0.0 to 0.4*B*.	Occasional to pattern on 5 to 6 ft (1.5 to 1.8 m) center.	None to occasional local application 2 to 3 in. (50 to 75 mm).
	B. Conventional	Lights sets, 5 to 6 ft (1.5 to 1.8 m) center. Rock load 0.3 to 0.6*B*.	Pattern, 5 to 6 ft (1.5 to 1.8 m) center.	Occasional local application 2 to 3 in. (50 to 75 mm).
Fair 50 < RQD < 75	A. Boring machine	Light to medium sets, 5 to 6 ft (1.5 to 1.8 m) center. Rock load 0.4 to 1.0*B*.	Pattern, 4 to 6 ft (1.2 to 1.8 m) center.	2 to 4 in. (50 to 100 mm) on crown.
	B. Conventional	Light to medium sets, 4 to 5 ft (1.2 to 1.5 m) center. Rock load 0.6 to 1.3*B*.	Pattern, 3 to 5 ft (0.9 to 1.5 m) center.	4 in (100 mm) or more on crown and sides.
Poor** 25 < RQD < 50	A. Boring machine	Medium circular sets on 3 to 4 ft (0.9 to 1.2 m) center. Rock load 1.0 to 1.6*B*.	Pattern, 3 to 5 ft (0.9 to 1.5 m) center.	4 to 6 in. (100 to 150 mm) on crown and sides. Combine with bolts.
	B. Conventional	Medium to heavy sets on 2 to 4 ft (0.6 to 1.2 m) center. Rock load 1.3 to 2.00*B*.	Pattern, 2 to 4 ft (0.6 to 1.2 m) center.	6 in. (150 mm) or more on crown and sides. Combine with bolts.
Very poor⁺ RQD < 25 (Excluding squeezing or swelling ground)	A. Boring machine	Medium to heavy circular sets on 2 ft (0.6 m) center. Rock load 1.6 to 2.2*B*.	Pattern, 2 to 4 ft (0.6 to 1.2 m) center.	6 in. (150 mm) or more on whole section. Combine with medium sets.
	B. Conventional	Heavy circular sets on 2 ft (0.6 m) center. Rock load 2.0 to 2.8*B*.	Pattern, 3 ft (0.6 m) center.	6 in. (150 mm) or more on whole section. Combine with medium to heavy sets.
Very poor (Squeezing or swelling)	A. Boring machine	Very heavy circular sets on 2 ft (0.6 m) center. Rock load up to 250 ft (76 m).	Pattern, 2 to 3 ft (0.6 + 0.9 m) center.	6 in. (150 mm) or more on whole section. Combine with heavy sets.
	B. Conventional	Very heavy circular sets on 2 ft (0.6 m) center. Rock load up to 250 ft (76 m). Heavy sets.	Pattern, 2 to 3 ft (0.6 + 0.9 m) center.	6 in. (150 mm) or more on whole section. Combine with heavy sets.

Source: Adopted from Deere et al., 1969.

Note:* In good and excellent quality rock, the support requirement will in general be minimal but will be dependent upon joint geometry, tunnel diameter, and relative orientations of joints and tunnel.

** Lagging requirements will usually be zero in excellent rock and will range from up to 25% in good rock to 100% in very poor rock.

⁺ Mesh requirements usually will be zero in excellent rock and will range from occasional mesh (or straps) in good rock to 100% mesh in very poor rock.

*** *B* = tunnel width

Table 24.1.9. Summary of Important Rock Classifications

Classification (Approximate Time of Development)	Data Required	Methods Used	Support Recommended
Rock Quality Designation, RQD (1963)	• Core Measurement	• Index	• Steel sets (rock bolts shotcrete)
Rock Structure Rating, RSR (1972)	• Rock type • Joints • Groundwater	• Numerical ratings	• Steel sets (rock bolts, shotcrete)
Geomechanics, RMR (1973)	• RQD • Core strength • Joints • Groundwater	• Numerical ratings	• Shotcrete • Rock bolts • Cast concrete
NGI, Q-System (1974)	• RQD • Joints • Groundwater • Stress factor	• Numerical ratings	• Shotcrete • Rock bolts • Cast concrete

2. The structures are usually lined for their ultimate use. Control of line, grade, and geometry may be of greater impact.

24.1.5.3 Rock Tunnel Support and Lining

Rock tunnels are very similar to mining excavations except for two general, and sometimes critical, differences. First, their design lifetime is generally on the order of a century, or at least a few decades. Second, many civil tunnels will be continuously inhabited throughout their lifetimes by the general public. For these reasons, the final lining may have design considerations that are more conservative than those for mines.

Initial supports for civil tunnels are basically the same as for mines. In general, they consist of the use (individually or in combination) of rock bolts, dowels, shotcrete, wire mesh, and steel sets. The general goal of an initial support system is to provide the following characteristics:

1. Absence of rock falls.
2. Control of stress-induced local instability.
3. Absence of mass instability by "reinforcing" the rock, i.e., encouraging the rock to support itself.
4. Restriction of loosening.
5. Control of water.

Design of the initial support system follows techniques described elsewhere in this *Handbook* (see Chapter 10.5) and generally takes one of three general approaches:

1. Empirical:
   a. RQD index.
   b. Q rating.
   c. RSR rating.
   d. RMR rating.
2. Pseudo-theoretical:
   a. arch theory.
   b. beam theory.
   c. block configuration.
3. Theoretical:
   a. elastic.
   b. inelastic.
   c. block and ubiquitous joint analysis.
   d. finite element analysis.
   e. beam element analysis.

Final supports for such tunnels often consist primarily of a lining system to meet user requirements such as esthetics, or waterflow or airflow characteristics. Often this is the minimum lining that can be constructed because the initial support system usually reinforces, knits together, or confines the rock in such a manner that the rock actually provides the majority of its own initial support. It is also assumed that the final lining (usually shotcrete or cast-in-place concrete) is properly and tightly placed, and that any voids between lining and rock are properly and thoroughly contact-grouted. Under these conditions, there can be only very nominal (if any) movement or loosening of rock around the opening. With this minimal movement, the effectiveness of the self-supporting "rock arch" is not diminished, and the final lining likely will never be required to support other than nominal loosening loads.

As a check on the lining capacity, the writers suggest an application of (1) an analysis that accounts for lining-rock interaction (such as beam elements or finite elements) with no more than one diameter of rock load, and (2) an analysis that considers the lining resistance in shear to any wedges that were identified during construction and that could punch through as the result of the long-term deterioration of initial support elements.

24.1.6 MIXED-FACE TUNNELS

Tunnels are frequently driven in two vastly different materials, usually soft ground and overlying rock. Such tunnels are called *mixed-face tunnels* and present very special challenges to the tunnel engineer. This may be especially true when the mixed face consists of special cases such as flowing silt overlying stiff clay.

In these cases, tunnel design and construction are usually driven by the softer of the two materials, but ignoring the harder material can lead to serious problems. For example, if a TBM is used, it will normally have to be selected to cut the softer material and to provide maximum control for that same material. The frequent result is that the excavation process may tend to hang up on the harder material because it is more difficult to cut. Under these conditions, unless good control is maintained, the machine may continue to excavate and remove the soft material at a rate greater than that of the hard material. The obvious result is disproportionate removal or loss of control of the soft ground that can lead to (1) instability at the face, and (2) an increase in lost ground at the face over that attained in a full face of either of the materials. Similar problems occur with the work crews:

1. A soft-ground crew may not have the experience and skills necessary to shift to a drill and blast operation for part of the face.
2. Similarly, a hard-rock crew (especially if drill and blast techniques are being used) likely may not have the skills to provide the support and controlled excavation necessary to maintain stability of the soft-ground portion of the face.

Mixed-face tunneling can involve considerations of ground combinations, groundwater conditions, and construction methods that are legion and certainly beyond the scope of this *Handbook.* When mixed-face conditions are expected, the designers should start by studying Wilbur (1982) and Terzaghi (1977).

24.1.7 SPECIAL CONSIDERATIONS

The authors wish to acknowledge that a myriad of special considerations may arise regarding a civil tunneling design and construction project. Some have been discussed above, but these are by no means all that might be encountered. The following list is representative of other special considerations that may arise. Readers are referred to Terzaghi (1977), Bickel and Kuesel (1982), and O'Rourke (1984) for initial study, but as with mixed-face tunneling, for most cases it is recommended that the consulting services of a specialist be obtained for developing or reviewing the final design and/or the construction procedures:

1. Adjacent, crossed, stacked, or intersecting tunnels.
2. Compressed air for control of water or for face stability.
3. Controlled blasting.
4. Emergency provisions.
5. Freezing to control water or to improve stability.
6. Geophysical exploration techniques.
7. Groundwater control (during construction and permanently).
8. Grouting to control water, improve stability, or control settlement:
   a. chemical.
   b. cement.
   c. compaction.
   d. consolidation.
9. Hazardous conditions (e.g., chemicals, gas).
10. Muck removal and hoisting.

11. Other support construction types (spiling, breasting, forepoling, New Austrian Tunneling Method (NATM).
12. Seismic conditions.
13. Shield/tunneling machine design.
14. Special structures (e.g., portals, caverns, sunken tubes, nearby or intersecting tunnels).
15. Squeezing or swelling ground.
16. Ventilation, lighting, and other utilities.
17. Work and safety rules.

24.1.8 UNIQUE CHARACTERISTICS OF CIVIL TUNNELS

The segments above apply, for the most part at least, to all civil tunnels. There are, in addition, characteristics of each type of civil tunnel that make it unique from other tunnels. The following paragraphs summarize some of those unique characteristics for reference. Users of this *Handbook* should be aware that these unique characteristics may have a significant impact on the design and construction of civil tunnels. Their impact, in fact, may be far greater than might be inferred from the simple listing of characteristics.

24.1.8.1 Rail Transit Tunnels

1. Rail alignment restrictions require tight adherence to tunnel alignment, especially the horizontal alignment.
2. Ventilation requirements for the operating life of the tunnels must be considered.
3. Infiltration of water, gases, or ground contaminants must be tightly controlled.
4. Cathodic protection is required where direct current (dc) is used for train power.
5. Vibration impacts of transit trains on nearby structures may require use of isolation materials or special mounting pads.
6. The requirement to minimize damage from earthquakes must be considered in some areas.
7. Tunnel lining may require multiple embedment of utilities.
8. Cross passages/emergency access must be provided.

24.1.8.2 Highway Tunnels

1. Ventilation requirements for the life of the tunnels must be considered.
2. Infiltration of water, gases, or ground contamination must be tightly controlled.
3. The requirement to minimize damage from earthquakes must be considered in some areas.
4. Cross passages/emergency access must be provided.

24.1.8.3 Water Supply Tunnels

1. Alignment requirements can often be less severe than for vehicle tunnels.
2. Design for interior pressure may be required.
3. Smoothness of the final lining will impact flow capacity and may thus be a concern.
4. Embedments are usually not required.

24.1.8.4 Wastewater Tunnels

1. Horizontal alignment requirements may not be as severe, but vertical alignment requirements may be important to avoid low spots.
2. Exfiltration concerns may be important.
3. Settlement concerns may occur where tunnels must be shallow.
4. Embedments are usually not required.
5. Special designs are often required for inlet drop structures.
6. Corrosion-resistant liners are often required for long-term durability in acid-producing conditions.

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Chapter 24.2
STORAGE AND POWER GENERATION

DAVID C. WILLETT

24.2.1 INTRODUCTION

Large permanent openings in rock are used for a number of functions associated with the bulk storage of solids, liquids, and gases, and with the generation and storage of electrical energy. Openings for these purposes, although large, are generally excavated by "conventional" civil engineering (full-face or heading and bench methods) as opposed to mining methods such as block caving, stoping, etc. Occasionally, if the configuration of the mined space is appropriate, the opportunity is taken to make use of openings, previously excavated for the extraction of minerals, for storage or other purposes.

In North America in general, other than for a few hydroelectric facilities, and some relatively small oil and gas storages, there are very few examples of the excavation of caverns in rock for purposes other than mineral extraction. In other parts of the world, however, particularly in Europe, and to a lesser extent in Asia, the use of caverns excavated in rock for purposes ranging from waste disposal and bulk storage, to recreational and even residential, is widely accepted and economically viable.

In North America, relatively small (up to 100,000 yd^3 or 76,500 m^3) caverns have been excavated for the bulk storage of crude oil, and a number of high vapor-pressure hydrocarbons such as propane, butane, propylene, and ethane, in rock types ranging from shales to limestones and granite. In Europe, however, primarily Scandinavia, extensive use has been made of large caverns (sometimes in excess of 1,000,000 yd^3 or 765,000 m^3) excavated for the specific purpose of storing crude oil and refined oil products. The currently largest single excavated underground oil storage facility (32 million bbl or 5 M m^3) is located in Korea. However, a major component (73 million bbl or 11.6 M m^3) of the US Strategic Petroleum Reserve program of crude oil storage is located in a previously excavated salt mine at Weeks Island, LA.

Temporary storage of flood waters is provided in deep tunnels in a number of North American cities, including notably Chicago (the Tunnel and Reservoir Plan, or TARP project) and, more recently, Milwaukee.

In the Kansas City area, extensive use is made of underground space created by many excavations for limestone. Uses of this space include bulk storage of solids, cold and refrigerated storage, manufacturing, general warehousing, and even retail office facilities.

Turning to energy production and storage, several major hydroelectric generating stations have been located underground in rock caverns in North America, including the Raccoon Mountain facility of TVA, the 5000-MW Churchill Falls power plant in Labrador, and the 3000-MW "Le Grande 2" (LG-2) plant near James Bay in the province of Quebec. These openings constitute some of the largest permanent man-made excavations in existence. At LG-2, for instance, the powerhouse cavern measures 80 ft span by 120 ft high by 1000 ft long (25 by 37 by 305 m).

Studies have also been made of the feasibility of locating nuclear generating plants underground; in Europe, a number of small plants were constructed in underground locations in the 1960s and early 1970s.

A number of bulk energy storage concepts currently being studied envisage the use of large underground openings. These concepts include Superconducting Magnetic Energy Storage (SMES), Compressed Air Energy Storage (CAES), and Underground Pumped Hydro (UPH).

The development and use of underground openings for these nonmining purposes is discussed in the following sections.

24.2.2 MINED OPENINGS FOR STORAGE PURPOSES

24.2.2.1 Gases

Although, as noted above, a number of small underground storages for gases such as natural gas have been developed in North America by conventional excavation techniques, by far the largest underground gas storage facilities are located either in previously depleted natural gas fields or in caverns that have been solution-mined in bedded or domed salt deposits. The technology associated with these major storage facilities is not within the purview of this chapter. For caverns in salt, see Chapter 15.3.

Typically, a mined facility will be located in a stratum of shale with bulk permeabilities in the region of 10^{-6} mm/sec in order to limit the leakage of gas from the cavern. Access for excavation is obtained through a small (3- to 5-ft-, or 1- to 2-m-diameter) shaft drilled from the surface and steel-lined. The caverns are generally excavated by the room and pillar method (Chapter 18.1)

In general, the containment of gas in unlined rock caverns is achieved by the low permeability of the surrounding rock mass. However, in fractured rock, containment can also be achieved provided that the necessary groundwater gradient is present. Suh et al. (1986) proposes that the depth of groundwater at the ceiling of the cavern should be greater than the maximum pressure head within the cavern plus a factor (ranging up to about 20% of the pressure head) depending upon the shape of the cavern. Although a number of proposals (e.g., Sagefors and Svemar, 1986) for methods by which liquefied natural gas (LNG) might be stored underground have been put forward, no successful prototype installation has yet been achieved.

24.2.2.2 Liquids

PURPOSE-EXCAVATED CAVERNS. Although at the time of writing, no major (larger than 1,000,000 bbl, or 159,000 M m^3) underground storage facilities for hydrocarbons have been developed by excavation in North America, many large storages have been constructed elsewhere, particularly in Europe. The liquid is stored in unlined caverns, usually excavated in jointed crystalline rock. Containment of the stored liquid is ensured either by locating the excavated cavern at such a depth below the natural groundwater level that the natural inflow of groundwater into the cavern prevents the seepage of the contained liquid or vapor, or by supplementing the groundwater pressure by means of a water injection system located in separate tunnels above the crown of the storage cavern.

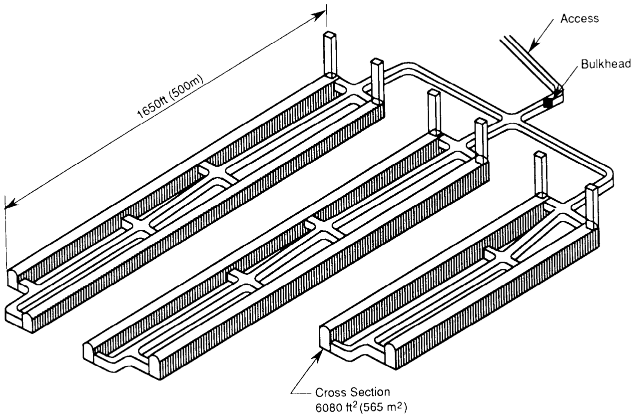

Fig. 24.2.1. Configuration of a typical large oil storage cavern in hard rock.

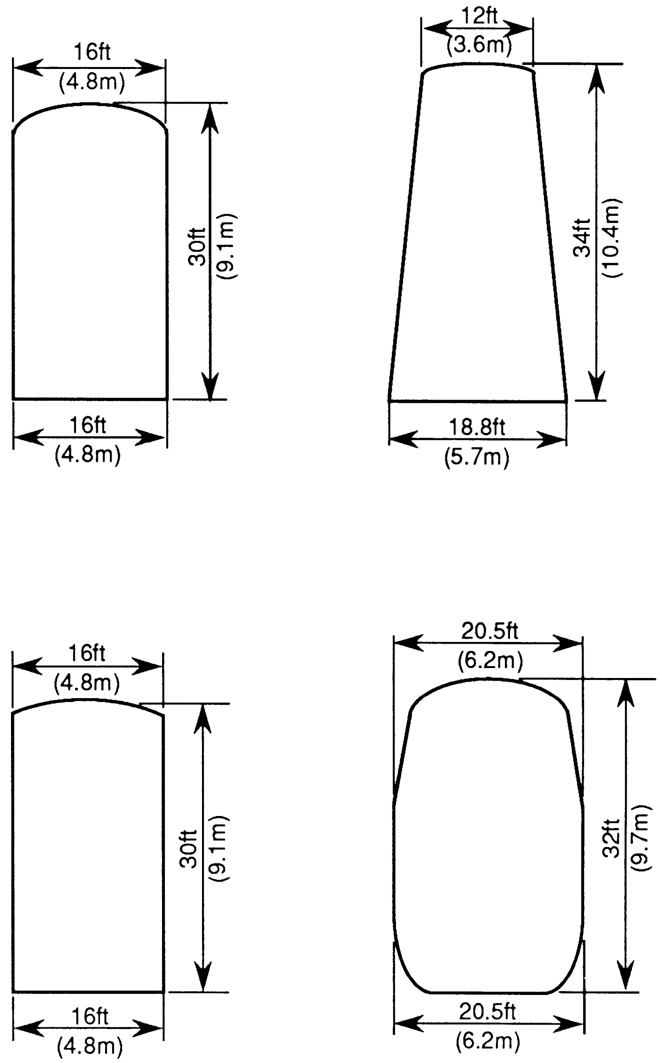

Fig. 24.2.2. Typical oil storage cavern cross sections.

Water entering the cavern collects as a "water bed" at the bottom of the stored oil and can be handled in one of three ways: fixed, variable level, or dry, depending upon the type of oil being stored and the relative economy of alternatives. Problems have been encountered with microbial growths at the oil-water interface with some types of oil or oil product.

A diagram of a typical large oil storage facility (Mongstad, Norway) is presented in Fig. 24.2.1. The principal characteristics of the caverns are shown as follows (Hosolen, 1986):

Volume stored:	8.17 million bbl (1.3 M m^3)
Cavern dimensions:	55 ft wide, 100 ft high, up to 1550 ft long (18 m wide, 33 m high, up to 510 m long)
Depth below ground level:	approximately 170 ft (55 m)

Overall Configuration—Depending upon the total volume to be stored, storage space is generally provided by one or more parallel unlined caverns excavated at some depth below rock surface in a competent rock mass. A key feature of the initial site selection will be the identification and characteristic of a suitable stratum; this will almost inevitably require a trade-off between the technical, logistical, economic, and environmental considerations.

Cavern Dimensions—Depending to some extent on the volume to be stored, but also upon the characteristics of the host rock, cavern sizes may range from 30 ft (10 m) span to as much as 60 or 70 ft (18 or 21 m) span and 100 or more ft (30 m) high. Typical cross sections of a number of caverns constructed by Neste Oy (Finland) are shown in Fig. 24.2.2. Walls of caverns intended for crude oil storage are often inclined inward towards the top to ensure that falling blocks do not strike sparks which might ignite the cavern contents.

Support—The basic principle in the design of reinforcement for any large underground opening is to help the rock mass support itself. As the majority of oil storage caverns excavated to date have been sited in hard competent crystalline rocks, reinforcement has been largely confined to rock bolting in the roof and walls; typically, bolt lengths in the roof have ranged from 0.15 to 0.30 times the cavern span, and in the walls from 0.10 to 0.20 times the cavern height (Johansson, 1986).

Supplementary reinforcement is often provided by pneumatically applied concrete (PAC); this is generally used either (1) un-reinforced to handle small potential fallouts or (2) reinforced with wire mesh in conjunction with rock bolts to provide more significant structural support in areas requiring arch continuity or wall support. More recently, fibre-reinforced PAC has been used with considerable success.

Containment—As discussed earlier (24.2.2.1), containment of both petroleum liquids and gases/vapors in caverns in jointed rock is ensured by providing a flow of groundwater into the cavern. In general, this can be obtained by locating the cavern so that the roof is at a depth below groundwater level, at least equal to the vapor pressure within the cavern. In situations where this is not feasible, artificial stimulation of flow into the cavern can be provided by the installation of either pressurized water injection tunnels or drillholes from the surface above the cavern.

Conversely, if groundwater conditions are such that there is excessive leakage into caverns, which in turn can lead either to excessive costs for pumping, or drainage of the water seal above the cavern, it may be necessary to embark on a program of grouting to reduce the flow.

Access—Access for construction is generally provided by means of an inclined road tunnel driven from the surface. When construction is completed, the tunnel is bulkheaded off, and such access as is required for the operation of the facility is obtained through vertical shafts to the surface, as shown in Fig. 24.2.1.

Construction—Caverns are generally constructed by conventional heading and bench methods. As noted above, roof and wall support during construction and for the permanent installation is provided by pneumatically applied concrete (PAC) if required, supplemented by wire mesh and rock bolts.

In planning the excavation, a number of considerations that are unique to oil storage come into play; these include

1. Maintenance of the groundwater level above the caverns
2. Scheduling the excavation to allow safe commissioning of the caverns finished first
3. Minimizing the size and extent of construction access that cannot be subsequently used for oil storage
4. Making best use of the opportunity for multiple face operation, with optimum scheduling of the various drilling, charging, scaling, and mucking crews

General—More than 100 major oil storage facilities of the type described above have been constructed in Europe. With the more recent oil surpluses, it is interesting to note that at least one of these storages has been converted to the storage of coal (Anon., 1988b).

USE OF EXISTING CAVERNS FOR STORAGE OF LIQUIDS. In North America, the only major underground oil storage facility in a mined cavern is located at Weeks Island, LA, in the previously excavated Weeks Island salt mine. This mine, located between 500 ft (150 m) and 850 ft (260 m) below ground level, was excavated in the period between 1910 and the present day. It was identified in 1976 as being suitable for conversion to oil storage, and the requisite modifications were made to facilitate the installation of the necessary equipment for the filling and withdrawal of crude oil.

In selecting the Weeks Island mine for oil storage, more than 3000 mines in the United States were examined for this possible use. The Weeks Island mine was selected from a short list of ten possible sites, principally on the basis of its capacity, its containment security, and its location relative to the major oil pipelines near the Gulf Coast.

24.2.2.3 Storage of Solids Underground

With the exception of radioactive wastes, discussed in Chapter 24.3, virtually all underground solids storage facilities in North America are based on the use of space previously excavated for the extraction of minerals. In a number of instances, however, these facilities have been extended for the specific purpose of providing additional space, the production of minerals—usually limestone—being of secondary importance.

Following procedures normally adopted for the extraction of limestone underground, these facilities have generally been excavated by the room and pillar technique. The majority are located in the general area of Kansas City; a listing and brief description of the principal currently operating facilities is set out in Table 24.2.1. Note that many are used not only for storage but also for a range of commercial and industrial purposes that benefit from the isolation and insulation provided by the underground environment, One of the largest refrigerated storage facilities in North America is located underground in this area.

In Norway, environmental pressures are such that a company involved in the processing of zinc ores has been required to develop an underground facility for the disposal of the solid wastes (Aarvoll et al., 1986). The caverns are excavated in a coarse-grained granitic gneiss.

24.2.3 MINED OPENINGS FOR POWER GENERATION FACILITIES

24.2.3.1 General

Worldwide, power plants for hydroelectric generation facilities have been constructed underground since the late 19th century. In North America, to date, 11 hydropower plants are located underground (Table 24.2.2).

The principal reason for locating these plants underground has been, in most cases, economy; the underground location is less costly than a similar scheme with the power plant at the surface. At the same time, a number of other benefits are often obtained, including aesthetic, environmental, and improved safety from external forces.

24.2.3.2 Siting

The decision to locate a hydroelectric power plant underground, and to select the most appropriate site having done so, requires the evaluation of a range of factors related primarily to the topography of the area, the distance between the planned headrace and tailrace, the need for surge chambers upstream or downstream of the power plant, and the depth of rock cover available along the planned route of the tunnels. Within this context, the known or projected geotechnical conditions related to rock quality, the presence of faults or other significant discontinuities and groundwater must be factored into the equation. The ultimate decision as to whether to site the power plant underground will, under most circumstances, be based on an assessment of the relative economy of the surface vs. underground siting.

From the geotechnical standpoint, the key factors relate to the ability of the proposed host rock to accommodate/support the excavation of a cavern, or a series of caverns, of a size required to house the necessary mechanical and electrical equipment without the need to provide costly and/or time-consuming support measures.

Guidance as to the quality of rock mass in which it is proposed to locate the required caverns can, in the first instance, be obtained through the application of one or more of the rock mass classification systems developed by a number of investigators (Hoek and Brown, 1982) (see also Chapter 24.1 and Section 10).

Development of the necessary input data upon which to base the rock-mass classification assessment will depend initially upon research of available literature and previous exploratory work. As the decision regarding the optimum location of the facility is firmed up, the need will arise to undertake exploratory work specifically aimed at providing the requisite data for the assessment and subsequently detailed design of the cavern.

24.2.3.3 Exploration

The optimum techniques to be adopted for obtaining the required data and, in fact, the type and extent of data required, will be different for each and every site. Many of the techniques used in conventional mining exploration (Chapters 4.3 and 4.4) are entirely appropriate for use here. Bearing in mind, however, that the ultimate objective is to develop permanent stable caverns, primary emphasis will be placed on obtaining information about the properties of the rock that affect its structural performance.

Typically, exploratory activities might include the following:

1. Initial assessment of regional geology using available geologic mapping, air photos, previous drilling records, photo interpretations, etc.

Table 24.2.1. Major US Underground Storage Sites

Location	Name	Description
Chester, IL	Midland Terminal	90 miles (145 km) south of St. Louis on the Mississippi River. 25-ft (8-m) ceiling heights, perfectly dry.
Lexington, KY	4th Street Tobacco Warehouse	Developing downtown. Will have document storage capability. Ceiling heights 25 to 50 ft (8 to 15 m). Active mine—aggregate.
Louisville, KY	Derby Developing Co.	Well located, in a near-suburban area. Expressway and zoo over the top. 28- to 110-ft (9- to 34-m) ceiling heights. 115 acres (47 hectares).
Carthage, MO	Carthage Marble Co. (CMC Inc.) PO Box 718 Carthage, MO 64836 (417) 358-2145	
Independence, MO	GeoSpace, Inc. 601 South M-291 Highway Independence, MO 64015	GeoSpace Executive Park is a subsurface development in the Metropolitan Kansas City area that will have 7.6 million net usable ft^2 (706,040 m^2) for office, manufacturing and warehouse, truck docks, parking, railways, roadways, and other common area. Slightly over one-half million net usable ft^2 (46,450 m^2) is currently developed in office, manufacturing and warehouse space.
Kansas City, MO	Grasis Corp.	Near Hunt Midwest. 250,000 ft^2, (23,225 m^2) total, 130,000 ft^2 (12,077 m^2) developed. Manufacturing facility with outside buildings, 25-ft (8-m) ceiling heights. Modified specifications.
Kansas City, MO	Hunt Midwest Enterprises, Inc. 8300 N.E. Underground Drive Kansas City, MO 64161	Land development company coordinating surface and underground development of 2500-acre (1012-hectare) master-planned community. Ceiling heights of 12 to 16 ft (4 to 5 m). Weatherproof docks. Rail served by Burlington Northern. Automatic fire protection.
Kansas City, MO	Prairie Mining	Unlimited space, in most affluent and prestigious area of Johnson County.
Neosko, MO	Southwest Lime Co. P.O. Box 816 Neosko, MO 64850 (417) 451-6806	
Springfield, MO	Griesemer Stone Company Rt. 2 Box 52 Springfield, MO 65802	1.2 million ft^2 (111,480 m^2) net usable space, sprinkled, dehumidified, lighting 10 foot-candles, 25-ft (8-m) stacking heights. Seven "buildings" 110,000 ft^2 (10,219 m^2)–340,000 ft^2 (31,586 m^2) in size, some cold storage. Rail to 6 of buildings, 30 boxcars. Truck docks (100 total), 24-hour security guard, dry storage for manufacturing (food, spices, Zenith TV picture tubes). 140,000 ft^2 (13,006 m^2) new space mined every day. Pillars on 80-ft (24-m) centers.
Warrenton, MO	Missouri Limestone	
Wampum, PA	Wampum Industrial Facility P.O. Box 486 Wampum, PA 16157 (412) 535-4308	Located on the Pennsylvania Turnpike near Pittsburgh. Over 2½ million ft^2 (232,250 m^2) of underground space offers clients current storage technology for all types of materials including classified documents, computer tapes and microfilm. 30-ft (9-m) diameter pillars. Unlimited floor loads. Specialized environmental control.

Table 24.2.2. Underground Hydroelectric Power Plants in North America

Name	State	Owner
United States		
Bear Swamp	MA	New England Power Company
Helms	CA	Pacific Gas and Electric
Northfield Mountain	MA	New England Electric Systems
Raccoon Mountain	TN	TVA
Boundary	WA	Seattle Power and Light
Bad Creek	NC	Duke Power
Balsam Meadow	CA	Southern California Edison
Canada		
Bersimis	PQ	Hydro Quebec
Chutes des Passes	PQ	Alcan
Churchill Falls	LB	Churchill Falls (Labrador) Corp.
Mica Creek	BC	BC Hydro
LeGrande II	PQ	Hydro Quebec

2. Detailed surface mapping, including identification of rock outcrops, strike, dip, jointing of exposed rock, etc., possibly supplemented by trenching to expose additional rock surfaces.

3. Geophysical exploration: of value primarily for extending information about near-surface features; can include such techniques as seismic refraction and reflection surveys, gravity meter, etc.

4. Diamond drilling: the staple and basic method of obtaining information about rock at depth. Key data to be obtained from the core include stratigraphy, jointing/fracturing, orientation, rock types, etc. May be supplemented by a variety of "down-hole" exploratory techniques, including borehole photography (still or video) and in situ stress determination by hydrofracturing techniques.

5. Exploratory adits: normally only developed once the location and orientation of the caverns have been essentially established. Provides first-hand information about in situ rock conditions and allows full-scale tests to determine parameters such as in-situ stress and bulk modulus of elasticity to be conducted.

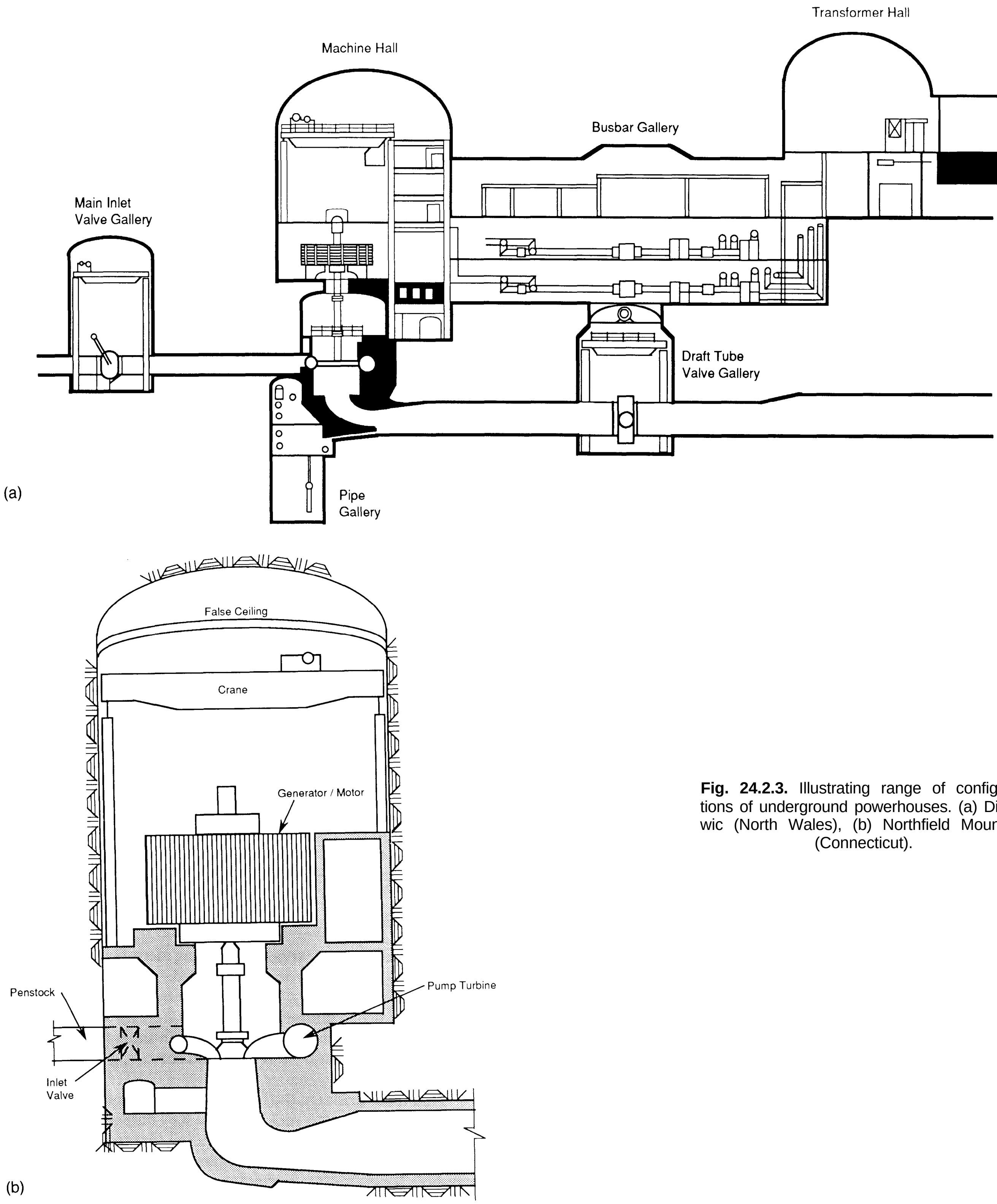

Fig. 24.2.3. Illustrating range of configurations of underground powerhouses. (a) Dinorwic (North Wales), (b) Northfield Mountain (Connecticut).

Typical adit dimensions range from 8 by 8 ft (2.4 by 2.4 m) up to 12 by 12 ft (4 by 4 m). Advantages of adits include the facility to allow bidders to obtain first-hand appreciation of rock conditions. Adits may also provide a useful first-stage for the excavation of large chambers.

24.2.3.4 Design

OVERALL ARRANGEMENT. Caverns for a hydroelectric facility must be designed to house the various mechanical and electrical components needed to generate and transmit electrical

Table 24.2.3. World's Largest Span Underground Power Plants

	Project	Country	Units (No.)	Units (MW)	Cavern Size Width (m)	Cavern Size Height (m)	Cavern Size Length (m)	Ground Cover (m)	Rock Type
1	Cirata	Indonesia	4	125	35	49.5	253	100	Pyroclastic, some shale
2	Imaichi	Japan	3	350	33.5	51	160	400	Sandstone, slate
3	Waldek II	Germany	2	239	33.5	40	106	260	Graywacke, sandstone, shale
4	Veytaux-Hongrin	Switzerland	4	60	30.5	26.8	137.5	330	Fractured limestone, schist
5	Fadalto	Italy	2	120	29.6	57.3	69.6	shallow	Sound limestone
6	El Cajon	Honduras	4	80	29.5	49	110	200	Karstic limestone
7	Grimsel II	Switzerland	4	75	29	19	140	200	Hard gneiss
8	Cabora Bassa	Mozambique	5	415	28.9	56	220	140	Unknown
9	Ferrera	Switzerland	3	62	29	25	143	150	Hard gneiss
10	Coo-Trois Ponts	Belgium	6	125	27.5	40	130	75	Thin beds phyllites, poor shales
11	Robiei	Switzerland	4	41	27	28.7	75	NA	Conglomerate, gneiss
12	Shintakasegawa	Japan	4	336	27	54.5	163	250	Good quality granites
13	LaGrande 2	Canada	16	333	26.5	47.3	483	100	Granitic gneiss w/ diorite
14	Helms	USA	3	350	25.3	38.1	102.4	370	Grandiorite
15	Kemano	Canada	16	100	25	42.7	213.4	430	Granitic, diorite
16	Motezic	France	4	228	25	42.5	145	380	High-quality granite
17	Churchill Falls	Canada	11	475	24.7	44.8	296.3	300	Gneiss, diorite, syenite
18	Dinorwic	UK	6	300	24.5	52.2	180.3	300	Good-quality slate
19	Mica	Canada	6	435	24.4	44.2	237.2	220	Schist, interbedded gneiss
20	El Toro	Chile	4	100	24.4	40	103	NA	Grandiorite

Source: Anon., 1989a.
NA = not available.
Conversion factor: 1 ft = 0.3048 m.

energy. Selection of the optimum configuration will require consideration of the structural characteristics of the rock mass in relation to the required dimensions and alternative arrangements available for the various components. Reference should be made to standard hydroelectric texts for a description of the dimensions and required configuration of the mechanical and electrical components. In many cases, the selection of the optimum arrangement revolves around the relative economy of housing all the components in one large cavern as opposed to using two or more smaller caverns. In making this selection, the cost of excavation and support of the alternative chamber sizes and the interaction of stresses around adjacent caverns require a good knowledge of the rock-mass characteristics provided by the exploration program discussed above.

The principal components to be housed within the overall power facility include:

1. Inlet valves (if required).
2. Turbine and generator (or pump/turbine and motor/generator).
3. Draft-tube gates or valves.
4. Transformers (if located underground).
5. Various mechanical and electrical subsystems.

Cross sections of two existing power plants illustrating the range of possible configurations are presented in Fig. 24.2.3 (a) and (b).

CAVERN DIMENSIONS. The required dimensions of the power plant cavern will depend primarily upon the size of the equipment to be installed and/or upon the number of the components to be located within the cavern. Space must be provided around the principal components to allow them to be installed and removed for maintenance and to accommodate the various electrical and mechanical subsystems, such as governors, excitation equipment, compressors, pumps, etc. Typically, spans in the range between 60 and 80 ft (18 and 24 m) are most frequently encountered; a listing of some of the world's largest span underground power plants is presented in Table 24.2.3.

SUPPORT. Because of the wide range of geotechnical conditions encountered in the construction of underground power plants, cavern support measures have ranged from a minimum of pattern rock bolting in high-quality rock to a full concrete lining with prestressed tendons in poor-quality rock. Fig. 24.2.4 illustrates a variety of cavern support measures in the following rock types:

1. Fortuna: Andesite, agglomerate and tuff.
2. Ohkawachi: Porphyyrite.

24.2.3.5 Construction

As with large oil storage caverns, the heading and bench method of excavation is generally adopted. This allows easier access for the installation of roof support measures during the initial stage of the work. The top heading may be excavated full-face or, for large spans, using a central subheading with side slashes. The sequence of excavation of the Churchill Falls caverns is shown in Fig. 24.2.5.

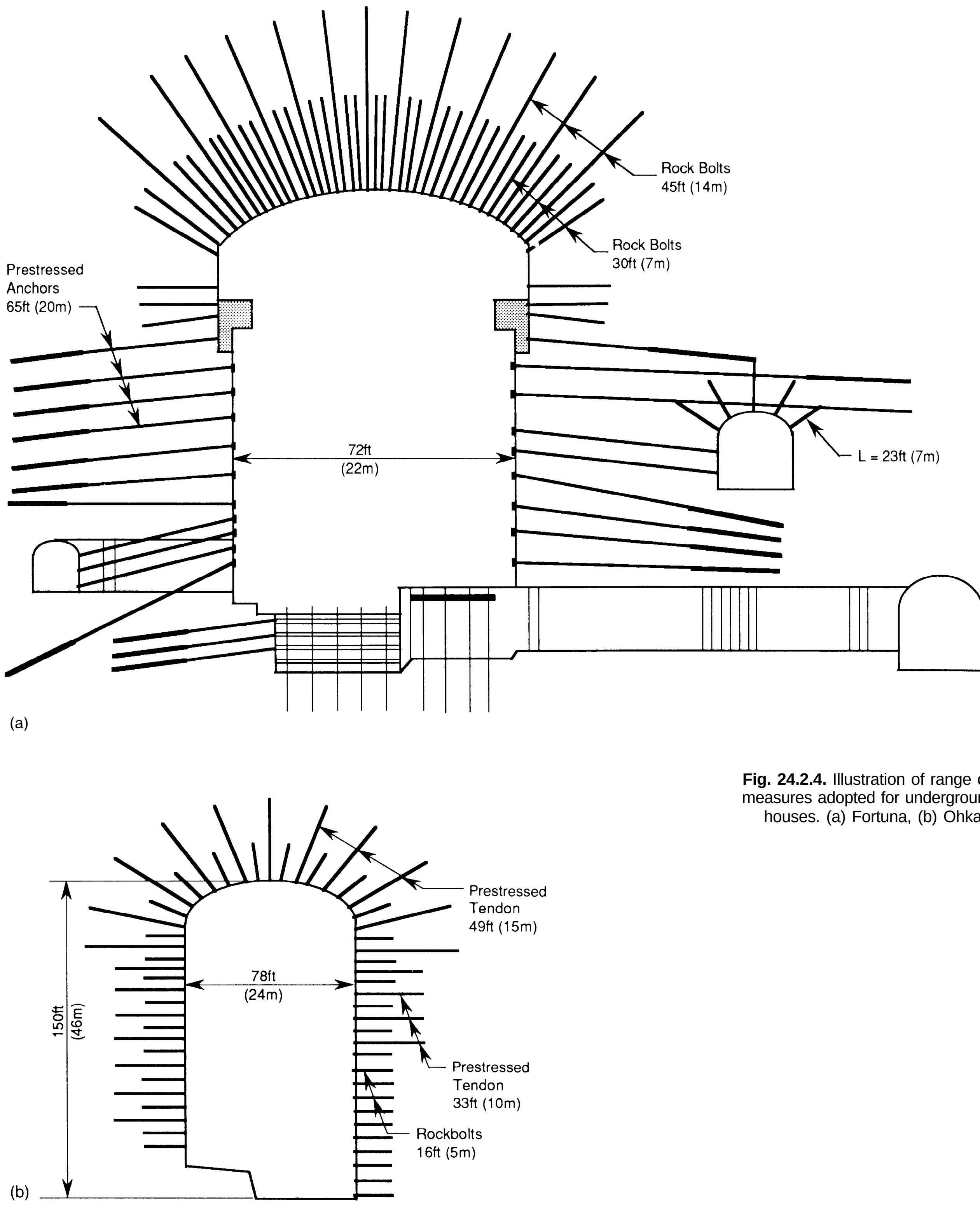

Fig. 24.2.4. Illustration of range of support measures adopted for underground power-houses. (a) Fortuna, (b) Ohkawachi.

To minimize damage and overbreak at the cavern walls, smooth blasting and presplitting techniques are used; the bench excavation may be drilled horizontally or vertically, depending upon the logistics of the access and spoil removal system.

24.2.4 ADVANCED ENERGY STORAGE CONCEPTS IN ROCK CAVERNS

In the period following the oil crisis of the mid- to late-1970s, a number of conceptual developments were proposed for

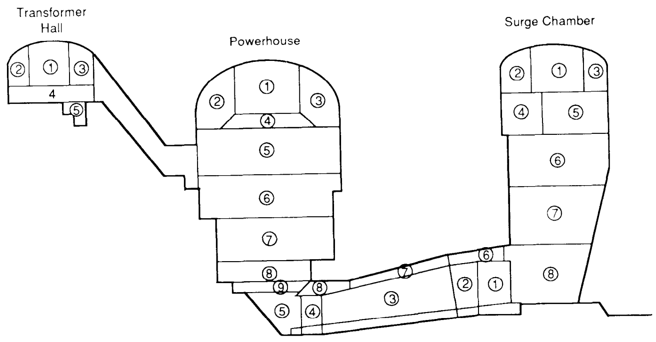

Fig. 24.2.5. Excavation sequence for Churchill Fails underground powerhouse.

the bulk storage of electrical energy in a number of forms; several of these concepts involved the use of large chambers excavated in rock. These included:

1. Underground pumped hydroelectric storage (UPH).
2. Compressed air energy storage (CAES).
3. Superconducting electromagnetic storage (SMES).

At the time of writing, none of these concepts has been developed into a prototype facility, but all three are still under active consideration.

24.2.4.1 Underground Pumped Hydroelectric Storage (UPH)

A definitive study of this concept was presented in a report commissioned by the Electric Power Research Institute (EPRI) (Anon., 1981). The concept proposes the use of caverns excavated at depths up to 5000 ft (1520 m) below ground level to be used as the lower reservoir of a pumped hydroelectric generating facility as illustrated in Fig. 24.2.6. The required excavated volume of these caverns could range up to 5 or even 10 million yd^3 (3.8 or 7.6 M m^3), depending upon the energy storage required. The EPRI report proposes that the reservoir caverns be excavated in the form of long parallel chambers, the size and configuration depending upon the characteristics of the host rock.

Although no UPH facilities have been constructed to date, a proposed facility near Akron, OH is noteworthy in that the use of an existing mine (the Norton limestone mine) is proposed to serve as the lower reservoir (Anon., 1989b), Fig. 24.2.7.

24.2.4.2 Compressed Air Energy Storage (CAES)

The compressed air energy storage concept involves the bulk storage of energy in the form of compressed air. The feasibility of the concept has been established by a prototype facility constructed at Huntorff in western Germany in 1979. At this facility, the air is stored at a pressure of 1000 psi (6.9 MPa) in two caverns, solution mined in salt. A similar facility has recently been completed by Alabama Electric Cooperative (Anon., 1988a).

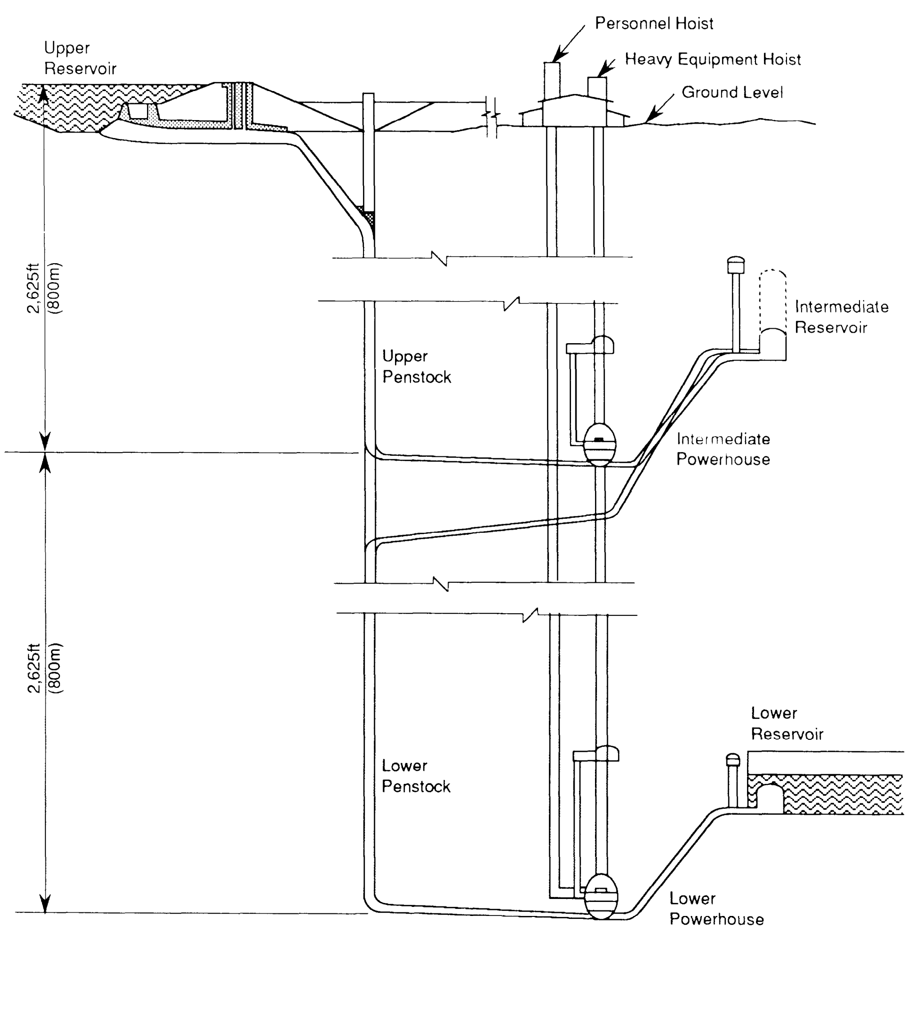

Fig. 24.2.6. Underground pumped storage concept.

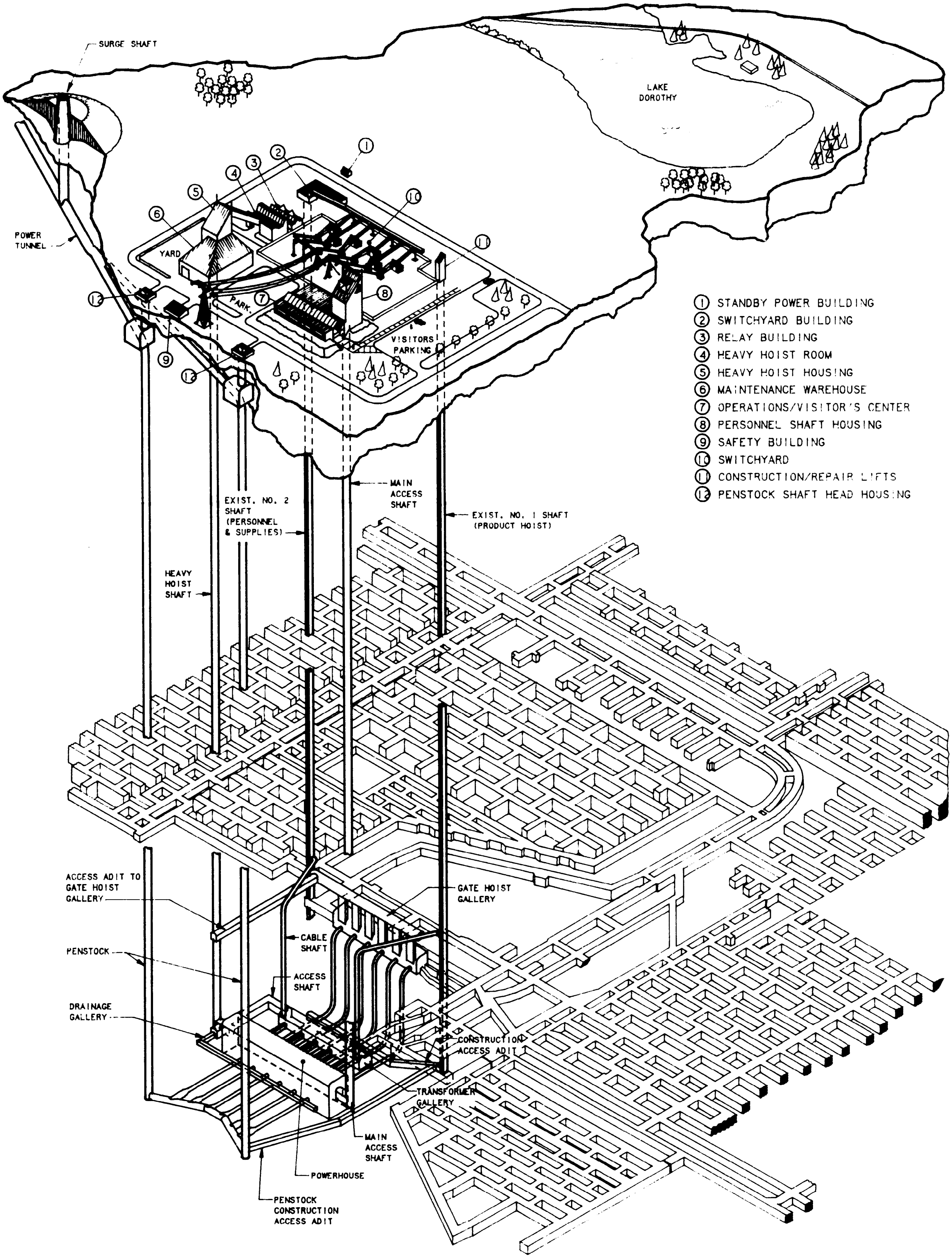

Fig. 24.2.7. Proposed underground pumped storage facility using an existing mine.

Studies of alternative means of storing large volumes of compressed air have included consideration of excavated caverns in hard rock (Anon., 1981). Possible cavern configurations have included large parallel caverns similar to those proposed for UPH and caverns excavated by room and pillar methods (Anon., 1988b).

24.2.4.3 Superconducting Electromagnetic Energy Storage (SMES)

Superconducting electromagnetic energy storage involves the storage of electrical energy in a closed electrical circuit cooled to subcritical temperatures. Laboratory scale tests have demonstrated the feasibility of the concept, and conceptual designs have been prepared for installations capable of storing up to 1000 MWh of electrical energy.

Because of the major forces, low temperatures, and high electrical fields involved in the concept, an underground location in rock has been proposed as the most appropriate for this development. No test or prototype installations have yet been constructed, although it is understood that the concept is under active consideration for the provision of the high-powered lasers required for the Strategic Defense Initiative (SDI) program.

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Chapter 24.3
WASTE REPOSITORIES

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The discussion that follows addresses the basic concepts of a high-level radioactive waste repository, the fundamental rationale behind the concepts, the criteria covering its performance, and the considerations that must be addressed in the creation of such a facility. A *repository* is a complex facility that must accommodate a variety of factors, including those defined by the natural laws, requirements of regulatory agencies, and requirements established by political compromise. At the time of the preparation of this chapter, no high-level radioactive waste repository had either been formally designed or built. Many concepts are still being analyzed, evaluated, and debated; the technical basis for the details of the design has not been formally or authoritatively established. Conceptual designs (Anon., 1979a, 1979b, 1981, 1983, 1984, 1986a, 1986b; MacDougall et al., 1987) have been prepared for repositories in which various approaches have been examined in addressing some of the issues; however, the final or acceptable approach has not been formally adopted.

Because of the early stages of development of the repository concept, this chapter discusses the concepts of a repository and the factors affecting the analysis, evaluation, and selection of approaches to design of the underground facility and its component parts.

For aspects of cavern construction and radioactive waste disposal, see Chapter 15.3.1.

24.3.1 INTRODUCTION

24.3.1.1 Repository Concept

The method selected (Anon., 1980) for the disposal of high-level radioactive waste in the United States is disposal in deep geologic formations. The concept is to encapsulate the high-level radioactive waste as a stable, solid form in a high-quality, hermetically sealed canister and place it deep within a stable geologic formation somewhere between 1000 to 3000 ft (300 to 900 m) below the surface. A conceptual illustration of a geologic repository is presented in Fig. 24.3.1.

High-level radioactive waste is a material that is known to be hazardous to man, and the hazardous nature will remain for an extended period of time. The hazard is associated with the radiation emitted (alpha, beta, or gamma) from the material. The degree of hazard generally decreases with time, and the rate of decrease is determined by the half-life of the radionuclides involved. Half-lives vary from a few seconds to 10s and 100s of thousands of years. Because of the initial high radiation intensity of the hazardous material and the duration of its persistence, a method of disposal of high integrity, beyond that of all previous waste disposal options, had to be identified. The method would require structures with a stability far beyond that of recorded history (approximately 10,000 years). The only structures that satisfied the time stability criterion were geologic structures, some of which have been in place for a period ranging from 250 million to a billion years. A geologic repository was conceived as a means of utilizing the earth as a physical barrier to separate the radioactive materials from man and his environment; the stability of the formation was to ensure its long-term effectiveness. A geologic repository is expected to contain a nominal quantity of radioactive waste of 77,000 tons (70,000t) in the form of spent fuel elements (Anon., 1982). When the basic constraints of underground facility design and requirements for dissipating the thermal energy from the radioactive decay are considered, a repository is likely to cover an area of approximately 2 to 3 mi^2 (5 to 8 km^2) underground. A repository, when constructed, will be a room and pillar configuration, involving on the order of 190 mi (300 km) of passageways and drifts. To construct such a facility, it is estimated that in excess of 20 million tons of rock (roughly 300 million ft^3 or 8.5 M m^3) may have to be removed and a large percentage subsequently replaced in the passageways and other openings.

The design of a high-level radioactive waste repository will have to satisfy numerous objectives, often conflicting, which will have to be met on a simultaneous basis. The life of a repository can be divided into two distinct time periods, and the different functions that are required to be performed during each will have a significant impact on the philosophy and approach to its design. The two time periods are the operational phase (30 to 100 years in duration) and the isolation phase (10,000 years in duration); discussion of the functions of the repository system in this paper is divided into these two categories.

During the operational phase, the objectives of the design will be to facilitate (1) rock removal required in creating the labyrinth of passageways, (2) the emplacement of radioactive waste, (3) operational safety for underground workers, (4) control of the air supply, and (5) protection against the release of radioactive materials, if there should be an accident of a radiological nature. In the discussion that follows, activities occurring in the operational phase that can have significant impact on the performance of the repository system in the isolation phase are generally presented in the discussion regarding the isolation phase. For the isolation phase, the design objectives will be to control the conditions around the package containing the radioactive waste, if necessary, to control and divert water away from the waste package, to control and dissipate decay heat from the waste package, to prevent perturbation of the initial geologic structure in which the repository is located, to provide effective man-made barriers to replace the natural rock removed in order to emplace the waste, and to prevent releases and transport of radioactive materials.

Greater detail is provided in subsequent segments regarding each of these design requirements.

24.3.1.2 Relationship Between Repository and the Natural Environment

The natural environment referenced here includes the geologic, hydrologic, and geochemical conditions that exist in the earth before and during the time the repository is performing its function. Geologic disposal was selected as the method for disposing of high-level radioactive waste because of the need for a situation where long-term stability (250,000 to 1,000,000 years) could be expected. Geologic formations, known to be stable for

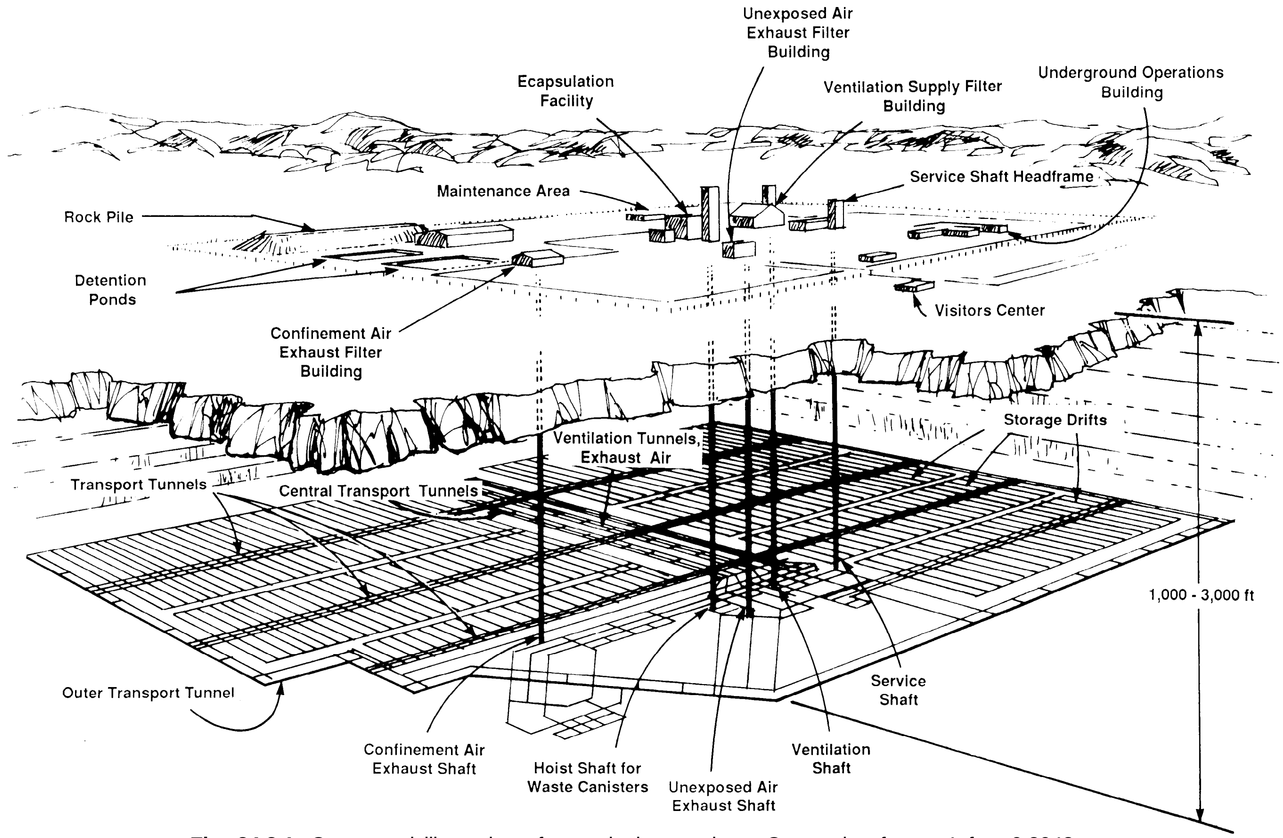


Fig. 24.3.1. Conceptual illustration of a geologic repository. Conversion factor: 1 ft = 0.3048 m.

periods of tens of millions of years, would serve as a significant physical barrier that would all but guarantee permanent separation of the solid, stable radioactive waste form from man's environment. While many naturally occurring materials can serve as effective barriers, the selection of a geologic medium would also be controlled primarily by its engineering and structural properties; a labyrinth of passageways over a 2- to 3-mi^2 (5- to 8-km^2) area would have to be designed and constructed to be structurally stable for a period of approximately 100 years.

Hydrologic conditions in the natural environment are also critical in that dissolution of or leaching of radionuclides from the waste form and the subsequent transport by water is considered the primary means by which radioactive materials can be brought back to the human environment. From this perspective, the smaller the amount of water available to interact with the waste, and the more benign from a chemical perspective, the better the situation. Because of this basic fact, there is a significant benefit to be obtained by placing a repository in the unsaturated (vadose) zone as opposed to the saturated zone (below the water table). The major difficulty is finding a location where the water table is 1000 to 3000 ft (300 to 900 m) below the surface. A repository placed below the water table, after the repository is closed and sealed, will eventually be resaturated with water. In such cases, there would be a significant quantity of water available for direct contact with the waste package. Depending on the depth below the water table, the hydrostatic head could be great enough to prevent the water from converting to vapor at 212°F (100°C), or lower temperatures at higher elevations, and thereby place superheated water in contact with the waste package. A situation with elevated temperatures and an electrolyte in direct contact with the waste package fosters a condition where corrosion rates and degradation mechanisms affecting the waste package are accelerated. If the waste package were to breach, then leach rates and dissolution rates for the waste form would be expected to be greater at elevated temperatures.

Repositories located in the vadose zone have a distinct physical advantage (Winograd, 1981) because the formation containing the repository will have significantly less free water and will not likely fill with water after the repository is sealed. (It is possible that a repository located in the vadose zone would flood after closure; however, it would require some significant force in nature to change the existing condition with regard to the water table.) Consequently, if there is no pool of standing water (natural electrolyte) in contact with the waste package, the potential for aqueous corrosion is significantly reduced. If the package were to be breached, then there is no significant quantity of water available to enter the waste package and leach or dissolve the waste form. With a paucity of water, there is substantially less capability for significant mass transport of radionuclides away from their point of emplacement. The possibility of transport of gas-phase radionuclides may, however, be greater in the vadose zone. While there may be many arguments about the details of hydrologic models for the unsaturated zone, the most elementary test is to consider the mass transport requirements to move radionuclides in the repository and the amount of water that will be available over time to affect the transport. While the velocity of water movement is often considered to be a critical factor (Anon., 1989a), the flux of water is a more fundamental parameter to consider in terms of the potential for mass transport of radionuclides.

The geochemistry of the environment can play an important role in the effectiveness of the repository, and there are numerous points that need to be considered. The chemical nature of the formation can affect the chemistry of the groundwater moving through it. With all other factors held constant, the less reactive the constituents in the water, the better the situation. Water will serve as an electrolyte, and chemical constituents in the water will affect how aggressive the electrolyte will be in the degradation of materials introduced into the system by man. The geochemistry of the geologic medium in and surrounding the repository proper is important from a chemical stability viewpoint. Heat from radioactive decay of the waste will raise the temperature of the rock, and elevated temperatures can lead to critical changes in the rock, such as driving off water of hydration from minerals in the formation. Change in chemical composition can produce physical and structural changes in the rock, primarily change in volume. Because of the volume of rock involved (several cubic miles or cubic kilometers), particular attention needs to be given to the active physical changes that might occur.

Geochemical conditions can also be positive in contributing to the isolation of waste. Certain minerals found in nature, for example, zeolites, have a chemical affinity for chemical constituents dissolved in water. When the water containing the dissolved constituents passes through solids containing zeolites, the constituents in the water will be removed from solution (Sherwood et al., 1975). If the dissolved constituents happen to be radionuclides, the zeolites would remove them from the water, thereby keeping radionuclides near the site of emplacement and promoting isolation.

The natural environment is critical to the repository in its ability to accomplish the objective for emplacing the waste in an operationally safe manner and for isolating the waste. When one considers the time required for waste isolation, the natural physical processes that will be in action continuously, and the size of the various components in the repository, as well as the repository, it is clear that the natural environment will dominate the control of release in comparison to the systems and materials introduced by man. The point to be made is that selecting the site for the repository with the proper combination of naturally occurring conditions is the most crucial part of the effort in creating a repository. A repository is significantly different from any other nuclear facility in that the major physical element of the repository that is, the site itself is in place at the outset of the effort, and man will have no role in putting the containment structure in place. It is essential to remember that when building a facility for man's use underground, it is not created by putting the materials structure in place but by taking material away.

24.3.1.3 Differences Between a Repository and Other Underground Structures

A repository will be a labyrinth of passageways covering an area on the order of 3 mi^2 (8 km^2) and, as such, might appear to be similar to other underground structures. While other underground facilities may be of similar size and extent, the purpose, requirements, and conditions of the repository make it significantly different. The penetration into the ground (shafts) and passageways are to support a primary objective, a method of getting the waste from the surface to its final resting place. Because of the long-term presence of the waste, the regulations governing its disposal strongly emphasize the requirement that no significant damage be done to the formation so that no unknown or difficult-to-seal pathway between the waste and the human environment be created.

The facility will also be different in that the waste introduced will have a gradual release of thermal energy (St. John, 1987) from radioactive decay. This will result in the ambient temperature of the repository increasing over a 200- to 400-yr period and then slowly returning to the natural ambient temperature after a total time of 400 to 800 yr. Depending on the location and timing of the waste emplacement, the temperature profile and associated thermal expansion and nonuniform stress fields could produce impacts on near-term operation and long-term isolation (Nitao, 1988). Temperature gradients can produce movement of water in the rock formation, especially for repositories in the vadose zone; controlling water movement in the repository is important and will require a good knowledge of the time, temperature, and location relationships. Temperature in the repository could eventually become a problem from an operational perspective. While a waste package may release energy at only a 1-kW rate (comparable to ten 100-W light bulbs in an emplacement hole) over a period of 40 years, the temperature of the rock surrounding the waste that absorbed the energy can reach a point where the conditions in the passageways will create a situation in which it is uncomfortable or difficult for humans to work. Consequently, the design must be able to accommodate innovative ways to keep the temperature of the work areas cool at a time when the repository is ready for closure. This is the timeframe when closure operations or retrieval operations would be implemented.

The long-term structural stability of the passageways is desired to support operations, yet the mined opening represents a preferential pathway for the movement of water underground. To avoid collapse of the openings, with a concomitant increase in the volume of formation in which water can move freely, it is considered prudent to backfill all passageways and openings as the surest method of preventing collapse. It is expected that the rock removed from the passages would be crushed, classified, and reinserted into the passageways. The crushed rock might be mixed with other materials to improve the ability of the backfilled passageways to resist deformation and to improve the ability of the backfill to retard the movement of water and radionuelides.

Shafts entering the earth will have to be backfilled and sealed. They clearly represent preferential pathways between the waste and human environment. The primary concern will be to seal the openings to ensure that water will not enter the repository and disrupt its initial condition.

24.3.1.4 Impact of Regulatory Approval Process on Design Approach

Before the repository can be built, a significant regulatory review and approval process must be completed. The specific technical requirements that must be satisfied are outlined in the *Code of Federal Regulations* (CFR) issued by the Nuclear Regulatory Commission (NRC). The technical regulations are presented in 10 CFR Part 60 (Anon., 1989a) while the details of the licensing procedure are outlined in 10 CFR Part 2 (Anon., 1990). Detailed information about the site, obtained from the site characterization program, along with a safety analysis report specified in 10 CFR 60.21 which examines and analyzes the repository design and its impact on short-term containment and long-term waste isolation characteristics, must first be reviewed and accepted by the NRC staff. Once accepted by the NRC staff, the case can be presented and argued before an Atomic Safety and Licensing Board. The Atomic Safety and Licensing Board currently consists of three judges, an environmental scientist, a physical scientist/engineer, and a lawyer. While this is the current structure, there are expectations the structure of the Board would change to reflect a broader scope of technical expertise. The panel constitutes an administrative law court, and the review is conducted within the context of a formal trial. In addition to the regulator (NRC) and the license applicant (Department of Energy or DOE), other parties, such as the representatives of the state in which the repository is to be built and whoever else is granted status by the Board, can be admitted to the proceedings as full participants with their lawyers challenging the technical data and analyses, and cross-examining scientists and engineers presenting the information about, and argument for, the proposed repository.

The objective of the Licensing Board is to identify issues of contention and hear the evidence in favor of and opposing the proposal, and to determine if the radiological hazard of the project falls within the approved guidelines. Because of the nature of the repository, that is, the length of time (10,000 yr) over which the repository will have to function, the basis on which to decide the safety aspects become somewhat diffuse. The rules established by the NRC under which to grant a construction authorization are embodied in 10 CFR Part 60. This rule identifies four major performance objectives pertinent to the postclosure performance of the repository that must be demonstrated in the regulatory review:

1. The prewaste emplacement groundwater travel time between the disturbed zone (around the repository) to the accessible environment must be greater than 1000 years.

2. The engineered barrier system cannot, on an annual basis, release more than 1 part in 100,000 of the radionuclides present at 1000 years after emplacement.

3. The waste package must provide essentially complete containment for 300 to 1000 years.

4. The release of radionuclides to the accessible environment over a 10,000-yr period cannot exceed the release limits (total curies) specified in 40 CFR Part 191 (a standard established by the Environmental Protection Agency) (Anon., 1986c).

These performance objectives appear to be well stated and quantifiable standards of judgment on which to base a decision. Fundamental questions pertain to the data set required, the analyses of that data set that will be considered acceptable, and the level of uncertainty or error bounds associated with the determinations that will be considered acceptable. It is evident from the time frames identified in each performance objective that they cannot be measured in the time proposed for the research and development or, for that matter, in a single human lifetime. Therefore, the basic parameters involved in each performance objective for a proposed site or design will have to be calculated based on models. In such a situation, there will be four different kinds of models involved in these estimates, which include a geologic model of the earth including its stratigraphy and spatial structure, a hydrologic model covering the location and movements of the groundwater, the physical models of the natural phenomena occurring at that location in that natural environment, and the mathematical models that reflect the various physical realities that allow for quantitative estimates of effects. Because of the complexity of these models, the spatial framework in which they must apply, and the temporal scale they must cover, it will be difficult, if not impossible, to validate the models in the traditional sense. Consequently, there will be substantial uncertainty as to the actual performance that will be achieved.

With this inherent uncertainty, there exists a fundamental motivation on the part of the regulators, who must approve the proposed site and the design of the underground facility within the context of "reasonable assurance" that it will perform, to compensate in order to ensure achievement of performance objectives. This approach is generally referred to as "conservatism." It is manifested in the assumptions that must be made about the conditions at the site and the bases established for the facility design. The basic philosophy that is manifest is the requirement to use conservative assumptions in models, considered to be inherently uncertain, in order to provide a reasonable picture of the likely outcome.

One approach of this kind has been utilized in the regulatory arena. Licensing of nuclear production and utilization facilities in the United States requires that building and system structures response to seismic events be analyzed (Anon., 1989b; Anon., 1977). Tectonic behavior of the earth is a physical process that cannot be confidently predicted or experimentally simulated, yet the understanding of the process and effects is important for predicting structural response to the seismic stimulus. Accordingly, regulatory precedence has been set in probabilistic prediction of tectonic behavior.

It is expected that this precedent will be applied to both the man-made surface facilities (100 years) and the underground facility (10,000 years). The longer time frames will lead to reduced confidence in the certainty in the predicted behavior. Through a combination of conservatism in the engineering design, conservatism in the prediction of the occurrence of tectonic phenomena, and conservatism in the magnitude of the tectonic phenomena, the applicant (DOE) must provide the requisite assurance to the NRC staff and the Licensing Board that the design is safe. Analysis (Subramanian et al., 1989; Koplik et al., 1982) of the design parameters and their impacts on safety have shown that there is minor reduction in risk to the public's health and safety by designing to extreme conservation and, under general circumstances, this avenue of design would not normally be considered prudent. However, in licensing cases, confidence in the safety of the design over long time periods is the imperative, and the enhanced conservatism is considered necessary to achieve "reasonable assurance."

24.3.1.5 Specific Factors That Must Be Considered

The design of the underground portion of the repository must accommodate a number of unusual design requirements. Three specific requirements include two different time frames for which the repository must perform, the need to rely on highly variable inhomogeneous material to provide for isolation of the waste, and the requirement to have a defense-in-depth concept by using multiple barriers to the release of radionuclides.

The repository will have to function in two different time frames. These time frames have been previously referred to as the operational phase and the isolation phase. The operational phase is the time when the repository is open; construction of new emplacement locations is underway; and receipt of and emplacing waste, executing the period of performance confirmation testing, and the closure, backfilling, and sealing phase is taking place. A point should be made about the process of constructing the repository. Initially, the shafts or ramps, underground maintenance areas, major passageways, and a limited number of emplacement areas will be constructed. It is estimated that it will require a minimum of five years to establish this underground base of operation. Once operational, both emplacement and construction activities will occur simultaneously in parts of the repository that are physically separated.

The time expected to emplace 77,000 tons (70,000 t) of high-level waste at the rate of 3300 tons (3000 t) per year is approximately 25 years (Anon., 1985; MacDougall et al., 1987). The NRC can require that the repository remain open for 50 years after the first waste is emplaced for the purpose of performance confirmation. At the end of the performance confirmation effort, the NRC must decide to either backfill and seal the repository or to retrieve the waste. For the retrieval operation, NRC regulations require that it be possible to retrieve all the waste in a time frame approximately equal to that required to emplace it. It is expected that the time to backfill all openings and passageways is less than the time to retrieve the waste (Anon., 1989a). Consequently, from a design perspective, the openings underground must be structurally stable for a time period of 80 to 100 years.

The second time frame that must be considered is the isolation period, which is nominally 10,000 years as specified in regulations (40 CFR Part 191). A longer time frame could be dictated by the decay of the radioactive materials in the repository.[a] The effectiveness of the repository in isolating the waste is most dependent on the intrinsic characteristics of the geologic, hydrologic, and geochemical environment in which the repository is located. The major point to be considered in the repository design is that the properties and character of the site, relied upon to meet the performance objectives, are not impacted by the construction, operation, or content of the repository to the extent that they no longer provide the means to isolate waste.

The decision as to the site where the repository will be located is the single most important decision that is made with respect to a repository. The site should be selected on the intrinsic characteristics of the geologic, hydrologic, and geochemical environment. This needs to be done in the framework of the regional geologic setting that provides some overall indication of long-term stability; stability that has to be judged in the context of tens of thousands to hundreds of thousands of years. The geologic medium is critical in that it becomes the engineering material in which the shafts and openings are constructed, and this will require stability in terms of a century. This judgment will have to be made in the estimates of conditions over a 2- to 3-mi^2 (5 to 8-km^2) area. The hydrologic regime is critical since water is the major means, in a stable geologic formation, by which radionuclides, contained in the solid waste, could possibly return to the human environment once buried 1000 ft (300 m) below the surface. Geochemical characteristics are critical since they will influence how aggressive a chemical environment the waste packages and waste form would have to resist over the required time and influence how closely temperatures would have to be controlled during the first 400 to 800 years the repository is in existence. The selection of the site establishes the boundary conditions that the designers must contend with in their attempt to establish a configuration for the underground facility to ensure that performance objectives can be achieved.

Site selection will affect the success of the efforts in a number of different ways. An ideal situation would be if the site were relatively elementary in structure and benign in character. Simplicity of geologic structure is desirable to limit the difficulty of characterizing the site and establishing an authoritative basis of "reasonable assurance" for the regulators and the Licensing Board that the site is understood well enough to approve its use.

A basic decision needs to be made regarding the hydrologic regime; that is, should the horizon in which the underground facility is located be above the water table (in the vadose zone) or below the water table. If the repository is located below the water table, the repository can be expected to resaturate with water after the repository is closed (unless it is in a sealed formation such as bedded salt). This is likely to create a more aggressive and corrosive environment for the first 400 to 800 years while the thermal energy is released with the waste package in intimate and constant contact with a hot or warm electrolyte. Corrosion and leaching potential in an electrolyte is likely to lead to failure of the waste package, leaching of radionuclides from the waste form, and transport of radionuclides in the abundant groundwater at a more rapid rate than if the groundwater were not present in a saturated condition. If the hydraulic head in the horizon is high, the temperature of the water in contact with the waste package could go above the boiling point of water, 212°F (100°C), due to the increased pressure.

Selection of a horizon in the vadose zone offers a potential for a more benign situation. In the vadose zone, there is far less potential for the waste package or waste form to be in constant contact with standing water (electrolyte). The waste package is more likely to experience conditions of low-temperature oxidation in a humid environment, as opposed to aqueous corrosion in an aggressive electrolyte. While water may come in contact on a periodic basis with the waste form, there is not likely to be continuous contact and flow of water to carry leached radionuclides downward through the repository and the geologic formation into the standing groundwater. The magnitude and nature of the flux of water past the waste package will determine the potential for significant mass transport of radionuclides out of the repository.

While the selection of a site can be the predominant step in creating a repository capable of achieving the desired performance objectives, it is recommended by the regulators that engineering measures, often called "engineered barriers," be utilized to help achieve performance objectives. The utilization of multiple barriers, both natural and engineered, is expected as part of an overall waste isolation system. The factor that is essential to keep in mind is the function the barrier is supposed to perform and the effectiveness of that barrier over the time frame that is involved. The current concept (Anon., 1988) is that the barriers to the movement of radionuclides from the point of waste emplacement include the waste form (for spent fuel, it is a uranium dioxide fuel pellet and the metal tubing in which it is sealed), the waste canister (a hermetically sealed metal canister), a backfill between the waste canister and the geologic medium, and the geologic medium between the backfill and the human environment. The first three items noted above would be considered an engineered barrier system in that they have been conceived, designed, and fabricated by man. All three of these items have been designed to retard the release of radionuclides. The effectiveness of each barrier needs to be evaluated on a case-by-case basis considering the materials involved, the hydrologic environment, and the chemical environment in which it must function. In general, three factors above all will control the effectiveness of the barrier: (1) the chemical affinity of the barrier material for the radionuclides, (2) the permeability of the barrier material to water, and (3) the physical size or dimension of the pathway through the barrier. Before final decisions are made, the true value of the engineered barrier needs to be evaluated in the context of the ability of the natural materials within the repository itself to retard the movement of radionuclides.

[a] A spent fuel element, depending on type—boiling water reactor (BWR) or pressure water reactor (PWR)—can have plutonium contents of 0.5 to 0.8% by weight. For a repository containing 77,000 tons (70,000 t) of heavy metal (in the form of spent fuel elements), a total plutonium content could range from 385 to 615 tons (350 to 560 t). The plutonium would eventually disappear by radioactive decay; the rule of thumb is that in 10 half-lives, the amount of the radionuclide would decrease by a factor of 1000. For Pu-239, with a half-life of 24,000 years, the 615 tons (560 t) would decrease to 1100 lb (500 kg) in 240,000 years. After 480,000 years, 20 half-lives, the total plutonium content in the repository (2 to 3 mi^2 or 5 to 8 km^2) would be 1.1 lb (500 g).

24.3.1.6 Basic Philosophical Principles that Affect the Approach to Repository Design

A number of basic considerations will affect the approach to the design of the underground portions of the repository. The most important consideration in the approach to the design is the fact that the earth, as it exists with its intrinsic positive and negative features, is the structure and material that will contain and isolate the waste. Characteristics that affect specific details of the design will be the strength of the rock and its time-dependent mechanical properties, the thermal conductivity, the chemical reactivity and stability of the rock, the hydrologic characteristics of the site and the region in which it is located, and the geologic structure of the formation. The designer will have had no influence in the creation of the geologic regime under consideration and will be able to do little to change the conditions in the geologic regime in which the wastes will be placed. The existing conditions are all that is available and the designer must work with them, utilizing them to achieve the isolation requirements.

The second point that needs to be considered is that the "underground facility" (the opening) itself will probably contribute little to achieve isolation. The most important feature in the design is the configuration of the facility, which will affect the distribution of the waste and, in effect, the thermal energy release distribution. The design and construction of the underground facility will do little to enhance the intrinsic ability of the earth to isolate the waste, but if not properly executed, the design and construction practices can affect the isolation in a negative manner. The primary function of the underground openings is to allow the waste to be moved from the surface to the point of final emplacement. They also provide for the removal of freshly mined rock and provide for ventilation. The concern for negative impacts focuses on several factors, including the creation of preferential pathways for radionuclides to move out of the repository and the potential for introduction of massive quantities of materials that could increase the mobility of the radionuclides. For example, organic materials are known to form chemically complex molecules with plutonium and, by doing so, to reduce the adsorption capability of the natural media for plutonium in aqueous solution (Sanchez et al., 1985). Organic materials in the form of diesel fuel, incompletely burned organics in engine exhaust, rubber on the ground from tire wear, and epoxy anchors in rock bolts are likely to be introduced. Care will be required in evaluating their effect and estimating the mass of materials introduced. The significance of the threat will be affected by the law of mass action and the tonnage of these materials that might be left in the repository when it is closed.

The third point is that the waste going into the repository is not a factor from a chemical compatibility viewpoint, but it can affect the design from a physical effects perspective. The effect is in the release of thermal energy from the decay of the radioactive materials. The release of the thermal energy into the geologic formation results in a temperature increase. The bulk of the thermal energy is released within the first 200 to 400 years. The control of the maximum increase in the temperature experienced by the geologic formation will depend on the energy content of the waste and the configuration of emplacement points. The energy content of the waste, especially in the form of spent fuel, will depend on the burn-up level experienced by the fuel while in the reactor and the aging time of the waste before emplacement. The maximum temperature desired in the repository will depend on a number of factors, including whether the repository is above or below the water table, the chemical reactivity and stability of the geologic medium, the coefficient of thermal expansion of the geologic medium, and the time to heat the surface of the passageways to a temperature in which it will be difficult for humans to work.

24.3.2 BASIC FUNCTIONS OF A REPOSITORY

24.3.2.1 Operational Phase

LIFETIME OF THE FACILITY. The lifetime of the repository facility is dictated by requirements to receive a specified amount of high-level radioactive waste and by the approach taken to the requirement to maintain the option to retrieve the waste. The requirement to maintain the option to retrieve the waste is discussed in detail in a subsequent section. It is sufficient to note here that there is a requirement to maintain the option to retrieve for a period of time and, if necessary, execute retrieval in a time period that is approximately equal to that required to emplace the waste. The Nuclear Waste Policy Act of 1982 (NWPA) (Anon., 1982) specifies that the first repository cannot receive more than the equivalent of 77,000 tons (70,000 t) prior to the development of a second repository. This requirement does not preclude the placement of more than 77,000 tons (70,000 t) in a repository; it only specifies that there be two repositories if more than 77,000 tons (70,000 t) is to be placed in one. The currently anticipated receipt rates for waste at the repository are specified by the DOE (Anon., 1985). The reference indicates that the repository should be accepting waste on a schedule that would result in a total of 77,000 tons (70,000 t) in a period of 25 yr. The schedule is predicated on a phased repository construction that would permit receipt of small amounts of waste in the first four years of operation. For purposes of this discussion, a period of 25 yr for emplacement of 77,000 tons (70,000 t) provides a reasonable estimate of the basic time period that is relevant to discussion of facility lifetime.

The required facility lifetime is based on the 25-yr time for receipt of waste and on the approach selected to maintain the option to retrieve the waste. For a repository for which it was planned to receive no more than 77,000 tons (70,000 t) of waste material, an estimate of the lifetime can be calculated in the following manner (Flores, 1986). The retrievability requirement of 10 CFR Part 60 states that the option to retrieve the wastes must be maintained throughout the period during which wastes are being emplaced and thereafter until the completion of the performance confirmation program and NRC review of the information obtained from that program. The requirement on retrievability in 10 CFR Part 60 indicates that to satisfy this objective, the Geologic Repository Operations Area (GROA) shall be designed so that any or all of the emplaced wastes can be retrieved on a reasonable schedule starting at any time up to 50 yr after waste emplacement operations are initiated, unless the Commission approves a different schedule. This different schedule would need to be established on a case-by-case basis consistent with the emplacement schedule and the planned performance confirmation program. Accordingly, activities related to retrieval could be occurring roughly 100 yr after the initiation of waste emplacement. Some flexibility in lifetime requirements could be accommodated within this schedule, however. Because the 50-yr requirement of 10 CFR Part 60 is twice as long as the period in which wastes are to be emplaced in this example, 25 yr will be devoted to simply monitoring the emplaced wastes and confirming the performance of the repository. A prudent engineering approach to retrieval could involve partial decommissioning of facilities, particularly those on the surface, and subsequent rehabilitation of the facilities important to the retrieval operations should it become necessary.

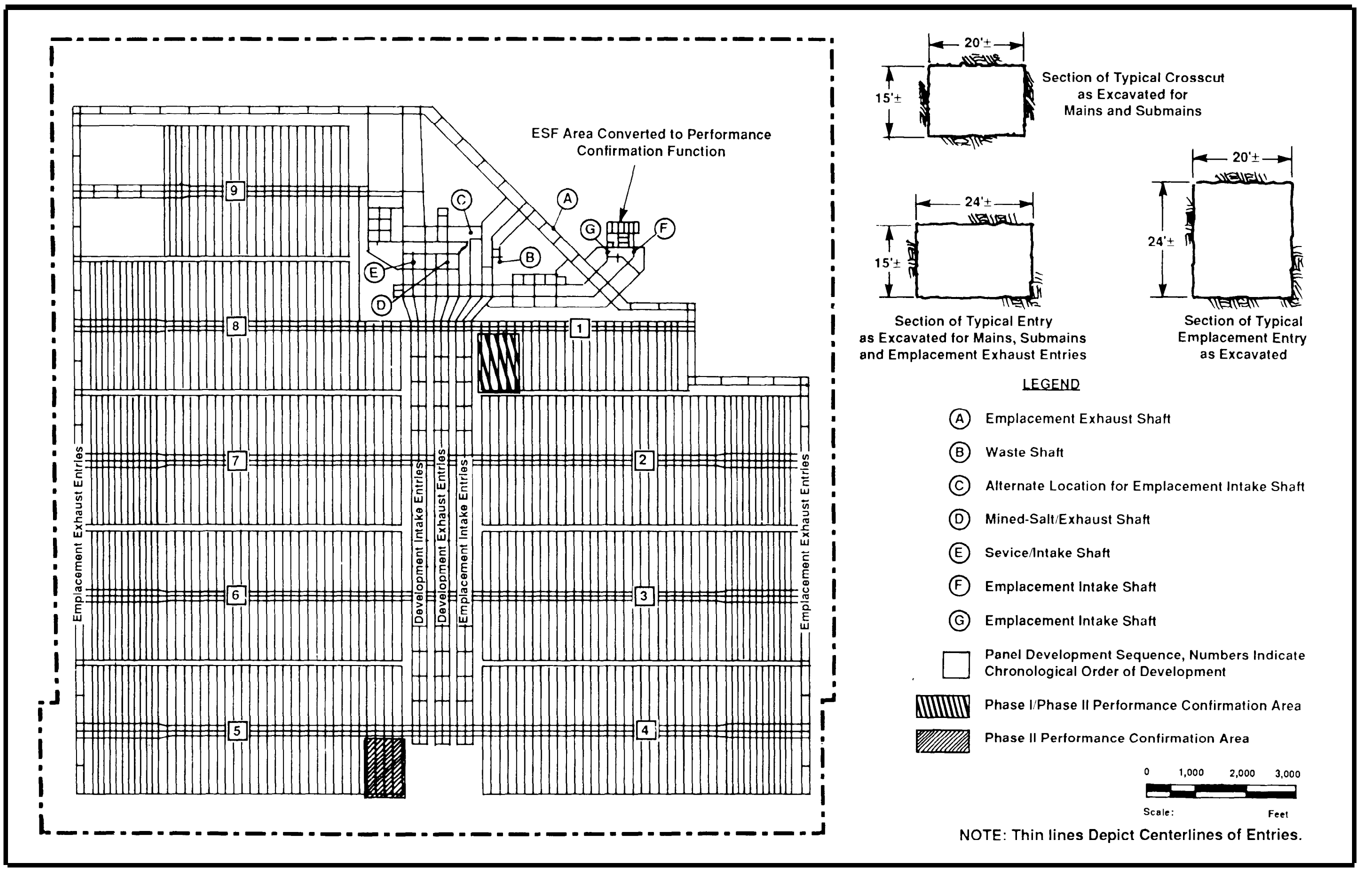


Fig. 24.3.2. Concept for a repository in a bedded salt formation. Conversion factor: 1 ft = 0.3048 m.

FACILITY DEVELOPMENT REQUIREMENTS. The development of the underground repository facility will probably proceed in much the same manner as the development of a comparably sized mine that could be developed under conditions that lend themselves to mechanization (MacDougall et al., 1987). Before a decision can be made regarding the suitability of a site for a repository, an exploratory shaft and a small area (up to several acres) for conducting investigations and tests must be constructed. The exploratory shafts could eventually be used in the operation of the repository, and the placement of the exploratory shaft and test area should consider the conceptual design of the orientation and configuration of the repository in the geologic formation. Conceptual illustrations of the underground portions of two different repositories are presented in Figs. 24.3.2 and 24.3.3. Fig. 24.3.2 reflects the design concepts for a repository in bedded salt formations that have no lateral boundary conditions created by the geologic formation. Fig. 24.3.3 presents the design of a proposed repository in a tuff formation that may be limited in the lateral directions by geologic structures. The primary purpose of the underground portion of the repository is to provide an area in which to dispose of waste materials. Thus the facility must provide access to a horizon selected for its isolation capabilities. Generally, this development would require multiple shafts to facilitate simultaneous construction and emplacement operations and maintain separate ventilation circuits. Depending on the depth of the selected horizon, there is no reason why one or more decline ramps could not be used in place of some shafts. Such ramps could actually offer safety advantages as compared to shafts and are, in fact, currently envisioned for use at the proposed repository site in a tuff formation at Yucca Mountain in Nevada (MacDougall et al., 1987).

Since it is a societal requirement that the first repository begin operation at the earliest possible date, it may not be possible to construct the entire repository before emplacement of waste begins. The initial development of the repository will cover the excavation of the underground area capable of receiving approximately 16,500 tons (15,000 t) of waste (five years receipt) prior to the initiation of operation (Anon., 1985). The size of the initial development is affected by the need to separate the ventilation systems for the area where emplacement of radioactive waste is occurring from the area where mining operations are occurring. The logic behind this situation is discussed under ventilation requirements.

The facility development must also consider the movement of excavated materials from two perspectives. First, the muck must be removed to enable the emplacement of waste canisters. Second, the muck could be used to backfill areas in which waste canisters had been placed. Backfilling could be accomplished concurrently with emplacement operations, but this decision would have to consider many factors and require specific approval by the NRC. Also, concurrent backfilling operations add an additional logistics problem to the underground operations, especially if all backfill materials must be crushed, classified to ensure particle size, and homogeneously mixed with added constituents to give the backfill desired geochemical or retardation characteristics. The use of decline ramps would also facilitate both the muck removal and backfilling operations, as a

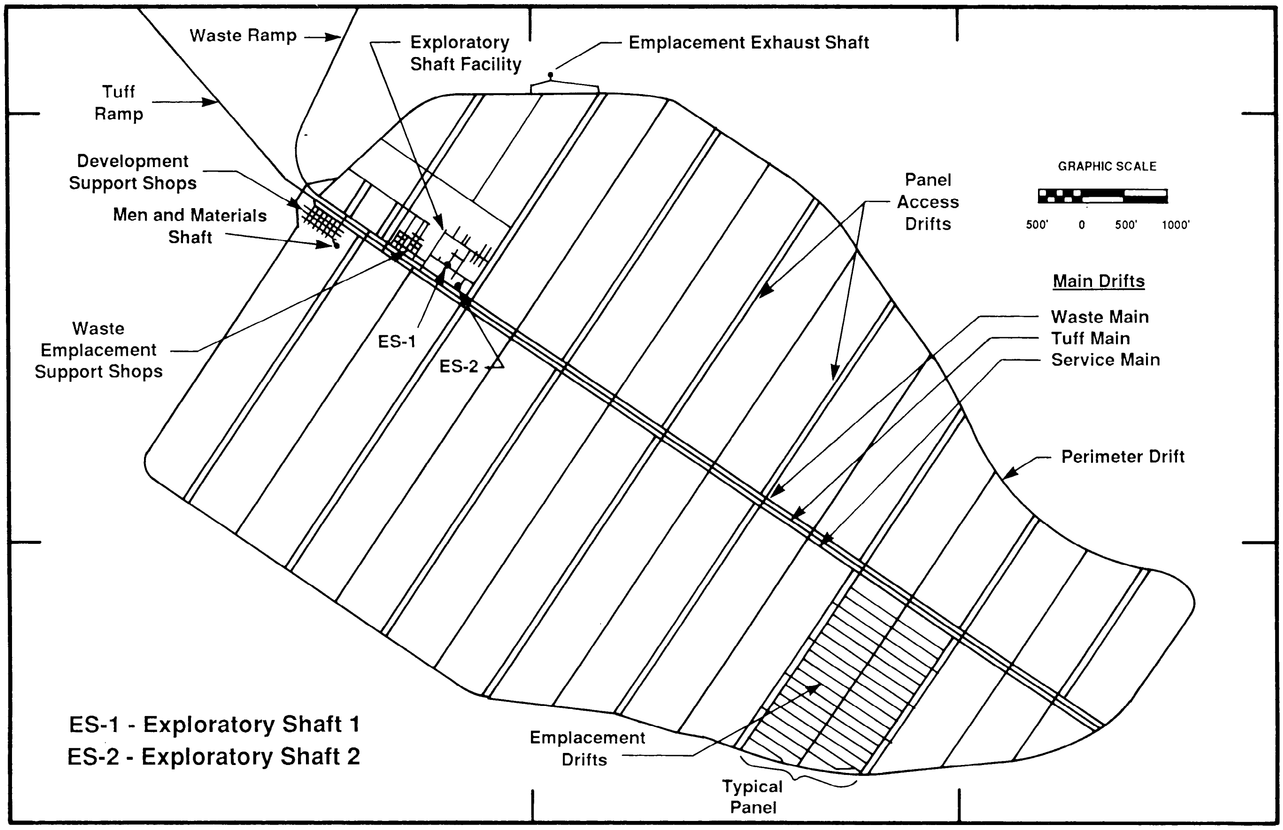


Fig. 24.3.3. Concept for a repository in a tuff formation. Conversion factor: 1 ft = 0.3048 m.

conveyor system could be implemented with possible attendant cost and efficiency benefits (MacDougall et al., 1987).

VENTILATION REQUIREMENTS. The regulatory requirements related to facility ventilation during the operational phase are derived from considerations that are related to public health and safety. Specifically, 10 CFR 60.133(g) contains a requirement that separate excavation and waste emplacement circuits be developed and maintained.

This requirement can be implemented in a conceptually straightforward manner by two ventilation systems designed to work in concert. The systems could be designed with a forced exhaust on the emplacement side of the ventilation circuit and a relatively positive ventilation pressure on the development or excavation side of the circuit. This is a safe arrangement in that loss of the ventilation fan(s) on either side of the circuit does not result in noncompliance with the regulatory requirement. If the exhaust fan(s) are lost, the positive pressure on the other side of the circuit should keep the ventilation flow in the correct direction; conversely, if the fan(s) are lost on the positive side of the circuit, the relatively negative pressures on the exhaust side of the circuit would prevent the airborne radionuclides, if released, from getting to the development side of the repository (MacDougall et al., 1987).

The ventilation systems do not have to be designed to work in concert, but there are apparent advantages in so doing. The primary disadvantage in a system where the circuits are not designed to work in concert is the potential for problems associated with leakage between the circuits.

The ventilation circuitry also is a primary element in compliance with the regulatory requirements to limit the release of radioactive materials that enable exposure of the public to airborne radionuclides during the operational period of the repository. Such radionuclides would generally only become available following an accident involving the rupture of a waste container. The method of choice to prevent releases following such an accident would probably be to use high-efficiency particulate-absorbing (HEPA) filters on the ventilation exhaust.

RETRIEVABILITY REQUIREMENTS. The principal reference to retrieval is contained in Section 122 of the NWPA (Anon., 1982) and states that the repository shall be designed and constructed to permit the retrieval of any spent nuclear fuel placed in such repository, during an appropriate period of operation of the facility, for any reason pertaining to the public health and safety, or the environment, or for the purpose of permitting the recovery of the economically valuable contents of such spent fuel.

The regulations, 10 CFR Part 60, require the implementation of these actions in 10 CFR 60.111(b) by stating that the geologic repository operations area be designed to preserve the option of waste retrieval throughout the period during which wastes are being emplaced, and thereafter until the completion of a performance confirmation program and Commission review of the information obtained from such a program. To satisfy this objective, the geologic repository operations area must be designed so that any or all of the emplaced waste could be retrieved on a reasonable schedule starting at any time up to 50 yr after waste emplacement operations are initiated, unless a different time period is approved or specified by the Commission. This different time period may be established on a case-by-case basis consistent with the emplacement schedule and planned performance confirmation program.

This requirement does not preclude decisions by the Commission to allow backfilling part or all, or permanent closure of, the geologic repository operations area prior to the end of the period of design for retrievability. A reasonable schedule for retrieval is one that would permit retrieval in about the same time as that devoted to construction of the geologic repository operations area and the emplacement of wastes.

The requirement to maintain the option to retrieve the emplaced waste results in either the requirement to maintain a stable facility, or to be able to reenter the facility and either remove any emplaced backfill, or in an extreme that could be encountered in temperature- or stress-sensitive rock formations, reexcavate the facility to recover the waste canisters.

For the single site selected for characterization by the US Congress, preliminary evaluations presented in the Site Characterization Plan for Yucca Mountain (Anon., 1988) indicate that stability can be maintained through the period of retrievability. Alternate evaluations (Brandshaug, 1989) produce similar conclusions and suggest that 50 yr of thermal and mechanical rock response has little effect on that development of joint slip beyond that produced from room excavation.

BACKFILL REQUIREMENTS. The term backfill in the repository has two specific connotations. In one case, it is used to identify the portion of the engineered barrier system that fills the volume in the emplacement hole between the waste package and the geologic medium or the host rock. In the second case, it refers to the material that will be used to fill the openings in the passageways and the shafts and/or ramps leading into the repository. While both applications have a common name and on the surface appear to have a common function, they are significantly different. Both backfills have a common interface where the emplacement holes intersect the passageway used to deliver the waste to the point of its permanent disposal.

The specific functions of the backfill that constitute the portion of the engineered barrier system adjacent to the waste canisters are covered in depth in that subsection. The primary function of the backfill that must be introduced into the openings, passageways, and shafts when it is decided that it is acceptable to close the repository is to provide some structural stability to the labyrinth of tunnels during the isolation phase. The backfill material will most likely be created by crushing and classifying the rock removed to create the opening in order to improve the packing density. The backfill is composed of a relatively noncompressible solid that, when filled into all the open space, can provide some structural support to the rock surrounding the opening. The efficiency of the backfill is dependent largely on the degree of compaction that can be obtained; its effectiveness will depend to some extent on the nature of the geologic medium in which the repository is located. If the repository is in a plastic, deformable medium with properties that are time dependent, there is still a possibility for subsidence in the opening since the backfill, if installed in particulate form in the field, would only achieve approximately 60% of maximum density. This leaves a 40% volume of void space that could slowly compact under the natural lithostatic load. In view of the structural strength of a nonplastic host rock and the low excavation ratio to be used in the repository, it is not likely that the openings will deform or collapse, thereby transferring the load to the backfill. If it becomes necessary to rely on the structural strength of the backfill to prevent subsidence, a careful program will have to be followed to ensure that the backfill does, in fact, provide structural resistance.

Since the backfill emplaced in the field has approximately 40% void space, it results in a fairly permeable material that will allow significant water movement if water enters the repository. Additions to the crushed host rock to alter its permeability are being considered (Fernandez and Freshley, 1984; Freshly, Dove and Fernandez, 1985a, 1985b). Bentonite, a mineral that increases in volume in the presence of water, has the ability to reduce the void space and permeability.

Backfill incorporating a material to minimize void space and permeability will be dependent on the geologic medium and its physical characteristics. For example, for a repository in salt, backfilling with bentonite is not likely to be needed. If any significant quantity of water enters a salt repository, it is a strong indication of a major failure in the seals, and the inclusion of a backfill in a water soluble media has little or no significance. For a repository in the vadose zone, a backfill with an expanding constituent seems to be of limited value; the value would be in a repository located beneath the water table in a formation that would be saturated with water.

CHEMICAL CONTAMINATION CONTROL. Chemical contamination control in the operational phase of the repository is of primary importance from an environmental perspective and an operational health and safety perspective. From either perspective, chemical contamination control at a repository is expected to be slightly different from that at other mines or underground facilities. First, environmental impacts will have to be mitigated and monitored as indicated by the provisions of the Resource Conservation and Recovery Act of 1976 (Anon., 1976). Secondly, all materials brought into the repository and their volume (tonnage) will have to be examined for mobility and interactive compatibility with the radionuclides.

RADIOLOGIC SAFETY REQUIREMENTS. The requirements of 10 CFR Part 60 related to radiological safety deal primarily with the release of airborne radionuclides during the operational phase of the repository. They are concerned with exposure of the public to these radionuclides. The requirements are manifested in a performance objective in 10 CFR Part 60.11 l(a) that states that the GROA shall be designed so that, until permanent closure has been completed, radiation exposures, radiation levels, and releases of radioactive materials to unrestricted areas will at all times be within the limits specified in 10 CFR Part 20 (Anon., 1989c). The regulation also sets forth general design criteria for the GROA in 10 CFR 60.131 to accomplish the goal as stated in the performance objective. These general design criteria specify that the design must incorporate means to limit the concentrations of radioactive materials in the air and means to limit the time required to work in the vicinity of the radioactive materials, including means to provide for limited access to such areas. The intent of these requirements can be met through the use of adequate shielding, designing equipment for ease of repair, and providing adequate space for ease of operations (MacDougall et al., 1987). The burden of compliance with this part of the regulation ultimately falls on the ventilation system which must be able to function to ensure that any such materials inadvertently released are not permitted to reach the public in amounts that violate the release standard.

An important concept in this part of the regulation is that of “structures, systems, and components important to (radiological) safety.” Generally, any item or activity that must be relied upon to to protect the public health and safety falls into this category. Also if an item or activity is not important to (radiological) safety, it is, in reality, not subject to regulation by the NRC. A notable example here is the safety requirements that are imposed on mines through the Mine Safety Health Administration (MSHA); NRC only regulates such aspects of the underground facility to the extent that worker protection may be necessary to provide reasonable assurance that all systems, structures, and components important to (radiological) safety can perform their intended functions. The structures, systems, and components important to (radiological) safety must be designed so that natural phenomena and environmental conditions anticipated at the GROA will not interfere with necessary safety functions.

The general design criteria in 10 CFR 60.131 describe particular design features that are intended to ensure the continuous operation of the structures, systems, and components important to (radiological) safety in the event of accidents or equipment failure. They require that the structures, systems, and components important to (radiological) safety be able to perform their intended functions in the event of fires, explosions or flooding. Consequently, they mandate, to the extent practicable, the use of noncombustible and heat resistant materials, as well as alarm and suppression systems. They further require instrumentation to monitor the performance of the structures, systems, and components important to (radiological) safety, as well as redundant means to ensure the continued function of the structures, systems, and components important to (radiological) safety in the event of an emergency. It is to be expected that, if a hoisting system is to be used to move the waste canisters underground, such system would fall into the category of a structure, system, or component important to (radiological) safety. Such hoists would have to be designed to preclude cage fall, be reliably locatable, incorporate interlocking systems that would fail in a safe manner, and include two independent indicators of when waste packages were ready to be loaded or unloaded.

Additional criteria related to radiologic safety for the surface facilities are found in 10 CFR 60.132. Generally, these criteria are related to safe handling and storage of radioactive materials at the surface. They result in a design that will accomplish fuel-rod handling and packaging into the waste canisters in a remotely operated hot cell facility. The design criteria also specify controls to be placed on effluent handling and disposal, as well as the treatment of low-level wastes generated at the site that do not need to be stored in the underground facility (MacDougall et al., 1987).

Closure Requirements. The regulatory requirements of 10 CFR Part 60 include particular requirements related to the permanent closure of the repository following its operational period. Specifically, the licensee is required to apply for an amendment to its license to receive and handle the radioactive wastes in order to close the facility. The application to amend the license will consist of an update to the original license application and environmental report, and must include information on the program of monitoring of the repository following permanent closure. Also, the application must include descriptions of the methods to be used to regulate or prevent activities that could impair the long-term isolation of emplaced wastes and to ensure that information relevant to the repository will be preserved for future generations. This will include the fabrication and use of monuments that have been designed to be as permanent as is practicable to identify the repository location. The closure would only be permitted if the performance confirmation program indicated through modeling of its performance that the facility was functioning as predicted.

From a design perspective, closure of the facility involves decommissioning of the underground facilities, including perhaps the removal of utilities and support components such as shaft liners. Sealing of the underground facility and its accesses, as well as boreholes drilled for exploration, is also an integral part of the facility closure. The removal of components, such as shaft liners, would primarily be undertaken to facilitate more effective sealing of the underground facility. It is incumbent upon the designers of the underground facility to anticipate possible methodologies for closure of the facility and incorporate means to facilitate closure into the design. A description of such design features will be included in the license application Safety Analysis Report as required under 10 CFR 60.21. The options available to the designer to meet this requirement are varied and generally are of the form of limiting or prohibiting activities that would preclude the effective closure of the facility.

Following the permanent closure of the facility and decontamination and decommissioning of the surface facilities, the licensee (DOE) will be required to apply for an amendment to terminate its license.

24.3.2.2 Isolation Phase

Waste Package Requirements. The core (the innermost part) of the engineered barrier system and the waste isolation system is the waste package. The regulations issued by the NRC establish a performance objective for the waste package in 10 CFR 60.113 that requires the waste package to provide for substantially complete containment for a period of 300 to 1000 yr.

This requirement is interpreted to mean that the characteristics of the waste package are such that radionuclides in a solid, liquid, or gaseous state will be held inside the boundaries of the package until the containment is breached. An interpretation of this requirement is that the waste package should have, as part of its structure, a hermetically sealed barrier (canister).

The waste package carries with it requirements to perform a number of functions. The specifications for the waste package (i.e., configuration, dimensions, fabrication methods, materials, test methods) must be developed to ensure the package can resist a series of expected and unanticipated events that it could likely experience in its nominal handling. The waste package must be considered as a candidate for the category of structures, systems, and components important to (radiological) safety. It should be kept in mind that some of the functions are time dependent and, after a minimum of 300 yr, these functions are not necessarily required. The functions the waste package could be expected to perform include:

1. Facilitate handling and transportation of the waste outside the repository facility.
2. Transmit thermal energy, released by the waste, to the environment.
3. Provide for handling in the repository (from surface to point of emplacement).
4. Provide a continuous barrier of extremely low permeability (hermetic seal) to prevent water from contacting the waste form and to prevent release of radionuclides (in a solid, liquid, or gas state).
5. Provide for retarding release of radionuclides once a hermetically sealed canister is breached.

Since the waste package is likely to be a structure, component, or system important to (radiological) safety, as well as to long-term isolation, it is expected that there will be extensive controls on the quality of this item. The nominal expectation is that the waste package will be fabricated from a material processed in a manner that is totally under the control of man. In this effort, the most significant decision is which material will be used. The material must satisfy two basic constraints: (1) it must be capable of meeting the containment requirement within the thermal and chemical environment in which it must perform, and (2) it must be capable of being fabricated into the hermetically sealed canister of the desired configuration. If the containment of radionuclides is an essential criterion used in obtaining approval to construct the repository, it is possible that the engineering technology available to fabricate the waste package can influence the selection of the site. The material selection is likely to be dictated by the environment that will exist in the repository (which includes the temperature profile during its lifetime, presence of an electrolyte, and chemical aggressiveness of the environment); the opposite can be true in that materials available to construct a waste package can set limits on the geologic, hydrologic, and geochemical conditions that can be selected for the repository.

Material selection will be more art than science. Two basic philosophies can be followed in the selection of the material. The philosophies center on the selection of a material with a high corrosion resistance capable of withstanding the environment with minimal effects or the use of a sacrificial material that will readily corrode, but by well-understood mechanisms, at well-established rates. The latent basis for successful selection of either material will depend solely on the ability to convince the NRC staff and the Licensing Board that mechanisms leading to failure are well understood and that the potential for failure, preventing achievement of the performance objective, is low.

Selection of materials that have high corrosion resistance and can provide the 300 to 1000 years of containment in the expected environment is an approach fostered by the expectation that groundwater (electrolyte) can and will be a major factor affecting the containment requirement. Metals and alloys that have high resistance to corrosion often exhibit failure characteristics (i.e., they fracture creating a breach in a waste package) that are considered unpredictable and difficult to evaluate. For example, stainless steel or titanium, which owe their high corrosion resistance to the passivation mechanism, often fail by stress corrosion cracking or hydrogen embrittlement. These failure mechanisms have long incubation periods and occur in a precipitous manner with little or no indication of the impending failure. While it may be possible to achieve the performance objective using such a material, it may be difficult to amass the data necessary to establish the compelling case that all failure mechanisms are well known and the knowledge can serve as the basis for such a decision.

An alternative approach is the selection of a material that has a moderate corrosion rate, but one for which the corrosion mechanism is well understood. The rationale for this approach is that with such material, other failure mechanisms, such as stress corrosion cracking, are not likely to occur in those materials. It is argued that it is possible to have confidence in the evaluations of these materials because of the apparent lack of the precipitous and difficult-to-characterize failure mechanism.

Regardless of which approach is chosen, the same fundamental data and analyses will be required to demonstrate that there will be reasonable assurance that the performance objective will be achieved and that the off-normal schemes of failure mechanisms have been examined and demonstrated as not likely to compromise performance. Data requirements will be driven by the NRC staff, the Licensing Board, and the other parties raising issues; neither approach can be given preferential treatment.

The development of the engineered barrier system and the waste isolation system requires an understanding of two different concepts: (1) the actions taken in the design and implementation of the system to achieve performance, and (2) the actions taken to demonstrate or provide reasonable assurance that performance objectives will be achieved. In this area, it is quite possible that all actions taken will provide for achievement of performance, but because it cannot be readily demonstrated and documented to the extent acceptable in an administrative law court, the proposed actions may not be accepted.

In the current mined geologic repository concept that is predicated on a situation where the United States has not developed an industry for reprocessing of spent fuel from commercial power reactors, the waste form expected for disposal is spent fuel elements (Anon., 1980). A fuel element is composed of an array (17 by 17, 15 by 15, or 8 by 8) of fuel rods (uranium dioxide ceramic pellets in stainless steel or zircalloy tubes) varying in length between 12 and 14 ft (3.6 and 4.2 m). A fuel element from a PWR weighs approximately 1320 to 1540 lb (600 kg to 700 kg), and for a BWR weighs approximately 550 to 660 lb (250 kg to 300 kg). These are illustrated in Fig. 24.3.4. A single fuel element or multiple fuel elements will be placed in a long, cylindrical canister and hermetically sealed. The number of fuel elements in a canister will depend on the thermal power output of each fuel element and the maximum allowable temperature in either the canister, the backfill that might surround the waste package, or the geologic media of the repository.

ENGINEERED BARRIER SYSTEM REQUIREMENTS. The engineered barrier system consists of the waste form, the waste package, and the backfill. The requirements of 10 CFR 60.113 establish a performance objective for the engineered barrier system that requires the release rate of any radionuclide from the engineered barrier system following the containment period shall not exceed one part in 100,000 per year of the inventory of that radionuclide calculated to be present at 1000 yr following permanent closure.

This barrier must be created using some material that has the properties, both chemical or physical, to support the execution of the function desired by the engineered barrier system. In addition to the materials, the method of assembling or fabricating the system into the desired configuration will also significantly affect the performance of the system. The functions that are minimally expected of the engineered barrier system are as follows:

1. Retard the release of radionuclides from the waste package.

2. Transfer the thermal energy, released within the waste package, to the geologic medium.

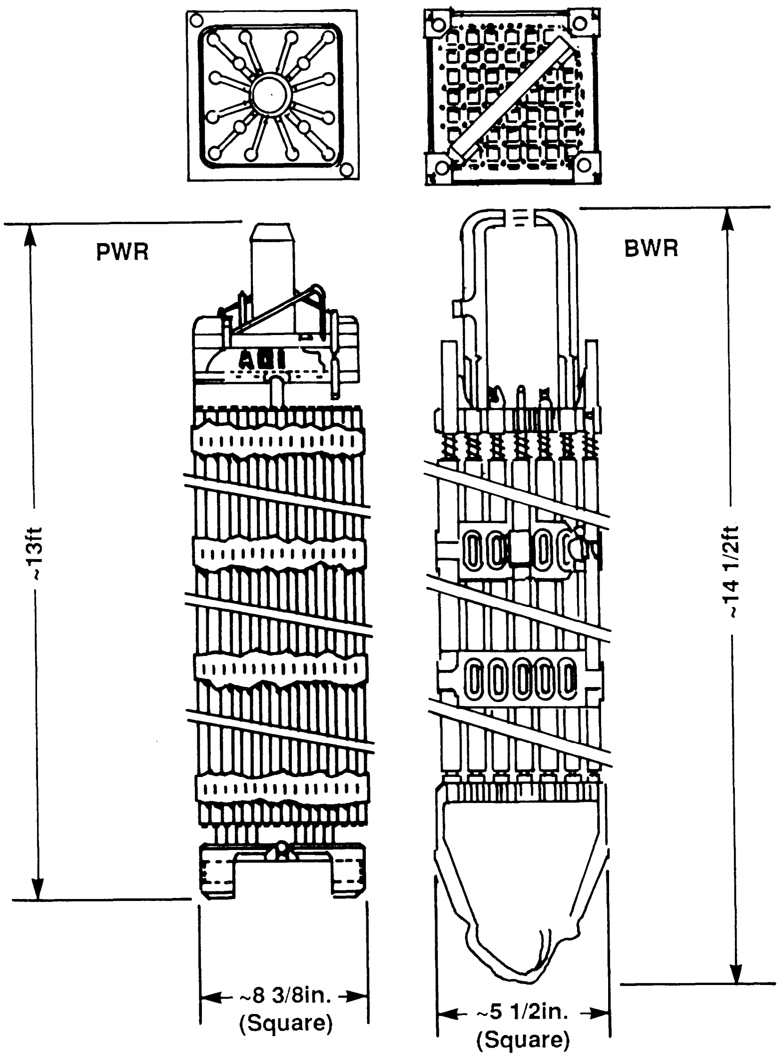
3. Control the rate at which water reaches the waste package.

4. Alter the groundwater chemistry that reaches the waste package or waste form.

Materials that usually receive preference for the engineered barrier system are ones that have high retardation coefficients. Materials that have those characteristics are usually complex inorganic oxides or silicates such as zeolites or bentonite. In powdered form, with a large surface to volume ratio, it can create a continuous form with uniform porosity and low permeability that can easily create a situation for effective sorption of radionuelides and control of water movement. However, the packing density of powders emplaced in the field to create a continuous barrier reach about 60% of theoretical density. The solid powders have poor thermal conductivity and, with 40% void space (be it water or air), create a good thermal insulator. This approach has the ability to keep the released thermal energy in the waste package, increasing its temperature.

FACILITY DESIGN AND CONSTRUCTION REQUIREMENTS. *Geometric Considerations*—The primary focus of compliance for postclosure performance of the repository is the performance objectives of 10 CFR Part 60. The regulation also embodies specific design criteria in 10 CFR 60.130 through 60.135. These specific design criteria are grouped to address, generally, surface facilities, operational phase requirements for the subsurface facilities and postclosure (isolation phase) performance requirements on the design. The postclosure design concerns, called additional design criteria in 10 CFR 60.133, specify design considerations intended to ensure that design and construction of the subsurface facilities of the repository are carried out in a manner that the repository assists the geologic setting in meeting the performance objectives.

The physical layout of the repository encompasses factors that, when varied, can have an impact upon the ability of the repository to comply with the performance objectives. The relevant factors include the depth of the repository and its geometric



	PWR	BWR
Fuel Element Array	17 × 17	8 × 8
Assembly Total Weight, lb (kg)	1,450 (658)	705 (320)
Uranium/Assembly, lb (kg)	1,015 (461)	415 (189)
Fuel Element OD, in. (mm)	0.37 (9.4)	0.49 (12.4)
Fuel Elements/Assembly	264	63

Fig. 24.3.4. Schematic diagram of commercial nuclear power reactor fuel elements. Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

configuration, including orientation of the drifts. The depth of the repository can have impacts upon compliance with the performance objectives. Generally, a deeper repository would have a longer travel path length for radionuclides transport. A repository located at a depth greater than 1000 ft (300 m) would be considered favorable under 10 CFR 60.122. This depth generally correlates to a depth deeper than typical depths of water wells. A repository deeper than 1000 ft (300 m) would be relatively safe from inadvertent penetration by development of a water well.

The geometric configuration of the underground repository also can be varied to result in different impacts on the ability of the repository to comply with the performance objectives. Although the actual wording of this requirement of 10 CFR 60.133(a)(1) is to select a configuration that contributes to containment and isolation, pragmatically, it may not be possible to design a layout that reliably contributes to isolation over the required 10,000-yr period of isolation. It is likely, however, that certain geometric configurations will result in lessened impact. Specifically, good mine development and layout practice dictates (Hoek and Brown, 1979) that natural fracture orientation and in situ stress state be considered in selecting drift configuration and orientation. This would also be expected to lessen postclosure impacts. A drift layout that would minimize support requirements would generally be expected to have the least impact on the ability of the site to isolate waste as it would be expected to minimize rock movements. An additional factor is relevant, however, for repository design. The effects of the stresses and displacements induced by the heat generated by the waste should also be considered in determining an acceptable configuration.

The intent of this portion of the regulation was addressed in the conceptual design of a repository to be located above the water table. In that design, a dipping repository configuration was to be developed. That facility would have been built in dipping strata; arguably, limiting the crossing of stratigraphic boundaries could contribute to isolation. Of more interest, relevant to the intent of this requirement, is the fact that such an attitude makes the repository self-draining, which is particularly appropriate for an unsaturated zone repository, should any significant quantity of water enter the repository.

Flexibility to Accommodate Specific Site Conditions—An important aspect of the regulatory requirements related to the design of the underground facility of the repository is the requirement of 10 CFR 60.133(a)(2) to maintain sufficient flexibility to accommodate site-specific conditions. Generally, the site-specific conditions that must be accommodated are those that are more likely to lead to a postclosure impact (loss of containment or isolation) or a preclosure impact (worker safety, excavation stability, or loss of ability to retrieve). The design of the underground facility of the repository needs to anticipate the possibility of encountering such conditions and have contingency plans in place to be able to deal with such conditions when encountered. Examples of site-specific conditions that would have to be accommodated include fault or shear zones with potential for rock mass movement or water-bearing zones where none were expected. The contingency plans would detail the manner in which the areas of anomalous conditions would be avoided or their effects mitigated.

The authorization to construct the repository facility will be granted subject to conditions imposed by the NRC that are structured to ensure that the actual construction of the repository is in rock conditions as described in the license application. To be in compliance with these imposed conditions requires that the calculations about compliance with the regulations bound the range of conditions that are encountered in the repository construction. If the conditions within which the repository is constructed are different from those for which the authorization was granted, the construction of the repository must be able to accommodate this. The contingency plans should provide the mechanism for accommodating these differing conditions. Should they be insufficient to do so, then the construction of the repository could not proceed until the effects of the differing conditions had been assessed and the construction authorization amended.

Underground Openings—The design of the underground openings must take into consideration several regulatory requirements related to the stability of the underground excavations. As previously noted, 10 CFR Part 60 imposes opening stability requirements or considerations on the facility for aspects of the facility related to retrievability or to worker safety to the extent that it is relevant to the successful operation of items important to safety. The typical MSHA-type safety considerations are beyond the purview of the NRC but are expected to be imposed on the repository underground facility through mandatory requirements of the licensee.

There are two additional design criteria of 10 CFR 60.133 that impose excavation stability-related requirements on the repository underground facility. The first of these is 10 CFR 60.133(e)(2), which states that openings in the underground facility must be designed to reduce the potential for deleterious rock movement or fracturing of the overlying or surrounding rock. The other requirement is found in 10 CFR 60.133(f), which states that the design of the underground facility shall incorporate excavation methods that will limit the potential for creating a preferential pathway for groundwater to contact the waste packages or for radionuclide migration to the accessible environment. These two requirements deal with conceptually similar aspects of impacts on the capability of the site to contain and isolate waste. They both deal with the creation of pathways, typically manifested as cracks or fractures, that would shorten travel times or otherwise promote preferential radionuclide transport.

Fluids and Materials Control—Fluids and materials control in the underground facility are relevant to the postclosure performance of the repository in that materials to be used in the underground facility have the potential to affect the transport of radionuclides to the accessible environment (e.g., see 24.3.1.6). Fluid and material control are implicitly recognized in 10 CFR 60.133(h), wherein the requirement that the engineered barrier system be designed to assist the geologic setting in meeting the postclosure performance objectives is stated. Also 10 CFR 60.130 explicitly requires the incorporation of specific design features that are needed for the facility to achieve compliance with the performance objectives. It is likely that fluid or material control could fall into this category.

Likely effects of materials used in the construction and operation of the underground facility are related to chemical complexing of the materials with the radionuclides, resulting in an enhanced travel time for the complexed ion relative to that of the radionuclide. Components of the epoxies used in rock bolt anchorage systems, as well as cementitious materials, have the potential to interact with the radionuclides, and they must be carefully examined in the context of the geochemical composition of the rock mass and the naturally occurring waters. For this particular example, a practical solution may be found in the use of bolting schemes utilizing concepts similar to those of the split set or swellex techniques, provided that the ferrous oxide materials resulting from the degradation of the steel bolts do not lead to a similar problem in complexing with the radionuclides.

The materials used can also degrade the expected corrosion response of the waste package if care is not exercised in the selection of the materials used in construction and operation of the underground facility. Alternately, a reasonable control program could permit the use of virtually any material, provided that means to prevent undesirable materials from contacting the waste packages could effectively be implemented. The previously mentioned concerns about organic materials (tire rubber, fuel and oil spills, etc.) may be amenable to control through the use of a scheme that placed a thin layer of fine particulate material on the floors of the drifts. Such a material could be effective in trapping the organics and could be subsequently scraped up at the time of closure of the facility.

For a repository in the unsaturated zone, the amount of water used in the construction and operation of the facility can also have an impact on the ability of the repository to comply with the release standards imposed by the performance objectives. Unrestricted use of water has the potential to modify the hydrologic properties of the site to the degree that the character of the unsaturated zone no longer functions to impede the transport of radionuclides. Specifically, this would occur if sufficient water were available to saturate a previously unsaturated volume of rock for a sufficient period of time to enable the development of saturated pathways to the accessible environment. The character of the unsaturated zone makes it unlikely that such pathways could be developed and maintained for sufficient periods to affect the performance of the site, if reasonable controls are placed on the amounts of water used in construction and operation of the facility.

Thermal Loads—The emplacement of the radioactive fuel elements into the rock at the repository horizon entails a significant quantity of heat energy that will be dissipated into the volume of rock surrounding the repository. The effects of this heat can be manifested through expansion of the rock and possible resulting fracturing and rock mass property degradation (St. John, 1987). The heat can also affect the geochemical and hydrologic properties of the rock mass as well as the hydrologic response of the area surrounding the repository. The effects of the heat can be managed during the operational period of the repository through ventilation and perhaps even involving refrigeration. However, the major effects of the heat load will be manifested during the postclosure time frame when the facility is not accessible for temperature control. The primary approach available to the designer to manage the effects of heat in this time frame is to control the areal power density. Practically, this is accomplished by separating the canisters sufficiently to limit the deleterious effects of the heat load.

The design criteria for the underground facility specify, in 10 CFR 60.133(i), that the underground facility shall be designed so that the performance objectives will be met, taking into account the predicted thermal and thermomechanical response of the host rock, the surrounding strata, and the groundwater system. This requirement is simply a statement that the demonstration of compliance with the postclosure performance objectives must consider the effects of the heat generated by the emplaced wastes. The effects of this heat need to be considered relative to compliance with the post-closure performance objectives. The effects of heat and their relationship to compliance with the postclosure performance objectives fall generally into one of two categories: (1) near-field, or roughly canister scale, effects; and (2) far-field, or roughly repository scale and larger, effects. The near-field effects, which involve higher temperatures and more severe rock mass property degradation, are important to the two performance objectives related to the waste packages. These performance objectives dictate a containment period (equivalent to a waste package lifetime) and a maximum permissible release rate for radionuclides from the canisters once containment is no longer effective. Temperature is an important parameter in determining the waste package environment, which together with the canister materials, form the basic engineered system which must comply with these performance objectives. The areal power density probably results in second-order effects on the waste package environment as compared to the effects of the canister temperature itself.

The repository scale effects of the heat due to the emplaced wastes must be considered in the demonstration of compliance with the prewaste emplacement groundwater travel time performance objective. The prewaste emplacement groundwater travel time performance objective must consider the effects of the heat of the emplaced wastes in calculating the extent of the disturbed zone, which is the boundary from which the travel time to the accessible environment is calculated. The disturbed zone is intended to be of sufficient extent such that the significant effects of the repository construction and functioning would be limited beyond it. Evaluations performed to date (Gordon et al., 1986) suggest that the effects of temperature are the most significant consideration in determining the extent of the disturbed zone.

Repository (and larger) scale temperature effects are important in the demonstration of compliance with the remaining postclosure performance objective, which addresses total radionuclide releases to the accessible environment. The effects of temperature need to be considered relative to the design of the underground facility, with regard to possible impacts on the ability of the repository to comply with the performance objective. Such impacts include effects of the heat of the emplaced wastes that could affect the rock mass properties in a manner that would limit the effectiveness of the site in preventing releases of radionuclides to the accessible environment. These effects

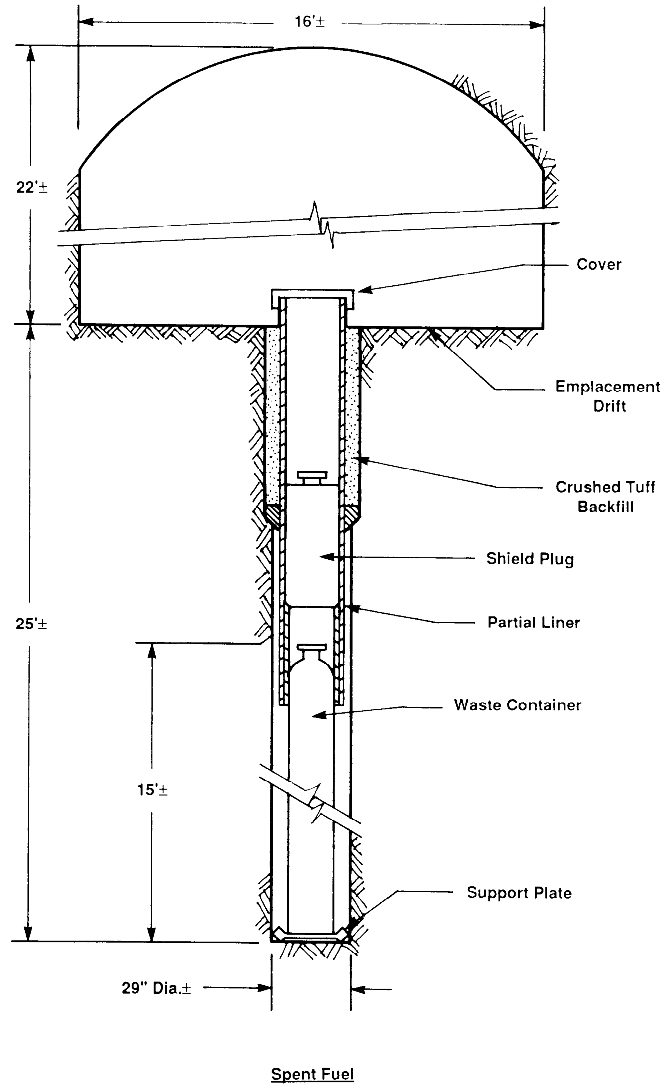


Fig. 24.3.5. Schematic diagram of single waste canister in vertical emplacement hole. Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m.

could be related to deleterious changes to the structural character of the rock mass such that preferential pathways could be created. Such effects could be related to structural instabilities in the excavation itself, resulting from thermally induced stresses, or they could be the result of larger-scale rock-mass responses such as thermally induced uplift or subsidence.

The design of the facility can accommodate such effects through design parameters including the areal power density, the extraction ratio, and the orientation and layout of the facility. The spacing of the waste canisters and the excavation extraction ratios effectively determine the decay heat power density for the repository. The areal power density can also be affected by the scheme chosen by which the waste canisters will be emplaced, either horizontal or vertical (MacDougall et al., 1987). In the vertical emplacement scheme, illustrated in Fig. 24.3.5, single canisters would be placed in the borehole in the floor of the drift. In the horizontal configuration, illustrated in Fig. 24.3.6, one or more canisters would be placed in long holes in the sidewalls of the drift. Vertical emplacement holes are generally limited to one canister, while long horizontal holes can be constructed to

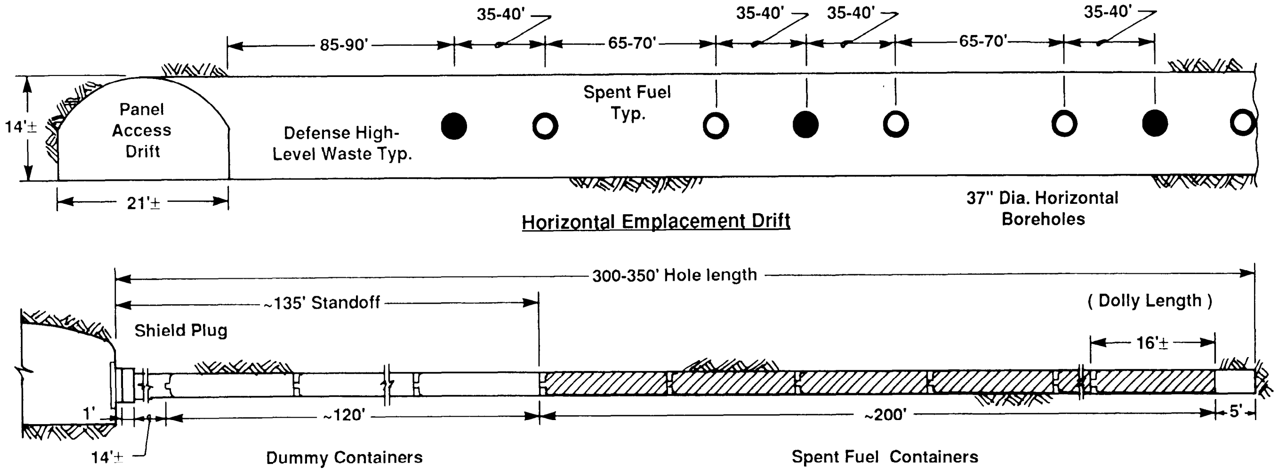


Fig. 24.3.6. Schematic diagram of multiple waste canisters in horizontal emplacement hole. Conversion factor: 1 ft = 0.3048 m.

accommodate numerous canisters. The horizontal emplacement scheme has two inherent advantages: (1) fewer drifts or passageways must be mined, and (2) a greater standoff distance between the drift wall and the last canister can significantly increase the time required for the temperature of the drift to increase over the nominal underground temperature. This second point could be critical in ensuring the long-term retrievability option. However, there are disadvantages with the horizontal concept in that construction of large-diameter (greater than 36-in. or 1-m) boreholes of the desired length has yet to be demonstrated in hard rock and the retrieval is probably less certain in the horizontal scheme.

On a scale larger than the repository, the effects of the heat resulting from the emplaced wastes have the potential to affect the hydrologic character of the surrounding rock. In a repository beneath the water table, such effects could be related to density changes of the water and concomitant affects on the direction of water movement. For a repository in the unsaturated zone, the effects are more likely to be related to drying of the rock mass and subsequent saturation of the rock mass as the vapors condense, resulting in relatively rapid flowpaths where none existed prior to the condensation. The amount of heat associated with the repository also raises questions of whether there is sufficient energy available to influence geologic processes in the region of the site; the possibility for such interactions must be considered in evaluations of compliance with the performance objective.

Sealing Requirements for Shafts and Boreholes—The general requirement for sealing of shafts and boreholes in 10 CFR 60.134 states that seals must be designed so that following permanent closure, the boreholes and shafts do not become pathways that compromise the ability of the repository to meet the postclosure performance objectives. This requirement further states that the materials and emplacement methods for seals shall be selected to reduce to the extent practical, the potential for creating a preferential pathway for groundwater to contact the waste packages and the potential for radionuclide migration through existing pathways. Techniques for sealing exploratory boreholes and shafts are in widespread use throughout the industry; it is unlikely, however, that such techniques have ever been examined for their ability to perform as intended on the time scale of interest for the postclosure performance of a repository (Fernandez and Freshley, 1984; Fernandez, Hinkebein, and Case, 1989).

The regulations issued by NRC (10 CFR Part 60) require that the repository be sealed when the repository is closed and decommissioned. The specific requirement states that the seals for shafts and boreholes shall be designed so that following permanent closure, the shafts do not become pathways that compromise the geologic repository's ability to meet performance objectives.

These sealing requirements are based on the assumption that a virgin site has no significant preferential pathways by which radionuclides can return to the surface. Drilling a borehole or constructing a shaft changes this basic situation by creating an opening extending from the surface to the horizon of the proposed repository. For large openings, two areas must be considered. The first is the opening itself in which a seal of some material will have to be placed. A conceptual illustration of a seal and its components is outlined in Fig. 24.3.7. The second is the annulus around the opening which will be disturbed by the creation of the shaft and in which the characteristic of the rock will be altered in a negative manner. In this volume, which is defined by a volume equal to about one radius of the opening, the permeability of the rock is judged to increase by two orders of magnitude, thereby making this portion of the repository more susceptible to movement of water.

Sealing of the shaft openings will require that a material of some type be placed in the opening. The material will have to be evaluated within the context of a number of factors. Obviously, it must have excellent long-term characteristics; that is, it will not change to a significant level with time; the material must be capable of making a tight interface with the host rock; it must form a strong mechanical or chemical bond with the host rock; and it cannot have significant shrinkage with time, or a crack will form at the interface between the seal and the host rock.

Practically, this sealing requirement will have to be addressed from at least two perspectives. First, detailed investigations will need to be carried out to ascertain the particular materials and sealing techniques that can be demonstrated to have the capability to comply with the intent of this requirement or at least support a convincing argument that they either have a high likelihood of success or that their nonperformance will not significantly affect the overall repository performance. (The latter argument may be appropriate for a repository in the unsaturated zone.) The desired materials and emplacement techniques will have to be compatible with the intrinsic character of the rock mass to be sealed. This can probably best be accomplished with materials that are geochemically similar to the host rock and that will not degrade significantly over the required time period. Materials with a potential to passively reinforce the intended performance of the seals would be ideal if they can be

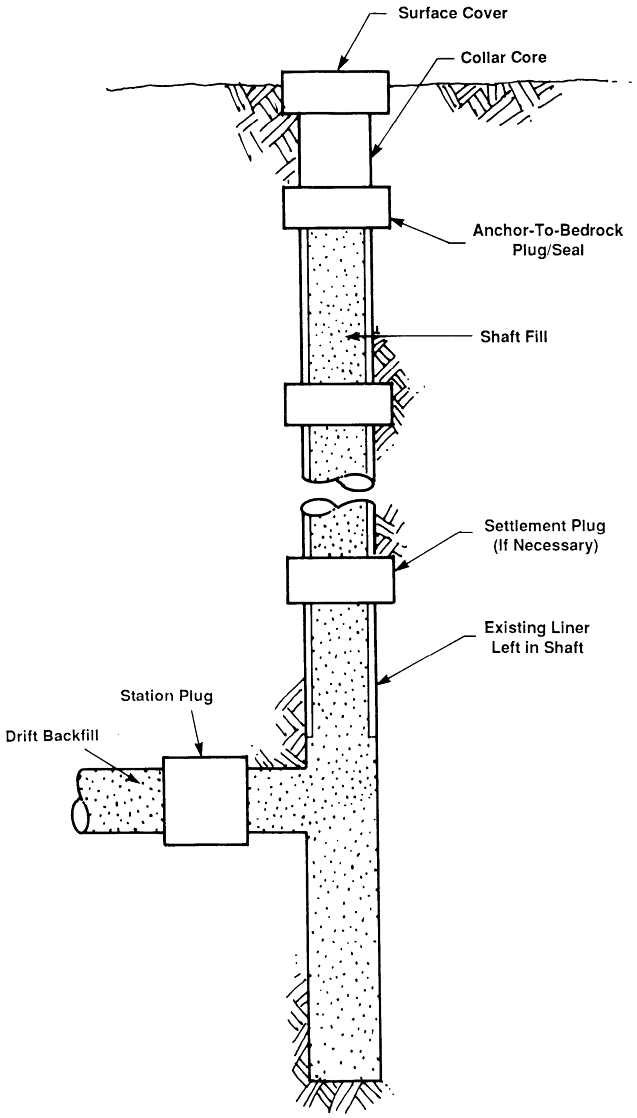


Fig. 24.3.7. Conceptual illustration of shaft seal configuration.

found. A material that could function in this manner would encourage precipitation of material in the groundwater encountering the seals that would clog any pathways that developed as the seals deteriorated.

The second perspective important to sealing that must be considered is related to the fact that it is unlikely that the sealing materials will be fully developed and the emplacement techniques finalized prior to the start of repository construction. This problem must be dealt with through an approach that recognizes the importance of limiting any impacts to the rock mass that would affect the ability to successfully emplace seals at a later date. Generally, this will require geochemical compatibility of materials used during construction with candidate sealing materials and excavation techniques that limit damage to the rock mass, particularly in areas where it is anticipated that seals will be emplaced.

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Chapter 24.4
MILITARY AND DEFENSE INSTALLATIONS

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24.4.1 INTRODUCTION

Native Americans, particularly those in the Southwest, are known to have used natural caves as their homes. Throughout North America and the remainder of the world, people have used natural and man-made caves and quarries for storage of materials and equipment and as a place to gather for security. These underground facilities restricted avenues of attack and made for easier defense of the facility.

Although the continental United States has been fortunate not to be involved with a war on its soil since the Civil War, US soldiers are aware of the benefits of underground facilities from their experiences in World Wars I and II, the Korean Conflict, and in Vietnam. These experiences vary from the common use of foxholes for personal protection to massive concrete bunkers along the Normandy beaches and the Maginot Line in France and Germany.

One of the most famous underground fortifications is the Rock of Gibraltar, which was captured by the British from the Spanish in 1704 after fierce fighting. The labyrinth of tunnels and chambers extend for more than 25 mi (40 km) within the Rock of Gibraltar. Today the Rock of Gibraltar serves as the headquarters for the Gibraltar Mediterranean Command of the North Atlantic Treaty Organization (Guthrie, 1987).

The development of nuclear weapons in the 1940s with their massive destructive power had a major impact on the use of shallow and deep underground facilities for defense purposes. This chapter of the *Handbook* describes the types of facilities that range from near-surface to deep underground and from man-made to natural, and describes the military and defense uses of underground facilities. Because of their purpose and use, it is not possible to describe these facilities in detail. However, the information provided should give a general idea of underground facilities for military and defense purposes.

As with any underground facility, the method of construction is a function of the geology, size of opening, depth, and degree of rock disturbance acceptable to the long-term operation of the facility. Most of the early deep underground defense facilities in rock were excavated with conventional drill-and-blast techniques. More recent facilities have been excavated with controlled blasting techniques where drillhole half rounds are clearly visible after muck removal. It is also possible to use full-face tunneling machines. However, to date, this has not been standard practice.

Because an underground facility is primarily dependent on the rock in which it is constructed for its strength or ability to resist nuclear weapon-generated ground shock, it is important that the excavation of the rock leave the remaining rock in an undisturbed state. A comparison of tunnel construction methods reveals that mechanical excavation techniques, especially full-faced tunnel boring machines, leave the rock surrounding the tunnel in essentially an undisturbed condition. Other methods of excavation, particularly uncontrolled drill and blast, can produce significant overbreak and rock fractures that extend several feet (meters) into the surrounding rock. Construction of underground facilities should therefore consider the in situ rock conditions and use an appropriate method of excavation to preserve the inherent strength of the rock.

Tunnel support is clearly a function of the type of rock and its associated in situ characteristics, groundwater conditions, depth of facility, and magnitude of predicted dynamic weapon-generated ground shock conditions. In most instances, deep underground facilities are located at a depth so as to avoid severe dynamic effects, and therefore standard support techniques used for civil and mining projects are most often used to reinforce tunnels for defense purposes.

In some instances it may be beneficial to install complex rock support systems. For example, to provide extra protection for unique or critical pieces of equipment or activities and personnel or to strengthen locally weak and jointed rock, it may be necessary to install composite structural liners of steel plate and reinforced concrete or structural liners of reinforced concrete or steel surrounded by crushable material. Lightweight, nonporous, low-strength concrete has been used as crushable backing for test structures at the Nevada Test Site.

Complex support systems are inefficient and very expensive to install. In most instances, large quantities of rock must be excavated to accommodate the large thicknesses of steel, concrete, and crushable packing.

Preliminary design of reinforcing for tunnels and shafts is normally accomplished by applying simple, closed-form methods of analysis (Newmark, 1969; Hendron and Aiyer, 1972). Final designs frequently rely on computer codes that have been calibrated against actual results from structural tests conducted at the Nevada Test Site. The actual design criteria are specified by the design agency and are frequently a function of the purpose and endurance requirements of the facility.

Because the audience for which this *Handbook* is intended is knowledgeable in the excavation and support techniques available to the underground mining and construction industries, and due to the many variations possible, excavation and support techniques and systems are not discussed in great detail.

24.4.2 TYPES OF UNDERGROUND FACILITIES

Underground facilities can be shallow or deep. They can be formed by new excavation or by the modification of naturally formed caves or previously excavated cavities and tunnels. This section will describe the primary types of facilities, near-surface cut and cover, shallow underground, deep underground, and existing excavations.

24.4.2.1 Near-surface Cut and Cover

Hardened concrete bunkers built in World War II, Minuteman missile silos in the Dakotas and Wyoming, and weapon-storage magazines are all excellent examples of defense facilities constructed as near-surface cut and cover facilities. These facilities are generally fabricated from reinforced concrete and are surface flush or covered with an earthen mound.

Construction of these facilities, for the most part, follows conventional civil construction procedures. However, the one major modification to conventional construction is the concrete thickness and quantity of steel reinforcement added to the concrete to form the structural elements of the facility. Special design manuals are available from both the US Army Corps of Engineers (Anon., 1986a) and the US Air Force (Schuster, Sauer, and Cooper, 1987) to aid in the design of near-surface cut and cover facilities.

Cut and cover facilities represent the first effort to improve survivability from conventional and nuclear weapon effects and terrorists. Fundamentally, they have been used to place operational military defense systems out of the way of the aboveground weapon-generated blast effects. These cut and cover facilities are relatively inexpensive to construct when compared to deep underground facilities but have lower limits of strength or "hardness." As nuclear weapons become more accurate, these facilities become less survivable and are less attractive for military and defense facilities.

24.4.2.2 Shallow Underground

This class of underground facility is generally less than 100 ft (30 m) deep and may be constructed as a cut and cover or as an underground excavation project. Command modules for the Minuteman missile silos were constructed as cut and cover facilities but have significant amounts of earth above them to enhance their protection from nuclear weapon effects. The decision to use a shallow underground facility in lieu of a near-surface cut and cover facility is generally dictated by the contents of the facility, its criticality of mission operation and tradeoffs between depth, and simply adding more concrete and steel to a surface-flush facility.

Shallow underground facilities are normally equipped with stronger blast doors and valves and more complex air filtration systems than the near-surface facilities to permit greater use in a nuclear contaminated environment. These facilities are more costly to construct but offer increased survivability to the contents of the facility.

Frequently, shallow underground facilities are provided additional protection against earth-penetrating weapons by placing large rock rubble or concrete slabs on the surface directly above the buried facility. The layers of rubble or concrete are designed to damage, deflect, or reduce the effectiveness of an attacking weapon.

Underground civil defense facilities throughout Scandinavia are constructed using drill-and-blast tunneling procedures. These facilities are seldom more than one or two tunnel diameters deep. In many instances, civil defense facilities in Scandinavia are excavated into the side of hard-rock hills.

24.4.2.3 Deep Underground

Deep underground facilities are generally several hundred feet (meters) deep and are constructed by tunneling horizontally into the side of a mesa or mountain with sufficient relief to provide the desired depth of earth cover. It is also possible to construct deep underground facilities by ramping or shafting vertically down to the desired depth of the facility. In most instances, where land is readily available, ramping down is more economical than shafting for muck removal during the construction of a large underground facility and for ease of buildback and routine operations.

Deep facilities can be excavated by drill and blast or mechanical means. Today mechanical means would be preferred because they result in less damage to the in situ rock around the opening.

Deep underground tunnels and cavities are not sensitive to damage from airblast effects if blast doors and valves of adequate resistance are provided for all openings connecting the underground facility with the surface. However, highly accurate nuclear weapons detonated as ground bursts within close proximity of a point directly above the facility generate a large ground shock that can collapse the rock tunnels. For maximum effectiveness and economy, it is important to design the facilities at a depth that offers balanced survivability against airblast effects and ground shock effects. Shock and vibration isolation for personnel and equipment, effective airblast doors and valves, and tunnel support systems must be balanced one against the other to eliminate overdesign of one or more of the protective systems.

24.4.2.4 Existing Excavations

Excavating rock and soil to develop underground facilities is expensive. Therefore, an existing cave, mine, or rock quarry that meets or nearly meets operational needs and provides the necessary level of protection against weapon effects and terrorist attacks represents a real cost savings. Even if some expansion and modification must be completed to meet all requirements, the development of existing or previously developed underground openings is attractive to the defense planner.

Unfortunately, mines are initially opened for the recovery of natural resources such as coal and minerals. Consequently, most mines are located in thin beds or seams or in highly fractured or weak host rock. Therefore, mine operations in the United States seldom produce underground space that is suitable for defense facilities.

Limestone quarries represent the one major exception and, if available, could be used for defense facilities. In the United States, few quarries of sufficient size exist except at Kansas City, which has been developed as an extensive commercial and industrial complex (Stauffer, 1974) (also see Chapter 24.2).

24.4.3 MILITARY USES OF UNDERGROUND FACILITIES

Underground facilities are defensive in nature. They are designed to provide protection against a direct attack from conventional and nuclear weapons and from terrorists so that the protected assets can be used to recover from that attack or to serve as a basis from which to launch a counterattack. The US government has developed a limited number of underground facilities. Currently, underground facilities are used for the storage of nuclear weapons; storage of other war fighting resources such as gas and oil; weapon emplacement; command, control, and communications centers; and nuclear weapons test and development. Research and development projects have focused on plans for the emplacement of strategic weapons in shallow and deep underground facilities.

Other countries throughout the world have developed underground facilities for a much wider variety of defense purposes. Sweden, as an example, is known to have underground hangars for military fighter aircraft, underground dry dock and repair facilities for submarines and small ships, and underground facilities for retractable radars and coastal artillery (Winqvist and Mellgren, 1988).

Civil defense is a major concern of every community in Scandinavia. According to published literature from Soviet/Warsaw Pact Countries, these countries also place major emphasis on developing whole cities and large civil defense facilities underground (Goure, 1978; Anon., 1988).

The next segment briefly describes these uses of underground facilities

24.4.3.1 Storage and Protection of Weapons, Equipment, and Supplies

This category of underground installation probably represents the simplest of all. Its primary function is to provide environmental, sabotage, and blast protection to stockpiled weapons, equipment, and supplies. With limited use by personnel, air handling, lighting, and overall safety requirements can be minimized to that required only for long-term preservation of the stored materials or equipment. Water tightness and humidity control represent the critical components of the environment of these installations. Temperature control is less critical since the earth surrounding the installation provides a very satisfactory and stable temperature environment, unless extraordinary conditions are encountered.

Weapon storage facilities constructed as cut and cover bunkers can be found at almost any military installation. US military forces have developed definitive designs for these shallow-buried facilities. Monzano Air Base in New Mexico is an example where tunnels were cut into the side of a hard-rock mountain for the specified purpose of storing weapons.

The use of salt domes and specifically excavated large cavities for the storage of crude oil and the use of underground quarries for the storage of equipment, records, and supplies are examples of using underground space for the storage of critical military resources. Underground limestone quarries and mines for other minerals are available in nearly every country. Many countries, including the United States, have either retrofitted some of this existing underground space for civil defense, including natural disasters, or developed detailed plans that could be implemented in relatively short order should international events dictate such action (Wright et al., 1976).

24.4.3.2 Command, Control, and Communications Centers

Communications represents one of the most critical elements of any military system. Without the capability to accurately transmit information to command, control, and gather data, it is exceptionally difficult to provide for the defense of a country. With the advent of intercontinental ballistic missiles, the need for secure command, control, and communications facilities in the continental United States has been greatly emphasized. President Reagan in October 1981 clearly stated that this country needed and would develop such secure facilities.

One of the oldest and largest deep underground facilities that forms a vital link in the defense of the United States is located in Cheyenne Mountain outside Colorado Springs, CO. This facility was constructed in the 1960s and consists of large openings, each containing three-story steel buildings on steel springs and shock absorbers. The rock is reinforced with standard grouted rock bolts. Fig. 24.4.1 shows one of the tunnels with its steel building (Norrell, 1963).

In the event of a nuclear war, vital information would be gathered by satellites. To ensure that the data collected would be available to national civilian and military leaders, special shallow-buried facilities have been constructed. According to a published article (Anon., 1986b), the US Air Force has constructed a satellite tracking station in Spokane, WA, capable of withstanding and remaining functional during a nuclear attack. This facility has a 52-in. (1320-mm) thick concrete roof and 18-in. (457-mm) thick concrete walls built integrally to ensure adequate protection against electromagnetic pulses associated with nuclear blasts.

Norway and Sweden have developed shallow-buried retractable radar systems as part of their coastal defense system. Fig. 24.4.2 shows an artist's sketch of one of these facilities. The facilities were constructed in hard rock, using carefully controlled blasting procedures. In peacetime, these radar facilities support civil aviation operations (Winqvist and Mellgren, 1988).

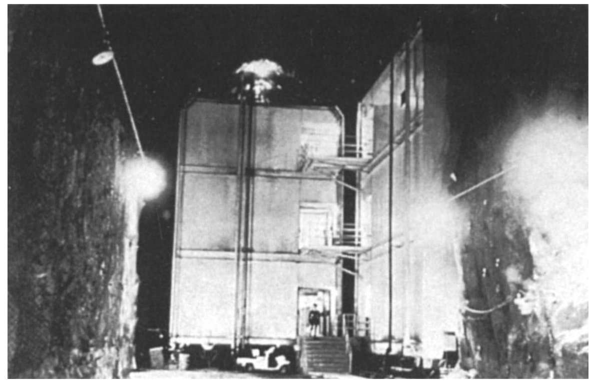

Fig. 24.4.1. Typical tunnel cross section and interior three-story building at Cheyenne Mountain complex (permission: US Air Force Space Command).

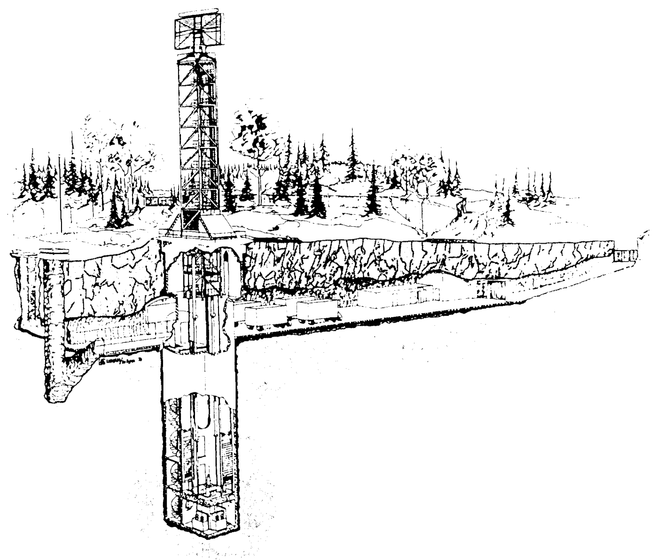

Fig. 24.4.2. PS-80 radar station with vertically retractable antennae (permission: Royal Swedish Academy of Engineering Sciences).

24.4.3.3 Weapon Emplacement and Transport

The Maginot Line still represents one of the best examples of a mechanized artillery system that was constructed in an underground facility. Modern-day versions of that system are currently operated by Norway and Sweden (Fig. 24.4.3). In these countries, the facilities are several stories deeper than those at the Maginot Line and are constructed in hard rock. From the ground surface, only the turret is visible. The separate units are independent as far as electricity, water, oil, and personnel requirements are concerned. Stringent demands are made on the environment in which personnel live (Winqvist and Mellgren, 1988).

The Royal Swedish Navy has developed an underground facility at Musko in hard rock for the support of submarines and small ships. These large tunnels are located in competent rock and, as shown in Fig. 24.4.4, require minimum rock reinforcement (Winqvist and Mellgren, 1988; Anon., 1986c).

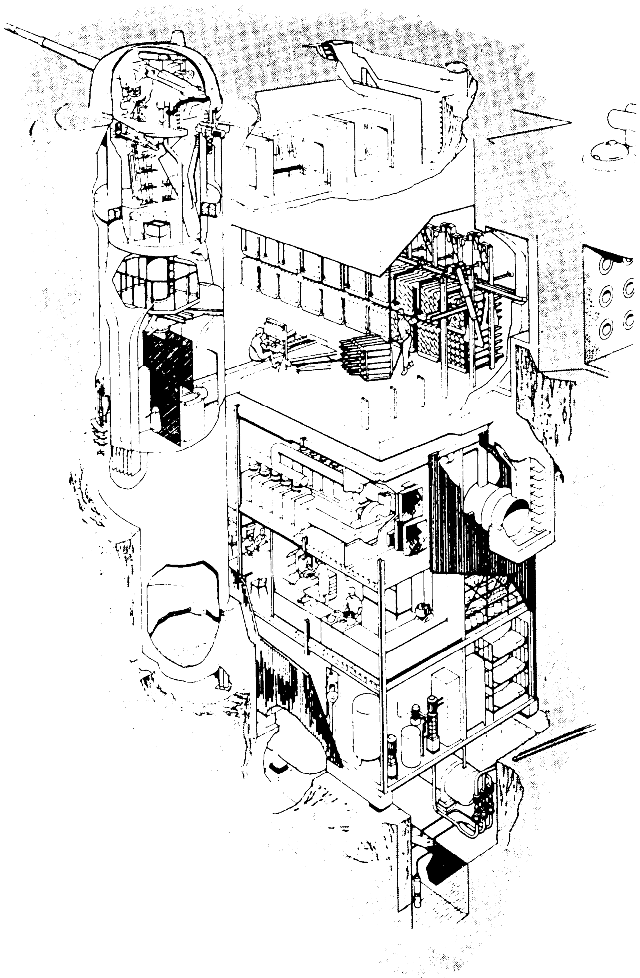

Fig. 24.4.3. A 120-mm ERSTA turret gun (permission: Royal Swedish Academy of Engineering Sciences).

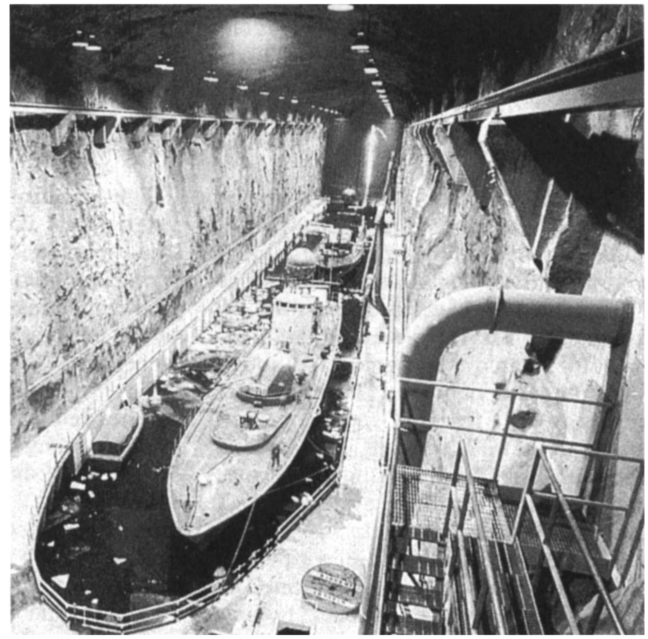

Fig. 24.4.4. Royal Swedish Navy facility at Musko in Sweden (permission: Tunnels & Tunnelling, London).

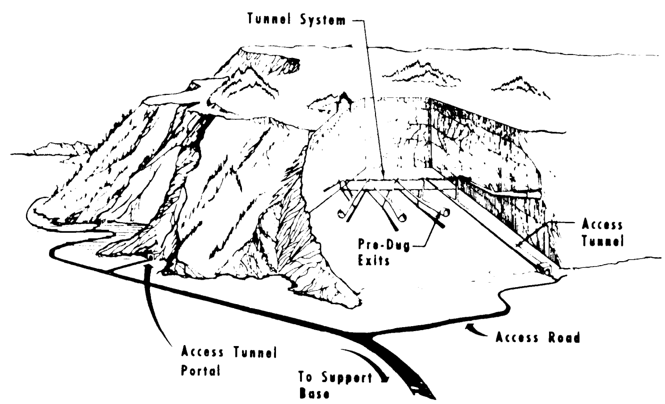

Fig. 24.4.5. Cutaway view of an artist's concept of a deep underground missile base (permission: DNA).

Strategic nuclear weapons have formed the backbone of the US deterrent posture for nearly 30 years. As new nuclear weapon systems are developed by potential enemies, it remains the goal of this country to protect its strategic retaliatory force (Berry, 1982). To this end, the US Air Force completed a major research and development project to design a deep underground missile basing system. One of the early artist's concepts is shown in Fig. 24.4.5. The basing concept consisted of several hundred miles (kilometers) of subway-size tunnels through which missiles on transporters would move to randomly assigned launch portals should the United States come under nuclear attack. Other, more closely spaced tunnels would be used for personnel living quarters, power plants, administrative offices, and other support activities. A unique requirement of this concept of missile basing was the need to "dig-out" through potential rubble zones using mechanical means of rock excavation. This concept was demonstrated by the US Air Force at the Nevada Test Site (Phelps, 1983).

24.4.3.4 Weapon Test and Development

In 1963, then President John F. Kennedy signed a treaty with the Soviet Union to halt all atmospheric testing of nuclear weapons. Immediately following that treaty, the United States and the Soviet Union began testing nuclear weapons deep underground. The testing procedure is designed to ensure that all weapon fission products are contained in underground cavities and tunnels. The Defense Nuclear Agency (DNA), as the designated agency to investigate nuclear weapon effects, and the Department of Energy (DOE), as the designated agency to develop nuclear weapons, conduct underground tests at the Nevada Test Site (NTS).

DOE tests are accomplished in deep vertical holes drilled into the flat valley floor of the NTS. The holes are drilled with blind down-hole drills and lined only where necessary, generally only to the depth of the alluvial overburden. After the nuclear device and the diagnostic instrumentation have been installed, the hole is backfilled with material in accordance with testing procedures to ensure full containment of the fission products.

DNA conducts its weapon effects tests in a series of underground tunnels. Two types of tunnel configurations have been

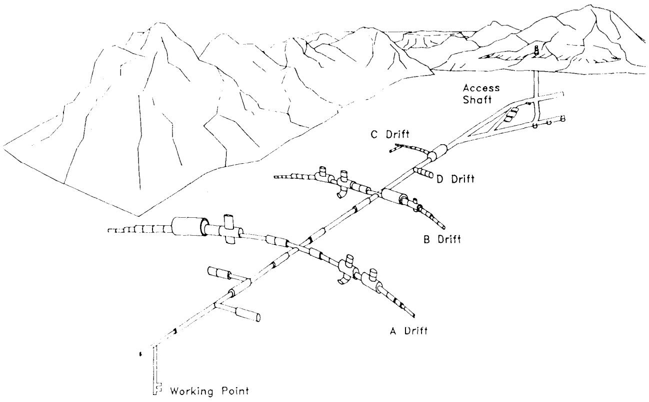

Fig. 24.4.6. Dedicated underground structures test (permission: DNA).

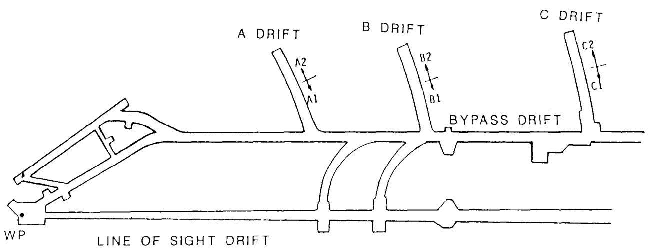

Fig. 24.4.7. Underground structures tests as part of line-of-sight weapon effects test (permission: DNA).

used for testing the hardness and survivability of underground tunnels and cavities. One tunnel configuration is a dedicated test of techniques specifically designed to provide the necessary support or reinforcement to the rock tunnels so that they can withstand the dynamic effects associated with nuclear weapon-generated ground shock. An example of this type test configuration is shown in Fig. 24.4.6. Various rock support or reinforcement techniques including rock bolts, composite concrete and steel liners, and concrete and steel liners with lightweight crushable concrete backpacking have been tested as part of these tests.

The second tunnel configuration used by DNA consists of a horizontal pipe, referred to as a line-of-sight (LOS) pipe, that guides selected elements of the nuclear fission process to impact on test objects, such as missile guidance components, so that the performance of these components can be evaluated in the selected nuclear weapon environment. As shown in Fig. 24.4.7, DNA frequently constructs a limited number of test tunnels, shown as drifts A, B, and C, for evaluation against the associated nuclear weapon-generated ground shock effects.

24.4.3.5 Protection of Personnel, Civil Defense

Civil defense, or the protection of a country's population in time of war, is the goal of every country. The United States, through its Federal Emergency Management Agency, develops plans for the safety and security of people from natural disasters as well as from wartime conditions (Wright et al., 1976). Specific shelter areas have been constructed in the basements of office buildings in large metropolitan areas (Chester and Zimmerman, 1987). In addition, plans have been made for the use of existing underground space in the event of a major war. Other countries, including the Soviet Union, have elaborate system of underground facilities for civil defense purposes (Goure, 1978; Anon., 1988).

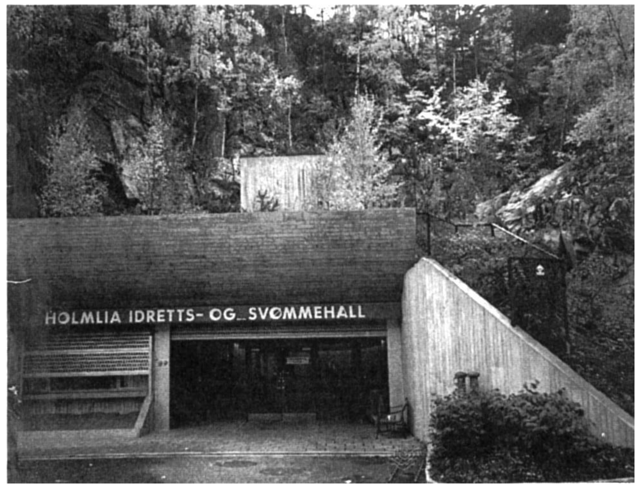

Fig. 24.4.8. Entrance to underground civil defense and sports facility at Holmlia, Norway (photograph: D. Piepenburg).

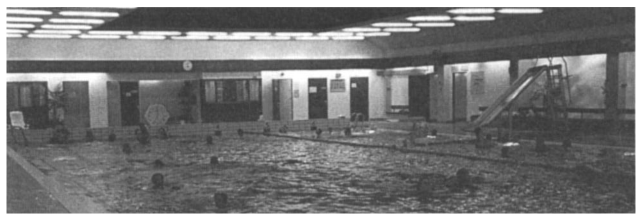

Fig. 24.4.9. Swimming pool in underground civil defense and sports facility, Holmlia, Norway (photograph: D. Piepenburg).

Sweden and Norway have developed an elaborate system of underground civil defense facilities (Winqvist and Mellgren, 1988; Rygh, 1982; Broch and Rygh, 1988). The extent of these facilities can best be portrayed by the fact that there is shelter space for some 6 million people in Sweden and 2 million people in Norway. This corresponds to 70% of the Swedish population and 50% of the Norwegian population.

In both Norway and Sweden, underground shelters are designed into the masterplan for each community. Moreover, these facilities are designed to meet peacetime sports activity requirements for young and old alike. Shown in Fig. 24.4.8 is the entrance to the Norwegian underground facility at Holmlia. Once inside, the facility is much like any other large sports facility with its swimming pool (Fig. 24.4.9) and its basketball and other indoor courts (Fig. 24.4.10).

The underground civil defense facilities in Norway and Sweden are equipped with air filtration systems, water storage facilities, and, as shown in Fig. 24.4.11, blast doors. Decontamination rooms are also provided near the blast doors to ensure that the inside environment remains free of radiation fallout and other contaminants that might adhere to personnel clothing.

Finland has also developed an extensive network of underground facilities for civil defense. Their underground facilities are constructed in hard rock and are designed for use during peacetime. Some underground facilities in Finland are used for

Fig. 24.4.10. Open courts in underground civil defense and sports facility, Holmlia, Norway (photograph: D. Piepenburg).

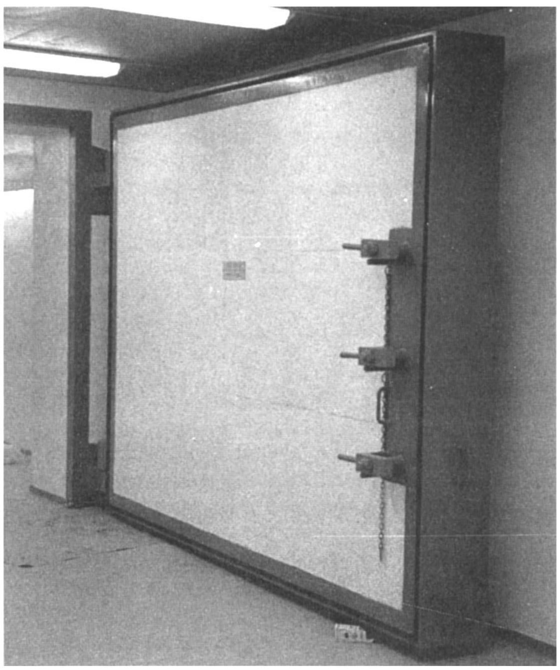

Fig. 24.4.11. Blast doors at Holmlia, Norway, civil defense and sports facility (photograph: D. Piepenburg).

sports halls, car parking, and at least one is used as a concert hall (Saari and Roinisto, 1986).

24.4.3.6 Industrial Plants and Storage of Supplies

The final activity to be discussed where underground space can be effectively used for defense purposes is industrial plants and storage of supplies. For the purposes of this discussion, the authors focus on the facilities at the greater Kansas City underground complex (also see Chapter 24.2).

Kansas City is located on a massive limestone formation that has afforded it significant opportunities for the development of factories, warehouses, and offices in underground openings (Stauffer, 1974). The physiographic and geologic features of this limestone formation are ideal for the development of underground facilities for industrial and commercial purposes. Six of the primary features are shown numbered in Fig. 24.4.12. These features are (1) a massively bedded limestone, (2) sufficient thickness, (3) an overlying impermeable shale, (4) level stratigraphy, (5) a competent overburden, and (6) natural accessibility.

The effective development of this underground space was prompted by several factors:

1. Availability of open space as a byproduct of limestone quarry operations.
2. Energy efficiency of operating in an underground facility where temperature variations are minimized.
3. Quality of the environment for manufacturing sensitive instruments.
4. Relative low cost of developing and maintaining the facility.

The underground industrial space at Kansas City also houses a lawnmower parts factory that uses the high bearing capacity of the rock floors for its heavy tooling and milling equipment. A sailboat factory uses the underground facility because the humidity is easily controlled during the setting of lacquers and glues used in the manufacturing process. Similarly, printing shops and automotive manufacturers use the quarried space to take advantage of the easily controlled humidity. According to Stauffer (1974), metal machinery and factory equipment do not rust. Spare parts can be stored for long periods without damage. Such companies as Allis Chalmers Farm Equipment and Ford Motor Co. store parts in the quarries at Kansas City.

Low or easily controlled humidity and the ease with which overall security can be maintained are attractive features to companies for storage of critical records. Large storage vaults protect records for use in case of natural or wartime disasters.

24.4.4 SUMMARY

The structures described in this chapter represent but a select number of the underground facilities used primarily for defense purposes. Many countries have developed extensive complexes of these facilities. However, the United States, because of its isolated location relative to the European and Asian continents, has not found it necessary to invest the same level of resources in underground facilities.

Technically, the capabilities exist in this country to construct underground facilities and reinforce them to withstand the ground shock effects produced by nuclear weapons. However, because the initial costs can be high, other forms of defense have been pursued by the US Department of Defense.

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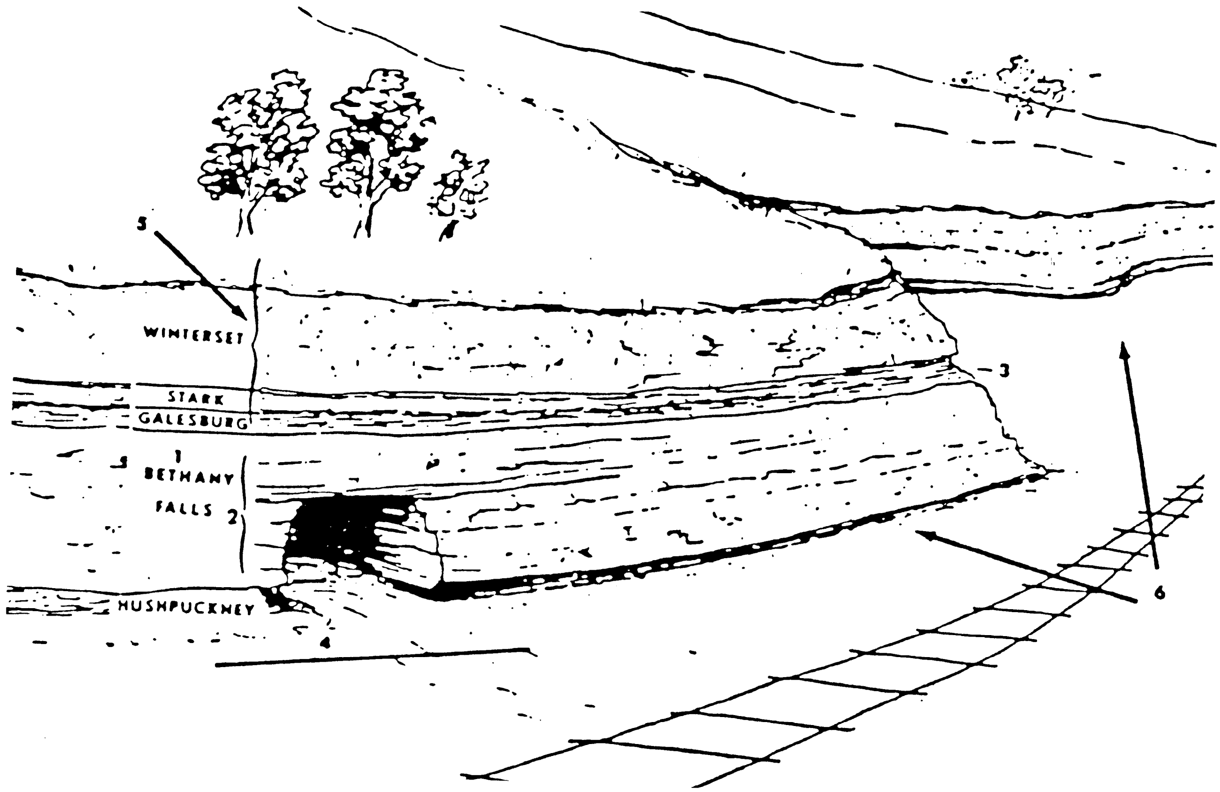

Fig. 24.4.12. An artist's sketch showing six features contributing to the development of underground space (permission: US National Academy of Sciences, Tunneling Technology Newsletter).

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Section 25 Postmining Operations

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Chapter 25.0
INTRODUCTION

RICHARD B. STEIN

Section 25 has been assembled to give the geological, mining, or metallurgical engineer an overview of some of the important downstream operations that typically follow mining activities. This subject is obviously too broad and complex to be covered in detail within this *Handbook*. However, the engineer will be interested to learn about the basic fundamentals, and especially how they might affect the economics of the total project from prospecting and mining through to metals recovery and marketing. In this way, it is hoped that the engineer will have a better overall appreciation for the technical and commercial prospects of a mining and metal-producing project.

Chapter 25.1
METALLURGICAL SAMPLING AND TESTING

ALLAN D. TAYLOR

In the first stages, the value of mineral deposits is often judged from chemical analysis of the available samples. However, very soon there arises a need to make an economic assessment of the potential project. Unless the ore will be shipped and sold without treatment, a determination must be made of the amount of valuable mineral that will be recovered into salable products and the quality of these products. This requires a metallurgical testing program. Chronologically, in the life of a mine, it commences during the mineral exploration stage and continues through exploitation (Section 5).

The ultimate objectives of the metallurgical test program are to:

1. Define the optimum treatment scheme.
2. Define the product characteristics.
3. Define the geometallurgy of the deposit.
4. Develop the production schedule for the project.
5. Develop criteria for the commercial plant.
6. Produce the final metallurgical report.

The objectives listed are all interrelated, and no single one can be attained without considering the impact of others. Consequently, attaining the objectives must be approached in stages, with each phase of the work designed to produce a higher confidence level for the combined objectives. For example, it would be foolish to think the optimum treatment scheme could be defined independently of the commercial plant criteria; these criteria will in turn determine the plant costs, the product characteristics, and the schedule of production that can be maintained.

The final metallurgical report should contain all of the information developed in the test program (both positive and negative) and the information required for an independent reviewer to assess the quality and reliability of the work. It should express the confidence that could be justifiably placed in a project evaluation based on that work. Further, the report should be a complete and reliable source for all process-related information required by an independent contractor to design and construct the commercial treatment plant, including major equipment criteria.

One early goal of the program will be to develop a clear understanding of what has been termed the *geometallurgy* of the deposit. This includes the spatial distribution of the values and their metallurgical response characteristics. This understanding is absolutely essential to the development of a rational mining and treatment plan, even on a conceptual basis. Only the depth and detail of that understanding, and the confidence level that may be assigned to the projected results, will vary as project development proceeds.

Developing the metallurgical testing program to define the geometallurgy of the deposit and select the optimum treatment scheme involves the utilization of all of the available information on geology, mineralogy, chemistry, and metallurgy, Also required are the mining plan or plans and their influence on the production of the ore that will be delivered to the processing facilities over the effective life of the operation. The development of this knowledge into a clear definition of the project is iterative, and will only be complete when the deposit has been mined out. The understanding must be developed to an appropriate level for each stage of the project development, from preliminary assessment through prefeasibility, detail design, and operation of the project.

In order to proceed with definition of the geometallurgy, the sample collection and testing program must be carefully planned and executed. Unfortunately, there is no simple formula for the development of the plan. The quantity and distribution of samples taken and the testing to be done must be assessed on the basis of the available knowledge. Each succeeding phase of the test program must be designed to bring the level of that knowledge to a new, higher plateau.

The samples that are readily available for early testing may be quite limited, but it must be recognized that the information derived can be no better than the samples. If they are not reasonably representative of the ore to be produced, this phase of the work may have to be repeated. In the meantime, decisions based on the results may be wrong.

If the deposit is near surface, and trench samples are available, there is usually no problem in obtaining samples for metallurgical testing. For deeper deposits, the drill cores taken for geological, mineralogical, and chemical analysis will often provide suitable material for the initial bench-scale metallurgical testing. However, while some metallurgical unit operations can be adequately assessed on the basis of laboratory testwork, others may require larger-scale testing. Crushing and rod and ball mill grinding can often be sufficiently proven by laboratory testing, but autogenous and semi-autogenous grinding almost always require pilot-plant-scale testing, at least prior to final plant design, and this may require several hundred tons of sample. Bulk flotation, which is a common means for concentrating many sulfides and some oxide and nonmetallic ores, can often be assessed on the basis of laboratory tests. The differential flotation of complex ores, such as copper-molybdenum, may require larger-scale testing, since the secondary mineral content may be too low to allow the collection of adequately sized samples from laboratory tests. Gold ores, and particularly placer gold deposits, are notoriously difficult to assess, partly because of the very low quantity of valuable mineral—which is the target—and the tendency for it to be concentrated in nuggets. Very large samples may be required for accurate assessment. In any case, sufficient sampling and testing must be done to provide the necessary information, whatever the difficulty.

Since mineral deposits are seldom if ever uniform, it is always a mistake to try to produce a single representative sample and to place faith on the results of testing. The appropriate approach is to identify the extremes and the more average types of ore. Then samples from each of these groups are tested until sufficient understanding has been developed to support an acceptable level of confidence in the interpretation of the deposit characteristics. Naturally, a great deal of care must be taken in applying variability established for one unit process to a different one, since the basis of separation may be different.

Once underway, the metallurgical test program must be carefully monitored to assure that all test data are analyzed to provide the maximum guidance for subsequent testwork. The test directors must be alert to indications of the need for modifying the program, and must be willing to make the appropriate changes when needed. However, the program cannot be allowed to drift from one idea to another without clear objectives. The attempt to merely follow the directions indicated by the results of current tests is a program destined for gross inefficiency and possible failure.

25.1.1 ORE CHARACTERIZATION

In most mineral deposits, there is sufficient variation to justify the development of several categories to describe different types of ore. Chemical analysis is one appropriate property, as is mineralogy, but neither is sufficient. The ore types should properly be based on the expected metallurgical response as well.

It is always best to have a substantial amount of petrographic work done on many samples before establishing the types, since simple sight criteria can be misleading. The petrographic work should be directed to yield the maximum of metallurgical information, since differences that are not significant from a geologic or ore genesis view may be critical to the metallurgy. In particular, the size of the valuable and gangue mineral grains is of vital interest to the metallurgist, and relatively slight variation in the sizes can have a profound effect on mineral dressing response. The degree of interlocking of valuable and gangue minerals is also of prime interest, as well as the particular minerals involved in the interlocking. Naturally, the relative amounts of the different species is important, since many ores contain more than one mineral carrying value, and each mineral will usually have a different metallurgical response. The gangue minerals are also important, since each one may cause different degrees of difficulty in the separation process.

For example, even small amounts of talc can make flotation difficult, and relatively small changes in carbonates may mean the difference in the economic viability of an acid leach procedure. Even low levels of oxidation may be very important in certain cases such as flotation, since even a slight tarnish of the mineral surface may drastically affect its separation characteristics. Certain other properties, including hardness, brittleness, fracture patterns, specific gravity, etc., may be important, as well as rheological functions of slurries derived from ground products. Unfortunately, important differences often go undetected in the early examinations. Only alert cooperation between the petrographer and the metallurgist will provide the proper recognition of these characteristics, and of their importance.

Finally, the metallurgical tests become one more ore characterization tool, usually the final criteria, but are often more cumbersome than other procedures. Therefore, one of the goals of the test program is to develop correlations with other characterization methods that allow confident prediction of the metallurgical results to be expected from a particular ore type or a specific lot of ore, based on simpler evaluation methods.

25.1.2 SAMPLING

Each metallurgical test sample should be selected to provide the appropriate material for the testwork to be performed. The amount of sample, the method of taking it, the location within the deposit, and the chemical, mineralogic, and physical characteristics should be determined beforehand, to the extent possible. This requires that the metallurgical test program be defined in as much detail as possible prior to attempting to take the sample.

It must be recognized that proper sampling is expensive, and the subsequent handling and testing usually even more so. Further, the decisions that are based on the test results will have a major effect on the future of the property. Therefore, the entire operation merits the full attention of all parties that can contribute to the selection of the sampling criteria, the performance of the actual sampling, the characterization of the sample, and the subsequent handling and storage of the material.

The selection of the sample criteria will be based on the best available understanding of the geology, the chemical and mineralogic composition of the ore zones, and the mining plan. The objectives of the test program for the particular sample must also be considered.

The quantity of material needed to provide a representative sample will clearly increase as the maximum rock size in the sample increases. The mineralogical characteristics are also a factor in selecting the quantity of material needed for a representative sample. For example, ores containing only a few large, randomly distributed particles of the valuable constituent will require larger samples than more uniform ores. Guidelines for estimation of minimum sample size are available (Cooper, 1983; Ottley, 1985).

The actual sampling technique must not only produce the quantity of sample desired from the selected mineralized zone, but it must deliver the sample in the proper physical condition. Especially for bulk samples, care must be taken that the sampling technique does not destroy the essential physical characteristics of the material. For example, underground mine blasting procedures could produce material too fine or too fractured to provide appropriate samples for autogenous grinding tests.

Finally, the verification of the geological, mineralogical, chemical, and physical characteristics of the sample must be done carefully to assure that the sample does, in fact, have the characteristics desired, and is appropriate for the testing planned. It may be far better to discard a sample that does not meet the specified criteria than to proceed with the testwork on material that will not give the information desired, or may actually produce misleading results.

The last item, the handling and storage of the sample, must be carefully planned and performed to avoid both contamination and any physical or chemical alteration of the sample that could affect the test results. Many metallurgical processes are affected by the size of the material, the moisture content, and exposure to the atmosphere. For example, reactive sulfides may require storage under nitrogen, rather than air.

25.1.3 TESTING

Metallurgical testing may include mineral processing, extractive metallurgy, and refining, depending on the ore and the mineral values to be recovered. Mineral processing alone includes sizing, comminution, concentration, and dewatering. Concentration itself is a broad field, including gravity, froth flotation, magnetic, and electrostatic separation processes. Similarly, extractive metallurgy encompasses hydro-, pyro-, and electro-metallurgical procedures. These are broad fields, but certain basic principles can be addressed. For example, it has already been emphasized that metallurgical testing should be approached as an organized program, with the objectives defined and understood for each phase. Once it has been determined that a deposit contains potentially profitable quantities of values, the objective becomes the determination of the procedure that will most effectively produce the best marketable products. Given that objective, it is obvious that the metallurgical testing must be designed to produce much more than just an elegant technical solution. If the ore has to be ground and concentrated, it is essential to establish that the product will in fact be marketable. It is equally essential that the testing determine not only the conditions and the procedure involved, but also the size and number of commercially available equipment units that will be required, so that a plant design can be developed and the capital and operating costs can be estimated.

Perhaps the most valuable and satisfying professional experience comes from the development of an innovative solution to technical problems, but it must be recognized that major technical innovations are extremely rare, and even more rarely are they a commercial success in a new project. If established commercial procedures are incapable of yielding a profitable project design, it should be recognized that the development of even a very attractive innovation will ordinarily be an expensive and time-consuming program.

The following description of the phases of a complete test program is naturally only indicative of what may be appropriate. Metallurgical test programs specific to heap and dump leaching and in situ mining are found in Chapters 15.2 and 15.3, respectively.

25.1.3.1 Test Program: Phase I

The initial testing will select the most appropriate treatment scheme, whether simple sizing, with or without some comminution or size reduction, or more complex treatment such as concentration.

Additionally, a standardized test for evaluating additional samples as they become available should be developed and verified by sufficient replication to establish the repeatability and the variation that will be expected in the results when treating a single standardized sample. This should be done to allow proper assessment of the significance of future test results.

The standardized test must be applicable to the samples that will be derived from the ongoing exploration and development work. In some cases, the standard test may not need to cover the entire treatment scheme. For example, bulk flotation of complex sulfide deposits may be sufficient for a standardized test even though the final treatment plant may include complex differential flotation circuits.

25.1.3.2 Test Program: Phase II

Phase II work will ordinarily involve the application of the standard test developed in Phase I to as wide a selection of samples from the deposit as is practicable. This is done to confirm the suitability of the treatment scheme and also to further define the geometallurgy of the deposit.

It must be remembered that it is always possible that growing knowledge may require a change to the treatment scheme or to the sampling program, and if this occurs, the entire program, including sampling, should be reevaluated.

25.1.3.3 Test Program: Phase III

Most major metallurgical projects will require a final phase of demonstration testing in a pilot-scale facility to prove the process prior to final design and construction. This type of testing is always expensive, and from that point, the need for thorough planning is perhaps even more important than in the earlier phases.

One attractive approach is to build a pilot facility on the property and operate it until acceptable data have been developed. In this circumstance, the sample production can be carried on concurrently with the testing, and there is much greater flexibility in both the quantity and type of ore that can be made available. It is not necessary to produce excess sample, and there is no risk of running out of proper ore. Occasionally, such a plant becomes the prototype for the commercial facility and actually recovers enough values to offset all or at least a portion of the costs of the test program.

On the other hand, it is usually far less expensive to use an existing facility, if one is available. The access to skilled staff and superior support facilities are also important considerations.

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Chapter 25.2
STORAGE AND TRANSPORTATION OF MINERALS

WILLIAM L. PRICE

25.2.1 INTRODUCTION

Temporary storage of mineral products and their further transportation to processors or customers are essential operations.

Storage is a requirement in order to interface between two discontinuous mineral operations, whatever they may be. Few flowsheets can be planned as completely continuous operations from start to finish. Even though planned as a continuous operation, if any one operation stops for any reason without surge pile storage, then every other phase would have to stop—a catastrophic event. However, with even modestly sized surge pile storage, all the remaining phases can be kept in operation for a reasonable time.

Transportation, in particular, is a discontinuous or batch process as compared to a relatively continuous mining or processing operation. This intermittent movement of materials makes storage facilities at transportation points an absolute essential. The cost of transporting minerals to a processing plant or to the ultimate consumer can easily equal or exceed the cost of producing the mineral itself. This fact necessitates giving serious consideration to facility location with regards to transportation—for example, locating to take advantage of low-cost water transportation vs. rail or truck.

In selecting a storage or transportation system, the normal economic equation of high initial investment coupled with low operating cost vs. the reverse, applies dramatically. As an example, either low-cost, highly labor intensive front-end loaders or a high-cost automated reclaiming method with minimum labor cost may be chosen. Thus an economic evaluation of a number of alternatives must be made in order to select the optimum storage or transportation loading/unloading approach for a given situation.

With a long-term trend toward increased labor rates, the facility designer should also try to predict future labor costs. These could help justify additional first-cost expenditures for storage or transportation systems.

In storage and transportation a general trend over many years has been the replacement of batch-type operations with continuous operations. Examples:

Batch-Type Operations (High Labor Cost)	Continuous-Type Operations (Low Labor Cost)
1. Trucking to destination	1. Belt conveying
2. Loading individual railroad cars	2. 100-car unit trains
3. Using front-end loaders or clamshell grab buckets	3. Using continuous reclaimers, e.g., bucket wheel excavator

The following presentation covers the significant transportation and storage methods that have withstood the test of time. Many innovative schemes have surfaced over the last 50 years, and a few have proceeded as far as their initial operating installations. However, with the process of economic attrition, many have died with no further installations and will not be discussed due to space limitations.

25.2.2 STORAGE

25.2.2.1 Open Storage

A system of piles exposed to the weather predominates in the storage of mineral products, especially for the lower-valued commodities. Unless there are environmental reasons or the mineral can be damaged by the elements, open storage is a natural choice with a seemingly endless array of pile layouts. (Figs. 25.2.1 to 25.2.3) A typical open storage pile is shown in Fig. 25.2.4.

Creating an open stockpile in its simplest form is done by employing a sloped belt conveyor. The mineral discharged to the ground forms the stockpile, and as long as segregation or breakage is not a problem, the simplest open conical pile storage is provided. If the stacking conveyor is mounted on rails or is track- or wheel-mounted to permit it to travel, a variety of tent-shaped piles can be created with unlimited storage possibilities. With the tonnage in a conical pile increasing as the cube of the height, and the tent-shaped pile tonnage increasing as the square of the height, and with no modifying factors, the highest practical pile represents the lowest nominal storage cost per ton. These open piles can be extended outward with earthmoving equipment for additional "dead" storage. Pile height is governed by permissible ground loading, segregation by size, and degradation by falling from the conveyor discharge.

Reclaiming from open piles generally falls into two categories. First, the mineral can be recovered by gravity to some type of feeder that places it on the reclaim belt conveyor (Fig. 25.2.5). Second, mechanical devices can dig and elevate the stored mineral—again to be placed on a belt conveyor. The minimum investment approach involves using standardized earthmoving equipment such as front-end loaders or pan scrapers. This approach also requires the highest labor cost per ton. A gravity-type draw-down conical pile is the next most expensive and suffers from the fact that only about 25% of the pile is available as live reclaimable storage. The remaining 75% must be recovered at a higher cost with earthmoving equipment. To gain higher percentages of live storage, tent-shaped piles are recommended using rotary plows, raking devices (Fig. 25.2.6), or bucket wheel reclaimers (Fig. 25.2.7). While this class of equipment represents significantly higher investment, labor cost per ton is significantly lower since it can be operated remotely from a control tower and can be automated rather easily. The other obvious advantage is the improvement in the percentage of live storage. Crawler-mounted bucket wheel reclaimers may be substituted for rail-mounted equipment.

25.2.2.2 Covered Storage

The significant extra cost per ton of mineral product for covered storage is usually justified in one of two ways: either environmental control regulations require it, or the mineral product will be damaged by exposure to the elements. For low-tonnage requirements, *prefabricated storage bins* can represent an economical solution. They are available in many shapes and sizes and can be quickly erected. They allow gravity reclaim to trucks or conveyor belts with a minimum of engineering.

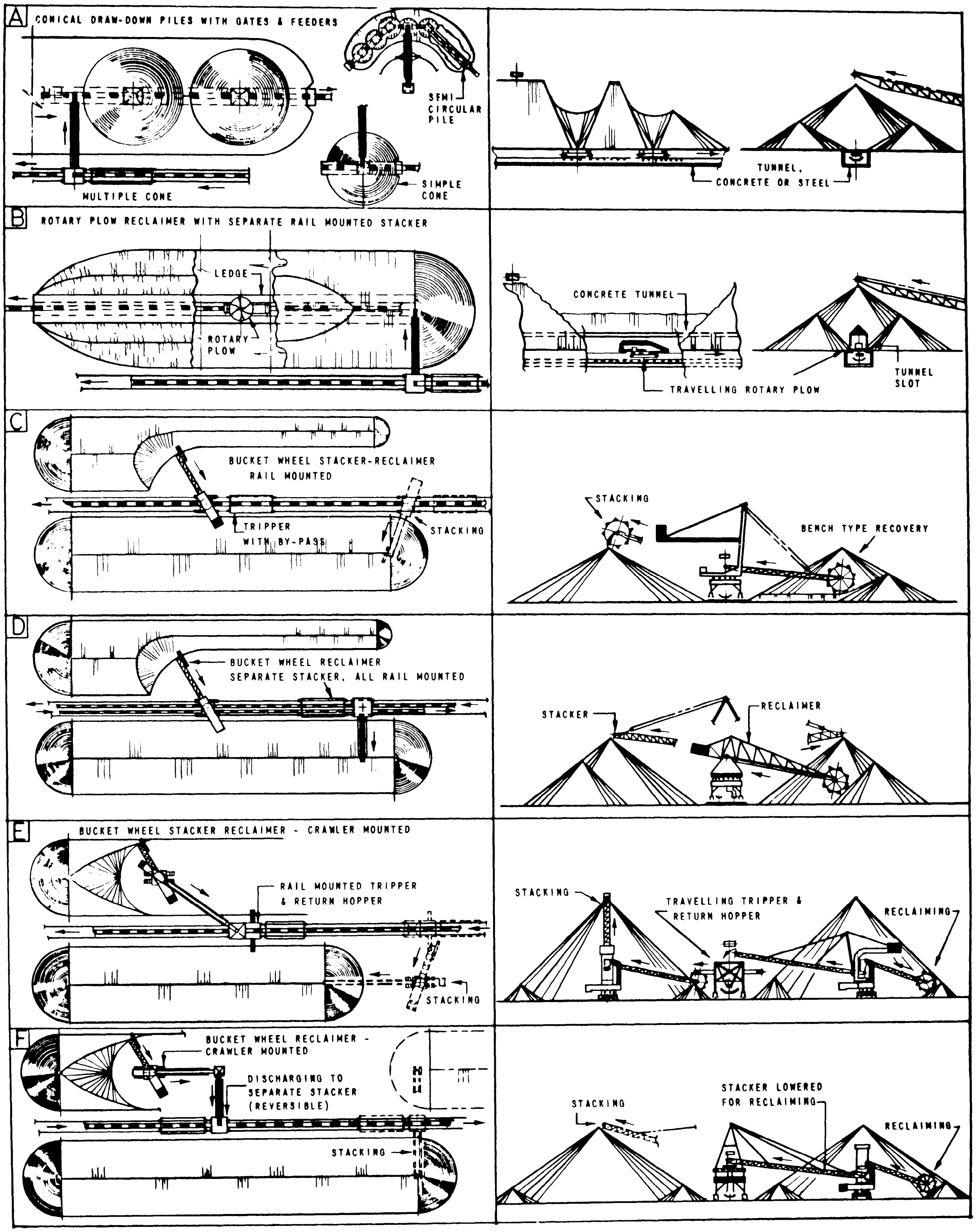


Fig. 25.2.1. Open storage system layouts. A. Gravity-fed conical draw-down piles with gates and feeders. B. Rotary plow reclaimer with separate rail-mounted stacker or alternate supported overhead conveyor with shuttle conveyor. C. Bucket wheel stacker-reclaimer, rail-mounted. D. Bucket wheel reclaimer with separate stacker, all rail-mounted. Stacker on inside or outside tracks. E. Bucket wheel stacker-reclaimer, crawler-mounted. Rail-mounted tripper and return hopper. F. Bucket wheel reclaimer, crawler-mounted. Rail-mounted stacker, reversible.

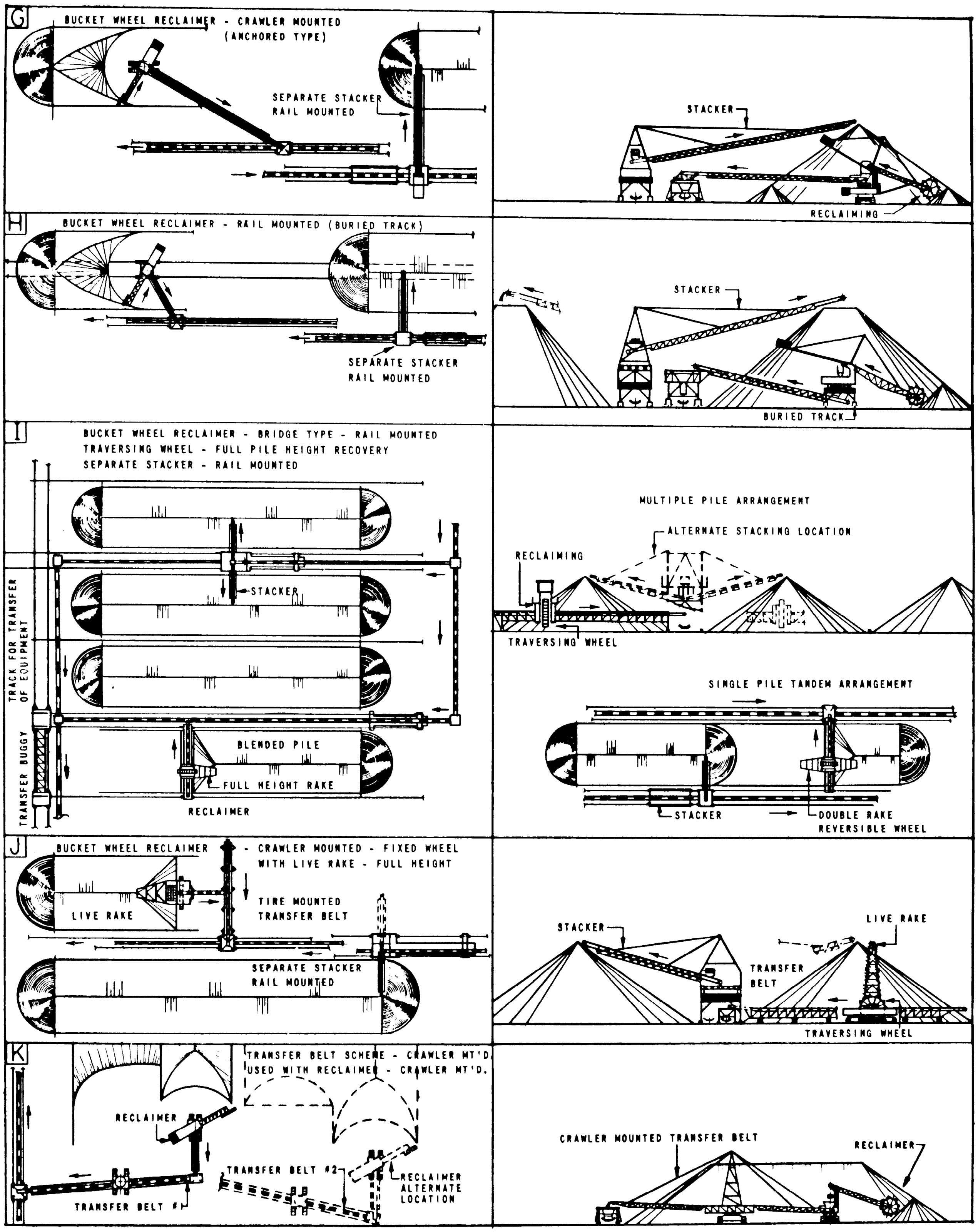


Fig. 25.2.2. Open storage system layouts. G. Bucket wheel reclaimer-crawler mounted-anchored type. H. Bucket wheel reclaimer-rail mounted-buried rail track. I. Blending bucket wheel reclaimer-bridge type-rail mounted. J. Crawler mounted reclaimer-traversing pile, loading a following conveyor. K. Crawler mounted reclaimer loading crawler mounted bridge type reclaiming belt.

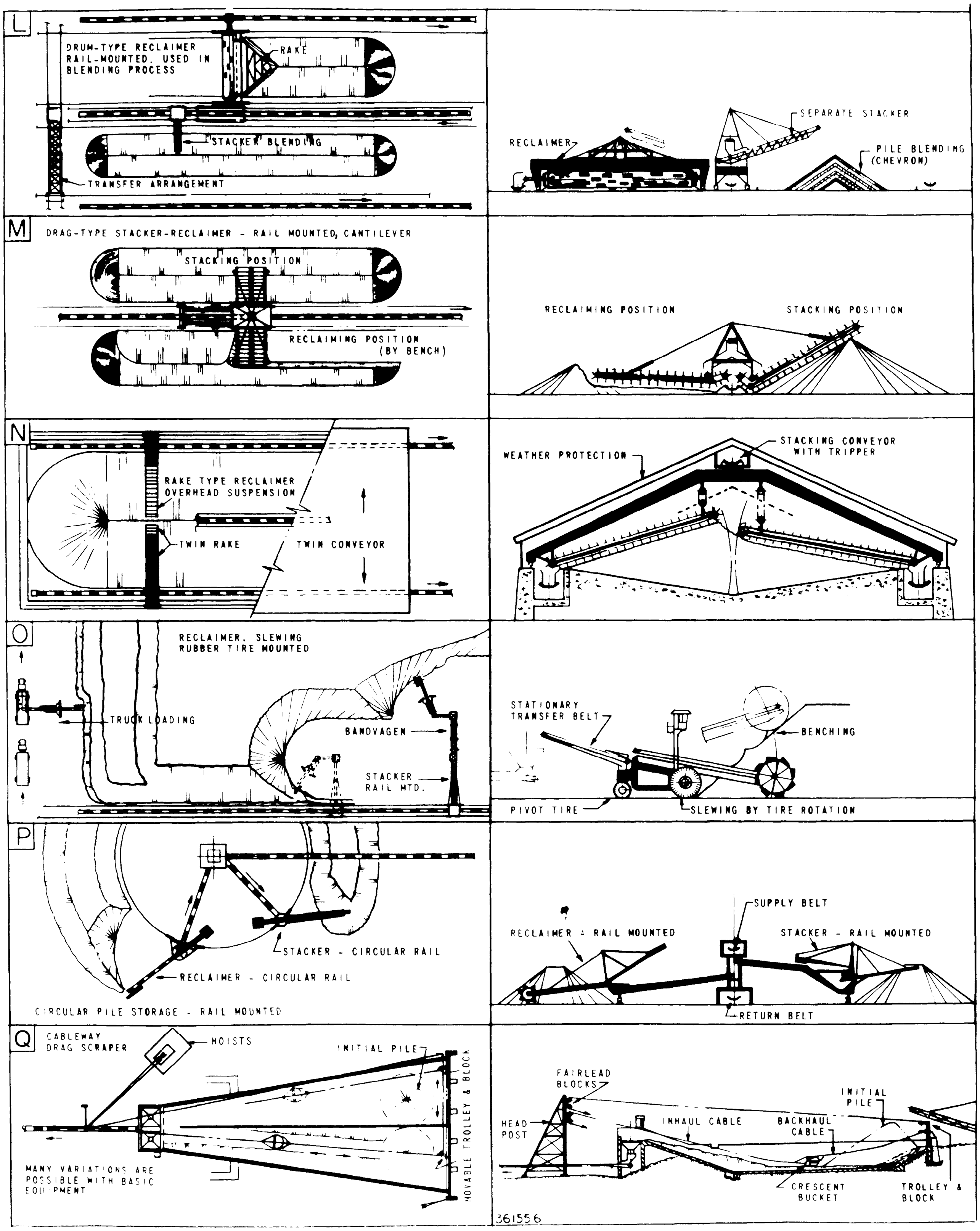


Fig. 25.2.3. Open storage system layouts. L. Blending drum type reclaimer, rail mounted. M. Drag type stacker/reclaimer with common belt. N. Covered rag type twin reclaimer with separate stacking belt. O. Rubber tire mounted reclaimer-developmental. P. Circular pile storage reclaiming-rail mounted. Q. Cableway drag scraper arrangement-low capacity.

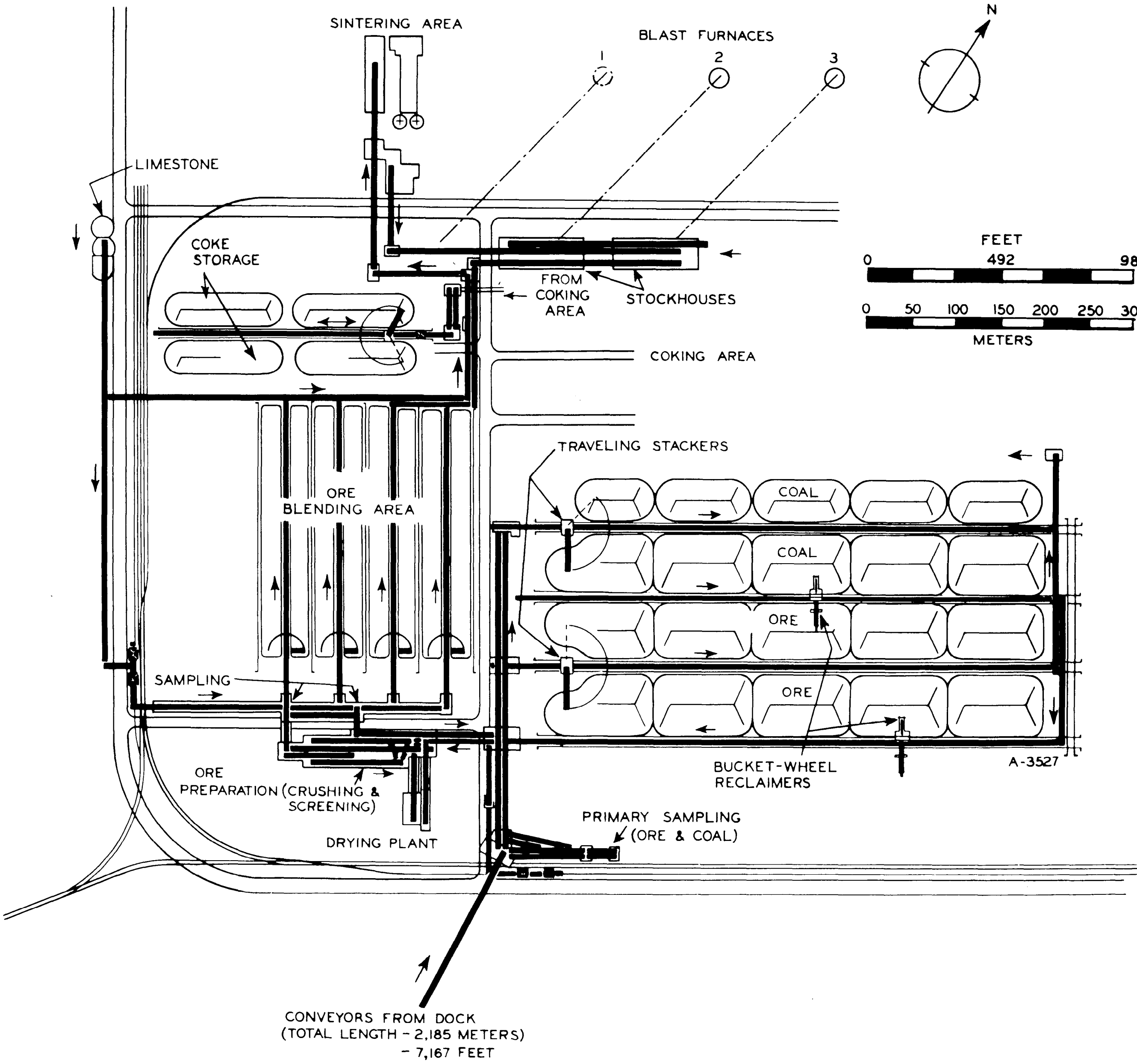


Fig. 25.2.4. Typical open pile storage complex for handling iron ore and coal for a steel mill (*courtesy:* A.T. Yu).

Silos offer economical storage in 12,000-ton (10,900-t) units (Fig. 25.2.8) and have become rather standardized in concept. For the storage of large tonnages of coal, the 70-ft (21-m) diameter slip-formed concrete silo is in common usage. It is fed by a long sloping belt conveyor sized to match the mine output. Reclaiming is by gravity via a set of gates in the silo bottoms. Cross-connected feeder belts bring the product to a common point. From there it can be loaded to unit trains, trucks, conveyor belts, or other systems. The silo contents have a basic problem of "rat-holing" with the mineral hanging up around the perimeter. If this collapses all at once, there could be potentially disastrous results. Concrete silos have failed structurally because of this type of hang-up and quick release of the silo contents.

Gravity-assisted reclaim for silo storage attempts to solve this problem. One popular approach is manufactured under the trade name "Eurosilo." This design offers a rotating horizontal screw or raking device traveling in a radial pattern. The rake moves mineral toward the center of the silo where it drops down a central vertical pipe to a discharge conveyor at grade level. Mineral hangups are thus prevented.

A *hemispherical roof structure* (Fig. 25.2.9) is often used in place of conventional open storage methods. The structural design follows conventional practice but tends to be individually designed for the specific situation at hand. To afford protection for tent-shaped piles, *rectangular roof structures* (free-span) (Fig. 25.2.10) are commonly used to provide covered storage for many mineral products. Although similar in configuration, they also are usually designed individually.

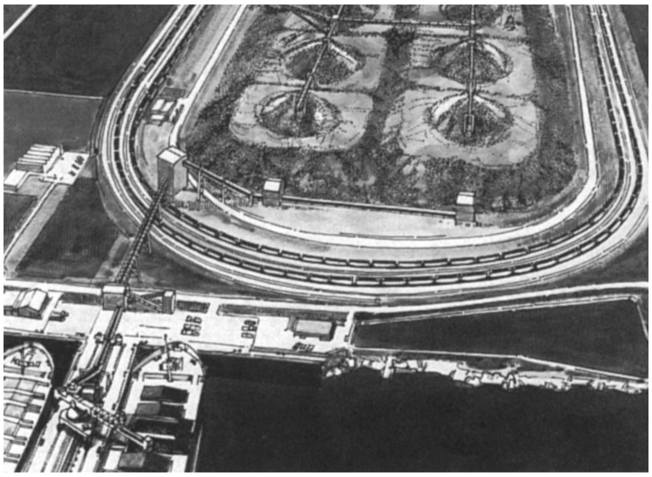


Fig. 25.2.5. Open storage piles using gravity drawn down to feed underground conveyor belts (*courtesy:* McNally-Wellman).

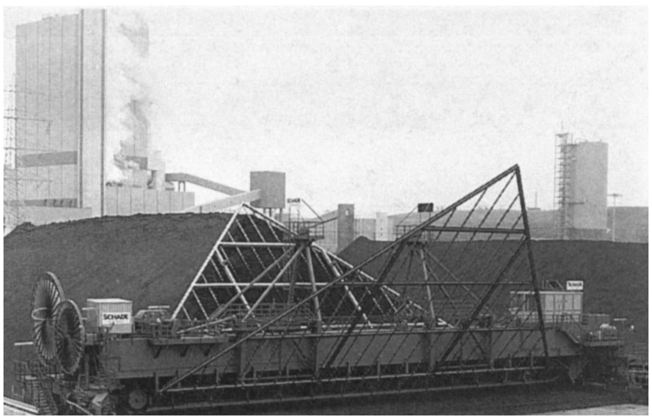


Fig. 25.2.6. Bridge scraper-reclaimer handling coal with 164-ft (50-m) span and 1125 tph (1020 t/h) capacity (*courtesy:* McNally-Wellman).

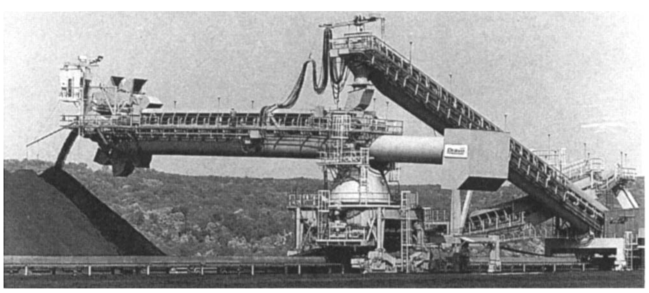


Fig. 25.2.7. Typical rail-mounted bucket wheel stacker-reclaimer used in the open storage complex (*courtesy:* McNally-Wellman).

Due to increased pressures to conform to environmental regulations, particularly dust control, the application of covered storage has seen a steady growth.

Fig. 25.2.8. Covered storage complex with rectangular covered roof at left for long-term storage. This feeds the concrete silo in center foreground. Unit train will pass under silo for flood loading (*courtesy:* PEBCO).

25.2.2.3 Blending and Mixing

In addition to the normal problems of open or closed storage, the operator of the storage yard often discovers the *necessity for blending* the incoming mineral product. This is caused by a need for greater uniformity in the product leaving the storage yard than existed in the incoming mine product. One obvious reason for this need would be variations in the chemistry or grade of mined product in various sections of the mine. Whatever the reason, this new blending function will add cost to the storage and reclaiming function. The governing requirement becomes the *permitted variation* or uniformity required in the product shipped out (Fig. 25.2.11).

There exists a certain amount of natural mixing each time the product is handled. This is accentuated by conical pile storage and a crude mixing can be accomplished by selective reclaiming with a clamshell bucket or front-end loader to blend the shipped product.

A more expensive approach is to draw varying products from different storage bins onto a common belt conveyor. As shown in Fig. 25.2.11, a crude or sophisticated proportioning method can be used depending on the need. *Bin blending to a belt* may prove adequate for reasonable capacity storage requirements. If not, a more complex and expensive *chevron-type open storage pile* is recommended (Fig. 25.2.12). This approach embodies the storage of tremendous tonnages on the ground, which may be required for other reasons. With a traveling stacker that continuously traverses the length of the tent-shaped pile, very thin layers from different mine sources are bedded on the continuously growing pile. Essential to this blending system is a *pile arrangement* which permits simultaneous bedding of one pile while a previously bedded pile is being reclaimed and shipped out.

To accomplish the above, the reclaimer is moved to the end of the rails and then onto traversing tracks set at right angles. The reclaimer is then moved to the next track where a prepared, blended pile is ready for reclaiming. A *full-height reclaimer* is required with a full height rake to insure the best blending by the traversing bucket wheel or wheels. A somewhat better product uniformity is inherent when reclaiming with a drum-type reclaimer (Fig. 25.2.13) as it automatically recovers mineral across the full width of the pile at one time. Other types of blending reclaimers include rake-type reclaimers. On large installations, there can be multiple twin-pile arrangements that afford larger capacity storage.

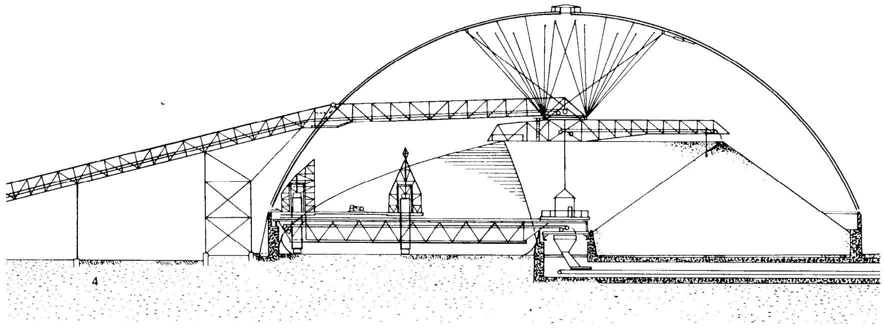


Fig. 25.2.9. Typical storage and reclaim arrangement under a hemispherical roof structure (*courtesy:* A.T. Yu).

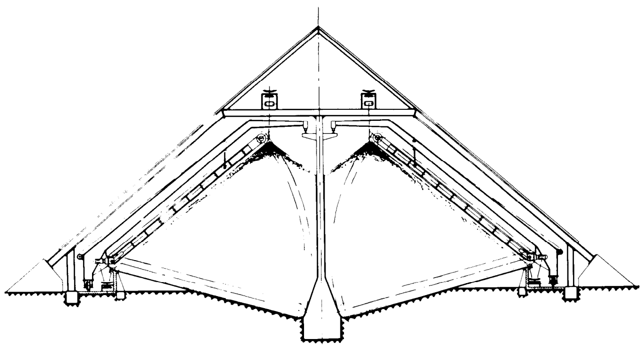


Fig. 25.2.10. Cross section of covered storage facility. Pile is prepared from twin overhead belt trippers. Twin scrapers reclaim plain or blended pile to ground belt conveyors to load out product.

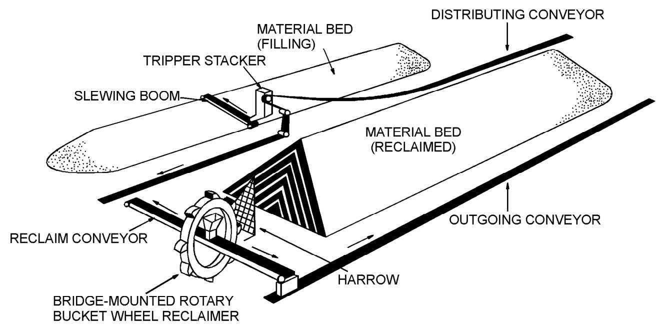


Fig. 25.2.12. Flow diagram of a typical bed-blending system using chevron-type bedding and single traversing reclaiming bucket wheel (*courtesy:* A.T. Yu).

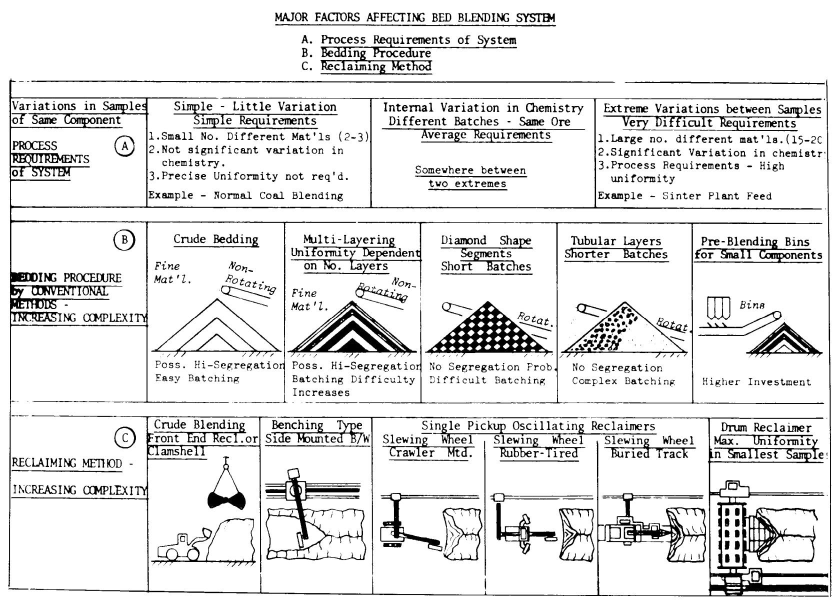


Fig. 25.2.11. Major factors affecting bed-blending systems including process requirements, bedding procedure, and reclaiming method.

25.2.3 TRANSPORTATION

For basic materials handling coverage, see Chapter 9.3 in this *Handbook.*

25.2.3.1 Belt Conveyors

Belt conveyors, because of their simplicity and universality, have quickly become indispensable in the layout of a storage facility. They permit maximizing the use of ground space and allow the designer tremendous flexibility in his flow scheme. The current parameters for belt conveyors include 20, 35, and 45° troughing idlers with the 35° replacing the older 20° as the most popular roll configuration today. Use of the former permits a gain of 28% in capacity. Commercial belting and hardware are available up to 120 in. (3.0 m) in width. An important factor in selecting minimum normal belt width is to select a figure at least three times the maximum lump size.

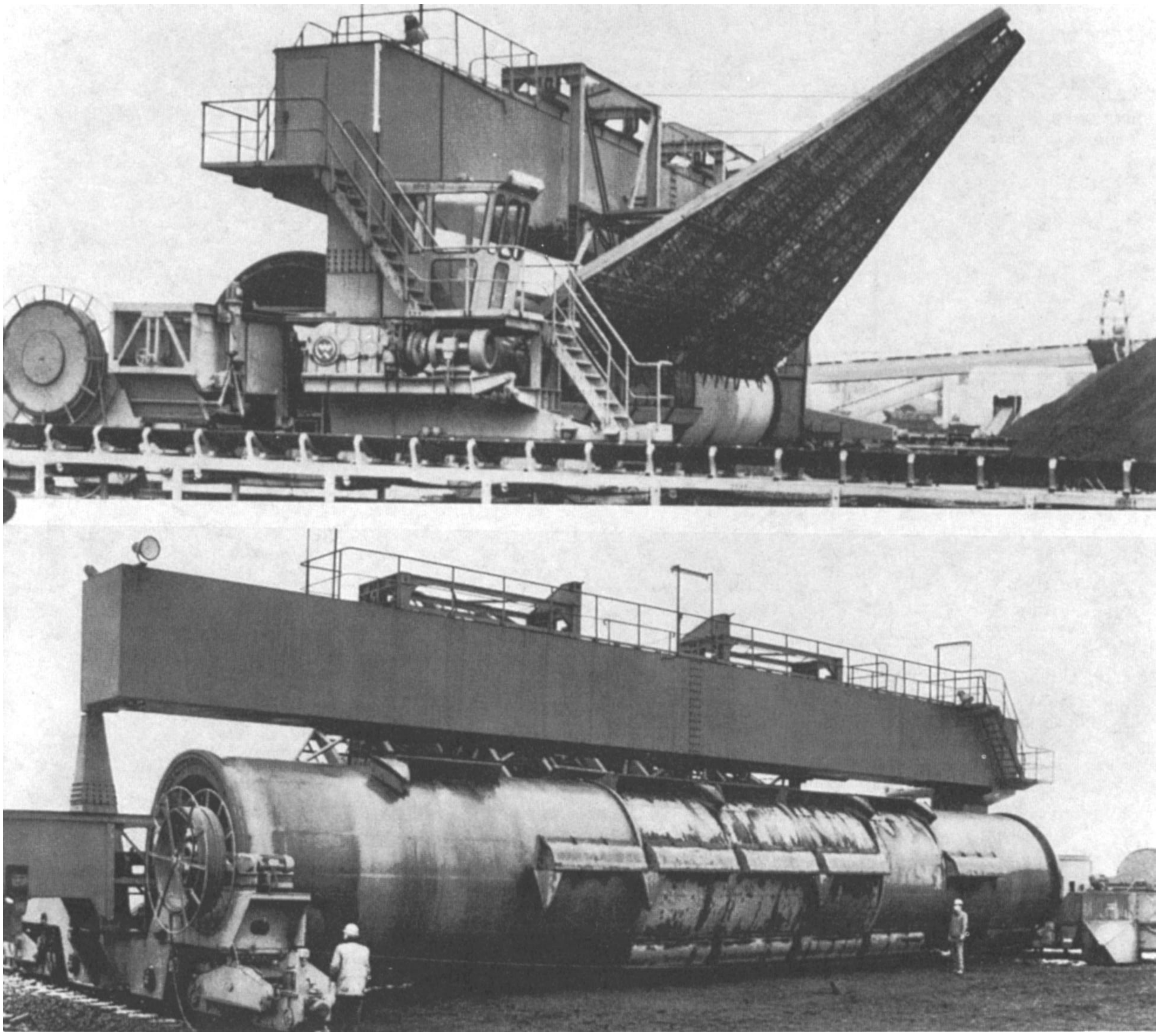


Fig. 25.2.13. Arrangement of drum-type blending reclaimer that samples full width of blended pile continuously as it advances.

For conventional belt layouts with unsophisticated loading chutes, belt speeds from 300 to 600 fpm (1.5 to 3 m/s) are workable. Belt speeds of 900 fpm (4.5 m/s) are common with properly designed loading and transfer points, especially on overland belts.

Actual *belt conveyor designs* for normal uses involve empirical calculations as prescribed in the Conveyor Equipment Manufacturers' Association (CEMA) manual. These designs readily lend themselves to computerized calculations that are quickly performed. The more recent development of using steel cable belting extends the belt conveyor's possibilities far beyond those achieved with original cord belting.

The most common conveyor operating problem is misalignment of the belt flight causing the belt to track to one side or the other. If not corrected immediately, the belt will destroy itself with edge wear against the structure.

In multi-flight layouts, particularly, *belt transfer points* are critical. If not designed correctly, the results can produce excessive belt wear, poor tracking, degradation of the mineral, and excessive dusting, among other problems. In laying out an *overland belt conveyor,* (Fig. 25.2.14) the designer should try for as few transfer points as possible to achieve the lowest capital investment. Many factors affect the layout including the general topography and location of power supply points that can affect drive locations for the various flights.

One significant option for the above is a unique design named the *cable-belt conveyor* (Fig. 25.2.15). This utilizes continuous twin wire ropes from beginning to end. This system requires only one drive station, which tensions the wire ropes rather than the belting. With the belting riding on top of the wire ropes, it serves as the product carrier only.

Another innovative approach simplifying the task of the overland conveyor designer is the use of *curved belt layout* Where a conventional overland belt layout would require various transfer points to miss existing obstacles, it is now possible to run continuous flights by laying out a curve to circumvent the obstacles. This has obvious cost advantages.

A radical departure from previous conveyor belt practice involves the *pipe belt* (Fig. 25.2.16). Here, instead of laying open in the conventional trough, the belt is rolled into a tubular pipe

Fig. 25.2.14. Ten-mile (16-km) long cable belt overland conveyor from mine to open storage. Bucket wheel stacker-reclaimer supplies barge loading river dock (*courtesy:* McNally-Wellman).

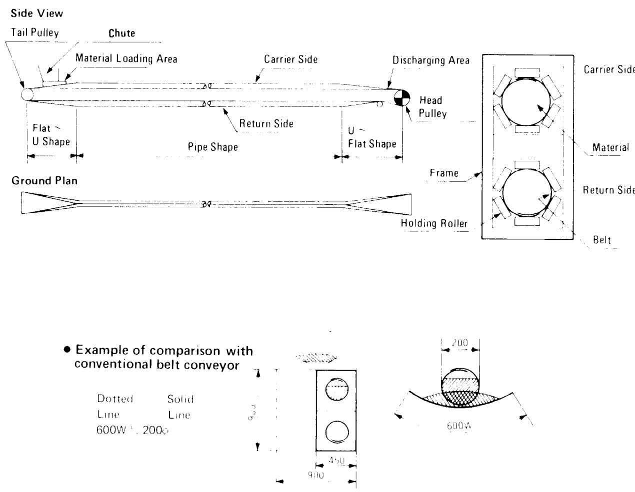


Fig. 25.2.16. Pipe conveyor configuration. Belting is restrained in tubular cross section (*courtesy:* Pipe Conveyor Co. Ltd.).

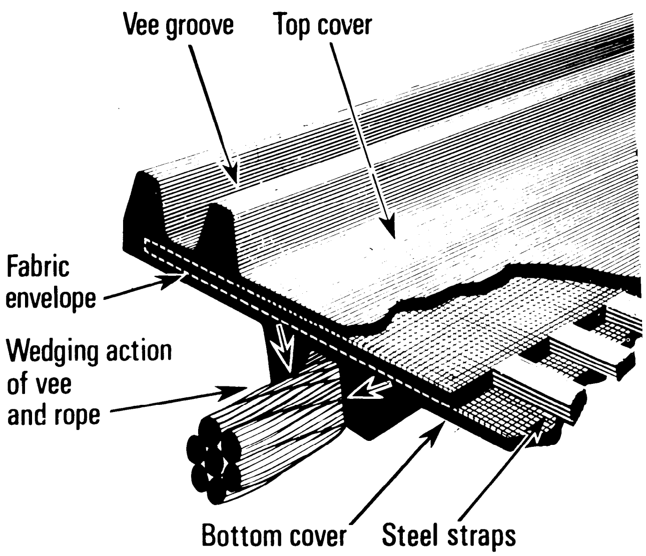


Fig. 25.2.15. Cable belt conveyor. Typical cross section.

cross section. The belt is restrained by special idlers. By acting as a pipe, spillage is minimized with external wind protection no longer needed for environmental reasons. In addition, since the mineral can no longer roll back, much steeper belt angles are a possibility with their consequent savings in required length and investment.

Steep-angle belts eliminate the basic disadvantage of the sloping belt; which requires a long and costly horizontal distance to gain elevation. One successful approach has been the *flexible side wall belt* (Fig. 25.2.17). The belt cross section utilizes a flat rubber bottom strand with corrugated rubber sides extending upwards at right angles. These enclose vertical flights that act as pockets to trap the mineral. This configuration permits loading the belt in the horizontal position where, as soon as required, by use of restraining pulleys, the belt can turn at a right angle to a vertical run. It can return to the horizontal again at the top to discharge the mineral. Minimal horizontal distance is required for this slope giving the designer almost complete flexibility in layout.

Another steep-angle approach employs a *sandwich belt* (Fig. 25.2.18) riding atop the steeply sloped and troughed carrying belt. Perfected by use on bucket wheel excavators, this arrangement permits extremely steep conveyor angles to be used with obvious advantages. Inverted troughing idlers are required above the top sandwich belt to prevent gaps between the upper and lower belts.

Shiftable ground belts can service a mobile belt discharge that is normally moved at right angles to the ground belt to follow an excavator. This arrangement permits use of ground belts fed by a bucket wheel excavator or other digging device. These must periodically advance into a new face at right angles to the belt line. To permit this belt shifting, the belt frame is loosely built and rests on ground sleds. The sleds are connected longitudinally by a rail section. When it becomes necessary to shift the ground belt, a crawler-mounted dozer travels parallel to the belt with a roller hook set over the rail section. While advancing, the belt frame snakes over in the direction of the dozer pull. This is repeated as often as necessary to obtain the required lateral shifting.

Traveling trippers are used to discharge from any desired point along a long horizontal belt. This offers great flexibility for storage layouts by making tent-shaped piles where the tripper can dump the mineral onto a traveling stacker to make a storage pile alongside.

Belt enclosures are a necessity where wind can blow fines from a belt or the mineral must be protected from the elements. Vertical side boards are often adequate for wind problems while complete half-circle covers, usually of corrugated metal, cover the entire troughed belt. Often the belt is completely enclosed in a structural steel tube with sufficient space for man access alongside. This design affords the maximum environmental protection and permits heating of the belt at reasonable expense if required.

25.2.3.2 Truck Transportation of Mineral Products

The transportation of mined products by truck within the mine and from mine to processing plant is covered elsewhere in this *Handbook.* These mine-related operations tend to utilize off-the-road-type trucks and equipment of a very specialized nature with extremely large capacities.

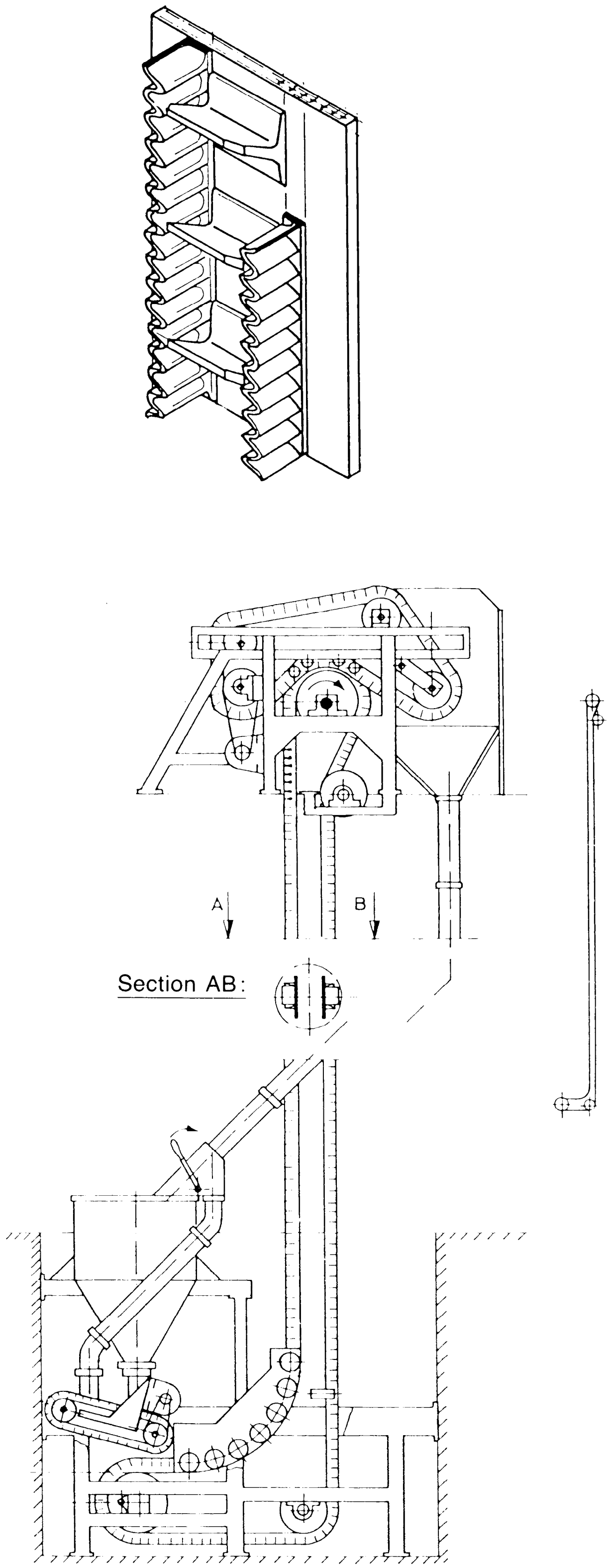


Fig. 25.2.17. Steep-angle conveyor using flexible side wall belts with accordian sides and pockets (*courtesy:* Scholtz).

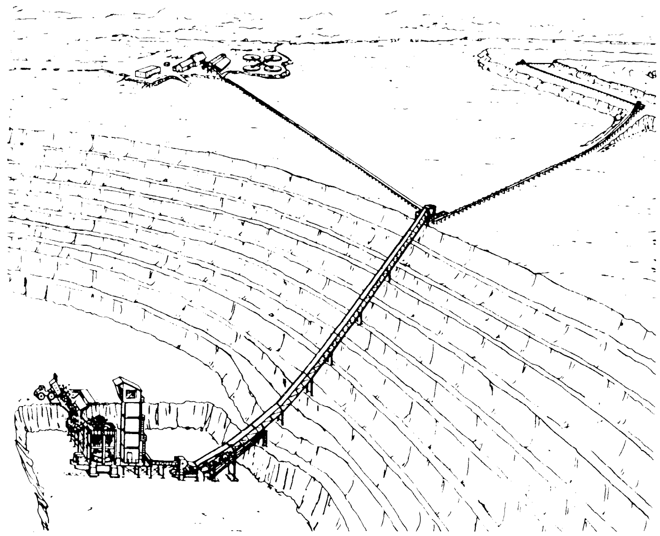


Fig. 25.2.18. Steep-angle conveyor in pit using sandwich belt retainer construction (*courtesy:* Continental Conveyor Co.).

Fig. 25.2.19. Loading minerals into the frame-type trailer is done by front-end loader. Smooth interior avoids mineral buildup (*courtesy:* Fruehauf Corp.).

Transportation of mined products to market by truck tends to be less significant on a tonnage basis than other methods of transportation described herein. Truck transportation has a high cost per ton-mile, which limits it to special situations. Bulk movement is described since most minerals will move in this fashion rather than in packaged or break-bulk shipments.

Trucks designed for the movement of minerals are divided into two general classifications. The first is an *end dump* design. Figs. 25.2.19 and 25.2.20 illustrate two configurations, namely frame-type vs. frameless construction. As always, the objective is to gain more payload. Aluminum construction is another alternative to achieve this purpose. The other classification is that of *bottom dump* design as shown in Fig. 25.2.21. This, of course, requires a special unloading facility, normally a long grated hopper leading to feeders and belt conveyors. This design offers a fast unloading cycle time.

Transporting minerals with small particle size is done economically with *closed pressure trucks* (Fig. 25.2.22). These are designed to pneumatically handle dusty, small-sized products. Thus, air pressure is used to discharge and move the mineral product for a considerable distance through hose or piping. Discharging vertically to a storage bin is the normal procedure. The truck body itself is compartmented so that the cargo can slide by gravity to the discharge ports where the airflow sends it through the discharge pipe, typically 4 in. (102 mm) in diameter.

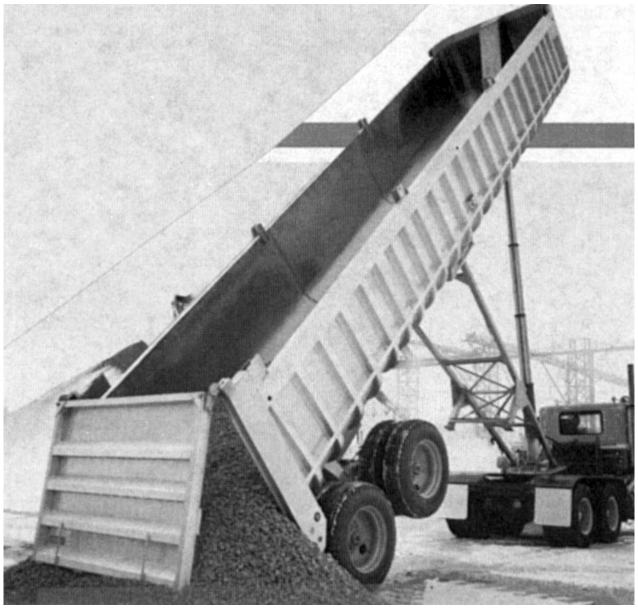


Fig. 25.2.20. Frameless trailer dumping mineral contents at desired location (*courtesy:* Fruehauf Corp.).

Fig. 25.2.21. Bottom-dump tandem trailers being loaded simultaneously by twin conveyor belts. Entire bottom is opened for discharge over receiving hopper (*courtesy:* Continental Conveyor Co.).

The advent of this type of truck has practically eliminated the use of bucket elevators to elevate fine products to bin storage.

Some containerization of sophisticated bulk minerals occurs and standardized truck containers are used.

Size and weight limits for truck transportation have steadily increased over the years. The growth of interstate highways offers faster turnaround times, and a new 13 ft 6 in. (4.43 m) vertical clearance allowance simplifies routing. One basic advantage of truck transportation is that the loading and unloading facilities are less expensive than those for competing systems. This factor could be particularly significant if the mineral deposit

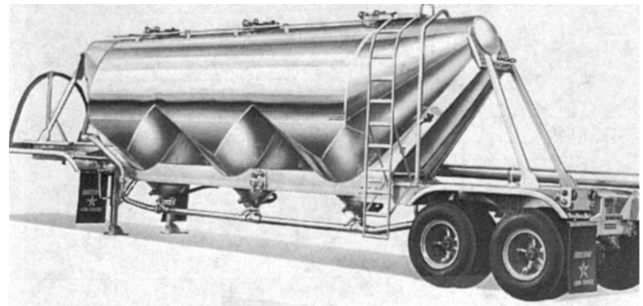


Fig. 25.2.22. Pressurized trailer with 1000-ft^3 (28-m^3) capacity allows air discharge of finely-sized mineral product via hose to overhead bins. Examples: cement, lime, etc. (*courtesy:* Fruehauf Co.).

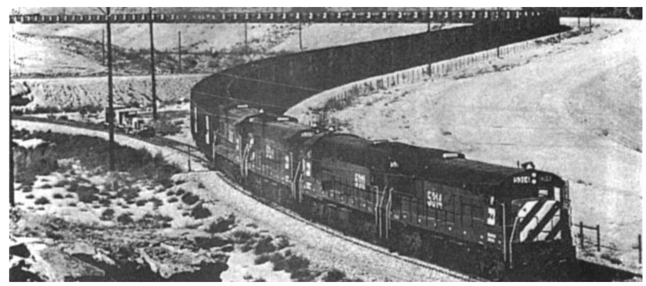


Fig. 25.2.23. Hundred-car unit train enroute from mine to final destination with four power units.

has a very short predicted life and cannot justify elaborate loading and unloading facilities.

25.2.3.3 Rail Transportation

The mining engineer normally has practically no control over the actual transportation of the mined product once the loaded rail car is turned over to a common carrier railroad. He does, however, retain control of the loading and unloading of the actual cars and can be instrumental in the selection of the type of train used. Several possible alternatives are discussed in some detail to help in the selection of the method to be used.

A partial train load will fit many situations and is normally loaded in an unsophisticated manner. For a typical user loading out small to medium tonnages of mined product per year, the normal procedure is for the common carrier to drop off the requested number of empty cars. Since allowed loading time is not critical, the plant operator moves the empties under the loading conveyor or storage bin with a car puller or shifting locomotive. Movements are matched to the loading tonnage rate required. If accurate car weights are required, weigh belt conveyors or track scales can be used.

The most significant breakthrough in keeping the railroads competitive was the introduction of the *unit train* concept in the early 1960s (Fig. 25.2.23). For the first time, a complete trainload of up to 100 cars was handled as a unit in all phases of the operation: loading, transport, and unloading. This mass movement of mined product gave rise to a new set of operating rules whereby the loading and unloading times must be rigidly controlled, typically four hours each. These concepts required new systems and equipment to accomplish the necessary handling rates. Assuming the unit train consisted of 100 cars at 100 tons

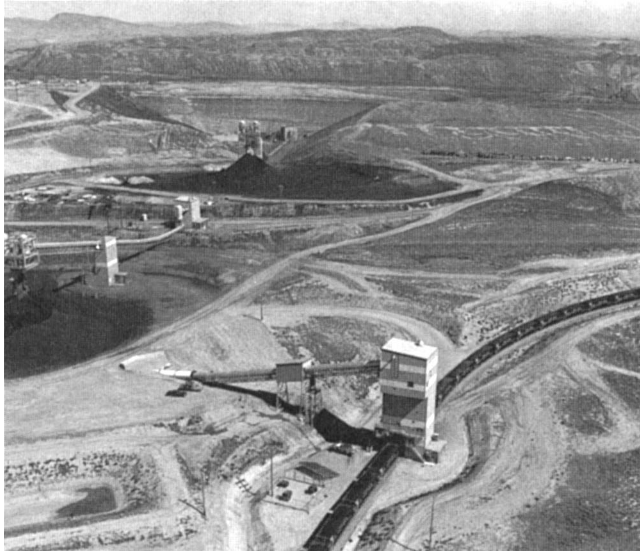


Fig. 25.2.24. Unit train loading facility. Storage bin fed by sloping conveyor stores sufficient coal for continuous loading of 100-car unit train with 100 tons (91 t) per car. (*courtesy:* A.T. Yu).

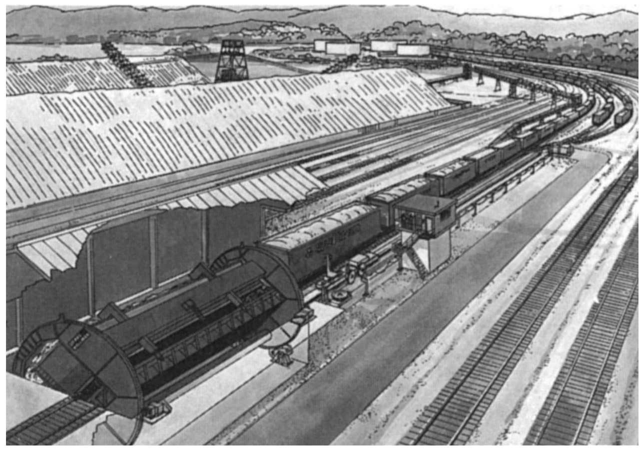


Fig. 25.2.25. Unit train car dumper facility. Each car is dumped after being advanced one car length by train positioner. Adjacent open storage piles reclaimed by rake reclaimer (*courtesy:* McNally-Wellman).

(91 t) each, it became necessary to store at least 10,000 tons (9070 t) of mined product in a state immediately accessible to the train loading points. The most popular method that evolved is the use of a 12,000-net-ton (10,885-t) capacity concrete silo that is filled continuously from the mine while awaiting the unit train. When the train arrives, the mined product is loaded into the cars in a four-hr period. One popular method of accomplishing this is by running the train directly beneath the silo bottom and "flood loading" each car as it passes beneath the loading chute (Fig. 25.2.24). Thus an average loading rate of about 2500 tons (2268 t)/hr is achieved. To achieve maximum efficiency with the unit train concept, it is essential that the cars be loaded to their weight limit as accurately as practical. This may require a secondary topping off operation. If there is significant loss from the top of the open cars in transit, crusting agents can be applied to the mineral surface to cut down on windage loss in transit.

When a high-tonnage unloading rate is not required and capital expenditure is to be minimized, the use of *bottom-dump hopper cars* is a logical choice. Each car can be shifted over a hopper beneath the tracks and the bottom gates manually opened to discharge the contents to the hopper, feeder, and conveyor belts that move the mineral products to their destination. Car movers are normally adequate for this service.

In the case of the unit train, a *rotary car dumper* is generally used where the quick discharge of a loaded railroad car (hopper type) is required. With this device, each car is turned upside down and the mineral contents discharged to a large hopper that feeds belt conveyors leading to the ultimate destination (Fig. 25.2.25). Normally, it is necessary to uncouple each car so that it enters the dumper unattached (Fig. 25.2.26). The only exception to this is when a unit train is designed with rotary couplers that permit dumping without breaking the train. With stricter environmental regulations (25.2.4), dust control at a car dumper is becoming mandatory and requires a complete enclosure and an exhaust system feeding a bag-house filter.

To achieve a high unloading rate, it becomes essential to move the cars in and out of the dumper quickly. One popular method employs a shifter locomotive, either manually or remotely controlled. These power units can handle a group of cars or an entire unit train. An alternate choice is a mechanical, remotely controlled train positioner that can handle a full unit trainload of cars. This device drops a power arm over the coupler and advances one car length before disengaging to return to the next car. Rotary couplers are required to avoid uncoupling cars.

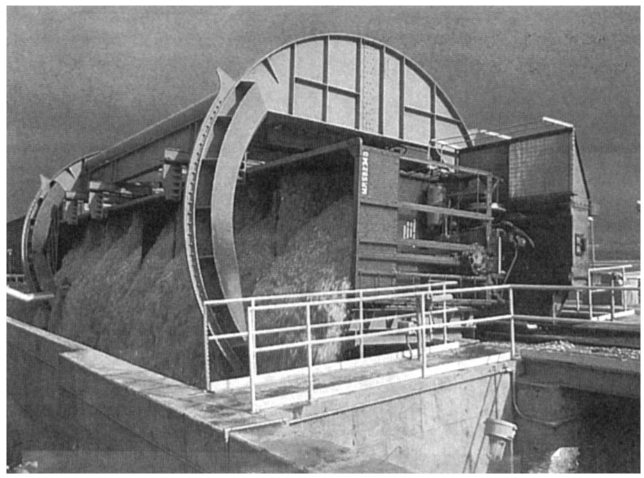


Fig. 25.2.26. Detailed view of mechanism in car dumper showing vertical clamps that hold car against bottom platen. A complete enclosure would be needed to control dusting (*courtesy:* McNally-Wellman).

The highest unloading rate possible can be achieved by the use of special *high trestle side-discharge cars* (Fig. 25.2.27). As the train approaches the trestle over the designated stock pile discharge point, each loaded car is tripped open by a device alongside the track and empties its complete contents where desired in a matter of seconds.

There are two significant factors to be evaluated in considering this trestle unloading system. The first is the additional significant investment for the cars with special quick-opening doors. The second factor is the selection of a location where

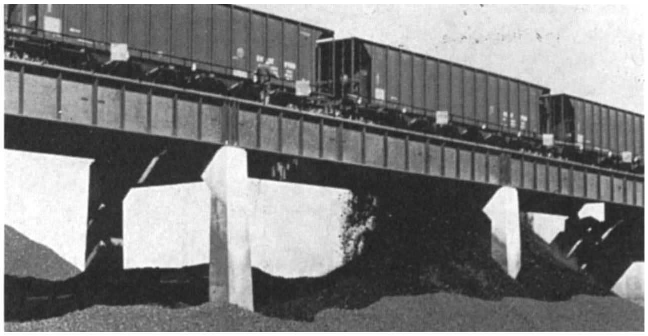


Fig. 25.2.27. Rapid discharge rail car facility. Cars are remotely opened while atop the trestle. Discharge is complete within seconds. Doors are automatically closed (*courtesy:* Ortner Car Co.).

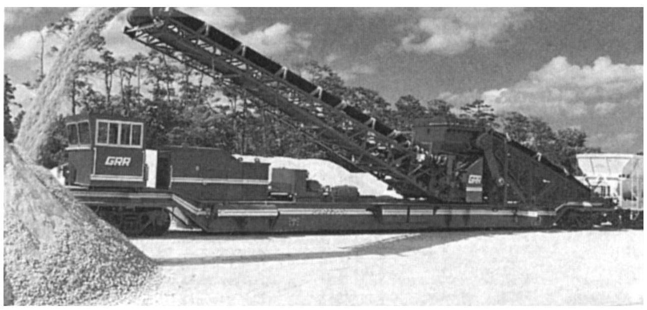


Fig. 25.2.28. "Dump-Train" arrangement features continuous conveyor belt under cars for full length of train. Discharge boom can be swiveled to desired position.

dropping the mineral product from trestle height to the ground will not create an environmental dust problem. If these two problems are addressed successfully, extremely high rates of discharge are possible.

One unique train unloading approach is that of the *self-unloading train* (Fig. 25.2.28) recently placed in operation. This innovative design consists of a train of about 30 cars permanently connected and containing a conveyor belt running under the full length of the train. The belt restrains the loaded mineral product at the bottom of each car hopper. Held loosely while traveling, the full-length belt is tightened at the straight-line unloading destination. Similar to a self-unloading ship, the train-long belt draws product from all of the cars and discharges the entire trainload at one end via a stacking conveyor that feeds the storage pile.

In extremely cold weather, if the loaded cars are permitted to sit idle or the trip is time consuming, the mineral product may become frozen in the cars. This situation can present serious operating problems in attempting to unload the cars. After thawing the car, large frozen lumps must be broken up on the car dumper grating in order to pass through to the feeders and belts. If freezing in the cars is the normal situation, then car thawing facilities become a necessity. The latest system utilizes microwave heating.

25.2.3.4 Marine Transportation

In planning for the transportation of mineral products to market, the use of marine transportation should be considered.

Fig. 25.2.29. Traveling clamshell grab bucket used to unload deep water vessels (*courtesy:* McNally-Wellman).

It becomes a logical choice for many reasons. If speed of delivery is not essential, high annual tonnages justify the investment in specialized equipment, and overseas destinations are often involved. Even if the marine transportation segment must be coupled with overland movement, it is normally possible to justify marine loading and unloading equipment due to the savings inherent in low-cost water movement. Recent years have seen low-value mineral products move competitively for unprecedented distances by water. Some examples would be the movement of coal from West Virginia to Florida by water, barite shipment from China to ports on the Gulf of Mexico, cement clinker movement across the oceans, and shipment of low-cost aggregates from Europe to the USA. Obviously, low-cost backhauls can be a governing factor.

For *ocean transport of minerals,* the primary governing factor which determines vessel size and configuration is whether the ship must pass through the Panama Canal. Vessels which can do so are grouped loosely as *"Panamax"* types, approximately 40,000-deadweight-ton (39,339-t) capacity, with 42-ft. (13.8-m) draft and less than 110-ft. (36-m) beam. Coincidentally, many US ports have the same draft limitation. When it is decided to ignore the Canal limitations, the maximum ship size becomes an economic one. There are now many ships of 100,000 to 300,000-deadweight-ton (98,340- to 295,044-t) capacity, with even larger sizes available for special situations. Planned backhauls of non-mineral cargoes can affect the basic design of these bulk mineral carriers. The selection becomes involved with the dredged depths of desired ports of call. Even this normal port depth limitation can be circumvented by "offloading" in deep water outside the port.

The *unloading of ocean vessels* becomes a significant cost-per-ton factor. Following the historical days when bulk carriers were unloaded by hand, the standard method of unloading minerals has been with the conventional *clamshell grab bucket unloader,* either whirler type or straight-line configuration (Fig. 25.2.29). To increase hourly capacity, the size of the unloading bucket has grown steadily. However, buckets beyond present capacity seem uneconomical at this time due to cycle time disadvantages relative to the other unloading systems described below.

While the cleanup of the ship's hold was originally done by main bucket alone, practically all unloaders now lower earth-

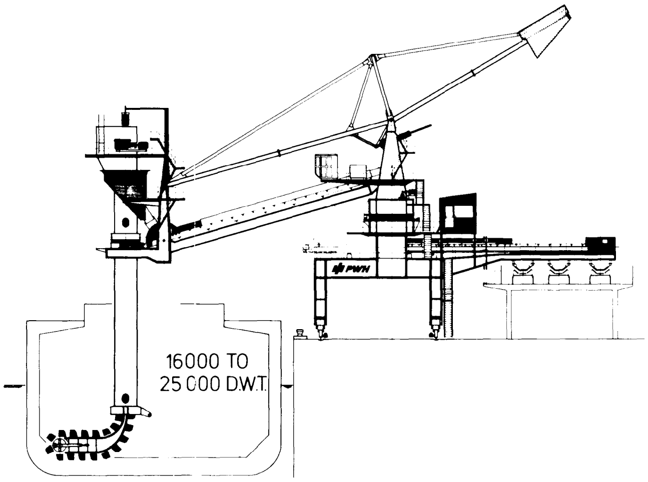


Fig. 25.2.30. Continuous ship unloader using bucket chain system to unload ocean-going ship (*courtesy:* P.W.H.).

moving-type equipment into the hold to speed up the cleanup and turnaround time for the ship. The "efficiency index," or the actual unloading rate achieved from start to finish divided by the free digging rate is normally much lower than one (usually 0.5, or less).

The desire to improve this index number and minimize elapsed unloading time prompted the search for improved equipment and led to development of *continuous ship unloaders.* One type is the dock-supported *bucket chain continuous ship unloader* (Fig. 25.2.30). Where the operator is faced with large, open hatch ships and sufficient annual tonnage, the higher cost of this unloader can be justified when considering the demurrage costs of the ship while in port.

The actual configuration of the vertical bucket arrangement varies as does the method of loading the buckets. Some variations include steel buckets on steel chains, steel buckets on rubber belting, and rubber pocket belts loaded by digging wheels. After being elevated to a sufficient height above the ship's deck, the mineral product is transferred to a horizontal belt conveyor headed towards shore and then to the dock belt conveyor and storage pile.

As contrasted to the bucket chain approach, the *vertical-screw continuous ship unloader* employs an encased vertical screw to dig the mineral in the ship's hold, then elevate it to the horizontal transfer belt. This design tends to have a lighter machinery weight allowing lighter dock construction. *Other continuous unloading methods* include vertical twin "sandwich belts" that trap the mineral between belts and elevate it out of the ship's hold. "Marine legs" trap the mineral between a steel chain flight and steel casing to allow vertical movement. *Pneumatic unloaders* are used on powdery mineral products.

Lake, coastal, and gulf transport is a significant factor in the economics of US mineral transportation. The basic difference between this type vessel and ocean ships is one of size and draft. Although there are some *self-unloading ships* used in ocean bulk trades, the preponderance of this type of equipment is found on the Great Lakes and Gulf of Mexico (Fig. 25.2.31). The relatively short turnaround times for mineral cargoes can justify the extra ship cost and the lost payload caused by carrying this extra machinery. The normal configuration consists of one or two conveyor belts extending for the length of the ship beneath the hoppers in the ship's hold. These discharge to the belt by gravity.

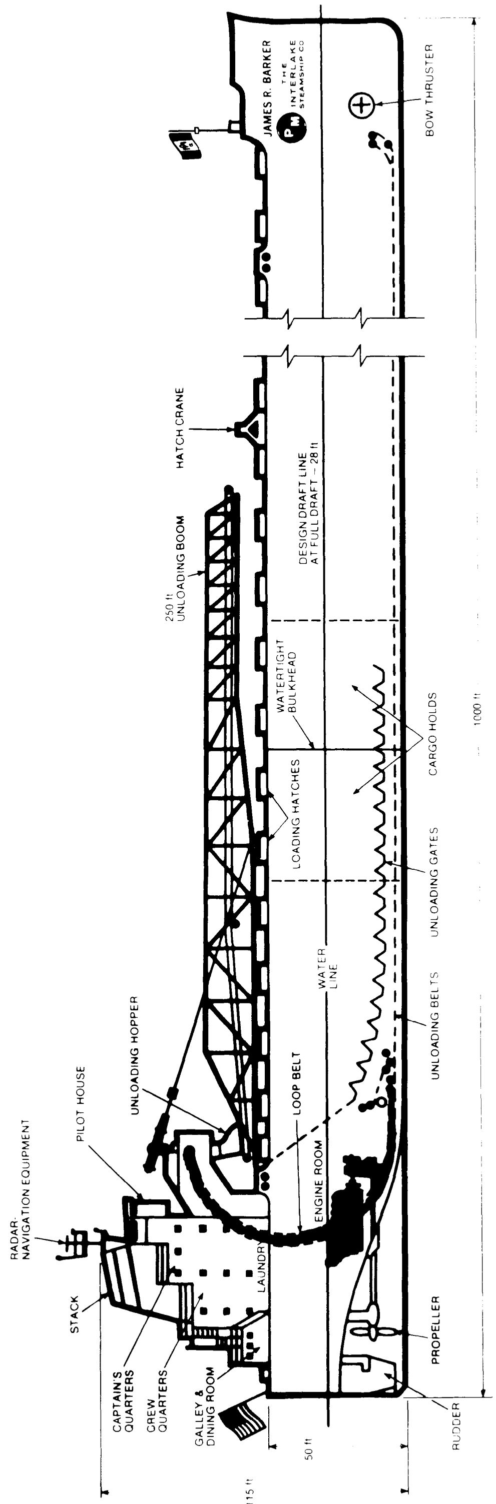


Fig. 25.2.31. Cross section of loop belt self-unloading ship. Deck boom conveyor can swing to either side (*courtesy:* Stephens Adamson). Conversion factor: 1 ft = 0.3048 m.

Once the belt conveyor reaches the end of the ship, there are various methods of elevating the mineral product. The height above deck must be sufficient to reach the normally pivoting discharge belt which unloads to a hopper on the dock. Some of these elevating methods include: "sandwich belt" arrangements, circular wheels to elevate, and rubber "pocket belts." Extremely high unloading rates, up to 20,000 tons (18,142 t)/hr, are possible. The user on shore has practically no investment on the dock except for the take-away belt to land storage.

Inland marine transportation permits tremendous tonnages of mineral products to be moved to destination via the Mississippi river system (Fig. 25.2.32). Together with all of its tributaries, this covers the basic heartland of the United States from mine to ultimate user and at an extremely low cost per ton-mile. For example, over 100 million tons (90 Mt) of coal/yr are moved from mine to power plants and other users by this means of transportation. Since most power plants require cooling water and the inland rivers provide the cheapest transportation possibility, a river location is very advantageous for a coal-burning power plant.

Since the inland river system is based on slack-water navigation with locks and dams, it is the lock sizes that determine the *limitations to equipment sizes* (Fig. 25.2.33). Early lock sizes of 56 by 360 ft (18.3 by 118 m) set a barge size of 26 by 175 ft (8.5 by 57.4 m) that would carry some 900 tons (817 t) in a 9-ft (2.74-m) draft channel. A group of six of these barges with a towboat became a standard tow. Beginning in 1929, lock sizes of 110 by 600 ft (33.5 by 183 m) on the Ohio River set a barge size of 35 by 195 ft (10.7 by 59.4 m) carrying some 1500 tons (1362 t) in a 9-ft (2.74-m) deep channel. The larger locks increased the standard size tow to 15 barges with towboat, moving over 22,000 tons (19,956 t) in one tow (Fig. 25.2.34). In the 1970s, modernization of lock sizes to 1200 by 110 ft (366 by 33.5 m) allowed the standard tow to pass through in a single lockage.

Economic towboat sizes range from 2000 to 6000 hp (1492 to 4476 kW) with some being powered with up to 10,000 hp (7460 kW) when moving a maximum of 30 barges down the open river section of the Mississippi River to the Gulf Coast.

Unloading of river barges began with the familiar clamshell bucket, and in the 1960s the *continuous barge unloader* made its appearance (Fig. 25.2.35). Since the standard open river barge offers access to all its mineral cargo, the entire contents can be unloaded in two or three longitudinal passes dependent on the bucket ladder configuration. Free digging rates of 5000 tons (4535 t)/hr are common using the 1500-ton (1361-t) capacity barge. These high-speed units have become the standard method of unloading at most power plants.

25.2.3.5 Overland Slurry Pipeline Transportation*

Unlike all other forms of transportation addressed in this chapter, *slurry pipelining* utilizes a stationary system with the mineral in motion. In truck, rail, and marine systems, the units are in motion, with the mineral payload being motionless relative to the transporting vehicle.

An overland slurry transportation facility is a system of pipelines, pump stations, and associated process facilities working in conjunction to move a water-crushed mineral mixture, (Stripling and Holter, 1981). At the head of the system is the slurry preparation facility. Grinding and slurrification equipment produce a controlled solid-water mixture for injection into the pipeline system. The pipeline is sized and designed by considering the hydraulic characteristics of the solid-liquid two-phase flow. Pump stations are positioned at appropriate locations along the pipeline to input energy. Several types of pumps may be utilized, but again consideration must be made for the two-phase nature of the flow. The terminal point of a slurry system will have facilities and equipment for storing, thickening, and dewatering the slurry. With growing environmental concerns, the water from the liquid-solid separation step may require clarification or other treatment prior to discharge. Throughout the transportation facility, instrumentation and controls will be used to measure variables (flow, pressure, density, etc.) and implement an operational philosophy.

The purpose of this section is to describe the components of a slurry transportation system and to show the special design considerations required for each component. Criteria are presented for the overall system design considering the interrelationships of the various components.

ECONOMIC FEASIBILITY. Slurry transport has proven to be technically and economically viable for coal, iron ore concentrates, sulfide concentrates, limestone (for Portland cement manufacture), and phosphate minerals. Economically feasible slurry pipeline projects tend to have large volume point-to-point transportation requirements and/or special conditions such as very rugged, inhospitable terrain between the transportation points. Slurry pipelines have significant initial capital costs but low operating costs. Slurry pipelines, therefore, have relatively low exposure to cost escalation. Finally, each project is a special situation and the transportation economics should be developed for the alternatives (rail, truck, marine, and slurry pipeline) and then compared.

PIPELINE HYDRAULIC CONSIDERATIONS. The design of a slurry pipeline is determined largely by hydraulic considerations. The solid particle size distribution selected for the slurry will have a major impact on the whole transportation system (including pumps, slurry preparation, and utilization facilities). The size consist that is finally selected should minimize cost of preparation, pipeline pumping, and dewatering while maintaining favorable hydraulic characteristics within the pipeline.

Once the solid particle size distribution has been defined, the slurry composition is established as to percentage of solids and liquid. Depending on the particular mineral, solids concentrations may typically vary from 40 to 65% by weight.

Hydraulic design involves consideration of the critical velocity or lowest permissible mean flow velocity, which insures deposit-free transport of the solid phase. If the average flow velocity is below critical velocity, the solid particles will begin to settle out and deposit on the bottom of the pipe. The migration of these particles in the lower pipe cross section can lead to excessive pipeline wear and restriction or blockage of the pipe cross section where pipeline slopes change from negative to positive. Frictional pressure losses also increase when the particles travel along the bottom pipe wall. Because of these adverse effects, it is advisable to operate slurry pipelines at flows above critical velocity.

Pipe friction pressure loss, particle attrition, pipeline plugging, transient flow, slack line flow and cavitation are other hydraulic characteristics that must be considered when designing a slurry pipeline. These and other aspects are discussed in more detail by Zandi (1971) and Stripling and Holter (1981).

PUMP STATION DESIGN. Slurry pipelines will have one or a number of pump stations at appropriate locations. The size and exact placement of the stations will be determined by a steady-state hydraulic analysis. Both centrifugal and positive displacement pumps can be used to pass slurries. The choice is dictated by the required discharge pressure. Frazier (1971), Willis and Truscott (1978), Sellgren (1979), and Childs (1979) further discuss the specifics of slurry pumps.

Centrifugal slurry pumps are low-head, high-volume devices and are often used for in-plant pumping plus providing suction to mainline pumps. A number of parallel, positive displacement

* This section contributed by Travis E. Stripling, Pipeline Engineering Dept., Brown & Root U.S.A., Inc., Houston, TX.

Fig. 25.2.32. Map of US inland rivers system showing coal-burning power plants receiving coal shipments by water.

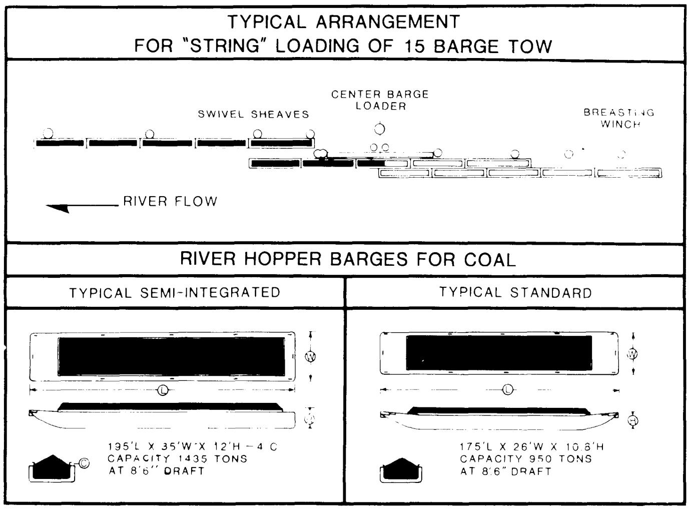


Fig. 25.2.33. Inland river barges—dimensions and loading arrangement at dock.

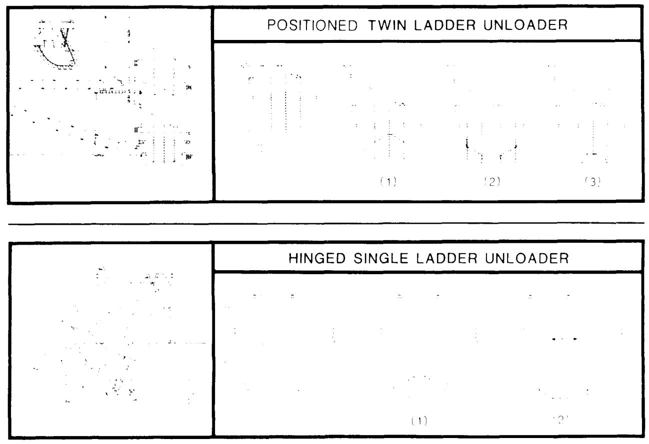


Fig. 25.2.35. Twin ladder and single ladder arrangements for continuous barge unloader.

Fig. 25.2.34. Typical inland river 15-barge tow of 1500-ton (1360-t) capacity.

machines are appropriate for mainline pumping. Flexibility can be provided by using variable speed drives on some mainline pumps. Pulsation dampeners, accumulators and/or signal biasing circuits should be used to reduce flow-induced pressure pulsations. Standpipes can be a very effective method of reducing pulsations on the suction side of positive displacement pumps.

Slurry pipeline systems need not operate as "tight-line" systems. Each pump station may well have an agitated storage tank on its suction side.

SLURRY PREPARATION. The solid portion of the slurry must be crushed to the specified size distribution before entering the system. Crushing and/or grinding may be necessary. Bernstrom (1977) has discussed the preparation process for a coal slurry. Wasp, Kenny, and Gandhi (1977) describe several slurrification techniques using devices such as gravity feed chutes and hopper-shaped sumps.

SLURRY UTILIZATION. In order to build flexibility into a slurry transport system, storage facilities will be required at the terminal locations. This storage capacity "uncouples" the pipeline facility from the ultimate activities at the terminal. In this manner, the pipeline can be operated in a steady-state condition, and fluctuations in volume can be made up with stored product.

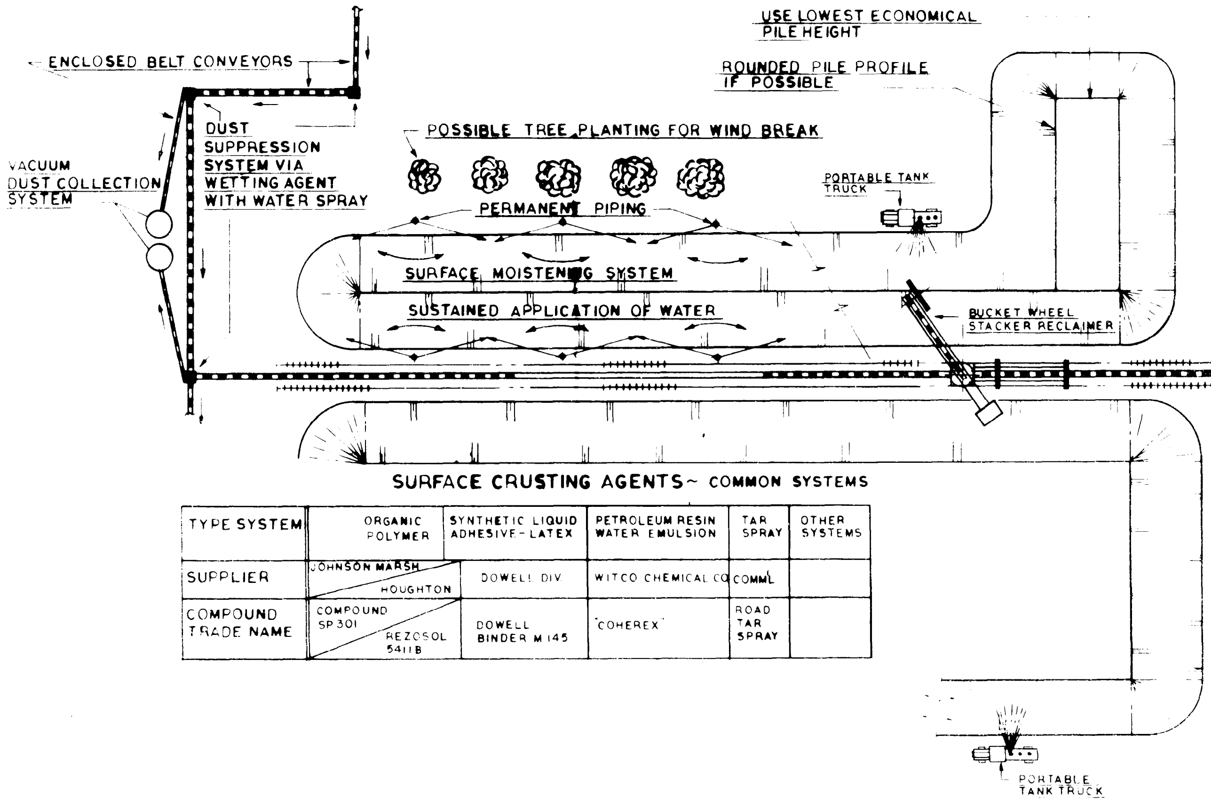
Various techniques are available for dewatering a mineral slurry. Vacuum and pressure filters, centrifuges, hydrocyclones, and thermal drying are examples which may be used alone or in combination.

INSTRUMENTATION AND CONTROLS. The control of a slurry pipeline system is similar to that for regular crude oil pipelines in several respects. A supervisory control and data acquisition (SCADA) system is used to monitor and control process variables at various locations from a central control point. Some pump stations may be fully or partially automated and require few if any operating personnel. The initial or primary pump station should use flow as the principal control parameter with process override systems providing for flow, station suction, and discharge pressure control. Booster or secondary pump stations should be controlled on suction pressure with a discharge pressure override system. Control valves should be used sparingly and limited to startup, shutdown, and bypass-type situations.

The initial pump station area is a critical location for process monitoring and control. The slurry must be inspected to assure the solid-liquid mixture has the required properties before it is passed into the pipeline. Injection of an off-specification slurry into the hydraulic system could cause significant problems. Density of the slurry must be monitored, since improper values of density can lead to formation of slurry plugs or create undesirable friction pressure drops. Indeed, a section of pipe before the first mainline pump station should be selected to determine the actual friction pressure drop to be expected in the pipeline system. Slurry particle size distribution should be checked periodically. Finally, slurry acidity or alkalinity should be maintained at a level that will reduce erosion/corrosion of the pipeline.

Liptak (1967) and Andrew (1974) present a general discussion of slurry instrumentation and controls. Buckwalter (1976) and Zahid (1977) discuss particular aspects associated with positive displacement pumping.

OVERALL SYSTEM OPTIMIZATION. The construction of a slurry transportation system can be an expensive undertaking. Thus, for a particular project, there is a considerable economic



TYPE SYSTEM	ORGANIC POLYMER	SYNTHETIC LIQUID ADHESIVE-LATEX	PETROLEUM RESIN WATER EMULSION	TAR SPRAY	OTHER SYSTEMS
SUPPLIER	JOHNSON MARSH / HOUGHTON	DOWELL DIV	WITCO CHEMICAL CO	COMM'L	
COMPOUND TRADE NAME	COMPOUND SP 301 / REZOSOL 5411 B	DOWELL BINDER M 145	"COHEREX"	ROAD TAR SPRAY	

Fig. 25.2.36. Generalized approach for dust control system for open storage piles including pile surface moistening, surface crusting agent, transfer point dust suppression, and vacuum dust collection system.

incentive to optimize the total system. This optimization can be achieved by selecting the slurry preparation, pipeline-pumping, slurry utilization system with minimum cost. However, this selection process is no small task. A pilot plant study may be required for the various elements of the system to determine a cost equation that is a function of several variables. Crushing-grinding-preparation costs must be determined for slurries of different particle size distribution, concentration, and quality. Test loop studies are necessary to estimate the pressure loss in pipelines of various diameters flowing with slurries at different velocities, concentrations, and particle size distributions.

25.2.4 ENVIRONMENTAL CONSIDERATIONS

Environmental regulations with regards to storage and transportation are in a state of flux at the present time since governmental regulations are not firmly resolved and depend somewhat on local interpretation. When any new storage project is under consideration, it is essential that a detailed environmental study be made and that an environmental impact statement be prepared, if required by the permitting process.

The two most significant environmental effects possible are in the areas of (1) surface water drainage from open storage piles and (2) dust control. The latter includes both wind-borne dust from pile surfaces and dust from transfer points as the mineral is handled. A generalized approach to dust control is shown in Fig. 25.2.36. The particle-size ranges for various types of dust are given in Fig. 25.2.37.

Surface drainage predictions require topographic layouts with stream locations pinpointed. Soil porosity is critical and

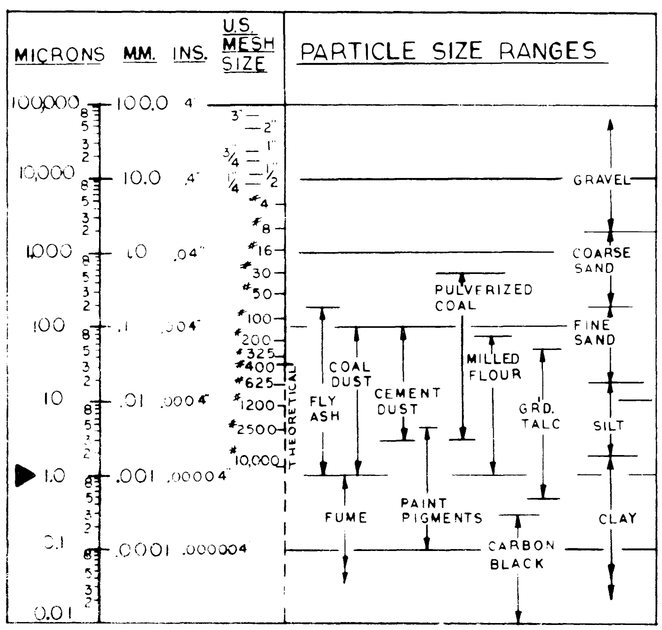


Fig. 25.2.37. Dust particle size range with comparisons for various commodities.

core sampling can help predict the path of contaminated surface drainage. With topographical data and proposed pile layouts, collection ponds can be laid out to intercept surface drainage. Here water treatment can be accomplished so that the runoff from the settling pond will be acceptable as it enters the local streams. Control of acidity through pH readings and the addition of alkaline reagents such as lime is a very common corrective measure.

Dust control at open storage piles requires a study of the prevailing wind situation and the location of populated areas near the site. One potential trouble spot occurs when the fine mineral is dropped from a significant height to the top of the open storage pile. This problem is best controlled by an adjustable height stacking conveyor belt to limit free fall. Dust generated from the surface of open piles can be controlled somewhat by the addition of a water spray system. When the mineral fines are subject to long-term storage, a surface crusting agent can help alleviate loss of wind blown fines from the pile surface.

Transfer point dust is normally controlled by water spray headers set up in the belt conveyor transfer housing or by dry air evacuation systems. To make the water more effective in wetting down the dust, a surfactant is metered into the water supply to the spray headers. By reducing surface tension, the water can more effectively penetrate the dust particles for dust control.

Conveyor belt covers aid significantly in controlling the action of wind as it passes over the moving belt. The ultimate in such dust control occurs when the entire belt conveyor is enclosed in a fabricated steel tube.

Dust collection, while an expensive process, is often mandated by local conditions. At the significant dust generation points, including transfer points, the area is hooded and the offending dust is vacuumed off and piped to a filter location, usually a bag filter setup where the dust is removed from the air.

With an ever-heightened public awareness of environmental pollution, the mineral storage operator can anticipate increased attention to dust problems. This will necessitate additional expenditures and require more vigilant daily operating controls. In many locations, this pollution problem has led to covered storage replacing open storage.

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Chapter 25.3
MINERAL PROCESSING

W.J. SCHLITT, M.I. CALLOW, V.P. KENYEN, AND R.S. PIZARRO

25.3.0 INTRODUCTION

W.J. SCHLITT

Mineral processing is, in general, the most important downstream operation for mining engineers. This is because most mined material—from hard-rock ores to coal to industrial minerals—is subjected to some type of comminution or beneficiation operation. In most cases, several unit operations are performed sequentially in order to produce a marketable product or feed to an extraction operation such as leaching or smelting.

A tremendous number of unit operations are included under the broad heading of mineral processing. The breadth reflects the need to beneficiate everything from dense, competent hard-rock ores to solid-fuel minerals like coal or oil shale to a wide range of industrial minerals that include hard, dense materials; soft, sticky clays; and light, fluffy minerals like vermiculite.

Comminution or size reduction is typically the first unit operation to which mined material is subjected. *Crushing* describes the breakage at coarse sizes, while *grinding* covers size reduction for fine particles. The run-of-mine ore is typically subjected to primary crushing in jaw or gyratory crushers, then secondary/tertiary crushing using cone crushers, roll crushers, hammer mills, or impactors. Crushing is generally followed by grinding. In this operation, material is reduced from about 12 to 25 mm (0.5 to 1.0 in.) down to very fine particle sizes. The grinding may employ rod mills, ball mills, autogenous or semiautogenous mills, pebble mills, roller mills, fluid energy mills, or other types of grinding mills.

Concurrent with crushing and grinding are the *classification* operations that sort and separate the crushed or ground particles on the basis of size and/or weight. The classification may be done by *screening* using stationary or vibrating screens or with sieve-bend screens. *Hydrocyclones* or mechanical classifiers (spiral, rake, or drag units) may also be used.

Comminution and classification are often followed by *concentration.* This is commonly done by *flotation,* a method that collects values, or less frequently, rejects gangue constituents in a stable froth. This involves use of aeration and/or agitation in conjunction with reagents that affect the surface chemistry of the fine particles. Another approach to concentration involves *gravity systems* (heavy media separation, centrifugal specific gravity separation, jigging, tabling, and flowing film concentration). Gravity operations are less common than flotation, but are still widely used in coal preparation, beach sand processing, and placer operations. *Magnetic* and *electrostatic methods of concentration* and separation also find some application in mineral processing, for example, magnetite concentration.

Other operations of importance in mineral processing include mixing, thickening, clarifying, filtering, and drying. A wide range of mechanical equipment is used in these applications. Important ancillary operations include materials handling and feeding, particle agglomeration (pelletizing, briquetting, or sintering), and such activities as dust control, water treatment, and tailings disposal.

The following segments are not intended to be comprehensive—there are the *SME Mineral Processing Handbook* and many other authoritative publications for this purpose. However, the most universal operations are covered in order to give the miner some perspective on what happens to his ore once it leaves the mine. These key unit operations include crushing, grinding, screening, hydrocycloning, flotation, gravity concentration, and magnetic separation. Each of these topics is covered in sufficient depth to give the reader some feel for the principles and practices involved.

25.3.1 CRUSHING

M.I. CALLOW AND V.P. KENYEN

Crushing is the first stage in the comminution unit operation. Two very different crushing methods are used depending on the type of rock to be crushed. As a broad principle, hard abrasive rocks are crushed by compression between wear-resistant surfaces, and softer less-abrasive materials are crushed by impact, shear, or also by compressive mechanisms.

Hard-rock, run-of-mine material has to be reduced to the desired product size in progressive stages of compressive crushing because the reduction ratio available in each stage is limited to about 6:1. Hard-rock crushers—typically, jaw, gyratory, and cone crushers—crush by nipping the rock between stationary and moving wear-resistant surfaces. The reduction ratio in each stage is controlled by the angle between the two surfaces that is required to hold the rock as the surfaces move together in the crushing motion. If the angle is too large, the rock will not be gripped, and if too small, the available reduction ratio will not be exploited. The rock is subjected to multiple impacts as it falls by gravity through the crusher.

In processes that subsequently grind the ore to finer sizes in rod mills and/or ball mills, the crushing plant product is typically in the size range of –38 mm (–1-1/2 in.) to as fine as –9.5 mm (–3/8 in.). This is normally accomplished in three or four stages designated as the primary, secondary, tertiary, and quaternary crushing stages. An example of the performance of each stage is as follows:

Crushing Stage	Typical Crushers	Feed Size Range	Product Size Range
Primary	Jaw or Gyratory	Up to 1524 mm (60 in.)	152 to 305 mm (6 to 12 in.)
Secondary	Cone	152 to 457 mm (6 to 18 in.)	37 to 127 mm (1¼ to 5 in.)
Tertiary	Cone	19 to 152 mm (3/4 to 6 in.)	5 to 25 mm (3/16 to 1 in.)

In processes that subsequently grind the ore in autogenous or semi-autogenous mills, only primary crushing is required.

The main factors that control the selection of hard-rock crushing equipment and the design of a crushing circuit are

1. Tonnage rate.
2. Top size and size distribution of feed.
3. Desired product size.
4. Method of feeding.
5. Ore characteristics.
   a. Work index.

b. Bulk density.
c. Abrasivity.
d. Compressive strength.
e. Clay content.

A large selection of crushing methods is available for crushing softer, less abrasive ores. In addition to the compressive crushing described above, crushing by impactors and hammermills provides a far greater reduction ratio since the technique is not restrained by the angle of nip. Traditionally, impact crushers have been used on ores containing less than 5% silica. Phosphates, limestone, barite, and coals are examples. Impact crushers have also been used on higher-silica, sticky ores, such as Wyoming uranium sandstones with high clay content, but maintenance was extremely high.

The purpose of this segment is to describe the general application of crushing techniques. Departures from normal practice, of which there are many, can be examined and explored in more detail in other publications.

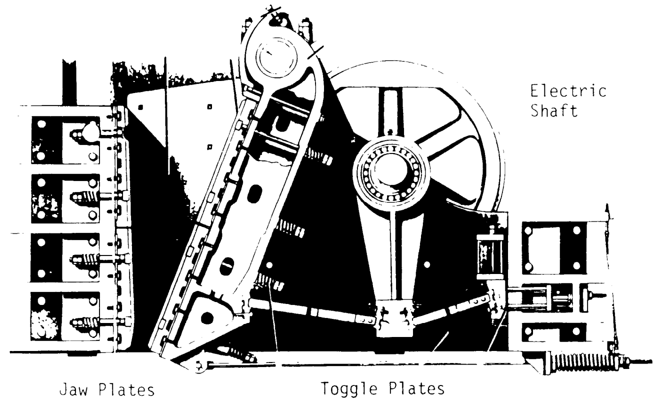


Fig. 25.3.1.1. Double-toggle jaw crusher (*courtesy:* Allis Chalmers).

25.3.1.1 Definition of Terms

Terms commonly used in defining some properties of the feed material, crusher sizing, and operating criteria are as follows.

FEED CHARACTERISTICS.

Bond Work Index—This index is most commonly used to calculate the power (measured in kilowatts or horsepower) required to crush a rock from a given feed size to a given product size. It is determined by a twin-pendulum impact crusher test on –76 mm + 51 mm (–3 in. + 2 in.) pieces of rock, square in cross section. Crusher power is calculated from the standard Bond equation (Eqs. 25.3.1.1 and 25.3.1.2) as illustrated later in this segment.

Compressive Strength—The compressive strength of a rock is determined by crushing cylindrical samples 51 mm (2 in.) in diameter by 51 mm (2 in.) in length in a universal tester. This provides a comparative measure of the resistance of a rock to compressive crushing.

Abrasion Index—This index is a measure of the abrasivity of the rock during crushing. A number of test methods are available, such as the Los Angeles Abrasion Test and the Deral Abrasion Test, but the most commonly used method is Bond's method in which a 76- by 25- by 6-mm (3- by 1- by 1/4-in.) piece of 500 Brinell SAE 4325 chrome-nickel-molybdenum steel rotates in a falling stream of ore under standard conditions and the weight loss is measured. The abrasion index A_i is then used to empirically predict wear rates of crusher liners.

CRUSHER SETTINGS. Crushers are adjusted to produce a selected product size. There are three main parameters which are interrelated.

Closed-side Setting (CSS)—This is the minimum distance between the surfaces at the closed position.

Open-side Setting (OSS)—This is the maximum distance between the surfaces at the open position.

Throw—This is the distance in the direction of compression that the moving wear surface travels between the open-side setting and the closed-side setting.

Both the open-side and closed-side settings are used in tabulations of crusher performance, and care should be taken to check the type of setting used in each case.

25.3.1.2 Primary Crushers

Gyratory crushers are used as the primary crusher for high-tonnage open pit operations with throughputs above 450 t/h (500 tph) because jaw crushers do not have enough capacity. Below 450 t/h (500 tph), jaw crushers generally are used because of their lower cost. Jaw crushers are also selected for underground crushing plants if the shaft diameter limits the size of equipment that can be handled. For surface installations with over 450 t/h (500 tph), both crushers should be considered. Some of the differences between jaw and gyratory crushers are as follows:

1. The capacity of a gyratory crusher is much greater than that of a jaw crusher handling the same size of feed.
2. Gyratory crushers can be choke fed by direct dumping from trucks. Jaw crushers cannot be choke fed; they require a scalping grizzly and a feeder.
3. Jaw crushers typically require less maintenance. Since the liners are reversible, the scrap metal loss is usually low.
4. The ratio of full load to idling power is about 2.2 for jaw crushers and 3.3 for gyratory crushers. Therefore, the gyratory that idles more also saves more while idling. In terms of reduction tons, the gyratory does double the work for the same expenditure of energy.
5. A gyratory primary crushing installation will normally cost more than a jaw crusher installation. There are, however, a number of trade-offs, and an engineering estimate is often required.

25.3.1.3 Jaw Crushers

There have been many types of *jaw crushers* designed and built since the first US patent was issued in 1830. They fall into three groups: Blake type (double toggle), Dodge type, and single toggle (overhead eccentric). The Blake-type double-toggle crusher (Fig. 25.3.1.1) and the single-toggle crusher (Fig. 25.3.1.2), however, dominate the market.

The double-toggle crusher is preferred for heavy-duty crushing on hard material. It is heavier and more expensive than a single-toggle crusher for a given duty. Justification for selecting a double-toggle crusher will therefore be needed for each application. Some of the factors in this analysis are

1. Compressive strength of the rock.
2. Work index.
3. Feed size and reduction ratio.
4. Abrasion index.
5. Life of the mine.
6. Capital cost.

The size of a jaw crusher is measured by its feed opening (gape) and length. For example, 610 by 914 mm (24 by 36 in.), or 2436, indicates a distance of 610 mm (24 in.) from fixed jaw

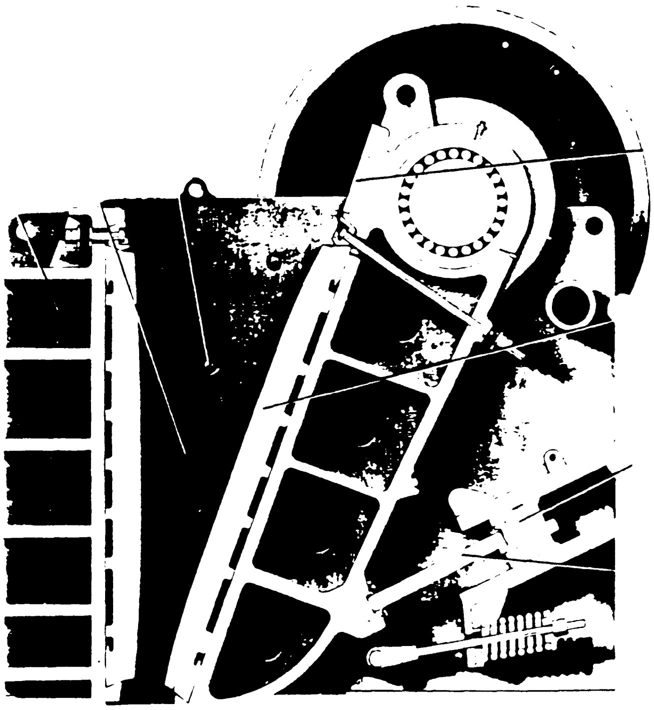


Fig. 25.3.1.2. Single-toggle jaw crusher (*courtesy:* Allis Chalmers).

plate to moving jaw plate where the feed enters, and 914 mm (36 in.) is the dimension across the jaw plates.

DESCRIPTION.

Double-toggle Jaw Crusher—The double-toggle crusher drives the moving jaw through an eccentric shaft, which moves the front and rear toggles up and down at each revolution of the shaft, closing the jaws on the downstroke and opening them on the upstroke. The toggles are held in the toggle seats by the spring-loaded tension rods. Minor setting changes to the crusher are made by changing shims between the main crusher frame and the rear toggle block. Major adjustments are made by changing the length of the front toggle.

Crusher Sizing—Jaw crushers are sized on the basis of the maximum lump size to be crushed and/or the tonnage rate to be crushed. Maximum lump size should not exceed 80% of the gape. For example, a 610- by 914-mm (24- by 36-in.), or 2436 crusher, will accept a maximum lump size of 610 by 0.8 = 488 mm (19 in.). In actual operations, the crusher will occasionally accept lumps up to the gape size, as long as bridging does not occur.

In open pit operations, the maximum lump size may be controlled by the size of the shovel dipper. Under these circumstances, a guide to crusher size can be obtained from Table 25.3.1.1.

Approximate capacities of double-toggle jaw crushers are available from crusher manufacturers and are provided in Table 25.3.1.2.

Single-toggle Crusher—This type of crusher is also called an overhead eccentric crusher. Reference to Fig. 25.3.1.2 will illustrate the reason for this name. The overhead eccentric is an integral part of the moving plate, which moves up and down with the rotation of the eccentric. The single toggle is pivoted at the base of the moving plate and causes the jaw to move in an arc. This results in a pinching action between the moving and stationary plates. The toggle is held in the toggle seats by the compression spring. Major adjustments to the crusher setting

Table 25.3.1.1. Smallest Jaw Crusher for Various Shovel Dippers

Capacity of Dipper		Jaw Crusher Size Recommended	
yd^3	(m^3)	in.	(mm)
$\frac{3}{4}$	(0.57)	30 × 36	(762 × 914)
1	(0.76)	30 × 36	(762 × 914)
$1\frac{1}{2}$	(1.15)	36 × 42	(914 × 1067)
$1\frac{3}{4}$	(1.34)	42 × 48	(1067 × 1219)
2	(1.53)	42 × 48	(1067 × 1219)
$2\frac{1}{2}$	(1.91)	48 × 60	(1219 × 1524)
3	(2.29)	48 × 60	(1219 × 1524)
$3\frac{1}{2}$	(2.68)	48 × 60	(1219 × 1524)
4	(3.06)	56 × 72	(1422 × 1829)
5 to 10	(3.8 to 7.6)	66 × 86	(1676 × 2184)

Source: Fuller Co.

are made by changing the toggle length. Minor adjustments are made by changing shims between the toggle block and the mainframe. The toggle is designed to shear or break in the event tramp material enters the crusher.

The single-toggle crusher is of much lighter construction than the double-toggle Blake-type crushers. For example, a 610 by 914-mm (24 by 36-in.) single-toggle crusher weighs about 14,512 kg (32,000 lb), whereas a double-toggle crusher of the same size weighs 25,678 kg (65,000 lb).

CRUSHER SIZING. Similar to the double-toggle crusher, the single-toggle crushers are selected either on the basis of maximum lump size to be crushed, or the tonnage rate to be crushed, or both.

2.5.3.1.4 Gyratory Crushers

A *gyratory crusher* is a gravity-type machine. Materials flow from top to bottom. It receives a large, coarse feed, usually run-of-mine, and its product usually requires additional crushing. Gyratory crushers can be used for primary or secondary crushing. This segment describes gyratory crushers for primary crushing. Secondary gyratory crushers are described under secondary crushing.

DESCRIPTION. A primary gyratory crusher is illustrated in Fig. 25.3.1.3. The crushing action is caused by the closing of the gap between the mantle line (movable) mounted on the central vertical spindle and the concave liners (fixed) mounted on the main frame of the crusher. The gap is opened and closed by an eccentric on the bottom of the spindle that causes the central vertical spindle to gyrate. The vertical spindle is free to rotate around its own axis. The crusher illustrated is a short-shaft suspended spindle type, meaning that the main shaft is suspended at the top and that the eccentric is mounted above the gear. The short-shaft design has superceded the long-shaft design in which the eccentric is mounted below the gear.

Gyratory crushers have been built in three types: suspended spindle, supported spindle, and fixed spindle. The supported-spindle and the suspended-spindle types are most common.

There are two suspended-spindle gyratory crusher designs: the short shaft and the long shaft. The short-shaft design is the modern design in which the eccentric is located above the level gear. The main frame has two sections; the upper shell supports the concaves while the lower shell contains the eccentric and the pinion shaft. The bevel gear and pinion provide rotation of the eccentric. The crusher is either belt-driven or has direct drive.

Table 25.3.1.2. Data on Jaw Crushers–Blake (Double Toggle)–Selected from Manufacturers' Catalogs

Make	Size feed opening, in.*	Approx. shipping weight in lb	Maximum rpm of flywheel	Motor hp	Approximate capacity per hour when discharge opening is set to the sizes shown below—in tons of 2,000 lb—materials weighing 100 lb/ft³ when crushed. Discharge opening in in. = Closed-side setting (CSS): 1	1¼	1½	2	2½	3	4	5	6	7	8	9	10
Traylor (Fuller)	10 × 20	11,500	300	20	14	17	20	25	34								
Kennedy Van Saun	10 × 24	14,000	275	15	16		22	25	33								
Traylor (Fuller)	15 × 24	23,000	275	30		23	27	34	42	50							
Kennedy Van Saun	14 × 24	20,000	275	25			26	33	41	50							
Traylor (Fuller)	24 × 36	64,000	250	75				77	95	114	150						
Kennedy Van Saun	24 × 36	71,500	200	75					95	113	150						
Traylor (Fuller)	30 × 42	98,000	200	100					125	150	200	250	300				
Allis-Chalmers**	32 × 42	108,400	200	100							250	290	330	360	400		
Traylor (Fuller)	36 × 48	135,000	180	125						208	270	330	390	450			
Kennedy Van Saun	36 × 48	163,500	200	150							270	330	389	445			
Traylor (Fuller)	42 × 48	155,000	170	150							290	350	410	475	535		
Allis-Chalmers**	42 × 48	159,000	180	125–150								380	420	470	510	540	580
Kennedy Van Saun	48 × 60	310,000	175	200							350	425	500	615	685		
Traylor (Fuller)	48 × 60	245,000	140	200							352	428	500	618	686		
Allis Chalmers**	48 × 60	271,000	170	150–200									480	530	570	610	660
Traylor (Fuller)	56 × 72	440,000	120	250								490	575	680	820		

** Open-side Setting OSS, Allis Chalmers 60 × 84,420,000, Allis Chalmers 66 × 84,420,000
Conversion factors: 1 in. = 25.4 mm, 1 lb = 0.4536 kg, 1 hp = 0.7457 kW, 1 lb/ft³ = 16.02 kg/m³.

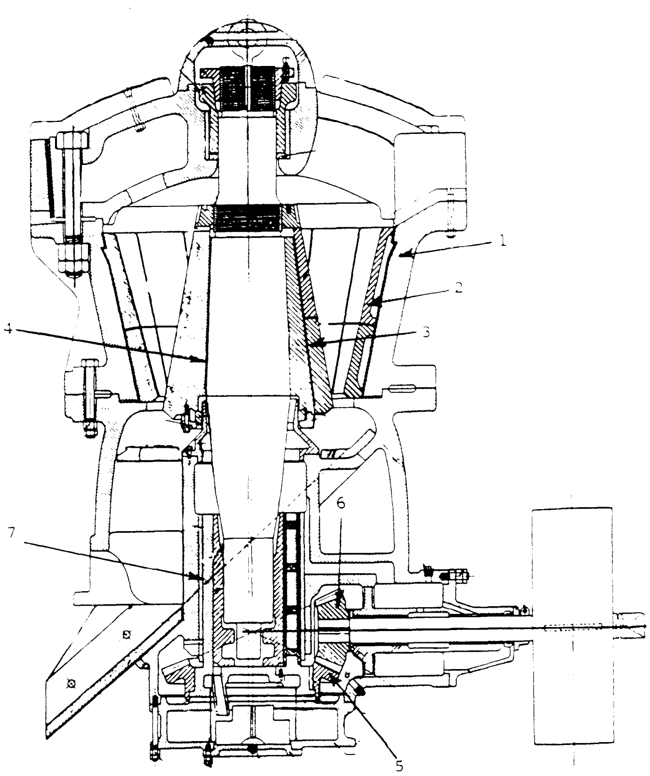


Fig. 25.3.1.3. Sectional view of short-shaft suspended gyratory crusher (*courtesy:* Fuller Co.).

DEFINITION OF GYRATORY CRUSHER TERMS.

Crusher Size—Gyratory crushers are designated either by the gape and mantle diameter or by the size of the receiving opening.

Table 25.3.1.3. Smallest Gyratory Crusher for Various Shovel Dippers

Capacity of Dipper yd³	(m³)	Gyratory Crusher Size Recommended in.	(mm)
¾	(0.57)	16 or 20	(406 or 508)
1	(0.76)	16 or 20	(406 or 508)
1½	(1.15)	20 or 26	(508 or 660)
1¾	(1.34)	26 or 30	(660 or 762)
2	(1.53)	26 or 30	(660 or 762)
2½	(1.91)	36 or 42	(914 or 1067)
3	(2.29)	42 or 48	(1067 or 1219)
3½	(2.68)	42 or 48	(1067 or 1219)
4	(3.06)	48 or 60	(1219 or 1524)
5 to 10	(3.8 to 7.6)	60 or 72	(1524 or 1829)

Source: Fuller Co.

Crusher Speed—The speeds listed in the tables are pinion shaft rpms.

Crusher Settings—Both closed-side settings (CSS) and open-side settings (OSS) are used for gyratory crushers. In contrast to the jaw crusher setting where closed-side settings are used, open-side settings are used in sizing gyratory crushers.

GYRATORY CRUSHER SIZING. The crusher is selected on the basis of feed size or tonnage rate or both. A rule of thumb is that 80% of the feed to the crusher should have a maximum one-way dimension of less than two-thirds of the nominal feed opening. This has limited value since in most cases, the average size distribution of the feed to the crusher is, at best, an educated guess. A more useful approach is to relate crusher size to the size of the shovel dipper in the pit. This is shown in Table 25.3.1.3.

Capacities of short-shaft, primary gyratory crushers, as provided in manufacturers' catalogs, are shown in Table 25.3.1.4, and the percentage of product passing a square opening is shown in Table 25.3.1.5.

Table 25.3.1.4. Data on Short-shaft Gyratory Crushers Selected from Manufacturers' Catalogs

Make	Size	Approx. shipping weight in lb.	Pinion rpm	Motor hp	Approximate capacity tons (2000 lb) per hr at discharge openings shown for material weighing 100 lb/ft³ — Open side setting of discharge opening in in.																				
					2	2½	3	3½	4	4½	5	5½	6	6½	7	7½	8	8½	9	9½	10	10½	11	11½	12
Traylor (Fuller)	30–60	200,000	425	200	345	420	495	560	625		765		910												
Allis-Chalmers	30–55	148,000	600	300		420	510	570	650	730	810	900													
Traylor (Fuller)	36–60	280,000	375	250		505	595	665	750		915		1070		1220										
Allis-Chalmers	36–55	163,000	600	300					600	680	760	850													
Traylor (Fuller)	42–66	315,000	375	250			675	770	860		1045		1220		1395		1550								
Allis-Chalmers	42–65	266,500	514	400						880	1000	1120	1250	1450	1650										
Nordberg (Rexnord)	42–70	251,800	380	300							780	870	950	1040	1120	1200									
Traylor (Fuller)	48–70	500,000	340	350				945	1050		1380		1500		1710		1910		2100						
Allis-Chalmers	48–74	504,000	514	500							1700	1850	2000	2150	2300	2500	2700								
Nordberg (Rexnord)	48–80	489,000	330	500									1515	1610	1720	1830	1940	2060							
Traylor (Fuller)	54–74	600,000	340	400					1250		1520		1790		2015		2250		2500		2700				
Allis-Chalmers	54–74	534,000	514	500								1800	1950	2100	2250	2400	2550								
Nordberg (Rexnord)	54–80	525,000	330	500									1440	1535	1640	1740	1850	1950							
Traylor (Fuller)	60–89	800,000	300	500							2150		2500		2925		3200		3450		3800		4100		
Allis-Chalmers	60–89	881,000	514	600									2500	2670	2840	3100	3260	3430	3600						
Traylor (Fuller)	72–90	980,000	300	700									2600		3050		3350		3600		4000		4300		
Nordberg (Rexnord)	60–102	1,200,000	300	800													2800	3040	3275		3740		4215		4685
Allis-Chalmers	60–109	1,277,000	450	1000														4300	4620	4940	5260	5580	5900	6250	6600
Traylor (Fuller)	84–110	1,300,000	275	1000											3500		3875		4250		4625		5000		

Conversion factors: 1 in. = 25.4 mm, 1 lb = 0.4536 kg, 1 hp = 0.7457 kW, 1 lb/ft³ = 16.02 kg/m³.

Table 25.3.1.5. Percentage of Product Passing a Square Opening Equal to Open-side Setting of Gyratory Crusher

Feed material	Run of mine	Scalped	Scalped and recombined with fines
Limestone	90	85	88
Granite	82	75	80
Trap rock	75	70	75
Ores	90	85	85

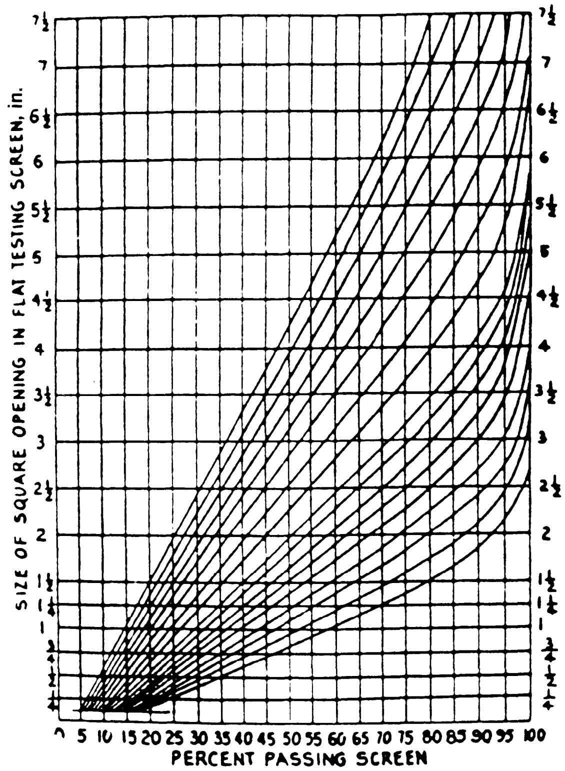


Fig. 25.3.1.4. Typical crusher size-distribution for under 7½ in. (190 mm) opening. Conversion factor: 1 in. = 25.4 mm.

Discharge Product Size. Figs. 25.3.1.4 and 25.3.1.5 illustrate typical discharge product size distributions for gyratory crushers. These size distributions should only be considered as good approximations based on actual screen analyses. In practice, the distribution will vary depending on many factors, one of which is the rock type. To determine the expected product from a crusher set at a predetermined discharge opening, obtain from Table 25.3.1.2 the approximate percentage of feed that will pass an opening equivalent to the crusher setting. The intersection of two axes will provide the product curve for the specific application.

25.3.1.5 Power Calculation

The crusher motor size is determined using Bond's crusher work index. The relation used for calculating the approximate unit power consumption is

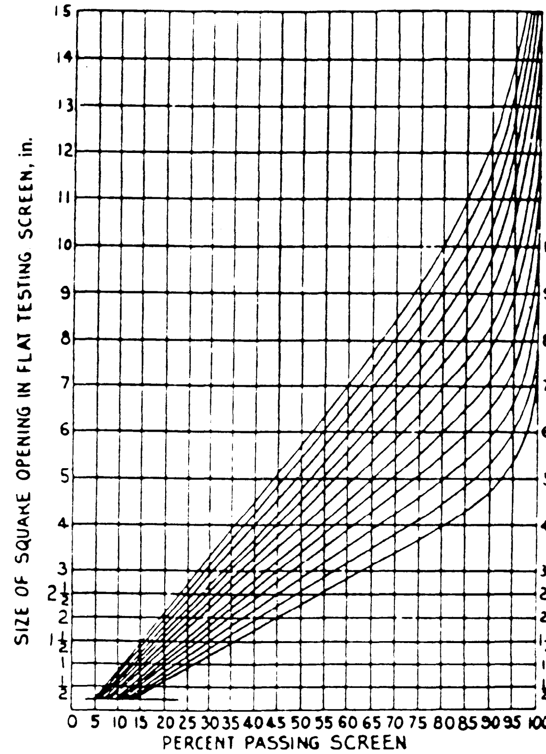


Fig. 25.3.1.5. Typical crusher-product size-distribution curves for open-side discharge over 7½ in. (190 mm). Conversion factor: 1 in. = 25.4 mm.

$$W = W_i \times 11.0 \times (\sqrt{F_{80}} - \sqrt{P_{80}}) / (\sqrt{F_{80}} \times \sqrt{P_{80}}) \qquad (25.3.1.1)$$

where W is specific power required in kWh/t, P_{80} is 80% passing size of the product in μm, and F_{80} is 80% passing size of the feed in μm or 2/3 of the crusher feed opening.

To find total power in kW, the calculated specific power is multiplied by a factor of 0.75 for primary crushing and 1.00 for secondary crushing:

$$\text{Total kW} = (\text{crusher capacity}) \times (W) \times (\text{factor}) \qquad (25.3.1.2)$$

This method of power calculation can be used for calculating the approximate power of primary jaw and gyratory crushers, secondary gyratory crushers, and cone crushers. For empirical values of work index for different materials, see Table 25.3.2.4.

An example of the use of the formula follows.

Example 25.3.1.1. Assume a copper ore with Bond crusher work index of 14.2. The design feed rate is 1800 t/h (2000 tph). An Allis Chalmers 5474 crusher is selected with an open-side setting of 165 mm (6-1/2 in.).

Solution. Feed size F_{80} = 914,400 (36 in.) (66% of crusher feed openings).

$$\text{Product Size } P80 = 139{,}700 \text{ mm (5.5 in.)}$$

$$W = 14.2 \times 11.0 \times (\sqrt{914{,}400} - \sqrt{139{,}700}) / (\sqrt{914{,}400} \times \sqrt{139{,}700})$$

$$W = 0.255 \text{ kWh/t (0.310 hp/ton)}$$

$$\text{Total power} = 0.255 \times 1800 \times 0.75 = 345 \text{ kW (465 hp)}$$

25.3.1.6 Cone Crushers

Cone crushers are exclusively used for secondary and tertiary crushing. The cone crusher operates in much the same way as the gyratory crusher except it operates at a higher speed. It has a flatter crushing chamber designed to give higher capacity and reduction ratio. The aim is to increase retention time of material in the crushing chamber or to reduce the velocity of material through the crushing chamber to allow multiple impacts to occur (higher reduction ratio).

The two most common types of cone crushers used in metal mining in North America are the Symons cone crushers manufactured by Nordberg and the Hydrocone crushers manufactured by Boliden-Allis (formerly Allis Chalmers). There are several other types manufactured throughout the world; the information contained here, however, is based on data from Nordberg and Allis Chalmers.

DEFINITION OF TERMS.

Crusher Sizing—Nordberg designates the crusher sizes by the mantle diameter measured in feet. At present, crushers ranging in size from 0.6 to 3 m (2 to 10 ft) are manufactured. Nordberg's descriptions of Symons cone crushers in general are vague but understood by the industry through common usage. Allis Chalmers' designations are a little more specific. These define both the feed opening and mantle diameter in inches. For example, Model 10-84 means a crusher with 254-mm (10-in.) feed opening and 2134-mm (84-in.) mantle diameter.

DESCRIPTIONS.

Symons Cone Crushers—Nordberg has two major types of cone crushers, a standard cone crusher that is used for secondary crushing applications and a short-head cone crusher that is used for tertiary crushing applications. Both types are made in sizes ranging from 0.6 to 3 m (2 to 10 ft). Over 1000 2-m (7-ft) crushers are in operation, but the 3-m (10-ft) crusher has found only limited application and is now being superseded by a high-capacity smaller unit (MP 1000).

The major difference between standard and short-head cone crushers is the shape of the crushing cavities. Standard cone crushers are designed to make a product ranging from 12.5 mm (1/2 in.) to more than 63.5 mm (2-1/2 in.) in both open and closed circuits. They may be furnished with fine, medium, coarse, or extra coarse cavities to meet variations in feed and end product size.

Short-head crushers as shown in Fig. 25.3.1.6 have a steeper head angle and a more parallel crushing cavity than standard machines. They are designed to produce a crushed product ranging from 3.2 mm (1/8 in.) to 19.0 mm (3/4 in.) in closed circuit with the appropriate screens. They may also be furnished with fine, medium, coarse, or extra coarse crushing cavities to meet variations in feed size and desired product size.

Nordberg also produces three other series of cone crushers: Omnicone, Gyrodisc, and Waterflush™ Wet Cone (Fig. 25.3.1.7) crushers. Omnicone requires less headroom and is available in sizes 940 to 1524 mm (37 to 60 in.) (note the change in measuring mantle diameter in inches instead of feet). Omnicone crushers should be considered as an alternative for mobile plants and low-tonnage plants.

Gyrodisc is a fine-product crushing machine and is capable of crushing hard, clean material down to –4.76 mm (4 mesh). It could, therefore, be a potential replacement for rod mills in a conventional crushing and grinding flowsheet. In the Gyrodisc crusher, crushing is achieved by a combination of impact and attrition. It will not handle wet sticky material nor will it handle material with excessive fines. Gyrodisc crushers are available from 914 to 2134 mm (36 to 84 in.).

Wet Cone crushers (WF Series) use Waterflush™ technology in which water is added to the crusher to flush fines through the crushing cavity. It is claimed that flushing produces a significant amount of flaky, shaped particles that break easily during ball milling. This results in elimination of the rod mill and a major reduction in overall comminution power requirements.

All Nordberg cone crushers are driven through a pinion mounted on a countershaft. The pinion drives the gear, which rotates the eccentric. The eccentric causes the main shaft head and mantle liner to gyrate, opening and closing the gap with the stationary bowl liner. The main shaft and head are supported on a spherical bronze socket liner. All assemblies except for the countershaft are removed from the top of the machine.

The crusher setting is adjusted by rotating the bowl, which has external threads that are screwed into a matching set of internal threads on the adjustment ring mounted on the main frame. On current models, the bowl is rotated by hydraulic rams mounted on the main frame. Tramp iron entering the crusher causes a spring release system to enlarge the crushing gap, allowing the tramp iron to pass. If the crusher becomes jammed, clearing jacks powered by a portable hydraulic raise the pack adjustment ring and bowl permitting noncrushable material to pass. The Omnicone has no springs, and protection is provided by a nitrogen/hydraulic accumulator system.

Approximate capacities for the standard and short-head crusher models are given in Tables 25.3.1.6 and 25.3.1.7. The capacities shown are neither maximum nor minimum and are based on results secured in actual practice. The figures are in short tons and apply to materials weighing 1602 kg/m^3 (100 lb/ft^3) and are based on a properly graded feed.

Screen analyses of crusher product at different crusher settings are listed in Table 25.3.1.8. It will be noted that for average ore, the tables are based on 70% of the product being finer than the constant closed-side setting. For hard ore, the tables are based on 60% passing the closed-side setting.

ALLIS CHALMERS HYDROCONE CRUSHERS: Allis Chalmers does not distinguish between a standard and short-head design but uses the same basic crusher for both secondary and tertiary crushing. The appropriate crusher is selected for each service by changing the crushing chamber design. Up to six different chamber designs are available for the 2134 mm (84 in.) (mantle diameter) model, which is the largest crusher in the series; fewer alternatives are available in the smaller sizes.

Allis Chalmers offers six sizes of crushers. Approximate open circuit capacities are shown in Table 25.3.1.9. Qualifications listed in the table should be noted.

25.3.1.7 Other Primary, Secondary, and Tertiary Compression Crushers

There are other *compression crushers* that should be considered for particular applications.

Mineral Sizers, manufactured by the MMD Group of Companies, Derbys, UK, can handle material up to 1.0 m^3 (35 ft^3) with a maximum throughput of up to 3180 t/h (3500 tph). The crusher has twin shafts which can be fitted with a range of teeth or picks. It is suitable for use on materials with compressive strengths in excess of 276,000 kPa (40,000 psi). The crusher is not suitable for low-abrasion ores.

The Stamler feeder/breaker, manufactured by the W.R. Stamler Corp., Millersburg, KY, is also a rotary pick breaker. The unit consists of a dump hopper, feeder, and single rotary pick breaker illustrated in Fig. 25.3.1.8. It can handle material up to a compressive strength of 207,000 kPa (30,000 psi) on a continuous basis. Maximum feed size is 1.2 by 1.8 by 1.8 m (4 by 6 by 6 ft) to produce a 203-mm (8-in.) product at up to

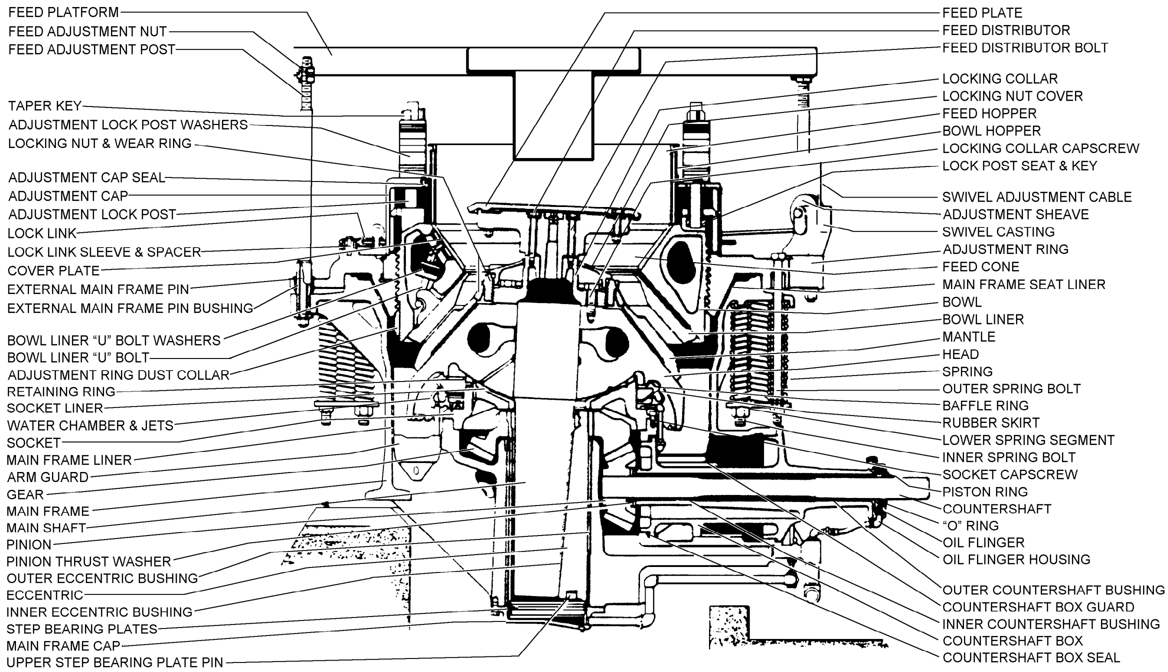


Fig. 25.3.1.6. Symons short-head, extra-heavy-duty cone crusher (*courtesy:* Rexnord, Inc.).

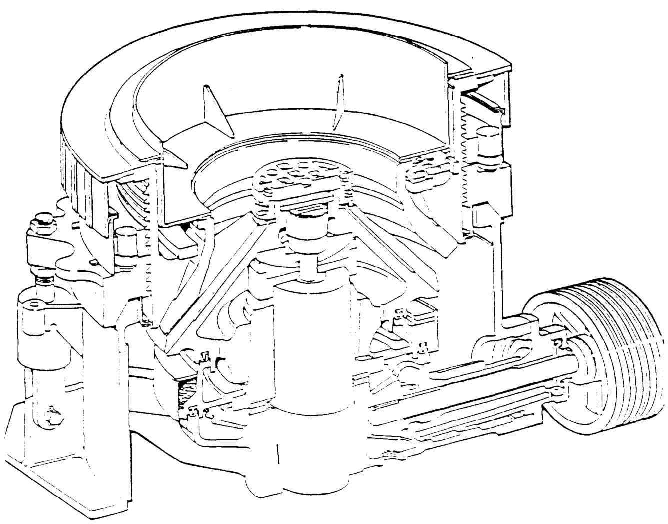


Fig. 25.3.1.7. Water flush cone crusher (*courtesy:* Nordberg, Inc.).

1800 t/h (2000 tph). The crusher is most suitable for low-abrasion ores.

Roll crushers, after a long and illustrious career, have virtually disappeared as tertiary crushers in the metal mining industry. They are still used to crush coal, and recently interest is being shown in a high-pressure roll crusher for very fine crushing. This roll crusher is described in the grinding section.

IMPACTORS AND HAMMERMILLS. There are numerous types of impactors and hammermills. The more common types are as follows:

A. Impactors.
  1. Common single rotor.
  2. Common double rotor.
  3. Single rotor reversible.
  4. Double rotor impactor with center feed.
B. Hammermills.
  1. Non-reversible hammermills.
  2. Non-reversible non-clog hammermills.
  3. Single rotor reversible hammermills.
  4. Ring-type hammermills.

Capacity—The capacity of a machine of given width is affected by the diameter of the hammer circle, the mass of moving parts, the tip speed, and the manner of feeding. Feed must penetrate the hammer circle if the full crushing surface is to be utilized. This calls for a balance between falling velocities and rotor velocities that differs considerably among various machines on the market.

Tables 25.3.1.10 and 25.3.1.11 list the approximate capacities for Universal impactors and hammermills manufactured by the Pettibone Corp.

25.3.1.8 Impact Crushing

Impact crushers crush by striking pieces of rock while in free fall and hurling them at high speed against stationary surfaces. Because the impact crusher depends for its effectiveness upon high velocity, wear is greater than in the slower-moving jaw and cone-type crushers. For this reason, its use is normally limited to relatively soft, friable, and sticky rocks that are characteristic of many nonmetallic mineral deposits.

Modern usage differentiates between the impactor and the hammermill. The impactor relies primarily on the impact of hammers (fixed or free-swinging) and secondarily upon pieces striking one another or steel surfaces; the hammermill relies on both the centrifugal impact force of free-swinging hammers and the attrition and shear action between these hammers and well-placed grates suspended at the bottom just below the hammer circle.

A recent innovation in impact crushing is the Barmac autogenous crushing mill, which is being applied with success to

Table 25.3.1.6. Standard Symons Cone Crusher Capacity Charts

STANDARD SYMONS CONE CRUSHERS—CAVITIES • FEED OPENINGS • PRODUCT SIZES • CAPACITIES

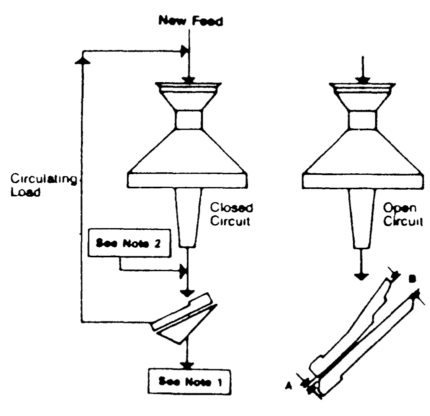
OPEN CIRCUIT—CAPACITIES IN TONS (2000 LB) PER HOUR PASSING THROUGH THE CRUSHER AT INDICATED DISCHARGE SETTING "A"

Size	Type of cavity	Recommended minimum discharge setting A	Feed opening with min. recommended discharge setting A		¼" (6 mm)	⅜" (9 mm)	½" (13 mm)	⅝" (16 mm)	¾" (19 mm)	⅞" (22 mm)	1" (25 mm)	1¼" (31 mm)	1½" (38 mm)	2" (51 mm)	2½" (64 mm)
			B Closed side	B Open side											
2 ft. (600 mm)	Fine	¼" (6 mm)	2¼" (57 mm)	2⅞" (72 mm)	18	20	25	30	34	40	45	50	60		
	Coarse	⅜" (9 mm)	3¼" (83 mm)	4⅜" (109 mm)		20	25	30	35	45	50	60	75		
	Extra Coarse	½" (13 mm)	4" (100 mm)	4⅜" (109 mm)			25	30	40	50	55	70	80		
3 ft (900 mm)	Fine	⅜" (9 mm)	3 5/16" (83 mm)	4 1/16" 102 mm)		50	65	80	90	100					
	Coarse	½" (13 mm)	6⅜" (159 mm)	7" (175 mm)			65	80	100	120	130	150	180		
	Extra Coarse	1" (25 mm)	6½" (163 mm)	7⅛" (178 mm)							130	150	180		
4 ft (1200 mm)	Fine	⅜" (9 mm)	5" (127 mm)	5¼" (131 mm)		70	100	120	140	155	170	185			
	Medium	½" (13 mm)	6⅛" (156 mm)	6¼" (156 mm)			110	130	150	160	180	200	220		
	Coarse	¾" (19 mm)	7⅛" (178 mm)	7⅝" (191 mm)					155	170	200	220	270	340	
	Extra Coarse	1" (25 mm)	9¼" (231 mm)	10" (250 mm)							210	230	280	350	
4¼ ft (1275 mm)	Fine	½" (13 mm)	4⅜" (109 mm)	5⅜" (137 mm)			120	140	160	170	180	200			
	Medium	⅝" (16 mm)	7½" (188 mm)	8¼" (210 mm)				145	175	190	220	250	280		
	Coarse	¾" (19 mm)	8⅝" (216 mm)	9⅝" (241 mm)					190	215	240	275	325	385	
	Extra Coarse	1" (25 mm)	9½" (238 mm)	10⅜" (259 mm)							260	300	335	395	
5½ ft (1650 mm)	Fine	⅝" (16 mm)	7½" (188 mm)	8⅜" (209 mm)				200	225	250	285	325	360		
	Medium	⅞" (22 mm)	8½" (213 mm)	9⅝" (241 mm)						285	320	370	420	460	
	Coarse	1" (25 mm)	9⅝" (241 mm)	10¾" (269 mm)							330	390	480	500	700
	Extra Coarse	1½" (38 mm)	13¼" (331 mm)	14½" (368 mm)									475	525	750
7 ft HD (2100 mm)	Fine	¾" (19 mm)	10⅛" (253 mm)	11⅛" (278 mm)					420	450	550	680	800		
	Medium	1" (25 mm)	12⅛" (303 mm)	13⅜" (334 mm)							670	800	890	1100	
	Coarse	1¼" (31 mm)	13⅜" (334 mm)	14¾" (369 mm)								870	930	1200	1400
	Extra Coarse	1½" (38 mm)	16¾" (425 mm)	18⅛" (460 mm)									970	1300	1500
7 ft SHD								CONSULT FACTORY FOR CAPACITIES							

CLOSED CIRCUIT—CAPACITIES IN TONS (2000 LB) PER HOUR BASED ON CLOSED CIRCUIT OPERATION

Size	Type of cavity	Recommended minimum discharge setting A	Feed opening with min. recommended discharge setting A: B Closed side	Feed opening with min. recommended discharge setting A: B Open side	Effective square opening on circuit screen: 1/4" (6mm)		3/8" (10mm)		1/2" (13mm)		5/8" (16mm)		3/4" (19mm)		7/8" (22mm)		1" (25mm)		1 1/4"(31mm)		1 1/2"(38mm)		2" (50mm)		2 1/2"(63mm)	
					Recommended closed side settings for closed circuit operation: 1/4" (6mm)		3/8" (10mm)		3/8" (10mm)		1/2" (13mm)		5/8" (16mm)		3/4" (19mm)		3/4" (19mm)		7/8" (22mm)		1" (25mm)		1 1/4"(32mm)		1 1/2"(38mm)	
					Note 1	Note 2	Note 1	Note 2	Note 1	Note 2	Note 1	Note 2	Note 1	Note 2	Note 1	Note 2	Note 1	Note 2	Note 1	Note 2	Note 1	Note 2	Note 1	Note 2	Note 1	Note 2
2 ft (600 mm)	Fine	1/4" (6 mm)	2 1/4" (57 mm)	2 7/8" (72 mm)	10	20	12	24	16	28	20	30	24	35	28	42	32	42	40	50	48	58				
	Coarse	3/8" (9 mm)	3 1/4" (83 mm)	3 3/4" (95 mm)			12	24	16	28	20	30	24	35	28	42	32	42	40	50	48	58				
	Extra Coarse	1/2" (13 mm)	4" (100 mm)	4 3/8" (109 mm)							20	30	24	35	28	42	32	45	40	55	48	65				
3 ft (900 mm)	Fine	3/8" (9 mm)	3 5/16" (83 mm)	4 1/16" (102 mm)			35	65	40	70	45	75	60	90	70	105	80	105	95	120	120	150	145	170		
	Coarse	1/2" (13 mm)	6 3/8" (159 mm)	7" (175 mm)							45	75	60	90	75	110	85	110	100	125	120	150	145	170		
	Extra Coarse	1" (25 mm)	6 1/2" (163 mm)	7 1/8" (178 mm)											75	110										
4 ft (1200 mm)	Fine	3/8" (9 mm)	5" (127 mm)	5 1/4" (131 mm)			50	95	55	95	70	105	85	125	100	150	120	155	130	160	165	200				
	Medium	1/2" (13 mm)	6 1/8" (156 mm)	6 1/4" (156 mm)							70	110	100	145	110	160	125	165	140	170	170	210	195	236		
	Coarse	3/4" (19 mm)	7 1/8" (178 mm)	7 5/8" (191 mm)											115	175	130	175	150	185	170	210	195	235		
	Extra Coarse	1" (25 mm)	9 1/4" (231 mm)	10" (250 mm)												175		175		185						
4 1/4 ft (1275 mm)	Fine	1/2" (13 mm)	4 3/8" (109 mm)	5 3/8" (137 mm)							95	135	105	155	125	185	145	190	175	220	205	205				
	Medium	5/8" (16 mm)	7 1/2" (188 mm)	8 1/4" (210 mm)									120	175	135	200	155	205	190	235	215	260	235	280		
	Coarse	3/4" (19 mm)	8 5/8" (216 mm)	9 5/8" (241 mm)											140	210	165	215	190	245	220	270	250	295		
	Extra Coarse	1" (25 mm)	9 1/2" (238 mm)	10 3/8" (259 mm)																	225	275	260	310		
5 1/2 ft (1650 mm)	Fine	5/8" (16 mm)	7 1/2" (188 mm)	8 3/8" (209 mm)									180	235	180	265	210	270	240	300	300	360				
	Medium	7/8" (22 mm)	8 1/2" (213 mm)	9 5/8" (241 mm)															250	310	310	375	330	390		
	Coarse	1" (25 mm)	9 5/8" (241 mm)	10 3/4" (269 mm)																	315	380	340	400	400	465
	Extra Coarse	1 1/2" (38 mm)	13 1/4" (331 mm)	14 1/2" (368 mm)																			350	410	420	490
7 ft HD (2100 mm)	Fine	3/4" (19 mm)	10 1/8" (253 mm)	11 1/8" (278 mm)	CONSULT FACTORY FOR CAPACITIES																					
	Medium	1" (25 mm)	12 1/8" (303 mm)	13 3/8" (334 mm)																						
	Coarse	1 1/4" (31 mm)	13 5/8" (334 mm)	14 3/4" (369 mm)																						
	Extra Coarse	1 1/2" (38 mm)	16 3/4" (425 mm)	18 1/8" (460 mm)																						
7 ft SHD					CONSULT FACTORY FOR CAPACITIES																					

Source: Nordberg Manufacturing Co.



Note 1 Net finished product (screen undersize)   **Note 2** Tons per hour passing through crusher (net finished product plus recirculating load)

Table 25.3.1.7. Short-head Symons Cone Crusher Capacity Charts

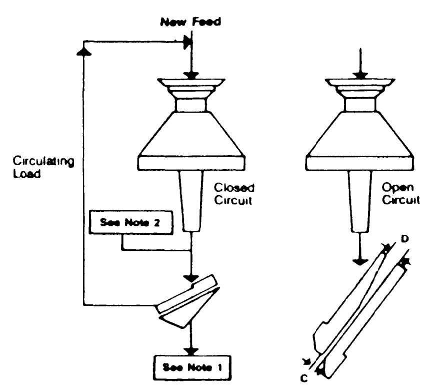
SHORTHEAD SYMONS CONE CRUSHERS—CAVITIES • FEED OPENINGS • PRODUCT SIZES • CAPACITIES
OPEN CIRCUIT—CAPACITIES IN TONS (2000 LB) PER HOUR PASSING THROUGH THE CRUSHER AT INDICATED DISCHARGE SETTING C

Size	Type of cavity	Recommended minimum discharge setting C	Feed opening with min. recommended discharge setting C: D Closed side	D Open side	1/8″ (3 mm)	3/16″ (5 mm)	1/4″ (6 mm)	3/8″ (10 mm)	1/2″ (13 mm)	5/8″ (16 mm)	3/4″ (19 mm)	1″ (25 mm)
2 ft (610 mm)	Fine	1/8″ (3 mm)	3/4″ (19 mm)	1 3/8″ (35 mm)	10	18	20	30	40			
	Coarse	3/16″ (5 mm)	1 1/2″ (38 mm)	2″ (51 mm)		18	24	32	45	55		
3 ft (914 mm)	Fine	1/8″ (3 mm)	1/2″ (13 mm)	1 5/8″ (41 mm)	30	45	60	75	100			
	Medium	1/8″ (3 mm)	1 5/16″ (33 mm)	2 3/8″ (60 mm)	30	45	60	75	100	110		
	Coarse	1/4″ (6 mm)	2″ (51 mm)	3″ (76 mm)			65	80	105	125	140	
4 ft (1219 mm)	Fine	3/16″ (5 mm)	1 1/8″ (29 mm)	2 1/4″ (57 mm)		55	85	95	135	145		
	Medium	5/16″ (8 mm)	1 3/4″ (44 mm)	2 7/8″ (73 mm)				100	145	160		
	Coarse	1/2″ (13 mm)	2 3/16″ (56 mm)	3 1/2″ (89 mm)					155	180	200	
	Extra Coarse	5/8″ (16 mm)	3 1/2″ (89 mm)	4 5/8″ (117 mm)					160	185	210	240
4 1/4 ft (1295 mm)	Fine	1/8″ (3 mm)	1 1/8″ (29 mm)	2 1/2″ (64 mm)	40	65	90	115	150	180		
	Medium	1/4″ (6 mm)	2 1/8″ (54 mm)	3 1/2″ (89 mm)			90	115	150	180		
	Coarse	5/16″ (8 mm)	2 3/4″ (70 mm)	4 1/8″ (105 mm)				120	175	200	220	250
	Extra Coarse	5/8″ (16 mm)	3 7/8″ (96 mm)	5 1/4″ (133 mm)							230	260
5 1/2 ft (1676 mm)	Fine	3/16″ (5 mm)	1 3/8″ (35 mm)	2 3/4″ (70 mm)		100	150	180	230			
	Medium	1/4″ (6 mm)	2 1/8″ (54 mm)	3 1/2″ (89 mm)			150	180	230	280	310	
	Coarse	3/8″ (10 mm)	3 7/8″ (98 mm)	5 1/4″ (33 mm)				210	280	310	340	370
	Extra Coarse	1/2″ (13 mm)	4 5/8″ (117 mm)	5 1/4″ (133 mm)					280	310	340	370
7 ft HD (2134 mm)	Fine	3/16″ (5 mm)	2″ (51 mm)	4 1/8″ (105 mm)		210	300	360	400	450		
	Medium	3/8″ (10 mm)	3 3/4″ (95 mm)	5 1/4″ (133 mm)				390	450	500	560	
	Coarse	1/2″ (13 mm)	5″ (127 mm)	7″ (178 mm)					500	530	600	660
	Extra Coarse	5/8″ (16 mm)	6″ (152 mm)	8″ (203 mm)						560	650	720
7 ft SHD					CONSULT FACTORY FOR CAPACITIES							

CLOSED CIRCUIT—CAPACITIES IN TONS (2000 LB) PER HOUR BASED ON CLOSED CIRCUIT OPERATION

Size	Type of cavity	Recommended minimum discharge setting C	Feed opening with min. recommended discharge setting C: D Closed side	Feed opening with min. recommended discharge setting C: D Open side	Effective square opening on closed circuit screen: 1/8″ (3mm); Recommended closed side settings for closed circuit operation: 1/8″ (3mm); Note 1	Note 2	3/16″ (5mm); 3/16″ (5mm); Note 1	Note 2	1/4″ (6mm); 3/16″ (5mm); Note 1	Note 2	3/8″ (10mm); 1/4″ (6mm); Note 1	Note 2	1/2″ (13mm); 3/8″ (10mm); Note 1	Note 2	5/8″ (16mm); 1/2″ (13mm); Note 1	Note 2	3/4″ (19mm); 5/8″ (16mm); Note 1	Note 2	1″ (25mm); 3/4″ (19mm); Note 1	Note 2
2 ft (610 mm)	Fine	1/8″ (3 mm)	3/4″ (19 mm)	1 3/8″ (35 mm)	6	12	10	20	13	20	15	23	25	32	30	45				
	Coarse	3/16″ (5 mm)	1 1/2″ (38 mm)	2″ (51 mm)			10	20	13	20	18	27	26	35	35	50				
3 ft (914 mm)	Fine	1/8″ (3 mm)	1/2″ (13 mm)	1 5/8″ (41 mm)	15	30	20	40	30	45	50	65	70	85	80	110				
	Medium	1/8″ (3 mm)	1 5/16″ (33 mm)	2 3/8″ (60 mm)	15	30	20	40	30	45	50	65	70	85	80	110	95	135		
	Coarse	1/4″ (6 mm)	2″ (51 mm)	3″ (76 mm)							55	70	75	90	85	115	105	145	130	150
4 ft (1219 mm)	Fine	3/16″ (5 mm)	1 1/8″ (29 mm)	2 1/4″ (57 mm)			30	60	40	60	60	95	80	105	100	140				
	Medium	5/16″ (8 mm)	1 3/4″ (44 mm)	2 7/8″ (73 mm)									85	110	100	140	120	175		
	Coarse	1/2″ (13 mm)	2 3/16″ (56 mm)	3 1/2″ (89 mm)											115	160	145	210		
	Extra Coarse	5/8″ (16 mm)	3 1/2″ (89 mm)	4 5/8″ (117 mm)													150	220	170	205
4 1/4 ft (1295 mm)	Fine	1/8″ (3 mm)	1 1/8″ (29 mm)	2 1/2″ (64 mm)	20	40	35	70	55	80	75	110	100	130	125	170				
	Medium	1/4″ (6 mm)	2 1/8″ (54 mm)	3 1/2″ (89 mm)							80	120	105	140	125	170				
	Coarse	5/16″ (8 mm)	2 3/4″ (70 mm)	4 1/8″ (105 mm)									110	150	140	200	175	250	200	260
	Extra Coarse	5/8″ (16 mm)	3 7/8″ (98 mm)	5 1/4″ (133 mm)													175	250	200	260
5 1/2 ft (1676 mm)	Fine	3/16″ (5 mm)	1 3/8″ (35 mm)	2 3/4″ (70 mm)			65	130	90	140	135	200	175	220	210	270				
	Medium	1/4″ (6 mm)	2 1/8″ (54 mm)	3 1/2″ (89 mm)							135	200	175	220	210	270	245	350	280	340
	Coarse	3/8″ (10 mm)	3 7/8″ (98 mm)	5 1/4″ (133 mm)									180	240	220	280	250	360	320	385
	Extra Coarse	1/2″ (13 mm)	4 5/8″ (117 mm)	5 1/4″ (133 mm)											220	280	250	360	320	395
7 ft HD (2134 mm)	Fine	3/16″ (5 mm)	2″ (51 mm)	4 1/8″ (105 mm)			120	240	160	230	240	360	315	390	360	450	420	500		
	Medium	3/8″ (10 mm)	3 3/4″ (95 mm)	5 1/4″ (133 mm)									315	420	360	500	420	550	500	600
	Coarse	1/2″ (13 mm)	5″ (127 mm)	7″ (178 mm)											380	540	450	650	550	660
	Extra Coarse	5/8″ (16 mm)	6″ (152 mm)	8″ (203 mm)													465	680	550	690
7 ft SHD	CONSULT FACTORY FOR CAPACITIES																			

Source: Nordberg Manufacturing.



Note 1 Net finished product (screen undersize)

Note 2 Tons per hour passing through crusher (net finished product plus recirculating load)

Table 25.3.1.8. Feed Material Rating (Average/Hard) for Cone Crushers
Open Circuit Product Analysis* (% Passing) for Given Closed Side Setting of Crusher

Product Size Inch (mm)	Closed Side Setting of Crusher, in. (mm)							
	¼ (6)	⅜ (10)	½ (13)	¾ (19)	1 (25)	1¼ (32)	1½ (38)	2 (51)
4 in. (102 mm)								100 / 100
3 in. (76 mm)							100 / 100	91 / 92
2½ in. (64 mm)						100 / 100	95 / 95	82 / 79
2 in. (51 mm)					100 / 100	93 / 94	86 / 84	70 / 60
1½ in. (38 mm)				100 / 100	92 / 92	80 / 76	70 / 60	51 / 40
1¼ in. (32 mm)				95 / 95	82 / 79	70 / 60	57 / 46	41 / 30
1 in. (25 mm)			100 / 100	86 / 84	70 / 60	54 / 43	44 / 33	32 / 22
¾ in. (19 mm)		100 / 100	92 / 92	70 / 60	51 / 40	39 / 29	32 / 22	25 / 16
½ in (13 mm)	100 / 100	86 / 84	70 / 60	44 / 33	32 / 22	26 / 17	22 / 14	17 / 11
⅜ in. (10 mm)	94 / 92	70 / 60	51 / 40	32 / 22	25 / 16	20 / 13	17 / 11	13 / 8
¼ in. (6 mm)	70 / 60	44 / 33	32 / 22	22 / 14	17 / 11	14 / 9	12 / 7	9 / 6
3/16 in. (5 mm)	50 / 40	32 / 22	25 / 16	17 / 11	13 / 8	11 / 7	9 / 6	7 / 4
No. 6 (3 mm)	34 / 24	23 / 15	18 / 12	12 / 8	10 / 6	8 / 5	7 / 4	5 / 3
No. 8 (2 mm)	25 / 16	17 / 11	13 / 8	9 / 6	7 / 4	6 / 4	5 / 3	4 / 2

*Note: These table values will vary with the method of feeding, selection of crusher cavity, the weight, cleanliness, and moisture contents of the material and its fracture pattern. Accurate values should be established by actual testing.

Source: Nordberg Manufacturing Co.

hard abrasive material only previously crushed by compressive crushers.

The basic principle of impact breaking is the total use of rotor energy in the reduction of the material. Kinetic energy is imparted to a particle of the material, and this energy is used chiefly to break down the bonding of the particle structure. The particle breaks along its planes of weakness and, as a result, produces a natural grain-shaped product.

Product gradation is a function of rotor speed and friability of the material. An increase in rotor speed or friability of material leads to an increase in the proportion of fines produced. Conversely, a decrease in rotor speed or friability of material will decrease the proportion of fines in the product. This is illustrated in Table 25.3.1.12.

The desired product gradation is an important factor in choosing between an impactor and a hammermill. This is illustrated by the size-distribution curve. Fig. 25.3.1.9 indicates the differences in products obtained from crushing an average limestone in both an impactor and a hammermill operating with approximately the same hammer tip speed. The hammermill yields an overall finer product gradation.

25.3.1.9 Crushing Circuit Design

A typical three-stage crushing circuit in metal mining application is shown in Fig. 25.3.1.10. This illustrates the step-by-step reduction of a run-of-mine ore in the primary, secondary, and tertiary crushing stages to a feed size suitable for the subsequent grinding stage. There are many variations in circuit design, but the basic design and equipment selection techniques are common to all circuits.

The operating schedule for each stage of crushing determines the tonnage rate for that stage. Typically, the primary crusher operates at the same schedule as ore deliveries from the mine. The surge pile after the primary crusher will allow the secondary and tertiary crushers to operate on a different schedule. In the example shown, all stages of crushing operate on the same schedule.

As stated earlier, the gyratory crusher is sized on the basis of feed size and/or tonnage rate.

The tonnage and size distribution of the gyratory crusher product will determine the double-deck screen size and the tons of feed fed to the secondary cone crusher. The secondary crusher

Table 25.3.1.9. Hydrocone Crusher Capacities

Crusher size, in.	Horsepower† range (electric)	Hydrocones: Crushing chamber in.	mm	Type	Top shell spider arm‡	Open circuit capacities,* tph (2000 lb): Closed side setting, in. and (mm) ¼ (6)	⅜ (10)	½ (13)	⅝ (16)	¾ (19)	⅞ (22)	1 (25)	1¼ (32)	1½ (38)	2 (51)	2½ (64)
22		1	25	Fine	3	13										
(559 mm)	25–30	2	51	Medium	3	20	24	26	28							
		3	76	Coarse	3		34	36	38	40	45					
36		2	51	Short	3	75	80	85	90	100						
(914 mm)	75–125	2	51	Fine	3		70	75	80	95						
		4	102	Medium	2 or 3		75	80	90	100	110	115				
		5½	140	Medium coarse	2			85	100	110	120	130	140	150		
		7	178	Coarse	2					120	140	160	175	190		
45		2½	64	Short	3		110	125	135	140						
(1143 mm)	100–200	3	76	Fine	3		80	95	105	110						
		6	152	Medium	2 or 3			145	155	160	165	175	190	200		
		7½	191	Medium coarse	2					170	180	195	210	225		
		9	229	Coarse	2					195	215	250	265	265	290	
51		2	51	Short	3		145	160	170	190	200	210				
(1295 mm)	125–200	3½	89	Fine	3		120	125	130	140	145	155	170	190		
		5	127	Medium fine	3			140	145	155	160	170	185	205		
		6	152	Medium	2 or 3			165	185	205	220	235	255	275		
		7	178	Medium coarse	2				190	210	225	245	265	285		
		10	254	Coarse	2					215	240	265	290	315	350	
60		3	76	Short	3		160	190	215	225	235	245				
(1524 mm)	200–300	4	102	Fine	3		155	165	175	185	195	205				
		7	178	Medium	2 or 3				275	290	300	310	330	350		
		10	254	Medium coarse	2						310	325	350	385	420	
		12	305	Coarse	2							350	400	450	525	600
84		3	76	Short	3		360	390	430	470	510	540				
(2134 mm)	300–500	5	127	Fine	3		400	400	470	515	535	560				
		7	178	Medium fine	3			470	520	535	550	570	640	690		
		10	254	Medium	2 or 3					575	625	675	750	850		
		13	330	Medium coarse	2						650	700	800	900	1100	
		17	432	Coarse	2									950	1200	1300

Source: Allis Chalmers.

Table 25.3.1.9.—Cont.

Hydrocone crushers with 3-arm spider top shells					Closed circuit, § tph (2000 lbs) — Product size, in. (mm) — A: Net finished product passing screen opening shown (tph)— B: Approx. total tph passing through crusher (net finished product plus circulating load)															
Crusher size, in.	Horsepower† range (electric)	Crushing Chamber			1/8 (3)		3/16 (5)		1/4 (6)		3/8 (10)		1/2 (13)		5/8 (16)		3/4 (19)		1 (25)	
		in.	mm	type	A	B	A	B	A	B	A	B	A	B	A	B	A	B	A	B
22 (559 mm)	25–30	1	25	Fine	6	10	7	11	8	13										
		2	51	Medium			11	18	14	23	18	28	22	31	24	34				
36 (914 mm)	75–125	2	51	Short	35	70	40	75	45	80	60	95	65	100	70	105	80	120		
		2	51	Fine			35	65	40	70	55	85	60	90	65	95	75	100		
		4	102	Medium							55	85	60	90	70	105	80	120	85	125
45 (1143 min)	100–200	2½	64	Short			50	105	70	125	90	135	100	150	105	160	125	170		
		3	76	Fine					50	85	55	90	60	95	65	105	80	120		
		6	152	Medium							100	105	110	165	120	180	125	185	135	200
51 (1295 mm)	125–200	2	51	Short			60	125	75	150	120	175	125	190	135	200	160	225	175	250
		3½	89	Fine					55	110	80	120	85	130	90	140	105	160	130	190
		5	127	Medium fine							90	135	95	145	100	155	115	175	140	205
		6	152	Medium									125	195	140	215	165	235	190	270
60 (1524 mm)	200–300	3	76	Short					95	160	125	190	155	230	175	260	190	270	205	290
		4	102	Fine							120	180	130	195	140	210	160	225	270	240
		7	178	Medium											200	300	225	330	250	350
84 (2134 mm)	300–500	3	76	Short					220	340	270	400	315	460	360	520	420	565	485	645
		5	127	Fine							320	475	345	525	380	565	450	615	500	670
		7	178	Medium fine									375	565	420	625	465	640	510	680

* Capacities shown are based on open-circuit operation and on properly sized, clean, dry, scalped, friable feed weighing 100 lb/ft.^{3}

† Actual horsepower required depends on stone hardness, ratio of reduction, etc. Consult Allis-Chalmers before purchasing motors.

‡ Hydroclone crushers with 2-arm spider top shells are normally used as secondary crushers and operated in open circuit. Those with 3-arm spider top shells are normally used as third- or fourth-stage crushers, sometimes operated in open circuit, but more often in closed circuit.

§ Capacities are based on closed-circuit operation and on properly sized, clean, dry, scalped, fiable feed weighing 100 lb/ft^3. Maximum Hydroclone control pressure, 500 psi. After Allis-Chalmers.

Conversion factor: 1 ton = 0.9072 t.

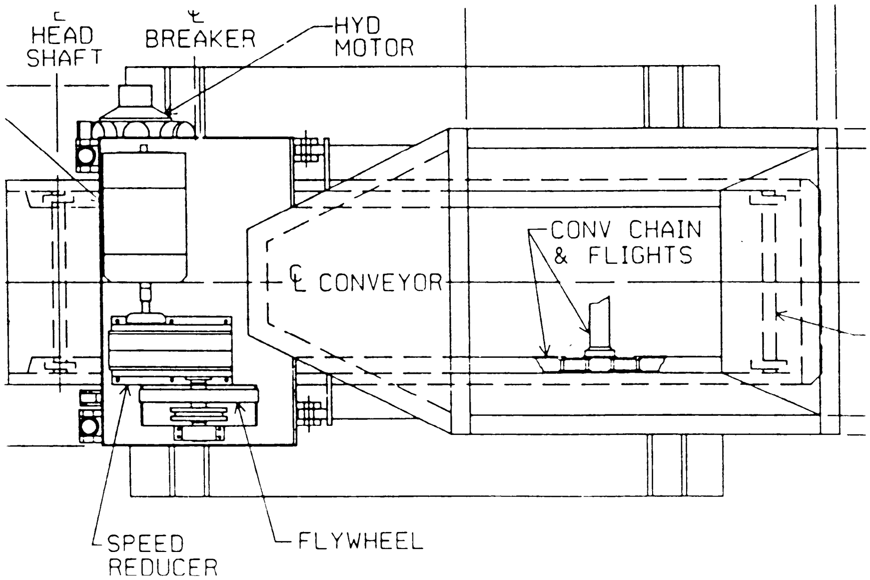


Fig. 25.3.1.8. Stamler feeder breaker (*courtesy:* W.R. Stamler, Inc.).

Table 25.3.10. Capacity of Impactors

Model Single Rotor	RPM Range	Cont. hp kW	Capacity tph (t/h)	Nominal Prod. in. (mm)
3755	450–900	200–250 hp (150–186 kW)	Up to 400 tph (363 t/h)	3 in. (76 mm)
4555	450–900	250–350 (186–261 kW)	Up to 500 tph (454 t/h)	3 in. (76 mm)
4560	400–850	250–350 hp (186–261 kW)	Up to 600 tph (544 t/h)	4 in. (102 mm)
5165	400–850	300–450 hp (224–336 kW)	Up to 750 tph (680 t/h)	6 in. (152 mm)
6185	400–600	400–600 hp (298–447 kW)	Up to 1400 tph (1270 t/h)	6 in. (152 mm)
7495	400–450	750–1000 (559–746 kW)	Up to 2000 tph (1814 t/h)	6 in. (152 mm)
Model Double Rotor				
3645	500–1000	200–300 hp (150–221 kW)	Up to 400 tph (363 t/h)	3 in. (76 mm)
4650	400–850	300–400 hp (224–298 kW)	Up to 500 tph (454 t/h)	4 in. (102 mm)
5760	300–650	500–1200 hp (373–895 kW)	Up to 2000 tph (1814 t/h)	6 in. (152 mm)

Source: Pettibone Universal, Inc.

Table 25.3.1.11. Capacity of Hammermills
HammerMaster

Model Number	Rotor Disk Thickness, in. (mm)	Hammer Wt. lb (kg) Manganese	Hammer Circle Dia., in. (mm)	Feed Opening in. (mm)	Speed RPM Road Rock	Speed RPM AgLime	Hp Required Req. Diesel Electric	Capacity tph $\frac{3}{16}$ in. Lime Dust	Capacity tph 1 in. Road Rock	Capacity tph $1\frac{1}{2}$ in. Road Rock
2636	$2\frac{1}{2}$	60	36	15 × 26	900	1100	175–200	40–60	90–150	110–180
	(63.5)	(27.24)	(91.44)	(406.4 × 660.4)			150	36–54	82–136	100–163
4136	$2\frac{1}{2}$	60	36	16 × 41	900	1100	250–300	55–90	120–185	170–220
	(63.5)	(27.24)	(91.44)	(406.4 × 104.14)			200	50–82	109–168	154–220
5136	$2\frac{1}{2}$	60	36	16 × 51	900	1100	300–450	70–110	150–230	210–275
	(63.5)	(27.24)	(91.44)	(406.4 × 129.54)			250	63–100	136–209	190–249
LimeMaster										
2636	$2\frac{1}{2}$	60	35	13 × 26		1800	300–450	30–50		
	(63.5)	(27.24)	(914.4)	(330.2 × 660.4)			250	27–45		
4136	$2\frac{1}{2}$	60	36	13 × 41		1800	450–550	45–75		
	(63.5)	(27.24)	(914.4)	(330.2 × 1041.4)			400	41–68		
4242	$2\frac{1}{2}$	60	42	13 × 42		1500	450–550	55–85		
	(63.5)	(27.24)	(1066.8)	(330.2 × 1066.8)			500	50–77		

Source: Pettibone Universal, Inc.

Table 25.3.1.12. Difference in Products from Impactor and Hammermill Crushers

CUMULATIVE PERCENT PASSING

Size, in.	2	3	4	5	6	2	3	4	5	6	2	3	4	5	6	2	3	4	5	6	Size, in.
10					97				95	90			98	90	81			95	85	75	10
9				98	93			98	93	87			95	85	76			91	79	69	9
8				95	89			95	90	82			90	79	71		99	85	72	62	8
7			98	91	84		98	91	84	77		97	81	73	65		93	76	64	56	7
6			93	85	79		95	86	79	71		90	76	66	60		85	67	56	50	6
5		96	87	79	75	98	90	73	60	64	98	80	68	60	54	95	71	59	50	45	5
4		90	80	72	68	95	72	65	59	57	90	71	60	54	48	85	62	50	44	40	4
3½	98	85	75	68	64	92	67	51	55	53	83	65	55	50	44	73	55	46	41	37	3½
3	93	80	70	63	58	87	70	61	55	50	76	60	52	47	41	64	50	42	38	34	3
2¾	90	76	67	60	55	83	67	58	52	47	72	57	49	44	39	61	47	40	35	32	2¾
2½	87	73	64	57	53	79	64	55	50	45	68	54	47	41	37	57	45	38	33	30	2½
2¼	83	69	61	54	50	77	61	53	47	42	64	52	44	39	35	53	42	36	31	28	2¼
2	79	65	58	51	47	74	58	50	44	39	60	49	41	37	33	50	39	33	29	26	2
1¾	74	61	54	48	44	68	54	46	41	36	55	46	38	34	30	46	36	30	26	23	1¾
1½	70	56	50	44	40	64	50	43	37	33	51	42	35	31	27	42	33	27	23	20	1½
1¼	64	51	45	40	36	57	45	38	33	29	46	38	32	27	24	36	30	24	20	17	1¼
1	58	45	40	35	32	50	40	34	28	25	41	33	28	24	21	31	25	20	17	14	1
⅛	54	42	37	33	29	47	37	31	26	23	38	30	25	22	20	28	23	18	15	12	⅛
¾	50	40	34	30	26	43	33	26	23	20	35	27	23	20	18	25	20	16	13	10	¾
⅝	45	35	31	27	23	38	29	24	19	18	32	24	20	17	16	23	17	13	11	9	⅝
½	40	31	27	24	20	34	25	20	16	15	27	21	17	14	11	18	14	10	9	6	½
⅜	34	26	22	19	16	28	20	16	14	12	22	17	13	11	9	15	11	8	7	4	⅜
¼	26	20	16	15	12	20	15	12	10	8	16	12	9	8	7	10	7	5	4	3	¼
4M	21	16	13	11	9	16	12	9	8	6	12	8	7	6	5	7	5	4	3	2	4M
8M	11	9	8	7	6	9	7	6	5	4	7	5	4	4	3	4	3	3	2	1	8M
Crusher Size and RPM	3445 @ 600 RPM 3850 @ 600 RPM					3445 @ 450 RPM 3850 @ 500 RPM 4654 @ 550 RPM					3850 @ 400 RPM 4654 @ 450 RPM 5660 @ 450 RPM					4654 @ 350 RPM 5660 @ 350 RPM					

Note: The results shown above indicate an average analysis of stone crushed by a single-rotor, two-hammer crusher working in dry, free-crushing medium hard limestone. No guarantee of product sizing can be inferred, since stone varies from area to area along with other local conditions. These are typical gradations for the speeds shown. Coarser or finer gradings can be achieved by making speed changes.
Source: Rexnord, Inc.
Conversion factor: 1 in. = 25.4 mm.

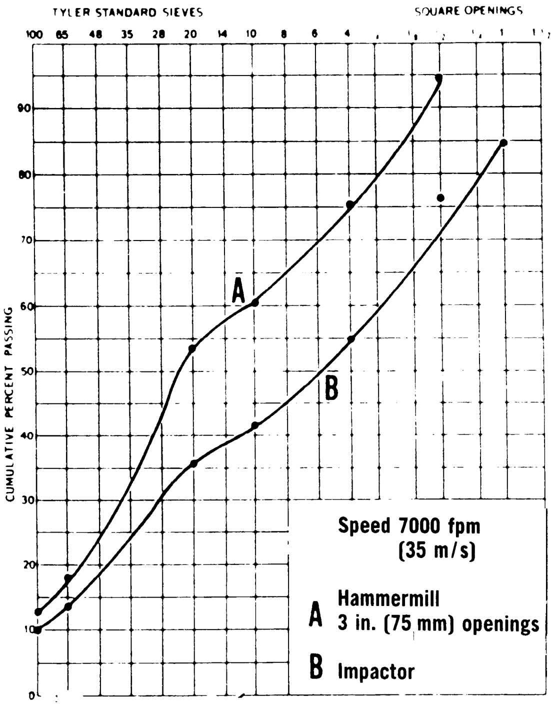


Fig. 25.3.1.9. Product gradation for Pennsylvania Hammermill vs. Pennsylvania impactor; see Table 25.3.1.12 (*courtesy:* Pennsylvania Crusher Corp.).

is sized from the standard cone capacity tables based on feed rate to the crusher and the type of cavity required to yield the desired product gradation.

Tertiary crushers are sized from the short-head cone capacity tables based on the tonnage rate, screen size and crusher setting. The relationship between the crusher setting and screen size determines the circulating load that can be calculated from the production gradation curves and screen efficiency.

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25.3.2 GRINDING

M.I. CALLOW AND V.P. KENYEN

Grinding is the further reduction of crushed products to a size suitable for a subsequent concentration process. This is normally achieved by tumbling the crushed ore together with a grinding media, in a rotating cylinder or a stirred vertical cylinder. The grinding media may be steel balls, rods, or rock pebbles. Most of the energy is consumed in keeping the mill or stirrer and media in motion, and only a very small part is used to perform the useful work of breaking the ore that surrounds the media.

Grinding is statistical in nature. The feed ore has a particle size distribution that depends on the ore characteristics and the crushing process. When ground, there is a certain probability that any given piece will be broken. The objective of grinding circuit design is therefore to select conditions that increase the probability of breaking particular size particles. This must be accomplished at the optimum combination of capital and operating costs to minimize the total cost over the life of the mine. Grinding is a low-efficiency, power-intensive process and typi-

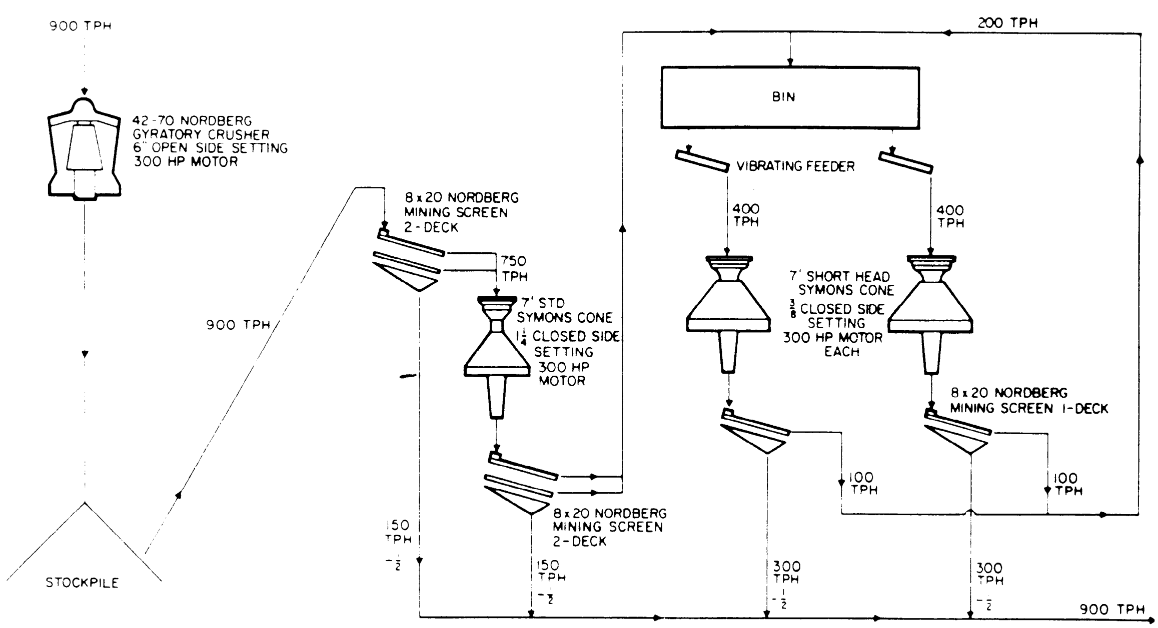

Fig. 25.3.1.10. Three-stage crusher plant flowsheet for producing ball mill feed (*courtesy:* Rexnord, Inc.). Conversion factors: 1 in. = 25.4 mm, 1 ton = 0.9072 t.

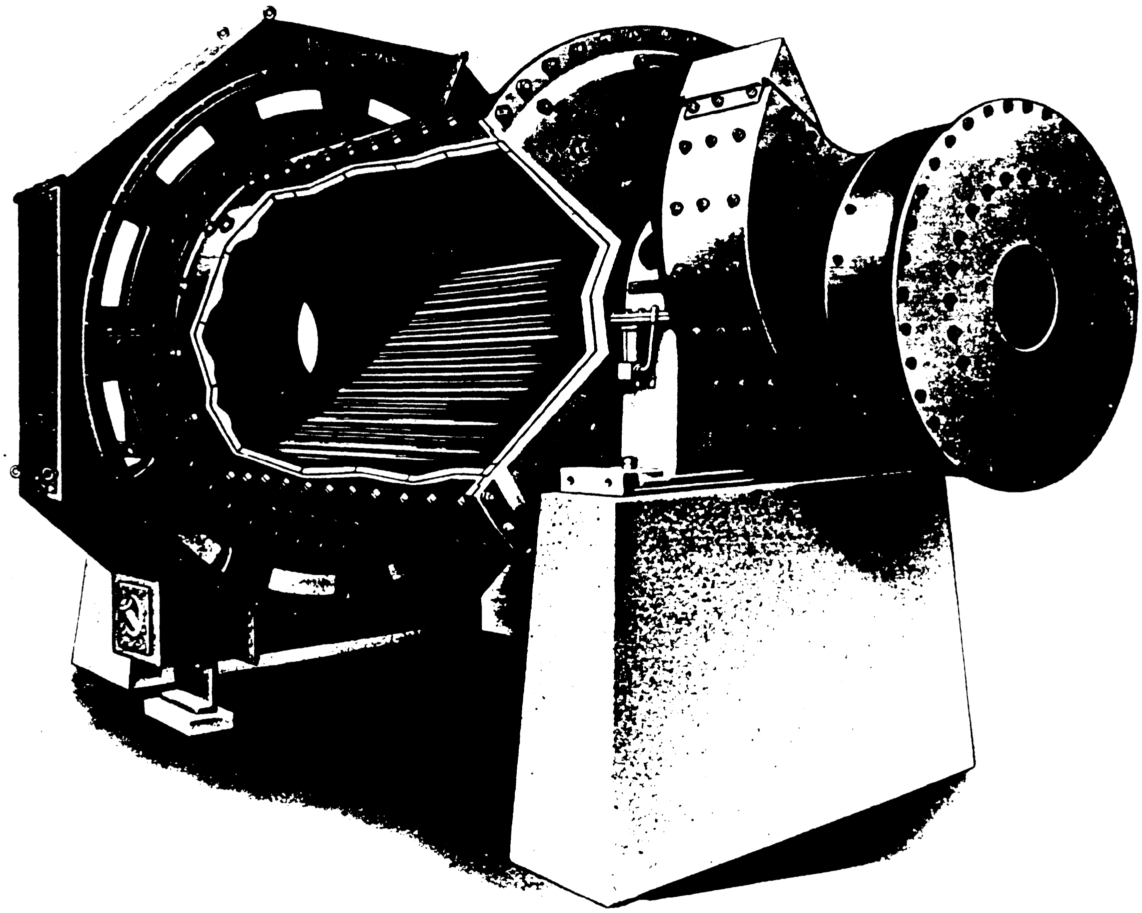

Fig. 25.3.2.1. Cutaway view of rod mill (*courtesy:* Allis Chalmers).

tally can account for up to 40% of the total plant direct operating cost.

This segment basically describes wet grinding of mineral-bearing ores by autogenous and semi-autogenous mills, rod mills, ball mills, and tower mills.

25.3.2.1 Mill Types

Mills are identified by the type of grinding media. Each type of mill is adapted to certain grinding applications that must be evaluated to select the optimum mill for each grinding task. Common mill types and their characteristics are discussed in the following.

ROD MILLS. Rod mills are horizontal, cylindrical, rotating shells loaded with rods that grind the ore by tumbling within the shell. Fig. 25.3.2.1 shows a cutaway view of an overflow rod mill. To prevent the conditions leading to rod-charge tangling, the length-to-diameter ratio is maintained at 1.4 to 1.6. Where this ratio becomes less than 1.25, the risk of tangling increases rapidly. The practical limit of a good quality rod (i.e., a rod that will not tangle) is 6.1 m (20 ft). Because of this limitation, the largest rod mill size is 4.6 m diameter by 6.6 m long (15 by 21.5 ft), using 6.1-m (20-ft) rods and drawing from 1640 to 1715 kW (2200 to 2300 hp).

Rod mills accept feed up to about 50 mm (2 in.) and produce a product in the size range of – 3000 to – 270 μm (–4 to –35 mesh). Grinding action is by line contact between the rods extending the length of the mill. Rods tumble and spin in roughly parallel alignment simulating a series of roll crushers. This results in preferential grinding of coarse material and minimizes production of slimes.

Of the three main types of rod mill—overflow, end peripheral discharge, and center peripheral discharge—only the overflow mill is in common usage. Wet grinding rod mills are normally used in the mineral processing industry. Dry grinding is used in some areas; however, it is confronted with problems and should be avoided except where absolutely necessary. The advantages and disadvantages of rod milling are as follows:

Advantages.

1. Rod mills operate at lower speed than ball mills since the rods are rolled and not cascaded.

2. For an equivalent grind, a rod mill uses less steel than a ball mill because of the lower speed and better contact between the media and ore.

Disadvantages.

1. The rod charge must be maintained in good working condition, and broken and worn rods must be removed. Rod mills usually require greater operator attention. It is important that the rods stay essentially parallel to one another. If rods become misaligned, grinding action is lost and, more importantly, rod tangles occur.

2. Maximum rod length is limited to about 6.1 m (20 ft). This in turn limits the length, diameter, and hence capacity of rod mills.

3. The heavier rods acting upon the lifters and liners result in greater maintenance requirements on the liner systems.

Table 25.3.2.1. Calculated Power Draw for Wet Overflow Rod Mills

Mill size, ft	Mill diam., ft		Length, ft		Rod weight, ton (40% charge, new rods)	Average mill speed			Mill hp at pinion shaft		(Rod length)/(inside diam.)	Optimum ratio of reduction
	Inside shell	Inside liners	Mill inside linders	Rods		Rpm	% of critical (inside diam.)	Peripheral (inside diam.), fpm	New rods	Worn in charge		
3 × 5	3.0	2.5	4.0	3.5	1.4	36.1	74.5	284	8	7	1.40	15.0
3 × 7	3.0	2.5	6.0	5.5	2.1	36.1	74.5	284	13	12	2.20	19.0
4 × 7	4.0	3.5	6.0	5.5	4.1	30.6	74.7	336	28	26	1.57	15.9
4 × 9	4.0	3.5	8.0	7.5	5.6	30.6	74.7	336	38	35	2.14	18.7
5 × 9	5.0	4.5	8.0	7.5	9.3	25.7	71.1	363	65	60	1.67	16.3
5 × 11	5.0	4.5	10.0	9.5	11.8	25.7	71.1	363	82	76	2.11	18.6
6 × 11	6.0	5.5	10.0	9.5	17.6	22.4	68.5	387	126	117	1.72	16.6
6 × 13	6.0	5.5	12.0	11.5	21.3	22.4	68.5	387	153	141	2.09	18.5
7 × 12	7.0	6.5	11.0	10.5	27.2	20.8	69.2	424	208	192	1.62	16.1
7 × 14	7.0	6.5	13.0	12.5	32.4	20.8	69.2	424	248	229	1.92	17.6
8 × 11	8.0	7.5	10.0	9.5	32.8	19.4	69.3	457	264	243	1.27	14.3
8 × 13	8.0	7.5	12.0	11.5	39.6	19.4	69.3	457	319	295	1.53	15.7
8½ × 13	8.5	8.0	12.0	11.5	45.1	19.0	70.0	477	375	347	1.44	15.2
9 × 13	9.0	8.5	12.0	11.5	50.9	17.9	68.1	478	420	388	1.35	14.8
9½ × 13	9.5	9.0	12.0	11.5	57.1	17.4	68.1	492	480	443	1.28	14.4
9½ × 15	9.5	9.0	12.0	11.5	67.0	17.4	68.1	492	564	520	1.28	15.5
10 × 14	10.0	9.5	13.0	12.5	69.1	16.5	66.4	492	577	533	1.32	14.6
10 × 16	10.0	9.5	15.0	14.5	81.2	16.5	66.4	492	669	618	1.53	15.6
10½ × 14	10.5	10.0	13.0	12.5	76.6	16.1	66.4	505	651	601	1.25	14.2
10½ × 16	10.5	10.0	15.0	14.5	85.8	16.1	66.4	506	755	647	1.45	15.2
11 × 15	11.0	10.5	14.0	13.5	91.2	15.8	66.8	521	792	731	1.29	14.4
11 × 17	11.0	10.5	16.0	15.5	104.7	15.8	66.8	521	909	839	1.48	15.4
11½ × 17	11.5	11.0	16.0	15.5	114.9	15.5	67.0	536	1018	939	1.41	15.0
12 × 17	12.0	11.5	16.0	15.5	125.5	15.1	66.8	546	1125	1038	1.35	14.7
12 × 19	12.0	11.5	18.0	17.5	141.8	15.1	66.8	546	1270	1172	1.52	15.6
12½ × 17	12.5	12.0	16.0	15.5	136.7	14.8	66.9	558	1243	1143	1.29	14.4
12½ × 19	12.5	12.0	18.0	17.5	154.4	14.8	66.9	558	1404	1296	1.41	15.3
13 × 18	13.0	12.5	17.0	16.5	157.9	14.3	66.0	562	1436	1325	1.32	14.6
13 × 21	13.0	12.5	20.0	19.5	186.7	14.3	66.0	562	1697	1566	1.56	15.8
13½ × 19	13.5	13.0	18.0	17.5	181.2	14.0	65.9	572	1666	1538	1.35	14.7
13½ × 21	13.5	13.0	20.0	19.5	201.9	14.0	65.9	572	1856	1714	1.50	15.5
14 × 19	14.0	13.5	18.0	17.5	195.4	13.6	65.2	577	1801	1663	1.30	14.5
14 × 21	14.0	13.5	20.0	19.5	217.7	13.6	65.2	577	2007	1853	1.44	15.2
14½ × 21	14.5	14.0	20.0	19.5	234.2	13.3	64.9	585	2176	2008	1.39	15.0
15 × 21	15.0	14.5	20.0	19.5	251.2	13.0	64.6	592	2349	2168	1.34	14.7

Conversion factors: 1 ft = 0.3048 m, 1 ton = 0.9072 t, 1 hp = 0.7457 kW.

Rod mills normally carry 35 to 65% rod charge by volume. The limits on charge level are (1) keeping the feed end trunnion open so that feed will get into the mill, and (2) keeping the rod charge low so rods will not work their way into discharge openings where they can cause rod tangling.

The following expression is used to determine the power draw by a rod mill (English units):

$$kW_r = 1.07\ D^{1/3}\ (6.3 - 5.4\ V_p)\ fC_s \qquad (25.3.2.1)$$

where kW_r is kilowatts per short ton (2000 lb) of rods, D is mill diameter inside liners in ft, V_P is fraction of mill volume loaded with rods, and fC_s is fraction of critical speed.

Table 25.3.2.1 lists many sizes of rod mills, giving speed, loading, and power data.

BALL MILLS. Ball mills are used following rod mills for grinding the ores to a finer size. They are also used as primary mills following fine crushing of ores. Ball mills accept feed in the size range 6.4 to – 25 mm (¼ to – 1 in.). The product size range can vary between – 270 to – 37 µm (– 35 to – 400 mesh). The ore is normally ground with cast or forged steel balls. Grinding action is by point-to-point contact between balls and shell liners and between the balls themselves. Ball mills vary in length-to-diameter ratio (L/D) ranging from less than 1:1 to greater than 2:1. There are no fixed rules on proper L/D ratios because these vary with the circuits used, ore type, feed size, and overall grinding requirement. Typically, a short ball mill has a lower retention time; consequently, it produces a coarser product. A long ball mill, sometimes called a *tube mill,* has a higher retention time and produces a finer product with less tramp oversize.

There are two common types of ball mills: grate discharge and overflow discharge.

Grate Discharge Ball Mills—Grate discharge mills are normally used when minimum fine production is needed. As ball mill sizes have increased over the years, overflow mills have become predominant, and grate discharge mills are rarely selected today.

Overflow Ball Mills—The overflow ball mill is illustrated in Fig. 25.3.2.2. As the name implies, in an overflow mill, the slurry discharges from the mill by overflowing the discharge trunnion. Over the years, the size of ball mills has increased steadily as the size of concentrators has increased significantly. The largest mills manufactured to date are by Rauma-Repola of Finland for a

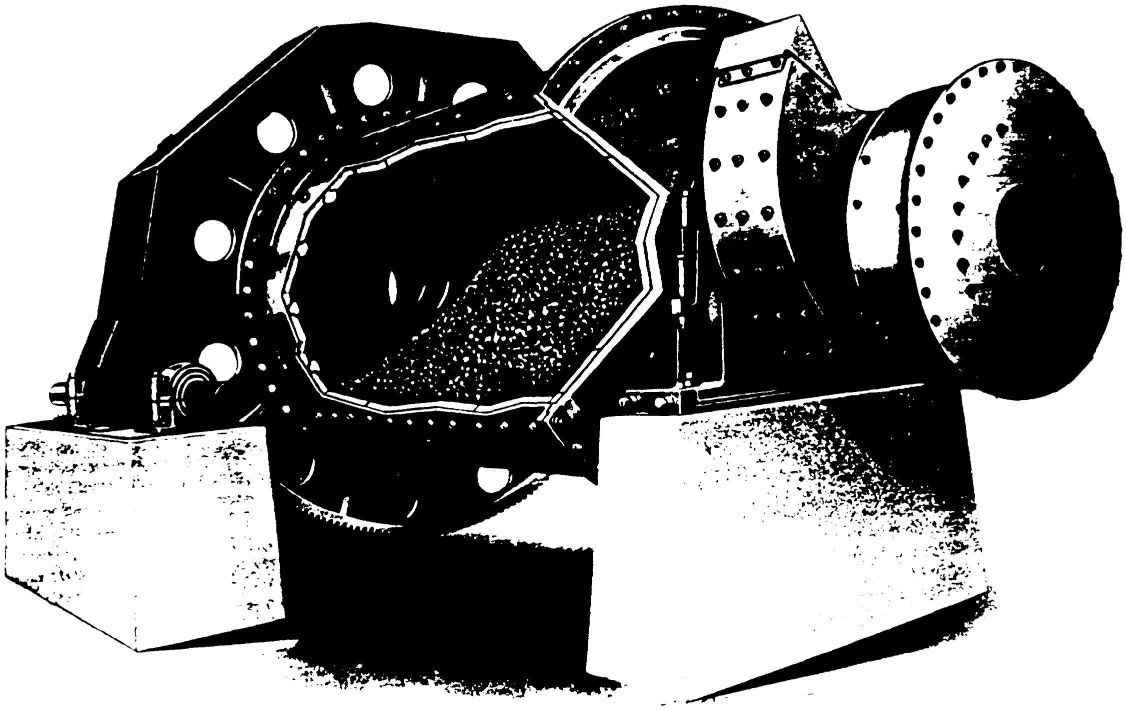

Fig. 25.3.2.2. Cutaway view of overflow ball mill (*courtesy:* Allis Chalmers).

Russian client. The mills are 6.4 m (21 ft) in diameter by 9.6 m (31.5 ft) in length, and each mill is driven by a 9620-kW (12,900-hp) wrap-around motor in which the drum carries the rotor windings that rotate within the stator mounted around the outside of the mill. Ball mills up to 5.5 m (18 ft) in diameter by 7.3 m (24 ft) in length with 4470-kW (6000-hp) motors and conventional drives are currently operating.

PEBBLE MILLS. Pebble mills are similar to ball mills but use nonmetallic grinding media such as porcelain balls, flint pebbles, and most often the ore itself as the grinding media. The bulk density of a pebble charge is less than the bulk density of the steel charge of a ball mill; consequently, for a comparable power draw, the size of a pebble mill is larger than a ball mill. Pebble mills are often used as dry mills and where iron contamination is detrimental to product quality.

Wet closed-circuit pebble mills are gaining acceptance as secondary autogenous mills. Even though they are not popular in North America, they have been widely adopted in South Africa.

In the metal mining industry, pebble mills follow primary autogenous or semi-autogenous primary grinding. Sized pebbles (ore) are either screened out of the ore stream in a crushing plant or extracted from the primary autogenous mill and fed to the pebble mill in a controlled manner to maintain constant power draw. For more detail on pebble mill applications, readers are referred to Crocker (1985).

AUTOGENOUS MILLS. Autogenous mills use large pieces of either run-of-mine or primary crushed ore as grinding media. This is different from the pebble mill which uses sized or graded rocks as grinding media. The predominant grinding mechanism in autogenous grinding is attrition with limited grinding by impact. For an ore to successfully grind autogenously, the ore must be competent, and it must break along grain boundaries at the desired product size. Another requirement is that the finer sizes should break easily and should be removed from the mill; otherwise, there will be a critical size buildup.

Critical size is the size that is too small to serve as effective grinding media and is yet too large to be broken down by large pieces. This size ordinarily ranges from 51 to 19 mm (2 to 3/4 in.) in diameter. If ore is hard to break, it will accumulate in the mill charge and decrease mill throughput.

Autogenous grinding has two advantages: it reduces metal wear and eliminates secondary and tertiary crushing stages. Thus it offers a savings in capital and operating costs.

Autogenous mills are available for both wet and dry grinding. The development of the dry-grinding AeroFall mill in the 1950s triggered the later North American development of wet grinding mills that now dominate the metal mining industry. Wet autogenous mills had considerable success in grinding hard, competent, and heavy iron ores in the 1960s, but as the application of autogenous grinding has widened, semi-autogenous grinding has evolved as the dominant technique, particularly on copper ores. The diameter of North American autogenous mills is normally two to three times the length, while South African and some European mills have approximately 1:1 diameter-to-length ratio. The ore charge is usually 25 to 35% of the mill volume.

Autogenous mills have grate discharges to retain the coarse grinding media in the mill.

An autogenous mill is shown in Fig. 25.3.2.3.

SEMI-AUTOGENOUS MILLS. Sometimes the ore may not have competent rocks to act as grinding media, or the ore body occasionally may have poor quality rocks that are not suitable grinding media. In such situations, *semi-autogenous mills* may be used.

In semi-autogenous grinding, the competent rocks are supplemented with steel balls in an amount of 2 to 10% of the total mill volume. The mill geometry of the semi-autogenous mill is the same as the autogenous mill with modifications in the structural design to account for the increased weight of media, pull, and impact. Ball sizes are typically from 76 to 127 mm (3 to 5 in.) in diameter.

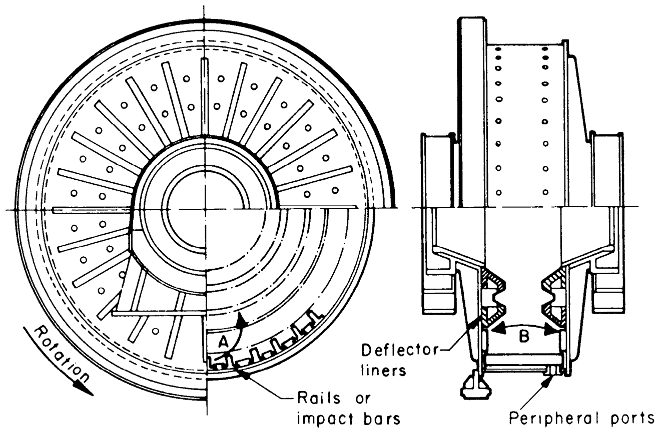

Fig. 25.3.2.3. Cross section of AeroFall mill (*courtesy:* AeroFall Mills, Ltd.).

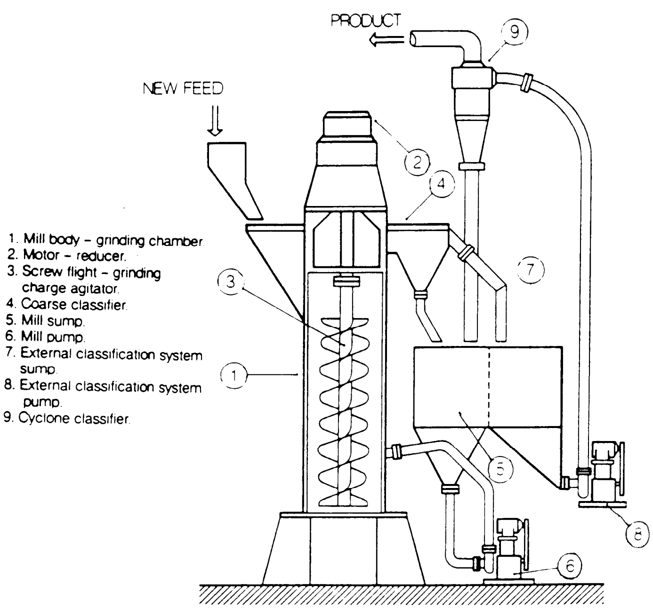

Fig. 25.3.2.4. Typical tower mill wet grinding system.

TOWER MILLS. The tower mill, invented in Japan, was originally used for fine regrinding. As the tower mill has become more accepted in North America, the range of application has expanded, and now tower mills are used in place of ball mills in low-tonnage secondary grinding circuits.

The tower mill illustrated in Fig. 25.3.2.4 consists of a vertical grinding vessel with a central shaft screw-shaped impeller. The screw impeller rotates and moves the grinding media up through the central portion of the grinding compartment and down through the outer portion. Grinding media such as steel balls, ceramic pebbles, or natural pebbles are charged into the grinding chamber and cover the screw flight agitator. Material to be ground is fed into the top, center, or bottom of the main body depending on the application and size of the feed. Grinding occurs by a combination of impingement and rubbing among the grinding balls and grinding balls against the screw and mill liners.

Tower mills are used both as dry- and wet-type mills. Dry grinding systems use air classifiers and fans, and wet grinding systems use a cyclone classifier to remove the product.

Advantages claimed by the tower mill manufacturers are a smaller area of installation, less noise (usually below 85 dB), simplicity of foundation, lower installation cost, and lower operating cost. Concurrent grinding and leaching can also be accommodated.

OTHER GRINDING MACHINES. Other, less-used grinding mills include roller mill, fluid energy mill, cage mill, vibratory ball mill, and vibra energy mill. For details of these equipment, readers are referred to the *SME Mineral Processing Handbook* (Weiss, 1985).

25.3.2.2 Mill Sizing and Selection

Rod mills and ball mills are sized based on the Bond work index obtained from standard Bond ball mill or rod mill grindability tests that are described elsewhere (e.g., see 25.3.1.5). It is much more difficult to size autogenous mills since there are no small-scale tests, such as the Bond work index tests, that adequately define the autogenous grinding characteristics of an ore. Autogenous grinding is also much more sensitive to change in ore type than conventional mills such as rod and ball mills.

It is therefore imperative that as a first step, a competency test is made on different types of ore from the same ore body. If the rocks are competent, it should be followed with a pilot plant test. A 1.5- or 1.8-m (5- or 6-ft) diameter mill is normally required to provide sufficient data to confidently size an autogenous or semi-autogenous mill.

BASIC POWER CALCULATION FOR ROD AND BALL MILLS. Once the work index (W_i) has been determined, the power requirement and mill size can be calculated from Bond's Third Theory equation as follows, rearranged from Eq. 25.3.1.1:

$$W = (10\ W_i/\sqrt{P_{80}}) - (10\ W_i/\sqrt{F_{80}}) \qquad (25.3.2.2)$$

where W is power consumption in kWh/ton of feed, P_{80} is size in μm that 80% of the product passes, and F_{80} is size in μm that 80% of the feed passes.

This calculation gives the power required at the following standard reference points:

1. Rod milling: wet open-circuit grinding in a 2.4-m (8-ft) diameter (inside liners) rod mill.

2. Ball milling: wet closed-circuit grinding in a 2.4-m (8-ft) diameter (inside liners) ball mill.

3. The calculated power is the power required at the pinion shaft of the mill and includes mill trunnion bearing and gear and pinion losses, but does not include motor losses or losses in any other drive component such as reducers and clutches.

CORRECTION FACTORS. In studying plant mill performance, a number of factors have been found that cause the work indices calculated from plant operating data to vary from the work indices obtained from Bond grindability tests. These factors, basically efficiency and inefficiency factors resulting from equipment size, grinding media, and circuit used, are the following:

CF_1 Dry grinding.
CF_2 Open-circuit ball milling.
CF_3 Mill diameter.
CF_4 Oversized feed.
CF_5 Grinds finer than 80% passing 74 μm (200 mesh).
CF_6 Too small or too large reduction ratio, rod mill.
CF_7 Too small a reduction ratio, ball mill.

The basic power W calculated in Eq. 25.3.2.2 should therefore be adjusted in accordance with the correction factors (CF_1 to CF_7) as follows:

CORRECTION FACTOR l—CF_1 (dry grinding) = 1.3.

Table 25.3.2.2. Open-circuit Ball Milling Correction Factor

Reference % Passing	Correction Factor 2 (CF_2)
50	1.035
60	1.05
70	1.10
80	1.20
90	1.40
92	1.46
95	1.57
98	1.70

Table 25.3.2.3. Material Size Conversion For 80% Passing Feed or Product Size

	80% Passing Equivalent	
Material size	μm (F) or (P)	$\sqrt{F}$ or $\sqrt{P}$
99%—1½ in.	25,000	158.0
99%—1 in.	18,000	134.0
99%—¾ in.	12,000	109.5
99%—½ in.	8,500	92.2
99%—⅜ in.	6,000	77.4
99%—3 mesh	4,200	64.8
99%—4 mesh	3,000	54.8
99%—6 mesh	2,100	45.8
99%—8 mesh	1,500	38.7
99%—10 mesh	1,000	31.6
99%—14 mesh	800	28.3
99%—20 mesh	550	23.4
99%—28 mesh	400	20.0
99%—35 mesh	270	15.4
99%—48 mesh	150	12.25
99%—65 mesh	105	10.25
99%—100 mesh	72	8.48
99%—150 mesh	55	7.42
99%—200 mesh	36	6.00
99%—325 mesh	20	4.47

CORRECTION FACTOR 2—CF_2 (open-circuit ball milling); see values in Table 25.3.2.2.

CORRECTION FACTOR 3—CF_3 (diameter) $CF_3 = (8/D_1)^{0.2}$ (or 0.914 minimum value) where D_1 is the mill diameter inside liners. The gain in grinding efficiency with increasing mill diameter ends at about 3.8 m (12.5 ft) inside the liners. The use of the diameter efficiency factor beyond this point does not give the correlation between W_{io} and W_i. The diameter efficiency factor for diameters larger than 3.8 m (12.5 ft) inside the liners should be the same as used for mills 3.8 m (12.5 ft) inside the liners; namely, 0.914.

CORRECTION FACTOR 4—CF_4 (oversized feed).

Optimum Feed Size F_o.

Ball Mill:

$F_o = 4000 \times \sqrt{(13/W_i)}$

If $F_{80} > F_o$, then

$CF_4 = [R_r + (W_i - 7) \times (F_{80} - F_o)/F_o]/R_r$

where $R_r = F_{80}/P_{80}$.

Rod Mill:

$F_o = 16{,}000 \times \sqrt{(13/W_i)}$

If $F_{80} > F_o$, then $CF_4 = 1.000$.

CORRECTION FACTOR 5—CF_5 (fineness of grind).

If $P_{80} < 74$ μm (200 mesh), then $CF_5 = (P_{80} + 10.3) / (1.145 \times P_{80})$

If $P > 74$ μm (200 mesh), then $CF_5 = 1.000$.

CORRECTION FACTOR 6—CF_6 (ratio of reduction, rod milling).

Optimum Reduction Ratio: $R_{r_o} = 8 + 5L/D_1$

where L = mill length, inside liners.

Reduction Ratio: $R_r = F_{80}/P_{80}$

$CF_6 = 1 + (R_r - R_{r_o})^2/150.$

CORRECTION FACTOR 7—CF_7 (low ratio of reduction, ball milling).

If $R_r < 3$, then

$CF_7 = [20\ (R_r - 1.35) + 2.6] / [20\ (R_r - 1.35)]$

If $R_r \geq 3$, then $CF_7 = 1.000$.

An example of the application of correction factors is as follows.

Example 25.3.2.1. Assume a ball mill operating in closed circuit. Work index is 13.0, feed size is 95–99% – 12.5 mm (½ in.), product size is 75% –74 μm (200 mesh), feed rate is 54.5 t/h (60 tph).

Solution. First, convert the feed and product sizes to 80% passing sizes. From Table 25.3.2.3, material size conversion, select 8500 as the F_{80} size for 99% passing 12.5 mm (½ in.). From the product size 75% passing 74 μm (200 mesh), the P_{80} size can be determined from a distribution curve or approximated from the Gates-Gaudin-Schumann relationship as follows:

Size 2 = (% passing size 2/% passing size 1)^2 × size 1

$P_{80} = (80/75)^2 \times 74$ μm $= 84$ μm

Inserting the 80% passing sizes and work index in Bond's formula gives:

$W = (10W_i/\sqrt{P_{80}}) - (10W_i/\sqrt{F_{80}})$

$W = (10 \times 13)/\sqrt{84} - (10 \times 13)\ \sqrt{8500} = 12.8$ kWh/t

The power required is:

(12.8 kWh/t) × (60 tph) = 770 kW (1030 hp)

From the mill size tables (25.3.2.1) in this segment, the horsepower calculated for an overflow ball mill corresponds to a 3.5 m (11.5 ft) in diameter by 4.6 m (15 ft) in length mill. The diameter inside the shell is 3.5 m (11.5 ft), and the diameter inside the liners D_1 is 3.4 m (11 ft).

The basic power W is corrected as follows:

In this example, correction factors CF_1 and CF_2 are not applicable.

CORRECTION FACTOR CF_3: $(8/11)^{0.2} = 0.938$

CORRECTION FACTOR CF_4:

$F_o = 4000 \times \sqrt{(13/W_i)} = 4000 \times \sqrt{(13/13)} = 4000$

$F_{80} = 8500$

since $F_{80} > F_o$, reduction ratio $R_r = \dfrac{8500}{84} = 101.2$

therefore, $CF_4 = [101.2 + (13 - 7) \times (8500 - 4000)/4000] / 101.2 = 1.067$

CORRECTION FACTOR CF_5:

Since $P_{80} = 84$ μm and $P_{80} > 74$ μm, $CF_5 = 1$

CORRECTION FACTOR CF_6 is not applicable.

CORRECTION FACTOR CF_7 is not applicable.

The final calculated horsepower is obtained by multiplying the base horsepower by each applicable correction factor:

$$(770) \times (0.938) \times (1.067) \times (1) = 771 \text{ kW } (1031 \text{ hp})$$

In this instance, there is not much difference from the uncorrected value. The mill size corresponding to the calculated horsepower is therefore the same, 3.5 m (11.5 ft) in diameter by 4.6 m (15 ft) in length.

POWER CALCULATION FOR AUTOGENOUS AND SEMI-AUTOGENOUS MILLS. It has been found that as a general rule, power requirements to grind a tonne (ton) of ore in a test mill to a given size can usually be duplicated in a large commercial mill. This assumes that the test work is carried out under carefully controlled conditions with good power measurements. It is generally found that in the small test mill, the load should be maintained between 25 and 30% of mill volume. The total power drawn under these conditions can be used for scale up to larger mills and to obtain unit power requirements per tonne (ton) of ore treated.

Various researchers and manufacturers have used many formulas for arriving at the total power requirements of a commercial mill. Many of the early wet autogenous mills were underpowered and had to operate at lower load levels, particularly if a ball charge had to be added to obtain the throughput. Such a situation increases both the liner and ball wear. It is therefore essential to have enough installed power on the commercial mill.

In general, most scale-up formulas use an exponent of the mill diameters and a direct comparison of mill lengths to arrive at a multiplying factor for the scale-up of the test mill power. The exponent used for mill diameters has varied from 2.4 to 3.0.

For wet mills, an exponent of 2.65 should be used to arrive at the power to drive the commercial mill provided that good power measurements have been taken in the test mill at a known charge level between 25 and 30%. The power on the test mill is always given in new kW, and the power on the commercial mill is generally measured as gross power. Therefore, using the relationship (in SI units):

$$P_C = P_T \times (L_C/L_T) \times [(D_C/D_T)^{2.65}]/0.95 \qquad (25.3.2.3)$$

should give the required power of the mill motor on the commercial mill in which P_C is gross power of the commercial mill in kW, P_T is net power of the test mill in kW, D_C is nominal diameter of the commercial mill in meters, D_T is nominal diameter of the test mill in meters, L_C is nominal length of the commercial mill in meters, and L_T is nominal length of the test mill in meters. Gross power is the power input to the mill motor, and net power is the power actually consumed in grinding after deducting mechanical and electrical losses.

This equation has been used to check the actual power obtained on commercial mills against the calculated figures. These cases were taken from actual test work where it is known that the work was carefully carried out and correctly supervised. In most cases the calculated figure from test work is generally conservative or safe for the design of commercial power requirements.

To find the nominal capacity of the mill in tonnes (tons) per hour, 90% of the gross power should be divided by the net grinding power per tonne (ton) obtained in the test mill.

Analyses of the data from operating dry AeroFall mills shows that mill power varies as the diameter raised to the 2.8 power and as the effective length. Specifically, the power drawn by mills of standard design treating normal ores under optimized conditions may be calculated from (in SI units):

$$P = 0.00100 \times W \times D^{2.8} \times L \qquad (25.3.2.4)$$

where P is motor input in kW for direct driven mills, W is the bulk density of the mill load in kg/m^3, D is mill diameter (inside shell liner) in m, and L is mill length inside the liners in meters.

The calculated power requirements of a commercial mill should also be compared with operating mills with corrections for differences in size and in bulk density of the mill loads.

All manufacturers have their own methods for arriving at mill power. It is therefore important to submit the test data to the manufacturer to verify that the mill will have the correct installed motor size.

In summary:

1. Take careful power measurements on the test mill with the load in mill between 25 and 30% of total mill volume.

2. Scale-up by formula to obtain mill motor size for commercial mill.

3. Compare this power requirement with mills of comparable size using correction for difference in bulk density of the mill load or the load plus ball charge.

4. Obtain nominal capacity of the mill by dividing 90% of the mill input power by the net kWh/t (ton) obtained in the test work.

5. Submit test data to mill manufacturer for verification that the commercial mill is correctly sized and powered to produce the required tonnage.

OTHER CONSIDERATIONS.

1. The Bond Third Theory also provides a simple yet accurate mathematical formula for comparing and studying grinding circuits and variables within these circuits.

The comparison of work indices is given as work index calculated from the plant operating data W_{io} divided by the work index obtained from the Bond grindability test W_i:

$$W_{io}/W_i \qquad (25.3.2.5)$$

Using the work index obtained from the grindability test as the standard for comparison can be a measure of the efficiency or inefficiency of the operating mills. Some plants do this on a regular basis.

2. The feed to the standard Bond ball mill grindability test is – 3.36 mm (–6 mesh). The coarser fraction of a – 12.5-mm (– 1/2-in.) single-stage ball mill feed is not included in the feed to the grindability ball mill. The – 12.5-mm (– 1/2-in.) feed to a standard Bond rod mill grindability test includes the coarse fraction of a single-stage ball mill feed. To obtain the complete grindability picture of an ore when selecting a single-stage ball mill, it is advisable to run both rod and ball mill grindability tests.

If there is a difference in the work indices obtained from the rod mill and the ball mill grindability tests, which frequently occurs, then, particularly if the rod mill test work index is higher, a two-step calculation should be made to determine the required grinding power. The rod mill work index should be used to calculate from the plant ball mill feed size to 80% passing 2100 μm. The calculation from 2100 to the desired product size is made using the work index from the ball mill grindability test. The sum of these two gives the total power per tonne (ton) required for grinding.

3. For rod mills, the power drawn varies as the diameter D to the 2.33 power ($D^{2.33}$), and for ball mills, the power drawn varies as the diameter to the 2.3 power ($D^{2.3}$). This is a change from the previously stated exponential factors of from 2.4 to 2.6.

4. The weight of the steel charge in a mill can be estimated as follows (in SI units):

$$C_w = (D^2/4)\ \pi\ V_p L Cd/1000 \qquad (25.3.2.6)$$

where C_w is charge weight in tonnes, D is mill diameter inside liners in m; V_p is percentage of mill volume loaded with charge; L for rod mills is rod length in m, or for ball mills, it is the effective length of grinding compartment in m; and Cd is density of the media including voids in kg/m^3.

When the actual weight per cubic meter including voids for the media is not available, the following weights including voids can be used to estimate the weight of media (charge) in the mill:

Table 25.3.2.4. Average Bond Work Indices of Materials

Material	Sp Gr	W_i	Material	Sp Gr	W_i
Andesite	2.84	22.13	Lead ore	3.44	11.40
Barite	4.28	6.24	Lead-zinc ore	3.37	11.30
Basalt	2.89	20.41	Limestone	2.69	11.61
Bauxite	2.38	9.45	Limestone for cement	2.68	10.18
Cement clinker	3.15	13.49	Manganese ore	3.74	12.46
Cement raw material	2.67	10.57	Magnesite-burned	5.22	16.80
Chrome ore	4.06	9.60	Mica	2.89	134.50
Clay	2.23	7.10	Molybdenum ore	2.70	12.97
Clay, calcined	2.32	1.43	Nickel ore	3.32	11.88
Coal	1.63	11.37	Oil shale	1.76	18.10
Coke	1.51	20.70	Phosphate fertilizer	2.65	13.03
Coke, petroleum	1.78	73.80	Phosphate rock	2.66	10.13
Copper ore	3.02	13.13	Potash ore	2.37	8.88
Coral	2.70	10.16	Potash salt	2.18	8.23
Diorite	2.78	19.40	Pumice	1.96	11.93
Dolomite	2.82	11.31	Pyrite ore	3.48	8.90
Emery	3.48	58.18	Pyrrhotite ore	4.04	9.58
Feldspar	2.59	11.67	Quartz	2.64	12.77
Ferrochrome	6.75	8.87	Quartzite	2.71	12.18
Ferromanganese	5.91	7.77	Rutile ore	2.84	12.12
Ferrosilicon	4.91	12.83	Sandstone	2.68	11.53
Flint	2.65	26.16	Shale	2.58	16.40
Fluorspar	2.98	9.76	Silica	2.71	13.53
Gabbro	2.83	18.45	Silica sand	2.65	16.46
Galena	5.39	10.19	Silicon carbide	2.73	26.17
Garnet	3.30	12.37	Silver ore	2.72	17.30
Glass	2.58	3.08	Sinter	3.00	8.77
Gneiss	2.71	20.13	Slag	2.93	15.76
Gold ore	2.86	14.83	Slag, iron blast		
Granite	2.68	14.39	furnace	2.39	12.16
Graphite	1.75	45.03	Slate	2.48	13.83
Gravel	2.70	25.17	Sodium silicate	2.10	13.00
Gypsum rock	2.69	8.16	Spodumene ore	2.75	13.70
Ilmenite	4.27	13.11	Syenite	2.73	14.90
Iron ore	3.96	15.44	Tile	2.59	15.53
Hematite	3.76	12.68	Tin ore	3.94	10.81
Hematite, specular	3.29	15.40	Titanium ore	4.23	11.88
Oolitic	3.32	11.33	Trap rock	2.85	21.10
Limonite	2.53	8.45	Uranium ore	2.70	17.93
Magnetite	3.88	10.21	Zinc ore	3.68	12.42
Taconite	3.52	14.87	Average of all ores		
Kyanite	3.23	18.87	tested	—	13.81

New rods 6250 kg/m^3 (390 lb/ft^3).

Worn-in rod charge (culled) 5770 kg/m^3 (360 lb/ft^3); (not culled) less than 5610 kg/m^3 (350 lb/ft^3) depending upon amount of broken rods in the charge.

Forged steel balls worn-in 4640/m^3 (290 lb/ft^3).

Cast iron balls worn-in 4160 kg/m^3 (260 lb/ft^3) or less, depending upon quality.

5. If a work index is not available from test work or equivalent plant operations, then the values in Table 25.3.2.4 can be used to obtain a first approximation.

GRINDING RODS. Grinding rods range in size from 38.1 to 101.6 mm (1.5 to 4 in.) in diameter and are normally 3.04 to 6.10 m (10 to 20 ft) in length, depending on the length of the mill. Their size and shape practically dictate that they be made from rolled steel bars. The rods are cut to their specified length, usually with a cut-off wheel, which produces a square cut. They are machine-straightened, usually to a tolerance of 6.35 mm (0.25 in.) maximum deviation from a straight line in 1.52 m (5 ft). Some rod users may specify a tolerance of 3.18 mm (⅛ in.) in 1.52 m (5 ft). Straight rods are easier to charge with a rod-charging machine; they are less prone to tangling of the charge and are less likely to break prematurely in service. They also do a better job of grinding.

For many years, practically all grinding rods were made from an unalloyed steel that contained 0.80 to 1.00% carbon, 0.30 to 0.90% manganese, 0.040% maximum phosphorus, and 0.050% maximum sulfur. They were used in their as-rolled condition with a hardness range of 240 to 300 Brinell. The larger-diameter rods tend to be softer than the smaller-diameter rods. AISI grade 1095 steel was usually specified. This type of steel is still extensively used for grinding rods, though it is now being replaced at a number of operations by harder and more abrasion-resistant grades of low-alloy steel (Table 25.3.2.5).

GRINDING BALLS. Grinding balls for tumbling mills range in size from 127-mm (5-in.) maximum diameter down to about 19-mm (¾-in.) minimum diameter. Currently, there is very little production of balls larger than 102-mm (4-in.) diameter, except for semi-autogenous mills. Balls smaller than 19-mm ¾-in.) diameter may be supplied on special order from the larger suppliers (Table 25.3.2.6).

GRINDING ROD AND BALL WEAR RATES. Bond (1958) had made extensive studies of the abrasiveness of various metals and minerals. By determining the work index W_i and abrasion index

Table 25.3.2.5. Grinding Rod Data

Rod Size (In.)	Wt/Ft (Lb)	Ft/Ton*	Surface Area (ft²/ton)
1½	6.0	333.3	131
2	10.8	185.2	98
2½	16.8	119.1	78
3	24.0	83.3	65
3½	32.8	61.0	56
4	42.8	46.7	49
5	67.0	29.9	39

* This column enables a quick calculation of the number of rods required. Assume 25 tons of 3½″ diameter × 19′6″ rods are required.

$$\text{No. of rods} = \frac{61 \times 25}{19.5} = 78 \text{ rods.}$$

Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 lb = 0.4536 kg, 1 ton = 0.9072 t.

Table 25.3.2.6. Grinding Ball Data

Ball Diam. (In.)	Wt. Each (Lb)	No./Ton	Surface Area (ft²/ton)
¾	0.063	31,994	392.2
1	0.148	13,497	294.1
1¼	0.290	6,911	235.3
1½	0.501	3,999	196.1
2	1.187	1,687	147.1
2½	2.318	864	117.6
3	4.006	500	98.0
3½	6.361	315	84.0
4	9.495	211	73.5
5	18.544	108	58.8

Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 lb = 0.4536 kg, 1 ton = 0.9072 t.

A_i in a specially designed laboratory machine, he was able to predict the wear of rods and balls in specific grinding operations. His relationships on rod and ball wear are as follows.

Wet rod mills:

$$\text{Rod wear in kg/kWh} = 0.16\,(A_i - 0.020)^{0.30} \qquad (25.3.2.7.)$$

Wet ball mills:

$$\text{Ball wear in kg/kWh} = 0.16\,(A_i - 0.015)^{1/3} \qquad (25.3.2.8.)$$

Dry ball mills:

$$\text{Ball wear in kg/kWh} = 0.023\,\sqrt{A_i} \qquad (25.3.2.9)$$

The foregoing equations were developed from the actual wear data supplied from a large number of grinding operations. Since there has been a continuing trend toward the use of more abrasion-resistant rods and balls since the collection of Bond's data, it will be necessary to reduce the constants in Bond's equations to bring them in line with this trend.

Heat-treated alloy-steel rods, as supplied by US manufacturers, cost 12 to 15% more than the 1095 steel rods. As is indicated in Table 25.3.2.7, 1095 steel rods wear at rates that are from 36 to 191% faster than the alloy rods. It appears, therefore, that the use of the alloy rods can be economically justified in rod-milling operations.

Table 25.3.2.7. Consumption of Grinding Rods in Various Ores

Type of ore	Rod diam., in.	Wear rates, lb/ton		Difference, %
		Alloy	High carbon	
Taconite	4	0.82	1.25	52
Ilmenite	2½	0.43	1.25	191
Lead-zinc	3½	0.83	1.15	39
Copper	4	0.65	0.90	38
Uranium	2½	0.58	0.90	55
Copper	4	0.61	0.83	36
Copper	3	0.36	0.53	47

Source: Armco Steel Corp. Data from 1095 steel rods.
Conversion factors: 1 in. = 25.4 mm, 1 lb = 0.4536 kg, 1 ton = 0.9072 t.

The consumption of balls for various grinding operations is shown in Table 25.3.2.8.

25.3.2.3 Mill Liners

Some of the more successful sections that have been used for rod mill liners are illustrated in Fig. 25.3.2.5. The choice of the particular design from the six basic types shown in this figure is influenced by numerous factors that include mill speed and diameter, size of rods used, abrasiveness and size of the mill feed, discharge level of the pulp, and toughness of the material in the liners. There is no one liner design that is best for all mills; however, the single-wave design (similar to that shown in Fig. 25.3.2.5a is currently the most popular for heavy-duty service in modern large-diameter mills. When installed, it is typically about 76.2 mm (3 in.) thick in the valleys and 152 to 178 mm (6 to 7 in.) thick at the top of the wave.

There is a fairly definite correlation between the size of balls in a mill and the design used for liners. The larger sizes of balls are used for coarse mill feeds, which tend to produce high wear rates on the liners. Consequently, the liners are usually made thicker and with higher or more waves or ribs to extend their life. In fine or secondary mill grinding with small balls, the wear rates on the liners are slower, which, in turn, permits the use of thinner liners with lower waves or fewer lifters. The smaller balls also produce less impact so that more brittle materials may be used in the liners.

Fig. 25.3.2.6 illustrates the more successful cross-sectional designs of shell liners made from cast ferrous materials. Cast liners, while still very popular, are now faced with increasing competition from rolled steel, rubber, and ceramic liners.

LINER LIFE. Consumption of steel and iron rod and ball mill linings, when expressed in pounds per ton of mill feed, may range from a high value of about 0.25 kg/t (0.5 lb/ton) for a ball mill wet grinding abrasive ore down to a low value of about 0.0005 kg/t (0.001 lb/ton) for dry grinding soft limestone or cement clinker.

Table 25.3.2.9 lists typical wear rates of liners in 15 high-tonnage milling installations in the United States and Canada that are involved in primary grinding of low-grade ores.

Since about 1961, there has been a marked growth in the use of rubber linings throughout the world. Much of this growth is due to the development of successful designs by the rubber manufacturers. The principal use of rubber linings has been in

Table 25.3.2.8. Consumption of Balls in Various Grinding Operations

Line	No. of plants	Ball mill feed	Type of grind	Ball type	Lb/ton Max	Lb/ton Min	Lb/ton Avg	Lb/kWh Max	Lb/kWh Min	Lb/kWh Avg
1	10	Crushed nonferrous ores	Wet primary	3	1.53	1.05	1.26	0.215	0.156	0.175
2	11	Rod-milled nonferrous ores	Wet primary	3	1.20	0.17	0.88	0.202	0.163	0.181
3	5	Rod-milled taconites†	Wet primary	3	0.95	0.50	0.71	0.123	0.077	0.101
4	15	Crushed raw cement rock‡	Wet	1,2,3,5,6	3.00	0.14	1.03	—	—	0.042
5	17	Crushed raw cement rock‡	Dry	1,2,3,5,6	0.30	0.03	0.12	—	—	0.006
6	35	Cement‡	Dry finish	1,2,3,5,6	1.23	0.09	0.42	0.032	0.002	0.011
7	48	Cement—USA	Dry finish	1,2,3	0.64	0.04	0.26	0.019	0.001	0.007
8	2	Hematite concentrate	Dry fine	3,6	2.00	0.90	1.45	0.071	0.069	0.070
9	1	Crushed phosphate rock	Dry	3	—	—	0.10	—	—	—
10	1	Crushed phosphate rock	Wet	3	—	—	0.40	—	—	—
11	2	Coal	Dry	3	0.20	0.05	0.12	—	0.003	—
12	1	Silica sand	Wet	3	—	—	2.20	—	—	—

† Wear per ton of rod-mill feed in two-stage grind. Part of silica is removed between the rod mill and ball mills.
‡ From private survey of 35 cement plants in various countries. The kW for raw cement rock includes that used in feed preparation.
Source: Pit and Quarry, July 1973. Conversion factors: 1 lb/ton = 0.5000 kg/t, 1 hp = 0.7457 kW.

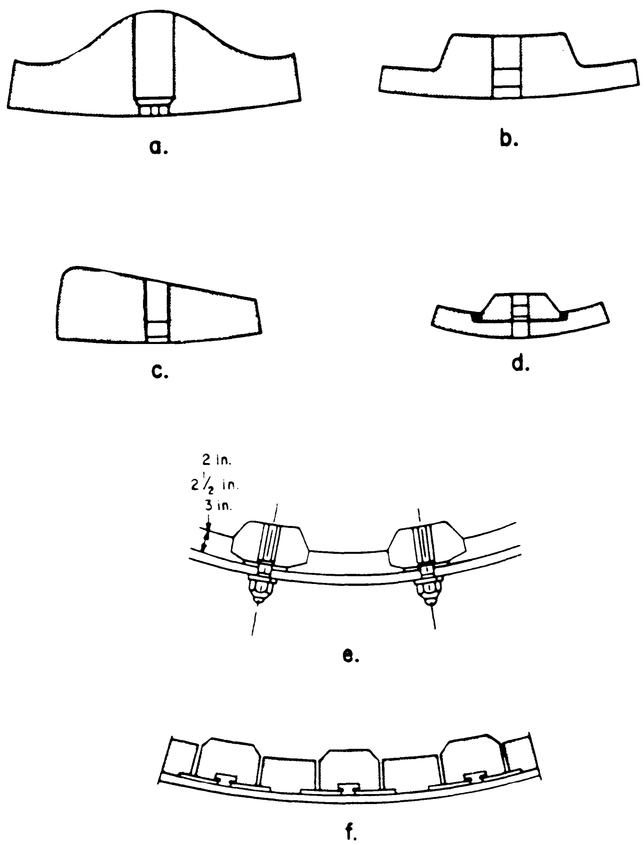

Fig. 25.3.2.5. Rod-mill shell liners, typical cross sections. Conversion factor: 1 in. = 25.4 mm.

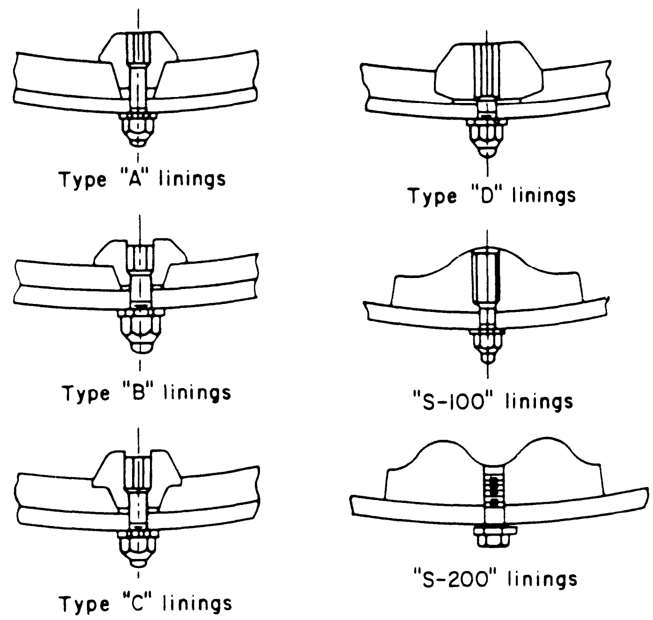

Fig. 25.3.2.6. Rolled-steel shell liners, typical cross sections (*courtesy:* USX Corp.).

fine-grinding operations, where they are capable of providing excellent wear life. The liners generally involve the use of molded plates wedged between molded lifter bars, as shown in Fig. 25.3.2.7.

25.3.2.4 Mill Drives

Many different types of drives can be used on grinding mills, depending on the size of the mill and torque requirements. Drive types range from V-belts to multi-pinion to gearless drives. Only the most popular types are discussed here.

RING GEAR AND SINGLE PINION. Fig. 25.3.2.8 shows several pinion mill drives. The single pinion (Fig. 25.3.2.8a and b) is by far the most popular drive arrangement because of its simplicity. The pinion can be driven by a slow-speed motor (approximately 150 to 250 rpm) or a combination of a higher-speed motor (600 to 1200 rpm) and speed reducer. This type of drive may use single-helical, double-helical, or spur gears.

This type of drive is commonly used on mills up to 2980 kW (4000 hp) and in some cases can be used on mills up to 3730 kW (5000 hp). Flexible couplings or air clutches are used to connect the drive motor and/or speed reducer to the pinion shaft. Use of the clutch reduces the starting torque requirements of the motor and the power demand on the electrical system.

RING GEAR AND MULTI-PINION DRIVES. Fig. 25.3.2.8 also shows dual pinion drives. Currently, this arrangement is the

Table 25.3.2.9. Typical Wear Rates of Mill Linings in Primary Grinding of Various Ores

Plant No.	Mill feed, in.	Unit size, ft	Liner wear, × lb/ton Rod mills	Liner wear, × lb/ton Ball mills	Liner wear, × lb/ton Total	Liner wear, lb/kWh Rod mills	Liner wear, lb/kWh Ball mills	Liner wear, lb/kWh Combined
Single Stage Ball Mills								
1	Copper ore, −¾	10 × 10 grated	—	0.080	—	—	0.0107	—
2	Copper ore, −½	12 × 12 grated	—	0.084	—	—	0.0095	—
3	Copper ore, −½	10½ × 11 o'flow	—	0.090	—	—	0.0110	—
4	Copper ore, −¾	16½ × 19 o'flow	—	0.095	—	—	0.0102	—
5	Molybdenum ore, −⅜	9 × 9 grated	—	0.095	—		0.0176	—
6	Molybdenum ore, −⅜	13 × 12 o'flow	—	0.101	—	—	0.0180	—
7	Copper ore, −½	10½ × 12 grated	—	0.121	—	—	0.0171	—
8	Molybdenum ore, −⅜	13 × 12 o'flow	—	0.150	—	—	0.0233	—
Double Stage, Rod and Ball Mills								
9	Taconite iron ore, −¾	1-RM-10½ × 15 2-BM-10½ × 15	0.040	0.030*	0.070*	0.0133	0.0042	0.0069
10	Taconite iron ore, −¾	1-RM-10½ × 16 2-BM-10½ × 14	0.042	0.031*	0.073*	0.0140	0.0052	0.0081
11	Copper ore, −¾	1-RM-12½ × 16 2-BM-12½ × 14	0.057	0.031	0.088	0.0190	0.0044	0.0088
12	Copper ore, −¾	1-RM-10½ × 15 2-BM-10½ × 15	0.067	0.027	0.094	0.0264	0.0052	0.0117
13	Copper ore, −¾	1-RM-12½ × 16 2-BM-12½ × 14	0.062	0.035	0.097	0.0177	0.0054	0.0097
14	Molybdenum ore, −¾	1-RM-12½ × 15 1-BM-12½ × 15	0.052	0.055	0.107	0.0148	0.0127	0.0136
15	Copper ore, −½	1-RM-10 × 16 2-BM-10½ × 13	0.100	0.046	0.146	0.0250	0.0077	0.0146
16	Taconite iron ore, −1	1-RM-11½ × 14 1-BM-11½ × 14	0.120	0.033*	0.153*	0.0257	0.0046	0.0129

* Per ton of feed to rod mills. Part of silica removed between rod mills and ball mills.
Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 lb/ton = 0.5000 kg/t, 1 hp = 0.7457 kW.

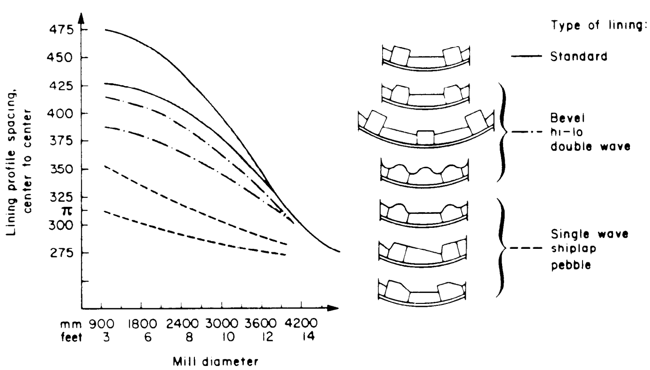

Fig. 25.3.2.7. Shell-liner profiles and spacings for rubber liners.

most popular for drives 3730 to 8950 kW (5000 to 12,000 hp). As in the case of the single pinion drive, slow- or high-speed motors can be used, as well as air clutches or flexible couplings.

The major problem with this type of drive is maintaining the load division between the pinions. The load division can be accomplished by either mechanical or electrical means, or a combination of both. The most popular and successful method to date has been electrical load division.

GEARLESS DRIVE. Fig. 25.3.2.9 shows the gearless or "wrap-around" drive. This type of drive consists of an ultra-low-speed synchronous motor running at mill speed, driving the mill without the use of gears. The motor can be "wrapped" around the mill with the rotor mounted on the mill shell or mill trunnion. Alternatively, it may be independently mounted and directly connected to the mill trunnion.

This type of drive offers variable speed with either constant torque or constant horsepower. In general, gearless drive becomes economical somewhere between 11,200 to 14,900 kW (15,000 to 20,000 hp).

25.3.2.5 Grinding Circuits

The grinding circuit reduces the feed material to a size needed for further processing. Both wet and dry grinding are used. In dry grinding, the moisture content is generally kept below 5%. If subsequent processing is a dry process, then dry grinding is favored. Wet grinding requires less power for a given grind and is typically chosen if there are no special circumstances.

Open- and Closed-circuit Grinding—Grinding may be done either by open or closed circuit. In open circuit, grinding material makes one pass through the mill and is discharged to subsequent processing. In closed-circuit grinding, each pass through the mill is followed by classification. The coarse material is returned to the mill for additional grinding while the fine material is discharged to subsequent processing.

Open-circuit Grinding—Open-circuit grinding should be considered if the following apply:

1. There is another stage of grinding that follows.
2. The reduction ratio is small.
3. The size gradient is not critical. Some undersize and oversize can be tolerated.

Closed-circuit Grinding—Closed-circuit grinding produces a closer-sized-product range than open-circuit grinding. Mill discharge in a closed circuit is separated by a classifier into coarse and fine material. The coarse material recycles to the mill, and the fine material discharges to the next processing stage. The size split is controlled by the classifier.

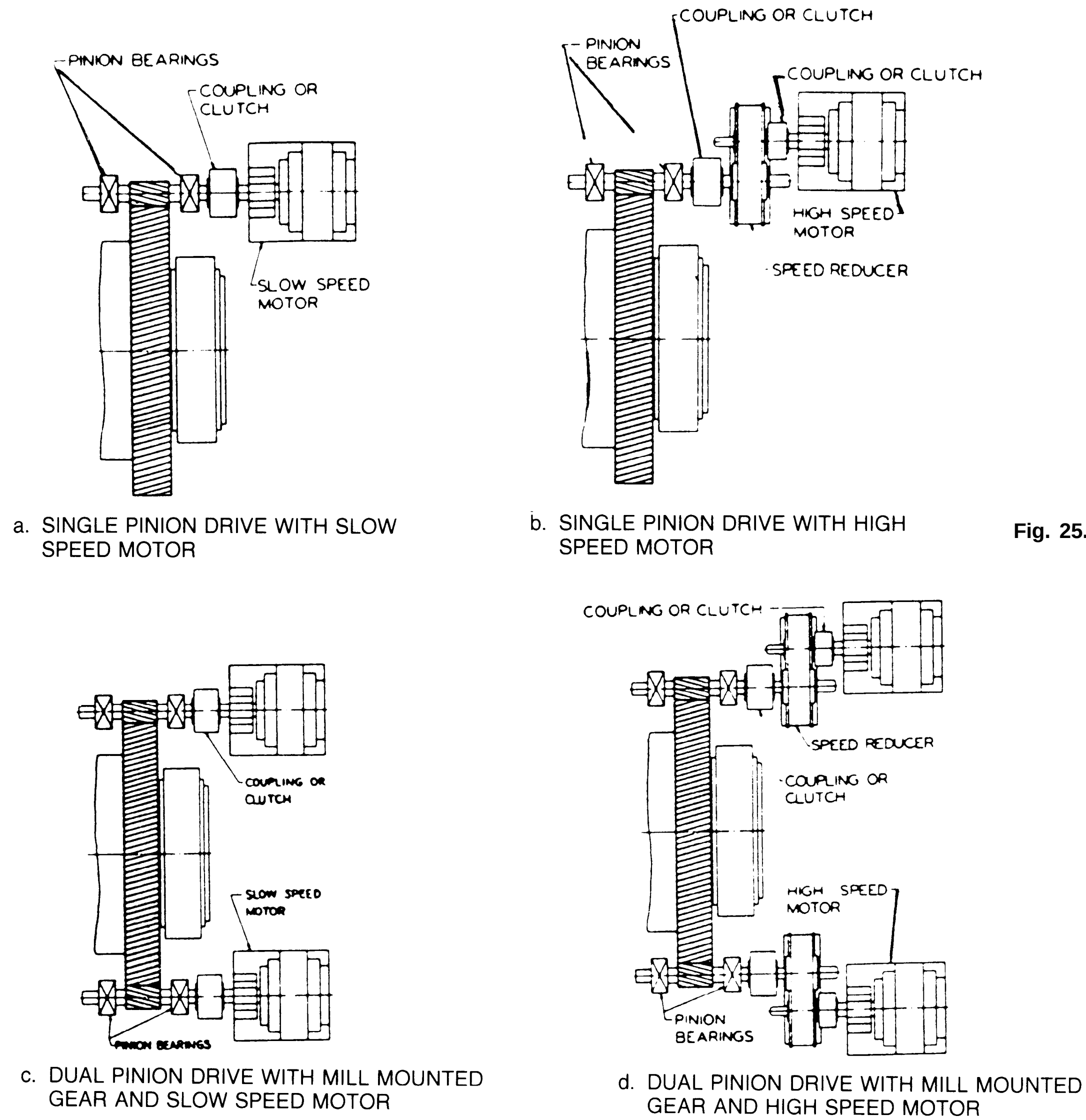

Fig. 25.3.2.8. Pinion mill drives.

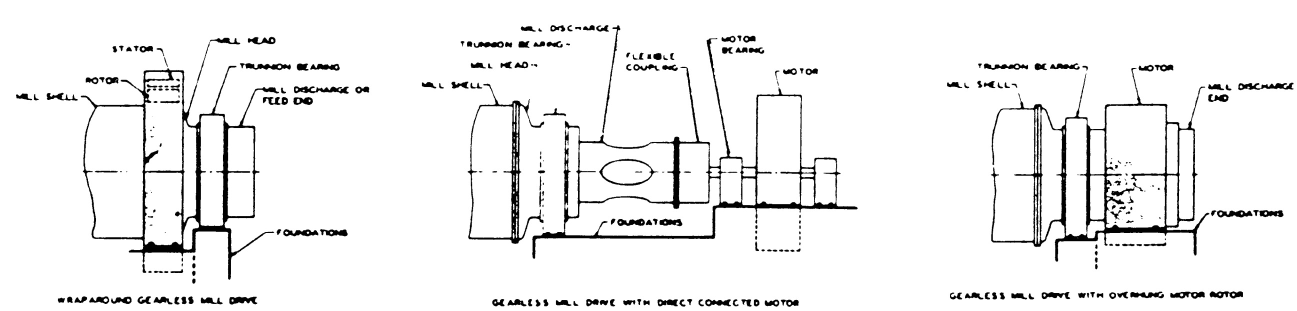

Fig. 25.3.2.9. Wrap-around gearless mill drives.

Closed-circuit grinding creates a *circulating load* which is defined as the ratio of the weight of the coarse material returned to the mill by the classifier to the weight of new feed.

SINGLE AND MULTISTAGE GRINDING. Grinding circuits may be either single stage or multistage. Single-stage grinding uses one mill per stage. Single-stage grinding may be either open or closed circuit in a rod, ball, autogenous, or semi-autogenous mill. Multistage grinding may use a combination of rod, ball, autogenous, or semi-autogenous mills in open or closed circuits.

CIRCUIT SELECTION. For years, an overlap has existed between fine crushing and coarse grinding, particularly coarse rod milling. In the last 25 years, the use of autogenous or semi-autogenous grinding (SAG) to do the work of secondary/tertiary crushing and rod milling (i.e., grinding primary crusher discharge to produce feed for a second grinding stage or, in some cases, final product) has added fuel to the argument. More recently, autogenous and SAG mills capable of drawing about 8950 kW (12,000 hp) with capacities of 450 to 1800 t/h (500 to

Table 25.3.2.10. Grinding Circuit Type (Primary Crusher Product to Beneficiation Feed)

	Circuit Parameter	1 Primary Autogenous	2 Primary Semi-Autogenous	3 Primary Autog. Secondary Pebble Mill	4 Primary Semi-Autog. Secondary Ball Milling	5 Cone Crusher Rod Mill Pebble Mill	6 Cone Crusher Rod Mill Ball Mill	7 Cone Crusher Single Stage Ball Mill	8 High Energy Automated Crushing—Single Stage Ball Mill
1	Capital Cost	2	1	5	3	8	7	6	4
2	Operating Cost								
	2.1 Maintenance	1	8	2	7	5	6	4	3
	2.2 Metal Wear	1	8	2	7	4	6	5	3
	2.3 Energy	8	7	6	5	3	2	4	1
3	Energy Efficiency	8	7	6	5	3	1	4	2
4	Medium Availability, Size, Qty.								
	4.1 Metallic: US	—	3	—	6	4	5	2 (3–3½ in.)	1 (3 in.)
	4.2 Metallic: Ex US	—	3	—	3	5	5	1	1 (3 in.)
	4.3 Rock	8	—	7	—	6	—	—	—
5	Capacity Fluctuations	7 (3–400%)	8 (2–600%)	6	5	4	1	3	2
6	Degree of Automation Required	Automation Absolutely Required	Automation Absolutely Required	Automation Absolutely Required	Automation Absolutely Required	Automation Necessary (Pebble Addition)	Manual Possible	Manual Possible	Automation Cone Crushing Req'd Ball Mill Preferred
7	Time to Bring Circuit into Stable Operation	2–3 Years	2–3 Years	(2-Stage Balancing) 2–3 Years	2–3 Years	1 Year	2–3 Months	1–2 Months	1–2 Months
8	Testwork Required					Secondary Pebbles			
	8.1 Media Competency	x	x	x	x	x	—	—	—
	8.2 Bond Grindability	x	x	x	x	x	x	x	x
	8.3 Impact W_i—Low Energy	x	x	x	x	x	x	x	x
	8.4 Abrasion Index	x	x	x	x	x	x	x	x
	8.5 Autog. Pilot Plant	x	x	x	x	—	—	—	—
	8.6 High Energy Impact	—	—	x	—	x	x	x	x
9	Ability to Control Product	8	7	6	5	4	1	3	2
10	Potential for Stage Classification	1	1	8	5	7	6	4	3
11	Circuit Availability	87–95%	80–92%	87–95%	80–92%	94–96%	94–96%	96–98%	96–98%
12	Feed Size Required for First Stage of Grinding	8–9 in. Normal	8–9 in. Normal	8–9 in. Normal	8–9 in. Normal	13,200 μm	13,200 μm	10,000 μm Closed Circuit Crushing Required	7,000 μm Closed Circuit Crushing Required
13	Ability to Handle Damp or Sticky Ore	2	1	4	3	5	6	7	8

Source: Engineering & Mining Journal, Feb. 1987.
Conversion factor: 1 in. = 25.4 mm.

the question of capital cost and the advantages and disadvantages of using one large machine to do a major portion of the work required.

At the same time, work has been done to improve the capacity, energy utilization, and availability of cone crushers for secondary and tertiary crushing to make finer feed for the rod mills and single-stage ball mills. Today mechanical reliability is available, and its demonstrated ability to crush to 80% passing 6.4 mm (¼ in.) encroaches on some of the work traditionally done by rod mills. In fact, for some applications, they eliminate rod mills completely in that single-stage ball mills can accept this product as feed if it contains only a small oversized feed factor.

Obvious in both of these developments is an attempt to eliminate, or at least reduce substantially, the wear of steel and iron grinding media and wear parts, particularly steel grinding rods.

Comparison of circuits utilizing autogenous or semi-autogenous grinding with those using cone crushing is the subject of a number of papers. Two international symposia contained papers comparing autogenous and semi-autogenous grinding with conventional circuits (circuits with secondary and tertiary crushing). From these, it is apparent that the energy consumption in the comminution machines is lower in conventional circuits than in circuits using autogenous or semi-autogenous primary mills; and, in general, the reverse is true for operating cost. Operating cost depends on the amount of work done by secondary/tertiary crushing and grinding in the conventional circuit and that done by the primary mill in a circuit using an autogenous or semi-autogenous mill.

RANKING CIRCUITS. Comminution consultants for Boliden-Allis have identified eight basic comminution circuits (see Rowland, 1987). In practice, there are variations of these circuits, such as in-circuit beneficiation steps and crushers in the primary autogenous grinding circuit, to fit specific ore characteristics. Circuits in which the coarse part of the primary mill discharge is crushed in a cone crusher, then either returned to the primary mill as circulatory load or fed to the secondary stage, are called ABC circuits (autogenous mill-ball mill-crusher circuits) (Table 25.3.2.10).

Among the parameters, item 8, test work, is of particular importance. It spells out the need for ore testing and lists specific tests required to specify equipment for each circuit. This is the first step in circuit selection, as it defines the physical characteristics of the ore needed to select the comminution circuit and the process equipment needed in that circuit.

Both the rankings and their significances can vary with each ore deposit. Therefore, an independent comparison should be made for any specific study.

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25.3.3 SCREENING

V.P. KENYEN

Screening is a unit operation that is an essential part of many different processes. The term applies to the mechanical separation of particles on the basis of size. This separation is accomplished on a screening surface with uniformly sized apertures. Particles larger than these apertures are rejected and pass over the surface, while those smaller (in at least two dimensions) are accepted and pass through. Many other terms are applied to this operation including sizing, sifting, sieving, and separation.

Most commonly, screens are used in conjunction with size-reduction operations, but they are found in other applications such as washing, dewatering, desliming, hedusting, scalping, concentration, and conveying.

Screens are used for processing a wide variety of materials, ranging from the fineness of talc up to boulders as large as 1.8 by 1.8 m (6 by 6 ft). Openings in screening surfaces may be as small as 127 μm (0.005 in.) or as large as 200 or 250 mm (8 or 10 in.), with some as large as 460 mm (18 in.).

There are two types of screens: stationary and dynamic (vibrating). Tables 25.3.3.1 and 25.3.3.2 show the principal types and applications of stationary and dynamic screens in use today.

Table 25.3.3.1. Types of Stationary Screens

Type and Mode of Operation	Application
1. *Grizzly:* Consists of a set of parallel bars, rods, or rails with a uniformly clear opening; often tapered from feed to discharge end to provide self-cleaning characteristics. Opening size is set by the maximum size of rock that can be safely delivered to and handled by equipment following. Grizzlies may be level, for use over bins or inclined for use ahead of crushers or belts.	Scalp out large rock using grizzly between truck dump and bin. Allow fines to bypass crusher, thus reducing load on machine.
2. *Static sieve:* Consists of a set of parallel profile bars or wires positioned at right angles to line of flow, and usually sloped at incline of 45° or more to the horizontal. Bars may form a flat, curved, or multi-sloped surface. Generally used only on wet screening operation.	Used for size separation, scalping, entering in range of 2 mm (10 mesh) and finer.

Source: Weiss, 1985.

Table 25.3.3.2. Types of Dynamic Screens

Type and Mode of Operation	Application
1. *Vibrating screen:* A screen with motion in a vertical plane which operates, generally, above 600 cpm at less than 25.4-mm (1-in.) stroke. Two arrangements are available; inclined or horizontal screening surface. Available with mechanical or electromagnetic vibrators that apply motion either to screen frame or screening surface.	See also inclined and horizontal.
2. *Inclined vibrating screen:* A vibrating screen arranged with an inclined screening surface. Material is fed in at the upper end and flows down the incline aided by the vibrating force. Circular stroke stratifies the bed, bringing fines down to the screen surface for separation. Coarse discharges over the end of the screen.	High-capacity units for separation of a wide range of particle sizes. Also used for scalping and trash removal.
3. *Horizontal vibrating screen:* A vibrating screen arranged with a horizontal screening surface. Material is fed in at back end and is moved over the length of screen by a positive pulsating stroke that throws the material up and forward, and withdraws deck down and backward. Material remains on the screen longer, and efficiency and accuracy of sizing are high.	Close sizing of medium-sized particles. Dewatering or media-recovery. Installations where headroom is limited.
4. *Oscillating screen:* A screen with linear motion, generally, in the larger stroke and slower speed (100 to 400 rpm) range.	Generally, used in -13 mm $+$ 250 μm ($-\frac{1}{2}$ in. $+$ 60 mesh) range. Light, free-flowing materials can be separated at 74 μm (200 mesh) and less.
5. *Reciprocating screen (shaking screen):* A screen with substantially linear motion in the plane of the main frame. Stroke is normally in range of 25 to 102 mm (1 to 4 in.), with speed from 30 to 200 cpm. Unit is slightly inclined and may be suspended from rods or cables, or supported from base by durable flat springs.	Units may be used for both conveying and size separation. Low capacity and high maintenance are handicaps. Good for accurate sizing of large lumps.
6. *High-speed screen:* A relative term referring to the operating frequency of a screen. Generally applied to those operating in excess of 3000 rpm or cycles per minute. Vibrator or exciter may be mechanical or electromagnetic.	Generally used for fine and ultra-fine screening.
7. *Vibrating grizzly:* Consists of a set of parallel bars or rails with a uniform clear opening space set for the maximum size of rock that can be safely delivered to and handled by the equipment that follows. Mechanical or electromagnetic vibrator provides positive force to move material permitting use of flatter slopes than those required by stationary grizzlies (10° vs. 45°).	Used to scalp out large rock for crushing, fines pass through readily because of large openings and are bypassed around the crusher. Often combined with feeder to save space and headroom.
8. *Revolving screen:* This is also referred to as trommel, scrubber, or barrel screen and is a cylinder on a shaft, mounted on rollers, with the screen surface forming the circumference. Screening surface may be wire cloth or perforated plate. The cylinder is open at both ends and inclined at a slight angle. These screens revolve at low speeds, usually 15 to 20 rpm. Both capacity and efficiency of these units are low when compared with vibrating screens.	Used for scrubbing and washing wet, sticky materials, and making a rough size separation. Used for wet sizing of sand and gravel in wet-process sand plants.

Source: Weiss, 1985.

25.3.3.1 Vibrating Screen Description

Vibrating screens are the most widely used in the mineral processing industry. The largest conventional vibrating screens installed to date are two 3.6 by 8.5 m (12 by 28 ft) Allis-Chalmers screens at Ok Tedi. Fig. 25.3.3.1 shows the main elements of a vibrating screen. Some of these are discussed below.

When selecting the type of deck coverings, the abrasive and sticky nature of the ore must be considered. Metallic decks have long been the surface of choice. Lately, however, there has been a major increase in the use of rubber and polyurethane screen decks. These screens come as small panels that can be replaced individually. Rubber decks are manufactured by Linetex, Trelleberg, and Skega among others, and polyurethane decks are manufactured by Polydeck Screen Corp.

Generally, the function of the top deck of multisurface screens is to relieve the load on the lower deck(s). Therefore, accurate sizing by the top deck is not necessary. A good practice is to use double-tapered, stepped cast grizzly bars, usually made of manganese or manganese steel for impact and abrasion resistance. Alternate materials such as high-chromium iron (26% Cr) are sometimes used; however, that material is so brittle that it does not stand up well under impact conditions.

One of two configurations is normally used on the lower or sizing surfaces (decks). These are the long-slot and rod types. The long-slot, or Ty-rod type of woven wire cloth provides the advantages of both increased throughput capacity and anti-blinding action as a result of secondary vibrations in the long wires. The rod-type deck is constructed from individual rods of high-carbon oil-tempered steel with a hardness of 46 to 48 Rockwell C. Rods are held in rubber end holders and intermediate spacers. The rubber allows the rods to move with secondary vibration to help alleviate buildup of sticky ore and relieve wedge blinding. This type of deck gives reasonably good life and high capacity.

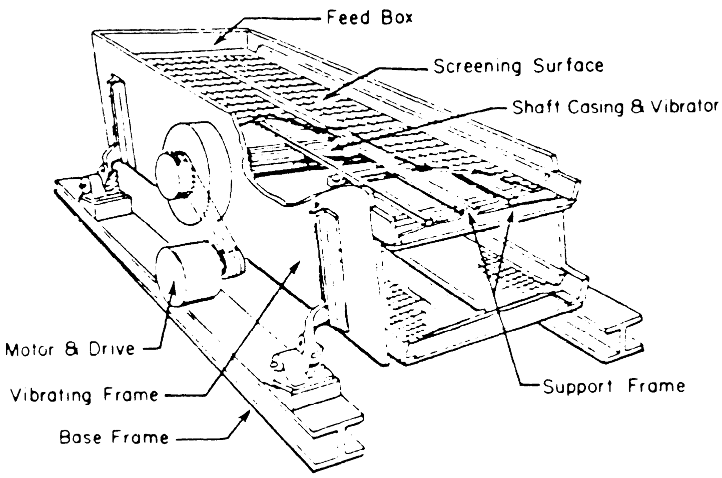

Fig. 25.3.3.1. Main elements of a vibrating screen (Weiss, 1985).

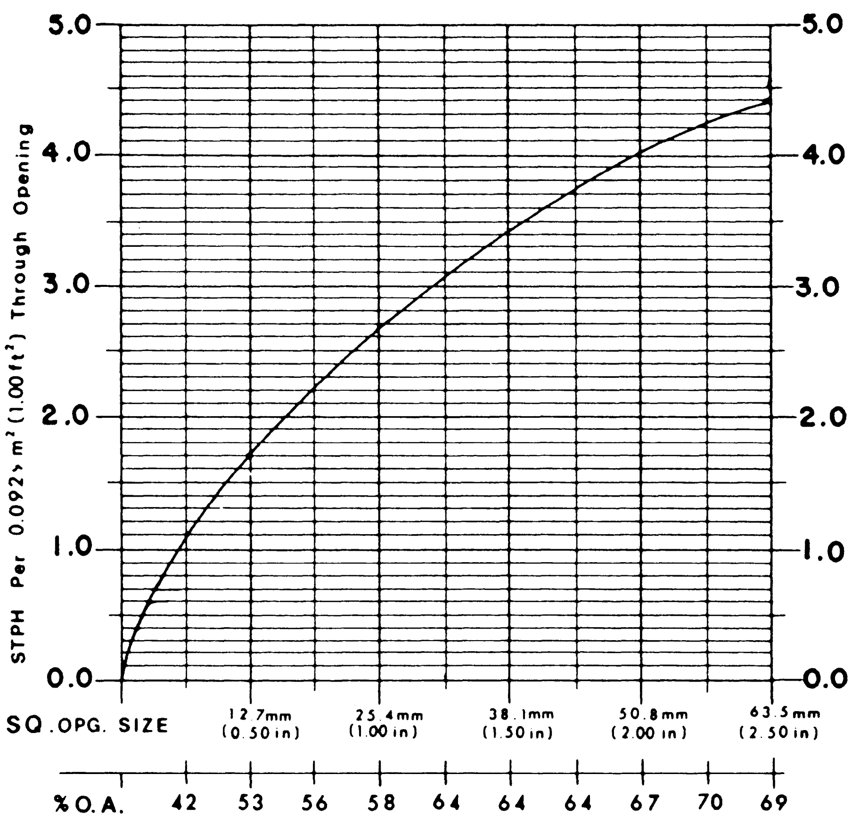

Fig. 25.3.3.2. Basic screen capacity (for ores having 1602-kg/m^3 or 100-lb/ft^3 bulk density) (Weiss, 1985).

25.3.3.2 Vibrating Screen Size Selection

The generally accepted approach to determining the screen area is the throughput method, which is based upon the amount of material (weight per hour) that will pass through 0.0929 m^2 (1.0 ft^2) of a screen deck having a specific opening.

The total required screen area A in square meters is given by the relation,

$$A = U/(C \times BD \times F \times E \times S \times D \times O \times W) \quad (25.3.3.1)$$

where U is tonnes per hour of screen undersize in the feed, C is basic screen capacity from Fig. 25.3.3.2, BD is bulk density expressed as the fraction found by dividing the density of the feed material (kg/m^3 or lb/ft^3) by 1602 kg/m^3 or 100 lb/ft^3, and F, E, S, D, O, and W are modifying factors defined in the following.

BASIC CAPACITY. The basic capacity curve representing the value of C for the various openings is based on a material with a bulk density of 1602 kg/m^3 (100 lb/ft^3). Since most metallic ores have similar screening characteristics, the C value for any ore can be determined by a simple ratio of densities.

The curve shown in Fig. 25.3.3.2 should be used for metallic ores only.

MODIFYING FACTORS. Many variables affect the screenability of a particular material. Only those that significantly affect the sizing of a screen are discussed here.

Table 25.3.3.3. Fines and Efficiency Factors

Halfsize, %*	Fines F	Efficiency E
0	.44	
10	.55	
20	.70	
30	.80	
40	1.00	
50	1.20	
60	1.40	
70	1.80	2.25
80	2.20	1.75
85	2.50	1.50
90	3.00	1.25
95	3.75	1.00

* Interpolate for percentages not shown.
Source: Weiss, 1985.

Table 23.3.3.4. Slotted Deck Opening Factors

Typical Deck Preparations	Length/ Width Ratio	Slotted Opening Factor S
Square and Slight Rectangular Openings	<2	1.0
Rectangular Openings Ton-Cap	>2 but <4	1.15
Slotted Openings Ty-Rod	>4 but <25	1.2
Parallel Rod Decks	>25	SP 1.4* RA 1.3

*SP = Slots parallel to flow, RA = Slots right angles to flow
Source: Weiss, 1985.

Fines Factor F—The *fines factor* is a measure of the amount of material, in the feed to a screen deck, which is less than one-half the size of the opening in the deck. It is a measure of the difficulty of screening when compared to 40% fines (F = 1.00), which is the C value used for the basic capacity. Table 25.3.3.3 gives the values for fines factors.

For calculating the fines factor, use the percentage of half-size coming to the particular deck, expressed as a percentage of the feed to that deck. For example, if the top deck has a 38mm (1.50-in.) opening and the bottom deck has a 12.5mm (0.50-in.) opening, and the feed analysis is 70% passing 38 mm, 50% passing 12.5 mm, and 35% passing 6.3 mm, then the fines factor for the bottom deck would be 35/70 = 50%.

Efficiency Factor E—The *efficiency of separation* is expressed as the ratio of the amount of material that actually passes the opening divided by the amount in the feed that should pass. "Commercially perfect" screening is considered to be 95% efficient. Therefore, the factor for 95% is 1.00 as indicated in Table 25.3.3.3. For scalping and crusher relief applications, an efficiency of 80 to 85% is generally acceptable.

Slot Factor S—The *slotted opening factor* compensates for the tendency of more particles to pass through the screening medium due to a fewer number of cross bars or wires obstructing the particle paths through the deck. The values for various slot configurations are given in Table 25.3.3.4.

Deck Factor D—The *deck factor* (Table 25.3.3.5) allows for the fact that stratification does not take place at the extreme end of the deck, so the bulk of the fines does not fall through until the material has traveled a short distance down the deck. Therefore, except for the top deck, the entire feed does not land on the feed end, which results in some of the screening area being ineffective.

Table 25.3.3.5. Deck Factor

Deck	Deck Factor D
Top	1.00
2nd	0.90
3rd	0.80

Source: Weiss, 1985.

Table 25.3.3.6. Wet Screening Factor

Size Opening (In. square)	W
1/32	1.25
1/16	3.00
1/8 and 3/16	3.50
5/16	3.00
3/8	2.50
1/2	1.75
3/4	1.35
1	1.25
+2	1.00

Source: Weiss, 1985. Conversion factor: 1 in. = 25.4 mm.

Open Area Factor O—The *open area factor* gives the ratio between the actual open area of the screen and the wire deck openings assumed in the standard capacity curve.

For example, if a deck with 36% open area was used for a 24-mm (0.95-in.) separation, the factor would be 36/58 or 0.62. Alternately, if a deck with 72% open area was used, the factor would be 72/58 or 1.24.

Wet Screening Factor W—A factor is applied when *wet screening* is used. The benefit derived from wet screening varies with the opening size, and the factors are given in Table 25.3.3.6.

The advantage of wet screening is realized only when the correct amount of water is used. The recommended volume for efficient wet screening varies from a minimum of 11.35 L/min (3 gpm) to 18.92 L/min (5 gpm) per 0.765 m^3 (1.0 yd^3) of feed material.

Example 25.3.3.1. To illustrate the selection procedure, the following sample problem is used.

Iron ore is fed at 272 t/h (300 tph). Bulk density of the ore is 2082 kg/m^3 (130 lb/ft^3), with 8% moisture. Size analysis of the ore is as follows:

Size Fraction	Cumulative % Passing
38 mm (1½ in.)	100.00
25 mm (1 in.)	98.00
19 mm (¾ in.)	92.00
12.5 mm (½ in.)	65.00
6.3 mm (¼ in.)	33.00

Separation required: 12.7 mm (½ in.), using dry screening.

Solution.

$A = (300\text{ tph}) \times (.65) / (C \times BD \times F \times E \times E \times S)$

$C = 1.7\text{ t}/0.09\text{ m}^2$, $BD = 2{,}082/1{,}602 = 1.30$, $F = 0.86$ (33%),

$E = 1.00$ (95%), $D = 1.00$ (top), and $S = 1.2$ (Ty-Rod cloth)

$A = 195\text{ tph}/(1.7)(1.3)(0.86)(1.00)(1.00)(1.2)$

$= 195\text{ tph}/(2.28\text{ tph}/0.0929\text{ m}^2/\text{ft}^2) = 7.95\text{ m}^2\ (85.5\text{ ft}^2)$

Table 25.3.3.7. Rate of Material Travel on Inclined Circle-Throw Screens with Counterflow Rotation

Angle, °	Flow rate, m/min (fpm)
18	18.29 (60)
20	24.39 (80)
22	30.48 (100)
25	36.58 (120)

Source: Weiss, 1985.

The 7.95-m^2 (85.5-ft^2) area represents the net effective screen area required. To allow for the space taken up by tension rails, center clamps, and openings that are blocked by cloth support bars, 10% is added to the net area.

$$7.95 + 0.79 = 8.74 \text{ m}^2 \text{ } (94.0 \text{ ft}^2).$$

A commercial screen with at least 8.74 m^2 (94.0 ft^2) of area should be selected, keeping in mind that the length-to-width ratio should be at least 2:1 for effective screening. Therefore, the tentative size of the screen should be 1.83 × 4.87 m (6 × 16 ft), with a total area of 8.91 m^2 (96.0 ft^2).

The depth of bed of material passing over the deck must be checked to be sure it is within acceptable limits. The depth of bed at the discharge end of the deck should not be more than four times the size of the deck opening. For example, if the deck opening is 12.5 mm (½ in.), the depth of bed at the discharge end should not be more than 50.8 mm (2 in.).

To determine the depth of bed, refer to Fig. 25.3.3.3, which gives the capacity per centimeter (2.54 in.) of depth for various widths, based on a rate of material travel of 18.29 m/min (60 fpm).

Rates of travel for various angles of installation are given in Table 25.3.3.7. the bed depth obtained from fig. 25.3.3.3 is adjusted by the ratio of 18.29 m/min (60 fpm) divided by the rate of travel of the tentative installation angle. Crushing plant screens are usually installed at 20 to 25°, with the majority being at 20°.

Checking the example shows that a 1.83-m (6-ft) wide screen will have 4.6 t/h/cm (116 tph/in.) depth of bed. The depth of bed passing over the deck, therefore, is 105/4.6 = 22.8 mm (0.90 in.). This is below the maximum allowed depth of 50.8 mm (2.0 in.). If the screen is installed at 20°, the depth of bed will be somewhat less.

In conclusion, one 1.83 × 4.87 m (6.0 × 16.0 ft) inclined, circle throw screen will be sufficient to screen the iron ore.

When calculating double-deck screen areas, each deck should be treated as a separate problem.

RECIRCULATING LOAD. When the screen is in closed circuit with a crusher, the approach to screen selection is slightly different. The total feed rate to the screen including recirculating load and the size analysis of the total feed should be calculated. For an example of how to size a screen with a recirculating load, the reader is referred to Section 3E on screening of the *SME Mineral Processing Handbook* (Weiss, 1985).

25.3.3.3 High-speed Screens

High-speed screens are used for very fine separations. The screen produces a high-frequency (3600 rpm) and low-amplitude vibration that breaks the fluid surface tension and promotes close screen/particle contact.

An example of a typical high-speed screen is the Derrick Multifeed screen illustrated in Fig. 25.3.3.4. A particularly interesting application is the successful replacement of cyclones in some closed-circuit grinding circuits where the higher separation efficiency of the screen has resulted in significant increases in capacity.

25.3.3.4 Sieve-bend Screens

Sieve-bend screens are high-capacity screening devices that utilize stationary, concave, wedge-bar screening surfaces. A complete sieve bend is comprised of a feed distribution chamber, a structure for holding the screen surface, and collection chambers or chutes for oversize and undersize materials. In devices used in metallurgical applications, provision is usually made for reversal of the direction of flow over the surface, either by rotating

Table 25.3.3.8. Typical Sieve Bend Types and Sizes

Type sieve bend	Radius, in.	Surface length, in.	Available widths, ft	Typical bar spacing, mm	Max. capacity feed, gpm per ft width	Required installation height, ft	Required floor area, ft^2 per ft width	Approx. weight, lb	Capital cost, $*
Gravity feed fixed surface, 50°	36	30	2	0.35 to 3.50	350	6	7 × 3	600	1,900
			4				7 × 5	900	2,500
			6				7 × 7	1,150	2,900
Gravity feed fixed surface, 45°	80	60	2	0.35 to 3.50	350	8	8 × 3	850	2,600
			4				8 × 5	1,200	3,600
			6				8 × 7	1,550	4,300
Gravity feed reversible surface, 60°	30	30	4	0.35 to 3.50	200	8	6 × 8	2,000	6,000
Pressure feed reversible, 270°	20	94	1½	0.20 to 0.60	300	7	9 × 8	2,100	6,300
Gravity feed fixed surface plus rapped, 45°	64	50	2	0.07 to 0.30	150	9	9 × 3	900	4,700

* Note: Carbon steel flanged feed distributor, housing, flanged oversize and undersize collecting chambers including 304 SS screen surface, 1970 prices.

Source: Weiss, 1985.

Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 ft^2 = 0.0929 m^2, 1 lb = 0.4536 kg, 1 gpm = 3.785/min.

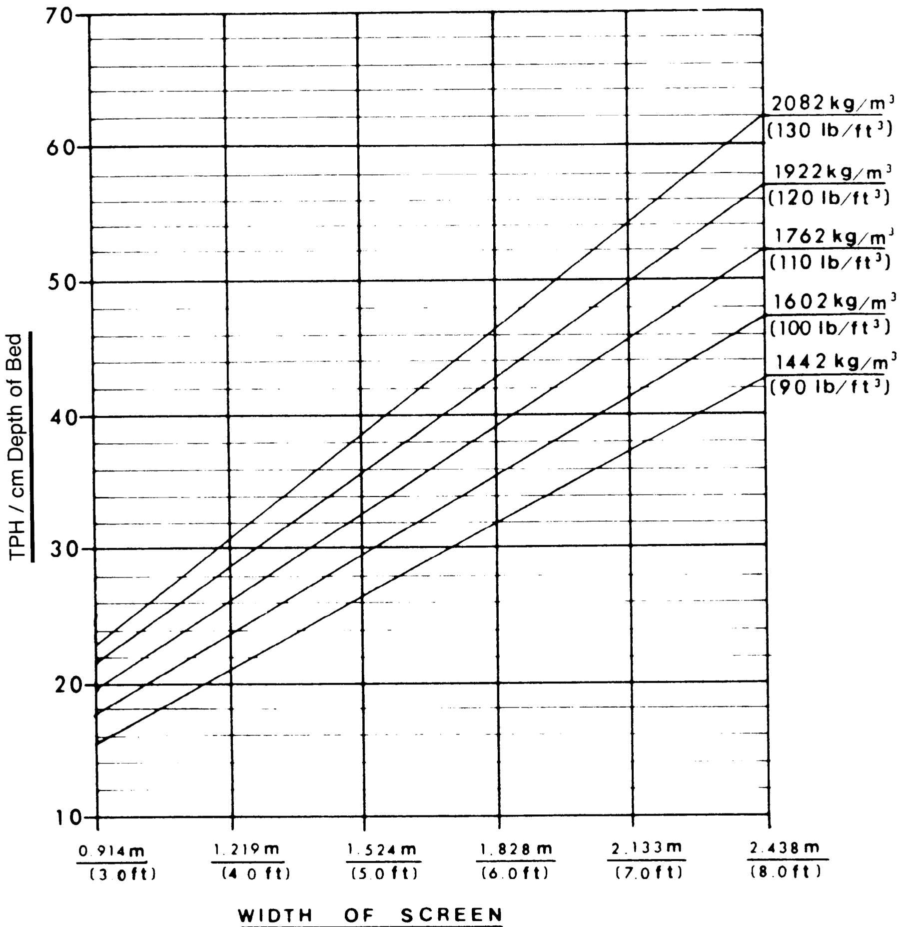

Fig. 25.3.3.3. Depth of bed chart (based on 18.29-m/min, or 60 fpm flow rate) (Weiss, 1985).

the surface or by changing the feed nozzle position. The size of the unit varies in height from 1.5 to 2.4 m (5 to 8 ft) and in width from 0.61 to 2.4 m (2 to 8 ft).

The orientation of the wedge-bar surface is such that the slots between the wedge bars are perpendicular to the flow of feed material over the surface. The feed is distributed evenly and tangentially over the width of the screen surface. A variety of means is used to accomplish this including simple gravity weir feeds, long slotted nozzles, and a multiplicity of round nozzles. A typical gravity-fed weir-type nozzle is shown in Fig. 25.3.3.5. Most sieve bends are fed by gravity in such a manner that a falling height of approximately 500 mm (20 in.) is provided before the feed meets the screen surface. In some cases where fine bar spacings (200 μm or less) are used, it is necessary that the feed stream meets the screen surface at a higher velocity. In such circumstances, pressurized nozzle feed arrangements are used.

For fine separations (200 μm or less), gravity feeding with a rapped or vibrated surface can be used. Such a device, known as a Rapifine screen, has found application in iron ore and coal beneficiation. A variety of low-frequency (6 to 10 raps/min) mechanical rappers mounted behind the screen deck are used. They impact the surface with a force of 1.4 to 3.4 J (1 to 2.5 ft-lb). Sieve bends are constructed of carbon steel, stainless steel, and composite materials.

Sieve-bend screens are used for separations in the range of 0.15 to 3 mm (0.0059 to 0.118 in.). Unlike hydrocyclones whose classification efficiency depends on the specific gravity of the particle and percentage solids of the feed, the efficiency of sieve-bend screens depends upon the particle size. As opposed to hydrocyclones, it can be fed at much higher percent solids, such as 45 to 50% solids by volume.

SIZING OF SIEVE-BEND SCREENS. To determine the size of the sieve bends, first the diameter of separation should be determined. By definition, this is the diameter of particle that reports 95% to the oversize fraction. Then the slot opening is determined from Fig. 25.3.3.6, which gives the slot opening for a given separation size. Using Fig. 23.3.3.7, the capacity of screen surface for a given slot opening is then determined. Data given in Fig. 23.3.3.7 are only for gravity-fed screens using bar widths of 2.3 mm (0.090 in.), and correlate the maximum volumetric feed rate in liters (gallons) per minute per linear meter (foot) of sieve-bend screen width. Reference to Table 25.3.3.8 then provides the specification for the screen so that number of linear meters (feet) required for given process capacity can be determined.

25.3.3.5 Other Screens

Revolving screens or trommels find application in breaking up and screening loosely cemented materials and alluvials. They are mainly used on tin and gold dredges, and other alluvial operations and in washing and sizing sands and gravels. In washing operations, water is jetted on to the material through nozzles from a spurge pipe down the center of the trommel.

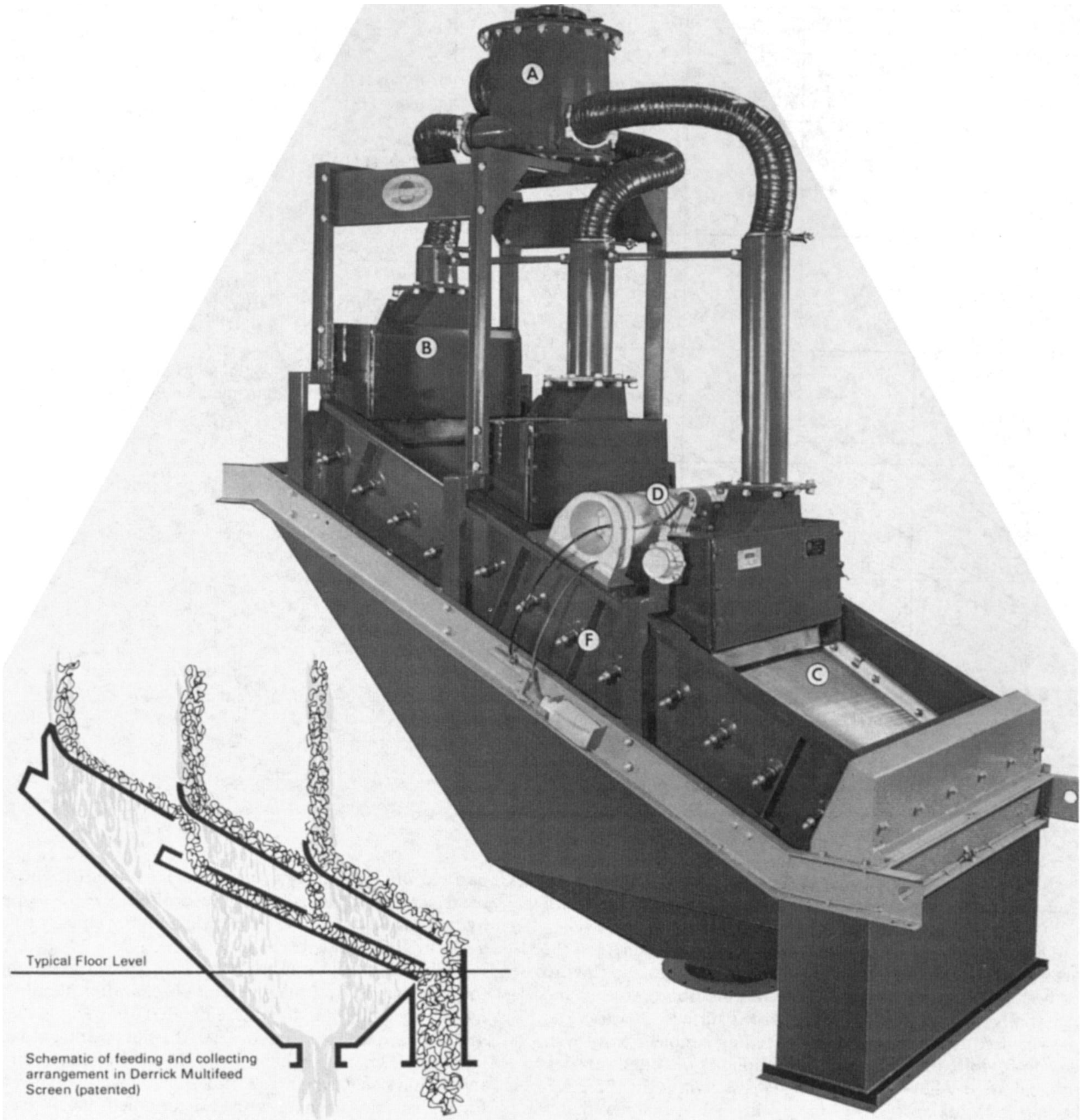

Fig. 25.3.3.4. Derrick multifeed screen (*courtesy:* Derrick Manufacturing Company).

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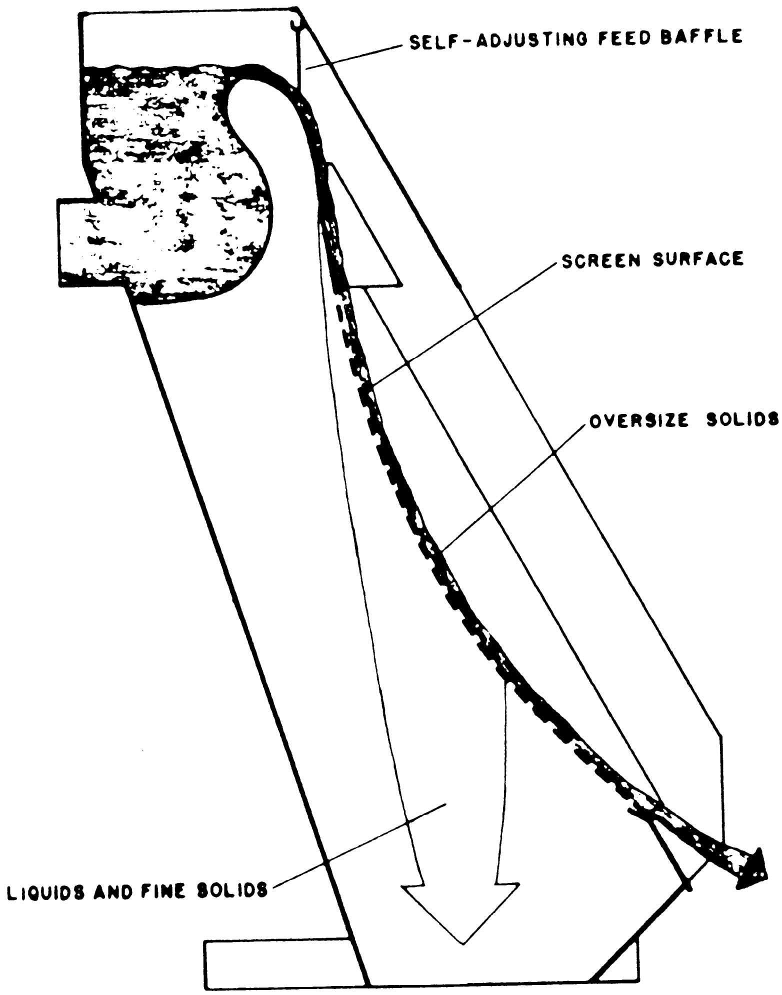

Fig. 25.3.3.5. Gravity-fed sieve-bend screen (Weiss, 1985).

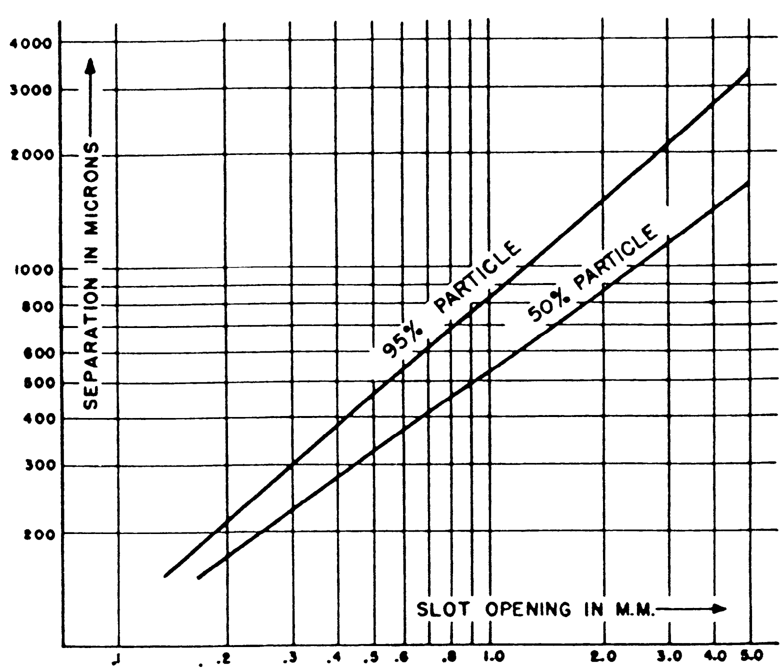

Fig. 25.3.3.6. Diameter of separation as a function of bar spacing (Weiss, 1985). Conversion factor: 1 in. = 25.4 mm.

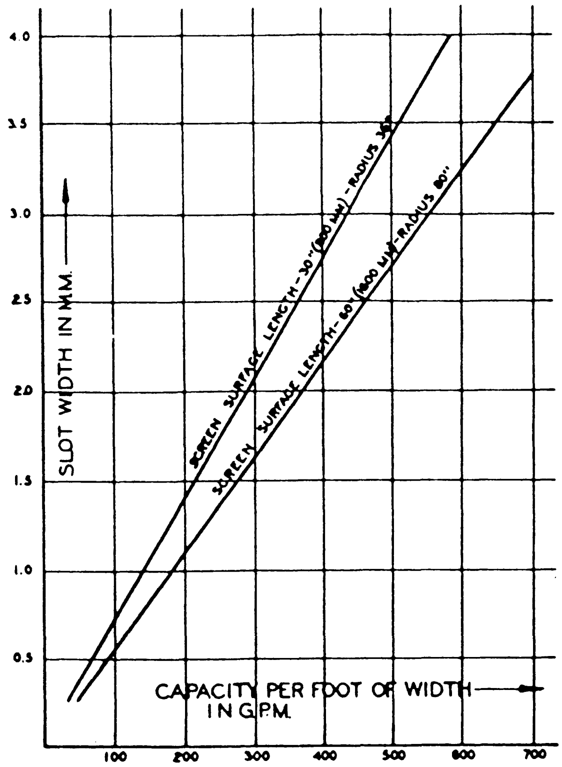

Fig. 25.3.3.7. Volumetric feed rate per lineal foot of sieve-bend screen width as a function of slot width (Weiss, 1985). Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 gpm = 3.785 L/min.

25.3.4 HYDROCYCLONING

V.P. KENYEN

Used to accomplish size separation of solids carried in slurries, the *hydrocyclone,* or liquid cyclone, as it is often called, is in most cases an extremely simple mechanical device with no moving parts. It is cylindroconical in shape, varying in size from a few millimeters (tenths of inches) to as large as perhaps 254 mm (10 in.) in diameter. The length-to-diameter ratio varies over a large range from 2:1 to 10:1. Weights of hydrocyclones vary from a few hundred grams (ounces) to several thousand kilograms (pounds), depending upon size and materials of construction.

The hydrocyclone depends on external power for its operation. Most commonly, a continuous flow centrifugal pump is used. Occasionally, it can also be operated using gravity feed systems provided sufficient head is available. Feed velocity head and pressure head are converted to both angular and linear acceleration, creating a cyclone effect where the angular acceleration increases as the feed liquid moves from the outside wall of the cyclone toward the axis of rotation. As the angular acceleration increases, the centrifugal forces also increase, causing the separation of particles by size and/or specific gravity.

Fig. 25.3.4.1 shows a typical cyclone geometrical configuration, while Fig. 25.3.4.2 illustrates the action of a cyclone.

25.3.4.1 Cyclone Components

The following terminologies are normally used in the classification of hydrocyclones.

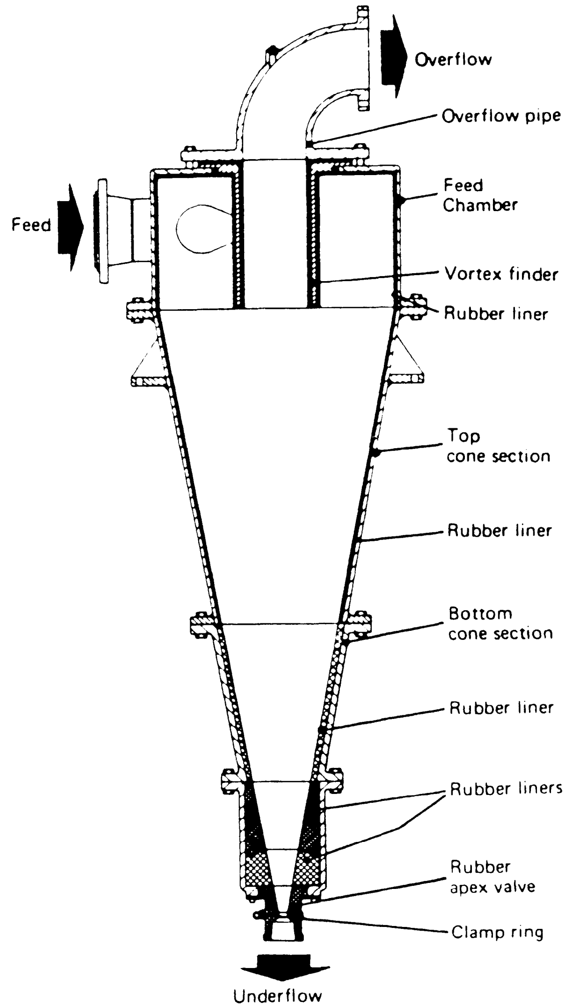

Fig. 25.3.4.1. Typical cyclone geometrical configuration (*courtesy:* Dorr-Oliver Co.).

FEED NOZZLE. Usually, the feed nozzle is tangent to the inside diameter of the cyclone. Since there are no moving parts within the cyclone, or power attachment directly connected to it, the spinning action required to develop the separating forces depends upon the feed velocity of the slurry entering the feed chamber tangentially. For a given size of cyclone, high pressure, caused by high velocity, yields a finer separation. While the smaller-diameter cyclones often respond to higher pressure in the range of 407 to 690 kPa (59 to 100 psi) gage, the normal operating velocity of most hydrocyclones is 39 to 66 m/s (12 to 20 fps).

VORTEX FINDER. At the top of the feed section, there is a flat plate through which projects the overflow or vortex finder. The vortex finder extends down into the feed chamber, terminating below the juncture of the cylindrical feed section with the conical section of the cyclone. The function of the vortex finder is to discharge the finer fraction of the feed to the process. Normally, the vortex finder is the largest nozzle in the liquid cyclone, and it discharges most of the water or liquid being fed to the cyclone. Fig. 25.3.4.3 gives the approximate relationship between cyclone diameter and feed nozzle size.

APEX VALVE. The smallest cross section of the cone on the hydrocyclone forms the apex nozzle. The apex orifice is the

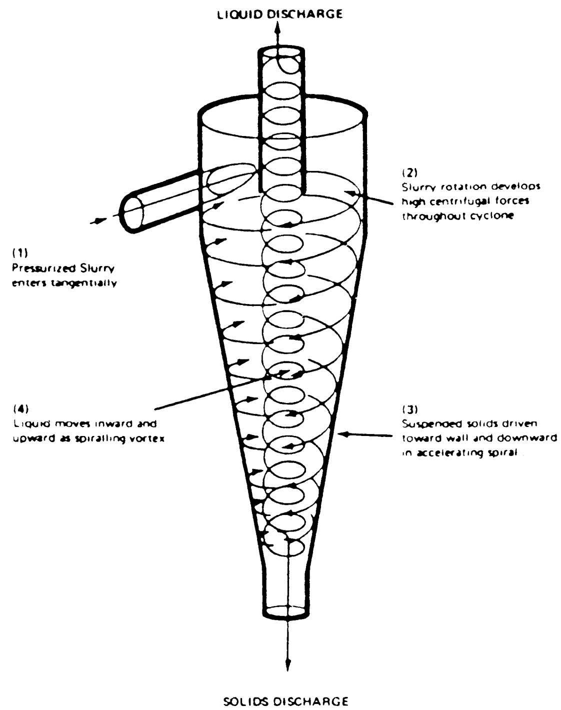

Fig. 25.3.4.2. Typical action within the cyclone (*courtesy:* Picenco International, Inc.).

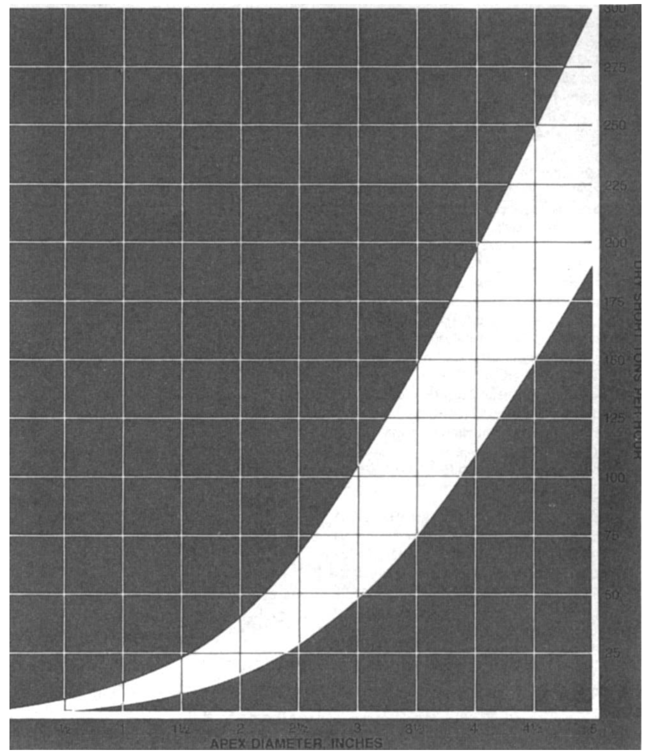

Fig. 25.3.4.4. Apex capacity chart (Krebs Engineers). Conversion factors: 1 in. = 25.4 mm, 1 tph = 0.9072 t/h.

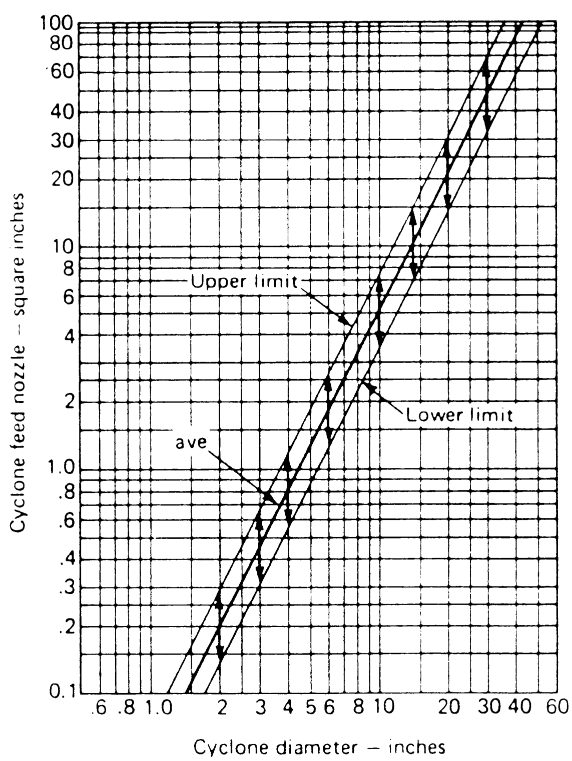

Fig. 25.3.4.3. Approximate cyclone feed-nozzle area vs. cyclone diameter (Weiss, 1985). Conversion factors: 1 in. = 25.4 mm, 1 in.^{2} = 645 mm^2.

smallest orifice in a hydrocyclone, and the coarse fractions of the particles are discharged through it at a higher percentage solids than the feed. Apex valves are made of either hard abrasion-resistant materials or natural gum rubber.

APEX SELECTION. The apex size is critical to proper cyclone performance. For each application, the required circulating load establishes the amount of solids which must be passed through the cyclone underflow.

Underflow density of 50 to 53% by volume is typical for primary grinding circuits; for regrind circuits, 40 to 45% by volume is typical. Therefore, using these guidelines, the apex can be sized.

Fig. 25.3.4.4 shows the approximate apex solid capacity for various apex diameters.

CLASSIFICATION AND SIZE OF SEPARATION. Historically, *classification* is defined as the particle size at which 1 to 3% report to the overflow, with the coarser particles reporting to the cyclone underflow. This is also referred to as the "mesh of separation." Lately, classification is defined as the size at which 50% of the solids reports to the overflow and the other 50% reports to the underflow. This theoretical particle size is defined as "*D*50." Fig. 25.3.4.5 shows an actual and corrected recovery curves. Note the actual curve does not reach zero probability because part of the underflow product is carried into the underflow as water or void filling solids. When the necessary correction is applied, a "corrected" recovery curve is obtained.

To further utilize this graph and make it applicable to any *D*50 point, each particle size is divided by the *D*50 value and a "reduced" recovery curve, as shown in Fig. 25.3.4.6, is obtained. Recent investigations have shown that this curve remains constant over a wide range of cyclone diameters.

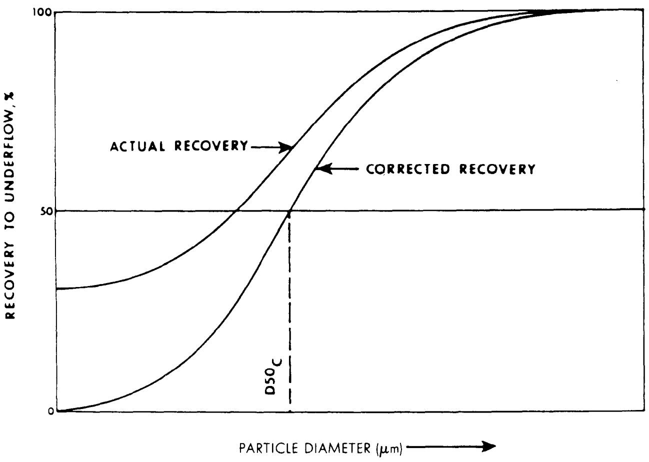

Fig. 25.3.4.5. Particle recovery curves (Arterburn, 1982).

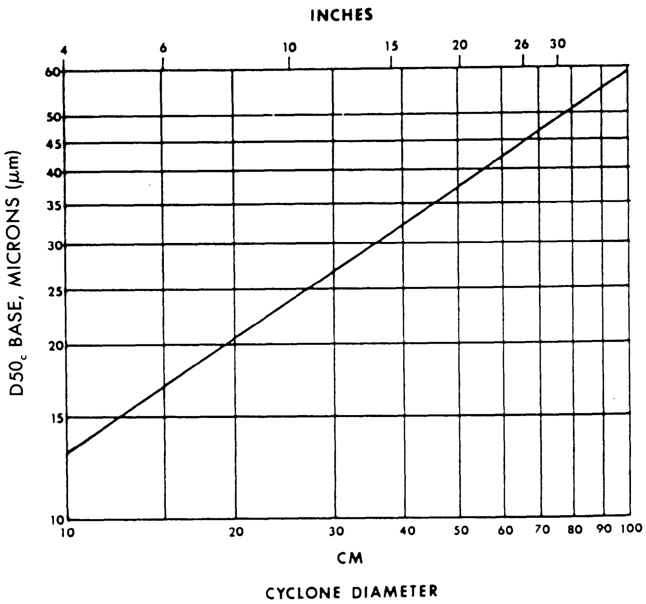

Fig. 25.3.4.6. Cyclone diameter vs. $D50_c$ for standard cyclone (Arterburn, 1982).

Factors affecting sharpness of separation include specific gravity of the solid particle mix, shape and surface character of the particles, pulp viscosity, pulp density, and size analysis vs. desired cut point.

25.3.4.2 Typical Applications

The application of cyclones is based on their ability to separate particle suspensions in accordance with particle size, density, and solids concentration. A general listing of cyclone applications is shown in Table 25.3.4.1. Cyclone operation can be divided into four types of operations: (1) coarse separations, 840 μm (about 20 mesh) maximum to 74 μm (200 mesh) minimum; (2) fine separations, desliming from 74 μm (200 mesh) maximum to 5 μm minimum; (3) desanding or desliming separations where one product is washed exceptionally clean by one or more stages of cycloning; and (4) combinations of coarse separations and fine separations using a coarse primary step followed by second-stage fine separation. Examples of cyclone applications are described here.

CLOSED-CIRCUIT GRINDING. Cyclones are widely used in closed-circuit grinding operations. While the cyclone is more flexible than certain other types of wet-classifying devices, it is less flexible as a classifier in a closed-circuit grinding system than in open-circuit classification systems. This is because the quality of the cyclone feed depends upon the rate and grindability of the ore being fed to the grinding system, and the ability of the grinding mill to grind it to the desired liberation size. Some changes can be made in the size, geometry, and operating conditions of the cyclone to modify the classification in a closed-circuit grinding system, and therefore control the circulating load, but the effect of these modifications is limited. The classification ability of the cyclone is more a function of correct mill sizing and operational control than the constraints that can be placed on the cyclone.

There is considerable controversy regarding the use of classification devices in closed-circuit grinding systems. Generally, the higher the circulating load, the narrower the size range of the final ground product. Conversely, the absence of a circulating load (i.e., an open-circuit grind) will tend to produce the largest oversize and the finest slime.

The cyclone is an inefficient classification device that tends to return to the grinding unit a good deal of finished material finer than the size required for subsequent concentrating process. At the same time, there is a heavy-media concentrating effect in cyclone classification, so that for a given liberation size, the cyclone tends to produce an overflow product slightly coarser than some other types of classifiers. It appears that this heavy-media effect can be an important feature when considering cyclones in closed-circuit grinding operations.

DESLIMING. Cyclones are used to deslime flotation feed, which minimizes reagent consumption and improves both recovery and grade. They are also used to produce a relatively slime-free sand material that is used for tailings-dam berm building and underground backfill. For such backfill, the mill tailings are often cycloned once at the mill, then again with clean water just before emplacement. The object is to maximize permeability so that water will drain freely away from the backfill sand. Desliming has been used in the production of specialty sands, such as glass sand, sand blasting sand, and filter sand, where the elimination of certain size of fine material is required.

Desliming is often used in the preparation of coal, and in some cases to wash slimy material away from coarser ore fractions that are amenable to solution leaching. Desliming, or more correctly, classification, is used on certain grades of clay in order to maintain approximate size-range consistency.

WASHING. Washing is another form of desliming. In some cases, however, it is construed as the washing out of a leach solution from ore solids, such as in the countercurrent washing of leach tailings. Cyclones are used for repeated washing of ores that are treated by flotation in order to remove reagent-consuming slimes or to remove undesirable reagents remaining from previous stages of flotation.

FEEDING AND DISCHARGING. Proper delivery of the slurry to the cyclone or cyclones is very critical. A velocity of 2 to 3 m/s (7 to 10 fps) is high enough to prevent particles from settling, even in horizontal pipe sections, but low enough to minimize wear.

If the slurry is to be distributed to several cyclones operating in parallel, extreme care should be given to the design of the slurry distribution system, and a radial type manifold is recommended. This is a system where the cyclones are fed from a central circular chamber. When properly designed, the central chamber becomes a mixing area, and the line velocity is lowered

Table 25.3.4.1. Applications of Hydrocyclones

Type of application	Kind of operation	Application: Purpose	Application: Description
Clarifying of turbid liquids	Main line	Complete clarification	Ore washing water
	Bypass line		Coal washing water
		Partial clarification	Drilling mud regeneration
			Waste water
Prethickening	Before dumps	Formation of angle of repose	Backfill tailings
	Before screens	Reduction of effluent and capacity increase	Quartz sand
	Before filters		Alumina
	Before centrifuges		Residue salts
	Before thickeners		Potash, coal
		Reduction of rake load	Mineral concentrates
Post-thickening	Effluent of screens, centrifuges, and filters	Recovery	Precipitates
			Residues
Sorting	Equal settling	Small particle size range	Decoaling of sand, leach precipitates, mineral mixtures
	Unsymetrical size distribution	Probability differences	Clays
	Heavy-media washing	Coarse fraction	Ores, coal
Washing by dilution	Re-thickening of washed slurries	Multi-stage washing	Uranium, precious metals
		Counter-current washing	Leaching
	Closed-circuit operations: coarse fractions,	Closed-circuit grinding	Ores, cement rock
	fine fractions	Crystal sizing	Alumina
Wet classifying	Classification	Classified products	Tailings dams, sand, filters, nonmetallics
	Degritting	Wear protection	Ores, sand
		Protection against oversize	
		Fine grades production	Chalk, graphite, kaolin, bentonite
	Desliming	Slime removal	Preflotation treatment
		Fines removal from coarse products	Sand
			Backfill tailings
			Drilling mud

Source: Weiss, 1985.

to approximately 0.6 to 0.9 m/s (2 to 3 fps). This will ensure that each cyclone is fed with the same slurry concentration.

To conserve space, both overflow and underflow launders are mounted inside the circle of cyclones, usually surrounding or underneath the radial feed manifold. The most economical construction usually has both the overflow and underflow launders welded around the central feed pipe, assuming it is fed from below. In some instances, the cyclones are mounted vertically, in which case an annulus for collection of underflow products is necessary.

25.3.4.3 Sizing and Selection of Cyclones

The objective in designing a cyclone is to produce an overflow from the cyclone that has a certain size distribution, defined as the percentage passing a specified micron size. An empirical relationship shown in Table 25.3.4.2 is used to relate the overflow size distribution to the $D50$ required to produce the specified separation.

Several factors influence classification. Important among them are percentage solids by volume, pressure drop, specific gravity, and viscosity. There are numerous other factors, such as inlet area and vortex finder size, as well; however, these variables are relatively minor and may be neglected for the preliminary sizing and selection.

CORRECTION FACTORS. Correction factors are applied to determine the size of separation or $D50$ (application) a cyclone can achieve:

$$D50\,(\text{application}) = D50(\text{base}) \times C_1 \times C_2 \times C_3 \quad (25.3.4.1)$$

Table 25.3.4.2. Relationship of $D50_c$ to Overflow Size Distribution

Required Overflow Size Distribution (Percent Passing) of Specified Size, μm	Multiplier (To Be Multiplied Times Size, μm)
98.8	0.54
95.0	0.73
90.0	0.91
80.0	1.25
70.0	1.67
60.0	2.08
50.0	2.78

Example: Produce an overflow of 80% passing 149 μm (100 mesh).
Multiplier from Table 1 at 80% passing = 1.25.
Size for application = 149 μm (100 mesh).
$D50_C$ required = 1.25 × 149 = 186 μm for application.
Source: Arterburn, 1982.

where $D50$ (base) is the µm size that a "standard cyclone" can achieve operating under the base conditions (Fig. 25.3.4.6) C_1 is the correction for the influence of cyclone feed concentration (percent solids by volume) (Fig. 25.3.4.7), C_2 is the correction for the influence of pressure drop (Fig. 25.3.4.8), and C_3 is the correction for the influence of specific gravity (Fig. 25.3.4.9).

FLOW RATE. The volume of water that a given cyclone can handle is related to the pressure drop across the cyclone. The relationship between the flow rate and pressure drop for several different sizes of cyclone is shown in Fig. 25.3.4.10.

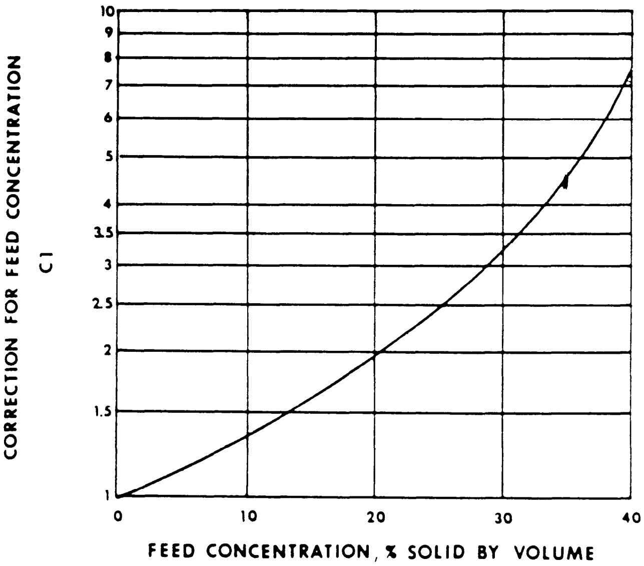

Fig. 25.3.4.7. Influence of feed concentration on separation (Arterburn, 1982).

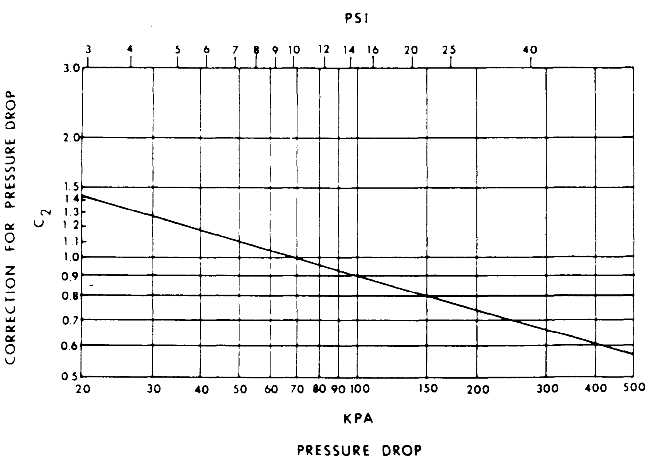

Fig. 25.3.4.8. Influence of pressure drop on separation (Arterburn, 1982).

Some correction is necessary to correct the "water" flow rate for the "slurry" flow rate. However, for preliminary estimates, a correction is not made since the capacity of the cyclone treating a slurry is greater than the one treating water.

The vortex finder size and inlet area of a cyclone also have an effect on the volumetric flow rate that a cyclone can handle. Increasing the vortex finder or inlet areas increases the capacity. Conversely, decreasing the vortex finder or inlet areas decreases the capacity.

25.3.4.4 Cyclone Sizing

Example 25.3.4.1. Select the size and number of cyclones for a rod mill/ball mill circuit where new feed to the rod mill is 200 t/h (220 tph). Both mill discharges join together at the cyclone feed sump and are pumped to the cyclones. The overflow is to be 60% – 74 μm (–200 mesh) at 40% solids by weight. The

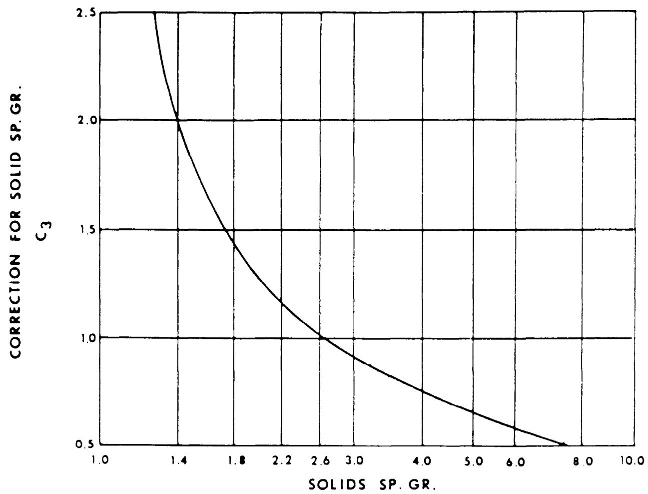

Fig. 25.3.4.9. Influence of solids specific gravity on separation (in. water) (Arterburn, 1982).

underflow becomes ball mill feed. The specific gravity of the solids is 2.9, and the estimated circulating load is 225%.

Solution.

STEP 1. Calculate the material balance from known information.

Overflow:	
Solids	200 t/h (220 tph)
Water	300 t/h (330 tph)
Slurry	500 t/h (550 tph)
Solids by wt.	40%
Sp. Gr. slurry	1.355
Slurry	102 L/s (1624 USGPM)
Underflow:	
Solids	450 t/h (495 tph)
Water	150 t/h (165 tph)
Slurry	600 t/h (660 tph)
Solids by wt	75%
Sp. Gr. slurry	1.966
Slurry	85 L/s (1342 USGPM)
Feed (sum of overflow and underflow):	
Solids	650 t/h (715 tph)
Water	450 t/h (495 tph)
Slurry	1100 t/h (1210 tph)
Solids by wt	59.1%
Sp. Gr. slurry	1.632
Slurrys	187 L/s (2967 USGPM)
Solids by vol	33.2%

STEP 2. Calculate the required $D50$ for the specified overflow of 60% passing 74 μm.

From Table 25.3.4.2, the multiplier at 60% passing 74 μm = 2.08. $D50$ (application) = 74 × 2.08 = 154 μm.

STEP 3. Calculate the cyclone diameter.

From Figs. 25.3.4.7, 25.3.4.8, and 25.3.4.9, obtain C_1, C_2, and C_3, respectively.

C_1, correction for feed density = 4.09

C_2, correction for P = 1.1

(Pressure drop assumed to be 50 kPa (7 psi)

C_3, correction for specific gravity of solids = 0.93

Calculate the cyclone diameter.

$D50$ (application) = $D50(\text{base}) \times C_1 \times C_2 \times C_3$

$154 = D50(\text{base}) \times 4.09 \times 1.1 \times 0.93$

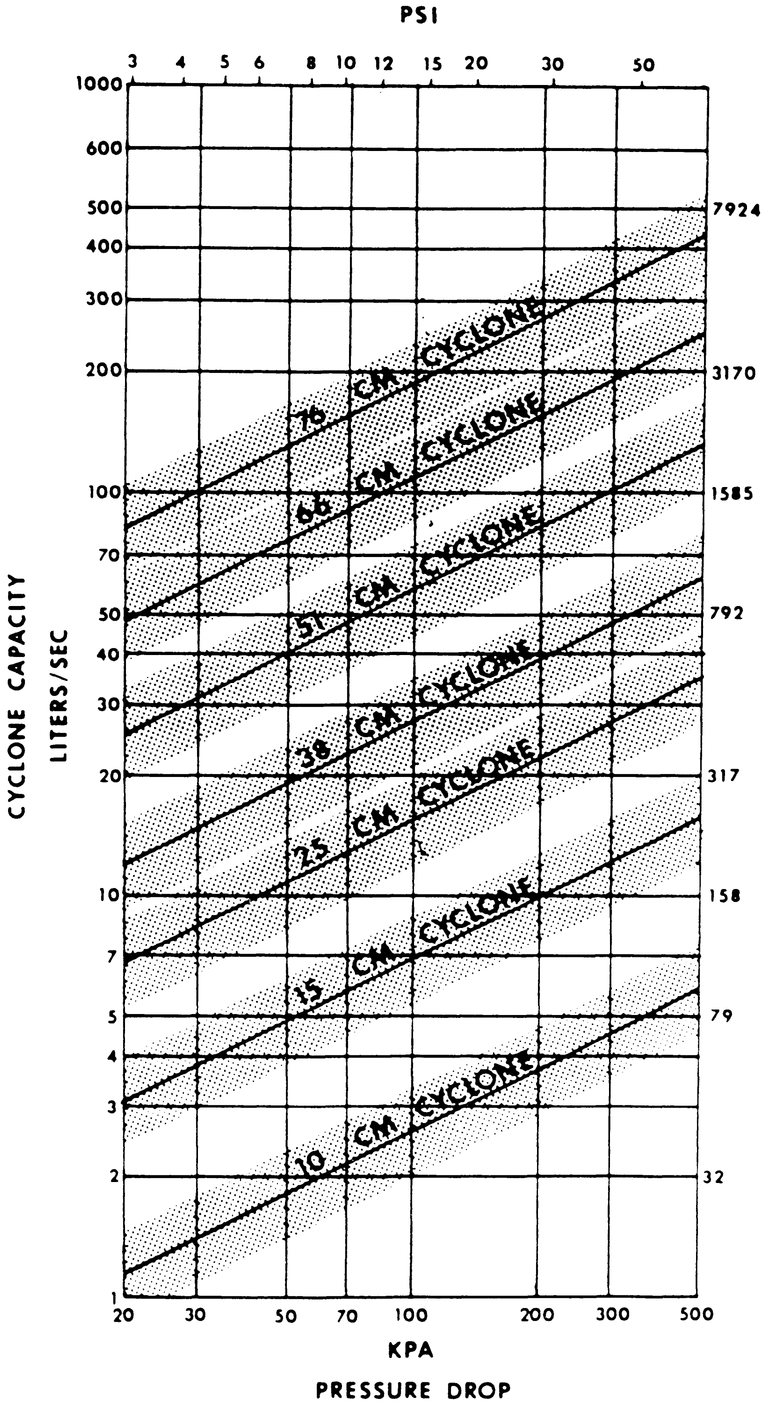

Fig. 25.3.4.10. Water capacity for standard cyclone (Arterburn, 1982).

*D*50 (base) = 37 μm

Referring to Fig. 25.3.4.6, use a 510-mm (20-in.) diameter cyclone.

STEP 4. Calculate the number of cyclones required.

From Fig. 25.3.4.10, using a 510-mm (20-in.) diameter cyclone at 50 kPa (7 psi) pressure:

Flowrate = 40 L/s (634 USGPM)

Total feed flow to the cyclones = 187 L/s (2964 USGPM)

Number of cyclones required = 187 L/s at 40 L/s per cyclone
= 4.7 or 5

STEP 5. Calculate the apex size.

Total underflow = 450 t/h (495 tph)

Underflow per cyclone = 450/5 = 90 t/h (99 tph)

From Fig. 25.3.4.4, use a 94-mm (3.75-in.) diameter apex.

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25.3.5 FLOTATION

R.S. Pizarro

Flotation is a physicochemical process for the separation of finely divided solids from one another. Separation of these (dissimilar) discrete solids from each other is effected by the selective attachment of the particle surface to a gas bubble or a liquid, whichever the case may be. The attachment of the particle to either gas or liquid phase is, in most cases, greatly assisted by a modification of the particle surface by surface active (surfactant) chemicals.

The extremely complex environment in which separation takes place is made up of three phases: (1) the liquid phase (generally water), which is chemically and physically very active; (2) the gaseous phase (generally air), which is relatively simple; and the (3) solid phase, which can be considered as infinitely variable. Gas bubbles act as balloons and provide the necessary buoyancy to carry selected minerals to the surface of the three-phase pulp where a stable froth holds the mineral, allowing it to be skimmed off as a concentrate. In the meantime, materials that have not been preferentially attached to air bubbles remain submerged and pass out of the process as a tailing.

It is safe to say that without the development of froth flotation as known today, the mining industry would never have reached its present state of development. Virtually the entire world supply of copper, lead, zinc, nickel, silver, and numerous other metals is first collected from low-grade ores as a flotation froth. Flotation has made it possible to utilize both low-grade and complex ores that would have been worthless had it been necessary to rely on the classical methods of gravity concentration. In a world that calls for ever-increasing tonnages of metals, flotation has made possible the exploitation of hundreds of millions of tons of material that otherwise would not have been economic.

Historically, the earliest attempts at flotation were concentrated on the recovery of copper, lead, and zinc. These were followed in turn by recovery of other metallic minerals including nickel, molybdenum, manganese, chromium, cobalt, and tungsten. Work was pushed on the recovery of nonmetallic or industrial minerals, and today huge tonnages of phosphate and potash are produced for fertilizers and detergents. Industrial minerals such as mica, fluorite, feldspar, beryl, and barite are upgraded by froth flotation. Fine coal, formerly discarded as waste to pollute the countryside, is recovered by flotation. In Saskatchewan, the recovery of potassium chloride (sylvite) from sodium chloride (halite) has had a tremendous impact on the provincial economy. All in all, it has been estimated that at least 100 different minerals are currently being recovered by flotation.

Outside the mining world, flotation has been applied in the chemical, paper, and food processing industries. Some typical applications in these fields are recovery of waste solids from process streams; recovery of silver from photographic solutions and film; removal of oils, fats, greases, and fibers from plant flows; removal of sulfur from rayon spinning baths; recovery of cellulose from pulp mill waste; removal of ink from waste paper; separation of wheat hulls from kernels, and recovery of bitumen from tar sands.

The effect of flotation upon industry is well illustrated by its effects on copper mining. In the face of rapidly increasing costs from 1935 to 1960, annual copper production in the United States increased from 313 to 1116 kt (345,000 to 1,230,000 tons). At the same time, the average grade of ore treated dropped from 1.57 to 0.72% Cu. Today open pit mines are operating profitably with cutoff grades as low as 0.25% Cu.

To gain an understanding of flotation theory and mechanics, it is necessary to study in depth the chemical and physical properties of surfaces as well as to establish the effect that changes in composition of the bulk phases have on the nature of the three interfaces: solid-gas, solid-liquid, and liquid-gas. With but few exceptions, inorganic solids are almost completely wetted by an aqueous phase. Accordingly, in order that flotation be able to proceed, the solid-liquid interface must be partially (and preferably completely) replaced by a solid-gas interface. The replacement of fluid by gas is contrived by the addition of suitable reagents to the aqueous phase, the resultant surface reaction leaving the solid surface with a *hydrophobic* (water repelling) film. Essentially, flotation chemistry is based on reactions at the interfaces that cause water to be displaced in favor of air to which the particle can remain attached, if and when it contacts mobile bubbles.

In order to successfully exploit the differences in surface properties of minerals, they must be ground to approximately 205 µm (65 mesh). Light minerals such as coal are sometimes separated at sizes as coarse as 1650 µm (10 mesh) under relatively quiescent conditions of froth formation. However, it has been noticed that collision, acceleration, and erratic motion all tend to shear large particles from their sites on the bubble walls. At the opposite end of the sizing scale, the surface characteristics of all particles in the pulp tend to become similar. With 3- to 5-µm particles, it is almost impossible at this time to first control and then exploit the surface properties to the degree necessary to differentiate between gangue and valuable mineral. For all practical purposes, one considers flotation limited to sizes between 205 µm (65 mesh) and 37 µm (400 mesh) (Currie, 1973).

25.3.5.1 Froth Flotation—Summary Of Process

In the flotation process, adhesion is obtained between surface coated mineral particles and air bubbles that are rising through a pulp. Enough buoyancy is provided by the bubble to cause the combination to rise and to form a reasonably stable froth that can be removed by skimming. The steps that make up the unit operation of flotation are the following:

1. The ore is ground in water to at least 297 µm (48 mesh).
2. The pulp thus formed is diluted with water to a consistency between 25 and 45% solids by weight.
3. Small quantities of surfactant chemicals are added to the pulp to modify the surfaces of specific minerals.

4. Another reagent specifically chosen to affect the mineral to be recovered by flotation is added. It gives the mineral particle an aerophilic surface (i.e., water repellent).

5. Another reagent is then added that assists in establishing a stable froth at the surface.

6. The chemically treated pulp, in a suitable container, has air introduced either by agitation or by the direct addition of low-pressure air.

7. The mineral-bearing froth that rises to the surface is skimmed off. The impoverished pulp passes on through a series of containers or cells in order to provide both time and opportunity for mineral particles to contact coursing air bubbles and to be recovered in the froth.

In any presentation of flotation theory, it does no harm to emphasize strongly that flotation is a surface phenomenon. An *aerophilic* (or hydrophobic) particle in a flotation system is one that is strongly attracted to an air interface; on the other hand, a *hydrophilic* particle in the same environment tends to stay covered with water. The differential conditions that promote separation of aerophilic particles from hydrophilic ones are surface phenomena. In this chapter, the development of flotation theory is predicated on the outstanding importance of the hydrated envelope surrounding all mineral particles in a pulp. Efforts will be made to establish the forces that attach both physicochemical and air films to the solids in the pulp. Each and every particle has both surface and mass. The chemical envelope applied to the particle develops either the aerophilic or hydrophilic character of the surface. Thus the nature of the underlying matter has, in the final analysis, very little effect on the environment. Literally, we float surfaces, and only incidentally what lies beneath the surface is recovered (Currie, 1973).

25.3.5.2 Particle Surface

It is usual to regard the changes which take place on the surface of a mineral particle in a pulp as resulting from an equilibrium state between hydrated ions (complexes) in solution and the charged surface of the particle. If the polar strength of the water molecule is greater than the attraction binding a mineral ion into its crystal lattice, then the mineral ion will migrate into the aqueous phase. The stronger the water molecule attracts, the wetter the particle surface becomes.

Surface is defined as a two-dimensional unit: length and breadth lacking thickness. However, this is not a descriptive enough picture for the study of surface chemistry. A particle that has been naturally sheared from its main mass, as for instance in a crusher or grinding mill, has exposed on its surface many partial faces of unit cell forms. The atoms of these cells are arranged according to definite patterns, and bonds, which result from the ability of a nucleus to give or receive electrons, are developed in each cell unit. There are three types of recognized primary chemical bonds: covalent, ionic, and metallic. Covalent and ionic bonds are the important forces in the flotation system.

1. *Covalent bonding* is the type of bond that exists between atoms, both of which are electron acceptors. Typical covalent bonds exist when the atoms involved are identical, as when two chlorine atoms come together and pool two electrons to form molecular chlorine. However, covalent bonding does not require bonded atoms to be identical. SO_2 and CO_2 both exhibit covalent bonding.

2. *Ionic bonding* is the type of bond produced when one atom of a pair readily accepts an electron, and the other atom readily parts with one. A transfer of an electron takes place, and a structure that is composed of ions is the result.

Ionic bonding differs from covalent bonding in that any cation will attract any anion and will repel any other cation. This really means that the bonding is not a permanent state that involves two ions; rather, it is an indiscriminate affair between any anion and any cation. Covalent bonding can be thought of as directional while ionic bonding is nondirectional. In a solid state, ionic bonds will bring about the condition of ions of alternating polarity. In the liquid state, the ionic bond does not produce an ordered array of ions, but statistically more cations surround anions than like ions surround each other. A solid or liquid whose bonds are ionic is capable of indefinite expansion in all directions, remembering always that at the solid or liquid surface there exists a force condition unlike that within the body of the substance.

25.3.5.3 Flotation Reagents

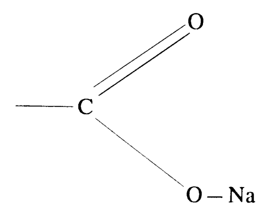
COLLECTORS. *Collectors* are the substances used in flotation to render certain selected minerals water repellent. In practically every case, collectors are organic substances. The minerals are made water repellent by the adsorption of collector ions or molecules on the mineral surface. Under these conditions, the energy level of the hydrated (water-wetted) mineral surface is reduced to a point where the formation of a three-phase contact perimeter is possible when the mineral particle contacts a bubble.

A characteristic of most collectors is a complex molecule lacking symmetry in structure and consisting of two parts that differ in properties—one polar and one nonpolar.

The nonpolar part of the molecule is a hydrocarbon radical that undergoes little or no reaction with water dipoles, and that accordingly has pronounced water repellent properties. This water repellent property is due to the presence in the hydrocarbon radical of only very weak van der Waals forces while the hydrocarbon chain bonding is strong (-C-C).

In contrast to the nonpolar part of the molecule, the polar part can react with water. Compounds which have distinguishable polar and nonpolar parts are called heteropolar. A typical heteropolar molecule is sodium oleate ($C_{17}H_{33}COONa$). The hydrocarbon radical is the nonpolar part and has practically no reactivity with water.

The polar part is



In the adsorption of these collectors on a mineral surface, the nonpolar part is oriented to the water phase and the polar part toward the mineral phase. This orientation is what actually makes the mineral surface water repellent.

Classification of Collectors—Collectors are classified according to their ability to dissociate into ions in an aqueous solution and with regard to the type of ion that produces the water-repellent effect. Where a collector dissociates into an anion and cation, the ion that is the direct cause of the water repellent action is considered as the active ion and the other as non-active. The water-repellent ion structure always includes a hydrocarbon radical, the presence of which ensures that the mineral is water repellent. However, hydrocarbons do not exist in the free state and are unable to attach themselves directly to a mineral surface. Therefore, the repellent ion structure includes a second group of atoms (in addition to the hydrocarbon) to form a link between the hydrocarbon and mineral surface. This linking group is called the solidophil group. Thus a water-repellent collector ion includes a hydrocarbon radical linked to a solidophil group. The

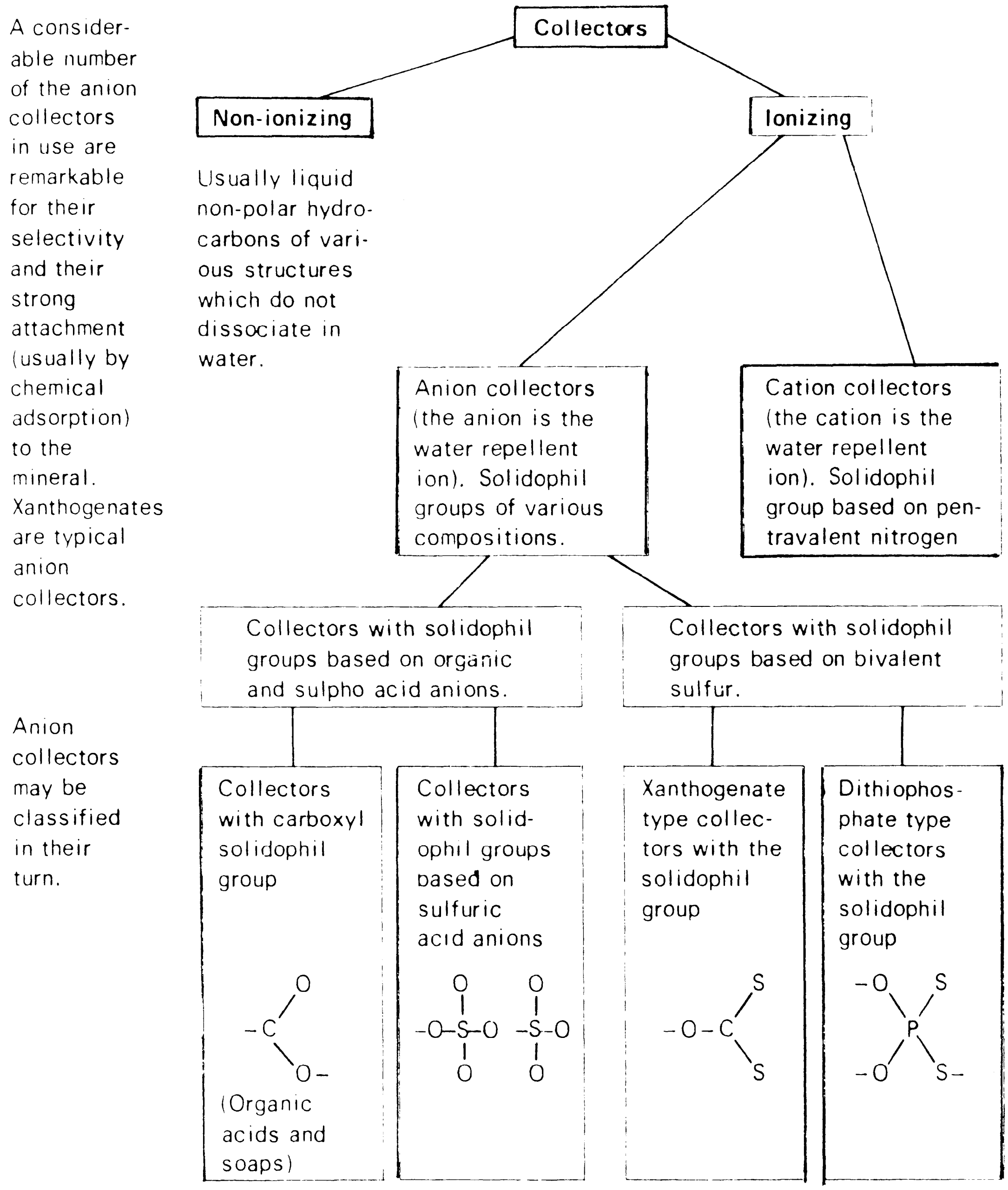
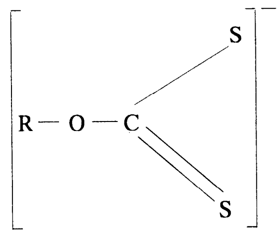


Fig. 25.3.5.1. Classification of flotation collectors (Currie, 1973).
Note: Collectors are classified according to the above with regard to ionic dissociation, anion and cation activity in relation to the mineral surface and solidophil group structure. Anion collectors are, therefore, those in which the anion (negatively charged) renders the mineral water repellent. Cation collectors are those where water repellence is secured by the cation (positively charged).

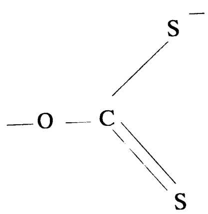
water-repellent effect is directly associated with the length of the hydrocarbon group, while the solidophil controls both the strength and selectivity of the ion attachment to the mineral surface.

As shown in Fig. 25.3.5.1, collectors can be classified according to ionic dissociation, anionic and cationic activity in relation to the mineral surface, and solidophil group structure. Anion collectors are, therefore, those in which the anion (negatively charged) renders the mineral water repellent. Cation collectors are those where water repellence is secured by the cation (positively charged).

Anion Collectors—Many of the anion collectors in use are noted for their remarkable selectivity and strong attachment (e.g., xanthate). In this type of collector, the solidophil group contains bivalent sulfur. Xanthates are salts of xanthic acid, which in turn is a product of an alcohol and carbon disulfide.

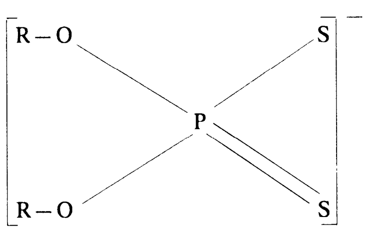
The active anion is 

and the connecting group is



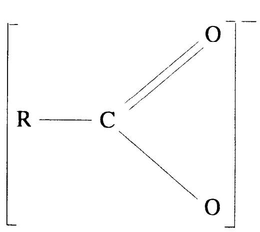
Xanthates are among the most highly selective and effective collectors for sulfide mineral flotation.

Another anion collector with bivalent sulfur in the solidophil is the dithiophosphate group (trade name Aerofloat). Like xanthate, the dithiophosphate collectors are of major importance in sulfide flotation. Dithiophosphates differ from xanthates in that they have pentavalent phosphorous in the solidophil group in place of carbon. The anion is



The cation is either Na or H, and attachment to a sulfide mineral surface is even stronger than with a xanthate.

Anions with a carboxyl solidophil group (e.g., oleic acid, sodium oleate) find application in the flotation of minerals that have the alkali earth metals as cations, that is, Ca, Ba, Mg, and Sr as well as some carbonate, oxide, and sulfate minerals. The anion is

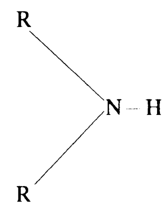


Cation Collectors—The characteristic feature of a cation collector is that the cation is the water-repelling agent, again consisting of a hydrocarbon radical and a solidophil group chemically bonded to it. The principal element in most cation collectors is nitrogen as an amino group. The anions of this type of collector are usually halides that take no active part in the reaction with the mineral.

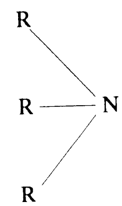
In contrast to anion-type collectors where collector attachment to the mineral is very strong, cation collector attachment is usually weak, and desorption can be made to take place by reducing concentration, that is, by diluting the pulp.

Cation collectors are used principally for flotation of silicates and certain rare metal oxides. As yet, application of this type of collector is limited.

One of the most commonly used cation collectors is the amine group, which is an organic complex derived from NH_4 and NH_3 when hydrogen atoms are replaced by aliphatic, aromatic or heterocyclic radicals. When one atom of hydrogen is replaced, a primary amine is formed with the general formula $R - NH_2$. The replacement of two hydrogen atoms gives



and the replacement of three hydrogen atoms gives



In an aqueous solution, an amine is complexed thus:

$$R - NH_2 + H_2O \rightleftharpoons [RNH_3]^+ + OH^- \qquad (25.3.5.1)$$

When an amine is used to float silica for example, the reactions are believed to be:

$$[SiO_3^-] + [RNH_3]^+ \xrightarrow{\text{ads.}} SiO_3 - NH_3R \qquad (25.3.5.2)$$

The attachment of a collector ion to a mineral surface by adsorption results in a pronounced weakening of the fields of force set up by the surface atoms (or ions) in the crystal lattice of the mineral. The bonds between water dipoles and the mineral surface have been broken and replaced by the more stable and stronger bonds set up by the collector reagent radicals. It is believed that this latter type of bond is more efficient from the aspect of usage of the surface (tension) energy. The requirements of the second law of thermodynamics have been met; namely, an overall decrease in the system free energy and adsorption is accordingly a spontaneous reaction. In these circumstances, the appearance of the water-repellent collector group in the hydrated or double-layer zone at the mineral surface (the water repellent groups having no reaction with water) reduces the stability and thickness of the hydrated layer. With the reduction of the hydrated layer, the development of a three-phase contact perimeter is possible, and an air bubble can attach itself to the mineral. For example, the strength of attachment of lead sulfide particles to an air-water interface in pure H_2O is 9.2 dynes/cm (0.63×10^{-3} lbf/ft) of the attachment line and 59.7 dynes/cm (4.1×10^{-3} lbf/ft) in a 5.5-mg/L (46×10^{-6} lb/gal) solution of potassium ethyl xanthate (Currie, 1973).

FROTHERS. *Frothing agents* are surface active substances, which by concentrating at the air-water interface help to keep air bubbles dispersed and prevent their coalescence. Frothing agents increase the stability of the flotation froth by decreasing the free surface energy of the bubble.

Frother Action—When a mineral particle surface has been made water repellent by the collector, stability of attachment of the particle to the bubble depends on frother efficiency. Frothers are heteropolar surface-active organic substances that can be adsorbed on an air-water interface.

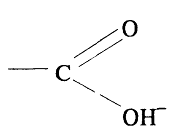
Pure liquids with nonpolar, polar, or heteropolar structures (i.e., water, kerosene, alcohol) do not produce stable froths when agitated with air. The bubbles that are produced break down quickly. However, if a small amount of a heteropolar surface-active substance is added to the water, a fairly stable froth will follow on the introduction of air in dispersed form. When the surface-active molecules react with the water, the water dipoles react with the polar groups and hydrate them; little or no reaction occurs with the nonpolar hydrocarbon grouping. The result is to force the hydrocarbon grouping into the air phase. In effect, adsorption of the frother in the surface layer of the water has taken place, with the nonpolar group being air oriented.

Consider a system in which the heteropolar molecules of a liquid of low surface tension γ in dynes/cm have been more or less evenly dispersed throughout water. If a low-solubility liquid such as pine oil (γ = 27 dynes/cm or 1.8×10^{-3} lbf/ft) has been added, the action will proceed along the following lines. All the molecules continue to move as before, so that from time to time, one of the pine oil molecules reaches the surface. Its condition is considerably different from a molecule of water at the surface. The downward force of attraction of the pine oil is slightly less than one-third of the force exerted on the pine oil by a molecule of water in the same plane; the net result is that the main body of water exerts an outward force against the pine oil. The molecules of pine oil thus find their way to an interface and stay there, resulting in the lowering of the surface tension at the interface with production of foam.

Bubble wall strength is increased by the intense reaction between the polar group of the frother and water dipoles. It requires considerably greater force to destroy these walls by bubble collision than when bubbles come into contact in the absence of a frother. If, for instance, any strong local pressure is exerted on the bubble skin, the distortion in the surface increases the area and hence decreases frother concentration in the surface. This results in a local increase in surface tension with more resistance to collapse (Fig. 25.3.5.2).

Frother Composition and Structure—The most effective frothers include in their composition one of the following polar groups:

Hydroxil: OH^-

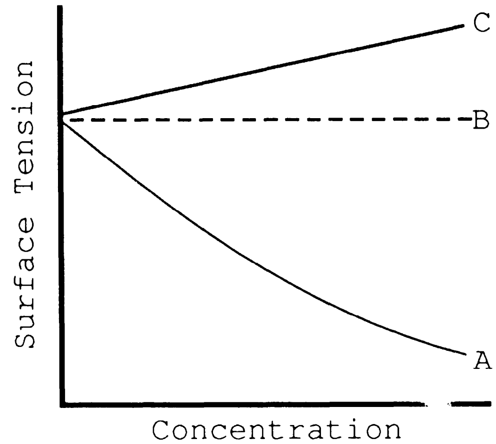
Carboxyl: 

Carbonyl: $=C=O$

Amino: NH_2^-

Sulfo: $—OSO_2\ OH$ or $SO_2\ OH^-$

The hydrophobic group consists of a hydrocarbon chain or a cyclic hydrocarbon radical. The most common frothers are



Legend:
A. Addition of surfactant with frother properties.
B. Pure H_2O γ = 72.75 dyne/cm (5.0×10^{-3} lbf/ft)
C. Addition of inorganic compound (non-frother) - electrolyte

Fig. 25.3.5.2. Changes in surface tension of water with additions of other compounds

those containing the OH group, that is pine oils, $C_{10}H_{17}OH$; cresols, $CH_3C_6H_4OH$; and alcohols such as $CH_{11}OH$. The OH group has strong hydrophilic properties and is only sparingly adsorbed on minerals, thus producing a minimum collecting effect.

Frothers must be soluble to some extent in order to get a uniform distribution and good surface active properties. Solubility will vary from 0.001% up to the 3 to 4% range. As a general rule, the presence of 1 or 2 polar groups in the molecule will give it suitable frothing properties, and an increase in the number of polar groups with the same hydrocarbon radical does not improve frothing. Frother strength is also connected with the structure and length of the nonpolar group. For instance CH_3OH does not form a froth; C_2H_5OH is a weak frother while the frothing properties improve as we go from ethyl to propyl, butyl, and amyl alcohols. To be effective as a frother, a minimum of six carbon atoms is required in the nonpolar group. If the nonpolar group becomes too long, frothing power drops off due to the decrease in solubility. Cetyl alcohol ($C_{16}H_{33}OH$) has no frothing power because of its very low solubility; introducing a sulfo group into a molecule of this alcohol makes it soluble and a very good frother ($C_{16}H_{33} \cdot SO_4H$). Thus a combination of frothers may sometimes be better than any of the individual frothers.

Frothers with a hydroxyl polar group (–OH) alcohol have no collector properties, and for this reason they are preferred over other frothers. Frothers with the carboxyl group (COOH) are both frothers and collectors.

To obtain equivalent froth, it is necessary to add the following amounts of the respective frothing agent per tonne of water:

terpineol	77 g
hexyl alcohol	63 g
heptyl alcohol	42 g
octyl alcohol	25 g

While the most important function of a frother is to form a stable froth which permits removal of a concentrate, frothers have other valuable uses in a flotation circuit. These are

1. Result in the formation of finer bubbles—that is, improve air dispersion in a flotation cell.
2. Prevent coalescence of separate air bubbles.
3. Decrease the rate at which bubbles rise to the surface of the pulp.
4. Affect collector action.
5. Increase bubble wall strength and the stability of the froth formed when mineralized bubbles rise to the surface.

Frother Application—Pine oil is a widely used frother whose frothing power is dependent on its content of terpineol, $C_{10}H_{17}OH$. Once a reliable source of supply is established, changes should be made with caution since it is not always possible to get a product of constant composition from different suppliers.

Cresol or cresylic acid, $CH_3C_6H_4{\cdot}OH$, is produced from coal tar and is a strong frother. It is toxic and burns.

Methylisobutylcarbinol, propylene glycol amyl ester (Dowfrother 250), and other high-molecular-weight alcohols are equally effective compared with pine oil and cresylic acid, and consumption is about 40% of the older style reagents. The synthetic products have a guaranteed composition that makes for ease of control in the plant.

In flotation practice, a variety of froth conditions are encountered that are influenced by the type of frother and collector used, type of ore solids present, mineral slimes, pH of pulp, etc. For instance, in the flotation of sulfide ores, a froth column 50 to 70 mm (2 to 3 in.) deep is usual, and it carries a well-mineralized layer at its surface. This type of froth will allow the removal of its mineral load down to an almost barren froth level, uncolored except for a small quantity of slime. This condition is also met with in nonmetallics, but more likely in nonmetallic flotation is the heavily flocculated oily curd held up by a thick bubble cushion. This type of froth occurs when a fatty acid, used as a collector and frother, causes a drop in froth volume as it is removed.

FLOTATION MODIFIERS. In addition to providing collector adsorption to produce floatability, surface treatment must also control selectivity through preferential adsorption of collector by the mineral to be floated and prevention of its adsorption by other minerals. This is rarely possible by use of collectors alone, since any mineral assemblage will usually contain more than one mineral that responds to a particular collector. Thus thiol collectors for sulfide minerals can develop floatability with all sulfides. Amines used to float silica from hematite ores will also float hematite if the pH is lowered. Similar conditions prevail with most collectors under practical conditions. The only circumstance permitting absolute selectivity is that of a naturally floatable mineral such as molybdenite, which in principle requires only a frother for flotation; in fact, the main problems with molybdenum ores are flotation of coarse middlings and the mechanical carryover of fine gangue. It is evident, therefore, that most flotation systems require additional reagents to control selectivity, with the effect described as depression, and the reagents, depressants.

Depressant—A *depressant* is a reagent that inhibits or prevents absorption of a collector by a mineral and hence prevents its flotation.

Activator—An *activator* is a reagent that enhances or assists adsorption of a collector.

pH Modifier—A *pH modifier* is a reagent that changes the hydrogen ion concentration of the pulp, the purpose being to either increase or decrease the adsorption of collector as may be derived. The following demonstrates how modifying agents function:

1. pH control

Lime	CaO
Soda ash	Na_2CO_3
Caustic soda	NaOH
Acid	H_2SO_4, HCl, HF

2. pH modification

With few exceptions, the effectiveness of all flotation agents depends greatly on the concentration of hydrogen or hydroxil ion in the pulp solution. One of the primary objects of flotation research is to discover the optimum pH for any given combination of reagents and ore. Most flotation plants treating sulfide ores operate with an alkaline pulp to give optimum metallurgy as well as to keep corrosion at a minimum. Very few plants operate in an acid circuit; this is usually only the case when minerals are being floated from the tailing of an acid leach.

The stability of reagent coatings can be described by their solubility products. The less soluble a compound, the less it dissociates into ions. Considering the flotation of galena using either xanthate or aerofloat as a collector and NaOH for pH control, the reactions are

$$PbS + NaOH \rightleftharpoons Pb(OH)_2 + 2Na^+ + S^= \qquad (25.3.5.3)$$

$$Pb(OH)_2 + (C_2H_5OS_2)^- \rightleftharpoons$$
$$Pb(OH) \cdot C_2H_5OCS_2 + OH \text{ (adsorbed)} \qquad (25.3.5.4)$$

Alkalinity is most commonly regulated with CaO and Na_2CO_3. NaOH can also be used but is more expensive. On account of its availability and low cost, lime is used in all circuits where the calcium ion is not objectionable, including most zinc and copper circuits. Some minerals, notably pyrite and galena have a tendency to adsorb calcium ions in preference to a collector and are thus not so easily floated in a lime pulp. This difficulty is eliminated by the use of soda ash which precipitates naturally occurring calcium ions as well as producing the necessary alkalinity.

Cationic Agents—Although most metal ions can be considered as resurfacing agents, there are only a few in common use. Copper ion is used in the flotation of sphalerite, which normally does not adsorb xanthate to any extent. The copper ion is added to the pulp as $CuSO_4$; copper ion replaces zinc ion in the mineral lattice, forming a CuS film on the sphalerite. The sphalerite then behaves as a copper mineral on which xanthate can be adsorbed and the mineral floated.

The electromotive series of metals can be used to determine the suitability of any metal as an active agent for any other particular metal. Any metal in the series will displace from solution those metals appearing below it. Copper ions would thus be suitable for activating nickel, cobalt, iron, and zinc sulfides before the addition of xanthate collector. Calcium is the most common reagent added to resurface pyrite and other ion minerals to depress them (render them completely wettable) in the presence of xanthate. Calcium and barium ions are used to resurface silica and other acidic minerals to make them floatable with fatty acid type collectors (activation).

Anionic Agents—The principal anion-active resurfacing agents include sulfides, cyanides, carbonates, and phosphates as well as certain oxidizing and reducing agents. The control of surface oxidation with heat causes considerable variation in flotation selectivity.

Oxidizing agents such as sodium hypochlorite (NaOCl) and potassium permanganate ($KMnO_4$) are effective as depressants for Cu, Fe, and Co sulfides in lime pulps. As an example, the separation of copper and molybdenum sulfides can be carried out by depressing the copper sulfide with sodium hypochlorite.

Reducing agents are also of real importance in flotation. In the flotation of galena, sodium sulfite (Na_2SO_3) or sodium bisulfite ($NaHSO_3$) is used to control excess oxidation. These compounds are particularly effective on ores containing copper minerals. Without a reducing agent, the copper mineral during grinding tends to oxidize, becoming more soluble. The resultant copper (and/or lead) ion may then activate the sphalerite, causing it to float in the lead concentrate. The addition of the reducing agent Na_2SO_3 prevents the oxidation and the resultant activation of sphalerite.

Sodium sulfide (Na_2S) is used to resurface cerussite ($PbCO_3$), cuprite (Cu_2O), azurite ($2CuCO_3 \cdot Cu(OH)_2$), and other oxide minerals, with a sulfide, to make them floatable (activated) with xanthate collectors. In small amounts, sodium sulfide may be used to reactivate pyrite, pyrrhotite, and other sulfides that have been depressed by lime or cyanide; however, the sulfide ion used in excess will depress all sulfide minerals.

Cyanides, usually added as sodium cyanide, are used for the resurfacing and depression of iron, copper, and zinc sulfide minerals. The ions of these metals form very stable complex compounds with cyanide. It has been observed that minerals with metallic ions that do not form such compounds with cyanide (e.g., Pb, Bi, Sn, Sb, and As) are not depressed by cyanide. The greater the solubility of a metal xanthate in cyanide, the less stable is the attachment of the collector to the corresponding mineral surface. The reactions are thought to be

$$ZnS \rightleftharpoons Zn^{++} + S^{=} \tag{25.3.5.5}$$

$$Zn^{++} + 2CN^{-} \rightleftharpoons Zn(CN)_2 \tag{25.3.5.6}$$

$$Zn(CN)_2 + 2CN^{-} \rightleftharpoons Zn(CN)_4^{=} \tag{25.3.5.7}$$

The $Zn(CN)_4$= radical then forms the insoluble complex with other cations, thus preventing collector adsorption. The same reactions occur with Fe and Cu, with the requirement of high CN^- concentrations in the case of Cu to achieve the desired result.

Another way to prevent activation of a mineral such as sphalerite is to add a salt of the ion that becomes displaced during activation (e.g., Zn^{++}). For instance, we could write for activation,

$$ZnS \overset{Zn^{++}}{} + Cu^{++} \rightleftharpoons ZnS \overset{Cu^{++}}{} + Zn^{++} \tag{25.3.5.8}$$

If we increase the concentration of the Zn^{++} in the solution by introducing, say, $ZnSO_4$, we can cause the above reaction to reverse and thus prevent activation.

The action of other depressants such as Na_2SO_3, $Na_2S_2O_3$, and NH_4OH is similar to the depressive action of cyanide as described above.

The depressive effect of $K_2Cr_2O_7$ and K_2CrO_4 is regarded as due to adsorption of CrO_4= anions on the surfaces of galena and pyrite in a type of chemical absorption.

25.3.5.4 Flotation Machines

CURRENT MODELS. Flotation machines now in use can be divided into four classes.

Mechanical—The most common type is characterized by a mechanically driven impeller that agitates the pulp and disperses air into it. Each impeller produces nearly ideal mixing in its vessel. It is normal to have a number of impellers in series. Within this class, there are two subdivisions:

1. Pulp flow. "Cell-to-cell" machines have weirs between each impeller; "open-flow" machines do not.

2. Aeration. "Supercharged" machines receive air from a blower; "self aerating" machines use the vortex created by the impeller to induce air.

Pneumatic—Machines in this class have no impeller and rely upon compressed air to agitate and/or aerate the pulp.

Froth Separators—In these, the incoming feed is introduced from above, onto the froth bed, and not into the pulp zone.

Column—The essential feature of column machines is a countercurrent flow of air bubbles and slurry, or of pulp and wash water.

A few machines do not fall neatly into any of these classes.

Mechanical and pneumatic machines have been in use for several decades whereas froth separators and columns are both some 20 years old. The mechanical machines have dominated the industry for many years, particularly in the West. Pneumatic types are now rarely seen, and froth separators have yet to gain wide acceptance. However, column flotation is gaining popularity based on its low capital and operating costs, particularly in copper-cleaning flotation circuits.

The dominance of mechanical machines is more a question of commercial realities than of design excellence. Major western manufacturers make only this type, and most flotation engineers are familiar with no other. The alternative designs are sold into limited geographical and/or commodity areas and encounter resistance elsewhere because of the lack of reference installations. No one wishes to be the buyer of a potentially disastrous machine. This conservatism is reinforced by an incomplete understanding of the mechanisms of flotation and a very slow rate of product development.

DESIGN DEVELOPMENTS—MECHANICAL MACHINES. A very comprehensive review of flotation machines was produced by Harris (1976). Designs created since Harris wrote, of which there are many, are here designated as "new". These and some of the more interesting older designs still available are described below, while small machines of semi-industrial or pilot plant size are, in general, excluded.

In recent years, new machines have been developed and many older designs modified. New designs are intended to have a specific technical advantage (better air dispersion, better flotation of fine particles, reduced reagent consumption, etc.), while most of the modifications to existing machines are intended to reduce overall costs, use new materials, or rectify a design defect.

Cost reduction is achieved in several ways:

1. Larger machines. As machine size increases, plant capital costs are reduced because machine weight and building volume are reduced. There are fewer banks of cells and mechanisms that imply lower costs for electricals, reagent feeding, process controls, pumps and piping, etc. Operating costs are reduced because of lower power consumption, reduced maintenance and operating labor, reduced total cost of wear parts, etc.

2. Mechanical redesign. Modern machines are generally simpler, easier to maintain, and cheaper to manufacture (in real terms) than their predecessors.

Modern materials such as cast and sprayed polyurethanes, abrasion resistant plate, ceramics, and troweling compounds can give enhanced protection against abrasion, corrosion, and the action of flotation reagents.

The first mechanical machines were of the "cell-to-cell" type. They were complicated in comparison with designs such as the Forrester pneumatic machine, and this led to the introduction of the "open-flow" concept. Over a period of some 30 years, these two types were cautiously developed to a unit capacity of about 2.8 m^3 (100 ft^3). At this point, the open-flow machines were in the ascendancy, for several reasons:

1. Simpler, cheaper construction.
2. Simpler maintenance.
3. Better suspension of coarse solids.
4. Higher unit capacity.

However, cell-to-cell machines were particularly useful in small plants and multi-stage cleaner floats (such as fluorspar) where the pumping action of the impellers permitted the transfer of intermediate flows without external pumps. A period of relatively rapid development then occurred in which open-flow machines were introduced up to 14 m^3 (500 ft^3) while conventional cell-to-cell machines developed more slowly to 5.67 m^3 (200 ft^3) and, eventually, also to 14 m^3 (500 ft^3). The most important application of these large cell-to-cell machines was in fine-coal cleaning where the users reported a significant improvement of selectivity over open-flow designs. More recently, open-flow machines have become available in sizes up to 42.5 m^3 (1500 ft^3).

Concurrent with this expansion of the more conventional mechanical units, the Maxwell cell was introduced in sizes up to 57 m^3 (2000 ft^3).

MECHANICAL CELL MANUFACTURERS. Aker Trondelag A/S is a well-established heavy engineering firm in Trondheim, Norway. The Aker flotation machine is a relatively new design developed in 1977–78. Initial sales were confined to the domestic market where the machine was well received; it has now become available elsewhere. Aker machines up to 40 m^3 (1412 ft^3) are now in use treating a variety of sulfide ores, upgrading iron ore, and separating quartz from feldspar. They are in use as roughers, scavengers, and cleaners.

The original design criteria for the Aker flotation machine were a high air-dispersion capacity, good solids suspension, and a quiescent pulp surface; these appear to have been achieved. The machine is said to be easy to maintain.

Booth Co. of Salt Lake City, UT, makes a flotation machine that is probably little known outside the western US and Peru where it is in use principally in rougher flotation. This machine is self-aerating and has a relatively shallow tank and an aerating impeller not far below the pulp surface. To promote pulp agitation, an axial flow propeller is mounted some distance lower on the impeller shaft. The tank is a truncated rectangular form with no interior baffles and a vertical plate between each impeller to restrict the pulp flow to the base of the tank.

The specific airflow through the pulp (based upon the manufacturer's figures) is noticeably higher than for any other machine, but is paid for by a high power consumption. In the event that the rate of flotation is more rapid in the Booth machine than in other designs, the higher power cost may be justified. Unlike all other machines, this airflow is unregulated.

Denver Equipment Division, Joy Industrial Equipment Co., of Colorado Springs, CO, sells a range of single mechanism D-R cells (redesigned since the Harris review) ranging from 2.8 m^3 (100 ft^3) to 36.1 m^3 (1275 ft^3). In addition, special purpose coal flotation machines up to 14.2 m^3 (500 ft^3) are available that use smaller impellers than their mineral counterparts or lower impeller speed. The company no longer manufactures double-mechanism machines.

The Denver D-R machine is used for a wide range of commodities, and for roughing, scavenging and cleaning operations. The coal machine, with its smaller impeller and lower power consumption, was developed to handle the lower-density pulps typical of the industry.

Galigher Co. of Salt Lake City, UT, has redesigned the large Agitair machines, added larger units, and stopped producing multiple-spindle machines. The new range of Agitair cells extends from 5.7 m^3 (200 ft^3) to 42.5 m^3 (1500 ft^3) and includes coal flotation machines from 8.5 m^3 (300 ft^3) to 28.3 m^3 (1000 ft^3) that use impellers from the next smallest mineral flotation machine.

The original Agitair stabilizer occupied all the outer areas of the tank. A circular stabilizer was developed, much like the stabilizer used in the Aker and OK machines, and this stabilizer is incorporated in the new designs.

The new tanks are somewhat deeper than their predecessors, having a modified truncated rectangular form and featuring a "double break" to prevent sanding. The tanks are also baffled to reduce excessive turbulence at the froth lip and have a vertical division plate between cells that constrains the pulp flow to the impeller region.

The drive head assembly has been redesigned to facilitate easy adjustment of the impeller-to-stabilizer clearance and for easier maintenance. The largest machine incorporates a shaft-mounted reducer to enable a high-speed motor and small V-belt sheave to be used.

Like the Denver D-R, the Agitair machine has enjoyed wide acceptance over the whole spectrum of commodities and cell functions.

Units made by Wemco of Sacramento, CA, have been extensively used in the flotation of copper sulfides in the rougher circuits. Larger and more efficient cells of 42 m^3 (1500 ft^3) and more recently of 85 m^3 (3000 ft^3) have met with good acceptance in the industry. Some of the features of these machines are

1. Self-induced air supply.
2. Aeration mechanism located well above the tank bottom.
3. High slurry circulation.
4. Large design clearances.

Wemco cells are equipped with the one + one rotor-dispenser design that produces a large dispersion of fine air bubbles for maximum recoveries. The large clearances between the rotor and dispenser eliminate the erosive action that prematurely wears the impeller/diffusers. The machine is simple, sturdy, and easy to operate.

Outokumpu Oy of Espoo, Finland, is a state-owned mining organization well known for its sophisticated products. When Harris wrote his paper, the OK flotation machine was available in only two sizes (nos. 3 and 16). The range has now been increased and extends up to 50 m^3 (1500 ft^3).

The OK impeller is a radical departure from the flat turbine form that is most commonly used by other manufacturers. It suggests a half grapefruit and consists of a number of narrow vertical slots communicating with the air passage in the hollow shaft. The slots taper downwards, and the top of the impeller is closed by a horizontal disk. As the impeller rotates, pulp is accelerated in the slots and expelled near the point of maximum diameter; this pulp flow is replaced by fresh pulp which enters the slots near their base where the diameter and peripheral speed are less. Thus the impeller acts rather as a pump drawing in pulp at the base of the cell and expelling it upwards. Air is blown down the shaft and is mixed into the pulp flow by the pumping action. The impeller is surrounded by a simple circular stator, similar to that used in the Aker machine. The OK impeller tapers downwards and does not become sanded in during a shut down; it can be restarted under load.

The tanks up to 8 m^3 (280 ft^3) are rectangular, with baffles. The 38-m^3 (1340-ft^3) tank is U shaped, and the 16-m^3 (560-ft^3) tank is now available in either form. The U shape gives a lighter, cheaper construction, and must also contribute very substantially to the hydrodynamic performance of the mechanism.

The OK machine is used widely in the flotation of sulfides, and in the processing of precious metals ores, iron ore, clays, tin, and tungsten ores. It is used in hydrometallurgical plants as a gas dispersion agitator.

The Sala BFP flotation machines formerly made by Sala International AB of Sala, Sweden, have now been replaced by the AS series. Harris included data on three members of the latter, the AS2-6, AS2-8, and AS4-13 of 6-, 8- and 13-m^3 (210-, 280-, and 460-ft^3) capacity, respectively. The AS4-13 has now been deleted and the series has been extended considerably. AS2 machines have two impellers, while AS4 designates four.

The Sala AS series is noticeably different from its competitors, the differences stemming from the design principles adopted. Whereas most manufacturers seek to achieve ideal mixing, with vertical flows to promote solids suspension, the Sala design aims to minimize vertical circulation. It is believed by the designers that the naturally occurring stratification in the pulp is beneficial to the flotation process. To this end, the impeller is positioned just below a horizontal hood that extends out to, and supports, the stabilizer. All slurry flow into the impeller is from below.

The impeller is a flat disc, with vertical, radial blades on both surfaces; the disc edge is cut back behind the trailing edge of the blades. The upper blades expel air that is blown down a standpipe, and the lower blades expel slurry from the central base area of the tank.

The impeller design is claimed to give two important advantages. Firstly, it disburses the air into very closely sized, fine bubbles that are correctly sized for the flotation process, particularly for fine particles. Secondly, because the air dispersion and pulp circulation functions are performed by the two impeller surfaces, they are independent.

The Sala machines are used to process a wide range of mineral commodities: base metals, iron ores, coal, and industrial minerals.

Figs. 25.3.5.3 and 25.3.5.4 show schematically the main features of the different cells described.

COLUMN FLOTATION. A significant development in flotation over the past few years has been the increasing industrial use of flotation columns, primarily in Canada. The column differs dramatically from conventional mechanical flotation machines both in design and operating philosophy, which has been a principal reason for its slow acceptance by the mineral industry. The concept was developed in the 1960s (Wheeler, 1966; Boutin and Wheeler, 1967), and since then, plant and laboratory columns have been marketed commercially by Column Flotation Company of Canada, Ltd. In 1980–81 at Mines Gaspe, Canada, three columns replaced 13 conventional cleaners in a molybdenum upgrading circuit, with superior results (Coffin and Miszcdak, 1982). Since then many of the copper/molybdenum producers in British Columbia have installed columns for molybdenum cleaning, including Gibraltar, Lornex, Highmont, and Island Cooper. There now appears to be widespread interest in flotation columns.

The column is particularly attractive for applications involving multiple cleaning stages and can upgrade in a single stage compared with several stages of mechanical cells. This results in simpler, more controllable circuits. Importantly, the column itself is well suited to computer control.

Description—A flotation column resembles a counter-current bubble column. It is typically 10 to 13 m (35 to 45 ft) high and 0.5 to 2 m (2 to 7 ft) in diameter (either circular or square in cross section). A schematic diagram is shown in Fig. 25.3.5.5, with the flow of solids and water illustrated in Fig. 25.3.5.6. Reagent-conditioned feed enters the column 2 to 3 m (7 to 10 ft) from the top and flows downward against a rising swarm of gas bubbles. Tailings are withdrawn at the column bottom. The feed and tailings flow rates are measured, and the tailings flow is controlled at a rate slightly greater (1 to 15%) than that of the feed; this is called a positive bias. The bias is provided by washwater added from a distribution of pipes located just below the lip of the column, beneath the top of the froth. Fig. 25.3.5.6 shows that part of the washwater travels down the column and part exits with the concentrate. No feed water should exit with the concentrate. Gas enters the column at the bottom through a distribution of rubber or cloth spargers. There is no mechanical agitation.

Three zones of differing rheology can be identified (Fig. 25.3.5.5). Zone 1 is the particle collection zone, a three-phase counter-current bubble column, with gas bubbles 10 to 40 mm (0.4 to 1.6 in.) in diameter, superficial gas velocity v_g up to 50 mm/s (1.0 fpm) and superficial slurry velocity v_l of 5 to 20 mm/s (0.1 to 0.4 fpm). Zone 2 is the cleaning zone, a deep (1 to 2 m, or 3 to 6 ft) packed bubble bed generated by the downward flow of washwater; washwater superficial velocity v_w is 5 to 10 mm/s (0.1 to 0.2 fpm). The interface between zones 1 and 2 is very distinct. Zone 3 is a conventional froth, extending tens of millimeters (a few inches) above the wash water pipes. Its sole purpose is to act as a medium for the transport of collected solids over the lip of the column.

The objective of wash water addition and operation with a positive bias is to prevent feed water, and therefore gangue particles, from reaching the concentrate. Liquid tracer tests at Mines Gaspe (Dobby and Finch, 1985) showed that less than 1% of the feed water reports to the concentrate. The column virtually eliminates the recovery of hydrophilic gangue particles, in contrast to conventional flotation machines where recovery by physical entrainment is inevitable (Trahar, 1981). The unique cleaning action of a flotation column is largely responsible for its ability to upgrade a fine-sized concentrate in a single step.

Column Design—At Gibraltar, columns are made of readily available pipe; the launders of the smaller-diameter columns are also made of pipe. It is important that the columns are vertical to avoid channeling.

Wash water is added through an array of perforated steel pipes. The water should be added at minimum pressure, to give a sprinkle rather than a spray. Launder spray water is also used; a nozzle delivering a fan-spray was necessary to break the froth.

The air spargers are a series of 25- to 50-mm (1- to 2-in.) diameter pipes, perforated and covered with filter cloth. These cloths last from about two months (in high-hardness water) to ten months. There is some evidence that the cloth gives large-diameter bubbles (3 mm or 0.1 in.), and alternative materials are being examined. A back-up set of spargers can be included, to be switched on when a rupture in the running set is detected. Replacement of spargers will mean a column shut-down; a running spare column may, therefore, be worth considering.

The only troublesome area has been the tail port, which is prone to high wear at the control valve. The control valve is critical; it is the only moving part of the column, and its performance is crucial in controlling smoothly the small changes in flow rate demanded to meet the bias set-point. At times, the flow rate can be low enough to sand the tails line; consequently, pipe sizes should be designed for flow velocities well above the critical.

The original tail port design, a butterfly valve in the line to a pump box, was abandoned in favor of connecting the pump directly to the column and using an in-line dart valve (Fig. 25.3.5.7). The dart and dart seat are made of silicon carbide ceramic. At the time of writing, this has not been replaced for seven months; in comparison, neoprene lasted 10 days.

This final design has been quite successful. A dart with little taper gives a smooth response to instrument control. The taper is, however, critical and must be determined for each application. Advantages of the system are that the pump uses some of the

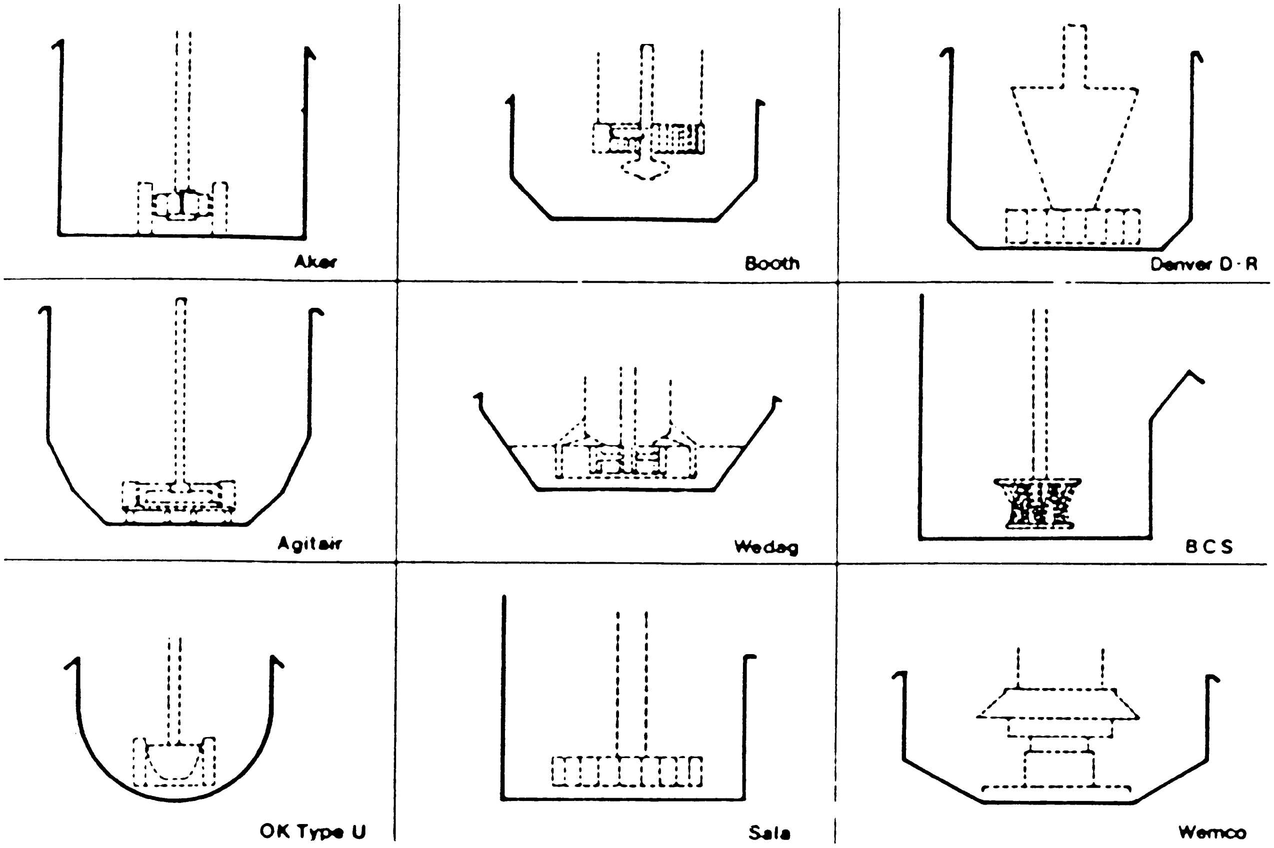


Fig. 25.3.5.3. Cell tank profiles of open-flow machines (Young, 1982).

column's static head, no spillages occur, and the flowmeter always operates under flooded conditions.

Column Operation— 1. *Axial Mixing in Collection Zone.* The collection zone is where particle recovery occurs. Therefore, mixing has a profound influence on recovery.

Residence time distribution studies, both solid and liquid, were conducted at Mines Gaspe on 0.5- and 1.0-m (18- and 36-in.) columns. The data were modeled by a plug flow dispersion model (Dobby and Finch, 1985), and two mixing parameters were extracted: (1) the mean residence time; and (2) the vessel dispersion number, $N = D/uL$, where u is particle velocity (liquid velocity plus relative particle settling velocity), L is length of the collection zone, and D is the axial dispersion coefficient. D was related to d, the column diameter in m (ft), as follows:

$$D = 0.063\, d \tag{25.3.5.9}$$

The vessel dispersion number and mean residence time can, therefore, be estimated for a given column size. When combined with a suitable flotation model (e.g., first-order with fast- and slow-floating fractions), the capacity/recovery relationship of the column can be predicted.

2. *Wash Water.* With respect to column control, the wash water has three functions: (1) to supply the make-up bias (which provides the cleaning action; (2) to maintain pulp level (i.e., zone 1, zone 2 interface location); and (3) to achieve the required water content in zone 3 to permit unhindered overflow of concentrate.

The bias must be positive ($Q_T/Q_F > 1$). The bias is controlled at a set point; commonly, Q_T/Q_F is 1.01 to 1.15. The minimum bias is desirable to minimize wash water demand and maximize residence time and, therefore, recovery. Concentrate grade does not seem to deteriorate at low bias, but close control is required.

The overflow must have a minimum water content for the solids to freely overflow, otherwise recovery drops. Overflow (i.e., concentrate) solids flowrate can be correlated with concentrate percentage solids. Since concentrate grade does not vary greatly, concentrate solids flow rate can be calculated for a given combination of solids feed rate, feed grade, and desired recovery (e.g., 98%). This is exploited in controlling column recovery.

3. *Gas Rate.* An attempt has been made to distinguish between the role of gas rate in the collection zone and that in the cleaning zone. A laboratory column 50 mm (2 in.) in diameter was modified to study the collection zone only. Mass recovery of a sample of final copper concentrate at Gibraltar was measured as a function of gas rate v_g. Fig. 25.3.5.8 shows that recovery (and therefore, the collection rate constant, k) reaches a plateau around 17 to 25 mm/s (0.3 to 0.5 fpm). This effect can be understood from the dependence of k on v_g, bubble diameter d_b, and bubble collection efficiency E_k, namely:

$$k \propto \frac{v_g\, E_k}{d_b} \tag{25.3.5.10}$$

Fig. 25.3.5.8 shows that, in contrast, the plant typically operates in the range of 25 to 40 mm/s (0.5 to 0.8 fpm). This may in part reflect coarser bubbles (with lower E_k) in the plant, but the suggestion is that extra air is required to carry material through the cleaning zone.

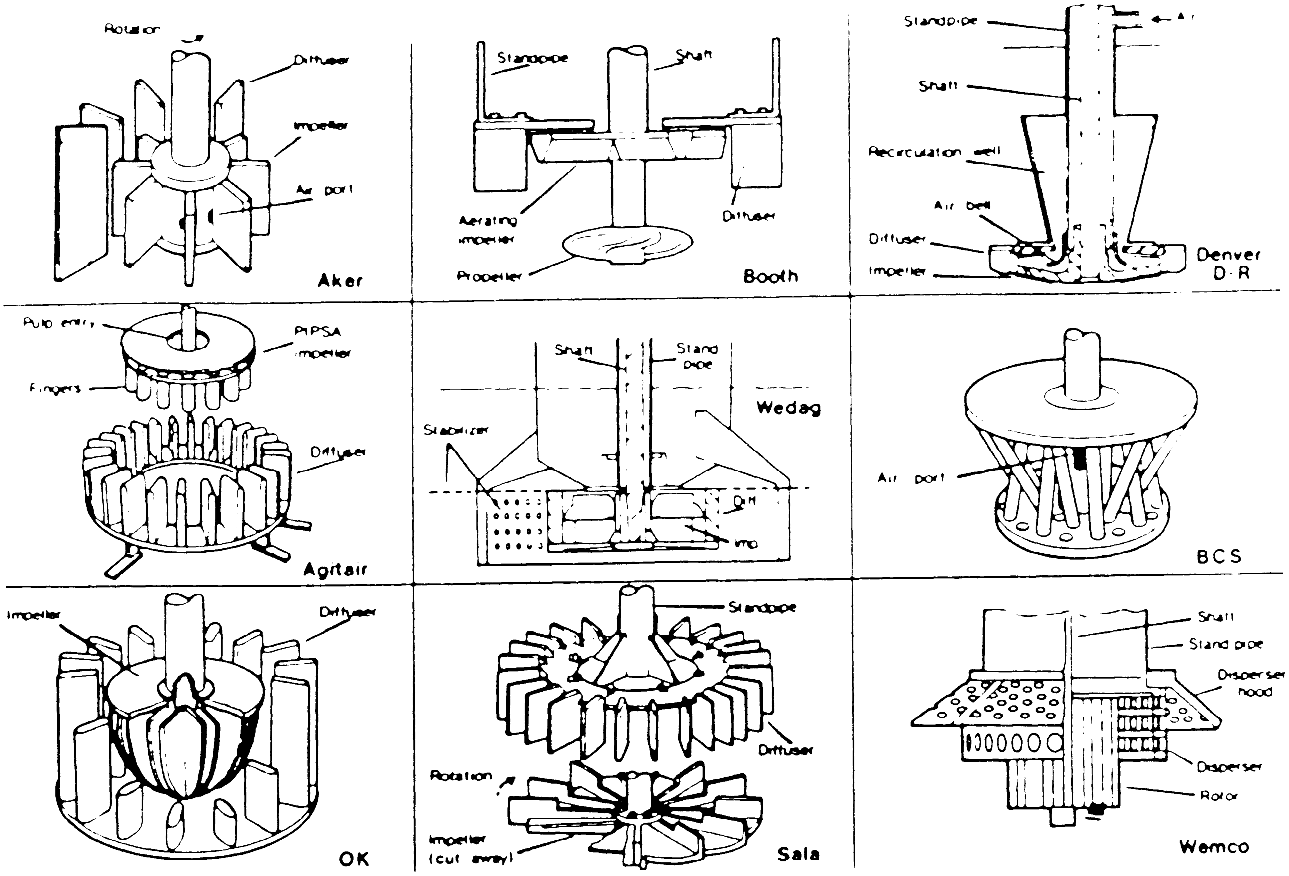


Fig. 25.3.5.4. Impeller mechanism details of open-flow machines (Young, 1982).

4. *Gas Holdup.* Mines Gaspe reported that recovery could be correlated with gas rate (Coffin and Miszcdak, 1982); this was not found at Gibraltar. This may be because bubble size is also a factor; a change in frother would change bubble size and alter the dependence of recovery on gas rate.

A measure, which includes both gas rate and bubble size, is the gas holdup, the fraction of air in the air/slurry mixture. Holdup varies directly with gas rate and inversely with bubble size (smaller bubbles rise more slowly giving longer retention time and increased holdup).

The maximum holdup is about 20 to 24%. Above this, solids start to accumulate in the column and eventually exit to tailings. This condition is identified by rapid changes in holdup (2 to 4%/min). This phenomenon is probably related to the transition from a homogeneous bubbling regime to a churning regime (Shah et al., 1982).

Instrumentation and Control—Manual control of a column is not feasible, instrumentation and some degree of automatic control is a necessity.

The control configuration under test at Gibraltar is shown in Fig. 25.3.5.6. There are three control loops: bias control, level control, and the wash water/gas holdup interaction.

Experience has shown that the bias and level control loops are the minimum requirement for stable operation of the column. The last loop is designed to control column performance. In comparison to conventional cells, the goal of performance control seems more attainable.

25.3.5.5 Concentrator Flotation Circuits—Commercial Applications to Copper Ores: San Manuel Concentrator

The San Manuel plant came into production in 1955 as a state-of-the-art, 36,000-t/d (40,000-tpd) operation. After crushing at mine and mill-site facilities, 13 grinding sections prepare the ore for flotation. Each grinding section has an open-circuited rod mill, whose product is sumped and pumped to cyclones that operate in closed circuit with two downstream ball mills. The feed to the grinding circuit has a Bond work index of 14 kWh/ton.

The ball mill cyclones overflow a 28%-solids pulp at 8% + 65 mesh and 58% − 200 mesh. Bulk Cu-Mo flotation roughers are arranged in eight lines, each of which produces a concentrate that grades 9 to 12% copper and a finished tailing.

The rougher concentrate combines with regrind mill discharge, and the pulp is pumped to regrind cyclones, which overflow a pulp for two-stage cleaning and underflow a spigot product for regrinding. The bulk cleaner concentrate is typically 76 to 78% –325 mesh and assays about 29.5% Cu and 1% Mo. This bulk Cu-Mo concentrate is thickened and conditioned with sodium cyanide and sodium hydrosulfide for flotation of molybdenite from the copper sulfides.

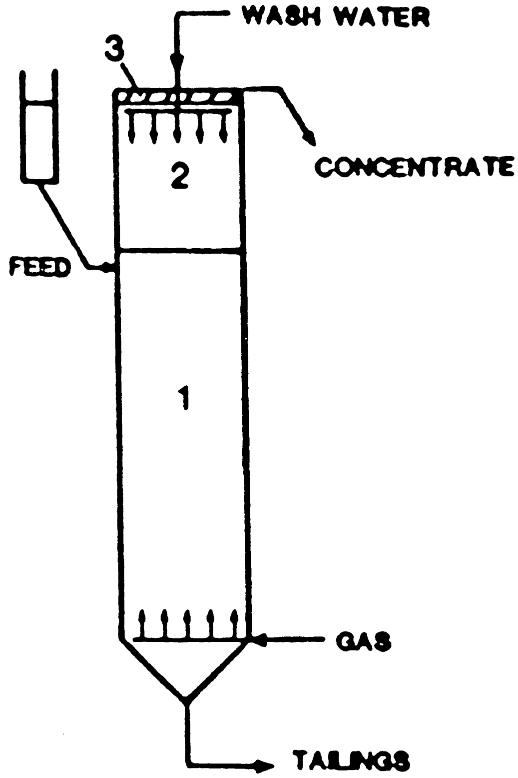


Fig. 25.3.5.5. Schematic illustration of flotation column. 1) Collection zone; 2) cleaning zone; 3) thin froth layer (Dobby, Amelunxen and Finch, 1985).

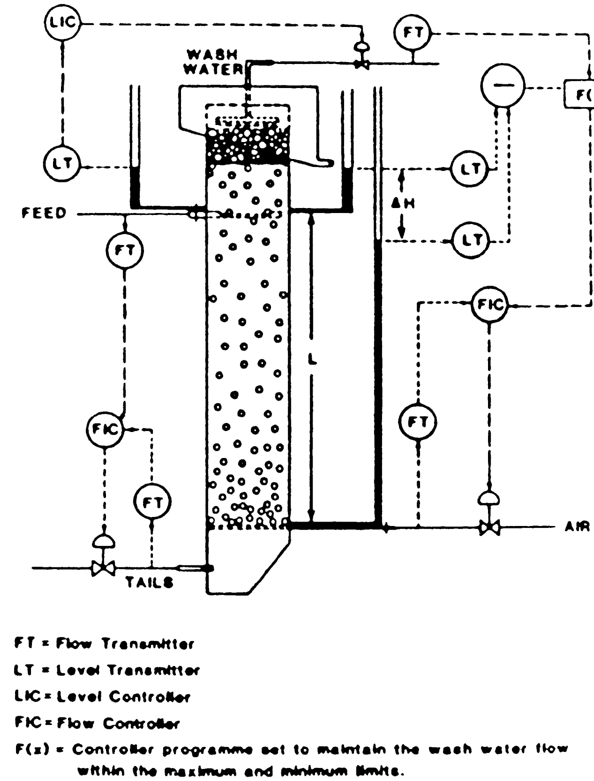


Fig. 25.3.5.6. Schematic diagram of column flotation unit showing flows of solids and water, plus control circuit loops (Dobby, Amelunxen and Finch, 1985).

Rougher flotation consists of five identical sections having a total of 780 Galigher 1.1-m^3 (40-ft^3) cells. The Galighers are belt driven, with one 11.2 kW (15 hp) motor for every two cells.

The 780 Galigher cells in rougher sections one through five are being replaced by 10 Maxwell 56.6-m^3 (2000-ft^3) cells and 80 Wemco 8.5-m^3 (300-ft^3) machines. Included in this project is the installation of on-line X-ray analytical equipment and mass-flow rougher control.

Downstream from the eight parallel flotation sections, eight ball mills, each functioning in closed circuit with cyclones, regrind the rougher concentrate to 76 to 78% –325 mesh. The reground concentrate is cleaned in two stages.

The original cleaner circuits are aligned in eight sections. Each section is equipped with two parallel lines of first cleaners, composed of 14 Agitair 12-m^3 (40-ft^3) cells per line, followed by a refloat of this first cleaner concentrate in an eight-cell final cleaner, also composed of the same Agitair cells.

COLUMN FLOTATION. In April 1986, two column flotation cells, operating in parallel, were installed to replace one of the eight conventional two-stage cleaner lines. Both cells were 12 m (40 ft) high; one was 1.5 m (5 ft) in diameter and the other 1.8 m (6 ft) in diameter. On the basis of operating experience with these cells, the column cells in the remaining seven lines will be standardized at 2-m (6-ft) diameter.

The new column cells replaced 28 first cleaners and eight final cleaners, each 12 m^3 (40 ft^3) in capacity. A test program was then outlined that involved evaluating the cells over a wide range of operating conditions. The major variables examined were the effects of varied pulp level, wash water, and airflow rates. Once an optimum range was established for each variable, automatic control was investigated. Clingan and McGregor (1987) discussed this column-cell experience but cautioned that the San Manuel experience may not be typical of concentrators at other mines, because the San Manuel ore body and mining method lend themselves to the production of a very consistent feed.

The San Manuel investigations involved a three-loop control strategy, as described by Amelunxen (1985). He also notes the need for enough column-cell wash water to replace water in the gangue that in conventional cells would be carried in the froth, a situation termed a positive bias. Another way of describing positive bias is to say that the tailing flow rate is greater than the feed flow rate. A conventional flotation machine always operates with a negative bias.

With other variables constant, a range of wash water flow rates from 0 to 10 L/s (0 to 160 gpm), equivalent to a bias of 0.85 to 1.25, was evaluated for each cell. As expected, the results indicated that the concentrate grade decreased below a bias of 1.00. No change was noted in copper or molybdenum recovery from a bias of 0.85 to 1.15, but above 1.15, molybdenum recovery decreased. No difference in metallurgical results was detected between a bias of 1.00 and 1.15, equivalent to about 3.8 to 7.6 L/s (60 to 120 gpm).

Airflow rate proved to be the most important operating parameter in the overall performance of the flotation columns. The ratio of airflow rate to cell volume, expressed as liters/min to liters (gallons/minute to gallons), can have dramatic effects on recovery and concentrate grade and was a valid way of correlating cell performance on the two columns of different size.

Because the airflow rate is the most important operating parameter, the sparger is the most important fixture on the column cell. The importance of generating small bubbles cannot

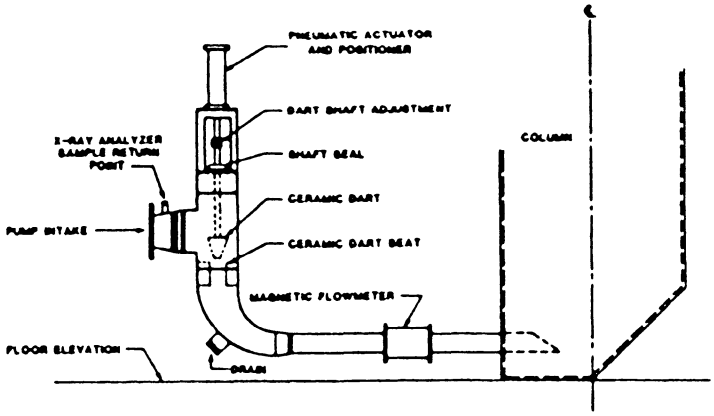


Fig. 25.3.5.7. Current design for tails port on column flotation unit (Dobby, Amelunxen and Finch, 1985).

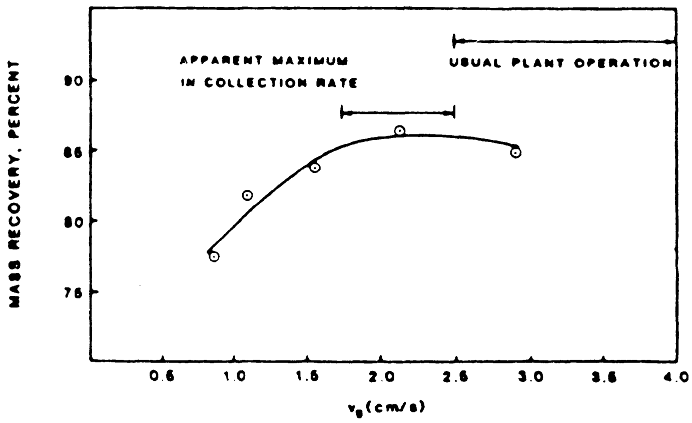


Fig. 25.3.5.8. Mass recovery of copper concentrate vs. superficial gas velocity in column flotation cell (Dobby, Amelunxen and Finch, 1985). Conversion factor: 1 in./sec = 2.54 cm/s.

be overstated (Clingan and McGregor, 1987), as these are more effective in contacting the copper mineral particles.

COMPARING CELL RESULTS. In terms of copper recovery and grade, the San Manuel authors report that results attained with single-stage column cleaners and two-stage conventional cleaners are about equal. Column cells produce a concentrate grading 29.83% Cu at a sulfide copper recovery of 90.36%. Conventional two-stage cleaners produce a concentrate grading 29.99% Cu at a sulfide copper recovery of 90.12%.

However, they observe that the copper concentrate grade at San Manuel is directly related to the degree of liberation in the regrind circuit. When comparing the conventional two-stage cleaners to the one-stage column cleaners at equivalent degrees of liberation, as measured at 44 μm (325 mesh), the one-stage column cleaner produces a higher concentrate grade.

Clingan and McGregor (1987) note that the molybdenum recovery in the bulk CU-MO column cell concentrate increased by a minimum of five percentage points compared to recovery by conventional two-stage cleaners. Such a remarkable gain greatly enhances the economics of the installation. While the San Manuel authors say that the reasons for the gain are now unknown, they point out that column cells are widely considered to be superior to conventional cells in the recovery of finer particle sizes. Since the bulk of the molybdenum losses at San Manuel are in the finer fractions, the results there may add some credence to this generalization.

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25.3.6 GRAVITY CONCENTRATION

V.P. Kenyen

Gravity concentration is the method of separating particles of mixed sizes, shapes, and specific gravities by using the force of gravity and/or centrifugal force. The nature of the process is such that size and shape classification are an inherent part of the process. It is not the difference in specific gravity that causes the differences in movement of particles but their weight in a common medium. A particle of high specific gravity and small size will have the same movement in a given fluid medium as another particle of lower specific gravity and larger size. Since a mixture of broken (mineral) grains has an extended size range, size control must be imposed so that the differential motion of the particles reflects the differences in specific gravity. (For size control, see 25.3.3 and 25.3.4 on screening and hydrocycloning.)

Gravity concentration provides a simple, high-capacity, low-cost means of concentrating ores. It works best for rich ores, for ores showing a coarse liberation size, for large volumes of ore with low-value minerals, for placer deposits, and as a preconcentration step.

Simultaneously achieving both a high recovery and a high grade is usually not possible in gravity concentration. Therefore, gravity concentration is commonly used as a rougher to achieve high recovery, followed by another type of process such as flotation (see 25.3.5) to make a high-grade concentrate. Lode ores are most difficult to process. During the comminution step, some fines are always produced, and these very fine particles are not effectively recovered by gravity concentration techniques. The efficiency of gravity methods for concentrating most heavy mineral placer deposits is automatically high since nature has already removed the nonrecoverable fines.

Gravity concentration is usually not competitive with either flotation or hydrometallurgy for the treatment of low-grade ores in which the valuable minerals are locked intimately with gangue in fine grains. The main problems preventing widespread adoption of gravity concentration techniques are twofold: (1) the need to separate, effectively, particles with small specific gravity differences; and (2) the need to develop efficient, high-capacity devices for the treatment of mineral fines (–75 μm).

The criterion proposed by Taggart for determining the applicability of gravity separations is given below. The relation is based on the equal settling ratios of two particles:

$$\text{concentration criterion} = (\rho_h - \rho_m)/(\rho_l - \rho_m) \qquad (25.3.6.1)$$

where ρ_h is the specific gravity of the heavy mineral, ρ_l is the specific gravity of the light mineral, and ρ_m is the specific gravity of the fluid medium.

The size range of applicability for gravity concentration, assuming viscosity effects can be ignored, is then obtained from Table 25.3.6.1. When the criterion is less than 1.25, gravity

Table 25.3.6.1. Values of Concentration Criterion

Criterion	Size of applicability
2.5	Separation easy down to fine sand, 74 μm (200 mesh)
2.5–1.75	Separation effective to 149 μm (100 mesh)
1.75–1.50	Separation possible to 1.68 mm (10 mesh), but difficult
1.50–1.25	Separation possible to 6.35 mm (¼ in.), but difficult
1.25	Relative processes impossible but absolute processes (e.g., heavy media) possible

Source: Aplan, 1985.

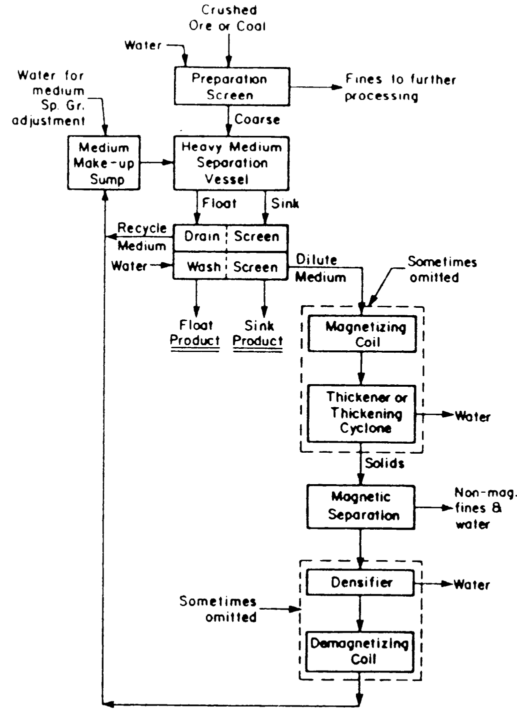


Fig. 25.3.6.1. Typical flowsheet for heavy media separation (Aplan, 1985).

concentration based on the differential particle velocity is not possible. However, concentration is possible by heavy media techniques using either true liquids or suspensions of solids such as ferrosilicon.

A number of *gravity separation* devices are available in the market. Primary among them are jigs, tables, spiral classifiers, and heavy media cyclones. Numerous other devices are available as well. Some of these are discussed in detail below.

25.3.6.1 Heavy Media Separation (HMS)

Heavy media concentration is an effective method of separating two or more minerals of different specific gravities. This requires a medium whose specific gravity is intermediate between the specific gravities of the two minerals to be separated. Other methods of gravity separation (e.g., jigs, tables, spirals) require close particle sizing and a specific gravity difference of at least one unit between the minerals to be separated. However, the HMS technique can be used over a wide size range and a narrow gravity difference.

Commercial separation in the specific gravity range of 1.3 to 3.8 is possible. The particle size treated ranges from 150 to 200 mm (6 to 8 in.) down to 3.3 mm (0.13 in. or 6 mesh). Particles greater than about 13 mm (½ in.) are usually treated in a "static" bath; a heavy media cyclone is generally used for particles less than 13 mm (½ in.) in size.

The basic steps involved in heavy media processes are (1) feed preparation, (2) heavy media separation, and (3) media removal and recovery. Many flowsheet variations are possible, and a typical flowsheet is given in Fig. 25.3.6.1.

HMS offers the following potential advantages:

1. Rejection of the waste at a coarse size, which causes less environmental problems.
2. Relatively low capital requirement.

Table 25.3.6.2. Materials Used for Heavy Medium

Material	Density, g/cm^3	Typical Size Range Used, μm
Shale*	2.0–2.6 (highly variable)	—
Quartz	2.65	Nominal 417 $\times$ 147 (35 $\times$ 100 mesh)
Barite	4.5	74 (200 mesh)
Magnetite	5.18	44 (325 mesh)
Ferrosilicon (15% Si)	6.8	208 (65 mesh)
Galena	7.8	208 (65 mesh)
Lead	11.35	—

* Used in some cyclonic heavy medium separations
Conversion factor: 1 lb/ft^3 = 0.01602 g/cm^3.
Source: Aplan, 1985.

Table 25.6.3.3. Types of Heavy Media Separators

Type of Equipment	Feed Size (mm) Top	Bottom
Quasi-Static Vessels	150–600	6
Wemco Drum, trough, Aitkens Classifier, Drew Boy Drum, O.C.C. Vessel, etc.		
Static Cones	100	2.5
Dynamic Separators	50	0.5
Heavy Media Cyclone DynaWhirlpool		

Source: Aplan, 1985. Conversion factor: 1 in. = 25.4 mm.

3. Relatively low operating and maintenance costs.
4. Ability to make sharp separations.
5. Ability to change the specific gravity of separation quickly to meet changing conditions.
6. Ability to remove products continuously.
7. Ability to treat a broad range of feed sizes.
8. Ease of start-up and shutdown without loss of separating efficiency.
9. High capacity with the use of relatively little floor space.
10. Supply of uniform feed to a subsequent concentration step.

DENSE MEDIA SEPARATION. The most important means of making a dense medium separation is to use a suspension of fine, heavy particles in water as a pseudo fluid. The use of this heavy medium to effect a separation between two particles of differing density is one of the major means of gravity concentration, and its use has been growing rapidly.

The settling rate of a relatively coarse particle (above about 1.6 mm or 10 mesh) settling in a suspension of fine particles (both assumed spherical) is given by Newton's equation:

$$V_m = d \times g \times [(4 \times \rho_s - \rho_m)/(3 \times Q \times \rho_m)]^{1/2} \quad (25.3.6.2)$$

where V_m is the maximum settling velocity in cm/s; Q is the coefficient of resistance, 0.4 for a sphere, and 2 to 3 for "blocky" particles; ρ_s is the specific gravity of the solid to be separated in g/cm^3; ρ_m is the apparent density of the suspension in g/cm^3; d is the particle diameter in cm; and g is the acceleration due to gravity (980 cm/s^2). Under the appropriate conditions, the density of a true fluid may be replaced by the apparent density of the suspension ρ_m in this equation.

The above equation does not show the effect of viscosity. However, if the medium becomes too viscous, it will reduce the particle settling rate and hence the efficiency of separation.

HEAVY MEDIUM SOLIDS. Table 25.3.6.2 lists materials that can be used as the heavy medium. The three most common media in use today are quartz, magnetite, and ferrosilicon, though galena was used extensively in the 1930s and 1940s for ore separations.

Magnetite is normally used for coal preparation, and ferrosilicon (FeSi) is used for ore separation. This is the 15% silicon grade that is an eutectoid composition. There are two types of FeSi—the ground FeSi manufactured in the United States and the atomized FeSi manufactured in Europe. Ground FeSi is used for separations at densities up to 3.2 g/cm^3 (200 lb/ft^3). If higher operating gravities are desired (up to 3.8 g/cm^3 or 240 lb/ft^3), atomized FeSi is used.

Process and Equipment—There are at least 74 heavy media separators that have been used at one time or another. These are divided into three basic groups: static separators, quasi-static separators, and dynamic separators. The effective size range of these separators is given in Table 25.3.6.3.

The type of equipment most suited for a given process is dependent upon the feed size and efficiency of the separating vessel. Pilot plant testing will be very helpful in determining the latter parameter.

Feed Preparation—The first step in heavy media separation is to prepare the feed by wet screening. This is done to (1) size the ore into different fractions for subsequent treatment by various concentrating devices, (2) remove the slimes that increase the viscosity of the medium, thus resulting in poor efficiency, and (3) prewet the ore or coal.

For coarse screening, woven wire is generally used, but for relatively fine screening, wedgewire deck should be considered. For sizing sticky, hard-to-screen ores, which have a tendency to blind the screens, flip-flow polyurethane screens are recommended. These have been successfully used for screening materials 1 to 30 mm (0.04 to 1.18 in.) in size. Because of the trampoline-like motion of the elastic screen panel, this type of screen is almost non-blinding.

Medium Recovery—The medium is recovered from the sink and float fractions by the drain and wash screens. DSM screens, sometimes referred to in the coal industry as sieve bends, are preferred as drain screens as these are very efficient in removing the fine medium. Where blinding is a problem, polyurethane screens are used.

When a single screen has the capacity to handle both the sink and float fractions, a divider is installed along the length of the screen to keep the products separate.

The medium recovered from the drain screens is returned to the medium sump for recirculation in the medium circuit. The medium removed by washing is returned to the dilute medium sump as it is too dilute and, in many instances, is contaminated with fine impurities. Therefore, prior to return to the medium sump for reuse, it should be cleaned and thickened. Wet magnetic separators, with permanent magnets, are normally used for cleaning and recovery.

Single-drum magnetic separators are common; however, when the medium is dilute and the feed volume is large, multiple drum separators should be used. For optimum results, the magnetic separators are fed at 30 to 35% solids by weight.

The efficiency of the magnetic separator depends on the magnetic susceptibility of the media and the rate at which the unit is fed. Satisfactory results are obtained when the separators are fed at the proper feed rate.

Magnetic separators are available in 762-, 914-, and 1219-mm (30-, 36-, and 48-in.) diameter with magnet widths to 3 m (10 ft). The respective capacities are 48, 63, and 85 m^3/h/m (515, 675, and 915 ft^3/hr/ft) of drum width.

Table 25.3.6.4. Typical Media Consumption in Heavy Media Separators

Type of Vessel	Media Loss (g/t)
Drum	150
WEMCO Cone	130
Cyclones	300
DynaWhirlpool	435

Conversion factor: 1 oz/ton = 31.25 g/t.

The importance of designing an adequate medium recovery circuit cannot be overemphasized. Attention should be given to details so that medium losses can be kept at a minimum. With increasing medium costs in mind, designers have been developing more sophisticated recovery circuits.

Media losses vary considerably. In general, higher losses are associated with the finer sizes of material processed and the fineness of the medium itself. Losses on the static-type separators are usually low. On cyclones or DynaWhirlpool (DWP) plants, the losses are generally higher. Table 25.3.6.4 shows media losses experienced in heavy media operations.

HEAVY MEDIA CYCLONE. Although cyclones were originally developed for use as classifiers or thickeners, it was later found that they could also serve effectively as heavy media separators.

As the medium and mineral particles are fed through the feed orifice, a vortex forms in the cyclone with a hollow air core extending from the overflow to the underflow orifice. Under the influence of centrifugal force, high specific-gravity particles move through the medium to the wall of the cyclone and descend in a spiral flow pattern to the underflow orifice. Those particles in the feed stream, lower in specific gravity than the feed medium, follow the major portion of the flow to the center of the core where they are caught in the high-velocity, upward, central current and are carried out through the overflow orifice.

In heavy media cyclone operation, the specific gravity of the medium is the main control within acceptable ranges of viscosity. However, other variables, such as cyclone size, feed pressure, cone angles, and orifice size, can also affect results. The principal design variables and their effects on operation follow:

Cyclone Diameter—Heavy-media cyclones are used in various sizes from 50 to 610 mm (2 to 24 in.) in diameter. The approximate volumetric capacities of these cyclones are shown in Table 25.3.6.5.

The performance of any separating device can best be shown by partition, or error, curves. Typical curves for the performance of a heavy media cyclone processing coal at two separating gravities are shown in Fig. 25.3.6.2. These curves represent the distribution to refuse of each specific gravity increment of the feed. They are drawn by plotting the recovery to refuse of each specific

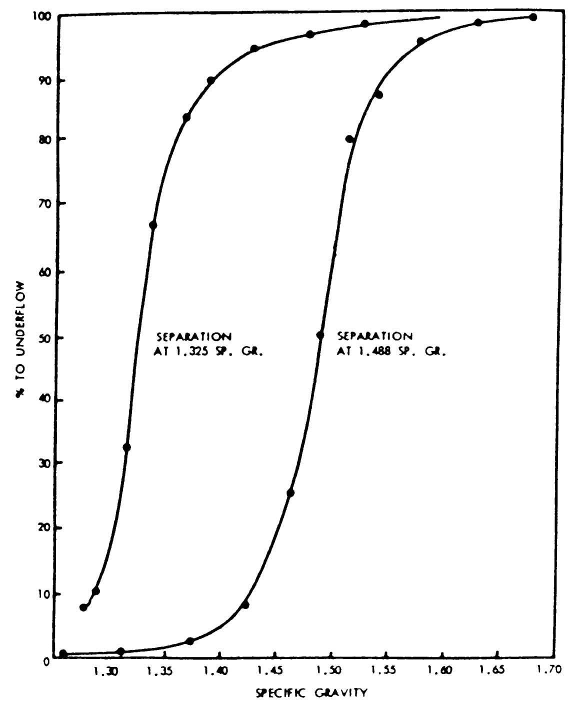


Fig. 25.3.6.2. Performance curves for heavy medium cyclone (Aplan, 1985).

gravity fraction of the feed against the midpoint of the specific gravity range involved. As a standard means of comparing the sharpness of separation of any process, the ecart probable (EP) method can be used. The ecart probable, or probable error, is defined as one-half the specific gravity interval corresponding to recovery values of 25 and 75%. The specific gravity of separation is the specific gravity point that corresponds to 50% recovery and is generally higher than the specific gravity of the medium.

DYNAWHIRLPOOL (DWP). Similar to the heavy media cyclone, the DynaWhirlpool is a stationary-body-type device used for making specific gravity separation, but it differs from the cyclone in its design and critical operating variables.

Design Features—The DynaWhirlpool vessel, shown in Fig. 25.6.3.3, consists of a hollow cylinder fitted with cover plates having central openings. The vessel has tangential orifices at either end of the cylinder. The medium is admitted under pressure through the bottom tangential orifice, and a vertical flow pattern is established in the cylinder. The flow discharged through the central bottom opening is lower in specific gravity

Table 25.3.6.5. Effect of Cyclone Diameter on Capacity

Diameter, in.	Material processed, tph			Volume processed at 8–10 psig, gpm	Orifice diam, in.		
	1.35 sp gr	2.0 sp gr	3.0 sp gr		Feed	Overflow	Underflow
2	0.5	0.7	1.0	10	½	1	⅜
8	7.5	11.0	17.0	150	1½	2½	1
14	25.0	37.0	56.0	450	4	6	5
20	50.0	74.0	110.0	750	6	8	6
24	70.0	103.0	156.0	1,000	8	10	7

Conversion factors: 1 in. = 25.4 mm, 1 psi = 6.8948 kPa, 1 gal/min = 3.79 L/min, 1 tph = 0.9072 t/h.
Source: Aplan, 1985.

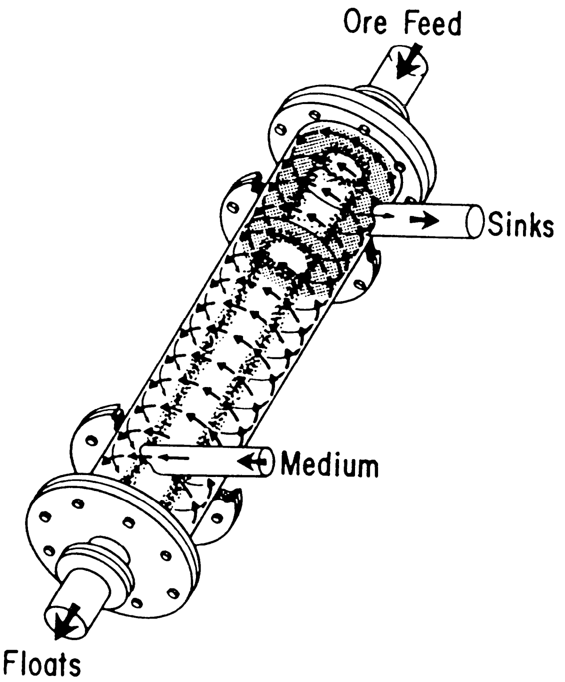


Fig. 25.3.6.3. Cutaway view of DynaWhirlpool vessel (Aplan, 1985).

than the feed medium. The material discharged from the upper tangential opening is higher in specific gravity than the feed medium.

By placing the cylinder in a slightly inclined position and selecting a smaller orifice for the top central opening than the bottom central opening, the device can be operated without medium discharging through the top central orifice. When ore particles and a portion of the feed medium are fed through this opening, they impinge on the rotating body of fluid. Particles of specific gravity lower than that of the rotating medium do not penetrate into the body of the medium and are discharged through the bottom central opening. Particles with a higher specific gravity than that of the rotating liquid settle toward the wall and are discharged through the top tangential opening. An advantage of this unit is that only the feed medium needs to be pumped, with the ore being fed directly into the device by gravity.

25.3.6.2 Jigging

In *jigging*, a mixture of ore particles is supported on a perforated plate or screen in a layer or "bed" with a depth many times the thickness of the largest particle. By subjecting the bed to an alternating rising and falling (pulsating) flow of fluid, the particles of high specific gravity travel to the bottom of the bed while the particles of lower specific gravity collect at the top.

FIXED-BED JIGS. All modern jigs are of the fixed-bed type in which the liquid pulse passes up and down through the jig bed, which is retained on a stationary screen. The different types of fixed-bed jigs are listed in Table 25.3.6.6. The older makes of jigs are described in Taggart (1951).

Variations in two modern jigs are discussed in the following.

Remer Jig—This jig has four important differences from the rest of the moving-hutch types: (1) the screen is continuous over all four compartments—stratification is therefore uninterrupted over the whole length of the jig; (2) all four compartments are actuated by a common mechanism; (3) the driving mechanism has two motions, a normal jig pulse of 80 to 120 strokes/min on which is imposed a fast (200 to 300 per min) pulse; (4) upflow water can be controlled individually in each of the four hutches.

Cleaveland IHC Jig—The special design features of this jig are (1) it is a circular jig in plan view; (2) the jig compartments are trapezoidal to reduce the effect of excessive top water; (3) the hydraulic-driven diaphragm stroke can be easily modulated; (4) the radial arrangement of the jig compartments saves space, provides for simplified feed distribution, and allows for the use of a raking device which levels the jig bed surface as it rotates.

METALLIC ORE JIGGING. In general, jigging operations have one of two objectives. Either they involve cobbing operations to discard waste rock before final processing of the crude concentrate, or they recover finished concentrates from ores in which the mineral grains are liberated.

Jigs are used to recover coal, diamonds, barite, concrete aggregate (from bank-run gravel), and heavy minerals of all kinds that occur in decomposed rocks or can be liberated by comminution without being reduced to slimes.

The greater part of the world's tin is concentrated by jigs. In the most modern tin dredges, diaphragm jigs such as the Yuba and more recently the Cleaveland seem to be preferred. In the concentration of fine heavy minerals from decomposed granitic rocks in Africa, good recoveries of tantalite, columbite, and cassiterite are obtained, down to 50 μm (300 mesh).

Use of jigs to recover gold and other precious metals is widely practiced since the high specific gravity differential makes good recoveries possible with very little operating attention. Jigs have the ability to adsorb large fluctuations in ore grade, tonnage rate, and dilution—all characteristics of placer and dredge operations. Also, since the metal concentrate is always hutched, losses due to theft are easily prevented. In dredge operations, diaphragm jigs are the most common, but in lode mining operations, the Denver Mineral Separation jig is commonly used.

Jig control has traditionally depended on the operator's skill. Modern practice may utilize instrumental control.

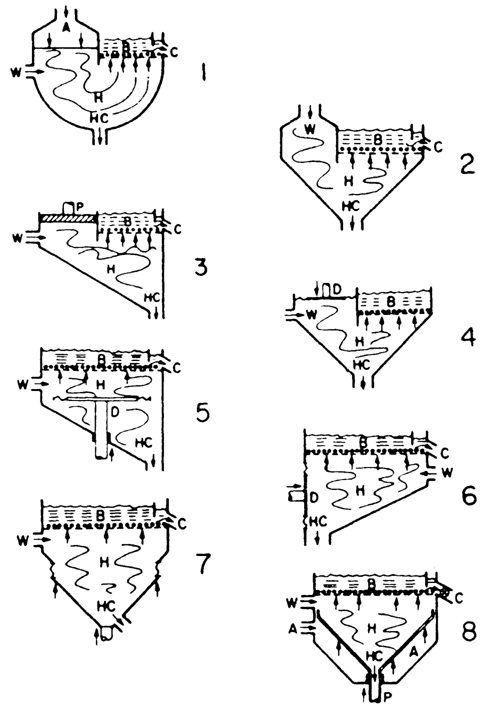
Given fixed machine parameters, practical operation requires adjustment of the state of bed dilution and particle stratification. To maintain this condition, the controls available to the operator are feed rate, rate of concentrate withdrawal, and hutch water addition. Any instrumentation and control should therefore include bed density sensing as well as solid flow measurement, with feed back loops to the feed, water, and discharge regulation devices.

Jigs use a large amount of water, so reclamation of water with recycling is standard practice. Clean water is not necessary, as clay and fully suspended slimes do not appreciably affect results up to a specific gravity of 1.05. However, provision must be made to clarify enough water to stay below this level. In some cases, chemical additions may help.

With few exceptions, but with some precautions, jig plants can be scaled up from pilot plant results. Every detail of the mechanical action of the pilot plant must be transferred to the industrial plant. Such factors as stroke speed, amplitude, and modulation; feed size distribution; and water velocity and distribution, must all be reproducible. For design purposes, it is useful to make specific gravity determinations on the pilot plant products and the plant feed and to construct distribution curves. These will permit efficiency comparisons with the specification curves for the industrial equipment. Within reasonable limits, imperfection coefficients from these curves are independent of feed changes.

Table 25.3.6.6. Fixed-bed Jig Types

Type and machines	Pulsing mechanism	Stroke modification
1. Baum		
Jeffrey	Air pulsion	Air discharge control
Link-Belt		
McNally-Pittsburgh		
Tacub, Batac		
2. Richards		
Pan-American Pulsator	Water pulsion	Water valve control
3. Hars		
Cooley, Collum	Mechanical piston	Differential piston action
Woodbury		
Denver-Hars		
4. Denver MS	Mechanical diaphragm	Water valve pulsion
5. Bendelari	Mechanical diaphragm (internal)	None
6. Yuba		
Jeffrey	Mechanical diaphragm (side)	Differential stroke, hydraulic or mechanical
Rouss		
Cleaveland-IHC		
7. Pan-American Placer		
Kraut	Moving hutch	Mechanical
Remer		
8. Russian		
MOBK	Pneumatic piston	Air release control



W = water inlet, H = hutch, B = jig bed, C = concentrate, D = diaphragm, P = piston.
Source: Aplan, 1985.

25.3.6.3 Wet Concentrating Tables

The modern *wet concentrating table* is a rectangular- or rhomboid-shaped deck, with riffles, that is operated in essentially a horizontal plane. A drive mechanism imparts a differential motion to the deck along its long axis while water flows by gravity along the short axis. Tables are currently employed for processing coal, barite, beach sands, chromite, glass sand, garnet, iron, manganese, mica, phosphate, potash, tantalum, tin, titanium, tungsten, and zircon.

The development of froth flotation in the 1920s, with its superior selectivity and capacity to concentrate finely ground ores, sharply reduced the use of shaking tables for sulfide ore concentration. As a consequence, only a few firms now manufacture tables for concentrating precious and base metal ores.

The separation effected on a wet concentrating table is the result of numerous mineral dressing principles simultaneously acting on the table feed. These include flowing-film concentration, hindered settling, consolidation trickling, and asymmetrical acceleration.

In practice, a slurry of solids and water is fed to the upper edge of the sloping table. As the suspended material moves across the table, it is caught and forms pools behind the longitudinal riffles. The differential shaking action of the deck causes size classification and specific gravity stratification. As a result, particles with similar specific gravities become arranged vertically according to size.

Once the beds are formed, the addition of more slurry and the action of the flow of cross water causes shearing of the top layers of the stratified particles, thereby forcing the lower specific gravity and coarser particles to cascade over the riffles toward the lower side of the table. The depth of the riffles and the bed thickness decrease from the drive mechanism end to the discharge end of the table. This results in continuous flowing film concentration of the increasingly finer-sized and higher-density particles as these particles move longitudinally along the table.

The differential drive mechanism is so designed that at the end of the backward stroke (deck moving toward drive end), the deck and hence the particles on the deck surface are momentarily at rest. The deck is then accelerated forward until, at the end of the stroke, the direction of travel is rapidly reversed. The particles on the deck, which moved with the deck on the forward stroke, will now slide forward owing to their momentum while the deck reverses its direction and starts on its backward stroke. Thus the particles always travel toward the discharge end of the table.

The idealized size and specific gravity stratification of the table action are shown in Fig. 25.3.6.4. Height and placement of the riffles and irregularities in feed rate, deck surface, table motion, and water supply and distribution all exert influences that will modify the idealized behavior of the feed pulp. The presence of middling particles, slimes, and marked differences in the size, shape, and porosity of particles of the same minerals also materially affect table action.

Table Operating Factors. To achieve maximum performance, attention must be given to the various operating conditions. Water and solids feed rates, size consist of the feed, and mechanical settings of the table are interdependent; prolonged changes of the table feed will require changes in the mechanical settings of the table.

Manufacturers' recommendations range from 230 to 285 rpm, with a 31.8 to 19-mm (1¼- to ¾-in.) stroke for coarse

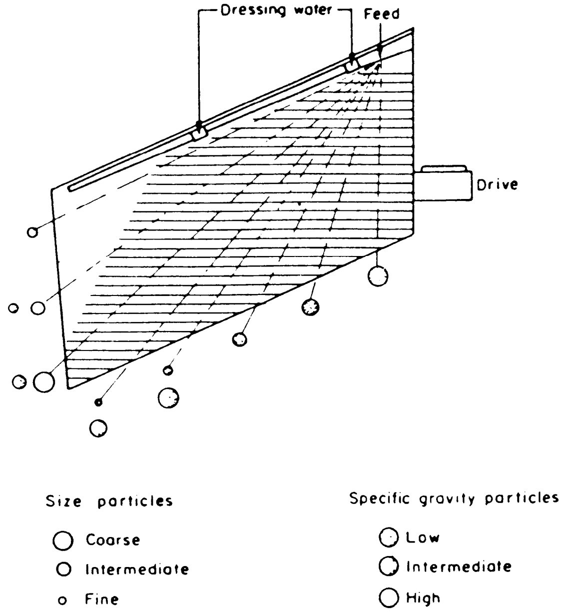


Fig. 25.3.6.4. Distribution of table products by particle size and specific gravity (Aplan, 1985).

sands. For finely sized material, the recommended range is 285 to 325 rpm, with a 19- to 10-mm (¾-in. to ⅜-in. stroke. Speeds and strokes for tabling coal are about the same as for ores.

NATURE OF FEED. Tabling involves both size and specific gravity separations that are influenced by the shape, porosity, and size consist of the mineral particles. Therefore, size classification of table feed is one practical way to improve table performance and increase table capacity. Modern multiple-cell hydraulic classifiers are simple, inexpensive, and can effectively sort crushed and ground materials by free and hindered settling. They discharge, at a constant rate, classified products suitable for tabling, sorting concentrate particles that have equal settling rates.

The largest particle size that can be effectively treated on tables is about 18 mm (¾ in.) when washing coal and about 1.65 mm (10 mesh) when washing ores. For ores of high specific gravity, the minimum effective size for beneficiation is below 44 µm (325 mesh).

RIFFLING. Tables are of two basic types: (1) sand tables, which are characterized by deep and extensive riffle systems necessary to effectively process coarser size material; and (2) slime tables, which are characterized by shallow riffles to minimize the disturbance in a bed of fine particles. The latter still allows the cascading of the fine material that is necessary for removing the low-density particles.

CAPACITY. The capacity of a table treating either metallic or nonmetallic ore is determined by the size and specific gravity of the material being tabled and whether the operation is a roughing or a cleaning one. The capacity of a table treating 1.65- to 1.17-mm (10- to 14-mesh) top-sized ore is about 1.8 t/h (2 tph) per deck. Slime tables producing a finished ore concentrate treat as little as 0.1 t/h (0.11 tph) per deck. The recommended capacity for coals having top sizes of 18 mm (¾ in.), 9.5 mm (⅜ in.), and 589 µm (28 mesh) is 13.5, 11, and 4.5 t/h (15, 12, and 5 tph) per deck, respectively. In actual practice where the washability of the feed is relatively good, overloads of up to 25% are often handled with little difficulty. However, where a high Percentage of near-separating gravity material is present, any overload to the table will greatly reduce the separation efficiency.

WATER CONSUMPTION. Water consumption is dependent on the size of feed and type of operation, that is, roughing or cleaning. In roughing operations, up to 2900 L of water/t of ore (700 gal/ton) is required. In cleaning, more water is used, ranging up to 4000 L/t (960 gal/ton). Slime ore treatment water requirements range from 3325 to as much as 4000 L/t (800 to 960 gal/ton).

A portion of the water is added directly to the table from a trough along the upper side (Fig. 25.3.6.4). This "dressing" water is necessary to assure complete mobility of the material on the table. Dressing water amounts to about 25% of the total water used in wet tabling.

TABLE FLOWSHEETS. Fig. 25.3.6.5 shows a typical table flowsheet for tantalite ore. The ore is processed over nine triple-deck tables in the primary circuit. In the cleaning circuit, the rougher concentrate is processed over three single deck tables. The tantalite concentrate is then dried. Coarse primary and secondary middlings are crushed and recirculated to maximize recovery.

Table 25.3.6.7 shows general operating parameters for a number of commercially available shaking tables as provided by the manufacturers.

25.3.6.4 Spirals

Since their introduction 30 years ago, spirals have become widely used. There are over 25,000 currently in use for concen-

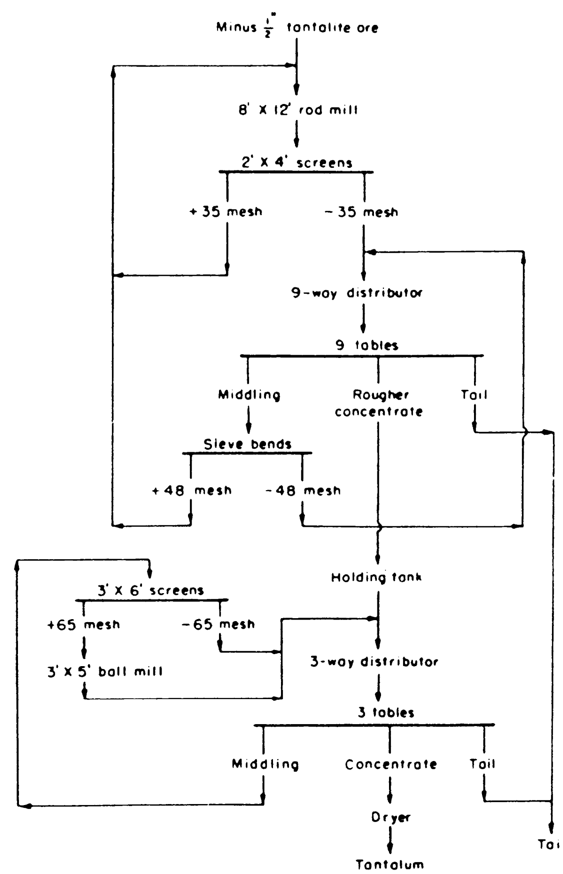


Fig. 25.3.6.5. Tantalite ore flowsheet (Aplan, 1985).

Table 25.3.6.7. Table Operating Parameters Suggested by Manufacturers

		Feed								
		Fine size		Coarse size					Horsepower	
Table model	Type	Top size	Capacity, ton per hr per deck	Top size	Capacity, ton per hr per deck	Speed, rpm	Stroke, in.	Deck size	Installed	Operating
SuperDuty and Concenco Tables, Deister Concentrator Co.										
No. 6	Ore	100 mesh	0.25	6 mesh	2.0	285–300	½–¾	6 ft 5 in. × 14 ft 1 in.	2	½
No. 666 (3 decks)	Ore	100 mesh	0.25	6 mesh	2.0	285–295	½–¾	6 ft. 5 in. × 14 ft 1 in.	3	3
No. 7	Coal	28 mesh	5.0	¾ in.	15.0	280–290	½–1¼	8 ft ¼ in. × 16 ft 9¼ in.	3	1
No. 77 (2 decks)	Coal	28 mesh	5.0	¾ in.	15.0	280–290	¾	8 ft ¼ in. × 16 ft 9¼ in.	3	3
Wilfley and Holman Tables, Wilfley Mining Machinery Co., Ltd.										
Wilfley No. 20	Ore	100 mesh	0.25	6 mesh	2.0	300–325	⅜–⅞	6 ft × 15 ft 6 in.	3	1
Wilfley No. 21	Ore	100 mesh	0.125	6 mesh	1.0	300–325	⅜–⅞	4 ft × 9 ft	2	1
Holman	Ore	100 mesh	0.25	6 mesh	2.0	270–280	⅜–⅞	5 ft 6 in. × 18 ft	2	¾–1
Wilfley No. 20C (3 decks)	Coal	28 mesh	5.0	¾ inch	13.0	230–270	1–1¼	7 ft 6 in. × 15 ft 6 in.	3*	1
Wilfley Tables (MSI Industries, Inc.)										
No. 6A and 11D (standard)	Ore	(†)	0.5	(†)	6.25	240–300	¾–1¼	6 ft × 15 ft	1½	½–¾
(oversize)	Ore	(†)	0.75	(†)	7.25	240–300	¾–1¼	7 ft × 15 ft	2	¾–1
No. 12	Ore	(†)	0.25	(†)	0.75	260–300	¾–1¼	3 ft 6 in. × 7 ft	1	¼–½

* Per deck.
† Not available.
Conversion factors: 1 in. = 25.4 mm, 1 ft = 0.3048 m, 1 tph = 0.9072 t/h, 1 hp = 0.7457 kW.
Source: Aplan, 1985.

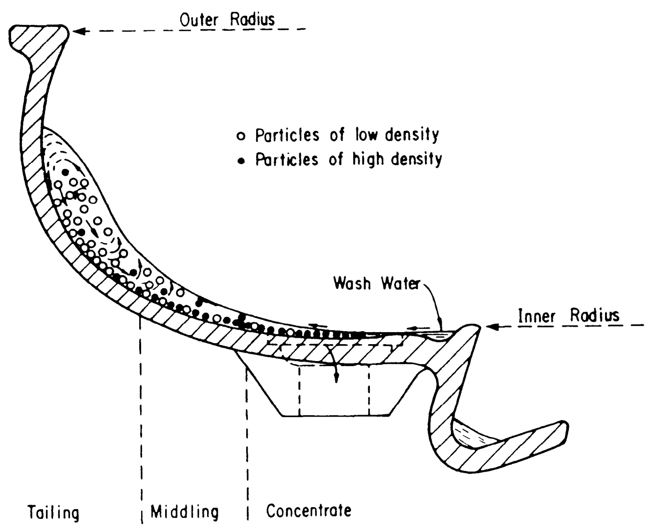


Fig. 25.3.6.6. Cross section of spiral trough (by permission from Humphreys Engineering Co.).

trating hematite, ilmenite, zircon, rutile, tungsten ore, chromite, and coal. Spirals usually have low capital and operating costs and low energy consumption.

A *spiral* consists of a curved channel. Wash water is supplied in an auxiliary trough from which it can be deflected into the main pulp stream. A cross section of the device is shown schematically in Fig. 25.3.6.6.

SPIRAL OPERATION. Feed pulp and wash water volumes are the most important operating parameters and vary between 57 and 113 L/min (15 and 30 gpm) for the feed and 11 and 57 L/min (3 and 15 gpm) for wash water. When either volume is too low, it causes a sluggish flow and sand bar formation occurs, resulting in a reduction of both concentrate grade and spiral capacity. On the other hand, excessive volumes cause the high specific-gravity mineral to be swept wide of the upper port, resulting in sand bar formation in the lower turns. This also results in lower recovery and capacity. The coarser the feed, the higher the optimum feed rate, since a higher pulp density is needed to crowd the large high specific gravity particles into the concentrate.

Feed size is usually between 1.16 mm and 74 μm (14 and 200 mesh), but with iron ore, up to 2.3 mm (8 mesh) material may be treated to make a rougher concentrate, provided a high pulp density is used. This prevents the larger high-specific-gravity particles from migrating towards the tailings. The resulting concentrate is difficult to clean, however, unless it is stripped of the +0.5 mm (28 mesh) material by classification. Usually, recovery falls off rapidly below 74 μm (200 mesh), but with a few high-specific-gravity minerals can be good down to 44 μm (325 mesh). Pulp density is usually between 20 and 30% solids.

Australian beach sand practice generally is to use 25 to 30% solids, but the operations are fairly good over the range 15 to 40% solids. Throughput varies from 0.4 to 2.2 t/h (0.4 to 2.4 tph) per machine, dependent on the feed size. On Australian beach deposits, the capacity of the normal 610- and 1067-mm (24- and 42-in.) spirals used by Rutile and Zircon Mines, Ltd. ranges from 0.4 to 2.2 t/h (0.4 to 2.4 tph), depending on the feed grade, degree of concentration, and recovery. The Wyong spirals have throughputs up to 3.7 t/h (4.1 tph) of feed carrying about 1% heavy minerals.

There are other types of gravity concentrators such as the flowing film concentrator, Bartles-Mozley concentrator, and Reichert Cone, which are used in special cases. Details on these are beyond the scope of this book, and readers are referred to the *SME Mineral Processing Handbook* (Weiss, 1985).

25.3.6.5 Laboratory Testing

The first step involved in developing a gravity separation flowsheet is to perform the heavy liquid analysis of the sample

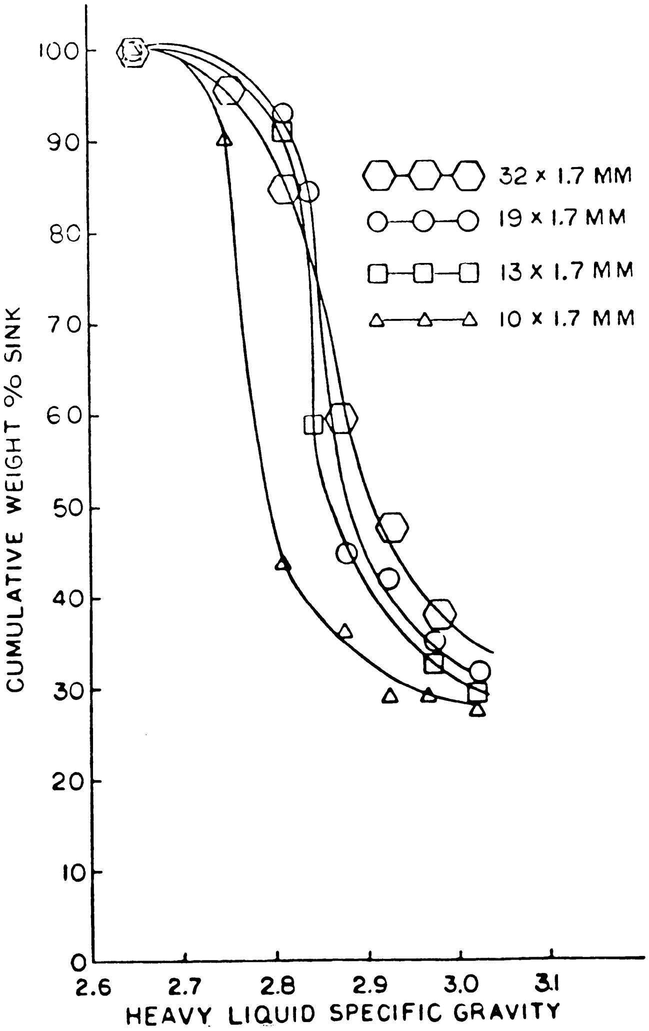


Fig. 25.3.6.7. Washability curve of a Mexican fluorite ore (Srinivasa, 1981). Conversion factor: 1 in. = 25.4 mm.

to determine the washability of the coal or ore. This analysis establishes the size of separation of the ore and the specific gravity of separation.

A complete sequential sink-float analysis of the ore, at close specific-gravity intervals, should be made in order to get a complete understanding of the ore's response to heavy liquid separation. Heavy liquids commonly used in the heavy liquid tests are methylene iodide (3.3 sp. gr.), tetra-bromoethane (2.96 sp. gr.), and methylene bromide (2.48 sp. gr.). Clerici solution is used for specific gravities exceeding 3.2. Ethylene dibutyl phthalate or carbon tetrachloride is used for dilution to intermediate densities. Caution should be exercised in handling the organic liquids. Most of them are toxic, and some are absorbed by the skin on contact. Laboratory tests should therefore be performed in a well-ventilated area.

Washability curves prepared from the heavy liquid analysis are given in Fig. 25.3.6.7. They show that the weight recovery of the sink fraction, at a given specific gravity, decreases at finer sizes. This indicates possible liberation of the values from the gangue particles. Grade-recovery curves (Fig. 25.3.6.8) show that the highest recovery is obtained at 19 mm (¾ in.) size. This indicates that there is an optimum size at which the best

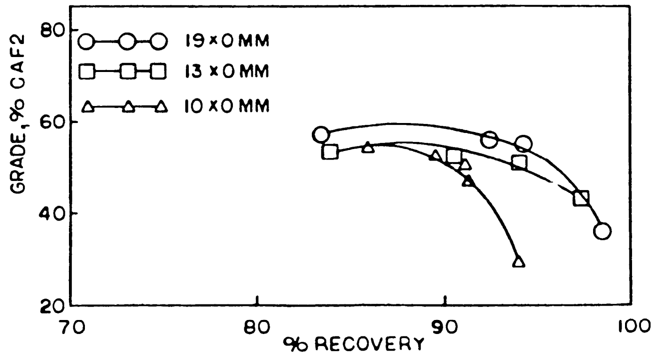


Fig. 25.3.6.8. Grade vs. recovery of a fluorite sample (Srinivasa, 1981). Conversion factor: 1 in. = 25.4 mm.

metallurgical results are obtained. It is therefore obvious that a washability curve alone will not provide the optimum conditions for separation. An analysis of the results is necessary.

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25.3.7 MAGNETIC SEPARATION

V.P. KENYEN

Magnetic separation is a means of concentrating or separating minerals with paramagnetic properties from nonmagnetic gangue particles. Preventive magnets are also used for the removal of tramp iron in crusher feed and other bulk material streams. This discussion is limited to concentration. Application of magnetic separators for removal of tramp iron is covered in several of the works listed in the bibliography.

25.3.7.1 Wet Magnetic Separators

There are two types of magnetic separators, wet and dry. The wet unit could be a drum separator, a magnetic filter, or a high-intensity wet magnetic separator.

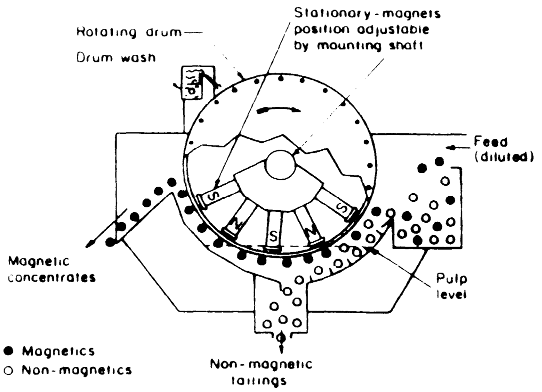

Fig. 25.3.7.1. Concurrent wet-drum separator (Bronkala, 1978, p. 471).

Wet drums are used to concentrate finely divided, strongly magnetic material. Three types of drum separators are available: concurrent, countercurrent, and counterrotation. Concurrent drums (Fig. 25.3.7.1) are used for cobbing of –6-mm (–0.25 in.) particles, countercurrent drums are used for finishing operations at –210 µm (–65 mesh), and counterrotation drums are used for roughing operations on –1.7 mm (–10 mesh) ore.

Wet magnetic drums are constructed of either permanent or electromagnets. A large majority of installations use permanent magnet drums because of their dependability and lower operating and maintenance costs. Electromagnetic drums are used where extremely high magnetic fields are desired or where simple electromagnetic field control is desired.

Commercial wet-drum separators come in diameters of 0.76, 0.91, and 1.2 m (2.4, 2.9, and 3.8 ft). Multiple (double or triple) drums are used for cobbing and finishing. Normal feed rates to the magnetic separators are as follows:

Cobbers 100 to 169 m^3/h/m (1075 to 1820 ft^3/hr/ft) of magnet width

Roughers 100 to 130 m^3/h/m (1075 to 1400 ft^3/hr/ft) of magnet width

Finishers 70 to 105 m^3/h/m (750 to 1130 ft^3/hr/ft) of magnet width

25.3.7.2 Dry Magnetic Separators

Based on the magnetic field intensity they develop, dry magnetic separators are classified as high-, moderate-, or low-intensity separators. High-intensity separators remove weakly magnetic particles, moderate-intensity separators remove moderately responsive magnetic particles, and low-intensity separators remove paramagnetic particles. The physical condition of the particles influences the selection of the dry magnetic separator.

There are two types of dry separators, the cross belt and the induced roll. The cross belt picks the magnetics off the feed belt and discharges them to the side. This has been used for concentrating wolframite and monazite.

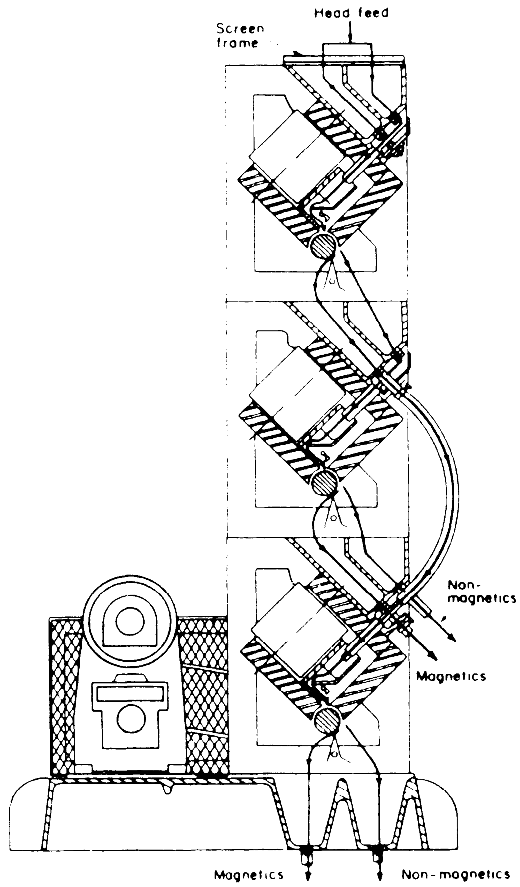

Fig. 25.3.7.2. Operating principle of induced-roll magnetic separator (Bronkala, 1978, p. 476).

The induced roll separator is used to concentrate and purify chemicals and minerals. The most frequently used purifying-type induced separator is the 0.76-m (30-in.) wide, three-field, twin-type separator having 1.5 m (60 in.) of total feed width. Such a unit is shown in Fig. 25.3.7.2 and will clean 2.7 to 6.4 t/h (3 to 7 tph) of product.

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Chapter 25.4
CHEMICAL AND ELECTROLYTIC PROCESSING

JOHN A. WELLS, WAYNE R. HOPKINS, AND RICHARD B. STEIN

The subject matter of this chapter is too vast to give even a cursory overview. There are many excellent text and reference books that treat individual topics in detail relating to chemical and electrolytic processing. Rather than attempt to treat the whole range of subject matter, this chapter will concentrate on new and emerging process technologies relating to chemical and electrochemical processes for recovery of gold, copper, zinc, aluminum, and magnesium.

25.4.1 GOLD PROCESSING

JOHN A. WELLS

The traditional circuit for gold recovery is the Merrill-Crowe method using zinc precipitation. It has been largely replaced by carbon-in-pulp and carbon-in-leach circuits, followed by electrowinning, and their newer technologies will be described in detail below.

25.4.1.1 Carbon-in-pulp

In the last decade, no subject has received greater attention in the field of extractive metallurgy than *carbon-in-pulp* (CIP) and the closely related *carbon-in-leach* (CIL) *processes.* This is a direct response to the relatively high price of gold that resulted in the development of many new mines.

The success of a CIP plant is centered around the adsorption operation. This section must work well, and gold adsorption in excess of 99.5% should be the target of both designers and operators. Such a figure is not always achieved, and reasons for this will be proposed.

Although CIP has been the popular choice for the majority of new plants, the Merrill-Crowe process has a history of successful applications. CIP is particularly suited to those ores that exhibit poor solids-liquids separation, as the filtration stage is eliminated. Adsorption tanks are much easier to install and operate than vacuum filtration (frequently two stages) with the associated repulping and vacuum systems.

Capital costs for CIP are generally 65 to 85% of those for Merrill-Crowe. However, for ore with high precious metal values, the Merrill-Crowe system has lower costs, as shown in the following:

Plant Capacity		Grade above which Merrill-Crowe capital costs are lower: Precious Metal Content*	
tpd	t/d	oz/ton	g/t
900	817	0.48	15
1800	1633	0.80	25
4500	4082	1.13	35

* Gold plus Silver content.

These numbers are site specific and based upon a generic study. They indicate why CIP is so popular, as there are few gold mines found today with grades in excess of those shown above. The reason for the less favorable economics of CIP at high grades is the large quantity of carbon that is required, and the large elution and regeneration section that is necessary to process it. Many gold mines contain an appreciable silver content, which puts the combined precious metal value well in excess of those shown above. For this reason, as well as the relatively lower kinetics of silver adsorption on carbon, new plants for silver and silver-gold deposits frequently continue to utilize Merrill-Crowe. Operating costs for CIP plants have been steadily reduced as improvements have been made to the process. Several estimates have been made comparing the two, and generally CIP operating costs are believed to be 80 to 90% of those of Merrill-Crowe.

CIP-FEED PREPARATION. The major problem experienced by the first generation of CIP plants was contamination of all stages of the process by the tramp oversize carried forward from the grinding circuit. This oversize material includes wood chips, plastic, and coarse solids. With the interstage screening systems installed in adsorption, this coarse material accumulates in the process and ultimately contaminates the carbon in all stages of adsorption, elution, and regeneration, and contributes to screen blockages and eductor chokes.

Many methods of removing this material from the system were developed, none of which were wholly successful. The long-term solution had to be prevention rather than cure, and this was achieved by the development of superior feed slurry screens and, more recently, by the development of the linear screen.

The selection of feed screens is therefore a critical element of CIP plant design. The screens are usually vibratory and utilize either wire mesh decks (which generally have a short life due to wear or more frequently tearing) or polyurethane screen panels. These latter panels have an extended life with good wear resistance but provide less open area and therefore, lower capacity in terms of cubic feet (cubic meters) of pulp per square foot (square meter) of screen area. The screens are typically 28 or 35 mesh, to remove both oversize and "near-size" material. An interesting innovation that works particularly well is the concept of reversing the slope of the screen, such that the feed end is lower than the discharge end. The oversize slowly moves up the screen. This effectively dewaters the oversize and prevents excess slurry from passing over the screen which is often a feature of "down slope" screens. Vibrating feed screens have a capacity of 100 to 130 $ft^3/ft^2/hr$ (30 to 40 $m^3/m^2/h$), but figures as low as 60 and as high as 200 $ft^3/ft^2/hr$ (18 to 60 $m^3/m^2/h$), have been reported (density and viscosity being major determinants of screen capacity). Non-vibrating screens have a much lower capacity and require frequent cleaning.

A recent innovation in CIP feed preparation is the linear screen shown in Fig. 25.4.1.1. These machines resemble typical belt filters and comprise a polyester filament cloth supported on rollers over a head and tail pulley. The slurry is distributed over the filter using a feed box that has a perforated plate bottom. The slurry drains through the cloth into the underpan, and the oversize is carried to the head of the screen from which it is

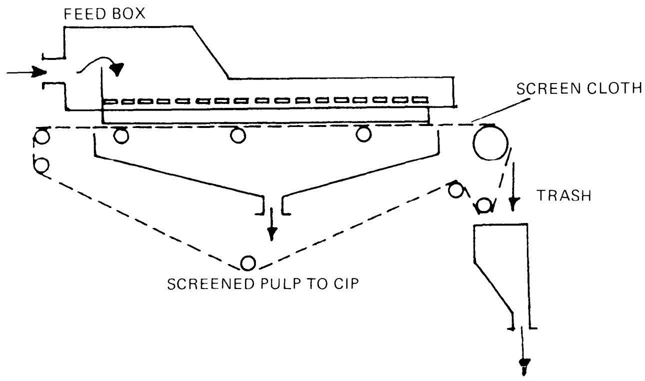

Fig. 25.4.1.1. Linear screen.

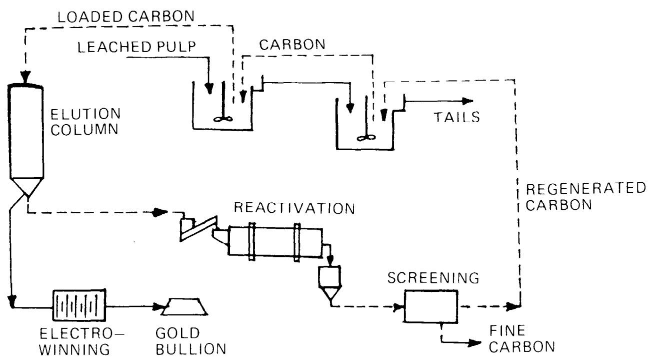

Fig. 25.4.1.2. Typical CIP circuit.

removed by a wash spray. These units often provide excellent results, with high throughputs of 165 $ft^3/ft^2/hr$ (50 $m^3/m^2/h$) at 99% availability and over 2500 hours cloth life. One mine reported that the woodchip content in the CIP pulp of 0.7 lb/1000 gal or 0.08 g/L was almost entirely eliminated.

One aspect of plant design that is frequently overlooked is the subsequent handling and disposal of the screen oversize. If the oversize product is wet, or a wash spray is used, the product may require secondary screen for dewatering. A small vibrating screen or sieve bend is suitable for this purpose. The solid residue should fall into a collecting box, which must be both big enough to hold a day's production and accessible enough for ease of removal. Alternatively, the residue can be repulped and pumped to the final tailing box.

CIP-ADSORPTION. A typical CIP adsorption section comprises a series of mechanically agitated tanks, often with a height-diameter ratio of 1:1, each tank providing a residence time of between 1.0 and 1.5 hours. A typical CIP circuit featuring adsorption tanks, elution, reactivation, and electrowinning is shown in Fig. 25.4.1.2.

Several publications have discussed the role of modeling for adsorption circuits, whereby the relationship between the key variables is developed (gold tenors, residence time, number of stages). For any given set of circumstances, the gold adsorption efficiency will increase with the number of stages. However, beyond a certain number of stages, the incremental return (i.e., gold recovery) is less than the additional costs incurred. Typically, gold adsorption is 60 to 70% per stage. As a very general approximation, six adsorption stages would appear to be the optimum for precious metal grades of 0.32 to 0.65 oz/ton (10 to 20 g/t). Fewer stages are not generally recommended due to the potential for short circuiting. Further stages may be added for particularly high-grade ores or if there is evidence of poor kinetics. However, it is advisable to keep the number of stages as low as possible in order to minimize capital and operating costs and the carbon inventory (high carbon inventories increase carbon consumption through attrition, increase the possibility of fouling, and increase gold in-process inventory). Plant design should however, be such that space is available for future expansion of adsorption capacity if required.

In order to achieve low barren solution values and achieve the target recovery of 99.5% gold adsorption, a critical parameter is the carbon activity (the maintenance of that is dependent upon the other unit operations which are described elsewhere).

The carbon should be loaded to as high a gold value as possible, and this value can be predicted by testwork and the use of the models referred to above. As a general rule, loadings of 30 oz/ton (1000 g/t) (gold/carbon) should be possible for 0.03 oz/ton (1 g/t) in the mill feed.

Mechanical agitation in CIP has largely displaced air agitation. It has been shown to be better both metallurgically and operationally. Power requirements are proportional to D^5 (D is impeller diameter). From operating data, this is typically 0.001 to 0.002 kW/ft^3 (0.05 to 0.10 kW/m^3) for 13- to 23-ft (4-to 7-m) diameter tanks.

The adsorption tanks may be installed in a straight line, with a drop between tanks of approximately 1.5 ft (0.5 m), or in an offset pattern. Other adsorption tank systems have been proposed, including one continuous vessel, separated into individual compartments by screens. However, the individual tank system remains the overwhelming choice at this time.

Transfer of carbon between stages is generally achieved by recessed-impeller-type pumps, suspended in the top of the tanks. These have replaced airlifts, which were used in the first CIP plants. Plant and test data conclusively show that carbon is hard and withstands pumping action. In general, the movement of carbon by pumping is recommended throughout CIP plants, and is preferable to eductors that require large volumes of high-pressure water and have a tendency to choke.

An interesting debate remains the issue of tank support. In South Africa, most tanks are installed with a concrete support, all tanks having the same dimension. An alternative to this is the false-bottom concept, which is popular in North America.

CIP INTERSTAGE SCREENS. Interstage screens are installed in each stage of adsorption to retain carbon but allow pulp to flow through the circuit. For finely ground gold ore pulps (i.e., typically 70% finer than 74 μm) the following screening sizes are usually employed for CIP/CIL operations.

Carbon Size	1–3 mm (6–16 mesh)
Prescreening	0.6 mm (28 mesh)
Interstage Screens	0.8 mm (20 mesh)
Final Catch Screens	0.6 mm (28 mesh)

Interstage screening is a critical feature of CIP operations. Many operations have performed below expectations due to capacity constraints or low efficiency of these screens.

25.4.1.2 Carbon-in-leach

Soon after the introduction of CIP, it was considered by some that the adsorption and leaching circuits could be combined into one unit operation. Reasons for this included the following:

1. The larger leach vessels would overcome interstage screen space limitations frequently experienced with the smaller adsorption tanks.

2. Extra dissolution had been observed on pulps passing through a CIP circuit, and it was considered that the addition of carbon early in the leach cycle might enhance the dissolution and, most importantly, prevent or reduce any preg-robbing tendencies in the ore.

3. Estimates of the capital expenditure indicated that the combined operation (CIL) would be 20% less costly than CIP.

Subsequent experience with CIP and CIL has shown that the actual results achieved in practice make comparison of the two more difficult than expected. Both systems work well in site-specific applications and achieve the desired results. The following general comments can be made.

1. Many of the early problems experienced with CIP were in retrospect due to peripheral problems such as poor screening and regeneration, rather than some fundamental problem in adsorption. However, when leaching and adsorption are combined in one vessel, technical problems increase rather than diminish. The resolution of poor gold recovery becomes more difficult to ascertain (i.e., is it poor dissolution, poor adsorption, or a combination of the two?). The concept of a simpler operation for CIL does not materialize in practice.

2. CIL is probably cheaper than CIP, but not much, and the difference is possibly less than 10% (and less than 1% of total project costs). Savings in tanks and agitators are partially or wholly offset by the greater carbon and gold inventory. (CIL inventories are approximately twice that of CIP. Greater carbon inventories in turn lead to greater carbon losses through attrition.) In addition, it can be argued that the higher carbon inventory will cause more adsorption of contaminants as the carbon stays longer in the adsorption circuit.

3. The CIL circuit operates with much lower carbon concentrations, typically 40 to 65 lb/1000 gal (5 to 8 g/L), and this results in the forward transfer of significantly greater volumes of pulp. This increases the load on the interstage screens, and calls for larger transfer pumps that frequently operate continuously. This high volume of pulp transfer undoubtedly contributes significantly to back mixing and in turn poor gold loading profiles and carbon distribution. The result is lower overall gold recovery.

4. Combining the processes can inhibit the optimization of the operational conditions for either process as the conditions can often be in conflict. Leaching may require high cyanide and lime levels and mixing rates. These are all detrimental to the adsorption process. Similarly, higher temperatures enhance leaching, but are detrimental to adsorption.

The reason that CIL is frequently installed in new plants is because some ores contain significant quantities (0.25% or more) of carbonaceous material. Under these circumstances, CIL, with the early addition of active carbon, reduces the effect of “preg robbers”. Whether this advantage is greater than the disadvantages of CIL is site-specific, and confirmatory test work and economic analysis should be completed prior to a final decision.

25.4.1.3 Carbon Regeneration

The successful application of CIP is dependent upon multiple use of the carbon through many adsorption-elution cycles. As the activity of the carbon is the main criterion that governs the gold recovery from solution, then it follows that some form of regeneration will be necessary.

Several types of kilns have have been proposed and tried for thermal regeneration. The conventional rotary kiln remains the most popular choice. Other kilns have been proposed, tested, or installed, resulting in various degrees of success. These include fluid bed, multiple hearth, resistive heating, conveyor belt, and indirect heat furnaces. Most operations continue to prefer the conventional rotary kilns.

During adsorption a whole host of organic and inorganic chemicals accumulate on the carbon (if they are present in the pulp), together with the precious metals. Thus water quality is important in the CIP process. Organics that may be present due to accidental spills or that are residual from effluent treatment plants (if such water is used by the plant) are particularly strongly adsorbed and foul the carbon. Thus the objectives of reactivation are to remove the adsorbates and return the carbon to, or near to its original quality in terms of activity. [Regeneration also includes periodic acid washes that remove the lime scale that can foul the carbon.]

25.4.1.4 Carbon Stripping (Elution)

Loaded carbon is periodically transferred from the number one contactor (either CIP or CIL) to the carbon stripping circuit. Stripping is also commonly referred to as elution.

In the stripping circuit, the gold is removed from the carbon into a pregnant solution. The gold (and silver) is generally recovered from this pregnant solution by electrowinning (as described in 25.4.1.5) or by zinc precipitation (Merrill-Crowe).

In stripping, the loaded carbon is first screened to separate the carbon from slurry or solution. The carbon is then transferred into a vertical strip column. A hot (230 to 266°F or 110 to 130°C) dilute solution, (usually a caustic/cyanide mix) is then pumped up through the column, this strips the adsorbed gold from the carbon. Usually a warm hydrochloric acid wash is also employed, either before or after stripping, to remove other materials, such as carbonates, that if left would ultimately blind the carbon.

Two approaches to stripping have been developed, namely the Zadra Process (common in North America) and the A.A.R.L. Process (developed and used in South Africa). Both systems have their proponents. As long as they are correctly designed, both systems work well and allow efficient stripping of the precious metals. Atmospheric stripping (temperature below 212°F or 100°C) is very slow and is seldom used.

25.4.1.5 Gold Electrowinning

Electrowinning is frequently used to recover gold from the pregnant solutions generated by the hot elution of gold from loaded carbon. It is a simple technology which has been successfully applied at many new gold mines throughout the world during the past 10 years. The electrowinning cells are typically small tanks, with 15- to 175-ft^3 (0.5- to 5.0-m^3) capacity, and are fabricated from a wide range of materials. The cells contain stainless steel plate anodes. The cathodes are usually in the form of steel wool in polypropylene cases. Gold is deposited at the cathode, together with evolution of hydrogen, and the predominant reaction at the anode is the generation of oxygen.

GOLD ELECTROWINNING CELLS. Parallel to the rapid development of carbon-in-pulp technology, considerable efforts have been made in cell development. As the technology has evolved, there has been a general acceptance of the rectangular, open-bath-type cell, with the other types being used less frequently. In this type of cell, the cathodes and anodes are suspended vertically in a rectangular tank, as shown in Fig. 25.4.1.3. The size of the tank and the numbers of electrodes are determined by the volume and tenor of electrolyte. The anodes are 316 stainless steel perforated plates or woven wire mesh (generally 1-mm wire, woven at a 3-mm pitch). The cathodes are polypropylene boxes, filled with stainless steel or mild steel wire wool. A stainless steel frame is incorporated into the cathode box to

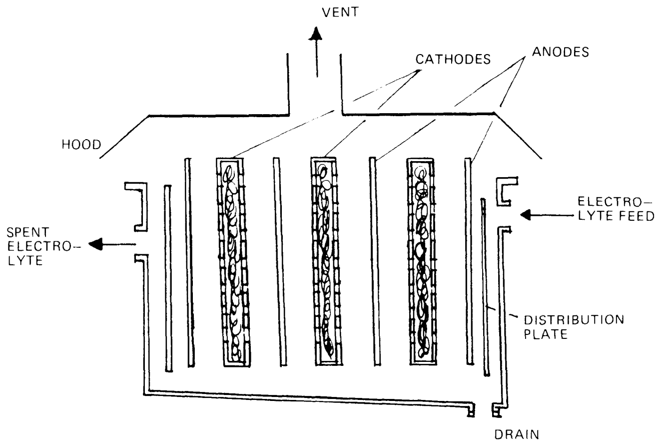

Fig. 25.4.1.3. Rectangular-type gold electrowinning cell.

add mechanical strength and provide electrical distribution across the whole cathode area. A typical cathode box would have dimensions of 12 by 150 by 150 in. (50 by 600 by 600 mm) and contain between 2 and 4 lb (1 and 2 kg) of steel wool. A steel wool packing density of 1.8 lb/ft^3 (30 kg/m^3) is the norm.

The cells are generally manufactured from either stainless steel (in which case, good insulation is necessary to prevent stray currents) or of plastic and fiberglass. Although lower in cost, care must be taken with plastic and fiberglass units in order to provide mechanical strength and durability. Operating data for a typical cell are as follows:

Cell size ($L \times W \times H$)	in.	$35 \times 28 \times 25$
	(mm)	$(900 \times 720 \times 640)$
Volume	ft^3	14
	(m^3)	(0.4)
Flow rate	ft^3/min	5
	(m^3/min)	(0.15)
Residence time	min	2.7
Cell current	A	500–800
Cell voltage	V	3–5
Number of cathodes		6
Number of anodes		7
E (single-pass efficiency)	%	70

25.4.2 COPPER PROCESSING

WAYNE R. HOPKINS

25.4.2.1 Solvent Extraction: Principles

For the commercial production of copper via solvent extraction, the common reagents are chelating, copper-specific species where the reactions are a hydrogen ion cycle as shown in Eq. 25.4.2.1 (and where R is the reagent molecule):

in extraction *in stripping*

$$2RH_{(org)} + Cu^{2+}SO_4{}^{2-}{}_{(aq)} \rightleftharpoons R_2Cu_{(org)} + H_2SO_{4(aq)} \quad (25.4.2.1)$$

with the direction of the reaction being controlled by hydrogen ion concentration.

These reagents demonstrate:

1. Fast reaction kinetics.
2. A high selectivity for the metal over a wide pH range.
3. Good stripping characteristics of the organic extract by change of parameters using acid levels in associated spent electrolyte of less than 1.2 lb/gal (150 g/L).
4. Rapid coalescence.
5. High solubility in the organic diluent and low solubility in the aqueous phase.
6. Low rate of degradation by hydrolysis, etc.
7. Low toxicity.
8. High flash point.

Diluent such as kerosene is used to reduce the viscosity of the extractant reagent and facilitate contact of the reagent with the aqueous solution. Such diluent is a mixture of paraffins, napthenes, and aromatics. Depending largely on the proportion of aromatic material, the diluent has some effect on the equilibrium, extraction kinetics, and phase separation. In copper systems, diluents are always high-flash-point (169 to 172°F, or 76 to 78°C) kerosenes to avoid the fire danger.

The stripping operation is the reverse of the extraction. In this step the loaded organic from the extractors is stripped of its loaded copper ions and converted back to a form appropriate for return to the extraction step. A chemical environment is employed that will cause the desired product to leave the organic phase and distribute into a concentrated aqueous solution from which it can be recovered. In addition, the stripping operation must restore the solvent to original activity for extraction.

25.4.2.2 Solvent Extraction: Equipment

The function of the liquid-liquid contactor is to bring two liquid phases together to allow maximum mass transfer between them and then to separate them prior to further contacting or processing.

There are a large number of contactor designs that fall into two categories: stagewise and differential. Column contactors are an example of differential contactors and are commonly used in the petroleum industry; they may be unagitated or agitated mechanically or by pulsing. Another example of a differential contactor is the centrifugal contactor. Mixer-settlers are stagewise contactors.

Several commercial firms offer proprietary mixer and settler designs that incorporate recent improvements, with the latter using the pumping type of mixer in order to avoid the cost of large interstage pumps. Of the earlier pumping mixers, the most widely used pump-mix impeller is the Denver-General Mills design, which has the turbine located at the bottom of the mixer tank. The turbine is a flat disc with radial blades beneath. It is mounted only 0.8 to 1.2 in. (20 to 30 mm) from the tank bottom, which then acts as a second impeller shroud and minimizes recirculation. Another type is the Davy design, which is mounted centrally in the mixer tank and is fed by a draft tube to the impeller eye. A compromise design has been developed using a Denver-type impeller at the tank center above a draft tube carrying a flat annular section that provides the second shroud. It is double shrouded with eight backswept vanes and may be fitted with spoiler blades. All these designs are used at tip speeds of less than 20 fps (400 m/min) to reduce secondary haze formation, and entrainments of organic continuous stage are normally 50 to 100 ppm.

The primary organic/aqueous dispersion formed in the mixer is allowed to disengage in the settler, where clean layers of each phase form by coalescence. The dispersed phase feed to the settler from the mixer should ideally consist of droplets with a narrow size range distribution. The bulk of the droplets produced are generally larger than 50 to 100 μm and are referred

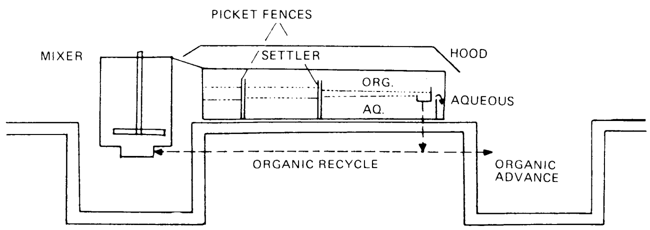

Fig. 25.4.2.1. Conventional mixer-settler (on grade).

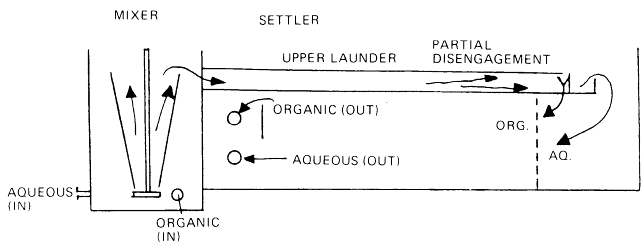

Fig. 25.4.2.2. Krebs mixer-settler.

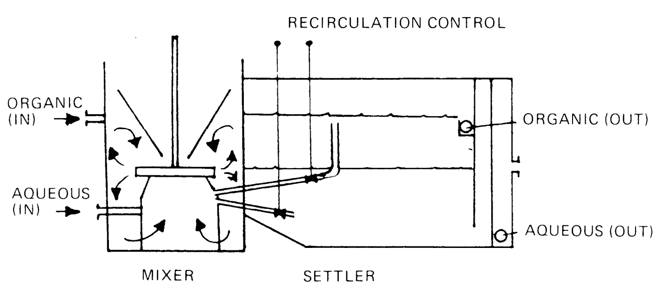

Fig. 25.4.2.3. Outokumpu mixer-settler.

to as the primary dispersion. Settlers are basically shallow rectangular tanks with a dispersion inlet at one end consisting of a baffle arrangement that directs the incoming mixture into the dispersion band existing in the settler. This introduction is kept at a controlled level by adjusting the height of the aqueous weir. This method of introduction reduces the formation of secondary entrainment by preventing the redispersion of the settled phase. At the opposite end of the settler, weirs separate clear organic and aqueous phases. Typical settler sizing is based on flowrates of about $3ft^3/ft^2/min$ ($5\ m^3/m^2/h$).

A typical configuration for the conventional pumping mixer and settler combination is shown in Fig. 25.4.2.1. The designs offered by Krebs (Fig. 25.4.2.2) and Outokumpu (Fig. 25.4.2.3) are the most recent in the field.

An important comment is useful as regards protection from potential fire hazards due to the flammability of the organic liquids. The fire protection system is normally arrived at on a case-by-case basis in conjunction with the operator's insurance underwriter. The aim of the firefighting system must be to extinguish any fire within about 10 minutes and to ensure that the adjacent mixer-settler units are cooled sufficiently to prevent extension of the fire. If a fire with the kerosene-type liquid is not extinguished in this period, then it is most probable that serious and extensive fire damage will result, putting the plant out of operation for a long period. The attainment of high temperatures during prolonged combustion will make the fire especially dangerous as the aqueous layer beneath the organic layer will boil and generate large quantities of steam which may eject burning organic droplets.

25.4.2.3 Electrowinning

Electrowinning is the final step in the production of metal from the recovery plant associated with a leaching operation. It consists of the electrowinning tankhouse, associated electrolyte pumps and tankage, materials handling equipment for starting sheet production, and product cathode packaging and shipping. Although the technology has been in place for about 100 years, the features of modern electrowinning (EW) tankhouses relate mostly to reduction in capital and operating costs and improvement in the copper cathode quality. The notable improvements in capital cost are due to (1) use of higher current density, (2) larger cathodes, and (3) use of improved corrosion-resistant materials of construction.

Use of a higher current density plays a very significant role in minimizing capital investment requirements for an electrowinning plant. Both pilot and commercial solvent extraction/electrowinning plants have shown that higher current densities are feasible for the relatively purer electrolytes produced. Operation of conventional type cells at 28 A/ft^2 (300 A/m^2) is now accepted.

Use of larger cathodes generally results in a better physical and chemical quality product. This is because the rough and often more impure copper occurs around the edges of the cathode. Use of larger cathodes results in capital investment cost savings.

1. The physical size of the cathode should be considered from a materials handling standpoint. This is particularly important when handling thin starting sheets to be sure they are not bent, etc.

2. The dimensions of the cathode should be chosen to allow interchangeability with other tankhouses. This may be important during startup when starting sheets may have to be purchased from outside sources to augment the production being built up in-house.

Taking all these into account, a size of 39- by 39-in. (1- by l-m) immersed cathode is appropriate. This is the commonest size in use. The spacing and number of cathodes per cell is based largely on personal experience and preference; however, there are some factors to consider:

1. Cathode spacings generally fall in the range of 3 to 4 in. (75 to 115 mm). The shorter dimension favors shorter cells (or more cathodes per cell) and a lower electrolyte IR drop. The wider spacing can result in improved flow conditions between electrodes, use of thicker (longer life) anodes, and some decrease in electrical shorts.

2. Over the years, the number of cathodes per cell has increased up to around 60. The main factors here are electrolyte flow patterns and keeping the cell short enough to allow for efficient crane handling of the cathodes. It is usual practice to choose a number divisible by 3 and (often, as well, by 2, e.g., 48). This simplifies cell-pulling techniques where every third cathode is pulled from the cell by the crane bale. Thus, as this pulling is done "live" with no interruption of power to disturb the anodic condition of the lead anodes, the current density on the cathodes remaining in the cell is increased only by 50% during the short duration of the pull.

Improvements in operating costs arise from fewer manual operations generated from fewer EW cells (when these are of maximum size and operate at the highest current density commensurate with product purity), improved tankhouse layout, and reduced stray current losses. For this last item, it is unusual for modern tankhouses to experience stray current losses in excess of 1%. This has largely been due to the widespread use of plastic piping systems for electrolyte recirculation in place of the earlier lead piping and the adoption of nonconducting cell linings.

Improvements in cathode quality have allowed the newer EW facilities to consistently produce product better than 99.99% Cu and, at least in one case, better than 99.999%. This improvement has arisen since the advent of solvent extraction of leach solutions, which enables electrowinning tankhouse operators to electrolyse very pure electrolytes. A difficulty remains in that the acid content required for effective stripping of the loaded organic produces an electrolyte of considerably higher acidity than that found in traditional leach solutions. This acid is sufficient to attack the lead anodes usually used in electrowinning, but the use of alloyed lead (with calcium, antimony, tin, etc.) and cobalt additions can control this phenomenon. It has been known since the 1920s that the presence of up to 100 ppm of cobalt in copper sulphate electrolyte modified the morphological structure of the oxide layer of a polarized lead anode, with a high proportion of PbO_2 being formed.

This earlier work has been rediscovered and the technique is now widely applied in tankhouses associated with solvent extraction plants. Cobalt levels are usually in the range of 60 to 100 ppm, partly dependent on current density. Very high-purity cathodes are obtained using such cobalt-modified anodes with usual lead levels of less than 5 ppm being reported. A similar preferred formation of a PbO_2 is observed if the anodes are pretreated by electrolysing in fluoride solutions before loading into the tankhouse cells. This is a well-known technique used with the lead/silver anodes of zinc electrowinning.

A drawback in the system is inherent in the solvent extraction unit operation. This is the carry-over of iron by the reagent and the necessary bleeding off of this iron from the tankhouse to control electrolyte iron levels. This bleed also causes cobalt losses which is a high-cost additive.

Of major concern in modern solvent extraction/electrowinning installations is the generation of acid mist which collects in the tankhouse building above the cells. This is caused by anodicly generated oxygen bubbles carrying the mist up from the electrolyte surface in the cell. In earlier leach/electrowin units, the problem was largely overcome by the use of foam-generating additives, such as Dowfax 2A1, which provide a foam blanket on the cell surfaces. Unfortunately, virtually all of the previously used surfactant materials are incompatible with solvent extraction circuits where their surfactant properties have deleterious effects on settling and extraction characteristics. AFFF type foams are suitable, but dosage control has been a problem. An effective solution is to use two or three layers of polyolefin balls of proprietary design on the cell surfaces. These reduce mist generation by 70 to 80%. There are objections to their use, usually quoted as “they get everywhere.” Electrowinning plants of large size associated with solvent extraction plants have been forced to use these ball covers to meet OSHA (Occupational Safety and Health Act) requirements, and as a result, excellent cell designs have been developed that retain the balls on the electrolyte surface very effectively.

If the proper precautions are taken, plants can comply with the local equivalent of the 1 mg/m^3 standard for an 8-hour exposure as required by OHSA in the United States. To achieve this level, the plant would operate at 0.8 to 1.0 lb/gal (100 to 130 g/L) H_2SO_4 and current density of approximately 18.5 to 23 A/ft^2 (200 to 250 A/m^2).

25.4.2.4 Electrorefining

Unlike an EW operation, which merely recovers copper, *electrorefining* actually purifies the copper. Electrolytic refining of copper starts with an impure copper anode cast directly from a refinery furnace. The impure anode and a pure copper cathode, usually a starting sheet, are suspended in an electrolyte of copper sulfate and sulfuric acid. Direct current applied to the cell causes the anode copper to go into solution with corresponding amounts of copper being plated out onto the cathode. The electrochemical reactions at the electrodes are equivalent, but reversed, and the overall component of cell voltage from the electrode reactions is zero. There is essentially no net change of copper content in the solution, the only changes in solution composition being increases in the contents of impurities in the electrolyte. Metals above Cu in the electrochemical series will not discharge at a cathode as long as copper ions are present. Thus copper can be separated from other metals and impurities in an electrolyte bath. Copper is sufficiently electropositive to hydrogen in this solution to allow its deposition at high-current efficiency within a wide range of concentration, electrolyte temperature, and current density.

In contrast to electrowinning where precious metal values in the ore body are left in the gangue during leaching, the electrorefinery is able to recover these important constituents. Such metals as gold, silver, platinum, palladium, selenium, and tellurium report through upstream processing and finally report the blister copper cast into the cell anodes. During electrolysis, these impurities fall to the cell floor (rather than deposit), forming slimes that are further treated for their recovery. This anode slime must be removed after every load of anodes or every second load have been consumed (i.e., every one or two months). The refinery has two conditions to deal with apart from deposition of copper at the cathode: increasing impurities in the electrolyte and problems due to the presence of significant amounts of slimes. As stated above, the latter are cleared from the anodes and cells, but the only method of electrolyte purification is to drain off and discard some electrolyte to keep the concentration of impurities below a certain economic limit.

Improvements in electrorefining have been made in recent times mostly in the realm of automation to reduce labor costs: (1) infrared spotters can detect shorts, (2) laser spotters can position crane loads exactly above the cells, and (3) “jumbo” cells can save on capital cost and reduced handling. These improvements are best exemplified by the Naoshima refinery in Japan and the 419,000-tpy (380,000-t/a) Amarillo plant of Asarco in Texas. Automated anode handling systems take loads of closely stacked anodes, mill or twist the lugs so they hang true in the cells, separate them at the correct cell spacing, and place them along conveyors or in racks ready for pick up by the crane. Starting sheet stripping lines are partially automated; blanks are stripped and prepared on a conveyor and accumulated at the correct spacing for crane pickup. The conveyor passes on to take up deposit blanks left by the crane; these are dip washed and stripped (usually manually) of deposits by strippers working on both sides of the blanks as they pass by on the conveyor. Blanks are then returned for reloading into the cells.

Improvements are also being made in the area of impurity control in the circulating electrolyte and better control of addition agents (glue, thiourea, chloride) used to promote smooth growth. The newest ideas for impurity control will see increasing use of solid and liquid ion exchange/solvent extraction, tailor-made for removal of specific impurities.

25.4.3 ZINC PROCESSING

WAYNE R. HOPKINS

25.4.3.1 Solvent Extraction

The production of special high-grade (SHG) zinc from sulfide concentrates utilizes a final zinc electrowinning step. The successful electrodispositon of zinc in this process depends upon solution purity, but there was no universally applicable solution purification process available until the advent of the Espindesa Process by Technicas Reunidas in Spain. This process employs a two-stage solvent extraction operation. The first stage extracts pure zinc chloride from an impure feed leach liquor, and the second stage produces a pure zinc sulfate solution from the zinc chloride. The extreme selectivity of the solvent extraction stages permits removal of those impurities deleterious to electrolysis.

An anion extraction reagent, a secondary amine, that removes the zinc as an anionic chloride complex is used in the first step. Extraction proceeds according to Eq. 25.4.3.1.

$$2R_2NH_2Cl_{(org)} + ZnCl_{2(aq)} \rightarrow (R_2NH_2)_2ZnCl_2{}^{2+}{}_{(org)} + 2Cl^-{}_{(aq)} \quad (25.4.3.1)$$

This is followed by a washing stage where the loaded organic is washed in acidified water to remove entrained aqueous feed and any other metal ions that may have been co-extracted with the zinc into the organic. The wash water is returned to the extraction stage, joining the feed. Next, the zinc is stripped from the organic with water in accordance with the reaction:

$$(R_2NH_2)_2ZnCl_{4(org)} + H_2O_{(aq)} \rightarrow 2R_2NH_2Cl_{(org)} + Zn^{2+}{}_{(aq)} + 2Cl^-{}_{(org)} \quad (25.4.3.2)$$

The organic extractant is regenerated and can be returned to the extraction stage. Only copper and cadmium are partially extracted with the zinc and pass on to the second part of the extraction process. In this second extraction step, the zinc is removed as the normal metal cation Zn^{++} using di-2-ethyl hexylphosphoric acid (D2EHPA) as the active reagent, which is also diluted in kerosene. The reaction is

$$2(RO)_2PO_2H_{(org)} + Zn^{2+}{}_{(aq)} \rightleftharpoons [(RO_2)PO_2]_2Zn_{(org)} + 2H^+{}_{(aq)} \quad (25.4.3.3)$$

Control of pH is important. At pH values above 2.8, Cu and Cd can co-extract with the zinc, while low pH reaction kinetics are slow. Loaded organic phase is first washed with dilute acid to remove aqueous entrainment, particularly chloride ions. Any copper and cadmium ions extracted with the zinc in the first stage are removed in the second-stage raffinate. Spent sulfate electrolyte from the zinc electrowinning plant is used to strip the loaded organic. This produces a strong electrolyte at 0.6 to 0.7 lb/gal (80 to 90 g/L) zinc. Any organic entrainment is reduced to 1 to 5 ppm by coalescence using settling tanks and charcoal filters.

25.4.3.2 Electrowinning

In this process step, the purified zinc sulfate solution is decomposed by direct current; zinc is deposited at the cathode, oxygen is liberated at the anode, and sulfuric acid is formed by the combination of hydrogen and sulfate ions. This produces the spent electrolyte that is used to leach the zinc calcine or to strip the solvent extraction organic reagent.

Cells are of concrete construction similar to copper electrowinning and are lined with PVC. The number of electrodes per cell is typically 35 to 50. The newer type anodes are constructed of a 1% silver-lead alloy. Alloyed anodes reduce voltage and slightly decrease the current efficiency but cause less contamination of the cathode with lead. Cathodes are constructed of aluminum sheet. Both anode and cathode have copper head bars. Typical electrode dimension are 40 by 25 in. (1000 by 600 mm). In some cases, where mechanical striping and handling are employed, electrodes up to 60 by 35 in. (1500 by 900 mm) are used. Cells are connected electrically in series using Walker multiple connections, and are usually arranged in units of 150 to 200 cells. The cells are arranged in tiers or "half sections" in a similar way to copper tankhouses, with current being carried to and from these and across the back busbars by copper busbar multileaf units.

Neutral solution is added to the electrolytic cells in controlled amounts, using a circulation circuit similar to that for copper electrowinning. The objective is to keep the zinc content and the acid content of the electrolyte constant. In the so-called low-acid plants, acid strength of the electrolyte is held at a level between 0.8 and 1.6 lb/gal (100 and 200 g/L) of sulfuric acid. The deposit character is modified using additions of glue, gum arabic, silicic acid, and strontium carbonate.

Current density, varies among plants from 18.5 to over 90 A/ft^2 (200 to over 1000 A/m^2). Most plants operate in the range of 55 to 65 a/ft^2 (600 to 700 A/m^2). In general, a higher current density means higher voltage per cell, higher acid strength needed, increased cooling load, faster deposition, less floor space and less solution to handle, better purification, closer electrode spacing, and a need for heating leach solutions. Choice of current density depends on how these various factors balance up at each plant location.

Frequent cleaning of cells is also necessary owing to the slimes that form on the anodes and on the cell bottoms. This is largely manganese dioxide, brought in with the electrolyte, where it is added to oxidize iron in the neutral leach. It forms on the anode as a slime coating, and although much of it falls off as a sludge into the bottom of the cell, the remainder has to be removed once it builds up beyond a certain point.

In modern installations, the deposited zinc is mechanically stripped from the aluminum cathodes. The principle of all mechanical stripping is similar; that is, the cathode is loosened by hammering or by short-travel, pneumatically operated knives, then the deposit is removed by a large knife moving across or down the face of the cathode. Mechanical stripping has been made possible by the ability to control the complete process of purification and electrolysis to achieve very consistent conditions. As a result, the adhesion of the zinc to the aluminum cathode remains constant, and the zinc can be removed using the same mechanical effort each time. Fully automated tankhouses are now operating with sophisticated computer logic control of all operations.

25.4.4 MOLTEN SALT ELECTROLYSIS

RICHARD B. STEIN

Three important metals are produced industrially by electrolysis in fused salts—aluminum, magnesium, and sodium—and their annual production is on the order of thousands of tons (tonnes): aluminum 1200 (1100), magnesium 220 (200), and sodium 75 (160). Aside from magnesium (where about 35% of its

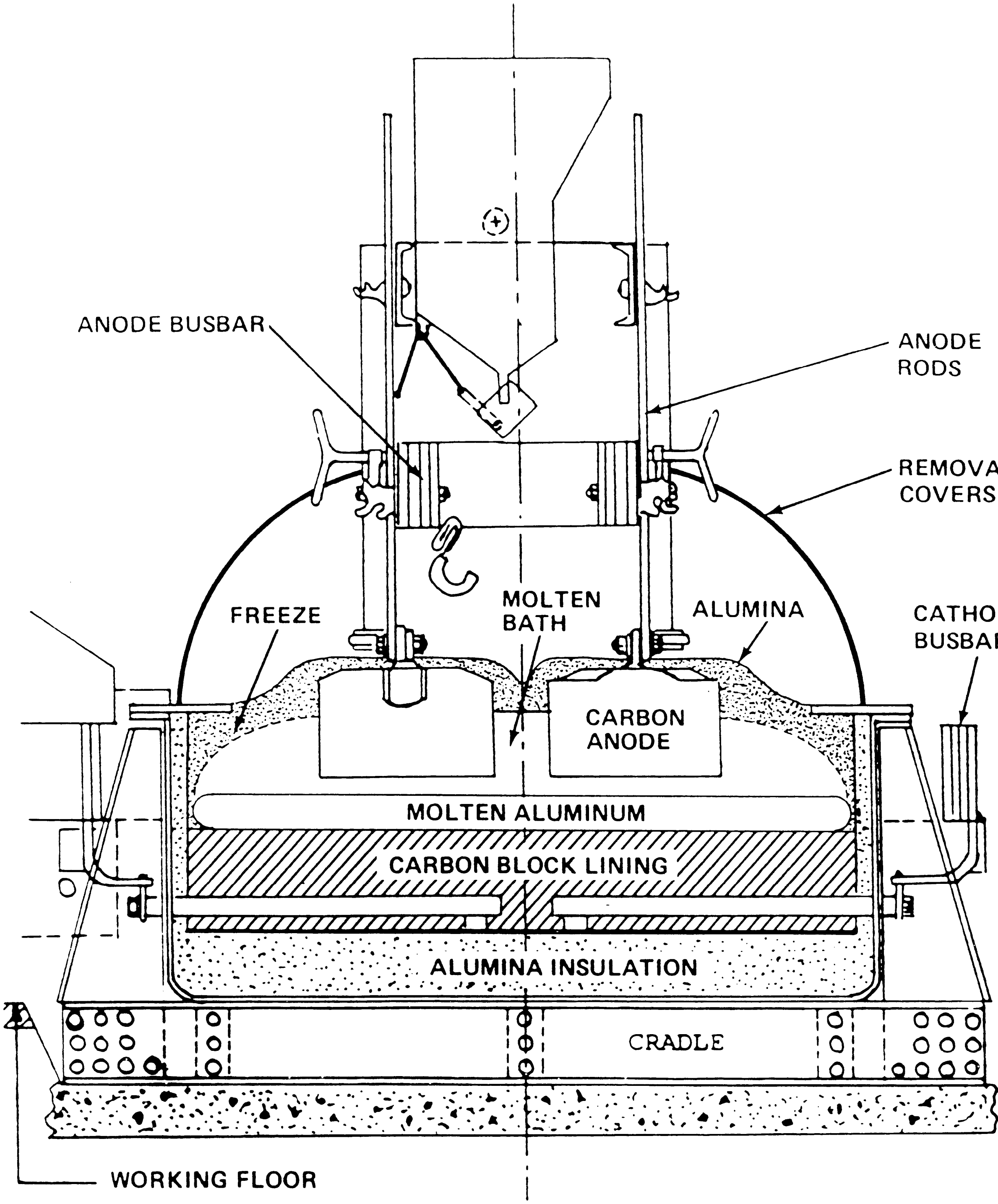

Fig. 25.4.4.1. Typical aluminum reduction cell.

production is by thermic reduction using ferrosilicon), there is no alternative method for other commercial-scale production. Magnesium and sodium are electrolyzed in mixed chloride baths at operating temperatures of 1076 to 1292°F (580 to 700°C) employing eutectic salt baths to lower the temperature. Aluminum electrolysis is practiced in a molten fluoride bath at about 1760°F (960°C).

25.4.4.1 Aluminum

Aluminum was an exotic rare metal until 1888 when Hall in America and Heroult in France independently discovered that aluminum oxide could be dissolved in the natural fluoride mineral cryolite, which is an equimolal mixture of sodium and aluminum fluorides. Cryolite melts at 1854°F (1012°C) and when

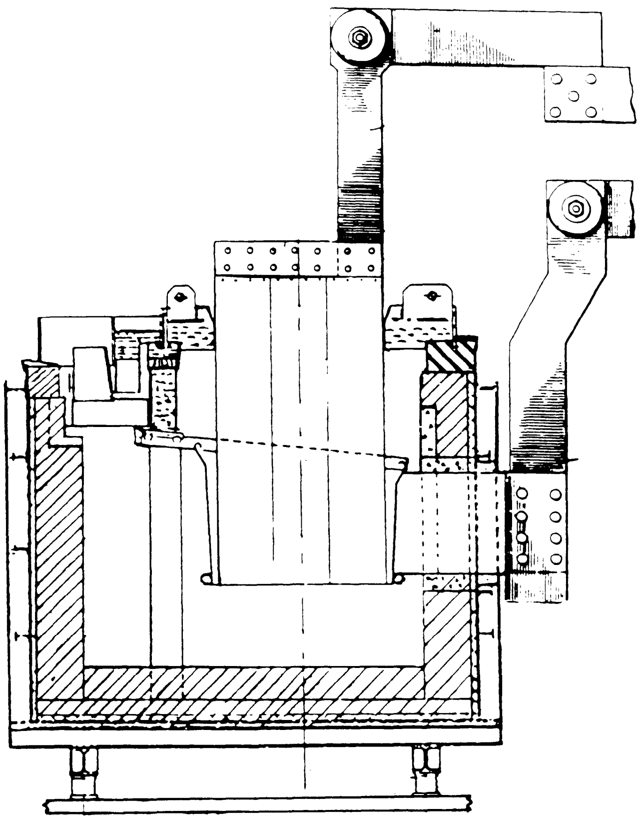

Fig. 25.4.4.2. Magnesium electrolyzer cell (Can. Patent No. 3,396,094).

it dissolves about 2 to 5% aluminum oxide, the bath is sufficiently molten to ionize the aluminum oxide. This bath will support electrolysis at 1724 to 1796°F (940 to 980°C) due to fluid action of bath additions, typically lithium, calcium, and aluminum fluorides.

Over the past 100 years, electrolytic cell design has matured, and in recent years it was driven by considerations of intensive capital costs, high energy costs, and need to eliminate fluoride emissions. Most modern aluminum reduction plants typically employ a line of cells of the type shown in Fig. 25.4.4.1, arranged in a potroom about 1300 ft (400 m) long and containing sufficient pots to produce 120,000 tpy (109,000 t/a) of metal.

The cells measure typically 33 to 46 ft (10 to 14 m) long, 10 to 16 ft (3 to 5 m) wide and 3 to 5 ft (1 to 1.5 m) high and are constructed of an outer steel shell, thermal insulation, carbon block lining, and suspended prebaked anodes of semi-graphitized petroleum coke. Even though the size and shape have continually changed because of the combined corrosive action of molten fluroides and molten metal, the materials of construction have remained essentially unaltered. The individual cells are hooded with covers, and an efficient gas collection and scrubbing system eliminates the cell gas emissions.

Within the cell electrolyte, the aluminum oxide is ionized, and metal is deposited at the cathode, and oxygen and carbon react at the bath-anode interface. The overall chemical reaction is

$$2Al_2O_3 + 3C \rightarrow 4Al + 3CO_2\uparrow \qquad (25.4.4.1)$$

whereby about 2 lb (0.9 kg) of Al_2O_3 and 0.5 lb (0.2 kg) of carbon (consumbable anode) make 1 lb (0.45 kg) of metal and 1.2 lb (0.54 kg) of gas to the scrubbers.

Economic trade-offs have driven the pot current to about 180,000 to 300,000 A at about 4.5 V dc across the cell. The cell inefficiences of ohmic resistance and polarization furnish the internal heat to maintain the high temperature. Power consumption is about 6.1 kWh/lb (13.5 kWh/kg) of metal product.

In operation, the cell contains about 0.5 to 4.0 in. (10 to 100 mm) of molten aluminum lying on the carbon cathode. A layer of 5 to 10 in. (120 to 250 mm) of molten electrolyte floats atop the metal pad, and above the molten electrolyte is a crust of frozen electrolyte. A special pot-tending, crust-breaking machine is used to make holes in the crust on a fixed schedule for alumina addition to restore the bath composition. Another special machine carries a vacuum crucible for aspirating a cell-day of production, typically one ton of metal, which is delivered to the casthouse for purification and ingot casting.

In addition to metal production and handling, a modern plant contains facilities for alumina storage; mixing, forming and baking of anodes; pot rebuilding; machinery services; pollution control; electric power; transportation, etc.

25.4.4.2 Magnesium

About three-quarters of the world's magnesium is produced by fused salt electrolysis, of which about one-half is based on sea water and the balance on brines, bitterns, carnallite, or other sources of magnesium chloride.

In the seawater process, the magnesium content of seawater is precipitated with calcined dolomite, and the resulting magnesium hydroxide is converted to chloride using recycled hydrochloric acid from the electrolytic cells. After recovery of magnesium chloride from the seawater, the cell feed is a hemihydrate, $MgCl_2 \cdot ½H_2O$, whose electrolysis requires consumable graphite anodes (as in aluminum reduction) at the rate of 0.1 lb/lb (kg/kg) of metal, plus external heat. Power consumption is said to be 8.0 kWh/lb (17.6 kWh/kg) for this process, which dates back to World War II.

The latest magnesium processes use anhydrous chloride and permanent graphite anodes which last from 2 to 5 years. The cells are completely sealed, and a typical cell will produce about 1 tpd (0.9 t/d) of metal and nearly pure chlorine gas at about 2.5 tons/ton (t/t) of metal. The process can start with chloride-containing brines that require purification, evaporation, dehydration, and chlorination to produce molten, anhydrous cell feed, or else can start with magnesite and carbon and use cell chlorine to make molten, anhydrous cell feed in specialized reactors upstream of the electrolysis.

In the modern diaphragmless cell in Fig. 25.4.4.2, the electrolyte is a mixture of $MgCl_2$-KCl-NaCl-$CaCl_2$ at 1292°F (700°C) with a density such that magnesium metal product floats, rather than sinks as in aluminum electrolysis. Each graphite block anode is surrounded by a steel cathode with sloping ends and sides which induce convective flow, causing metal globules to rise and coalesce in the upper collection pipe and then be conveyed to metal collection chamber at the front of the cell. Metal is removed once or twice per day by a vacuum crucible and taken to the casthouse for pouring into ingots. The power consumption for this type of electrolytic cell is about 6.35 kWh/lb (14 kWh/kg) of metal product, and compared to the seawater process consumes essentially no graphite and produces a pure chlorine byproduct.

Chapter 25.5
SALES AND MARKETING

BRIAN NOLK

The sales and marketing of mineral products is a distinctive activity within a mining company and operates in a relatively subjective mode rather than the more familiar scientific modes of geology, mining engineering, etc. This chapter attempts to explain the often volatile and seemingly erratic operations of the marketing process to the more classically trained person. Information on pricing and trading is presented in Chapter 2.3.

It is important to remember that the engineering and operationing functions of any company are affected by the success, or failure of the marketing staff, and that there is no point in mining or processing any material that cannot be sold at a profit. Geologic, engineering, and operations professionals are forced to work within a multi-year time frame for the discovery, delineation, study, and engineering development of mineral deposits. For example, a large copper deposit may require as long as seven years to develop. Due to the evergrowing complexity of the environmental requirements, this period is getting even longer. It follows that any economic considerations that can delay the process, or terminate it completely by undermining the viability of the entire deposit, are often considered contrary and unfair by those who have worked so hard and so well for the project.

However, the marketing strategies that a company may choose to employ can work to ameliorate the risks inherent in any mining venture. The more a company knows about the commercial viability of a project, and the ways the viability can be enhanced, the easier it is for that company to get on with the engineering and mining tasks. It is a matter of knowing which strategies exist and which may best fit a given enterprise.

Included below are some basics of marketing and sales for the layman to consider. The hope is that understanding the mechanics of these economic forces will make living with them easier.

25.5.1 MARKETING PHILOSOPHY

The linkage in the long chain between producers and end consumers, connecting each group to the next one down the material chain, is the sales and marketing activity. It must be remembered that a material is often sold at several stages of its life. In the case of copper, for example, sales activity can occur at the stage of ores and concentrates, intermediate metal (such as blister copper), refined metal (such as electrowon or electrolytic cathode), alloyed unwrought metal (such as brass billet), semifinished products, finished products, and back again as scrap.

At each step of this chain, the material is bought and sold or, if processed internally, at least priced for that purpose. At each step, the producer may seek to hedge his risks and to price his physical inventory for accounting and insurance purposes. Thus "sold" is meant to include the step in an internal process where the value of the material is determined. Again, to use copper as an example, a copper mining company will want to know the market for concentrates as well as that for blister, to determine whether it is worthwhile building, or more commonly, upgrading a smelter. The market for concentrates can often be completely different from that of the metal, and a business decision will have to be made regarding the disposition of concentrates.

The internal selling of concentrates to a company's own smelter is a valid marketing exercise. It helps a company evaluate whether its materials should be continued on down the chain, or whether the owner should halt the process and either find a substitute source or substitute material, or in some other way interrupt the chain.

The relative glut or shortage of a material at any stage in this chain will have some effect on the sales and marketing of the other materials along the chain, but the glut or shortage may not affect the entire chain equally. For example, if a disruption occurs at a major copper refinery, there may be a temporary shortage of refined metal and by-products due to the refinery problem. Accordingly, the price of copper cathode may jump sharply as available refinery capacity limits the immediate availability of refined metal.

What must be remembered by participants in the mining industry is that the increasing internationalization of the minerals commodities markets will cause any relative surplus or shortage of a material on the other side of the world to have some impact on the domestic market, either directly or indirectly. While this has been always true of many minerals, the impact is greater now than it ever was. This is something to remember when one discovers that the market price for the mineral product has dropped since last year, even though the domestic demand has remained roughly the same. If a new project in Brazil or China has come on-line in the interim, the economics of the world market have changed, and there is no use fighting facts.

25.5.2 TYPES OF TRANSACTIONS

It is necessary to sort out the types of transactions that fall within the marketing and sales framework. Some of the sales transactions at each of these individual steps are quite straightforward, involving individual contracts of sale with fixed prices and tonnages, and some are quite complex, involving multi-year contracts with periodic pricing linked to terminal exchange price formulas for various levels of tonnage, sometimes involving currencies other than the one the producer would prefer. A description follows of the various methods of marketing and pricing, from the most straightforward to the most complex, with emphasis on how these prices are used by the industry.

25.5.3 PRODUCER PRICING

Still quite common in the United States and some other areas of the world, producer pricing is a straightforward mechanism by which a minerals producer establishes a price for its product by announcing to key customers and the media a fixed price of the day (or month, quarter, etc.). Producers set prices they feel are reasonable and competitive, and which they have the authority to alter at their convenience. Consumers can contract in set amounts, commonly in increments of short tons per month for delivery, with the pricing set as "pricing in effect at time of delivery." In general, it is possible for producers to maintain producer pricing control of their products when there are few competing producers of the same material or where the products are so specialized that it is not a commodity at all. In those instances, producers can maintain contractual relationships with their specific customers, based on an annual or quarterly negotiation.

An example, in copper, would be specific purity specifications of copper such as oxygen-free, high conductivity (OFHC), or a specialized fabricated shape. Other materials which still use producer pricing are specialized ferroalloys with specific compositions.

Producers have other pricing options that they may from time to time employ. In times of price volatility, they may wish to abrogate some of their pricing responsibilities. Rather than forcing their marketing staff to make daily, or even hourly, determinations of fair selling value of the company's products, they may wish to hitch their entire marketing wagon to a free market indicator, such as an exchange price like the London Metal Exchange (LME) or Commodity Exchange (Comex) futures contract, or even to a published free market transaction quote, such as is published by international metals journals (e.g., *Metal Bulletin* or *Metals Week*). This system uses a deferred price system, since neither the seller nor the consumer knows the exact level of price that will be in effect when the metal is delivered. Nonetheless, they both know that it will be in the parameters of the existing free market. If a copper producer sets his copper cathode price on the basis of an exchange price—say, the Comex average settlement price for the month of shipment of this product— plus 4.5¢/lb (9.9¢/kg) for his margin, he can sell his metal forward. If he can then spot buy his raw materials—say, blister copper at Comex minus 15¢/lb (33¢/kg)—then he has effectively locked in a margin. Regardless of where the market goes, he is settled in at a profitable margin, assuming that 19.5¢/lb (43¢/kg) is sufficient to cover his refining and handling costs, with profit included.

The message is that both buyers and sellers need to choose the same pricing mechanisms, whether the producer price, or other free market indicators, whether a published free market quote or an exchange based price. Then both can proceed with some confidence to build in a margin which comfortably covers their operations.

25.5.4 EXCHANGE BASED PRICING

An area of increasing importance for the metals and minerals industry is the trading in futures and options contracts for these materials. No producers can afford to ignore the impact of exchange trading of metals as the basis of price discovery and hedging potential. Exchange trading of metals has a long history, with a recognizable forerunner of the London Metal Exchange setting up shop in the 1870s to trade in copper and tin. The LME is still a major center for trading in copper, as well as aluminum, lead, zinc, and nickel.

Other exchanges and markets have long established roles in the metals world, particularly Switzerland and London for gold, where various groups have helped establish a daily quote for bullion. In the United States, there are several exchanges with well established contracts for trading metals, including the Commodity Exchange, in New York and New York Mercantile Exchange and the three Chicago exchanges, the Chicago Board of Trade, the Chicago Mercantile Exchange, and the MidAmerica Commodity Exchange. The Kuala Lumpur Commodity Exchange has a longstanding contract for trading in tin, and the Tokyo Commodity Exchange has a platinum contract of growing importance to the industry.

Exchange trading of metals offers several important services to the industry as a whole, even for those who do not actively participate in the exchange trading itself. Chief among the services offered is *price discovery,* for the commodities traded and by extension to the comparable materials both up and down the materials chain, by the use of premiums and discounts from the exchange traded price.

The importance of prices, as discovered by terminal exchanges, is that they are within the scope of the various contract specifications, quite representative of world prices for the commodities in the various currencies and grades used by consumers. Thus people who might immediately question the value of a terminal market price for a metal that they buy or sell in a different geographic area can be made to see that the price discovery service offered by an exchange is, in many cases, one of the best systems over the longer term.

A good example of the phenomenon is in copper, where several exchanges offer contracts in the metal, all with varying specifications and in several different currencies, but usually within small percentage points of each other despite the widely diverging needs, supply factors, currency fluctuations, and underlying economic factors. The price set by the exchange(s) is the price of copper, like it or not, and that is the foundation of the exchanges.

Appendix

Table A. Conversion of English to SI Units

Table B. Strength Properties of Rocks

Table C. Engineering Properties of Rock

Table D. Properties of Common Minerals

Table E. Material Properties and Characteristics

Table F. Heat Value of Various Fuels

Table G. Geometrical Relationships and Mensuration Formulas

Table H. Standard Sizing and Meshes

Table I. Ship Measures

Table J. International Atomic Weights

Table K. Time Value of Money Factors for Discrete Compounding

Table A. Conversion of English to SI Units, Listed Alphabetically

(Symbols of SI units given in parentheses)

To convert from	to	Multiply by	
abampere	ampere (A)	1.000 000*	E+01
abcoulomb	coulomb (C)	1.000 000*	E+01
abfarad	farad (F)	1.000 000*	E+09
abhenry	henry (H)	1.000 000*	E -09
abmho	siemens (S)	1.000 000*	E+09
abohm	ohm (Ω)	1.000 000*	E -09
abvolt	volt (V)	1.000 000*	E -08
acre foot (US survey)[1]	meter^3 (m^3)	1.233 489	E+03
acre (US survey)[1]	meter^2 (m2)	4.046 873	E+03
acre	hectare (h)	4.046 873	E-01
ampere hour	coulomb (C)	3.600 000*	E+03
are	meter^2 (m^2)	1.000 000	E+02
angstrom	meter (m)	1.000 000*	E-10
astronomical unit	meter (m)	1.495 979	E+11
atmosphere (standard)	pascal (Pa)	1.013 250*	E+05
atmosphere (technical = 1 kgf/cm^2)	pascal (Pa)	9.806 650*	E+04
bar	pascal (Pa)	1.000 000*	E+05
barn	meter^2 (m^2)	1.000 000*	E-28
barrel (for petroleum, 42 gal)	meter^3 (m^3)	1.589 873	E-01
board foot	meter^3 (m^3)	2.359 737	E-03
British thermal unit (International Table)	joule (J)	1.055 056	E+03
Btu (International Table)•ft/h•ft^2•°F (*k*, thermal conductivity)	watt per meter kelvin (W/m•K)	1.730 735	E+00
Btu (International Table) in./h•ft^2•°F (*k*, thermal conductivity)	watt per meter kelvin (W/m•K)	1.442 279	E-01
Btu (International Table) in./s•ft^2•°F (*k*, thermal conductivity)	watt per meter kelvin (W/m•K)	5.192 204	E+02
Btu (International Table)/h	watt (W)	2.930 711	E-01
Btu (International Table)/ft^2	joule per meter^2 (J/m^2)	1.135 653	E+04
Btu (International Table)/h•ft^2•°F (*C*, thermal conductance)	watt per meter^2 kelvin (W/m^2 •K)	5.678 263	E+00
Btu (International Table)/s•ft^2•°F	watt per meter^2 kelvin (W/m^2•K)	2.044 175	E+04
Btu (International Table)/l°	joule per kilogram (J/kg)	2.326 000*	E+03
Btu (International Table)/lb•°F (c, heat capacity)	joule per kilogram kelvin (J/kg•K)	4.186 800*	E+03

[1]Conversion factors for land measure may be determined from the following relationships:

1 league = 3 miles (exactly)
1 rod = 16-1/2 feet (exactly)
1 section = 1 square mile (exactly)
1 township = 36 square miles (exactly)

Note: * means exact value.

To convert from	to	Multiply by
bushel (US)	meter^3 (m^3)	3.523 907 E-02
caliber (inch)	meter (m)	2.540 000* E-02
calorie (International Table)	joule (J)	4.186 800* E+00
calorie (kilogram, International Table)	joule (J)	4.186 800* E+03
cal (International Table)/g	joule per kilogram (J/kg)	4.186 800* E+03
cal (International Table)/g•°C	joule per kilogram kelvin (J/kg•K)	4.186 800* E+03
cal (thermochemical)/min	watt (W)	6.973 333 E -02
cal(thermochemical)/cm^2 •min	wattpermeter^2 (W/m^2)	6.973 333 E+02
carat (metric)	kilogram (kg)	2.000 000* E-04
centimeter of mercury (O °C)	pascal (Pa)	1.333 22 E+03
centimeter of water (4°C)	pascal (Pa)	9.806 38 E+01
centipoise	pascal second (Pa•s)	1.000 000* E-03
centistokes	meter^2 per second (m^2/s)	1.000 000* E-06
circular mil	meter^2 (m^2)	5.067 075 E -10
clo	kelvin meter^2 per watt (K•m^2/W)	2.003 712 E -01
cup	meter^3 (m^3)	2.365 882 E -04
curie	becquerel (Bq)	3.700 000* E+10
day (mean solar)	second (s)	8.640 000 E+04
day (sidereal)	second (s)	8.616 409 E+04
degree (angle)	radian (rad)	1.745 329 E -02
degree Celsius	kelvin (K)	$t_{\mathrm{K}} = t_{^\circ\mathrm{C}} + 273.15$
degree Fahrenheit	degree Celsius	$t_{^\circ\mathrm{C}} = (t_{^\circ\mathrm{F}} - 32)/1.8$
degree Fahrenheit	kelvin (K)	$t_{\mathrm{K}} = (t_{^\circ\mathrm{F}}+459.67)/1.8$
degree Rankine	kelvin (K)	$t_{\mathrm{K}} = t_{^\circ\mathrm{R}}/1.8$
°F h•ft^2/Btu (International Table) (R, thermal resistance)	kelvinmeter^2 per watt (K•m^2/W)	1.761 102 E-01
denier	kilogram per meter (kg/m)	1.111 111 E-07
dyne	newton (N)	1.000 000* E-05
dyne•cm	newton meter (N•m)	1.000 000* E-07
dyne/cm^2	pascal (Pa)	1.000 000* E-01
electronvolt	joule (J)	1.602 19 E-19
EMU of capacitance	farad (F)	1.000 000* E+09
EMU of current	ampere (A)	1.000 000* E+01
EMU of electric potential	volt (V)	1.000 000* E-08
EMU of inductance	henry (H)	1.000 000* E-09
EMU of resistance	ohm (Ω)	1.000 000* E-09
ESU of capacitance	farad (F)	1.112 650 E-12
ESU of current	ampere (A)	3.335 6 E-10
ESU of electric potential	volt (V)	2.997 9 E+02
ESU of inductance	henry (H)	8.987 554 E+11
ESU of resistance	ohm (Ω)	8.987 554 E+11
erg	joule (J)	1.000 000* E-07
erg/cm^2•s	watt per meter^2 (W/m^2)	1.000 000* E-03
erg/s	watt (W)	1.000 000* E-07
faraday (chemical)	coulomb (C)	9.649 57 E+04
faraday (physical)	coulomb (C)	9.652 19 E+04
fathom	meter (m)	1.828 8 E+00
fermi (femtometer)	meter (m)	1.000 000* E-15
fluid ounce (US)	meter^3 (m^3)	2.957 353 E-05

To convert from	to	Multiply by	
foot	meter (m)	3.048 000*	E-01
foot (US survey) [1]	meter (m)	3.048 006	E-01
foot of water (39.2°F)	pascal (Pa)	2.988 98	E+03
ft^2	$meter^2$ (m^2)	9.290 304*	E-02
ft^2/h (thermal diffusivity)	$meter^2$ per second (m^2/s)	2.580 640*	E-05
ft^2/s	$meter^2$ per second (m^2/s)	9.290 304*	E-02
ft^3 (volume; section modulus)	$meter^3$ (m^3)	2.831 685	E-02
ft^3/min	$meter^3$ per second (m^3/s)	4.719 474	E-04
ft^3/s	$meter^3$ per second (m^3/s)	2.831 685	E-02
ft^3/ton (tonnage factor)	$meter^3$ per tonne (m^3/t)	3.121 39	E -02
ft^4 (moment of section)[2]	$meter^4$ (m^4)	8.630 975	E-03
ft/h	meter per second (m/s)	8.466 667	E-05
ft/min	meter per second (m/s)	5.080 000*	E-03
ft/s	meter per second (m/s)	3.048 000*	E-01
ft/s^2	meter per $second^2$ (m/s^2)	3.048 000*	E-01
ft/ton (drilling factor)	meter per tonne (m/t)	3.359 8	E-01
footcandle	lux (lx)	1.076 391	E+01
footlambert	candela per $meter^2$ (cd/m^2)	3.426 259	E+00
ft•lbf	joule (J)	1.355 818	E+00
ft•lbf/h	watt (W)	3.766 161	E-04
ft•lbf/min	watt (W)	2.259 697	E-02
ft•lbf/s	watt (W)	1.355 818	E+00
ft•poundal	joule (J)	4.214 011	E-02
ft•ton (moment)	meter•tonne (m•t)	2.765 1	E-01
free fall, standard (*g*)	meter per $second^2$ (m/s^2)	9.806 650*	E+00
gal	meterper$second^2$ (m/s^2)	1.000 000*	E-02
gallon (Canadian liquid)	$meter^3$ (m^3)	4.546 090	E-03
gallon (UK liquid)	$meter^3$ (m^3)	4.546 092	E-03
gallon(US dry)	$meter^3$ (m^3)	4.404 884	E-03
gallon (US liquid)	$meter^3$ (m^3)	3.785 412	E-03
gal (US liquid)/day	$meter^3$ persecond(m^3/s)	4.381 264	E-08
gal (US liquid)/min	$meter^3$ persecond(m^3/s)	6.309 020	E-05
gal (US liquid)/hp•h (SFC, specific fuel consumption)	$meter^3$ per joule (m^3/J)	1.410 089	E-09
gamma	tesla (T)	1.000 000*	E-09
gauss	tesla (T)	1.000 000*	E-04
gilbert	ampere (A)	7.957 747	E-01
gill (UK)	$meter^3$ (m^3)	1.420 654	E-04
gill (US)	$meter^3$ (m^3)	1.182 941	E-04
grad	degree (angular)	9.000 000*	E-01
grad	radian (rad)	1.570 796	E-02
grain (1/7000 lb avoirdupois)	kilogram (kg)	6.479 891*	E-05
grain (lb avoirdupois/7000)/gal (US liquid)	kilogrampermeter^3(kg/m^3)	1.711 806	E-02
gram	kilogram (kg)	1.000 000*	E-03
g/cm^3	kilogram per $meter^3$ (kg/m^3)	1.000 000*	E+03
$gram\text{-}force/cm^2$	pascal (Pa)	9.806 650*	E+01
hectare	$meter^2$ (m^2)	1.000 000*	E+04
horsepower (550 ft•lbf/s)	watt (W)	7.456 999	E+02

[2] This is sometimes called the moment of inertia of a plane section about a specified axis.

To convert from	to	Multiply by	
horsepower (boiler)	watt (W)	9.809 50	E+03
horsepower (electric)	watt (W)	7.460 000*	E+02
horsepower (metric)	watt (W)	7.354 99	E+02
horsepower (water)	watt (W)	7.460 43	E+02
horsepower (U.K.)	watt (W)	7.457 0	E+02
hour (mean solar)	second (s)	3.600 000	E+03
hour (sidereal)	second (s)	3.590 170	E+03
hundredweight (long)	kilogram (kg)	5.080 235	E+01
hundredweight (short)	kilogram (kg)	4.535 924	E+01
inch (in.)	meter (m)	2.540 000*	E-02
inch of mercury (32°F)	pascal (Pa)	3.386 38	E+03
inch of mercury (60°F)	pascal (Pa)	3.376 85	E+03
inch of water (39.2°F)	pascal (Pa)	2.490 82	E+02
inch of water (60°F)	pascal (Pa)	2.488 4	E+02
in.^{2}	meter^2 (m^2)	6.451 600*	E-04
in.^{3} (volume; section modulus)	meter^3 (m^3)	1.638 706	E-05
in.^{3}/min	meter^3 per second (m^3/s)	2.731 177	E-07
in.^{4} (moment of section)^2	meter^4 (m^4)	4.162 314	E-07
in.•min^2/ft^6 (air resistance factor)	newton•second^2 p/m^8)	1.17	E-09
in./s	meter per second (m/s)	2.540 000*	E-02
in./s^2	meter per second^2 (m/s^2)	2.540 000*	E-02
kayser	1 per meter (1/m)	1.000 000*	E+02
kelvin	degree Celsius	$t_{°C} = t_K - 273.15$	
kilocalorie (International Table)	joule (J)	4.186 800*	E+03
kilocalorie (thermochemical)/min	watt (W)	6.973 333	E+01
kilogram-force (kgf)	newton (N)	9.806 650*	E+00
kgf•m	newton meter (N•m)	9.806 650*	E+00
kgf•s^2/m (mass)	kilogram (kg)	9.806 650*	E+00
kgf/cm^2	pascal (Pa)	9.806 650*	E+04
kgf/m^2	pascal (Pa)	9.806 650*	E+00
kgf/mm^2	pascal (Pa)	9.806 650*	E+06
km/h	meter per second (m/s)	2.777 778	E- 01
kilopond	newton (N)	9.806 650*	E+00
kWh	ioule (J)	3.600 000*	E+06
kip (1000 lbf)	newton (N)	4.448 222	E+03
kip/in.^{2} (ksi)	pascal (Pa)	6.894 757	E+06
knot (international)	meter per second (m/s)	5.144 444	E-01
lambert	candela per meter^2 (cd/m^2)	$1/\pi$ *	E+04
lambert	candela per meter^2 (cd/m^2)	3.183 099	E+03
langley	joule per meter^2 (J/m^2)	4.184 000*	E+04
league	meter (m)	[see footnote 12]	
light year	meter (m)	9.460 55	E+15
liter	meter^3 (m^3)	1.000 000*	E-03
maxwell	weber (Wb)	1.000 000*	E-08
mho	siemens (S)	1.000 000*	E+00
microinch	meter (m)	2.540 000*	E-08
micrometer (μm)	meter (m)	1.000 000*	E-06
mil	meter (m)	2.540 000*	E-05

Note: Use period (.) with abbreviation for "inch."

To convert from	to	Multiply by	
mile (international)	meter (m)	1.609 344*	E+03
mile (statute)	meter (m)	1.609 3	E+03
mile (US survey)[1]	meter (m)	1.609 347	E+03
mile (international nautical)	meter (m)	1.852 000*	E+03
mile (US nautical)	meter (m)	1.852 000*	E+03
mi^2 (international)	$meter^2$ (m^2)	2.589 988	E+06
mi^2 (US survey)[1]	$meter^2$ (m^2)	2.589 998	E+06
mi/h (international)	meter per second (m/s)	4.470 400*	E-01
mi/h (international)	kilometer per hour (km/h)	1.609 344*	E+00
mi/min (international)	meter per second (m/s)	2.682 240*	E+01
mi/s (international)	meter per second (m/s)	1.609 344*	E+03
millibar	pascal (Pa)	1.000 000*	E+02
millimeter of mercury (0°C)	pascal (Pa)	1.333 22	E+02
minute (angle)	radian (rad)	2.908 882	E-04
minute (mean solar)	second (s)	6.000 000	E+01
minute (sidereal)	second (s)	5.983 617	E+01
month (mean calendar)	second (s)	2.628 000	E+06
oersted	ampere per meter (A/m)	7.957 747	E+01
ohm centimeter	ohm meter (Ω•m)	1.000 000*	E-02
ohm circular-mil per ft	ohm $millimeter^2$ per meter (Ω•mm^2/m)	1.662 426	E-03
ounce (avoirdupois)	kilogram (kg)	2.834 952	E-02
ounce (troy or apothecary)	kilogram (kg)	3.110 348	E-02
ounce (UK fluid)	$meter^3$ (m^3)	2.841 307	E-05
ounce (US fluid)	$meter^3$ (m^3)	2.957 353	E-05
ounce-force	newton (N)	2.780 139	E-01
oz•in.	newtonmeter (N•m)	7.061 552	E-03
oz (avoirdupois)/gal (UK liquid)	kilogram per $meter^3$ (kg/m^3)	6.236 021	E+00
oz (avoirdupois)/gal (US liquid)	kilogram per $meter^3$ (kg/m^3)	7.489 152	E+00
oz (avoirdupois)/$in.^3$	kilogram per $meter^3$ (kg/m^3)	1.729 994	E+03
oz (avoirdupois)/ft^2	kilogram per $meter^2$ (kg/m^2)	3.051 517	E-01
oz (avoirdupois)/yd^2	kilogram per $meter^2$ (kg/m^2)	3.390 575	E-02
oz/ton (ore grade)	grams per tonne (g/t)	3.125	E+01
parsec	meter (m)	3.085 678	E+16
peck (US)	$meter^3$ (m^3)	8.809 768	E 03
pennyweight	kilogram (kg)	1.555 174	E-03
perm (0°C)	kilogram per pascal second $meter^2$ (kg/Pa•s•m^2)	5.721 35	E-11
perm (23°C)	kilogram per pascal second $meter^2$ (kg/Pa•s•m^2	5.745 25	E-11
perm in. (0°C)	kilogram per pascal second meter (kg/Pa•s•m)	1.453 22	E-12
perm in. (23°C)	kilogram per pascal second meter (kg/Pa•s•m)	1.459 29	E-12
phot	lumen per $meter^2$ (lm/m^2)	1.000 000*	E+04
pica (printer's)	meter (m)	4.217 518	E-03
pint (US dry)	$meter^3$ (m^3)	5.506 105	E-01
pint (US liquid)	$meter^3$ (m^3)	4.731 765	E 01
point (printer's)	meter (m)	3.514 598*	E 04
poise (absolute viscosity)	pascal second (Pa•s)	1.000 000*	E-01

To convert from	to	Multiply by	
pound (lb avoirdupois)	kilogram (kg)	4.535 924	E-01
pound (troy or apothecary)	kilogram (kg)	3.732 417	E-01
lb•ft^2 (moment of inertia)	kilogrammeter^2 (kg•m^2)	4.214 011	E-02
lb•in.^{2} (moment of inertia)	kilogram meter^2 (kg•m^2)	2.926 397	E-04
lb•min^2/ft^4 (air friction factor, K)	kilogram per meter^3 (kg/m^3)	1.855	E+06
lb/ft•h	pascal second (Pa•s)	4.133 789	E-04
lb/ft•s	pascal second (Pa•s)	1.488 164	E+00
lb/ft (loading factor)	kilogram per meter (kg/m)	1.488 156	E+00
lb/ft^2	kilogram per meter^2 (kg/m^2)	0.882 428	E+00
lb/ft^3 (specific weights)	kilogram per meter^3 (kg/m^3)	1.601 846	E+01
lb/gal (UK liquid)	kilogram per meter^3 (kg/m^3)	9.977 633	E+01
lb/gal (US liquid)	kilogram per meter^3 (kg/m^3)	1.198 264	E+02
lb/h	kilogram per second (kg/s)	1.259 979	E-04
lb/hp•h (SFC, specific fuel consumption	kilogram per joule (kg/J)	1.689 659	E-07
lb/in.^{3}	kilogram per meter^3 (kg/m^3)	2.767 990	E+04
lb/min	kilogram per second (kg/s)	7.559 873	E-03
lb/s	kilogram per second (kg/s)	4.535 924	E-01
lb/ton (powder factor)	kilogram per tonne (kg/t)	5.000	E-01
lb/yd^3 (powder factor)	kilogram per meter^3 (kg/m^3)	5.932 764	E-01
poundal	newton (N)	1.382 550	E-01
poundal/ft^2	pascal (Pa)	1 488 164	E+00
poundal•s/ft^2	pascal second (Pa•s)	1.488 164	E+00
pound-force (lbf)	newton (N)	4.448 222	E+00
lbf•ft	newton meter (N•m)	1.355 818	E+00
lbf•ft/in.	newton meter per meter (N•m/m)	5.337 866	E+01
lbf•in.	newton meter (N•m)	1.129 848	E-01
lbf•in./in.	newton meter per meter (N•m/m)	4.448 222	E+00
lbf•s/ft^2	pascal second (Pa•s)	4.788 026	E+01
lbf/ft	newton per meter (N/m)	1.459 390	E+01
lbf/ft^2	pascal (Pa)	1.788 026	E+01
lbf/in.	newton per meter (N/m)	1.751 268	E+02
lbf/in.^{2} (psi)	pascal (Pa)	6.894 757	E+03
lbf/lb (thrust/weight [mass] ratio)	newton per kilogram (N/kg)	9.806 650	E+00
quart (US dry)	meter^3 (m^3)	1.101 221	E-03
quart (US liquid)	meter^3 (m^3)	9.463 529	E-04
rad (radiation dose absorbed)	gray (Gy)	1.000 000*	E-02
rhe	1 per pascal second (1/Pa•s)	1.000 000*	E+01
rod	meter (m)	[see footnote [l]]	
roentgen	coulomb per kilogram (C/kg)	2.58	E-04
second (angle)	radian (rad)	4.818 137	E-06
second (sidereal)	second (s)	9.972 696	E-01
section	meter^2 (m^2)	[see footnote[l]]	
shake	second (s)	1.000 000*	E-08
slug	kilogram (kg)	1.459 390	E+01
slug/ft•s	pascal second (Pa•s)	4.788 026	E+01
slug/ft^3	kilogram per meter^3 (kg/m^3)	5.153 788	E+02
statampere	ampere (A)	3.335 640	E-10
statcoulomb	coulomb (C)	3.335 640	E-10
statfarad	farad (F)	1.112 650	E-12
stathenry	henry (H)	8.987 554	E+ll
statmho	siemens (S)	1.112 650	E-12

To convert from	to	Multiply by	
statohm	ohm(Ω)	8.987 554	E+11
statvolt	volt (V)	2.997 925	E+02
stere	meter^3 (m^3)	1.000 000*	E+00
stilb	candela per meter^2 (cd/m^2)	1.000 000*	E+04
stokes (kinematic viscosity)	meter^2 per second (m^2/s)	1.000 000*	E-04
tablespoon	meter^3 (m^3)	1.178 676	E-05
teaspoon	meter^3 (m^3)	4.928 922	E-06
tex	kilogram per meter (kg/m)	1.000 000*	E-06
therm	joule (J)	1.055 056	E+08
ton (assay)	kilogram (kg)	2.916 667	E-02
ton (long, 2240 lb)	kilogram (kg)	1.016 017	E+03
ton (metric)	kilogram (kg)	1.000 000*	E+03
ton (nuclear equivalent of TNT)	joule (J)	4.184	E+09
ton (refrigeration)	watt (W)	3.516 800	E+03
ton (register)	meter^3 (m^3)	2.831 685	E+00
ton (short, 2000 lb)	kilogram (kg)	9.071 817	E+02
ton/yd^3 (specific weight)	tonnes per meter (t/m^3)	1.186 66	E+00
ton (long)/yd^3 (specific weight)	kilogram per meter (kg/m^3)	1.328 939	E+03
ton (short)/h	kilogram per second (kg/s)	2.519 958	E-01
ton-force (2000 lbf)	newton (N)	8.896 111	E+03
tonne (t)	kilogram (kg)	1.000 000*	E+03
ton/yd^3 (specific weight)	tonnes per meter^3 (t/m^3)	1.186 55	E+00
torr (mm Hg, 0°C)	pascal (Pa)	1.333 22	E+02
township	meter^2 (m^2)	[see footnote[l]]	
unit pole	weber (Wb)	1.256 637	E-07
W•h	joule (J)	3.600 000*	E+03
W•s	joule (J)	1.000 000*	E+00
W/cm^2	watt per meter^2 (W/m^2)	1.000 000*	E+04
W/in.^{2}	watt per meter^2 (W/m^2)	1.550 003	E+03
yard	meter (m)	9.144 000*	E-01
yd^2	meter^2 (m^2)	8.361 274	E-01
yd^3	meter^3 (m^3)	7.645 549	E-01
yd^3/min	meter^3 per second (m^3/s)	1.274 258	E-02
yd^3/ton (stripping ratio)	meter^3 per tonne (m^3/t)	8.427 8	E-01
year (calendar)	second (s)	3.153 600	E+07
year (sidereal)	second (s)	3.155 815	

Source: Modified from ASTM/IEEE Standard Metric Practice.

Table B. Strength Properties of Rocks

Table B1. Uniaxial Compressive Strength for Rocks

Rock Type	Location	Uniaxial Compressive Strength (MPa)	Standard Deviation (MPa)	Number of Tests	Reference No.
Basalt		81.2	42.8	116	1
Basalt	Idaho and Washington	118.1	78.5	156	2
Coal	West Virginia	17.6	7.7	10	3
Coal	United Kingdom	20.6	N/A	283	4
Coal	United Kingdom	66.4	N/A	421	4
Coal	South Africa	40.1	N/A	638	5
Coal	South Africa	43.1	N/A	240	6
Coal	South Africa	6.2	2.3	37	7
Dolerite	United Kingdom	166.2	50.3	24	8
Dolerite Quartz	United Kingdom	333 . 3	89.2	22	9
Dolomite		100.8	29.5	16	10
Gneiss Granite		154.7	23.5	22	11
Gneiss Biotite		160.1	32.0	34	11
Granite		122.2	N/A	63	12
Granite	Wyoming	128.6	24.6	91	13
Granite (Barre)	Vermont	225.6	33.4	150	14
Granite		231.4	42.4	16	15
Granite		94.6	35.9	12	16
Granite	United Kingdom	206.9	103.6	11	9
Granite	Wyoming	142.4	21.0	24	17
Granite Aeirine		161.2	17.9	16	11
Granite Biotite		159.9	41.1	171	11
Granite Biotite muscovite		144.9	27.3	26	11
Granite Granodiorite	California	36.6	43.6	10	18
Granite Porphry		152.3	23.1	10	19
Granodiorite	United Kingdom	165.7	51.7	21	8
Granodiorite	California	116.0	45.5	43	18
Hematite		168.7	36.6	15	10

Iron Ore	France	34.4	12.1	18	20
Limestone	Marshall Islands	47.8	41.1	12	22
Limestone		90.9	41.2	16	23
Limestone		44.2	46.0	41	24
Limestone		93.0	28.9	47	12
Limestone		110.0	47.7	17	25
Limestone (Bedford)		66.2	29.7	150	14
Limestone (Conconino)		47.8	4.1	12	26
Limestone (Indiana)		75.3	36.0	17	15
Limestone (Indiana)		53.1	3.7	14	26
Limestone (Siyeh)		37.7	8.7	11	10
Limestone	United Kingdom	103.8	26.2	15	8
Magnetite		136.6	23.7	15	10
Marble (Chickies Quartzite)		98.6	22.6	14	27
Marble, Calcite	Ontario, Canada	136.8	75.6	20	28
Marble, Dolomite		123.3	58.1	35	28
Monzonite Quartz		164.2	2.8	31	11
Monzonite Quartz	Nevada	77.1	41.0	29	29
Norite, Westerly Granite	Rhode Island	210.2	10.5	15	30
Porphyry Dacite		101.4	31.8	55	31
Porphyry Rhyolite		226.0	62.3	14	10
Potash		32.8	12.9	56	21
Quartzite		143.7	50.4	20	10
Quartzite, Mica		112.4	37.3	55	25
Rhyodacite		142.9	45.4	14	31
Rhyolite Porphyry		200.5	100.3	11	32
Rock Salt		23.7	2.8	36	1
Sandstone (Carlile Shale)	Arizona	24.2	11.0	104	33
Sandstone		59.8	32.0	69	12
Sandstone (Gatun)	Panama	4.6	N/A	74	34
Sandstone (Berea)	Ohio	91.2	N/A	150	14

Sandstone, Limestone, Feldspar	Soviet Union	238.2	N/A	326	35
Schist, Biotite Quartz		32.6	10.5	17	36
Schist, Mica		38.3	16.1	57	25
Shale		12.9	17.0	23	1
Shale	West Virginia	22.8	15.5	14	3
Shale		116.4	29.8	59	10
Tonalite	Utah	85.1	36.7	33	37
Tuff		16.6	6.9	24	1
Tuff	Nevada	60.0	52.6	58	38, 39
Tuff	Nevada	65.3	50.1	19	40
Tuff, Pumice	Nevada	30.6	13.5	27	41
Tuff, Pumice	Nevada	22.7	9.5	28	42
Tuff (Indian Trail, Zeolite)	Nevada	52.4	19.8	44	43, 50

Table B2a. Tensile Strength for Rocks (Direct Pull Method)

Rock Type	Location	Tensile Strength (MPa)	Standard Deviation (MPa)	Number of Tests	Reference
Basalt, Calcite		8.6	1.4	12	44
Conglomerate		29.7	N/A	10	45
Iron Ore	France	3.2	0.8	18	20
Limestone		4.2	N/A	18	46
Limestone (Lenoir)		5.8	3.2	11	44
Sandstone (Carlile Shale)	Arizona	1.1	0.5	102	33
Sandstone		1.7	1.3	10	12
Sandstone, Calcereous		4.3	2.5	10	47
Schist, Biotite Quartz		3.1	2.1	15	36
Schist, Mica		3.1	2.1	15	25

Table B2b. Tensile Strengths for Rocks (Splitting Disk Method)

Rock Type	Location	Tensile Strength (MPa)	Standard Deviation (MPa)	Number of Tests	Reference
Dolomite		8.7	3.3	13	10
Granite (Barre)	Vermont	13.8	2.1	150	14
Granite (Barre)	Vermont	14.2	2.5	17	48
Limestone (Bedford)		7.5	3.6	34	14
Limestone (Indiana)		9.1	3.8	17	48
Iagnetite Silica		12.5	1.7	12	10
Porphyry Rhyolite		14.4	1.8	9	10
Sandstone		7.7	1.8	21	10
Sandstone (Berea)		7.1	5.2	33	14
Sandstolle (I'erea)		10.2	5.7	17	48
Sandstone (Darley Dale)	United Kingdom	6.0	1.0	9	49
Sandstone (Darley Dale)	United Kingdom	14.8	3.9	11	49
Shale		10.1	1.9	24	10

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Table B3a. Room Temperature Creep Coefficients for Rocks

Rock Type	Time (t) (s)	Differential Stress (Pa)	Strain	Constant
Andesite	1.2×10^6	10^4	10^{-5}	7.1×10^{-7}
Basalt	5.2×10^5	10^4	10^{-4}	8.0×10^{-6}
Gabbro	2.6×10^5	10^4	10^{-4}	8.3×10^{-7}
Granite	1.7×10^5	10^5	10^{-5}	8.5×10^{-6}
Granite	1.0×10^6	10^5	10^{-4}	7.2×10^{-6}
Granite	8.6×10^2	10^5	10^{-3}	1.7×10^{-4}
Granodiorite	2.6×10^5	10^4	10^{-4}	8.2×10^{-6}
Granodiorite	5.2×10^5	10^4	10^{-5}	7.8×10^{-7}
Limestone	8.6×10^4	10^5	7×10^{-3}	6.4×10^{-3}
Quartzite	1.7×10^6	10^4	10^{-3}	7.0×10^{-5}
Rhyolite	8.6×10^7	10^4	10^{-4}	5.5×10^{-6}
Rock Salt	3.1×10^6	10^4	10^{-2}	2.1×10^{-4}
Rock Salt	8.6×10^4	10^4	2×10^{-3}	3.0×10^{-3}
Sandstone	8.6×10^2	10^2	10^{-2}	1.7×10^{-3}
Shale	8.6×10^4	10^4	10^{-3}	9.1×10^{-5}
Shale	1.2×10^8	10^3	10^{-2}	5.4×10^{-4}

Source: Robertson, 1960.

Table B3b. Room Temperature Effect of Stress on Creep of Rocks

Rock Type	Maximum Strain (10^{-3})	Maximum Stress (MPa)	Exponent (n)	Hydrostatic Pressure (MPa)	Reference
Alabaster (in water)	1	360	1.8	0	Evans et al., 1937
Gabbro	0.01	10	1.0	0	Lomnitz, 1956
Granite	1	350	3.3	0	Evans et al.,1937
Granite	3	100	3.0	0	Matsushima, 1960
Granodiorite	0.2	10	1.0	0	Lomnitz, 1956
Halite	20	30	1.9	0	Kendall, 1958
Limestone	200	670	5.0	1000	Griggs, 1936
Limestone	7	140	1.7	0	Kendall, 1958
Limestone	5	350	3.0	70	Kendall, 1958
Limestone	4	410	7.9	100	Kendall, 1958
Limestone	300	580	3.7	200	Robertson, 1960
Limestone	180	630	1.8	300	Robertson, 1960
Limestone	100	560	1.8	400	Robertson, 1960
Shale	3	10	2.7	0	Phillips, 1932
Slate	1	360	1.8	0	Evans et al., 1937

Applicable Equation, $\dot{\varepsilon} = A\sigma^n$
Source: Robertson, 1960.

Table B3c. Temperature Relationships for Creep

Relationship	Restrictions	Reference
(1) $\dot{\varepsilon} = A \exp(-q/RT)$		Arrhenius, 1889
(2) $\dot{\varepsilon} = \dot{\varepsilon}_o \exp(-q/RT) \sinh \frac{\sigma}{\sigma_o}$		Kauzmann, 1941; Ree et al., 1960
(3) $\dot{\varepsilon} = \dot{\varepsilon}_o \exp(-q/RT)\, \sigma^n$	$n = 2$ to 8 $10^9 > \dot{\varepsilon}/D > 10^2$	Weertman, 1962
(4) $\dot{\varepsilon} = A \exp(-q/RT) \exp(B\sigma)$		Sherby et al., 1967
(5) $\dot{\varepsilon} = A(G/T) \exp(-q/RT)\ (\sigma/G)$		Weertman, 1957,1968; Barrett et al., 1965
(6) $\dot{\varepsilon} = A\sigma D_v / L^2 T$	$\varepsilon/D < 10^2$	Nabarro, 1948; Herring, 1950
(7) $\dot{\varepsilon} = A\sigma D_g / L^3 T$		Coble, 1963
(8) $\dot{\varepsilon} = A \exp(q - b\sigma/RT)$		Kauzmann, 1941

NOMENCLATURE

A,B,D,n	constants
D_g	diffusivity, grain boundary
D_v	diffusivity, volume
G	shear modulus
L	length
q	activation energy of a solid
R	universal gas constant
T	absolute temperature
$\dot{\varepsilon}$	creep or strain rate
σ	normal stress

Conversion factor: 1000 psi = 6.8948 MPa

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Table C. Engineering Properties of Rocks

Engineering property	Symbol	Nature: field(F) Laboratory (L) Derived (D)	Method of measurement	Units of measure	Use in engineered construction	Limitations	Ref.
Unit weight	γ	L	Volumetric displacement in water; weight per unit volume; usually weighed as oven dried but may be specified on several other bases.	kg/m^3	Weight, per unit volume, of entire rock; primary term in many computations; useful in computing in situ stress.	Expected statistical variances in the more porous rocks.	1 35 26
Moisture content	w	L	Oven drying.	%	Effect of moisture variations on other physical properties considered in data evaluation and assessment.	Varies seasonally and under influences of construction activities; of little use to refer to tabulations by other except as a condition of specific property test.	35 26
Porosity	n	L	20.106 mm^3 sample placed in porosimeter device.	%	Indication of ability to retain fluids or gases.	Often highly depth dependent.	45 26 21 35
Coefficient of permeability	k	L/F	Cylinder of rock placed in pereameter (L) or pressured water flow into borehold (F).	cm/s	Yield of water for dewatering purposes; determination of pore water distribution for support design of slope stability analyses, planning for grouting.	Field tests require careful isolation (packing) of borehold segment or interest; avoid high-pressure hydrotracturing; greatly influenced by discontinuities.	21
Compressive strength	c_o	L	Uniaxial or Triaxial conditions, in universal test machine; strain gages used for moduli determinations.	Pa	Index classification test; load bearing capacity; other properties according to Mohr failure concept; slope stability; mine pillar stress; subsidence; excavation, blasting, drilling and mole boring performance.	Avoid unrepresentative anisotropic fabric elements or discontinuities; select representative sample; consider data scatter; L/D ratio is quite important (standard is 2.1); peak strength is obtained.	T-15 13 30 T-24 T-25 34 28
Induced tensile strength	T_o	L	Point-bearing compression of compression of tabular cylindrical specimen (Brazilian test); more classical tensile pull test is not generally utilized.	Pa	Load-bearing capacity under tension.	Introduces a contributory compressive stress, therefore not a true determination of tensile strength; magnifies effect of microdefects; theoretically of greater T_o than direct-test value.	T-15 56 20 13 T-23 34 26

Engineering property	Symbol	Nature: field(F) Laboratory (L) Derived (D)	Method of measurement	Units of measure	Use in engineered construction	Limitations	Ref.
Unconfined shear strength	S_o	L/F	Direct shear box, utilizing natural discontinuities or planes or weakness; also triaxial compression test which initiates shear failure on basis of sample geometry or inherent weaknesses.	Pa; ø(deg); c(Pa)	Index classification test; load-bearing capacity; slope stability; mine pillar stress; subsidence.	Slope stability; residual friction angle ($ø_r$); strong effect by anisotropy.	T-15 32 25 38 16 28
Abrasivity	g	L	400g crushed rock rotated in paddle-and-drum test device.	Dimensionless	Resistance to abrasion as indicator of percussion drillability index.	Must be carefully related to petrographic nature of rock lithology and fabric in order to provide for meaningful assessment of effect.	9 52
Hardness			Small laboratory test holding devices for impact. Height or rebound of small diamond-tipped device.	Dimensionless	Indicator or relative hardness; useful in tunnel boring rate estimates.		
Shore scleroscope	H_r	L			As above	Multiple readings are required to overcome effect of sharp point impact on very small area.	59 58 63
Schmidt rebound	H_s	L		Dimensionless	As above	Softer rock breaks on impact; must use Type L device of minimal energy.	41
Taber	H_a	L		Dimensionless	As above		T-23
Total	H_t	L		Dimensionless			T-26 T-27
Sonic velocity	v	L	Core sample, held in accoustical bench; subjected to pulsation of electrically-generated mechanical vibration.	km/s	Computational input for calculation of E, G, and K.	Difficult to accommodate fabric anisotropies; difficult to know volume tested in situ; does not introduce time-dependent strain variations in derived properties	19 44 3
Seismic velocity Compressional Shear	 V_p V_s	F	Mechanical or explosive energy wave arrival sensed by geophone, timer, and recorder; measured on ground surface or in borehole configurations.	km/s	As above, indicator of overall maturity of rock due to averaging effect on wave travel paths, depending on geophone/energy source array.	Degree of saturation important; test does not introduce nonlinear, time-dependent strain-variation in derived properties; most valid in homogeneous and isotropic rock.	49 3 28

Poisson's ratio	u	D	Calculated from sonic/seismic velocity tests or by use of electrical resistance strain gages on compression tests.	Dimensionless; in range 0 to 0.5.	Computational input for calculation of stress distribution patterns and of predicted strain in elastic media; required for finite element modeling.	Difficult to extroplate laboratory measurement to field conditions; often estimated without testing; best approximation is from triaxial compression test at confinement equivalent to in situ conditions.	47 3 28
Modulus of rigidity	G	D	As above	Pa	Indicator of seismic design stiffness.	Strain-related.	49
Bulk modulus	K	D	As above	Pa		Strain-related.	49
Tangent modulus of elasticity (Young's modulus, or modulus of deformation)	E_r	L	Triaxial compression in universal test machine; electrical resistance strain gages.	Pa	The fundamental stress-strain relationship; input for static displacement computations and for dynamic, seismic analyses.	Requires accommodation of any anisotropy of rock fabric and model in situ conditions.	T-6 4 28
Secant modulus of elesticity	E_s	L	As above	Pa	Alternate expression of the fundamental stress-strain relationship.	As above.	3
Swelling slake durability	I_s I_D	L L	Rotational cyclic immersion in water; measure percent loss by weight; measure swelling displacement (d) and initial weight of specimen L (I_s = d/L x 100); measure slaking weight percentages lost in wetting cycles.	Dimensionless (0—100)	Indication of relative resistance to weathering on exposure to elements.	Semi-quantitative, in use, swell potential is proportional to confining pressure; slake potential relates to moisture exposure and water chemistry at site.	36 26

Conversion factors:

1 lb = 0.4536 kg
1 lb/ft^3 = 16.0185 kg/m^3
1 in./sec = 2.54 cm/s
1 psf = 47.88 Pa
1 psi= 6894.8 Pa

Table C1. Engineering Properties of Igneous Rocks

Rock type: geologic unit	Location	γ (kg/m^3)	C_o (Pa)	Ref.	H^a	V_p (km/s)	Ref.	u	G (Pa)	Ref.	E_t (Pa)	Ref.	E_s (Pa)	Ref.	Ref.
Amphibolite	McLeese Lake, B.C.	2920	2.65	7				0.24			1.75	10			7
Amphibolite, fine	Oorgaum, Mysore State, India	3070	4.23	8	92	5.79	3		4.58	10	1.04	11			20
Andesite, hypersthene	Palisades Dam, ID	2570	1.29-1.32	9				0.18					5.45	10	1
Anorthosite, Labradorite, C.	Ukrainian Shield, USSR	2770	2.27	8				0.36	3.41	10	9.28	10			2
Basalt, Lower Granite	Pullman, WA	2727	2.27-3.55	8	57:412:116	5.27	3				5.02	10			6
Basalt, Olivine, dense	Nevada Test Site, NV	2720									2.47	10			14
Basalt, Olivine, sl. vesicular		2660									2.86	10			14
Basalt, Olivine, Western Cascade	Medford, OR	2730	1.69-2.20	8				0.25					4.21	10	1
Basalt	Painesdale, MI	2850	2.30	8	69	4.63	3		2.68	10	6.15	10			20
	Ahmeek, MI	2940	2.58-3.59	8	79	5.15	3		3.17	10	7.79	10			20
Basalt, subaqueous	Eniwetok, PTT	2860	1.94	8	71			0.18			6.93	10			4
Basalt, vesicular	Bergstrom, TX	2550	7.44	7		4.65	3	0.13			3.74	10			19
		2580	8.34	7		5.04	3	0.19			4.05	10			19
Basalt, dense		2593	1.13	8				0.20			5.21	10			19
		2761	1.32	8		5.56	3	0.17			7.65	10			19
		2752	1.25	8		4.70	3				5.79	10			19
Charnokite (hypersthene granite)	Ukrainian Sheild, USSR	2730	2.47	8				0.22	2.75	10	6.73	10			2
Diabase, Medford	Cambridge, MA	2882	1.77	8	44:2.13:60										8
Diabase, Palisades	W. Nyack, NY	2932	2.41	8	59						8.19	10			8
Diabase; French Creek	St. Peters, PA	3060	3.01	8	58						9.94	10			8
Diabase, altered	Clinton Co., NY	2940	3.21	8	92	5.70	3		3.73	10	9.58	10			20
Diorite; Kennsington	Washington, DC	2820	8.09(7)-2.76[8]		50:8.7:150										8
Diorite, gneissic	Mineville, NY	3030	1.86	8	90	4.27	3		2.78	10	5.53	10			20
Diorite, augite, fresh	Keetley, UT	2740	3.33	8	82	5.55	3	0.25	3.37	10	8.41	10			21
Diorite, augite, sl. altered		2720	2.79	8	83	5.43	3	0.26	3.18	10	8.00	10			21
Diorite, augite, altered		2720	2.15	8	71	4.94	3	0.30	2.56	10	6.64	10			21
Diorite, biotite, porph., sl. altered		2690	2.28	8	77	4.97	3	0.27	2.83	10	6.68	10			21
Diorite, biotite, porph., s. altered		2660	1.80	8	67	4.75	3	0.22	2.45	10	6.01	10			21
Diorite, hornblende	Ishpeming, MI	3010	2.74	8	84	6.00	3	0.29	4.22	10	1.07	11			4
Gabbro; Salem	Beverly, MA	3060	1.33-1.49	8	52:6.47:129						8.76	10			8
Gabbro, altered	Clinton Co., NY	2930	2.77	8	82	5.36	3		3.36	10	8.48	10			20
Gabbro/diabase	Ukrainian Shield, USSR	3000	3.09	8				0.33	4.41	10	1.19	11			2
Gabbro/diabase	Karelian SSR, USSR	3190	3.14	8							1.17	11			2
Granite, f.	Grand Coulee, WA	2571	1.94	8	53:10.5:172	4.64	3				5.48	10			5
Granite, c.		2627	1.61	8	52:9.5:161	4.08	3				5.24	10			5
Granite, Pikes Peak	Colorado Springs, CO	2675	1.57	8	58						7.06	10			8
Granite, Barre	Barre, VT	2643	1.94	8	53						6.15	10			15
Granite, Pre-Cambrian	Loveland, CO	2630	7.21	7				0.14					2.69	10	1
Granite	Woodstock, MD	2650	2.51	8	98	4.51	3		2.54	10	5.46	10			20
	Tem Piute Dist., NV	2630	2.72	8	100	4.42	3		2.25	10	5.13	10			20
	Mt. Airy, NC	2600	2.10	8	90	2.44	3		1.02	10	1.57	10			20
Granite, biotite, m.	Karelian SSE, USSR	2700	2.39	8				0.25	2.41	10	6.93	10			2
Granite, gneissic; Lithonia	Lithonia, GA	2640	1.93	8	85	2.71	3	-0.19	1.18	10	1.91	10			3
		2640	2.13	8	85	2.50	3	-.023	1.09	10	1.64	10			3
		2660	2.09	8	89	2.62	3	0.02	8.96	9	1.86	10			3
		2620	2.05	8	85	1.08	3	-0.28	7.10	9	1.04	10			3

Granite, f.-m; unaweep	Grand Junction, CO	2670	1.74	8	59	3.17	3	-0.19	1.68	10	2.72	10			4
		2710	1.59	8	53	3.75	3	0.00	1.91	10	3.82	10			4
Granite, par. to foliation; unaweep		2730	1.61	8	44	3.93	3	0.12	1.90	10	4.23	10			4
Granite; unaweep		2660	1.74	8	37	3.17	3	-0.13	1.55	10	2.72	10			4
Granite, pink	Bergstrom, TX	2710				6.47	3	0.29	6.84	10	8.57	10			19
Granite, weathered		2650				5.83	10	0.29	4.72	10	5.75	10			19
		2620				5.33	10	0.30	4.65	10	5.36	10			19
Granodiorite	Bergstrom, TX	2689	4.07	7		5.72	3	0.70			5.84	10			19
		2703	1.39	8		5.89	3	0.22			7.99	10			19
		2699	8.51	7		5.80	3	0.17			6.87	10			19
		2702	1.29	8		5.72	3	0.19			7.3	10			19
		2700	1.15	8		5.93	3	0.19			7.10	10			19
Magnetite, ore	Mineville, NY	4230	1.41	8	72	2.72	3		1.86	10	3.14	10			20
Monzonite, porphyritic; Colville	Grand Coulee, WA	2575	1.49	8				0.18					4.14	10	1
		2575	1.71	8				0.15					4.21	10	1
Pegmatite	Star Lake, NY	2590	2.14	8	87	4.88	3		2.28	10	6.16	10			20
Pyroxenite	Clinton Co., NY	3450	1.70	8	70	1.98	3		1.03	10	1.31	10			20
Pyroxenite, fresh	Star Lake, NY	3430	1.82	8	60	6.03	3		5.03	10	1.24	11			20
Pyroxenite, heavily altered		2530	5.86	7	28	2.96	3		7.58	9	2.20	10			20
Quartz, diorite	Mountain Home, ID		8.74	7				0.05					2.14	10	1
Quartz, monzonite	Bergstrom, TX	2669	1.48	8				0.70			6.74	10			19
		2680	1.55	8				0.22			7.24	10			19
		2670	1.30	8				0.17			6.68	10			19
		2673	1.29	8				0.19			7.65	10			19
		2667	1.39	8				0.19			7.72	10			19
Rapakivi (granite)	Ukrainian Shield, USSR	2640	2.72	8				0.20	2.43	10	5.81	10			2
Shonkinite (dark syenite)	Clinton Co., NY	3350	1.85	8	78	3.23	3		1.94	10	3.54	10			20
Syenite	Kirkland Lake, ON	2820	3.03	8		5.12	3		2.83	10	7.38	10			20
Syenite, porphyritic		2700	4.34	8		5.12	3		3.03	10	7.10	10			20

Table C2. Engineering Properties of Metamorphic Rocks

Rock type; geologic unit	Location	γ (kg/m^3)	C_o (Pa)	Ref.	H[a]	V_p (km/s)	Ref.	u	G (Pa)	Ref.	E_t (Pa)	Ref.	E_s (Pa)	Ref.	Ref.
Argillite, Cambridge	Dorchester, MA	2810	1.36	8							8.41	10			8
	Cambridge, MA	2642	6.61	7											8
		2510	3.15	7	15:0.45:10										8
		2759	1.55	8	26:1.2						4.83	10			8
		2715	1.55	8							3.86	10			8
Gneiss, quartz diorite	Bethesda, MD	2775	9.60	7	64:5.17:139						7.24	10			8
Gneiss, schistose; Wissahickon	Washington, DC	2980	7.01	7	46:2.90:79										8
Gneiss, Dworssak	Orofino, ID	2804	1.62	8	48:						5.36	10			8
Gneiss, diorite: Idaho Springs	Montezuma Quad., CO	2865	8.41	7				0.06			6.41	10			18
Gneiss, granite	Mineville, NY	2750	2.12	8	99	3.63	3		1.96	10	3.85	10			20
Gneiss, granite, pegmatitic	Star Lake, NY	3040	1.53	8	75	4.66	3		2.88	10	6.67	10			20
Gneiss, pegmatitic		2650	1.96	8	81	4.11	3		2.12	10	4.46	10			20
Gneiss, augite	Hackettstown, NJ	3360	2.19	8	74	5.55	3	0.27	4.07	10	1.03	11			21
Gneiss, biotite		2910	1.61	8	74	4.79	3	0.24	2.71	10	6.72	10			21
Gneiss	Bergstrom, TX	2710				4.58	3	0.15	2.59	10	5.38	10			19
		2810				6.28	3	0.29	6.74	10	8.32	10			19
Greenstone	Mt. Weather, VA	3020	2.69	8	81	5.85	3		4.21	10	1.05	11			20
		2960	3.05	8	80	5.21	3		3.86	10	8.07	10			20
Greenstone, amygdaloidal	Catoctin, PA	3040	2.01	8	64	3.99	3	-0.21	3.07	10	4.90	10			3
Hematite, ore	Soudan, MN	5070	6.07	8	74	6.28	3		7.79	10	2.00	11			20
Hematite, ore; par. bedding	Bessemer, AL	3780	1.19	8	51	4.30	3		2.69	10	6.69	10			20
		3670	1.39	8	50	4.30	3		2.70	10	6.73	10			20
Hornfels	Tem Piute Dist., NV	3190	5.33	8		5.49	3		4.09	10	9.58	10			20
Marble, Cherokee	Tate, GA	2707	6.69	7	36						5.59	10			8
Marble, taconic	Rutland, VT	2707	6.21	7	31						4.79	10			8
Marble, perp. bedding	Cockeysville, MD	2870	2.12	8	56	4.18	3		2.61	10	4.93	10			20
Marble, par. bedding		2870	2.23	8	27				2.83	10	6.74	10			20
Marble, paleozoic	Ural Mountains, USSR	2710	1.49	8							7.67	10			2
Marble, dolomitic, f.	Karelian SSR, USSR	2820	2.74	8				0.26	3.00	10	8.94	10			2
Marble, Oro Grande	Oro Grande, CA	2720	1.65	8	56	5.40	3	0.30	3.03	10	7.86	10			3
		2680	5.52	7	42	4.90	3	0.16	2.80	10	6.52	10			3
Metarhyolite	Soudan, MN	2840	1.25	8	47	5.06	3		3.16	10	7.86	10			20
Quartzite, Wissahickon	Washington, DC	2804	4.71	7	38:2.78:63										8
Quartzite, phyllite lenses	Raven, Yugoslavia	2590				8.22	2				1.27	9			13
Quarzite, altered		2590				2.5	3				1.21	10			13
Quartzite, ferruginous	Kursk, USSR	3510	3.43	8							1.71	11			2
Quartzite, Biwabik	Babbitt, MN	2750	6.29	8		5.55	3	0.10	3.86	10	8.48	10			3
Quartzite, hematitic	Ishpeming, MI	4070	2.93	8	71	5.21	3	0.20	4.06	10	9.79	10			4
Phyllite, sericite	El Dorado Co., CA	2340	9.79	6									1.79	10	1
Phyllite, quartzose		2180	9.38	6									7.58	9	1
Phyllite, graphitic	El Dorado Co., CA	2350	6.69	6									9.65	9	1
Phyllite, green	Ishpeming, MI	3240	1.26	8	40	4.85	3		3.28	10	7.65	10			20
Schist, chlorite	Bethesda, MD	2813	2.53	7	37:1.89:51						3.10	10			8
Schist, biotite; Idaho Springs	Montezuma Quad., CO	2720	2.09	7									2.48	10	1
Schist, sericite	Superior, AZ	2700	1.62	8	82	4.72	3		2.62	10	6.00	10			20
Skarn, garnet-pyroxene	Star Lake, NY	3280	1.30	8	61	5.12	3		3.48	10	8.62	10			20
Slate, par. bedding, calcareous	Bangor, PA	2740	1.83	8	56						8.88	10			20
Tactite, epidote	Ophir, UT	2870	2.66	8	65	4.60	3	0.11	2.77	10	6.14	10			21

Table C3. Engineering Properties of Sedimentary Rocks

Rock type; geologic unit	Location	γ (kg/m^3)	C_o (Pa)	Ref.	H[a]	V_p (km/s)	Ref.	u	G (Pa)	Ref.	E_t (Pa)	Ref.	E_s (Pa)	Ref.	Ref.
Borax, ore: Ricardo	Boron, CA	2140	4.41	7	22								4.21	9	4
Chert, chalcedonic; Boone	Picker, OK	2560	3.60	8	96			0.09			5.34	10			3
Chert, dolomitic; Fort Payne	Smithville, TN	2630	2.10	8	74	3.35	3	0.00	1.65	10	3.54	10			4
		2670	2.02	8	67	4.48	3	0.14	2.37	10	5.62	10			4
Conglomerate; Roxbury	Boston, MA	2679	8.28	7	41:6.37:102										8
Conglomerate	Kirkland Lake, Ont.	2670	1.65	8		5.40	3		3.24	10	7.79	10			20
Dolomite, Lockport	Rochester, NY	2765	2.12	8	50:3.24:86						4.48	10			8
	Niagara Falls, NY	2579	9.10	7	44:						5.10	10			8
Dolomite, Bonne Terre	Bonne Terre, MD	2673	1.52	8	49:						6.63	10			8
Dolomite	Jefferson City, TN	2760	3.59	8	69:	5.30	3		3.17	10	7.79	10			20
Dolomite, Beekmantown	Wood Co., WV	2833						0.22	2.95	10	7.23	10			17
		3004						0.22	3.05	10	7.50	10			17
		2783						0.26	3.75	10	9.50	10			17
		2832						0.19	3.64	10	8.65	10			17
Dolomite, Maple Mill	Omaha, NE		3.47	7				0.36			4.79	10			18
		2827	4.32	7				0.05			2.74	10			18
		2818	1.13	8				0.12			6.13	10			18
		2528	4.45	7				0.51			4.39	10			18
		2507	7.08	7				0.09			2.14	10			18
		2531	6.09	7				0.40			8.62	10			18
Dolomite, jointed; Jurassic	Gojak, Yugoslavia	2800				2.51	3				1.27	10			13
Dolomite	Mascot, TN	2840	3.22	8	74	5.46	3		3.52	10	8.48	10			20
Graywacke, m.; Chico	Monticello Dam, CA	2440	4.88	7				0.03					1.24	10	1
		2490	5.07	7				0.02					9.65	9	1
Gypsum	Buffalo, NY	2262	1.25	7	18										8
Jaspillite, ferrugtinous, siliceous sandstone	Ishpeming, MI	3390	3.42	8	85	5.55	3		4.83	10	1.03	11			20
Limestone	Bedford, IN	2206	5.10	7	33:0.43:20	3.91	3				2.85	10			6
Limestone, Solenhofen	Bavaria, FGR	2621	2.45	8	54:1.75:72	5.78	3				6.38	10			6
Limestone, Ozark tavern	Carthage, MO	2659	9.79	7	49						5.59	10			8
Limestone, porous; redwall	Lee's Ferry, AZ	2440	1.33	8				0.18					1.65	10	1
Limestone, reef	Eniwetok, PTT	2300	3.42	7				0.16					3.79	10	1
Limestone, fossiliferous	Bedford, IN	2370	7.52	7	27	3.78	3		1.42	10	3.34	10			20
Limestone, fossiliferous, par. bed.	Bedford, IN	2370	6.85	7	27				1.56	10	3.91	10			20
Limestone, limonitic	Bessemer, AL	2920	1.72	8	61	4.75	3		2.82	10	4.54	10			20
Limestone, marly	Rifle, CO	2250	1.10	8	56	2.38	3		6.90	9	1.25	10			20
Limestone, marly; par. bed.	Rifle, CO	2180				3.11	3		6.76	9	2.14	10			20
Limestone, Martinsburg	Martinsburg, WV	2680	1.59	8	61	5.00	3	0.21	2.73	10	6.59	10			21
Limestone, Black River	Trenton, WV	2688						0.16	2.45	10	5.70	10			17
Limestone, dolomitic; Mesozoic	Turkmenian SSR, USSR	2700	2.10	8							7.62	10			2
Limestone, detrital	Moscow Syncline, USSR	2160	5.20	7							2.90	10			2
Limestone, chalky; Smokey Hill	Pickstown, SD	1410	8.27	6	10	1.34	3	0.30	1.59	9	2.90	9			3
		1710	1.65	7	13	1.74	3	-0.13	2.55	9	4.48	9			3
Limestone, dolomitic; Bonne Terre	Bonne Terre, MO	2660	1.75	8	51	5.09	3	0.22	2.85	10	6.96	10			3
		2780	1.98	8	59	5.88	3	0.29	3.76	10	9.72	10			3
		2710	1.96	8	49			0.05			1.99	10			3
		2690	1.96	8	33	5.36	3	0.22	3.13	10	7.65	10			3
		2670	1.46	8	48	3.78	3	-0.07	2.10	10	3.87	10			3
Limestone, fossiliferous; St. Louis	St. Genevieve, MO	2670	1.64	8	48	5.00	3	0.24	2.68	10	6.67	10			4
Limestone; Wyandotte	Omaha, NE	2546	1.15	7				0.24			2.11	9			18
		2605	4.90	7				0.64			1.61	10			18

Rock type; geologic unit	Location	γ (kg/m^3)	C_o (Pa)	Ref.	H[a]	V_p (km/s)	Ref.	u	G (Pa)	Ref.	E_t (Pa)	Ref.	E_s (Pa)	Ref.	Ref.
Limestone, silurian	Omaha, NE	2352	9.60	7				0.19			3.07	10			18
Limestone, Chickamauga	Smithville, TN	2740	1.73	8	53	4.39	3	0.14	2.33	10	5.30	10			4
		2730	1.73	8	52	3.08	3	0.22	1.17	10	2.72	10			4
Limestone, dolomitic, well-cemented	Pondera Co., MT	2710	1.68	8				0.31					7.65	10	16
Limestone, jointed; Jurassic	Gojak, Yugosalvia	2700				1.92	3				9.16	9			13
Marlstone, mahagony	Rifle, CO	2220	8.14	7	49	3.20	3	0.17	1.02	10	2.41	10			3
Marlstone, par. bed.; mahagony	Rifle, CO	2360	1.72	8	61	4.18	3	0.33	1.53	10	4.10	10			3
Marlstone, Maxville	E. Fultonham, OH	2190	5.59	7	23	3.38	3	0.13	1.10	10	2.50	10			4
Oil Shale, Parachute Creek	Rio Blanco, CO	2044	8.28	7				0.33					6.24	9	12
		2220	1.10	8				0.37					1.12	10	12
		2190	1.81	8				0.30					1.08	10	12
		2124	9.35	7				0.24					7.03	9	12
Quartzite, Baraboo	Baraboo, WI	2627	3.21	8	59						8.84	10			8
Quarzite	Bergstrom, TX	2610	6.45	7							2.76	6			19
		2570	1.26	8							3.56	10			19
		2610	1.75	8							5.91	10			19
		264	2.23	8							6.36	10			19
		2570	1.64	8							5.44	10			19
Salt; diamond crystal	Jefferson Island, LA	2163	2.14	7	23						4.90	9			8
Salt	Bergstrom, TX	2167	1.81	7		3.76	3				6.14	9			19
		2168	1.89	7		3.37	3	0.06			3.45	9			19
		2167	2.85	7		4.08	3				3.45	10			19
		2298	2.20	7		4.07	3	0.189			2.05	10			19
		2317	3.07	7				0.03			3.28	10			19
Sandstone, Navajo	Page, AZ	2015	4.35	7	30:0.04:6	2.52	3						1.53	10	6
Sandstone, Cambridge	Cambridge, MA	2647	4.93	7	27:0.44:18										8
Sandstone, Crab orchard	Crossville, TN	2531	2.14	8	47						3.92	10			6
Sandstone, f.; Tensleep	Casper, WY	2325	7.25	7				0.06					1.31	10	1
Sandstone, c.	Amherst, OH	2170	4.21	7	20	1.20	3		4.00	9	7.10	9			20
Sandstone, c., par. bed.	Amherst, OH	2170	3.55	7	20				4.65	9	1.09	10			20
Sandstone, ferruginous	Bessemer, AL	2930	2.35	8	65	4.05	3		2.42	10	4.96	10			20
	Monogalia Co., WV	2600	1.32	8	53	3.42	3	0.22	1.51	10	3.83	10			21
Sandstone	Huntington, UT	2200	1.07	8		2.44	3	-0.10	7.03	9	1.31	10			21
		2170	7.93	7		2.56	3	0.04	7.03	9	1.45	10			21
		2140	9.79	7		2.19	3	0.04	4.83	9	1.01	10			21
		2350	2.23	8		2.96	3	-0.11	1.17	10	2.07	10			21
		2330	1.91	8		2.87	3	-0.07	1.02	10	1.86	10			21
Sandstone; carboniferous	Donets Basin, USSR	2650	2.56	8				0.14	2.43	10	5.55	10			21
Sandstone, Thorold	Niagara Falls, Ont.	2460						-0.12			2.13	10			11
		2510						-0.18			3.31	9			11
Sandstone, calcareous, nonesuch	White Pine, MT	2600	1.58	8	62	4.63	3	0.16	2.39	10	5.53	10			3
Sandstone, cemented, Navajo	Huntington, UT	2880	1.24	8	50	2.77	3	-0.07	9.45	9	1.75	10			3
Sandstone, cemented; obl. bed.; Navajo	Huntington, UT	2370	3.38	7	54	3.38	3	0.05	1.41	10	2.71	10			3
Sandstone, uncemented; obl. bed.; Navajo	Huntington, UT	2130	5.59	7	32	2.29	3	-0.05	5.86	9	1.12	10			3
Sandstone, uncemented, par. bed.; Navajo	Huntington, UT	2130	3.31	7	36	2.10	3	-0.04	4.96	9	9.58	9			3
Sandstone, Graywacke; Kanawha	DeHue, WV	2600	1.41	8	55	2.93	3	-0.17	1.34	10	2.23	10			3
Sandstone, f.; Morrison/Bushy Basin	Long Park, CO	2540				2.62	3	-0.04	9.10	9	1.76	10			4
Sandstone, Shaly; St. Peter	Omaha, NE	2344	3.73	7				0.05			7.19	9			18
		2450	3.46	7				0.06			1.25	10			18

Sandstone, silty; Seminole	Tulsa, OK	2500	7.45	7	31	2.87	3		1.08	10	2.19	9			4
Sandstone; Homewood	Franklin, PA	2200	8.69	7	43	1.92	3	-0.11	4.69	9	8.27	9			4
Sandstone, Berea	Amherst, OH	2182	7.38	7	42:0.47:29	2.64	3				1.93	10			6
Shale, Rochester	Rochester, NY	2738	1.22	8	45:0.73:39						3.79	10			8
Shale, Brunswick	Highland Park, NJ	2631	8.29	7	38:0.70:31						1.38	10			8
Shale, Bertie	Buffalo, NY	2712	1.97	8	42:1.92:59						5.03	10			8
Shale, siderite, banded; Kanawha	DeHue, WV	2760	1.12	8	38	2.16	3	-0.43	1.17	10	1.33	10			3
Shale, calcareous; Wyandotte	Omaha, NE	2177	1.19	7				0.32			1.97	9			18
Shale, sl. weathered; Cherokee	Omaha, NE	2496	8.34	6				0.15			1.67	9			
Shale, Calcareous; Sheffeild	Omaha, NE	2602	5.98	6				0.14			3.09	10			18
Shale, Maqueketa	Omaha, NE	2618	4.25	7				0.01			7.32	9			18
Shale, carbonaceous; Chattanooga	Smithville, TN	2300	1.12	8	50	2.38	3	00.00	6.55	9	1.39	10			4
		2300	1.10	8	48	2.38	3	-0.02	7.10	9	1.34	10			4
Siltstone; Hackensack	Hackensack, NJ	2595	1.23	8	47:154:58	3.99	3				2.63	10			6
Siltstone, par. bedding; Maxville	E. Fultonham, OH	2660	3.65	7	20			0.13					4.81	10	4
		2680	3.45	7	19			0.26					8.68	10	4
Siltstone, poorly cemented; Bandera	Omaha, NE	2304	3.54	6				0.35			1.25	8			18

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Table D. Properties of Common Minerals

Mineral	Chemical Composition	Crystal System	Streak	Color	Specific Gravity	Hardness
Acanthite	Ag_2S	Morph.: isometric	Black, may mark paper	Gray-black	7.3	2-2.5
Allanite	Ce, La, Th epidote	Monoclinic	Colorless	Brown to pitch-black	3.5-4.2	5.5-6
Alunite	$KAl_3(SO_4)_2(OH)_6$	Hexagonal	Colorless	White, grayish, red	2.6-2.8	4
Amblygonite	$LiAlFPO_4$	Triclinic	Colorless	White, pale green or blue	3.0-3.1	6
Amphibole Group	Essentially hydrous Ca-Mg-Fe silicates	Monoclinic	Colorless	White, green, black	3.0-3.3	5-6
Andalusite	Al_2SiO_5	Orthorhombic	Colorless	Reddish-brown, flesh-red, olive-green	3.16-3.20	7.5
Anglesite	$PbSO_4$	Orthorhombic	Colorless	Colorless or white. Gray-brown when impure	6.2-6.4	3
Anhydrite	$CaSO_4$	Orthorhombic	Colorless	Colorless, white, blue, gray, red	2.89-2.98	3-3.5
Anthophylite	$(Mg,Fe)_7(Si_8O_{22})(OH)_2$	Orthorhombic and monoclinic	Colorless	Gray, clove brown, green	2.85-.3.2	5.5-6
Antlerite	$Cu_3(SO_4)(OH)_4$	Orthorhombic	Light green	Dark emerald-green	3.9±	3.5-4
Apatite	$Ca_5(PO_4)_3(F,Cl,OH)$	Appears amorphous	Colorless	White, yellow, brown, gray	2.6-2.9	3-5
	$Ca_5(PO_4)_3(F,Cl,OH)$	Hexagonal	Colorless	Green, blue, violet, brown, colorless	3.15-3.20	5
Aragonite	$CaCO_3$	Orthorhombic	Colorless	Colorless, white	2.95	3.5-4
Arsenopyrite	FeAsS	Monoclinic	Black	Silver- or tin-white	6.0-6.2	5.5-6
Atacamite	$Cu_2Cl(OH)_3$	Orthorhombic	Light green	Dark emerald-green	3.75-3.77	3-3.5
Azurite	$Cu_3(CO_3)(OH)_2$	Monoclinic	Light blue	Intense azure-blue	3.77	3.5-4
Barite	$BaSO_4$	Orthorhombic	Colorless	Colorless, white, blue, yellow, red	4.5	3-3.5
Bauxite	A mixture of Al hydroxides		Colorless	Yellow, brown, gray, white	2.0-2.55	1-3
Beryl	$Be_3Al_2(Si_6O_{18})$	Hexagonal	Colorless	Bluish-green, yellow, pink, colorless	2.65-2.8	7.5-8
Biotite	$K(Mg, Fe)_3(AlSi_3O_{10})(OH)_2$	Monoclinic	Colorless	Dark brown, green to black; may be yellow	2.95-3	2.5-3

Mineral	Chemical Composition	Crystal System	Streak	Color	Specific Gravity	Hardness
Borax	$Na_2B_4O_5(OH)_4 \cdot 8H_2O)$	Monoclinic	Colorless	Colorless or white	1.7±	2-2.5
Bornite	Cu_5FeS_4	Morph.: isometric	Black	Fresh surface bronish-bronze; purple tarnish	5.1	3
Brochantite	$CuSO_4 \cdot 3Cu(OH)_2$	Orthorhombic	Pale green	Emerald green to blackish green	3.9	3.5-4
Brucite	$Mg(OH)_2$	Hexagonal	Colorless	White, gray, green	2.39	2.5
Calcite	$CaCO_3$	Hexagonal	Colorless	Colorless, white, and variously tinted	2.71	3
Cassiterite	SnO_2	Tetragonal	Light brown	Brown to black	6.8-7.1	6-7
	SnO_2	Tetragonal	Colorless	Reddish-brown to black	6.8-7.1	6-7
Celestite	$SrSO_4$	Orthorhombic	Colorless	Colorless, white, blue, red	3.95-3.97	3-3.5
Cerussite	$PbCO_3$	Orthorhombic	Colorless	Colorless or white	6.55	3-3.5
Chalcocite	Cu_2S	Morph.: orthorhombic	Gray-black	Steel-gray. May tarnish to dead black on exposure	5.7	2.5-3
Chalcopyrite	$CuFeS_2$	Tetragonal	Black	Brass-yellow	4.1-4.3	3.5-4
Chlorite	$Mg, Fe_3(Si,Al)_4O_{10}(OH)_2(Mg,Fe)_3(OH)_6$	Monoclinic	Colorless	Green of various shades	2.6-2.9	2-2.5
Chromite	$FeCr_2O_4$	Isometric	Dark brown to black	Iron-black to brownish-black	4.6	5.5
Chrysocolla	$\sim Cu_4H_4Si_4O_{10}(OH)_8$	Amorphous	Very light blue	Light green to turquoise-blue	2.0-2.4	2-4
Cinnabar	HgS	Hexagonal	Bright red	Red to vermillion	8.1	2-2.5
Clinozoisite	$Ca_2Al_3O(SiO_4)(Si_2O_7)(OH)$	Monoclinic	Colorless	Grayish-white, green, pink	3.25-3.37	6-6.5
Cobaltite-Gersdorffite	(Co, Fe)AsS-NiAsS	Pseudoisometric	Black	Silver- or tin-white	6.33	5.5
Colemanite	$CaB_3O_4(OH)_3 \cdot H_2O$	Monoclinic	Colorless	Colorless, white	2.42	4-4.5
Copper	Cu	Isometric	Copper-red, shiny	Copper-red on fresh surface; black tarnish	8.9	2.5-3
Corundum	Al_2O_3	Hexagonal	Colorless	Colorless, gray, blue, red, yellow brown, green	3.95-4.1	9

Covellite	CuS	Hexagonal	Black. May mark paper	Blue; may tarnish to blue-black	4.6	1.5-2
Cuprite	Cu_2O	Isometric	Red-brown to Indian red	Red-brown to deep red. Ruby-red if transparent	6.0	3.5-4
Diamond	C	Isometric	Colorless	Colorless, yellow, red, blue, black	3.5	10
Dolomite	$CaMg(CO_3)_2$	Hexagonal	Colorless	Colorless, white, pink	2.85	3.5-4
Enargite	Cu_3AsS_4	Orthorhombic	Black	Gray-black	4.4	3
Enstatite	$MgSiO_3$	Orthorhombic	Colorless	Gray-brown, green, bronze-brown, black	3.2-3.5	5.5
Epidote	$Ca_2(Al,Fe)Al_2O(SiO_4)(Si_2O_7)(OH)$	Monoclinic	Colorless	Yellowish to blackish green	3.35-3.45	6-7
Erythrite	$Co_3(AsO_4)_2{\cdot}8H_2O$	Monoclinic	Pink	Red to pink	2.95	1.5-2.5
Fluorite	CaF_2	Isometric	Colorless	Colorless, violet, green, yellow, pink	3.18	4
Galena	PbS	Isometric	Gray-btack	Blue-black to lead-gray	7.6	2.5
Garnet	$A_3B_2(SiO_4)_3$	Isometric	Colorless	Usually brown to red. Also yellow, green, pink	3.5-4.3	6.5-7.5
Goethite	$\alpha FeO{\cdot}OH$	Orthorhombic	Yellow-brown, yellow ocher	Dark-brown to black	4.37	5-5.5
Gold	Au	Isometric	Gold-yellow, shiny	Gold-yellow	15.0-19.3	2.5-3
Graphite	C	Hexagonal	Black	Steel-gray to iron black	2.23	1-1.5
Grunerite	$(Mg,Fe)_7Si_8O_{22}(OH)_2$	Orthorhombic and monoclinic	Colorless	Gray, clove-brown, green	2.85-3.2	5.5-6
Gypsum	$CaSO_4{\cdot}2H_2O$	Monoclinic	Colorless	Colorless, white, gray. May be colored by impurities	2.32	2
Halite	$NaCl$	Isometric	Colorless	Colorless, white, red, blue	2.1-2.3	2.5
Hematite	Fe_2O_3	Hexagonal	Red-brown	Red to vermillion	5.2	1+
	Fe_2O_3	Hexagonal	Red-brown, Indian red	Dark brown to steel-gray to black	4.8-5.3	5.5-6.5

Mineral	Chemical Composition	Crystal System	Streak	Color	Specific Gravity	Hardness
Hemimorphite	$Zn_4(Si_2O_7)(OH)_2 \cdot H_2O$	Orthorhombic	Colorless	White, pale green, blue	3.4-3.5	4.5-5
Ilmenite	$FeTiO_3$	Hexagonal	Dark brown to black	Black	4.7	5.5-6
Jamesonite	$Pb_4FeSb_5S_{14}$	Monoclinic	Black	Gray-black	5.5-6.0	2-3
Kaolinite	$Al_2Si_2O_5(OH)_4$	Triclinic	Colorless	White; may be darker	2.6-2.63	2-2.5
Kernite	$Na_2B_4O_6(OH)_2 \cdot 3H_2O$	Monoclinic	Colorless	Colorless or white to gray	1.95	3
Kyanite	Al_2SiO_5	Triclinic	Colorless	Blue, usually darker at center	3.56-3.66	5-7
Lazurite	$(Na,Ca)_8(AlSiO_4)6(SO_4,S, Cl)_2$	Isometric	Colorless	Deep azure blue, greenish blue	2.4-2.45	5-5.5
Lepidolite	$K(Li, Al)_{2-3}(AlSi_3O_{10})(O, OH, F)_2$	Monoclinic	Colorless	Lilac, grayish white	2.8-3.0	2.5-4
Leucite	$KAlSi_2O_6$	Pseudoisometric	Colorless	Gray, white, colorless	2.45-2.50	5.5-6
Magnesite	$MgCO_3$	Hexagonal	Colorless	Colorless. White, yellow, gray, brown	3.0-3.2	3.5-5
Magnetite	Fe_3O_4	Isometric	Black	Black	5.18	6
Malachite	$Cu_2CO_3(OH)_2$	Monoclinic	Light green	Bright green	3.9-4.03	3.5-4
Manganite	$MnO(OH)$	Morph.: orthorhombic	Dark brown to black	Steel-gray to iron-black	4.3	4
Marcasite	FeS_2	Isometric	Black	Pale yellow to almost white	4.9	6-6.5
Margarite	$CaAl_2(Al_2Si_2O_{10})(OH)_2$	Monoclinic	Colorless	Pink, gray, white	3.0-3.1	3.5-5
Microcline	$KAlSi_3O_8$	Triclinic	Colorless	Colorless, white, gray, cream, red, green	2.54-2.56	6
Microcrystalline Quartz	SiO_2		Colorless	Light brown, yellow, red, green	2.6	7
Millerite	NiS	Hexagonal	Black	Brass-yellow. Slender crystals greenish	5.5	3-3.5
Molybdenite	MoS_2	Hexagonal	Black to greenish black	Blue-black	4.7	1-1.5
Monazite	$(Ce, La, Y, Th)PO_4$	Monoclinic	Colorless	Yellowish-to reddish-brown	5.0-5.3	5-5.5
Muscovite	$KAl_2(AlSi_3O_{10})(OH)_2$	Monoclinic	Colorless	Pale brown, green yellow, white	2.76-2.88	2-2.5
Natrolite	$Na_2Al_2Si_3O_{10} \cdot 2H_2O$	Orthorhombic	Colorless	Colorless, white	2.25	5-5.5
Nepheline	$(Na, K)AlSiO_4$	Hexagonal	Colorless	Colorless gray, greenish, reddish	2.55-2.65	5.5-6

Nickeline (Niccolite)	NiAs	Hexagonal	Black	Pale copper-red. May be silver-white, pinkish	7.8	5-5.5
Olivine	$(Mg, Fe)_2SiO_4$	Orthorhombic	Colorless	Olive to grayish-green, brown	3.27-4.37	6.5-7
Opal	$SiO_2 \cdot nH_2O$	Essentially amorphous	Colorless	Colorless, white, yellow, red, brown	1.9-2.2	5-6
Orpiment	As_2S_3	Monoclinic	Pale yellow	Lemon-yellow	3.49	1.5-2
Orthoclase	$KAlSi_3O_8$	Monoclinic	Colorless	Colorless, white, gray, cream, red, green	2.54-2.56	6
Pentlandite	$(Fe, Ni)_9S_8$	Isometric	Black	Brownish-bronze	4.6-5.0	3.5-4
Phlogopite	$KMg_3(AlSi_3O_{10})(OH)_2$	Monoclinic	Colorless	Yellowish-brown, green-white	2.86	2.5-3
Plagioclase	Various proportions of albite, $NaAlSi_3O_8$, and anorthite, $CaAl_2Si_2O_8$	Triclinic	Colorless	Colorless, white, gray bluish. Often shows play of colors	2.62 (ab)-2.76 (an)	6
Platinum	Pt	Isometric	Gray, shiny	White or steel-gray	14-19	4-4.5
Prehnite	$Ca_2Al(AlSi_3O_{10})(OH)_2$	Orthorhombic	Colorless	Apple-green, gray, white	2.8-2.95	6-6.5
Proustite	Ag_3AsS_3	Hexagonal	Bright red	Ruby-red	5.55	2-2.5
Pyrargyrite	Ag_3SbS_3	Hexagonal	Red-brown, Indian red	Deep red to black	5.8	2.5
Pyrite	FeS_2	Isometric	Black	Pale brass-yellow	5.0	6-6.5
Pyrolusite	MnO_2	Tetragonal	Black	Iron-black	4.7	1-2
Pyromorphite	$Pb_5(PO_4)_3Cl$	Hexagonal	Colorless	Green, brown, yellow, gray	6.5-7.1	3.5-4
Pyrophylite	$Al_2Si_4O_{10}(OH)_2$	Monoclinic	Colorless	White, apple green, gray. When impure, as in soapstone, dark green to almost black	2.8-2.9	1-2
Pyroxene Group	Essentially Ca-Mg-Fe silicates	Monoclinic	Colorless	White, green, black	3.1-3.5	5-6
Pyrrhotite	$Fe_{1-x}S$	Morph.: hexagonal	Black	Brownish-bronze	4.6	4
Quartz	SiO_2	Hexagonal	Colorless	Colorless, white, smoky. Variously colored.	2.65	7
Realgar	AsS	Monoclinic	Orange-yellow	Deep red	3.48	1.5-2
Rhodochrosite	$MnCO_3$	Hexagonal	Colorless	Pink, rose-red	3.45-3.6	3.5-4.5

Mineral	Chemical Composition	Crystal System	Streak	Color	Specific Gravity	Hardness
Rhodonite	$MnSiO_3$	Triclinic	Colorless	Rose-red, pink, brown	3.58-3.70	5.5-6
Rutile	TiO_2	Tetragonal	Colorless to light brown	Reddish-brown to black	4.18-4.25	6-6.5
Scapolite	Essentially Na, Ca aluminum silicate	Tetragonal	Colorless	White, pink, gray, green, brown	2.65-2.74	5-6
Scheelite	$CaWO_4$	Tetragonal	Colorless	White, yellow, green, brown	5.9-6.1	4.5-5
Scorodite	$FeAsO_4 \cdot 2H_2O$	Orthorhombic	Colorless	Leek green to liver-brown	3.1-3.3	3.5-4
Serpentine	$Mg_3Si_2O_5(OH)_4$	Monoclinic	Colorless	Olive- to blackish-green, yellow-green, white	2.3-2.66	2-5
Siderite	$FeCO_3$	Hexagonal	Colorless, brown	Light to dark brown	3.83-3.88	3.5-4
Sillimanite	Al_2SiO_5	Orthorhombic	Colorless	Hair-brown, brown, gray, grayish-green	3.23	6-7
Silver	Ag	Isometric	Silver-white, shiny	Silver-white on fresh surface. Black tarnish	10.5	2.5-3
Skutterudite-Nickel skutterudite	$(Co, Ni, Fe)As_3$-$(Ni,Co, Fe)As_3$	Isometric	Black	Silver- or tin-white	6.1-6.9	5.5
Smithsonite	$ZnCO_3$	Hexagonal	Colorless	Brown, green, blue, pink, white	4.35-4.40	5
Sodalite	$Na_6(AlSiO_4)_6Cl_2$	Isometric	Colorless	White, gray, blue, green	2.15-2.3	5.5-6
Sphalerite	ZnS	Isometric	Light to dark brown	Dark brown to coal-black. More rarely yellow or red	3.9-4.1	3.5-4
Spinel	$MgAl_2O_4$	Isometric	Colorless	Red, black, blue, green, brown	3.6-4.0	8
Spodumene	$LiAlSi_2O_6$	Monoclinic	Colorless	White, gray, pink, green	3.15-3.20	6.5-7
Staurolite	$Fe_2Al_9O_6(SiO_4)_4(O,OH)_2$	Pseudo-orthorhombic	Colorless	Red-brown to brownish-black	3.65-3.75	7-7.5
Stibnite	Sb_2S_3	Orthorhombic	Gray-black	Blue-black	4.5	2
Stilbite	$NaCa_2Al_5Si_{13}O_{36} \cdot 14H_2O$	Monoclinic	Colorless	White, yellow brown, red	2.1-2.2	3.5-4
Strontianite	$SrCO_3$	Orthorhombic	Colorless	Colorless white	3.7	3.5-4
Sulfur	S	Orthorhombic	Colorless, pale yellow	Pale yellow	2.05-2.09	1.5-2.5
Sylvite	KCl	Isometric	Colorless	Colorless or white	1.99	2

Talc	$Mg_3Si_4O_{10}(OH)_2$	Monoclinic	Colorless	White apple-green, gray. When impure as in soapstone, dark green to almost black	2.7-2.8	1
Tenorite	CuO	Triclinic	Dark brown to black	Pitchy black	6.5	3-4
Tetrahedrite	$(Cu, Fe, Zn, Ag)_{12}Sb_4S_{13}$	Isometric	Black; may have brown tinge	Steel-gray. May tarnish to dead black on exposure	4.7-5.0	3-4.5
Titanite	$CaTiO(SiO_4)$	Monoclinic	Colorless	Brown, gray, green, yellow	3.4-3.55	5-5.5
Topaz	$Al_2SiO_4(F,OH)_2$	Orthorhombic	Colorless	Colorless yellow, pink, bluish, greenish	3.4-3.6	8
Tourmaline	Complex boron silicate of Na-Ca-Al-Mg-Fe-Mn	Hexagonal	Colorless	Black, green, brown, blue, red, pink, white	3.0-3.25	7-7.5
Turquoise	$CuAl_6(PO_4)_4(OH)_8{\cdot}5H_2O$	Triclinic	Colorless	Blue, bluish-green, green	2.6-2.8	6
Ulexite	$NaCaB_5O_6(OH){\cdot}5H_2O$	Triclinic	Colorless	White	1.95	1
Uraninite	UO_2	Isometric	Dark brown to black	Black	9.0-9.7	5.5
Vesuvianite	Complex hydrous Ca-Mg-Fe-Al silicate	Tetragonal	Colorless	Green, brown yellow, blue, red	3.35-4.45	6.5
Wavellite	$Al_3(PO_4)_2(OH)_3{\cdot}5H_2O$	Orthorhombic	Colorless	Yellow, green, white, brown	2.33	3.5-4
Willemite	Zn_2SiO_4	Hexagonal	Colorless	Yellow-green, white blue, gray, brown	3.9-4.2	5.5
Witherite	$BaCO_3$	Orthorhombic	Colorless	Colorless, white	4.3	3.5
Wolframite	$(Fe, Mn)WO_4$	Monoclinic	Brown to black	Brown to black	7.0-7.5	5-5.5
Wollastonite	$CaSiO_3$	Triclinic	Colorless	Colorless, white, gray	2.8-2.9	5-5.5
Wulfenite	$PbMoO_4$	Tetragonal	Colorless	Yellow, orange, red, gray, green	6.8±	3
Zircon	$ZrSiO_4$	Tetragonal	Colorless	Brown, red, gray, green, colorless	4.68	7.5

Source: Klein and Hurlbut, 1985.

Properties of Common Minerals with Emphasis on Ore Minerals

This table contains properties of some common minerals arranged in alphabetical order. Emphasis has been placed on common ore minerals, including those minerals important in the examination of weathered outcrops. The properties were compiled primarily from the determinative tables found in

Klein, C., and Hurlbut, C.S., Jr., 1985, *Manual of Mineralogy,* 20th ed., John Wiley and Sons, New York, pp. 531-563.

The table normally presents the predominant crystal system found in hand specimens. Other crystal forms often exist for individual minerals. A more exhaustive list of these crystal systems can be found in the following reference:

Bayliss, P., Erd, R.C., Mrose, M.E., Sabina, A.P., Smith, D.K., 1986, *Mineral Powder Diffraction File,* International Center for Diffraction Data, Swarthmore, PA, 1390 pp.

Editing of this appendix table, selection of pertinent minerals, and addition of some of the ore minerals were performed by Dr. Willard C. Lacy, Lacy and Associates, Green Valley, AZ, and Dr. Donald E. Ranta, Phelps Dodge Mining Company, Lakewood, CO. The editors acknowledge their help.

Permission to use the information from Klein and Hurlbut was provided by John Wiley and Sons. Readers who need a determinative form of the table for identification of unknown minerals should refer to the tables provided in the reference above by Klein and Hurlbut.

Table E. Material Properties and Characteristics

Material	Bank Density (lb/ft^3)	Loose Density (lb/ft^3)	Angle of Repose (degrees)	Angle of Incline (degrees)
Alumina	--	60	22	18
Ammonium nitrate	--	45	--	--
Asbestos ore	--	81	30-44	--
Ashes, dry	--	35-40	40	22
Ashes, wet	--	45-50	50	25
Bauxite, ground,dry	--	68	20-31	17-20
Bauxite, run of mine	100-160	75-120	31	18
Bauxite, crushed, 3 x 0 in.	--	75-85	30-44	20
Clay, compact, natural bed	109	82	--	--
Clay, dense, tough or wet	111	83	--	--
Clay, dry	85	68	--	--
Clay, dry excavated	69	--	--	--
Clay, dry in lump, loose	--	60-70	35	20
Clay, fines	--	100-120	35	20-22
Clay, light (kaolin)	104	80	--	--
Clay and gravel, dry	100	71	--	--
Clay and gravel, wet	114	81	--	--
Chrome ore	--	125-140	30-44	--
Cinders, coal	--	40-45	35	22
Coal, anthracite	81-85	60-63	27	16
Coal, anthracite, sized	--	55-60	27	16
Coal, bituminous	70	50-52	45-55	22-24
Coal, bituminous, mined, sized	--	45-55	35	16
Coal, bituminous, mined, run-of-mine	--	45-55	38	18
Coal, bituminous, mined, slack, 1/2 in. & under	--	43-50	40	22
Coal, bituminous, strip, not cleaned	--	50-60	--	--
Coal, lignite	--	40-45	38	22
Coke	--	24-31	--	--
Coke, breeze, 1/4 in. & under	--	25-34	30-45	20

Material	Bank Density (lb/ft^3)	Loose Density (lb/ft^3)	Angle of Repose (degrees)	Angle of Incline (degrees)
Coke, loose	--	23-35	30-44	--
Coke, petroleum	--	35-40	--	20
Copper ore	141	100-160	--	20
Earth, dry	104	57-83	35	20
Earth, dry, loam	78	57-68	--	--
Earth, moist	100	75-85	--	22
Earth, wet	125	100-104	--	
Earth, wet, containing clay	--	100-110	45	23
Earth, sand, gravel	115	98	--	--
Earth, rock	93-119	71-91	--	--
Feldspar, 1/2 in. screenings	--	70-85	38	18
Feldspar, 1 1/2 to 3 in.	--	90-100	34	17
Feldspar, 200 mesh	--	100	30-44	--
Gneiss	168	96	--	--
Granite	167	90-111	--	20
Granite and porphyry	170	97	--	--
Graphite ore	--	65-75	30-44	--
Gravel, run-of-bank	--	90-100	38	18-20
Gravel, dry	91-120	46-107	--	--
Gravel, dry, screened	--	90-100	40	18
Gravel, wet	144	131	--	--
Gypsum	163-167	100-111	--	--
Gypsum, 1/2 in. screenings	--	70-80	40	21
Gypsum, 1 1/2 to 3 in.	--	70-80	30	15
Iron ore	--	100-200	35	18-20
Iron ore pellets	--	116-130	30-44	13-15
Iron ore, hematite	241-322	144-145	--	--
Iron ore, taconite	150-200	107-143	--	--
Kaolin	104	80	--	--
Lead ore	--	200-270	30	15
Lime, pebble	--	53-56	30	17
Limestone	163	99	--	--
Limestone, blasted	156	89-93	--	--
Limestone, crushed	--	85-90	38	18
Limestone, marble	170	97-101	--	--
Manganese ore	--	125-140	39	20
Mud, dry	80-110	66-91	--	--
Mud, wet	110-130	91-108	--	--
Nickel-cobalt sulfate ore	--	80-150	30-44	--
Rock, crushed	--	125-145	20-29	--
Rock, well blasted	148	99	--	--
Rock, soft, excavated with shovel	--	100-110	30-44	22
Rock, stone, crushed	120-145	89-107	--	--
Sand, bank, damp	--	105-130	45	20-22
Sand, bank, dry	--	90-110	35	16-18
Sand, dry	81-126	70-115	--	--
Sand, moist	126	110	--	--
Sand and gravel, dry	123	108	--	--
Sand and gravel, wet	144	125	--	--
Sandstone	144-153	96-110	--	--
Sandstone, broken	--	85-90	30-44	--
Shale, broken	--	90-100	20-29	--
Shale, crushed	--	85-90	39	22
Shale, riprap	104	78	--	--
Slag	136	110	--	--
Slate	170-180	131-139	--	
Stone, crushed	120-145	89-107	--	--
Sulphur ore	--	87	--	--
Trap rock	185	122-124	--	--
Zinc ore, crushed	--	160	38	22

Note: All material properties given here may vary considerably from those experienced in the field. Conversion factors: 1 lb/ft^3 = 16.0185 kg/m^3, 1 in. = 25.4 mm.

(*Sources:* Adapted by Johnny D. Shumate, University of Kentucky, Department of Mining Engineering from: Adler, L.A., and Naumann, H E., 1970, *Analyzing Excavation and Handling Equipment,* Research Division Bulletin 53, VPI and SU, Blacksburg, VA, pp. 9 and 194; Conveyor Equipment Manufacturers Association, 1988, *Belt Conveyors for Bulk Materials,* 3rd. ed., CEMA, pp. 42-50; Killebrew, C.E., 1968, Section 8.3, "Tractor Shovels, Tractor Dozers, Tractor Scrapers," *Surface Mining,* 1968, E.P. Pfleider, ed., AIME, New York, p. 466; Woodruff, S.D., 1966, *Methods of Working Coal and Metal Mines,* Vol. 3, Pergamon Press, New York, pp. 496-507.)

Table F. Heat Value of Various Fuels

Fuel	API Gravity	Btu/lb	Btu/gal
Aviation gasoline	68	20,420	120,700
Motor gasoline	58	20,120	125,800
Kerosene	42	19,810	134,700
Domestic fuel oil	32	19,450	141,200
Diesel fuel oil	28	19,350	143,100
Industrial fuel oil	18	18,930	149,400
Industrial fuel oil	11	18,590	153,900

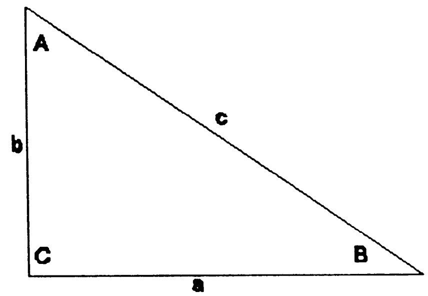
	Density 20°/4°C	Btu/lb	CAI/G
Crude Petroleum	0.828-.989	18,180-20,670	10,100-10,940
Lignite		6,347- 7,189	3,526-3,994
Subbituminous		9,207-10,557	5,115-5,865
Bituminous		10,958-14,134	6,088-7,852
Semibituminous		14,121-14,699	7,845-8,166
Semianthracite		13,702	7,612
Anthracite		12,577-13,351	6,987-7,417
Oven coke		14,300-14,410	7,946-8,006

	Sp Gr (Air = 1)	Heat of Combustion kg•cal/m^3	Flame Temp. °C (No Excess Air)
Natural gas	0.60-1.29	8,040-17,400	1965
Propane	1.55	20,950	2015
Butane	2.04	26,350	2005
Producer gas	.86	1,182	1665

Source: SME Mining Engineering Handbook, 1st ed., 1973.

Table G. Geometrical Relationships and Mensuration Formulas

Right Triangle

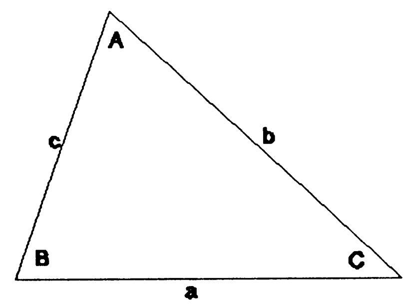


$$\sin A = \frac{a}{c}, \cos A = \frac{b}{c}, \tan A = \frac{a}{b}$$

$$c = \sqrt{a^2 = b^2}$$

$$\text{Area} = \frac{1}{2} a\, b$$

Oblique Triangle



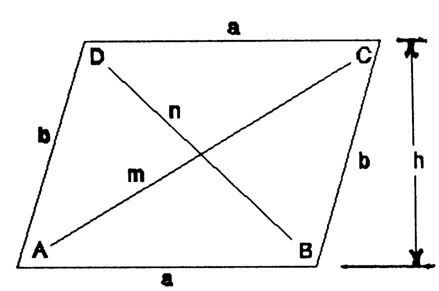
$$\frac{\sin A}{a} = \frac{\sin B}{b} = \frac{\sin C}{c}$$

$$c^2 = a^2 + b^2 - 2ab \cos C$$

$$\text{Area} = \frac{1}{2} ab \sin C$$

$$s = \frac{1}{2} a + b + c, \text{ Area} = \sqrt{s(s-a)(s-b)(s-c)}$$

Parallelogram



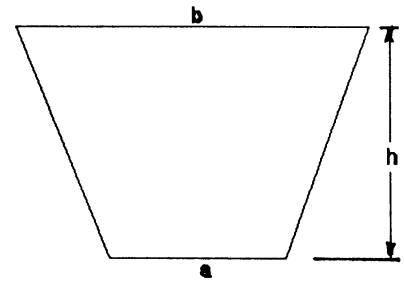
$$h = b \sin A, \text{ Area} = ah = ab \sin A$$

$$m = \sqrt{a^2 + b^2 + 2ab \cos A}$$

$$n = \sqrt{a^2 + b^2 - 2ab \cos A}$$

Trapezoid

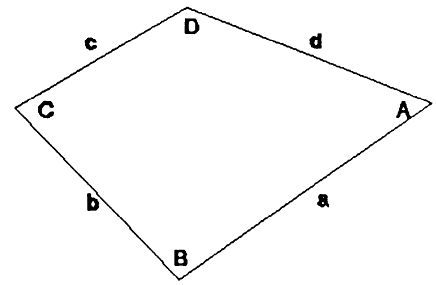
$$\text{Area} = \frac{1}{2}(a+b)\,h$$



Quadrilaterial

$$s = \frac{1}{2}(a+b+c+d)$$

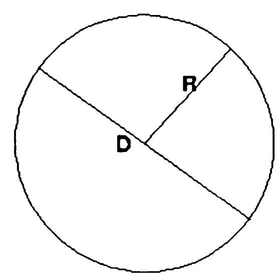
$$\text{Area} = \sqrt{(s-a)(s-b)(s-c)(s-d) - abcd\cos^2\frac{A+C}{2}}$$



Circle ($\pi = 3.14159...$)

$$\text{Circumference} = 2\pi R = \pi D$$

$$\text{Area} = \pi R^2 = \frac{1}{4}\pi D^2$$



Sector and Segment

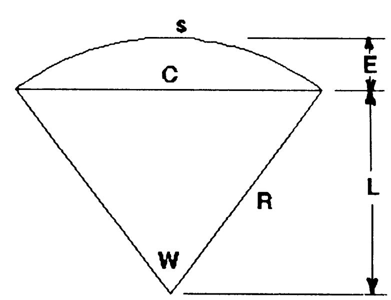
$$L = R\cos\frac{W}{2}$$

$$\text{cord length }(C) = 2\frac{R\sin W}{2}$$

$$\text{arc length }(s) = R\ \frac{W}{2},\ W(\text{radians})$$

$$\text{Area (sector)} = \frac{1}{2}R^2 W$$

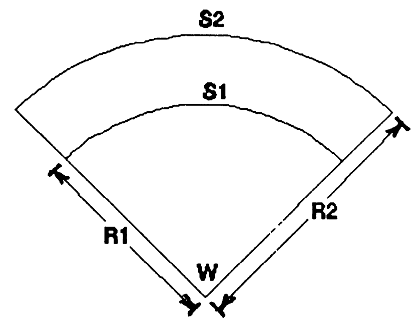
$$\text{Area (segment)} = \frac{1}{2}R^2(W - \sin W)$$



Annulus

$$\text{Area} = \frac{1}{2} W\,(R1 + R2)\,(R2 - R1)$$

$$\text{Area} = \frac{1}{2}\,(R2 - R1)\,(s1 + s2)$$

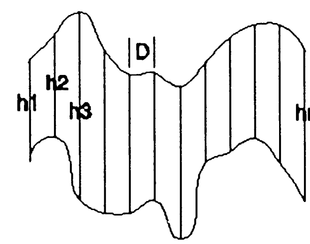


Plane Areas

Divide the two-dimensional area into n-1 strips of length h1, h2, h3, . . . hn (h1 and/or hn may be zero). D is the common separation between lines

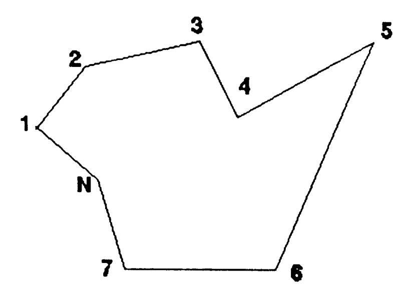
Trapezoidal Rule (approximate)

$$\text{Area} = D\left[\frac{1}{2}h\,1 + h\,2 + h\,3 + \ldots h\,(n-1) + \frac{1}{2}H\,(n)\right]$$



Simpson's Rule (approximate) where n is odd.

$$\text{Area} = \frac{1}{3}D\,[h1 + 4h2 + 2h3 + 4h4 + \ldots + 2h(n-2) + 4h(n-1) + h(n)]$$



Area by coordinates (exact) x1,y1 coordinates of point 1

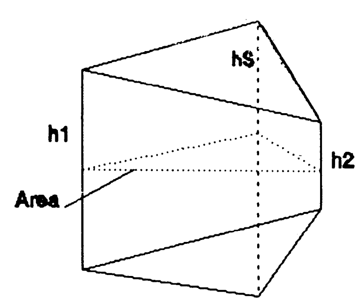
$$\text{Area} = \frac{1}{2}[x1\,(y2 - y(n)) + x2\,(y3 - y1) + \ldots + x(n)\,(y1 - y(n-1))]$$

$$\text{Area} = \frac{1}{2}[y1\,(x2 - x(n)) + y2\,(x3 - x1) + \ldots + y(n)\,(x1 - x(n-1))]$$

Triangular Prism

$$\text{Volume} = \frac{1}{3}A\,(h1 + h2 + h3)$$

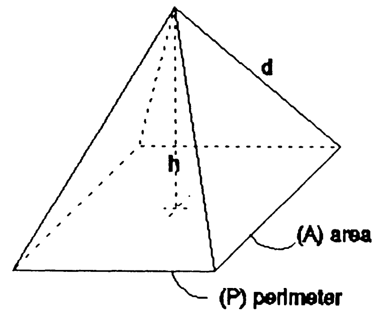
where A = area of right section



Pyramid

$$\text{Lateral surface area} = \frac{1}{2}(Pd)$$

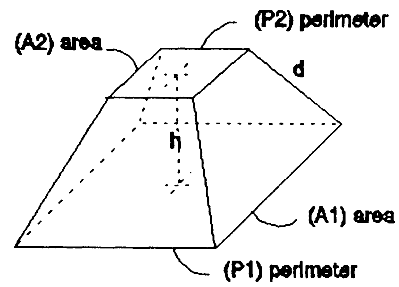
$$\text{Volume} = \frac{1}{3}(Ah)$$



Frustum of a Pyramid

$$\text{Lateral surface area} = \frac{1}{2}(P1 + P2)d$$

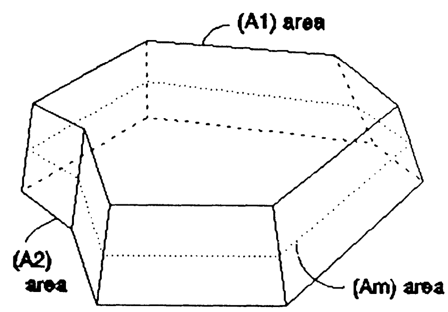
$$\text{Volume} = \frac{1}{3}h\left(A1 + A2 + \sqrt{A1A2}\right)$$



Prismatoid

Top and bottom surfaces must be parallel. Al is the surface area of the top, A2 is the surface area of the bottom, A(m) is the surface area of midsection, and h is the altitude.

$$\text{Volume} = \frac{1}{6}h\ (A1 + 4A(m) + A2)$$

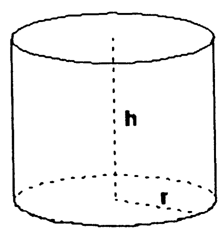


Right Circular Cylinder

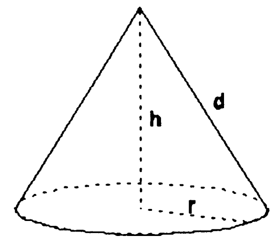
Lateral surface area = $2\pi\ r\ h$

Total surface area = $2\pi\ r\ (r+h)$

Volume = $\pi\ r^2\ h$



Right Circular Cone

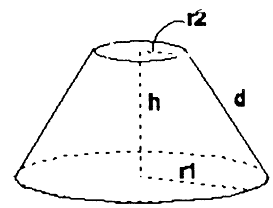


$$\text{Lateral surface area} = \pi\, r \sqrt{r^2 + h^2}$$

$$\text{Total surface area} = \pi\, r \left(r + \sqrt{r^2 + h^2} \right)$$

$$\text{Volume} = \frac{1}{3} \pi\, r^2\, h$$

Frustum of a Right Circular Cone

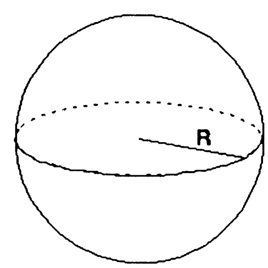


$$\text{Total surface area} = \pi[r1^2 + r2^2 + (r1 + r\ )]$$

$$\text{Lateral surface area} = \pi(r1 + r2)\sqrt{(r1 - r2)^2 + h^2}$$

$$\text{Total surface area} = \pi[r1^2 + r2^2 + (r1 + r2)\sqrt{(r1 - r2)^2 + h^2}\ ]$$

Sphere

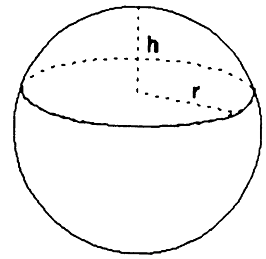


$$\text{Surface area} = 4\pi R^2$$

$$\text{Volume} = \frac{4}{3}\pi R^3$$

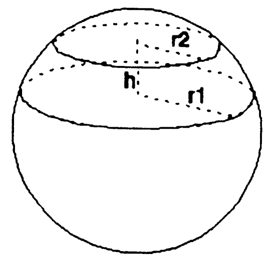
Spherical Segment with One Base

Surface area = $2 \pi R h$



$$\text{Volume} = \frac{1}{3} \pi h^2 (3 R - h) = \frac{1}{6} \pi h (3 r^2 + h^2)$$

Spherical Segment with Two Bases



Surface area = $2 \pi r h$

$$\text{Volume} = \frac{1}{6} \pi h \left[3 (r1)^2 + 3 (r2)^2 + h^2 \right]$$

Source: M.K. McCarter, University of Utah

REFERENCE

Eves, H., 1979, Mensuration Formulas, *CRC Standard Mathematical Tables,* CRC Press, Inc., W. H. Beyer, ed., Boca Raton, FL, pp. 139-150.

Table H. Standard Sizing Scale Based on the Standard 200-Mesh Screen $\sqrt{2}$

Size in.	Size mm	Size micrometers	Mesh	Sizing Method	Example
1.050	26.67				
0.742	18.85				River gravel
0.525	13.33				
0.371	9.423				
0.263	6.680		3		Pea gravel
0.185	4.699		4		
0.131	3.327		6		Fine gravel
0.093	2.362		8		
0.065	1.651		10		
0.046	1.168		14		
0.0328	0.833		20		
0.0232	0.589		28		
0.0164	0.417		35	Screen	
0.0116	0.295		48		Beach sand
0.0082	0.208		65		
0.0058	0.147		100		
0.0041	0.104		150		
0.0029	0.074	74	200		
	0.052	52	270		
	0.037	37	400	Classification	
		26			Fine silt
		18.5	800		
		13			
		9.25	1600		Blood cells
		6.5			
		4.62	3200		
		3.25			
		2.32			
		1.62		Microscope	Many germs
		1.16	A°		
		0.81			
		0.58	5800		Wave length visible light
	0.41				
		0.29			
		0.20			
		0.14			
		0.10		Centrifuge	
		0.07			
		0.05			
		0.035			Thinnest iridescent films by interference
	0.025				
		0.017			
		0.012			
		0.008			
		0.006			
		0.004			Very large molecules
		0.003			
		0.002			
		0.0015			
		0.001	10		
		0.0007			
		0.0005	5		Average unit crystal

Conversion Factors: 1A° = 1 x 10^{-10} m, 1 μ = 1 x 10^{-6}m, 1 mm = 3.937 x 10^{-2} in.
Source: *SME Mining Engineering Handbook*, 1st ed., 1973.

Table I. Ship Measures

Ship measurements are of 5 kinds. In the US and in British countries, the ton = 2,240 lb; where the metric ton is used, 2,204.6 lb.

(I) *Displacement* is total wt in tons of a vessel and its contents. Displacement "light" = wt of vessel without stores, bunker fuel, or cargo; displacement "loaded" = total wt, including the last 3 items.

(2) *Cargo tonnage* is expressed on the basis of either "weight" or "cubic measurement." "Measurement ton" is usually 40 ft^3 (as in the US); in some countries, 42 ft^3.

(3) Gross *tonnage* applies to the vessel itself, not to cargo, and is the contents in ft^3 of its closed-in spaces, divided by 100 (assuming a vessel-ton to be 100 ft^3). "Register" states both gross and net tonnage, and is about the same under US and British rules.

(4) *Net tonnage* is gross tonnage, less deductions for space occupied by crew, engine room and fuel, and for navigation; that is, the space available for passengers and cargo. As a ton of cargo usually occupies much less than 100 ft^3, see (2), cargo tonnage generally exceeds both net and gross tonnage.

(5) *Deadweight tonnage* is the number of tons a vessel can carry, of cargo, stores, and fuel. It is the difference between tons of water displaced when the vessel is "light" and when submerged to "load line" (which is often shown on the vessel's side by the "Plimsoll mark"). Capacity for weight cargo is less than dead-weight tonnage.

Examples of relative measurements. (*a*) An ordinary freight steamer: net tonnage, 4,000; gross tonnage, 6,000; deadweight capacity, 10,000 tons; displacement, loaded, 13,350 tons. (*b*) Oil tanker, Standard Oil Co. of California, launched 1938: net tonnage, 4,934; gross tonnage, 8,298; deadweight capacity, 12,800 tons; displacement, loaded, 17,320 tons.

Conversion factors: 1 ton = 0.9072 t, 1 ft^3 = 0.02832 m^3.
Source: Mining Engineer's Handbook, 1941, Robert Peele and J.A. Church, John Wiley and Sons, New York.

Table J. International Atomic Weights

Element	Symbol	Atomic No.	Atomic Weight	Valence
Actinium	Ac	89	227	—
Aluminum	Al	13	26.98	3
Americium	Am	95	(243)	3,4,5,6
Antimony	Sb	51	121.76	3,5
Argon	Ar	18	39.944	0
Arsenic	As	33	74.92	3,5
Astatine	At	85	(210)	1,3,5,7
Barium	Ba	56	137.36	2
Berkelium	Bk	97	(247)	3,4
Beryllium	Be	4	9.013	2
Bismuth	Bi	83	208.99	3,5
Boron	B	5	10.82	3
Bromine	Br	35	79.916	1,3,5,7
Cadmium	Cd	48	112.41	2
Calcium	Ca	20	40.08	2
Californium	Cf	98	(251)	—
Carbon	C	6	12.011	2,4
Cerium	Ce	58	140.13	3,4
Cesium	Cs	55	132.91	1
Chlorine	Cl	17	35.457	1,3,5,7
Chromium	Cr	24	52.01	2,3,6
Cobalt	Co	27	58.94	2,3
Copper	Cu	29	63.54	1,2
Curium	Cm	96	(247)	3
Dysprosium	Dy	66	162.51	3

Einsteinium	Es	99	(254)	—
Erbium	Er	68	167.27	3
Europium	Eu	63	152.0	2,3
Fermium	Fm	100	(257)	—
Fluorine	F	9	19.00	1
Francium	Fr	87	(223)	1
Gadolinium	Gd	64	157.26	3
Gallium	Ga	31	69.72	2,3
Germanium	Ge	32	72.60	4
Gold	Au	79	197.0	1,3
Hafnium	Hf	72	178.50	4
Helium	He	2	4.003	0
Holmium	Ho	67	164.94	3
Hydrogen	H	1	1.0080	1
Indium	In	49	114.82	3
Iodine	I	53	126.91	1,3,5,7
Iridium	Ir	77	192.2	3,4
Iron	Fe	26	55.85	2,3
Krypton	Kr	36	83.80	0
Kurchatovium	Ku	104	—	—
Lanthanum	La	57	138.92	3
Lawrencium	Lr	103	—	3
Lead	Pb	82	207.21	2,4
Lithium	Li	3	6.940	1
Lutetium	Lu	71	174.99	3
Magnesium	Mg	12	24.32	2
Manganese	Mn	25	54.94	2,3,4,6,7
Mendelevium	Md	101	(256)	—
Mercury	Hg	80	200.61	1,2
Molybdenum	Mo	42	95.95	3,4,6
Neodymium	Nd	60	144.27	3
Neon	Ne	10	20.183	0
Neptunium	Np	93	(237)	4,5,6
Nickel	Ni	28	58.71	2,3
Niobium	Nb	41	92.91	3,5
Nitrogen	N	7	14.008	3,5
Nobelium	No	102	(254)	
Osmium	0s	76	190.2	2,3,4,8
Oxygen	0	8	16	2
Palladium	Pd	46	106.4	2,4,6
Phosphorus	P	15	30.975	3,5
Platinum	Pt	78	195.09	2,4
Plutonium	Pu	94	(244)	3,4,5,6
Polonium	Po	84	(209)	—
Potassium	K	19	39.100	1
Praseodymium	Pr	59	140.92	3
Promethium	Pm	61	(145)	3
Protactinium	Pa	91	(231)	—
Radium	Ra	88	(226)	2
Radon	Rn	86	(222)	0
Rhenium	Re	75	186.22	—
Rhodium	Rh	45	102.91	3
Rubidium	Rb	37	85.48	1
Ruthenium	Ru	44	101.1	3,4,6,8
Samarium	Sm	62	150.35	2,3
Scandium	Sc	21	44.96	3
Selenium	Se	34	78.96	2,4,6
Silicon	Si	14	28.09	4
Silver	Ag	47	107.873 1	
Sodium	Na	11	22.991	1

Element	Symbol	Atomic No.	Atomic Weight	Valence
Strontium	Sr	38	87.63	2
Sulfur	S	16	32.066	2,4,6
Tantalum	Ta	73	180.95	5
Technetium	Tc	43	(97)	6,7
Tellurium	Te	52	127.61	2,4,6
Terbium	Tb	65	158.93	3
Thallium	Tl	81	204.39	1,3
Thorium	Th	90	(232)	4
Thulium	Tm	69	168.94	3
Tin	Sn	50	118.70	2,4
Titianium	Ti	22	47.90	3,4
Tungsten	W	74	183.86	6
Uranium	U	92	238.07	4,6
Vanadium	V	23	50.95	3,5
Xenon	Xe	54	131.30	0
Ytterbium	Yb	70	173.04	2,3
Yttrium	Y	39	88.91	3
Zinc	Zn	30	65.38	2
Zirconium	Zr	40	91.22	4

The value in parenthesis in the atomic weight column is, in each case, the mass number of the most stable isotope.

Source: SME Mining Engineering Handbook, 1st ed., 1973.

Table K. Time Value of Money Factors for Discrete Compounding

The time value of money relationships provided here are based upon the notations used in D.W. Gentryand T.J. O'Neil, *Mine Investment Analysis*, 1984, SME-AIME, New York, 502 pp.

Symbols used in the equations and tables are as follows:

F = a future sum of money in \$.

P = a present single sum of money in \$.

A = the amount of a single payment in a series of n equal payments, each payment to be made at the end of a period in \$.

i = effective interest rate per period, expressed as a decimal (e.g., 5% = 0.05).

n = number of interest periods (usually n = numbers of years; however, other periods, e.g., months, quarters, may apply instead).

g = an arithmetic amount by which a payment increases each period in \$.

The tables that follow provide seven times the value of money factors that are defined below along with a method for applying each factor.

(1) *Single Payment Compound Amount Factor:*

$$F/P_{i,n} = (1 + i)^n$$

Use this factor to multiply a present amount P to determine what future amount F will accumulate over n periods at an effective interest rate i per period.

(2) *Single Payment Present Worth Factor:*

$$P/F_{i,n} = \frac{1}{(1+i)^n}$$

Use this factor to multiply a future sum F to determine its equivalent present worth P where the present is n periods earlier at an effective interest rate i per period.

(3) *Uniform Series Compound Amount Factor:*

$$F/A_{i,n} = [(1+i)^n - 1]/i$$

Multiply the end-of-period payment A by this factor to determine the future sum F that will accumulate after n such payments at an effective interest rate i per period.

(4) *Sinking Fund Factor:*

$$A/F_{i,n} = \frac{1}{[(1+i)^n - 1]/i}$$

Use this factor to multiply a future sum F to determine the equivalent end-of-period payments A for n earlier periods at an effective interest rate i per period.

(5) *Present Worth Factor:*

$$P/A_{i,n} = \frac{[(1+i)^n - 1]}{i(1+i)^n}$$

Multiply this factor by and end-of-period payment A made for n periods to determine an equivalent present sum P at an effective interest rate i per period.

(6) *Capital Recovery Factor:*

$$A/P_{i,n} = \frac{i(1+i)^n}{\left[(1+i)^n - 1\right]}$$

Use this factor to multiply by a present amount P to determine an equivalent future series of end-of-period payments A over n periods at an effective interest rate i per period.

(7) *Uniform Gradient Series Factor:*

$$A/G_{i,n} = \frac{1}{i} - \frac{n}{i}(A/F_{i,n})$$

$$= \frac{1}{i} - \frac{n}{\left[(1+i)^n - 1\right]}$$

Use the above relationship to determine the end-of-period payments A that is equivalent in present value to n payments starting at \$0 at the end of period 1 and increasing by an amount G per period until the end of period n.

1/2% Interest Factors

n	Single Payment: Compound-amount factor (To find F Given P, F/P i,n)	Single Payment: Present-worth factor (To find P Given F, P/F i,n)	Equal Payment Series: Compound-amount factor (To find F Given A, F/A i,n)	Equal Payment Series: Sinking-fund factor (To find A Given F, A/F i,n)	Equal Payment Series: Present-worth factor (To find P Given A, P/A i,n)	Equal Payment Series: Capital-recovery factor (To find A Given P, A/P i,n)	Uniform gradient-series factor (To find A Given G, A/G i,n)
1	1.005	0.9950	1.000	1.0000	0.9950	1.0050	0.0000
2	1.010	0.9901	2.005	0.4988	1.9851	0.5038	0.4988
3	1.015	0.9852	3.015	0.3317	2.9703	0.3367	0.9967
4	1.020	0.9803	4.030	0.2481	3.9505	0.2531	1.4938
5	1.025	0.9754	5.050	0.1980	4.9259	0.2030	1.9900
6	1.030	0.9705	6.076	0.1646	5.8964	0.1696	2.4855
7	1.036	0.9657	7.106	0.1407	6.8621	0.1457	2.9801
8	1.041	0.9609	8.141	0.1228	7.8230	0.1278	3.4738
9	1.046	0.9561	9.182	0.1089	8.7791	0.1139	3.9668
10	1.051	0.9514	10.228	0.0978	9.7304	0.1028	4.4589
11	1.056	0.9466	11.279	0.0887	10.6770	0.0937	4.9501
12	1.062	0.9419	12.336	0.0811	11.6189	0.0861	5.4406
13	1.067	0.9372	13.397	0.0747	12.5562	0.0797	5.9302
14	1.072	0.9326	14.464	0.0691	13.4887	0.0741	6.4190
15	1.078	0.9279	15.537	0.0644	14.4166	0.0694	6.9069
16	1.083	0.9233	16.614	0.0602	15.3399	0.0652	7.3940
17	1.088	0.9187	17.697	0.0565	16.2586	0.0615	7.8803
18	1.094	0.9141	18.786	0.0532	17.1728	0.0582	8.3658
19	1.099	0.9096	19.880	0.0503	18.0824	0.0553	8.8504
20	1.105	0.9051	20.979	0.0477	18.9874	0.0527	9.3342
21	1.110	0.9006	22.084	0.0453	19.8880	0.0503	9.8172
22	1.116	0.8961	23.194	0.0431	20.7841	0.0481	10.2993
23	1.122	0.8916	24.310	0.0411	21.6757	0.0461	10.7806
24	1.127	0.8872	25.432	0.0393	22.5629	0.0443	11.2611
25	1.133	0.8828	26.559	0.0377	23.4456	0.0427	11.7407
26	1.138	0.8784	27.692	0.0361	24.3240	0.0411	12.2195
27	1.144	0.8740	28.830	0.0347	25.1980	0.0397	12.6975
28	1.150	0.8697	29.975	0.0334	26.0677	0.0384	13.1747
29	1.156	0.8653	31.124	0.0321	26.9330	0.0371	13.6510
30	1.161	0.8610	32.280	0.0310	27.7941	0.0360	14.1265
31	1.167	0.8568	33.441	0.0299	28.6508	0.0349	14.6012
32	1.173	0.8525	34.609	0.0289	29.5033	0.0339	15.0750
33	1.179	0.8483	35.782	0.0280	30.3515	0.0330	15.5480
34	1.185	0.8440	36.961	0.0271	31.1956	0.0321	16.0202
35	1.191	0.8398	38.145	0.0262	32.0354	0.0312	16.4915
40	1.221	0.8191	44.159	0.0227	36.1722	0.0277	18.8358
45	1.252	0.7990	50.324	0.0199	40.2072	0.0249	21.1595
50	1.283	0.7793	56.645	0.0177	44.1428	0.0227	23.4624
55	1.316	0.7601	63.126	0.0159	47.9815	0.0209	25.7447
60	1.349	0.7414	69.770	0.0143	51.7256	0.0193	28.0064
65	1.383	0.7231	76.582	0.0131	55.3775	0.0181	30.2475
70	1.418	0.7053	83.566	0.0120	58.9394	0.0170	32.4680
75	1.454	0.6879	90.727	0.0110	62.4137	0.0160	34.6679
80	1.490	0.6710	98.068	0.0102	65.8023	0.0152	36.8474
85	1.528	0.6545	105.594	0.0095	69.1075	0.0145	39.0065
90	1.567	0.6384	113.311	0.0088	72.3313	0.0138	41.1451
95	1.606	0.6226	121.222	0.0083	75.4757	0.0133	43.2633
100	1.647	0.6073	129.334	0.0077	78.5427	0.0127	45.3613

3/4% Interest Factors

n	Single Payment: Compound-amount factor (To find F Given P, F/P i,n)	Single Payment: Present-worth factor (To find P Given F, P/F i,n)	Equal Payment Series: Compound-amount factor (To find F Given A, F/A i,n)	Equal Payment Series: Sinking-fund factor (To find A Given F, A/F i,n)	Equal Payment Series: Present-worth factor (To find P Given A, P/A i,n)	Equal Payment Series: Capital-recovery factor (To find A Given P, A/P i,n)	Uniform gradient-series factor (To find A Given G, A/G i,n)
1	1.008	0.9926	1.000	1.0000	0.9926	1.0075	0.0000
2	1.015	0.9852	2.008	0.4981	1.9777	0.5056	0.4981
3	1.023	0.9778	3.023	0.3309	2.9556	0.3384	0.9950
4	1.030	0.9706	4.045	0.2472	3.9261	0.2547	1.4907
5	1.038	0.9633	5.076	0.1970	4.8894	0.2045	1.9851
6	1.046	0.9562	6.114	0.1636	5.8456	0.1711	2.4782
7	1.054	0.9491	7.159	0.1397	6.7946	0.1472	2.9701
8	1.062	0.9420	8.213	0.1218	7.7366	0.1293	3.4608
9	1.070	0.9350	9.275	0.1078	8.6716	0.1153	3.9502
10	1.078	0.9280	10.344	0.0967	9.5996	0.1042	4.4384
11	1.086	0.9211	11.422	0.0876	10.5207	0.0951	4.9253
12	1.094	0.9142	12.508	0.0800	11.4349	0.0875	5.4110
13	1.102	0.9074	13.601	0.0735	12.3424	0.0810	5.8954
14	1.110	0.9007	14.703	0.0680	13.2430	0.0755	6.3786
15	1.119	0.8940	15.814	0.0632	14.1370	0.0707	6.8606
16	1.127	0.8873	16.932	0.0591	15.0243	0.0666	7.3413
17	1.135	0.8807	18.059	0.0554	15.9050	0.0629	7.8207
18	1.144	0.8742	19.195	0.0521	16.7792	0.0596	8.2989
19	1.153	0.8677	20.339	0.0492	17.6468	0.0567	8.7759
20	1.161	0.8612	21.491	0.0465	18.5080	0.0540	9.2517
21	1.170	0.8548	22.652	0.0442	19.3628	0.0517	9.7261
22	1.179	0.8484	23.822	0.0420	20.2112	0.0495	10.1994
23	1.188	0.8421	25.001	0.0400	21.0533	0.0475	10.6714
24	1.196	0.8358	26.188	0.0382	21.8892	0.0457	11.1422
25	1.205	0.8296	27.385	0.0365	22.7188	0.0440	11.6117
26	1.214	0.8234	28.590	0.0350	23.5422	0.0425	12.0800
27	1.224	0.8173	29.805	0.0336	24.3595	0.0411	12.5470
28	1.233	0.8112	31.028	0.0322	25.1707	0.0397	13.0128
29	1.242	0.8052	32.261	0.0310	25.9759	0.0385	13.4774
30	1.251	0.7992	33.503	0.0299	26.7751	0.0374	13.9407
31	1.261	0.7932	34.754	0.0288	27.5683	0.0363	14.4028
32	1.270	0.7873	36.015	0.0278	28.3557	0.0353	14.8636
33	1.280	0.7815	37.285	0.0268	29.1371	0.0343	15.3232
34	1.289	0.7757	38.565	0.0259	29.9128	0.0334	15.7816
35	1.299	0.7699	39.854	0.0251	30.6827	0.0326	16.2387
40	1.348	0.7417	46.446	0.0215	34.4469	0.0290	18.5058
45	1.400	0.7145	53.290	0.0188	38.0732	0.0263	20.7421
50	1.453	0.6883	60.394	0.0166	41.5665	0.0241	22.9476
55	1.508	0.6630	67.769	0.0148	44.9316	0.0223	25.1223
60	1.566	0.6387	75.424	0.0133	48.1734	0.0208	27.2665
65	1.625	0.6153	83.371	0.0120	51.2963	0.0195	29.3801
70	1.687	0.5927	91.620	0.0109	54.3046	0.0184	31.4634
75	1.751	0.5710	100.183	0.0100	57.2027	0.0175	33.5163
80	1.818	0.5501	109.073	0.0092	59.9945	0.0167	35.5391
85	1.887	0.5299	118.300	0.0085	62.6838	0.0160	37.5318
90	1.959	0.5105	127.879	0.0078	65.2746	0.0153	39.4946
95	2.034	0.4917	137.823	0.0073	67.7704	0.0148	41.4277
100	2.111	0.4737	148.145	0.0068	70.1746	0.0143	43.3311

1% Interest Factors

n	Single Payment: Compound-amount factor (To find F Given P, F/P i,n)	Single Payment: Present-worth factor (To find P Given F, P/F i,n)	Equal Payment Series: Compound-amount factor (To find F Given A, F/A i,n)	Equal Payment Series: Sinking-fund factor (To find A Given F, A/F i,n)	Equal Payment Series: Present-worth factor (To find P Given A, P/A i,n)	Equal Payment Series: Capital-recovery factor (To find A Given P, A/P i,n)	Uniform gradient-series factor (To find A Given G, A/G i,n)
1	1.010	0.9901	1.000	1.0000	0.9901	1.0100	0.0000
2	1.020	0.9803	2.010	0.4975	1.9704	0.5075	0.4975
3	1.030	0.9706	3.030	0.3300	2.9410	0.3400	0.9934
4	1.041	0.9610	4.060	0.2463	3.9020	0.2563	1.4876
5	1.051	0.9515	5.101	0.1960	4.8534	0.2060	1.9801
6	1.062	0.9421	6.152	0.1626	5.7955	0.1726	2.4710
7	1.072	0.9327	7.214	0.1386	6.7282	0.1486	2.9602
8	1.083	0.9235	8.286	0.1207	7.6517	0.1307	3.4478
9	1.094	0.9143	9.369	0.1068	8.5660	0.1168	3.9337
10	1.105	0.9053	10.462	0.0956	9.4713	0.1056	4.4179
11	1.116	0.8963	11.567	0.0865	10.3676	0.0965	4.9005
12	1.127	0.8875	12.683	0.0789	11.2551	0.0889	5.3815
13	1.138	0.8787	13.809	0.0724	12.1338	0.0824	5.8607
14	1.149	0.8700	14.947	0.0669	13.0037	0.0769	6.3384
15	1.161	0.8614	16.097	0.0621	13.8651	0.0721	6.8143
16	1.173	0.8528	17.258	0.0580	14.7179	0.0680	7.2887
17	1.184	0.8444	18.430	0.0543	15.5623	0.0643	7.7613
18	1.196	0.8360	19.615	0.0510	16.3983	0.0610	8.2323
19	1.208	0.8277	20.811	0.0481	17.2260	0.0581	8.7017
20	1.220	0.8196	22.019	0.0454	18.0456	0.0554	9.1694
21	1.232	0.8114	23.239	0.0430	18.8570	0.0530	9.6354
22	1.245	0.8034	24.472	0.0409	19.6604	0.0509	10.0998
23	1.257	0.7955	25.716	0.0389	20.4558	0.0489	10.5626
24	1.270	0.7876	26.973	0.0371	21.2434	0.0471	11.0237
25	1.282	0.7798	28.243	0.0354	22.0232	0.0454	11.4831
26	1.295	0.7721	29.526	0.0339	22.7952	0.0439	11.9409
27	1.308	0.7644	30.821	0.0325	23.5596	0.0425	12.3971
28	1.321	0.7568	32.129	0.0311	24.3165	0.0411	12.8516
29	1.335	0.7494	33.450	0.0299	25.0658	0.0399	13.3045
30	1.348	0.7419	34.785	0.0288	25.8077	0.0388	13.7557
31	1.361	0.7346	36.133	0.0277	26.5423	0.0377	14.2052
32	1.375	0.7273	37.494	0.0267	27.2696	0.0367	14.6532
33	1.389	0.7201	38.869	0.0257	27.9897	0.0357	15.0995
34	1.403	0.7130	40.258	0.0248	28.7027	0.0348	15.5441
35	1.417	0.7059	41.660	0.0240	29.4086	0.0340	15.9871
40	1.489	0.6717	48.886	0.0205	32.8347	0.0305	18.1776
45	1.565	0.6391	56.481	0.0177	36.0945	0.0277	20.3273
50	1.645	0.6080	64.463	0.0155	39.1961	0.0255	22.4363
55	1.729	0.5785	72.852	0.0137	42.1472	0.0237	24.5049
60	1.817	0.5505	81.670	0.0123	44.9550	0.0223	26.5333
65	1.909	0.5237	90.937	0.0110	47.6266	0.0210	28.5217
70	2.007	0.4983	100.676	0.0099	50.1685	0.0199	30.4703
75	2.109	0.4741	110.913	0.0090	52.5871	0.0190	32.3793
80	2.217	0.4511	121.672	0.0082	54.8882	0.0182	34.2492
85	2.330	0.4292	132.979	0.0075	57.0777	0.0175	36.0801
90	2.449	0.4084	144.863	0.0069	59.1609	0.0169	37.8725
95	2.574	0.3886	157.354	0.0064	61.1430	0.0164	39.6265
100	2.705	0.3697	170.481	0.0059	63.0289	0.0159	41.3426

1 1/4% Interest Factors

n	Single Payment: Compound-amount factor (To find F Given P, F/P i,n)	Single Payment: Present-worth factor (To find P Given F, P/F i,n)	Equal Payment Series: Compound-amount factor (To find F Given A, F/A i,n)	Equal Payment Series: Sinking-fund factor (To find A Given F, A/F i,n)	Equal Payment Series: Present-worth factor (To find P Given A, P/A i,n)	Equal Payment Series: Capital-recovery factor (To find A Given P, A/P i,n)	Uniform gradient-series factor (To find A Given G, A/G i,n)
1	1.013	0.9877	1.000	1.0001	0.9877	1.0126	0.0000
2	1.025	0.9755	2.013	0.4970	1.9631	0.5095	0.4932
3	1.038	0.9635	3.038	0.3293	2.9265	0.3418	0.9895
4	1.051	0.9516	4.076	0.2454	3.8780	0.2579	1.4830
5	1.064	0.9398	5.127	0.1951	4.8177	0.2076	1.9729
6	1.077	0.9282	6.191	0.1616	5.7459	0.1741	2.4618
7	1.091	0.9168	7.268	0.1376	6.6627	0.1501	2.9491
8	1.105	0.9055	8.359	0.1197	7.5680	0.1322	3.4330
9	1.118	0.8943	9.463	0.1057	8.4623	0.1182	3.9158
10	1.132	0.8832	10.582	0.0946	9.3454	0.1071	4.3960
11	1.147	0.8723	11.714	0.0854	10.2177	0.0979	4.8744
12	1.161	0.8616	12.860	0.0778	11.0792	0.0903	5.3506
13	1.175	0.8509	14.021	0.0714	11.9300	0.0839	5.8248
14	1.190	0.8404	15.196	0.0659	12.7704	0.0784	6.2968
15	1.205	0.8300	16.386	0.0611	13.6004	0.0736	6.7669
16	1.220	0.8198	17.591	0.0569	14.4201	0.0694	7.2350
17	1.235	0.8097	18.811	0.0532	15.2298	0.0657	7.7009
18	1.251	0.7997	20.046	0.0499	16.0293	0.0624	8.1645
19	1.266	0.7898	21.296	0.0470	16.8191	0.0595	8.6264
20	1.282	0.7801	22.563	0.0444	17.5991	0.0569	9.0861
21	1.298	0.7704	23.845	0.0420	18.3695	0.0545	9.5439
22	1.314	0.7609	25.143	0.0398	19.1303	0.0523	9.9993
23	1.331	0.7515	26.457	0.0378	19.8818	0.0503	10.4528
24	1.347	0.7423	27.788	0.0360	20.6240	0.0485	10.9044
25	1.364	0.7331	29.135	0.0344	21.3570	0.0469	11.3539
26	1.381	0.7240	30.499	0.0328	22.0810	0.0453	11.8012
27	1.399	0.7151	31.880	0.0314	22.7960	0.0439	12.2465
28	1.416	0.7063	33.279	0.0301	23.5022	0.0426	12.6898
29	1.434	0.6976	34.695	0.0289	24.1998	0.0414	13.1311
30	1.452	0.6889	36.128	0.0277	24.8886	0.0402	13.5703
31	1.470	0.6804	37.580	0.0267	25.5690	0.0392	14.0074
32	1.488	0.6720	39.050	0.0257	26.2410	0.0382	14.4425
33	1.507	0.6637	40.538	0.0247	26.9047	0.0372	14.8756
34	1.526	0.6555	42.045	0.0238	27.5601	0.0363	15.3066
35	1.545	0.6475	43.570	0.0230	28.2075	0.0355	15.7357
40	1.644	0.6085	51.489	0.0195	31.3266	0.0320	17.8503
45	1.749	0.5718	59.915	0.0167	34.2578	0.0292	19.9144
50	1.861	0.5374	68.880	0.0146	37.0125	0.0271	21.9284
55	1.980	0.5050	78.421	0.0128	39.6013	0.0253	23.8925
60	2.107	0.4746	88.573	0.0113	42.0342	0.0238	25.8072
65	2.242	0.4460	99.375	0.0101	44.3206	0.0226	27.6730
70	2.386	0.4192	110.870	0.0091	46.4693	0.0216	29.4902
75	2.539	0.3939	123.101	0.0082	48.4886	0.0207	31.2594
80	2.702	0.3702	136.116	0.0074	50.3862	0.0199	32.9812
85	2.875	0.3479	149.965	0.0067	52.1696	0.0192	34.6560
90	3.059	0.3270	164.701	0.0061	53.8456	0.0186	36.2844
95	3.255	0.3073	180.382	0.0056	55.4207	0.0181	37.8671
100	3.463	0.2888	197.067	0.0051	56.9009	0.0176	39.4048

1 1/2% Interest Factors

n	Single Payment: Compound-amount factor	Single Payment: Present-worth factor	Equal Payment Series: Compound-amount factor	Equal Payment Series: Sinking-fund factor	Equal Payment Series: Present-worth factor	Equal Payment Series: Capital-recovery factor	Uniform gradient-series factor
	To find F Given P F/P i,n	To find P Given F P/F i,n	To find F Given A F/A i,n	To find A Given F A/F i,n	To find P Given A P/A i,n	To find A Given P A/P i,n	To find A Given G A/G i,n
1	1.015	0.9852	1.000	1.0000	0.9852	1.0150	0.0000
2	1.030	0.9707	2.015	0.4963	1.9559	0.5113	0.4963
3	1.046	0.9563	3.045	0.3284	2.9122	0.3434	0.9901
4	1.061	0.9422	4.091	0.2445	3.8544	0.2595	1.4814
5	1.077	0.9283	5.152	0.1941	4.7827	0.2091	1.9702
6	1.093	0.9146	6.230	0.1605	5.6972	0.1755	2.4566
7	1.110	0.9010	7.323	0.1366	6.5982	0.1516	2.9405
8	1.127	0.8877	8.433	0.1186	7.4859	0.1336	3.4219
9	1.143	0.8746	9.559	0.1046	8.3605	0.1196	3.9008
10	1.161	0.8617	10.703	0.0934	9.2222	0.1084	4.3772
11	1.178	0.8489	11.863	0.0843	10.0711	0.0993	4.8512
12	1.196	0.8364	13.041	0.0767	10.9075	0.0917	5.3227
13	1.214	0.8240	14.237	0.0703	11.7315	0.0853	5.7917
14	1.232	0.8119	15.450	0.0647	12.5434	0.0797	6.2582
15	1.250	0.7999	16.682	0.0600	13.3432	0.0750	6.7223
16	1.269	0.7880	17.932	0.0558	14.1313	0.0708	7.1839
17	1.288	0.7764	19.201	0.0521	14.9077	0.0671	7.6431
18	1.307	0.7649	20.489	0.0488	15.6726	0.0638	8.0997
19	1.327	0.7536	21.797	0.0459	16.4262	0.0609	8.5539
20	1.347	0.7425	23.124	0.0433	17.1686	0.0583	9.0057
21	1.367	0.7315	24.471	0.0409	17.9001	0.0559	9.4550
22	1.388	0.7207	25.838	0.0387	18.6208	0.0537	9.9018
23	1.408	0.7100	27.225	0.0367	19.3309	0.0517	10.3462
24	1.430	0.6996	28.634	0.0349	20.0304	0.0499	10.7881
25	1.451	0.6892	30.063	0.0333	20.7196	0.0483	11.2276
26	1.473	0.6790	31.514	0.0317	21.3986	0.0467	11.6646
27	1.495	0.6690	32.987	0.0303	22.0676	0.0453	12.0992
28	1.517	0.6591	34.481	0.0290	22.7267	0.0440	12.5313
29	1.540	0.6494	35.999	0.0278	23.3761	0.0428	12.9610
30	1.563	0.6398	37.539	0.0266	24.0158	0.0416	13.3883
31	1.587	0.6303	39.102	0.0256	24.6462	0.0406	13.8131
32	1.610	0.6210	40.688	0.0246	25.2671	0.0396	14.2355
33	1.634	0.6118	42.299	0.0237	25.8790	0.0387	14.6555
34	1.659	0.6028	43.933	0.0228	26.4817	0.0378	15.0731
35	1.684	0.5939	45.592	0.0219	27.0756	0.0369	15.4882
40	1.814	0.5513	54.268	0.0184	29.9159	0.0334	17.5277
45	1.954	0.5117	63.614	0.0157	32.5523	0.0307	19.5074
50	2.105	0.4750	73.683	0.0136	34.9997	0.0286	21.4277
55	2.268	0.4409	84.530	0.0118	37.2715	0.0268	23.2894
60	2.443	0.4093	96.215	0.0104	39.3803	0.0254	25.0930
65	2.632	0.3799	108.803	0.0092	41.3378	0.0242	26.8392
70	2.835	0.3527	122.364	0.0082	43.1549	0.0232	28.5290
75	3.055	0.3274	136.973	0.0073	44.8416	0.0223	30.1631
80	3.291	0.3039	152.711	0.0066	46.4073	0.0216	31.7423
85	3.545	0.2821	169.665	0.0059	47.8607	0.0209	33.2676
90	3.819	0.2619	187.930	0.0053	49.2099	0.0203	34.7399
95	4.114	0.2431	207.606	0.0048	50.4622	0.0198	36.1602
100	4.432	0.2256	228.803	0.0044	51.6247	0.0194	37.5295

2% Interest Factors

n	Single Payment: Compound-amount factor	Single Payment: Present-worth factor	Equal Payment Series: Compound-amount factor	Equal Payment Series: Sinking-fund factor	Equal Payment Series: Present-worth factor	Equal Payment Series: Capital-recovery factor	Uniform gradient-series factor
	To find F Given P F/P i,n	To find P Given F P/F i,n	To find F Given A F/A i,n	To find A Given F A/F i,n	To find P Given A P/A i,n	To find A Given P A/P i,n	To find A Given G A/G i,n
1	1.020	0.9804	1.000	1.0000	0.9804	1.0200	0.0000
2	1.040	0.9612	2.020	0.4951	1.9416	0.5151	0.4951
3	1.061	0.9423	3.060	0.3268	2.8839	0.3468	0.9868
4	1.082	0.9239	4.122	0.2426	3.8077	0.2626	1.4753
5	1.104	0.9057	5.204	0.1922	4.7135	0.2122	1.9604
6	1.126	0.8880	6.308	0.1585	5.6014	0.1785	2.4423
7	1.149	0.8706	7.434	0.1345	6.4720	0.1545	2.9208
8	1.172	0.8535	8.583	0.1165	7.3255	0.1365	3.3961
9	1.195	0.8368	9.755	0.1025	8.1622	0.1225	3.8681
10	1.219	0.8204	10.950	0.0913	8.9826	0.1113	4.3367
11	1.243	0.8043	12.169	0.0822	9.7869	0.1022	4.8021
12	1.268	0.7885	13.412	0.0746	10.5754	0.0946	5.2643
13	1.294	0.7730	14.680	0.0681	11.3484	0.0881	5.7231
14	1.319	0.7579	15.974	0.0626	12.1063	0.0826	6.1786
15	1.346	0.7430	17.293	0.0578	12.8493	0.0778	6.6309
16	1.373	0.7285	18.639	0.0537	13.5777	0.0737	7.0799
17	1.400	0.7142	20.012	0.0500	14.2919	0.0700	7.5256
18	1.428	0.7002	21.412	0.0467	14.9920	0.0667	7.9681
19	1.457	0.6864	22.841	0.0438	15.6785	0.0638	8.4073
20	1.486	0.6730	24.297	0.0412	16.3514	0.0612	8.8433
21	1.516	0.6598	25.783	0.0388	17.0112	0.0588	9.2760
22	1.546	0.6468	27.299	0.0366	17.6581	0.0566	9.7055
23	1.577	0.6342	28.845	0.0347	18.2922	0.0547	10.1317
24	1.608	0.6217	30.422	0.0329	18.9139	0.0529	10.5547
25	1.641	0.6095	32.030	0.0312	19.5235	0.0512	10.9745
26	1.673	0.5976	33.671	0.0297	20.1210	0.0497	11.3910
27	1.707	0.5859	35.344	0.0283	20.7069	0.0483	11.8043
28	1.741	0.5744	37.051	0.0270	21.2813	0.0470	12.2145
29	1.776	0.5631	38.792	0.0258	21.8444	0.0458	12.6214
30	1.811	0.5521	40.568	0.0247	22.3965	0.0447	13.0251
31	1.848	0.5413	42.379	0.0236	22.9377	0.0436	13.4257
32	1.885	0.5306	44.227	0.0226	23.4683	0.0426	13.8230
33	1.922	0.5202	46.112	0.0217	23.9886	0.0417	14.2172
34	1.961	0.5100	48.034	0.0208	24.4986	0.0408	14.6083
35	2.000	0.5000	49.994	0.0200	24.9986	0.0400	14.9961
40	2.208	0.4529	60.402	0.0166	27.3555	0.0366	16.8885
45	2.438	0.4102	71.893	0.0139	29.4902	0.0339	18.7034
50	2.692	0.3715	84.579	0.0118	31.4236	0.0318	20.4420
55	2.972	0.3365	98.587	0.0102	33.1748	0.0302	22.1057
60	3.281	0.3048	114.052	0.0088	34.7609	0.0288	23.6961
65	3.623	0.2761	131.126	0.0076	36.1975	0.0276	25.2147
70	4.000	0.2500	149.978	0.0067	37.4986	0.0267	26.6632
75	4.416	0.2265	170.792	0.0059	38.6771	0.0259	28.0434
80	4.875	0.2051	193.772	0.0052	39.7445	0.0252	29.3572
85	5.383	0.1858	219.144	0.0046	40.7113	0.0246	30.6064
90	5.943	0.1683	247.157	0.0041	41.5869	0.0241	31.7929
95	6.562	0.1524	278.085	0.0036	42.3800	0.0236	32.9189
100	7.245	0.1380	312.232	0.0032	43.0984	0.0232	33.9863

3% Interest Factors

n	Single Payment: Compound-amount factor	Single Payment: Present-worth factor	Equal Payment Series: Compound-amount factor	Equal Payment Series: Sinking-fund factor	Equal Payment Series: Present-worth factor	Equal Payment Series: Capital-recovery factor	Uniform gradient-series factor
	To find F Given P F/P i,n	To find P Given F P/F i,n	To find F Given A F/A i,n	To find A Given F A/F i,n	To find P Given A P/A i,n	To find A Given P A/P i,n	To find A Given G A/G i,n
1	1.030	0.9709	1.000	1.0000	0.9709	1.0300	0.0000
2	1.061	0.9426	2.030	0.4926	1.9135	0.5226	0.4926
3	1.093	0.9152	3.091	0.3235	2.8286	0.3535	0.9803
4	1.126	0.8885	4.184	0.2390	3.7171	0.2690	1.4631
5	1.159	0.8626	5.309	0.1884	4.5797	0.2184	1.9409
6	1.194	0.8375	6.468	0.1546	5.4172	0.1846	2.4138
7	1.230	0.8131	7.662	0.1305	6.2303	0.1605	2.8819
8	1.267	0.7894	8.892	0.1125	7.0197	0.1425	3.3450
9	1.305	0.7664	10.159	0.0984	7.7861	0.1284	3.8032
10	1.344	0.7441	11.464	0.0872	8.5302	0.1172	4.2565
11	1.384	0.7224	12.808	0.0781	9.2526	0.1081	4.7049
12	1.426	0.7014	14.192	0.0705	9.9540	0.1005	5.1485
13	1.469	0.6810	15.618	0.0640	10.6350	0.0940	5.5872
14	1.513	0.6611	17.086	0.0585	11.2961	0.0885	6.0211
15	1.558	0.6419	18.599	0.0538	11.9379	0.0838	6.4501
16	1.605	0.6232	20.157	0.0496	12.5611	0.0796	6.8742
17	1.653	0.6050	21.762	0.0460	13.1661	0.0760	7.2936
18	1.702	0.5874	23.414	0.0427	13.7535	0.0727	7.7081
19	1.754	0.5703	25.117	0.0398	14.3238	0.0698	8.1179
20	1.806	0.5537	26.870	0.0372	14.8775	0.0672	8.5229
21	1.860	0.5376	28.676	0.0349	15.4150	0.0649	8.9231
22	1.916	0.5219	30.537	0.0328	15.9369	0.0628	9.3186
23	1.974	0.5067	32.453	0.0308	16.4436	0.0608	9.7094
24	2.033	0.4919	34.426	0.0291	16.9356	0.0591	10.0954
25	2.094	0.4776	36.459	0.0274	17.4132	0.0574	10.4768
26	2.157	0.4637	38.553	0.0259	17.8769	0.0559	10.8535
27	2.221	0.4502	40.710	0.0246	18.3270	0.0546	11.2256
28	2.288	0.4371	42.931	0.0233	18.7641	0.0533	11.5930
29	2.357	0.4244	45.219	0.0221	19.1885	0.0521	11.9558
30	2.427	0.4120	47.575	0.0210	19.6005	0.0510	12.3141
31	2.500	0.4000	50.003	0.0200	20.0004	0.0500	12.6678
32	2.575	0.3883	52.503	0.0191	20.3888	0.0491	13.0169
33	2.652	0.3770	55.078	0.0182	20.7658	0.0482	13.3616
34	2.732	0.3661	57.730	0.0173	21.1318	0.0473	13.7018
35	2.814	0.3554	60.462	0.0165	21.4872	0.0465	14.0375
40	3.262	0.3066	75.401	0.0133	23.1148	0.0433	15.6502
45	3.782	0.2644	92.720	0.0108	24.5187	0.0408	17.1556
50	4.384	0.2281	112.797	0.0089	25.7298	0.0389	18.5575
55	5.082	0.1968	136.072	0.0074	26.7744	0.0374	19.8600
60	5.892	0.1697	163.053	0.0061	27.6756	0.0361	21.0674
65	6.830	0.1464	194.333	0.0052	28.4529	0.0352	22.1841
70	7.918	0.1263	230.594	0.0043	29.1234	0.0343	23.2145
75	9.179	0.1090	272.631	0.0037	29.7018	0.0337	24.1634
80	10.641	0.0940	321.363	0.0031	30.2008	0.0331	25.0354
85	12.336	0.0811	377.857	0.0027	30.6312	0.0327	25.8349
90	14.300	0.0699	443.349	0.0023	31.0024	0.0323	26.5667
95	16.578	0.0603	519.272	0.0019	31.3227	0.0319	27.2351
100	19.219	0.0520	607.288	0.0017	31.5989	0.0317	27.8445

4% Interest Factors

n	Single Payment: Compound-amount factor	Single Payment: Present-worth factor	Equal Payment Series: Compound-amount factor	Equal Payment Series: Sinking-fund factor	Equal Payment Series: Present-worth factor	Equal Payment Series: Capital-recovery factor	Uniform gradient-series factor
	To find F Given P F/P i,n	To find P Given F P/F i,n	To find F Given A F/A i,n	To find A Given F A/F i,n	To find P Given A P/A i,n	To find A Given P A/P i,n	To find A Given G A/G i,n
1	1.040	0.9615	1.000	1.0000	0.9615	1.0400	0.0000
2	1.082	0.9246	2.040	0.4902	1.8861	0.5302	0.4902
3	1.125	0.8890	3.122	0.3204	2.7751	0.3604	0.9739
4	1.170	0.8548	4.246	0.2355	3.6299	0.2755	1.4510
5	1.217	0.8219	5.416	0.1846	4.4518	0.2246	1.9216
6	1.265	0.7903	6.633	0.1508	5.2421	0.1908	2.3857
7	1.316	0.7599	7.898	0.1266	6.0021	0.1666	2.8433
8	1.369	0.7307	9.214	0.1085	6.7328	0.1485	3.2944
9	1.423	0.7026	10.583	0.0945	7.4353	0.1345	3.7391
10	1.480	0.6756	12.006	0.0833	8.1109	0.1233	4.1773
11	1.539	0.6496	13.486	0.0742	8.7605	0.1142	4.6090
12	1.601	0.6246	15.026	0.0666	9.3851	0.1066	5.0344
13	1.665	0.6006	16.627	0.0602	9.9857	0.1002	5.4533
14	1.732	0.5775	18.292	0.0547	10.5631	0.0947	5.8659
15	1.801	0.5553	20.024	0.0500	11.1184	0.0900	6.2721
16	1.873	0.5339	21.825	0.0458	11.6523	0.0858	6.6720
17	1.948	0.5134	23.698	0.0422	12.1657	0.0822	7.0656
18	2.026	0.4936	25.645	0.0390	12.6593	0.0790	7.4530
19	2.107	0.4747	27.671	0.0361	13.1339	0.0761	7.8342
20	2.191	0.4564	29.778	0.0336	13.5903	0.0736	8.2091
21	2.279	0.4388	31.969	0.0313	14.0292	0.0713	8.5780
22	2.370	0.4220	34.248	0.0292	14.4511	0.0692	8.9407
23	2.465	0.4057	36.618	0.0273	14.8569	0.0673	9.2973
24	2.563	0.3901	39.083	0.0256	15.2470	0.0656	9.6479
25	2.666	0.3751	41.646	0.0240	15.6221	0.0640	9.9925
26	2.772	0.3607	44.312	0.0226	15.9828	0.0626	10.3312
27	2.883	0.3468	47.084	0.0212	16.3296	0.0612	10.6640
28	2.999	0.3335	49.968	0.0200	16.6631	0.0600	10.9909
29	3.119	0.3207	52.966	0.0189	16.9837	0.0589	11.3121
30	3.243	0.3083	56.085	0.0178	17.2920	0.0578	11.6274
31	3.373	0.2965	59.328	0.0169	17.5885	0.0569	11.9371
32	3.508	0.2851	62.701	0.0160	17.8736	0.0560	12.2411
33	3.648	0.2741	66.210	0.0151	18.1477	0.0551	12.5396
34	3.794	0.2636	69.858	0.0143	18.4112	0.0543	12.8325
35	3.946	0.2534	73.652	0.0136	18.6646	0.0536	13.1199
40	4.801	0.2083	95.026	0.0105	19.7928	0.0505	14.4765
45	5.841	0.1712	121.029	0.0083	20.7200	0.0483	15.7047
50	7.107	0.1407	152.667	0.0066	21.4822	0.0466	16.8123
55	8.646	0.1157	191.159	0.0052	22.1086	0.0452	17.8070
60	10.520	0.0951	237.991	0.0042	22.6235	0.0442	18.6972
65	12.799	0.0781	294.968	0.0034	23.0467	0.0434	19.4909
70	15.572	0.0642	364.290	0.0028	23.3945	0.0428	20.1961
75	18.945	0.0528	448.631	0.0022	23.6804	0.0422	20.8206
80	23.050	0.0434	551.245	0.0018	23.9154	0.0418	21.3719
85	28.044	0.0357	676.090	0.0015	24.1085	0.0415	21.8569
90	34.119	0.0293	817.983	0.0012	24.2673	0.0412	22.2826
95	41.511	0.0241	1012.785	0.0010	24.3978	0.0410	22.6550
100	50.505	0.0198	1237.624	0.0008	24.5050	0.0408	22.9800

5% Interest Factors

n	Single Payment: Compound-amount factor (To find F Given P, F/P i,n)	Single Payment: Present-worth factor (To find P Given F, P/F i,n)	Equal Payment Series: Compound-amount factor (To find F Given A, F/A i,n)	Equal Payment Series: Sinking-fund factor (To find A Given F, A/F i,n)	Equal Payment Series: Present-worth factor (To find P Given A, P/A i,n)	Equal Payment Series: Capital-recovery factor (To find A Given P, A/P i,n)	Uniform gradient-series factor (To find A Given G, A/G i,n)
1	1.050	0.9524	1.000	1.0000	0.9524	1.0500	0.0000
2	1.103	0.9070	2.050	0.4878	1.8594	0.5378	0.4878
3	1.158	0.8638	3.153	0.3172	2.7233	0.3672	0.9675
4	1.216	0.8227	4.310	0.2320	3.5460	0.2820	1.4391
5	1.276	0.7835	5.526	0.1810	4.3295	0.2310	1.9025
6	1.340	0.7462	6.802	0.1470	5.0757	0.1970	2.3579
7	1.407	0.7107	8.142	0.1228	5.7864	0.1728	2.8052
8	1.477	0.6768	9.549	0.1047	6.4632	0.1547	3.2445
9	1.551	0.6446	11.027	0.0907	7.1078	0.1407	3.6758
10	1.629	0.6139	12.587	0.0795	7.7217	0.1295	4.0991
11	1.710	0.5847	14.207	0.0704	8.3064	0.1204	4.5145
12	1.796	0.5568	15.917	0.0628	8.8633	0.1128	4.9219
13	1.886	0.5303	17.713	0.0565	9.3936	0.1065	5.3215
14	1.980	0.5051	19.599	0.0510	9.8987	0.1010	5.7133
15	2.079	0.4810	21.579	0.0464	10.3797	0.0964	6.0973
16	2.183	0.4581	23.658	0.0423	10.8378	0.0923	6.4736
17	2.292	0.4363	25.840	0.0387	11.2741	0.0887	6.8423
18	2.407	0.4155	28.132	0.0356	11.6896	0.0856	7.2034
19	2.527	0.3957	30.539	0.0328	12.0853	0.0828	7.5569
20	2.653	0.3769	33.066	0.0303	12.4622	0.0803	7.9030
21	2.786	0.3590	35.719	0.0280	12.8212	0.0780	8.2416
22	2.925	0.3419	38.505	0.0260	13.1630	0.0760	8.5730
23	3.072	0.3256	41.430	0.0241	13.4886	0.0741	8.8971
24	3.225	0.3101	44.502	0.0225	13.7987	0.0725	9.2140
25	3.386	0.2953	47.727	0.0210	14.0940	0.0710	9.5238
26	3.556	0.2813	51.113	0.0196	14.3752	0.0696	9.8266
27	3.733	0.2679	54.669	0.0183	14.6430	0.0683	10.1224
28	3.920	0.2551	58.403	0.0171	14.8981	0.0671	10.4114
29	4.116	0.2430	62.323	0.0161	15.1411	0.0661	10.6936
30	4.322	0.2314	66.439	0.0151	15.3725	0.0651	10.9691
31	4.538	0.2204	70.761	0.0141	15.5928	0.0641	11.2381
32	4.765	0.2099	75.299	0.0133	15.8027	0.0633	11.5005
33	5.003	0.1999	80.064	0.0125	16.0026	0.0625	11.7566
34	5.253	0.1904	85.067	0.0118	16.1929	0.0618	12.0063
35	5.516	0.1813	90.320	0.0111	16.3742	0.0611	12.2498
40	7.040	0.1421	120.800	0.0083	17.1591	0.0583	13.3775
45	8.985	0.1113	159.700	0.0063	17.7741	0.0563	14.3644
50	11.467	0.0872	209.348	0.0048	18.2559	0.0548	15.2233
55	14.636	0.0683	272.713	0.0037	18.6335	0.0537	15.9665
60	18.679	0.0535	353.584	0.0028	18.9293	0.0528	16.6062
65	23.840	0.0420	456.798	0.0022	19.1611	0.0522	17.1541
70	30.426	0.0329	588.529	0.0017	19.3427	0.0517	17.6212
75	38.833	0.0258	756.654	0.0013	19.4850	0.0513	18.0176
80	49.561	0.0202	971.229	0.0010	19.5965	0.0510	18.3526
85	63.254	0.0158	1245.087	0.0008	19.6838	0.0508	18.6346
90	80.730	0.0124	1594.607	0.0006	19.7523	0.0506	18.8712
95	103.035	0.0097	2040.694	0.0005	19.8059	0.0505	19.0689
100	131.501	0.0076	2610.025	0.0004	19.8479	0.0504	19.2337

6% Interest Factors

n	Single Payment: Compound-amount factor (To find F Given P, F/P i,n)	Single Payment: Present-worth factor (To find P Given F, P/F i,n)	Equal Payment Series: Compound-amount factor (To find F Given A, F/A i,n)	Equal Payment Series: Sinking-fund factor (To find A Given F, A/F i,n)	Equal Payment Series: Present-worth factor (To find P Given A, P/A i,n)	Equal Payment Series: Capital-recovery factor (To find A Given P, A/P i,n)	Uniform gradient-series factor (To find A Given G, A/G i,n)
1	1.060	0.9434	1.000	1.0000	0.9434	1.0600	0.0000
2	1.124	0.8900	2.060	0.4854	1.8334	0.5454	0.4854
3	1.191	0.8396	3.184	0.3141	2.6730	0.3741	0.9612
4	1.262	0.7921	4.375	0.2286	3.4651	0.2886	1.4272
5	1.338	0.7473	5.637	0.1774	4.2124	0.2374	1.8836
6	1.419	0.7050	6.975	0.1434	4.9173	0.2034	2.3304
7	1.504	0.6651	8.394	0.1191	5.5824	0.1791	2.7676
8	1.594	0.6274	9.897	0.1010	6.2098	0.1610	3.1952
9	1.689	0.5919	11.491	0.0870	6.8017	0.1470	3.6133
10	1.791	0.5584	13.181	0.0759	7.3601	0.1359	4.0220
11	1.898	0.5268	14.972	0.0668	7.8869	0.1268	4.4213
12	2.012	0.4970	16.870	0.0593	8.3839	0.1193	4.8113
13	2.133	0.4688	18.882	0.0530	8.8527	0.1130	5.1920
14	2.261	0.4423	21.015	0.0476	9.2950	0.1076	5.5635
15	2.397	0.4173	23.276	0.0430	9.7123	0.1030	5.9260
16	2.540	0.3937	25.673	0.0390	10.1059	0.0990	6.2794
17	2.693	0.3714	28.213	0.0355	10.4773	0.0955	6.6240
18	2.854	0.3504	30.906	0.0324	10.8276	0.0924	6.9597
19	3.026	0.3305	33.760	0.0296	11.1581	0.0896	7.2867
20	3.207	0.3118	36.786	0.0272	11.4699	0.0872	7.6052
21	3.400	0.2942	39.993	0.0250	11.7641	0.8520	7.9151
22	3.604	0.2775	43.392	0.0231	12.0416	0.0831	8.2166
23	3.820	0.2618	46.996	0.0213	12.3034	0.0813	8.5099
24	4.049	0.2470	50.816	0.0197	12.5504	0.0797	8.7951
25	4.292	0.2330	54.865	0.0182	12.7834	0.0782	9.0722
26	4.549	0.2198	59.156	0.0169	13.0032	0.0769	9.3415
27	4.822	0.2074	63.706	0.0157	13.2105	0.0757	9.6030
28	5.112	0.1956	68.528	0.0146	13.4062	0.0746	9.8568
29	5.418	0.1846	73.640	0.0136	13.5907	0.0736	10.1032
30	5.744	0.1741	79.058	0.0127	13.7648	0.0727	10.3422
31	6.088	0.1643	84.802	0.0118	13.9291	0.0718	10.5740
32	6.453	0.1550	90.890	0.0110	14.0841	0.0710	10.7988
33	6.841	0.1462	97.343	0.0103	14.2302	0.0703	11.0166
34	7.251	0.1379	104.184	0.0096	14.3682	0.0696	11.2276
35	7.686	0.1301	111.435	0.0090	14.4983	0.0690	11.4319
40	10.286	0.0972	154.762	0.0065	15.0463	0.0665	12.3590
45	13.765	0.0727	212.744	0.0047	15.4558	0.0647	13.1413
50	18.420	0.0543	290.336	0.0035	15.7619	0.0635	13.7964
55	24.650	0.0406	394.172	0.0025	15.9906	0.0625	14.3411
60	32.988	0.0303	533.128	0.0019	16.1614	0.0619	14.7910
65	44.145	0.0277	719.083	0.0014	16.2891	0.0614	15.1601
70	59.076	0.0169	967.932	0.0010	16.3846	0.0610	15.4614
75	79.057	0.0127	1300.949	0.0008	16.4559	0.0608	15.7058
80	105.796	0.0095	1746.600	0.0006	16.5091	0.0606	15.9033
85	141.579	0.0071	2342.982	0.0004	16.5490	0.0604	16.0620
90	189.465	0.0053	3141.075	0.0003	16.5787	0.0603	16.1891
95	253.546	0.0040	4209.104	0.0002	16.6009	0.0602	16.2905
100	339.302	0.0030	5638.368	0.0002	16.6176	0.0602	16.3711

7% Interest Factors

n	Single Payment		Equal Payment Series				Uniform gradient-series factor
	Compound-amount factor	Present-worth factor	Compound-amount factor	Sinking-fund factor	Present-worth factor	Capital-recovery factor	
	To find F Given P F/P i,n	To find P Given F P/F i,n	To find F Given A F/A i,n	To find A Given F A/F i,n	To find P Given A P/A i,n	To find A Given P A/P i,n	To find A Given G A/G i,n
1	1.070	0.9346	1.000	1.0000	0.9346	1.0700	0.0000
2	1.145	0.8734	2.070	0.4831	1.8080	0.5531	0.4831
3	1.225	0.8163	3.215	0.3111	2.6243	0.3811	0.9549
4	1.311	0.7629	4.440	0.2252	3.3872	0.2952	1.4155
5	1.403	0.7130	5.751	0.1739	4.1002	0.2439	1.8550
6	1.501	0.6664	7.153	0.1398	4.7665	0.2098	2.3032
7	1.606	0.6228	8.654	0.1156	5.3893	0.1856	2.7304
8	1.718	0.5820	10.260	0.0975	5.9713	0.1675	3.1466
9	1.838	0.5439	11.978	0.0835	6.5152	0.1535	3.5517
10	1.967	0.5084	13.816	0.0724	7.0236	0.1424	3.9461
11	2.105	0.4751	15.784	0.0634	7.4987	0.1334	4.3296
12	2.252	0.4440	17.888	0.0559	7.9427	0.1259	4.7025
13	2.410	0.4150	20.141	0.0497	8.3577	0.1197	5.0649
14	2.579	0.3878	22.550	0.0444	8.7455	0.1144	5.4167
15	2.759	0.3625	25.129	0.0398	9.1079	0.1098	5.7583
16	2.952	0.3387	27.888	0.0359	9.4467	0.1059	6.0897
17	3.159	0.3166	30.840	0.0324	9.7632	0.1024	6.4110
18	3.380	0.2959	33.999	0.0294	10.0591	0.0994	6.7225
19	3.617	0.2765	37.379	0.0268	10.3356	0.0968	7.0242
20	3.870	0.2584	40.996	0.0244	10.5940	0.0944	7.3163
21	4.141	0.2415	44.865	0.0223	10.8355	0.0923	7.5990
22	4.430	0.2257	49.006	0.0204	11.0613	0.0904	7.8725
23	4.741	0.2110	53.436	0.0187	11.2722	0.0887	8.1369
24	5.072	0.1972	58.177	0.0172	11.4693	0.0872	8.3923
25	5.427	0.1843	63.249	0.0158	11.6536	0.0858	8.6391
26	5.807	0.1722	68.676	0.0146	11.8258	0.0846	8.8773
27	6.214	0.1609	74.484	0.0134	11.9867	0.0834	9.1072
28	6.649	0.1504	80.698	0.0124	12.1371	0.0824	9.3290
29	7.114	0.1406	87.347	0.0115	12.2777	0.0815	9.4527
30	7.612	0.1314	94.461	0.0106	12.4091	0.0806	9.7487
31	8.145	0.1228	102.073	0.0098	12.5318	0.0798	9.9471
32	8.715	0.1148	110.218	0.0091	12.6466	0.0791	10.1381
33	9.325	0.1072	118.933	0.0084	12.7538	0.0784	10.3219
34	9.978	0.1002	128.259	0.0078	12.8540	0.0778	10.4987
35	10.677	0.0937	138.237	0.0072	12.9477	0.0772	10.6687
40	14.974	0.0668	199.635	0.0050	13.3317	0.0750	11.4234
45	21.002	0.0476	285.749	0.0035	13.6055	0.0735	12.0360
50	29.457	0.0340	406.529	0.0025	13.8008	0.0725	12.5287
55	41.315	0.0242	575.929	0.0017	13.9399	0.0717	12.9215
60	57.946	0.0173	813.520	0.0012	14.0392	0.0712	13.2321
65	81.273	0.0123	1146.755	0.0009	14.1099	0.0709	13.4760
70	113.989	0.0088	1614.134	0.0006	14.1604	0.0706	13.6662
75	159.876	0.0063	2269.657	0.0005	14.1964	0.0705	13.8137
80	224.234	0.0045	3189.063	0.0003	14.2220	0.0703	13.9274
85	314.500	0.0032	4478.576	0.0002	14.2403	0.0702	14.0146
90	441.103	0.0023	6287.185	0.0002	14.2533	0.0702	14.0812
95	618.670	0.0016	8823.854	0.0001	14.2626	0.0701	14.1319
100	867.716	0.0012	12381.662	0.0001	14.2693	0.0701	14.1703

8% Interest Factors

n	Single Payment		Equal Payment Series				Uniform gradient-series factor
	Compound-amount factor	Present-worth factor	Compound-amount factor	Sinking-fund factor	Present-worth factor	Capital-recovery factor	
	To find F Given P F/P i,n	To find P Given F P/F i,n	To find F Given A F/A i,n	To find A Given F A/F i,n	To find P Given A P/A i,n	To find A Given P A/P i,n	To find A Given G A/G i,n
1	1.080	0.9259	1.000	1.0000	0.9259	1.0800	0.0000
2	1.166	0.8573	2.080	0.4808	1.7833	0.5608	0.4808
3	1.260	0.7938	3.246	0.3080	2.5771	0.3880	0.9488
4	1.360	0.7350	4.506	0.2219	3.3121	0.3019	1.4040
5	1.469	0.6806	5.867	0.1705	3.9927	0.2505	1.8465
6	1.587	0.6302	7.336	0.1363	4.6229	0.2163	2.2764
7	1.714	0.5835	8.923	0.1121	5.2064	0.1921	2.6937
8	1.851	0.5403	10.637	0.0940	5.7466	0.1740	2.0985
9	1.999	0.5003	12.488	0.0801	6.2469	0.1601	3.4910
10	2.159	0.4632	14.487	0.0690	6.7101	0.1490	3.8713
11	2.332	0.4289	16.645	0.0601	7.1390	0.1401	4.2395
12	2.518	0.3971	18.977	0.0527	7.5361	0.1327	4.5958
13	2.720	0.3677	21.495	0.0465	7.9038	0.1265	4.9402
14	2.937	0.3405	24.215	0.0413	8.2442	0.1213	5.2731
15	3.172	0.3153	27.152	0.0368	8.5595	0.1168	5.5945
16	3.426	0.2919	30.324	0.0330	8.8514	0.1130	5.9046
17	3.700	0.2703	33.750	0.0296	9.1216	0.1096	6.2038
18	3.996	0.2503	37.450	0.0267	9.3719	0.1067	6.4920
19	4.316	0.2317	41.446	0.0241	9.6036	0.1041	6.7697
20	4.661	0.2146	45.762	0.0219	9.8182	0.1019	7.0370
21	5.034	0.1987	50.423	0.0198	10.0168	0.0998	7.2940
22	5.437	0.1840	55.457	0.0180	10.2008	0.0980	7.5412
23	5.871	0.1703	60.893	0.0164	10.3711	0.0964	7.7786
24	6.341	0.1577	66.765	0.0150	10.5288	0.0950	8.0066
25	6.848	0.1460	73.106	0.0137	10.6748	0.0937	8.2254
26	7.396	0.1352	79.954	0.0125	10.8100	0.0925	8.4352
27	7.988	0.1252	87.351	0.0115	10.9352	0.0915	8.6363
28	8.627	0.1159	95.339	0.0105	11.0511	0.0905	8.8289
29	9.317	0.1073	103.966	0.0096	11.1584	0.0896	9.0133
30	10.063	0.0994	113.283	0.0088	11.2578	0.0888	9.1897
31	10.868	0.0920	123.346	0.0081	11.3498	0.0881	9.3584
32	11.737	0.0852	134.214	0.0075	11.4350	0.0875	9.5197
33	12.676	0.0789	145.951	0.0069	11.5139	0.0869	9.6737
34	13.690	0.0731	158.627	0.0063	11.5869	0.0863	9.8208
35	14.785	0.0676	172.317	0.0058	11.6546	0.0858	9.9611
40	21.725	0.0460	259.057	0.0039	11.9246	0.0839	10.5699
45	31.920	0.0313	386.506	0.0026	12.1084	0.0826	11.0447
50	46.902	0.0213	573.770	0.0018	12.2335	0.0818	11.4107
55	68.914	0.0145	848.923	0.0012	12.3186	0.0812	11.6902
60	101.257	0.0099	1253.213	0.0008	12.3766	0.0808	11.9015
65	148.780	0.0067	1847.248	0.0006	12.4160	0.0806	12.0602
70	218.606	0.0046	2720.080	0.0004	12.4428	0.0804	12.1783
75	321.205	0.0031	4002.557	0.0003	12.4611	0.0803	12.2658
80	471.955	0.0021	5886.935	0.0002	12.4735	0.0802	12.3301
85	693.456	0.0015	8655.706	0.0001	12.4820	0.0801	12.3773
90	1018.915	0.0010	12723.939	0.0001	12.4877	0.0801	12.4116
95	1497.121	0.0007	18701.507	0.0001	12.4917	0.0801	12.4365
100	2199.761	0.0005	27484.516	0.0001	12.4943	0.0800	12.4545

9% Interest Factors

n	Single Payment: Compound-amount factor (To find F Given P, F/P i,n)	Single Payment: Present-worth factor (To find P Given F, P/F i,n)	Equal Payment Series: Compound-amount factor (To find F Given A, F/A i,n)	Equal Payment Series: Sinking-fund factor (To find A Given F, A/F i,n)	Equal Payment Series: Present-worth factor (To find P Given A, P/A i,n)	Equal Payment Series: Capital-recovery factor (To find A Given P, A/P i,n)	Uniform gradient-series factor (To find A Given G, A/G i,n)
1	1.090	0.9174	1.000	1.0000	0.9174	1.0900	0.0000
2	1.188	0.8417	2.090	0.4785	1.7591	0.5685	0.4785
3	1.295	0.7722	3.278	0.3051	2.5313	0.3951	0.9426
4	1.412	0.7084	4.573	0.2187	3.2397	0.3087	1.3925
5	1.539	0.6499	5.985	0.1671	3.8897	0.2571	1.8282
6	1.677	0.5963	7.523	0.1329	4.4859	0.2229	2.2498
7	1.828	0.5470	9.200	0.1087	5.0330	0.1987	2.6574
8	1.993	0.5019	11.028	0.0907	5.5348	0.1807	3.0512
9	2.172	0.4604	13.021	0.0768	5.9953	0.1668	3.4312
10	2.367	0.4224	15.193	0.0658	6.4177	0.1558	3.7978
11	2.580	0.3875	17.560	0.0570	6.8052	0.1470	4.1510
12	2.813	0.3555	20.141	0.0497	7.1607	0.1397	4.4910
13	3.066	0.3262	22.953	0.0436	7.4869	0.1336	4.8182
14	3.342	0.2993	26.019	0.0384	7.7862	0.1284	5.1326
15	3.642	0.2745	29.361	0.0341	8.0607	0.1241	5.4346
16	3.970	0.2519	33.003	0.0303	8.3126	0.1203	5.7245
17	4.328	0.2311	36.974	0.0271	8.5436	0.1171	6.0024
18	4.717	0.2120	41.301	0.0242	8.7556	0.1142	6.2687
19	5.142	0.1945	46.018	0.0217	8.9501	0.1117	6.5236
20	5.604	0.1784	51.160	0.0196	9.1286	0.1096	6.7675
21	6.109	0.1637	56.765	0.0176	9.2923	0.1076	7.0006
22	6.659	0.1502	62.873	0.0159	9.4424	0.1059	7.2232
23	7.258	0.1378	69.532	0.0144	9.5802	0.1044	7.4358
24	7.911	0.1264	76.790	0.0130	9.7066	0.1030	7.6384
25	8.623	0.1160	84.701	0.0118	9.8226	0.1018	7.8316
26	9.399	0.1064	93.324	0.0107	9.9290	0.1007	8.0156
27	10.245	0.0976	102.723	0.0097	10.0266	0.0997	8.1906
28	11.167	0.0896	112.968	0.0089	10.1161	0.0989	8.3572
29	12.172	0.0822	124.135	0.0081	10.1983	0.0981	8.5154
30	13.268	0.0754	136.308	0.0073	10.2737	0.0973	8.6657
31	14.462	0.0692	149.575	0.0067	10.3428	0.0967	8.8083
32	15.763	0.0634	164.037	0.0061	10.4063	0.0961	8.9436
33	17.182	0.0582	179.800	0.0056	10.4645	0.0956	9.0718
34	18.728	0.0534	196.982	0.0051	10.5178	0.0951	9.1933
35	20.414	0.0490	215.711	0.0046	10.5668	0.0946	9.3083
40	31.409	0.0318	337.882	0.0030	10.7574	0.0930	9.7957
45	48.327	0.0207	525.859	0.0019	10.8812	0.0919	10.1603
50	74.358	0.0135	815.084	0.0012	10.9617	0.0912	10.4295
55	114.408	0.0088	1260.092	0.0008	11.0140	0.0908	10.6261
60	176.031	0.0057	1944.792	0.0005	11.0480	0.0905	10.7683
65	270.846	0.0037	2998.288	0.0003	11.0701	0.0903	10.8702
70	416.730	0.0024	4619.223	0.0002	11.0845	0.0902	10.9427
75	641.191	0.0016	7113.232	0.0002	11.0938	0.0902	10.9940
80	986.552	0.0010	10950.574	0.0001	11.0999	0.0901	11.0299
85	1517.932	0.0007	16854.800	0.0001	11.1038	0.0901	11.0551
90	2335.527	0.0004	25939.184	0.0001	11.1064	0.0900	11.0726
95	3593.497	0.0003	39916.635	0.0000	11.1080	0.0900	11.0847
100	5529.041	0.0002	61422.675	0.0000	11.1091	0.0900	11.0930

10% Interest Factors

n	Single Payment: Compound-amount factor (To find F Given P, F/P i,n)	Single Payment: Present-worth factor (To find P Given F, P/F i,n)	Equal Payment Series: Compound-amount factor (To find F Given A, F/A i,n)	Equal Payment Series: Sinking-fund factor (To find A Given F, A/F i,n)	Equal Payment Series: Present-worth factor (To find P Given A, P/A i,n)	Equal Payment Series: Capital-recovery factor (To find A Given P, A/P i,n)	Uniform gradient-series factor (To find A Given G, A/G i,n)
1	1.100	0.9091	1.000	1.0000	0.9091	1.1000	0.0000
2	1.210	0.8265	2.100	0.4762	1.7355	0.5762	0.4762
3	1.331	0.7513	3.310	0.3021	2.4869	0.4021	0.9366
4	1.464	0.6830	4.641	0.2155	3.1699	0.3155	1.3812
5	1.611	0.6209	6.105	0.1638	3.7908	0.2638	1.8101
6	1.772	0.5645	7.716	0.1296	4.3553	0.2296	2.2236
7	1.949	0.5132	9.487	0.1054	4.8684	0.2054	2.6216
8	2.144	0.4665	11.436	0.0875	5.3349	0.1875	3.0045
9	2.358	0.4241	13.579	0.0737	5.7590	0.1737	3.3724
10	2.594	0.3856	15.937	0.0628	6.1446	0.1628	3.7255
11	2.853	0.3505	18.531	0.0540	6.4951	0.1540	4.0641
12	3.138	0.3186	21.384	0.0468	6.8137	0.1468	4.3884
13	3.452	0.2897	24.523	0.0408	7.1034	0.1408	4.6988
14	3.798	0.2633	27.975	0.0358	7.3667	0.1358	4.9955
15	4.177	0.2394	31.772	0.0315	7.6061	0.1315	5.2789
16	4.595	0.2176	35.950	0.0278	7.8237	0.1278	5.5493
17	5.054	0.1979	40.545	0.0247	8.0216	0.1247	5.8071
18	5.560	0.1799	45.599	0.0219	8.2014	0.1219	6.0526
19	6.116	0.1635	51.159	0.0196	8.3649	0.1196	6.2861
20	6.728	0.1487	57.275	0.0175	8.5136	0.1175	6.5081
21	7.400	0.1351	64.003	0.0156	8.6487	0.1156	6.7189
22	8.140	0.1229	71.403	0.0140	8.7716	0.1140	6.9189
23	8.954	0.1117	79.453	0.0126	8.8832	0.1126	7.1085
24	9.850	0.1015	88.497	0.0113	8.9848	0.1113	7.2881
25	10.835	0.0923	98.347	0.0102	9.0771	0.1102	7.4580
26	11.918	0.0839	109.182	0.0092	9.1610	0.1092	7.6187
27	13.110	0.0763	121.100	0.0083	9.2372	0.1083	7.7704
28	14.421	0.0694	134.210	0.0075	9.3066	0.1075	7.9137
29	15.863	0.0630	148.631	0.0067	9.3696	0.1067	8.0489
30	17.449	0.0573	164.494	0.0061	9.4269	0.1061	8.1762
31	19.194	0.0521	181.943	0.0055	9.4790	0.1055	8.2962
32	21.114	0.0474	201.138	0.0050	9.5264	0.1050	8.4091
33	23.225	0.0431	222.252	0.0045	9.5694	0.1045	8.5152
34	25.548	0.0392	245.477	0.0041	9.6086	0.1041	8.6149
35	28.102	0.0356	271.024	0.0037	9.6442	0.1037	8.7086
40	45.259	0.0221	442.593	0.0023	9.7791	0.1023	9.0962
45	72.890	0.0137	718.905	0.0014	9.8628	0.1014	9.3741
50	117.391	0.0085	1163.909	0.0009	9.9148	0.1009	9.5704
55	189.059	0.0053	1880.591	0.0005	9.9471	0.1005	9.7075
60	304.482	0.0033	3034.816	0.0003	9.9672	0.1003	9.8023
65	490.371	0.0020	4893.707	0.0002	9.9796	0.1002	9.8672
70	789.747	0.0013	7887.470	0.0001	9.9873	0.1001	9.9113
75	1271.895	0.0008	12708.954	0.0001	9.9921	0.1001	9.9410
80	2048.400	0.0005	20474.002	0.0001	9.9951	0.1001	9.9609
85	3298.969	0.0003	32979.690	0.0000	9.9970	0.1000	9.9742
90	5313.023	0.0002	53120.226	0.0000	9.9981	0.1000	9.9831
95	8556.676	0.0001	85556.760	0.0000	9.9988	0.1000	9.9889
100	13780.612	0.0001	137796.123	0.0000	9.9993	0.1000	9.9928

12% Interest Factors

n	Single Payment: Compound-amount factor	Single Payment: Present-worth factor	Equal Payment Series: Compound-amount factor	Equal Payment Series: Sinking-fund factor	Equal Payment Series: Present-worth factor	Equal Payment Series: Capital-recovery factor	Uniform gradient-series factor
	To find F Given P $F/P\ i,n$	To find P Given F $P/F\ i,n$	To find F Given A $F/A\ i,n$	To find A Given F $A/F\ i,n$	To find P Given A $P/A\ i,n$	To find A Given P $A/P\ i,n$	To find A Given G $A/G\ i,n$
1	1.120	0.8929	1.000	1.0000	0.8929	1.1200	0.0000
2	1.254	0.7972	2.120	0.4717	1.6901	0.5917	0.4717
3	1.405	0.7118	3.374	0.2964	2.4018	0.4164	0.9246
4	1.574	0.6355	4.779	0.2092	3.0374	0.3292	1.3589
5	1.762	0.5674	6.353	0.1574	3.6048	0.2774	1.7746
6	1.974	0.5066	8.115	0.1232	4.1114	0.2432	2.1721
7	2.211	0.4524	10.089	0.0991	4.5638	0.2191	2.5515
8	2.476	0.4039	12.300	0.0813	4.9676	0.2013	2.9132
9	2.773	0.3606	14.776	0.0677	5.3283	0.1877	3.2574
10	3.106	0.3220	17.549	0.0570	5.6502	0.1770	3.5847
11	3.479	0.2875	20.655	0.0484	5.9377	0.1684	3.8953
12	3.896	0.2567	24.133	0.0414	6.1944	0.1614	4.1897
13	4.364	0.2292	28.029	0.0357	6.4236	0.1557	4.4683
14	4.887	0.2046	32.393	0.0309	6.6282	0.1509	4.7317
15	5.474	0.1827	37.280	0.0268	6.8109	0.1468	4.9803
16	6.130	0.1631	42.753	0.0234	6.9740	0.1434	5.2147
17	6.866	0.1457	48.884	0.0205	7.1196	0.1405	5.4353
18	7.690	0.1300	55.750	0.0179	7.2497	0.1379	5.6427
19	8.613	0.1161	63.440	0.0158	7.3658	0.1358	5.8375
20	9.646	0.1037	72.052	0.0139	7.4695	0.1339	6.0202
21	10.804	0.0926	81.699	0.0123	7.5620	0.1323	6.1913
22	12.100	0.0827	92.503	0.0108	7.6447	0.1308	6.3514
23	13.552	0.0738	104.603	0.0096	7.7184	0.1296	6.5010
24	15.179	0.0659	118.155	0.0085	7.7843	0.1285	6.6407
25	17.000	0.0588	133.334	0.0075	7.8431	0.1275	6.7708
26	19.040	0.0525	150.334	0.0067	7.8957	0.1267	6.8921
27	21.325	0.0469	169.374	0.0059	7.9426	0.1259	7.0049
28	23.884	0.0419	190.699	0.0053	7.9844	0.1253	7.1098
29	26.750	0.0374	214.583	0.0047	8.0218	0.1247	7.2071
30	29.960	0.0334	241.333	0.0042	8.0552	0.1242	7.2974
31	33.555	0.0298	217.293	0.0037	8.0850	0.1237	7.3811
32	37.582	0.0266	304.848	0.0033	8.1116	0.1233	7.4586
33	42.092	0.0238	342.429	0.0029	8.1354	0.1229	7.5303
34	47.143	0.0212	384.521	0.0026	8.1566	0.1226	7.5965
35	52.800	0.0189	431.664	0.0023	8.1755	0.1223	7.6577
40	93.051	0.0108	767.091	0.0013	8.2438	0.1213	7.8988
45	163.988	0.0061	1358.230	0.0007	8.2825	0.1207	8.0572
50	289.002	0.0035	2400.018	0.0004	8.3045	0.1204	8.1597

15% Interest Factors

n	Single Payment: Compound-amount factor	Single Payment: Present-worth factor	Equal Payment Series: Compound-amount factor	Equal Payment Series: Sinking-fund factor	Equal Payment Series: Present-worth factor	Equal Payment Series: Capital-recovery factor	Uniform gradient-series factor
	To find F Given P $F/P\ i,n$	To find P Given F $P/F\ i,n$	To find F Given A $F/A\ i,n$	To find A Given F $A/F\ i,n$	To find P Given A $P/A\ i,n$	To find A Given P $A/P\ i,n$	To find A Given G $A/G\ i,n$
1	1.150	0.8696	1.000	1.0000	0.8696	1.1500	0.0000
2	1.323	0.7562	2.150	0.4651	1.6257	0.6151	0.4651
3	1.521	0.6575	3.473	0.2880	2.2832	0.4380	0.9071
4	1.749	0.5718	4.993	0.2003	2.8550	0.3503	1.3263
5	2.011	0.4972	6.742	0.1483	3.3522	0.2983	1.7228
6	2.313	0.4323	8.754	0.1142	3.7845	0.2642	2.0972
7	2.660	0.3759	11.067	0.0904	4.1604	0.2404	2.4499
8	3.059	0.3269	13.727	0.0729	4.4873	0.2229	2.7813
9	3.518	0.2843	16.786	0.0596	4.7716	0.2096	3.0922
10	4.046	0.2472	20.304	0.0493	5.0188	0.1993	3.3832
11	4.652	0.2150	24.349	0.0411	5.2337	0.1911	3.6550
12	5.350	0.1869	29.002	0.0345	5.4206	0.1845	3.9082
13	6.153	0.1625	34.352	0.0291	5.5832	0.1791	4.1438
14	7.076	0.1413	40.505	0.0247	5.7245	0.1747	4.3624
15	8.137	0.1229	47.580	0.0210	5.8474	0.1710	4.5650
16	9.358	0.1069	55.717	0.0180	5.9542	0.1680	4.7523
17	10.761	0.0929	65.075	0.0154	6.0472	0.1654	4.9251
18	12.375	0.0808	75.836	0.0132	6.1280	0.1632	5.0843
19	14.232	0.0703	88.212	0.0113	6.1982	0.1613	5.2307
20	16.367	0.0611	102.444	0.0098	6.2593	0.1598	5.3651
21	18.822	0.0531	118.810	0.0084	6.3125	0.1584	5.4883
22	21.645	0.0462	137.632	0.0073	6.3587	0.1573	5.6010
23	24.891	0.0402	159.276	0.0063	6.3988	0.1563	5.7040
24	28.625	0.0349	184.168	0.0054	6.4338	0.1554	5.7979
25	32.919	0.0304	212.793	0.0047	6.4642	0.1547	5.8834
26	37.857	0.0264	245.712	0.0041	6.4906	0.1541	5.9612
27	43.535	0.0230	283.569	0.0035	6.5135	0.1535	6.0319
28	50.066	0.0200	327.104	0.0031	6.5335	0.1531	6.0960
29	57.575	0.0174	377.170	0.0027	6.5509	0.1527	6.1541
30	66.212	0.0151	434.745	0.0023	6.5660	0.1523	6.2066
31	76.144	0.0131	500.957	0.0020	6.5791	0.1520	6.2541
32	87.565	0.0114	577.100	0.0017	6.5905	0.1517	6.2970
33	100.700	0.0099	664.666	0.0015	6.6005	0.1515	6.3357
34	115.805	0.0086	765.365	0.0013	6.6091	0.1513	6.3705
35	133.176	0.0075	881.170	0.0011	6.6166	0.1511	6.4019
40	267.864	0.0037	1779.090	0.0006	6.6418	0.1506	6.5168
45	538.769	0.0019	3585.128	0.0003	6.6543	0.1503	6.5830
50	1083.657	0.0009	7217.716	0.0002	6.6605	0.1501	6.6205

20% Interest Factors

n	Single Payment: Compound-amount factor (To find F Given P, F/P i,n)	Single Payment: Present-worth factor (To find P Given F, P/F i,n)	Equal Payment Series: Compound-amount factor (To find F Given A, F/A i,n)	Equal Payment Series: Sinking-fund factor (To find A Given F, A/F i,n)	Equal Payment Series: Present-worth factor (To find P Given A, P/A i,n)	Equal Payment Series: Capital-recovery factor (To find A Given P, A/P i,n)	Uniform gradient-series factor (To find A Given G, A/G i,n)
1	1.200	0.8333	1.000	1.0000	0.8333	1.2000	0.0000
2	1.440	0.6945	2.200	0.4546	1.5278	0.6546	0.4546
3	1.728	0.5787	3.640	0.2747	2.1065	0.4747	0.8791
4	2.074	0.4823	5.368	0.1863	2.5887	0.3863	1.2742
5	2.488	0.4019	7.442	0.1344	2.9906	0.3344	1.6405
6	2.986	0.3349	9.930	0.1007	3.3255	0.3007	1.9788
7	3.583	0.2791	12.916	0.0774	3.6046	0.2774	2.2902
8	4.300	0.2326	16.499	0.0606	3.8372	0.2606	2.5756
9	5.160	0.1938	20.799	0.0481	4.0310	0.2481	2.8364
10	6.192	0.1615	25.959	0.0385	4.1925	0.2385	3.0739
11	7.430	0.1346	32.150	0.0311	4.3271	0.2311	3.2893
12	8.916	0.1122	39.581	0.0253	4.4392	0.2253	3.4841
13	10.699	0.0935	48.497	0.0206	4.5327	0.2206	3.6597
14	12.839	0.0779	59.196	0.0169	4.6106	0.2169	3.8175
15	15.407	0.0649	72.035	0.0139	4.6755	0.2139	3.9589
16	18.488	0.0541	87.442	0.0114	4.7296	0.2114	4.0851
17	22.186	0.0451	105.931	0.0095	4.7746	0.2095	4.1976
18	26.623	0.0376	128.117	0.0078	4.8122	0.2078	4.2975
19	31.948	0.0313	154.740	0.0065	4.8435	0.2065	4.3861
20	38.338	0.0261	186.688	0.0054	4.8696	0.2054	4.4644
21	46.005	0.0217	225.026	0.0045	4.8913	0.2045	4.5334
22	55.206	0.0181	271.031	0.0037	4.9094	0.2037	4.5942
23	66.247	0.0151	326.237	0.0031	4.9245	0.2031	4.6475
24	79.497	0.0126	392.484	0.0026	4.9371	0.2026	4.6943
25	95.396	0.0105	471.981	0.0021	4.9476	0.2021	4.7352
26	114.475	0.0087	567.377	0.0018	4.9563	0.2018	4.7709
27	137.371	0.0073	681.853	0.0015	4.9636	0.2015	4.8020
28	164.845	0.0061	819.223	0.0012	4.9697	0.2012	4.8291
29	197.814	0.0051	984.068	0.0010	4.9747	0.2010	4.8527
30	237.376	0.0042	1181.882	0.0009	4.9789	0.2009	4.8731
31	284.852	0.0035	1419.258	0.0007	4.9825	0.2007	4.8908
32	341.822	0.0029	1704.109	0.0006	4.9854	0.2006	4.9061
33	410.186	0.0024	2045.931	0.0005	4.9878	0.2005	4.9194
34	492.224	0.0020	2456.118	0.0004	4.9899	0.2004	4.9308
35	590.668	0.0017	2948.341	0.0003	4.9915	0.2003	4.9407
40	1469.772	0.0007	7343.858	0.0002	4.9966	0.2001	4.9728
45	3657.262	0.0003	18281.310	0.0001	4.9986	0.2001	4.9877
50	9100.438	0.0001	45497.191	0.0000	4.9995	0.2000	4.9945

25% Interest Factors

n	Single Payment: Compound-amount factor (To find F Given P, F/P i,n)	Single Payment: Present-worth factor (To find P Given F, P/F i,n)	Equal Payment Series: Compound-amount factor (To find F Given A, F/A i,n)	Equal Payment Series: Sinking-fund factor (To find A Given F, A/F i,n)	Equal Payment Series: Present-worth factor (To find P Given A, P/A i,n)	Equal Payment Series: Capital-recovery factor (To find A Given P, A/P i,n)	Uniform gradient-series factor (To find A Given G, A/G i,n)
1	1.250	0.8000	1.000	1.0000	0.8000	1.2500	0.0000
2	1.563	0.6400	2.250	0.4445	1.4400	0.6945	0.4445
3	1.953	0.5120	3.813	0.2623	1.9520	0.5123	0.8525
4	2.441	0.4096	5.766	0.1735	2.3616	0.4235	1.2249
5	3.052	0.3277	8.207	0.1219	2.6893	0.3719	1.5631
6	3.815	0.2622	11.259	0.0888	2.9514	0.3388	1.8683
7	4.768	0.2097	15.073	0.0664	3.1611	0.3164	2.1424
8	5.960	0.1678	19.842	0.0504	3.3289	0.3004	2.3873
9	7.451	0.1342	25.802	0.0388	3.4631	0.2888	2.6048
10	9.313	0.1074	33.253	0.0301	3.5705	0.2801	2.7971
11	11.642	0.0859	42.566	0.0235	3.6564	0.2735	2.9663
12	14.552	0.0687	54.208	0.0185	3.7251	0.2685	3.1145
13	18.190	0.0550	68.760	0.0146	3.7801	0.2646	3.2438
14	22.737	0.0440	86.949	0.0115	3.8241	0.2615	3.3560
15	28.422	0.0352	109.687	0.0091	3.8593	0.2591	3.4530
16	35.527	0.0282	138.109	0.0073	3.8874	0.2573	3.5366
17	44.409	0.0225	173.636	0.0058	3.9099	0.2558	3.6084
18	55.511	0.0180	218.045	0.0046	3.9280	0.2546	3.6698
19	69.389	0.0144	273.556	0.0037	3.9424	0.2537	3.7222
20	86.736	0.0115	342.945	0.0029	3.9539	0.2529	3.7667
21	108.420	0.0092	429.681	0.0023	3.9631	0.2523	3.8045
22	135.525	0.0074	538.101	0.0019	3.9705	0.2519	3.8365
23	169.407	0.0059	673.626	0.0015	3.9764	0.2515	3.8634
24	211.758	0.0047	843.033	0.0012	3.9811	0.2512	3.8861
25	264.698	0.0038	1054.791	0.0010	3.9849	0.2510	3.9052
26	330.872	0.0030	1319.489	0.0008	3.9879	0.2508	3.9212
27	413.590	0.0024	1650.361	0.0006	3.9903	0.2506	3.9346
28	516.988	0.0019	2063.952	0.0005	3.9923	0.2505	3.9457
29	646.235	0.0016	2580.939	0.0004	3.9938	0.2504	3.9551
30	807.794	0.0012	3227.174	0.0003	3.9951	0.2503	3.9628
31	1009.742	0.0010	4034.968	0.0003	3.9960	0.2503	3.9693
32	1262.177	0.0008	5044.710	0.0002	3.9968	0.2502	3.9746
33	1577.722	0.0006	6306.887	0.0002	3.9975	0.2502	3.9791
34	1972.152	0.0005	7884.609	0.0001	3.9980	0.2501	3.9828
35	2465.190	0.0004	9856.761	0.0001	3.9984	0.2501	3.9858

30% Interest Factors

n	Single Payment		Equal Payment Series				Uniform gradient-series factor
	Compound-amount factor	Present-worth factor	Compound-amount factor	Sinking-fund factor	Present-worth factor	Capital-recovery factor	
	To find F Given P F/P i,n	To find P Given F P/F i,n	To find F Given A F/A i,n	To find A Given F A/F i,n	To find P Given A P/A i,n	To find A Given P A/P i,n	To find A Given G A/G i,n
1	1.300	0.7692	1.000	1.0000	0.7692	1.3000	0.0000
2	1.690	0.5917	2.300	0.4348	1.3610	0.7348	0.4348
3	2.197	0.4552	3.990	0.2506	1.8161	0.5506	0.8271
4	2.856	0.3501	6.187	0.1616	2.1663	0.4616	1.1783
5	3.713	0.2693	9.043	0.1106	2.4356	0.4106	1.4903
6	4.827	0.2072	12.756	0.0784	2.6428	0.3784	1.7655
7	6.275	0.1594	17.583	0.0569	2.8021	0.3569	2.0063
8	8.157	0.1226	23.858	0.0419	2.9247	0.3419	2.2156
9	10.605	0.0943	32.015	0.0312	3.0190	0.3312	2.3963
10	13.786	0.0725	42.620	0.0235	3.0915	0.3235	2.5512
11	17.922	0.0558	56.405	0.0177	3.1473	0.3177	2.6833
12	23.298	0.0429	74.327	0.0135	3.1903	0.3135	2.7952
13	30.288	0.0330	97.625	0.0103	3.2233	0.3103	2.8895
14	39.374	0.0254	127.913	0.0078	3.2487	0.3078	2.9685
15	51.186	0.0195	167.286	0.0060	3.2682	0.3060	3.0345
16	66.542	0.0150	218.472	0.0046	3.2832	0.3046	3.0892
17	86.504	0.0116	285.014	0.0035	3.2948	0.3035	3.1345
18	112.455	0.0089	371.518	0.0027	3.3037	0.3027	3.1718
19	146.192	0.0069	483.973	0.0021	3.3105	0.3021	3.2025
20	190.050	0.0053	630.165	0.0016	3.3158	0.3016	3.2276
21	247.065	0.0041	820.215	0.0012	3.3199	0.3012	3.2480
22	321.184	0.0031	1067.280	0.0009	3.3230	0.3009	3.2646
23	417.539	0.0024	1388.464	0.0007	3.3254	0.3007	3.2781
24	542.801	0.0019	1806.003	0.0006	3.3272	0.3006	3.2890
25	705.641	0.0014	2348.803	0.0004	3.3286	0.3004	3.2979
26	917.333	0.0011	3054.444	0.0003	3.3297	0.3003	3.3050
27	1192.533	0.0008	3971.778	0.0003	3.3305	0.3003	3.3107
28	1550.293	0.0007	5164.311	0.0002	3.3312	0.3002	3.3153
29	2015.381	0.0005	6714.604	0.0002	3.3317	0.3002	3.3189
30	2619.996	0.0004	8729.985	0.0001	3.3321	0.3001	3.3219
31	3405.994	0.0003	11349.981	0.0001	3.3324	0.3001	3.3242
32	4427.793	0.0002	14755.975	0.0001	3.3326	0.3001	3.3261
33	5756.130	0.0002	19183.768	0.0001	3.3328	0.3001	3.3276
34	7482.970	0.0001	24939.899	0.0001	3.3329	0.3001	3.3288
35	9727.860	0.0001	32422.868	0.0000	3.3330	0.3000	3.3297

40% Interest Factors

n	Single Payment		Equal Payment Series				Uniform gradient-series factor
	Compound-amount factor	Present-worth factor	Compound-amount factor	Sinking-fund factor	Present-worth factor	Capital-recovery factor	
	To find F Given P F/P i,n	To find P Given F P/F i,n	To find F Given A F/A i,n	To find A Given F A/F i,n	To find P Given A P/A i,n	To find A Given P A/P i,n	To find A Given G A/G i,n
1	1.400	0.7143	1.000	1.0001	0.7143	1.4001	0.0000
2	1.960	0.5103	2.400	0.4167	1.2245	0.8167	0.4167
3	2.744	0.3645	4.360	0.2294	1.5890	0.6294	0.7799
4	3.842	0.2604	7.104	0.1408	1.8493	0.5408	1.0924
5	5.378	0.1860	10.946	0.0914	2.0352	0.4914	1.3580
6	7.530	0.1329	16.324	0.0613	2.1680	0.4613	1.5811
7	10.541	0.0949	23.853	0.0420	2.2629	0.4420	1.7664
8	14.758	0.0678	34.395	0.0291	2.3306	0.4291	1.9186
9	20.661	0.0485	49.153	0.0204	2.3790	0.4204	2.0423
10	28.925	0.0346	69.814	0.0144	2.4136	0.4144	2.1420
11	40.496	0.0247	98.739	0.0102	2.4383	0.4102	2.2215
12	56.694	0.0177	139.234	0.0072	2.4560	0.4072	2.2846
13	79.371	0.0126	195.928	0.0052	2.4686	0.4052	2.3342
14	111.120	0.0090	275.299	0.0037	2.4775	0.4037	2.3729
15	155.568	0.0065	386.419	0.0026	2.4840	0.4026	2.4030
16	217.794	0.0046	541.986	0.0019	2.4886	0.4019	2.4262
17	304.912	0.0033	759.780	0.0014	2.4918	0.4014	2.4441
18	426.877	0.0024	1064.691	0.0010	2.4942	0.4010	2.4578
19	597.627	0.0017	1491.567	0.0007	2.4959	0.4007	2.4682
20	836.678	0.0012	2089.195	0.0005	2.4971	0.4005	2.4761
21	1171.348	0.0009	2925.871	0.0004	2.4979	0.4004	2.4821
22	1639.887	0.0007	4097.218	0.0003	2.4985	0.4003	2.4866
23	2295.842	0.0005	5737.105	0.0002	2.4990	0.4002	2.4900
24	3214.178	0.0004	8032.945	0.0002	2.4993	0.4002	2.4926
25	4499.847	0.0003	11247.110	0.0001	2.4995	0.4001	2.4945
26	6299.785	0.0002	15746.960	0.0001	2.4997	0.4001	2.4959
27	8819.695	0.0002	22046.730	0.0001	2.4998	0.4001	2.4970
28	12347.570	0.0001	30866.430	0.0001	2.4998	0.4001	2.4978
29	17286.590	0.0001	43213.990	0.0001	2.4999	0.4001	2.4984
30	24201.230	0.0001	60500.580	0.0001	2.4999	0.4001	2.4988

50% Interest Factors

n	Single Payment		Equal Payment Series				Uniform gradient-series factor
	Compound-amount factor	Present-worth factor	Compound-amount factor	Sinking-fund factor	Present-worth factor	Capital-recovery factor	
	To find *F* Given *P* *F/P i,n*	To find *P* Given *F* *P/F i,n*	To find *F* Given *A* *F/A i,n*	To find *A* Given *F* *A/F i,n*	To find *P* Given *A* *P/A i,n*	To find *A* Given *P* *A/P i,n*	To find *A* Given *G* *A/G i,n*
1	1.500	0.6667	1.000	1.0000	0.6667	1.5000	0.0001
2	2.250	0.4445	2.500	0.4000	1.1112	0.9001	0.4001
3	3.375	0.2963	4.750	0.2106	1.4075	0.7106	0.7369
4	5.063	0.1976	8.125	0.1231	1.6050	0.6231	1.0154
5	7.594	0.1317	13.188	0.0759	1.7367	0.5759	1.2418
6	11.391	0.0878	20.781	0.0482	1.8245	0.5482	1.4226
7	17.086	0.0586	32.172	0.0311	1.8830	0.5311	1.5649
8	25.629	0.0391	49.258	0.0204	1.9220	0.5204	1.6752
9	38.443	0.0261	74.887	0.0134	1.9480	0.5134	1.7597
10	57.665	0.0174	113.330	0.0089	1.9654	0.5089	1.8236
11	86.498	0.0116	170.995	0.0059	1.9769	0.5059	1.8714
12	129.746	0.0078	257.493	0.0039	1.9846	0.5039	1.9068
13	194.620	0.0052	387.239	0.0026	1.9898	0.5026	1.9329
14	291.929	0.0035	581.858	0.0018	1.9932	0.5018	1.9519
15	437.894	0.0023	873.788	0.0012	1.9955	0.5012	1.9657
16	656.841	0.0016	1311.681	0.0008	1.9970	0.5008	1.9757
17	985.261	0.0011	1968.522	0.0006	1.9980	0.5006	1.9828
18	1477.891	0.0007	2953.783	0.0004	1.9987	0.5004	1.9879
19	2216.837	0.0005	4431.671	0.0003	1.9991	0.5003	1.9915
20	3325.256	0.0004	6648.511	0.0002	1.9994	0.5002	1.9940
21	4987.882	0.0003	9973.765	0.0002	1.9996	0.5002	1.9958
22	7481.824	0.0002	14961.640	0.0001	1.9998	0.5001	1.9971
23	11222.730	0.0001	22443.470	0.0001	1.9999	0.5001	1.9980
24	16834.100	0.0001	33666.210	0.0001	1.9999	0.5001	1.9986
25	25251.160	0.0001	50500.330	0.0001	2.0000	0.5001	1.9991

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